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Technical Report

Thor Douta Gold Project PFS
Thor Explorations Ltd.

Southeast, Senegal

In accordance with the requirements of National Instrument 43-101 "Standards of Disclosure for Mineral Projects" of the Canadian Securities Administrators

Qualified Persons:
D Claridge, FAusIMM
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AMC Project 0424072
Effective date 24 January 2026

1 Summary

1.1 Background and purpose of this report

This Technical Report (Thor Douta Gold Project Technical Report, Technical Report, or Report) provides the results of a Preliminary Feasibility Study (PFS) for Thor Explorations Ltd. (Thor or Company) on the Douta Gold Project (Douta, Property, or Project) in Southeast, Senegal. The Technical Report has been prepared by AMC Consultants (UK) Limited (AMC) on behalf of Thor. The Project comprises the Makosa, Makosa Tail, and Baraka 3 resources.

Thor is a Canadian mineral exploration company engaged in the acquisition, exploration, development, and operation of mineral properties located in Nigeria, Senegal, and Côte d'Ivoire. Thor holds a 100% interest in the Segilola Gold Project located in Osun State of Nigeria. Mining and production commenced at Segilola in 2021. Thor also holds a 100% interest in the Douta Project located in south-eastern Senegal.

Douta is planned to be an open pit operation targeting both oxide and sulphide Mineral Resources. The Project will be operated in two Phases. Phase1 – Oxide Ore (Phase 1) will treat oxide ore via a carbon-in-leach (CIL) circuit. Phase 2 – Primary Ore (Phase 2) will treat refractory Primary Ore, initially by roasting followed by CIL.

Thor is listed on both the Canadian TSX Venture Exchange (TSX-V:THX) and the Alternative Investment Market of the London Stock Exchange (AIM:THX). Thor is incorporated federally in Canada under Canadian law, with registered number BC0860183. Major shareholders of the Company are Cds & Co (Company Nominees & Brokerages) 43.2% holding of issued share capital, AFC Equity Investments 15.9%, Computershare Company Nominees Ltd. 8.1%, Segun Lawson 4.9%, Adbro Limited 3.4%, Sparkrod Nigeria Limited 3.3%, Hongkong Tiande Baorun Trade Limited 3.2%, and Nigerian Mining Corporation 3.1%.

This Technical Report has been prepared in accordance with the requirements of National Instrument 43-101 (NI 43-101), "Standards of Disclosure for Mineral Projects" of the Canadian Securities Administrators (CSA) for lodgement on CSA's System for Electronic Document Analysis and Retrieval (SEDAR+).

The effective date of this Technical Report is 24 January 2026.

The Qualified Persons (QPs) under NI 43-101 for the PFS are Mr Dominic Claridge, FAusIMM, Principal Mining Engineer (AMC); Mr Rob Chesher, FAusIMM(CP Met), RPEQ, Senior Principal Consultant (AMC); Mr Alfred Gillman, FAusIMM(CP) (Thor).

1.2 Mineral Resource and Mineral Reserve tables

Mineral Resources (Table 1.1) and Mineral Reserves (Table 1.2) are classified and reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves May 2014 (the CIM Definition Standards).

Table 1.1 Douta Mineral Resources

Classification	Volume (Mm^3)	Average density	Tonnage (Mt)	Grade (g/t Au)	Ounces MAu
Indicated	18.9	2.68	50.6	1.04	1.70
Inferred	3.4	2.72	9.3	0.92	0.27

Notes:
- CIM Definition Standards were followed for Mineral Resources.
- Mineral Resources have been constrained by optimised pit shells based on a gold price of US$4,000/oz.
- Mineral Resources are inclusive of Mineral Reserves.
- Calculated breakeven cut-off grades based on mining costs, metallurgical recovery, milling costs, and G&A costs range between 0.21 g/t to 0.33 g/t across the various deposits.
- Open pit Mineral Resources are reported in situ at a cut-off grade of 0.30 g/t Au which is the average of the individual calculated breakeven cut-offs.
- The Mineral Resource is considered to have reasonable prospects for economic extraction by open pit mining methods above a 0.30 g/t Au and within an optimised pit shell.
- High grade assays were capped at 10 g/t Au.
- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
- Effective date for the Mineral Resource is 24 January 2026.
- Tonnage and grade measurements are in metric units.
- Totals may not compute exactly due to rounding.

Source: Thor, 2025.

Table 1.2 Mineral Reserves

Area	Classification	Oxide		Transitional		Fresh		Total	
		Tonnes (Mt)	Au grade (g/t)	Tonnes (Mt)	Au grade (g/t)	Tonnes (Mt)	Au grade (g/t)	Tonnes (Mt)	Au grade (g/t)
Makosa Main	Probable	8.7	0.88	5.6	0.91	14.1	1.13	28.4	1.01
Makosa Tail	Probable	1.7	0.82	1.2	0.89	4.4	1.25	7.3	1.09
Total Makosa	**Probable**	**10.4**	**0.87**	**6.8**	**0.91**	**18.5**	**1.16**	**35.6**	**1.03**
Baraka 3	Probable	0.8	1.13	0.2	0.98	0.001	1.46	1.0	1.11
Douta total	Probable	11.1	0.89	7.0	0.91	18.5	1.16	36.6	1.03

Notes:
- CIM definition Standards were used for reporting of Mineral Reserves.
- Mineral Reserves are estimated using a long-term gold price of US$3,000 per troy oz for all mining areas.
- Mineral Reserves are stated in terms of delivered tonnes and grade before processing recovery.
- Mineral Reserves are defined by pit optimisation and are based on viable breakeven cut-offs as generated by material type, process destination and metallurgical recoveries.
- Metal recoveries are variable dependent on material type and mining area.
- Open pit dilution and geological ore loss are applied through the regularisation of the Mineral Resource models to an appropriate selective mining unit size.
- Effective date of Mineral Reserves is 24 January 2026.
- Tonnage and grade measurements are in metric units.
- Totals may not compute exactly due to rounding.

Source: AMC, 2026.

1.3 Property description

Thor owns and operates the Douta Project which includes several gold deposits in the central Makosa area together with several deposits located further to the south in the Baraka 3 area. It is Thor's intent to mine the gold deposits on the Douta Property and treat the ore through a planned CIL treatment plant that will be located in the vicinity of the Makosa deposits. A second stage refractory ore treatment plant is to be constructed to complement the CIL plant and treat primary ore mined later in the project life.

Douta is located in south-eastern Senegal, approximately 620 kilometres (km) south-east of the capital city of Dakar and 80 km north of the town of Kedougou. The Douta tenure and associated land holdings are shown in Figure 1.1.

Figure 1.1 Project location map

Source: Thor, October 2025.

1.4 History

The Douta permit was initially part of Randgold Resources's Kounemba permit. This land package comprising three licences (Kounemba, Kanoumering, and Tomboronkoto) was selected based on a mineralised structure that was interpreted from Landsat imagery to extend south from the Sabodala gold deposit (Endeavour Mining) and Niamia Permit in the North, where thick sequences of deformed volcaniclastic rocks including andesitic lithic tuff were found.

The late 2003 and early 2004 regional soil sampling program at 1,000 by 100 metres (m) spacing, identified 11 high-priority targets, for detailed work. Due to the low tenor of the Massawa anomaly, it was originally selected as a secondary target. A follow-up detailed soil sampling grid program was completed in mid-2005, and identified a 3.5 km long, 100 to 400 m wide soil anomaly at greater than 50 parts per billion (ppb) gold in soil.

The area east of Massawa (the present Douta licence) was relinquished by Randgold Resources in 2007-2008.

In 2009, International Mining Company (IMC) was granted this area under the name of Douta permit with an original licence area of 103 square kilometres (km^2). IMC entered into an option agreement in February 2011 with African Star Resources (ASR), a wholly owned subsidiary of Thor for a 70% interest in Douta. In September 2025 Thor acquired the remaining 30% interest from IMC, to take a 100% interest in Douta.

1.5 Geology and mineralisation

The Douta district occurs in the West African (Birimian) Paleoproterozoic metallogenic province, which extends from Senegal and Mali through north-eastern Guinea, Ivory Coast, Ghana, Burkina Faso, and as far east as Niger. Gold deposits in the West African metallogenic district, including those on the Douta permits, show many characteristics consistent with their classification as orogenic (mesothermal) gold deposits and prospects. Orogenic gold systems are structurally controlled deposits formed during regional deformation (orogenic) events.

The Douta Project is located in the West African Craton, within the 2,213 Ma to 2,198 Ma age Kedougou-Kenieba inlier, and features Birimian Dialé sedimentary formations in the east and mafic-volcaniclastic formations of the Mako Belt in the west (Figure 1.2). From south to north, the main structural feature of the exploration licence is the NNE-to-NE striking Main Transcurrent Shear Zone (MTSZ).

The properties comprising the Douta Project can be subdivided into two main resource areas, both of which are hosted by sedimentary and volcano-sedimentary rocks of the Dialé Group:

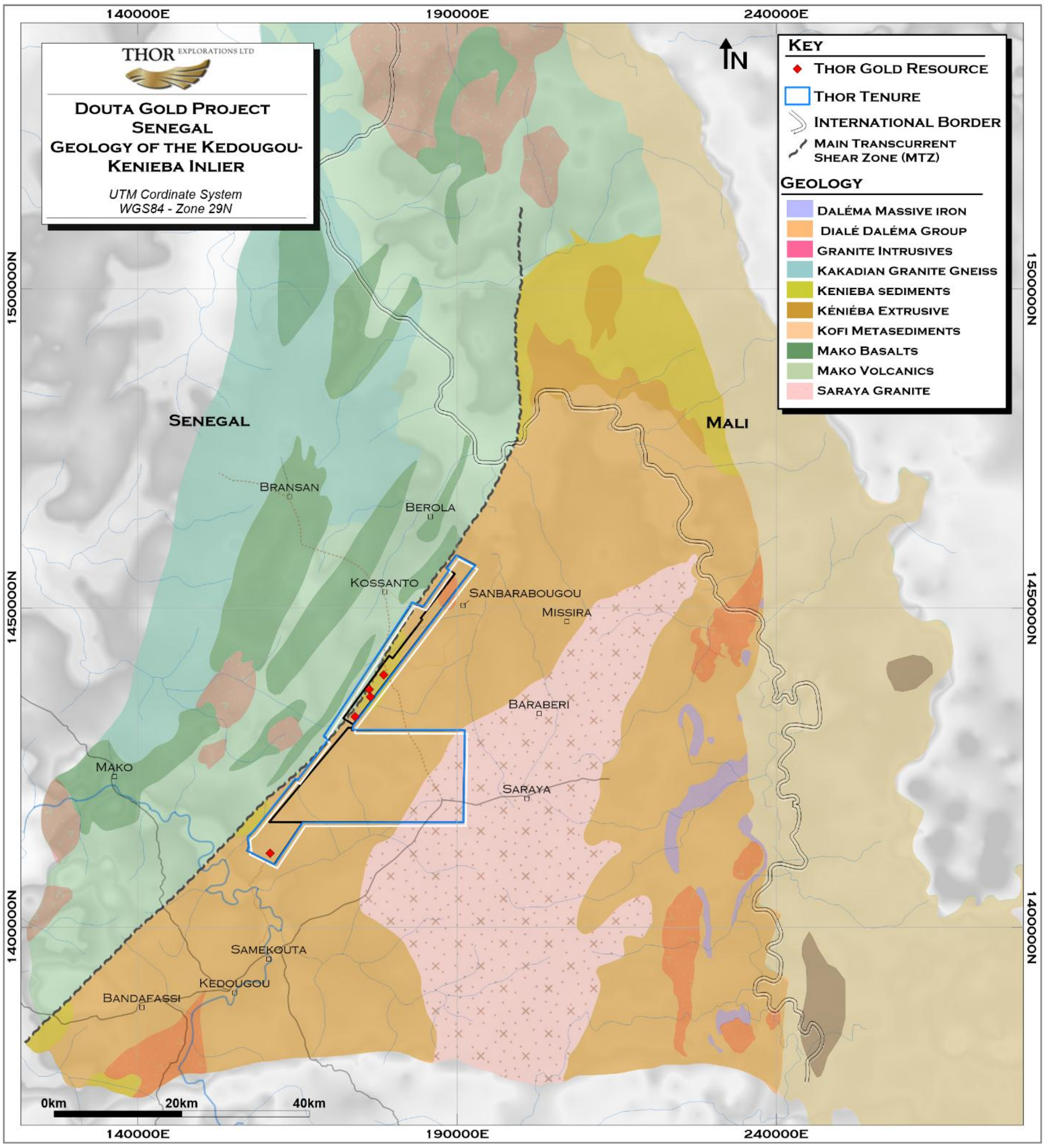
- Makosa, Makosa North, Makosa East, Makosa Tail
- Baraka 3 (consisting of Baraka 3 East and Baraka 3 West)

In addition, there are three exploration areas:

- Sambara
- Maka
- Mansa

Both resource areas present similar lithologies, which generally trend north-northeast with steep dips towards the north-west. The rocks have been metamorphosed to the lower greenschist facies. Turbidite sediments (greywacke), volcano sedimentary rocks, and graphitic shales dominate the sequence, which, in the vicinity of Makosa and Makosa Tail, is conformably intruded by a gabbro dyke. At Makosa East, granitic rocks have been intersected in the footwall sequence.

Figure 1.2 Geology of West Africa and Kedougou-Kenieba Inlier



Source: Thor, 2025.

1.6 Exploration

The Douta Project hosts a large but underexplored regional-scale gold system with strong exploration potential. Existing mineralisation, prospects, and targets remain open along strike and at depth within major structural corridors and the extensive area between them.

The Project comprises permits covering approximately 538 km^2 within the highly prospective Birimian Dialé metasediments. Key exploration areas are located along the MTSZ (hosting the Makosa Tail, Makosa, Makosa North, and Makosa East deposits). Approximately 30 of exploration areas of the MTSZ lie within the Project area.

Additional mineralisation potential exists within and adjacent to these structures, where early-stage exploration has identified numerous prospects along secondary and tertiary structural zones.

Extensive datasets support ongoing target generation, including drilling, soil, and termite mound sampling. Aeromagnetic and electromagnetic surveys are planned to generate further exploration targets. Beyond the defined Mineral Resources, numerous prospects exist that have yet to be drill tested. A phased, property-wide exploration program is ongoing, with data review, target evaluation, and drill prioritisation currently underway.

1.7 Mineral Resources

Mineral Resources for the Douta Project are estimated for five gold deposits and prospects located on the Douta Demande and the Douta West exploration permit. Separate block models cover the Makosa, Makosa Tail, and Baraka 3 East and West deposits. The Makosa block model encompasses the Makosa, Makosa North, and Makosa East deposits. Mineral Resources are reported at an effective date of 24 January 2026 (Table 1.3 and Table 1.4).

The methods, parameters, assumptions, and support data used for the Douta block models, which date back to 2023, were reviewed to ensure they remain current. Models have been updated as required to either include new information or revised cost assumptions such as gold price and operation costs.

The same overall approach was used for each model whereby block grade and density estimates are constrained by domains representing the mineralisation, lithology, and weathering surfaces. Mineral Resources are reported within pit shells generated by AMC. Only classified blocks greater than or equal to the open pit cut-off grades and within the open pit shells are reported.

QP for the Mineral Resource estimates is Alfred Gillman, General Manager (Exploration and Resources) to Thor is not independent. Mr Gillman is a QP in accordance with NI 43-101.

Table 1.3 Douta Project Mineral Resource estimate by deposit at 0.3 g/t Au cut-off

Classification	Deposit	Volume (Mm^3)	Average density	Tonnage (Mt)	Grade (g/t Au)	Ounces MAu
Indicated	Makosa North	3.67	2.69	9.9	1.08	0.34
Indicated	Makosa	7.62	2.71	20.6	1.06	0.70
Indicated	Makosa East	3.11	2.67	8.3	0.92	0.25
Indicated	Makosa Tail	3.92	2.71	10.6	1.03	0.35
Indicated	Baraka 3	0.54	2.11	1.1	1.43	0.05
Indicated	**Total**	**18.9**	**2.68**	**50.6**	**1.04**	**1.70**
Inferred	Makosa North	1.73	2.79	4.8	1.02	0.16
Inferred	Makosa	0.55	2.72	1.5	0.95	0.05
Inferred	Makosa East	0.50	2.68	1.3	0.87	0.04
Inferred	Makosa Tail	0.52	2.67	1.4	0.57	0.03
Inferred	Baraka 3	0.09	2.02	0.2	0.99	0.01
Inferred	**Total**	**3.4**	**2.72**	**9.3**	**0.92**	**0.27**

Notes:
- CIM Definition Standards were followed for Mineral Resources.
- Mineral Resources have been constrained by optimised pit shells based on a gold price of US$4,000/oz.
- Resources are reported inclusive of Reserves.
- Calculated breakeven cut-off grades based on mining costs, metallurgical recovery, milling costs, and G&A costs range between 0.21 g/t to 0.33 g/t across the various deposits.
- Open Pit Mineral Resources are reported in situ at a cut-off grade of 0.30 g/t Au which is the average of the individual calculated breakeven cut-offs.
- The Mineral Resource is considered to have reasonable prospects for economic extraction by open pit mining methods above a 0.30 g/t Au and within an optimised pit shell.
- High grade assays were capped at 10 g/t Au.
- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
- Bulk density is assigned according to weathering profile with a weighted average of 2.71.
- Effective date for the Mineral Resource is 24 January 2026.
- Sum of individual amounts may not equal due to rounding.

Source: Thor, January 2026.

1.8 Mining and Mineral Reserves

Mineral Reserves for the Douta Project are estimated for three gold deposits, namely the Makosa Main, Makosa Tail, and Baraka 3 deposits. The Makosa Main deposit consists of the Makosa, Makosa North, and Makosa East deposits stated in the Mineral Resource.

Mineral Reserves are reported with an effective date of 24 January 2026 (Table 1.4).

Table 1.4 Mineral Reserves

Area	Classification	Oxide		Transitional		Fresh		Total	
		Tonnes (Mt)	Au grade (g/t)	Tonnes (Mt)	Au grade (g/t)	Tonnes (Mt)	Au grade (g/t)	Tonnes (Mt)	Au grade (g/t)
Makosa Main	Probable	8.7	0.88	5.6	0.91	14.1	1.13	28.4	1.01
Makosa Tail	Probable	1.7	0.82	1.2	0.89	4.4	1.25	7.3	1.09
Total Makosa	**Probable**	**10.4**	**0.87**	**6.8**	**0.91**	**18.5**	**1.16**	**35.6**	**1.03**
Baraka 3	Probable	0.8	1.13	0.2	0.98	0.001	1.46	1.0	1.11
Douta total	**Probable**	**11.1**	**0.89**	**7.0**	**0.91**	**18.5**	**1.16**	**36.6**	**1.03**

Notes:
- CIM definition Standards were used for reporting of Mineral Reserves.
- Mineral Reserves are estimated using a long-term gold price of US$3,000 per troy oz for all mining areas.
- Mineral Reserves are stated in terms of delivered tonnes and grade before processing recovery.
- Mineral Reserves are defined by pit optimisation and are based on viable breakeven cut-offs as generated by material type, process destination and metallurgical recoveries.
- Metal recoveries are variable dependent on material type and mining area.
- Open pit dilution and geological ore loss are applied through the regularisation of the Mineral Resource models to an appropriate selective mining unit size.
- Effective date of Mineral Reserves is 24 January 2026.
- Tonnage and grade measurements are in metric units.
- Totals may not compute exactly due to rounding.

Source: AMC, 2026.

Both Geotechnical and Hydrogeological studies have been undertaken at the Douta project. While some gaps in data do exist, particularly for Baraka 3 (consisting of east and west zones), the overall level of information is suitable for a PFS level of study.

Given the geometry and grade of the orebodies, the deposits will be mined by open pits using conventional excavator / truck mining methods. Dilution and ore loss were accounted for through regularisation of the Mineral Resource models. Pit optimisations were undertaken for all four geological areas, Makosa Main, Makosa Tail, Baraka 3 East, and Baraka 3 West. Optimisation parameters were obtained from Thor and validated against existing operations and through discussions with suppliers.

The project schedule is divided into two Phases. Phase 1 treats the oxide / transitional mill feed through a conventional CIL process. Phase 2 treats primary fresh mill feed through a whole ore roasting circuit prior to the CIL circuit. Optimised shells were produced for all deposits for both Phases, with the Phase 1 optimisation shells being used to guide the development of Phase 1 pits, which were then used in conjunction with the Phase 2 shells to develop the ultimate, life-of-mine (LOM) pits. Ramps and berms were included in all designs with ramp width being dictated by proximity to the base of each pit and expected traffic flow. Where practical, a “goodbye cut” was designed at the base of each pit to maximise extraction of mill feed.

Utilising expected production rates and assumed mining equipment, and after comparison with nearby operations, a bench height of 6 m was chosen. Geotechnical conditions allow for up to three benches to be stacked between safety berms. It is expected that each bench will be mined as two flitches to help control dilution, although there may be some opportunity to mine full bench heights in areas of waste to aid production.

Makosa Main contributes 78% of the mill feed and 67% of the total material movement. Makosa Tail contributes 20% of the mill feed and 26% of the total movement with the remainder coming from the Baraka 3 pits (Table 1.5).

Table 1.5 Pit inventories

Phase	Total material			
	Ore tonnes (Mt)	Ore grade (g/t)	Waste tonnes (Mt)	Total tonnes (Mt)
Phase 1				
Makosa Main	12.6	0.91	31.6	44.2
Makosa Tail	2.3	0.90	6.1	8.4
Baraka 3 East	0.4	1.21	5.3	5.7
Baraka 3 West	0.5	1.01	3.9	4.4
Total Phase 1	**15.9**	**0.92**	**46.8**	**62.7**
Phase 2				
Makosa Main	15.8	1.09	84.1	99.8
Makosa Tail	4.9	1.19	41.9	46.8
Total Phase 2	**20.7**	**1.12**	**126.0**	**146.7**
Total	**36.6**	**1.03**	**172.7**	**209.3**

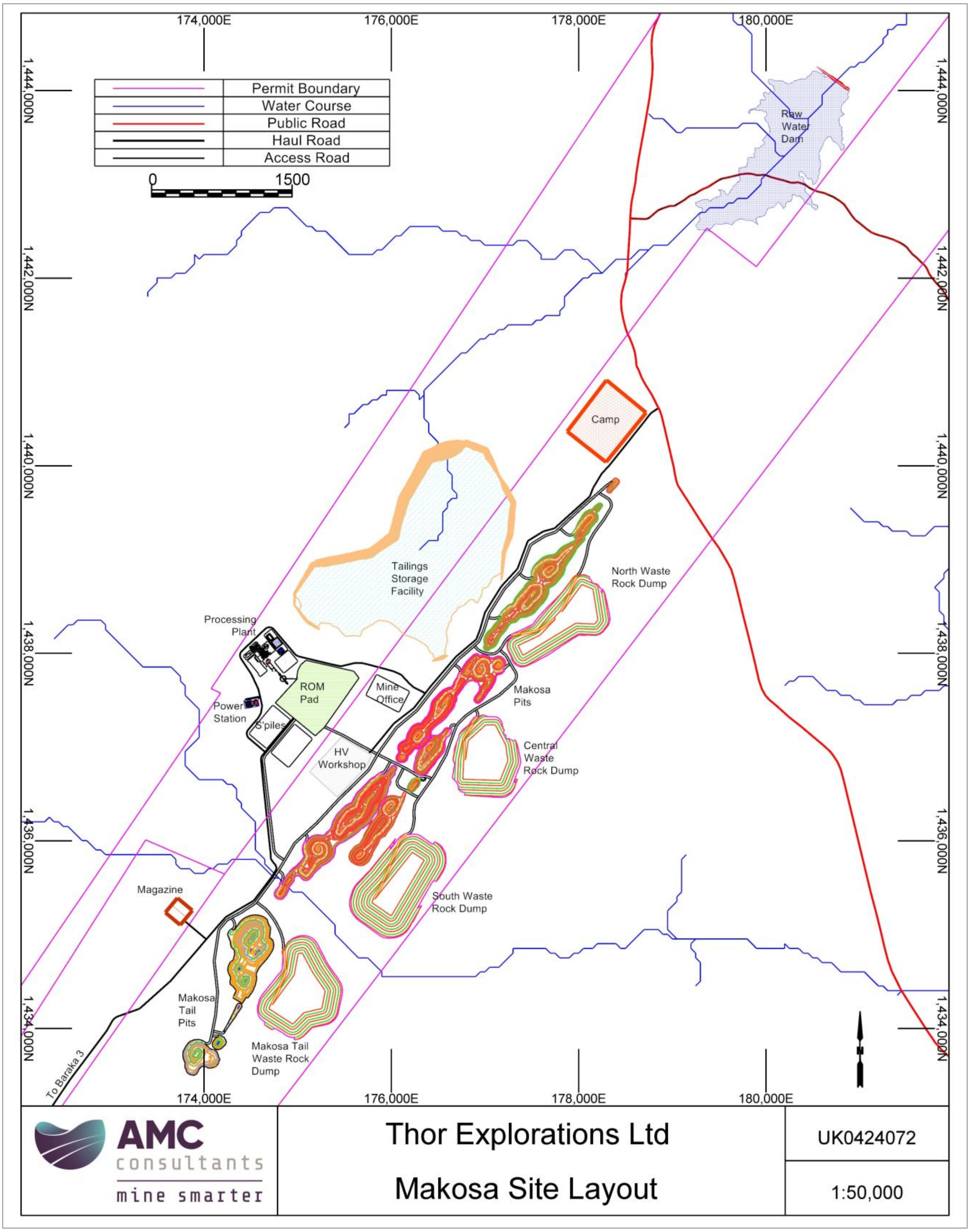
Source: AMC, January 2026.

There is scope for larger pits with increased Indicated Resources, improved geotechnical or financial conditions, and in the case of Baraka 3, a change or addition to the southern lease boundary.

Waste rock dumps were defined for all areas of the project. Given the terrain and land use, there is little restriction on waste dump footprint. However, there is a restriction to the footprint of several of the dumps due to the eastern lease boundary at Makosa Main.

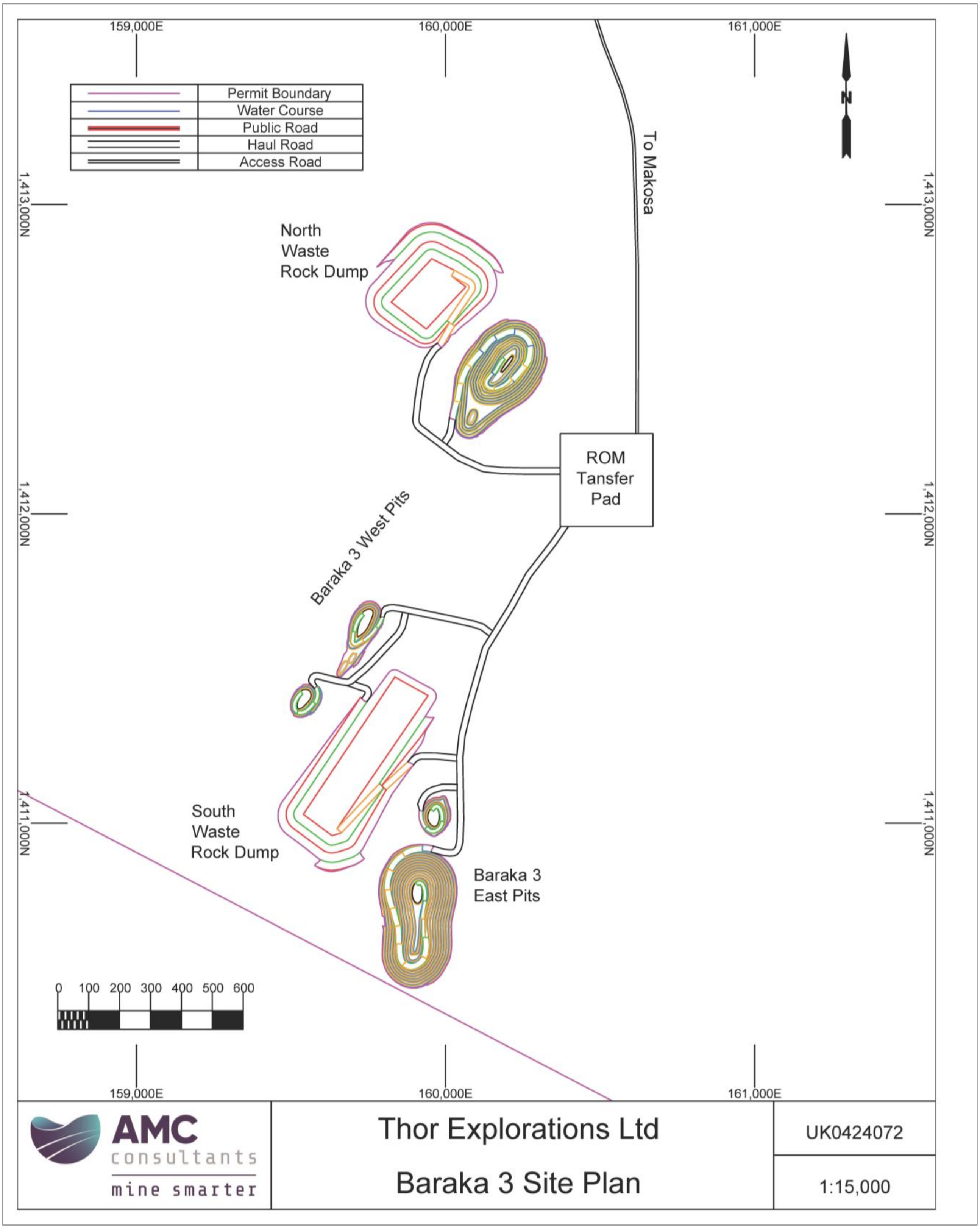
Run-of-mine (ROM) pads, stockpiles, surface haul roads, and access roads were designed in preliminary forms. Figure 1.3 shows the general arrangement for the Makosa site while Figure 1.4 shows that for the Baraka 3 site.

Figure 1.3 General arrangement for the Makosa site



Source: AMC, January 2026.

Figure 1.4 General arrangement for the Baraka 3 site



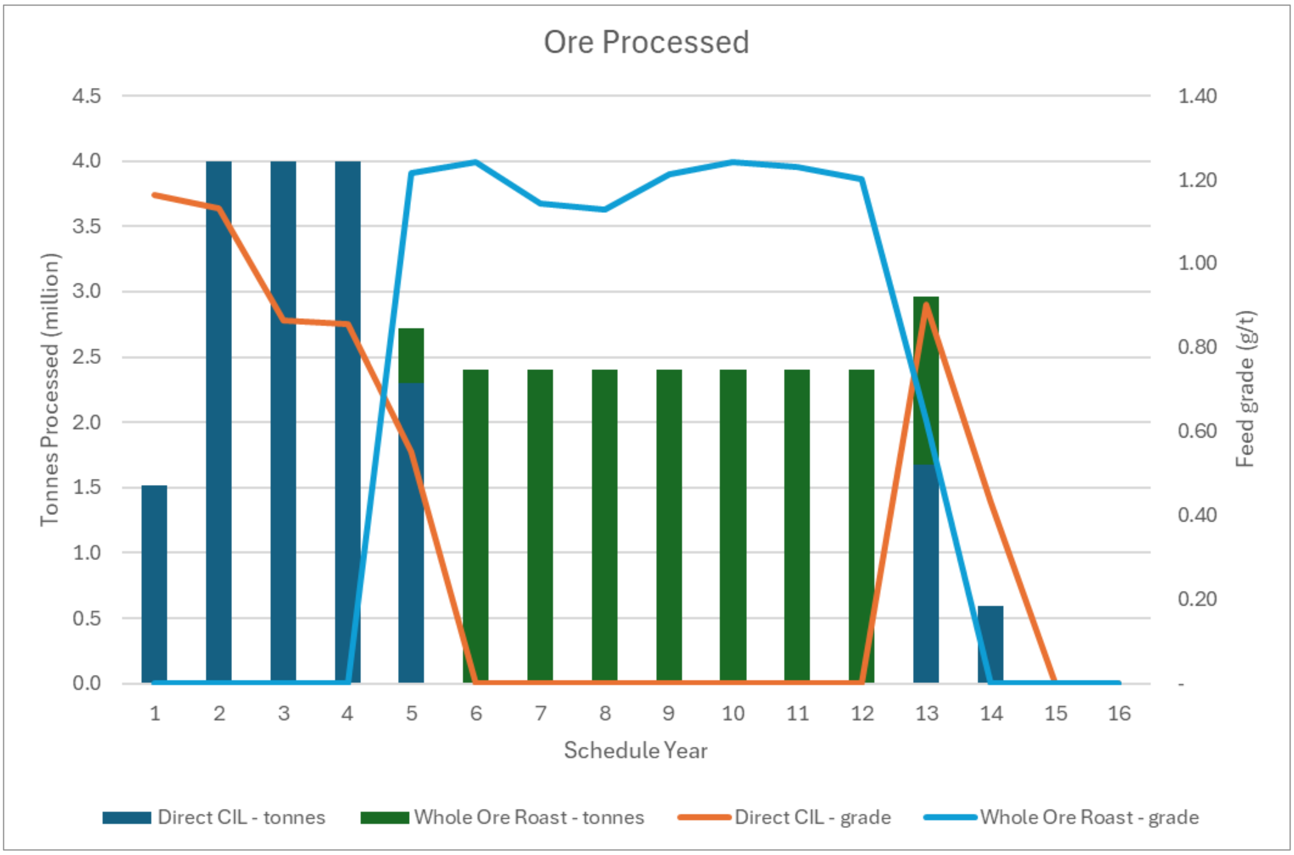
Source: AMC, January 2026.

Production is divided into two distinct Phases. Phase 1 processes oxide and transitional material through a standard CIL leach processing flowsheet while Phase 2 treats primary fresh ore through a roasting component prior to the CIL leach process. This approach allows for the deferment of capital associated with the construction of the roasters by focusing on oxide ore from all pits prior to commencing the mining of fresh ore.

The target throughput for Phase 1 is 4 million tonnes per annum (Mtpa) through the processing plant while the target for Phase 2 is 2.4 Mtpa, being comprised of 1.2 million tonnes (Mt) through each of the two roasting units.

A mining and processing schedule was prepared using the pit designs and processing targets. This schedule yielded a project life of over 13 years and yielded 1.03 million ounces of recovered gold. Phase 1 concludes in Year 5 with the exhaustion of the Phase 1 oxide pits. Phase 2 commences in Year 5 and continues until Year 13. At this point, the roasting units are deactivated and approximately 2.3 Mt of predominantly transitional mill feed, mined from the Phase 2 pits, is treated (Figure 1.5).

Figure 1.5 Annual processing



Source: AMC, January 2026.

The project has been developed on the assumption that a mining contractor will be utilised for all mining activities. Costs and equipment strategies have been developed around this assumption.

1.9 Mineral processing

The orebody is comprised of three broad ore types representing oxide, transition, and sulphidic ores. Gold is distributed throughout the ore types with variable grades but on average higher in the sulphidic ores. The oxide and transition ores are the result of weathering alteration and have been further classified as strongly oxidised (SOX), moderately oxidised (MOX), and weakly oxidised (WOX). SOX ores dominate the oxide ores progressing to lesser levels of oxidation through the transition ore types.

Gold is hosted in the general oxidised gangue lattice, in pyrite and arsenopyrite, and silicates.

SOX and MOX ores are amenable to whole ore cyanide leaching. WOX ores exhibit lower recoveries for cyanide leaching and the sulphidic ores exhibit poor whole ore cyanide leach recoveries. Alternate recovery processes have been investigated to improve gold recovery in the less oxidised and sulphidic ores.

Gold recoveries were developed from test work completed at Independent Metallurgical Operations Pty. Ltd. (IMO) and Mineral Processing and Powder Technology Institute of Northeastern University (NEU). IMO tested samples supplied by Thor from selected drillholes. Samples were selected by ore types classified as oxide, transition, and fresh and by orebody location. Sub samples of two master composites of fresh ore type (MC2, MC3) were delivered to NEU, China for testing of the suspension roasting process.

The recoveries determined from test work on the differing ore types are shown in Table 1.6.

Table 1.6 Recoveries by ore type

Ore type	Orebody	Process	Recovery (%)
Oxide	All	CIL	92.5
Transitional	Makosa (Excluding East)	CIL	82.65
Transitional	Makosa East	CIL	72.8
Fresh	Makosa	Suspension roast - CIL	81.0
Fresh	Makosa Tail	Suspension roast - CIL	88.0

Source: AMC, 2025.

IMO test work optimised CIL process parameters and recoveries for oxide and transition ore types. Oxide ores showed recoveries consistently more than 90% with low variation by orebody. All results were averaged to determine the final recovery. Transition ore types exhibited variation between Makosa and Makosa East locations. Recoveries were averaged for each of the two locations. IMO testing of fresh ore types gave low and variable results by CIL and these were excluded from the recovery data.

Fresh ores from Makosa and Makosa Tail were tested by suspension roasting and CIL extraction of the calcined product by NEU. Results were consistently higher than methods tested by IMO and have been utilised as recoveries for fresh ores in pit optimisations.

The major outcomes of the metallurgical testing programs can be summarised as:

- Oxide ores can be processed by conventional gravity and CIL which also mitigates against preg-robbing for those ores with organic carbon.
- Transition ores can be processed by CIL but further work is necessary to understand the variations observed.
- Fresh ores are not amenable to use of CIL processes and achieved poor recoveries even after flotation concentration and oxidative or high temperature treatment of the concentrates followed by cyanide leaching.
- Fresh ores can be treated by suspension roasting and CIL for gold recovery.

Gold distribution and fineness is variable between and within ore types as demonstrated by recovery responses. Oxidised ores have a higher proportion of coarser gold amenable to gravity recovery. Generally other ore types do not and display poorer gravity responses with the exception of two fresh ore samples which gave 12% and 20% recoveries. Transition ores have a more consistent response of approximately 5% gravity recoveries but show generally high CIL recoveries with one exception indicating that the gold is not locked in silicates or sulphides. The poor leaching recoveries of fresh ores which is improved by suspension roasting indicates that a proportion of the locked gold is both fine and recoverable.

The processing circuit is to be constructed in two phases as follows:

- Phase 1 will treat oxide and transitional ores via a conventional CIL circuit.
- Phase 2 will treat fresh ore via suspension roasting. The roasted product will be reground and treated in a reduced capacity CIL circuit (cf. Phase 1).

Design throughput for Phase 1 is 4 Mtpa of ore to comminution. The proposed process plant design for Phase 1 is based on established gravity / CIL technology, which consists of crushing, milling, gravity recovery of free gold followed by leaching / adsorption of gravity tailings, elution and gold smelting to recover gold from loaded carbon, cyanide destruction in plant tailings and tailings disposal. Services to the process plant will include reagent mixing, storage and distribution, power and water supply, oxygen generation and compressed air services. The plant will operate in the Phase 1 configuration until oxide ore is depleted which is nominally planned to be early Year 5 of the Project.

In Phase 2, the plant will treat 2.4 Mtpa of fresh, sulphide ore. A roasting circuit treating the comminution product stream will be added to the flowsheet. The suspension roasting process will expose refractory gold particles prior to cyanide leaching. The circuit will consist of a pre-roasting dewatering stage, pre-roasting product storage silo, suspension roasting, and calcine repulping and regrinding. The repulped and reground calcine will feed the first leach tank of the Phase 1 CIL circuit.

1.10 Infrastructure

1.10.1 Project overview and location

The Thor Douta Project is located in south-eastern Senegal, approximately 620 km east of Dakar and north of the town of Kédougou. The Makosa site hosts the majority of project infrastructure, while the Baraka 3 mining area is located approximately 35 km to the south and is dedicated to mining operations only. The project is located in a more rural area of Senegal, with limited existing infrastructure, requiring a largely self-sufficient development approach.

1.10.2 Access, logistics, and transport

Access to site is primarily via Senegal’s national road network, utilising the N1 from Dakar to Tembacounda, the N7 to Kédougou, and secondary tarred roads to Bembou, followed by a 20 km dirt road to site. The total road distance from Dakar is approximately 720 km. Roads are generally in good condition and are considered the most reliable logistics option for both personnel and freight.

Senegal’s rail infrastructure is limited and currently unreliable. The Dakar–Bamako metre-gauge railway passes through Tembacounda, but due to rehabilitation uncertainty and lack of freight handling facilities, rail transport is not considered dependable at this stage. Road transport is therefore recommended as the primary logistics route, with rail as a potential future opportunity only.

Air access is provided via Blaise Diagne International Airport near Dakar for international travel and freight, with a smaller domestic airport at Kédougou suitable for passenger movements only. The Port of Dakar has been identified as the preferred and most suitable port facility for sea freight.

1.10.3 Power supply and generation

The mine site is not connected to Senegal’s national grid, which is limited to the northern and western regions of the country, as a result the project is designed to be fully self-powered.

A local 32 megawatt (MW) Heavy Fuel Oil (HFO) / diesel power plant is planned, comprising 16 × 2 MW generators, generating power at 10.5 kilovolts (kV). The plant is designed to operate in parallel and is supported by a dedicated fuel storage facility providing a 20-day operating buffer. Fuel storage includes HFO storage tanks, diesel tanks, buffer tanks, treated HFO tanks, and day tanks.

1.10.4 Tailings Storage Facility (TSF)

The Tailings Storage Facility (TSF) has been designed by Knight Piésold for a 14-year LOM and will store tailings from both Phase 1 oxide (Years 1-5) and Phase 2 fresh ore processing (Years 5-14). The TSF footprint covers approximately 270 hectares (ha) and is located in a gently sloping valley draining to the northeast. Total tailings storage requirements are estimated at ~27.9 million cubic metres (Mm^3).

The TSF will be a fully lined conventional slurry facility, constructed in four stages to defer capital expenditure. The design includes a starter dam followed by three raises, increasing capacity progressively to approximately 38 Mt by Raise 3. Embankment crest elevations increase from 176 metres above sea level (mASL) (starter dam) to 190 mASL (final raise), with upstream and downstream slopes of 1V:3H.

Water balance modelling indicates increasing inflows over the LOM, with peak inflows reaching ~850,000 m^3/month. The TSF pond volume varies seasonally between approximately 0.2 Mm^3 (dry season) and 1.4 Mm^3 (wet season) under the base case. Results are sensitive to entrainment assumptions, and further tailings testing and site-specific climate monitoring are recommended.

1.10.5 Water supply and Water Storage Dam (WSD)

There is no municipal water supply available. A Water Storage Dam (WSD) will be constructed within the mining licence area to supply all raw, process, potable, and service water requirements. The process plant requires approximately 254 cubic metres per hour (m^3/h) during Phase 1 and 221 m^3/h during Phase 2.

The WSD has a footprint of approximately 1.22 km^2 and has been designed in accordance with Canadian Dam Association (CDA) guidelines. The embankment has a maximum height of ~5.5 m, crest width of 13.5 m, and total embankment fill of approximately 30,000 m^3. Water balance results indicate sufficient storage capacity to meet process water demands throughout the LOM, subject to confirmation of ground hydraulic conductivity through further investigation.

Potable water will be supplied via a treatment plant, with demand estimated at ~303 m^3/day to support an on-site camp for construction and for ongoing operations.

1.10.6 Site infrastructure and supporting facilities

The project includes approximately 11-12 km of internal gravel roads within the Makosa area and a 35 km access road to the Baraka pits. Roads are designed as dual-lane gravel roads with engineered fill and crushed stone wearing surfaces.

Supporting infrastructure includes administrative offices, workshops (mechanical and electrical), stores, training facilities, ablutions, wastewater management systems, and explosive magazines located away from operational areas for safety compliance.

An ancillary mobile fleet has been defined to support operations, including loaders, graders, dozers, forklifts, light vehicles, fire trucks, ambulances, and rescue vehicles.

1.10.7 Communications and telecommunications

In-pit communications will be provided via a wireless repeater-based system to ensure continuous coverage for mobile equipment. Site telecommunications are allowed for using Starlink satellite internet, which is expected to become an option in Senegal during 2025-2026, providing a cost-effective and reliable solution for remote operations.

1.10.8 Pit dewatering

Pit dewatering is required to manage both rainfall and groundwater inflows. The strategy combines in-pit sump pumping with ex-pit dewatering boreholes to depressurise the saprolite. Daily pit pumping requirements vary by pit.

Approximately 88 dewatering boreholes are required at an estimated average borehole yield of 3 litres per second (L/s).

1.10.9 Key outcomes

- The project is logistically viable using existing road infrastructure, with Dakar as the primary port and logistics hub.
- Self-sufficient power and water systems are required and have been appropriately sized.
- TSF and WSD designs are technically robust, staged to reduce upfront capital, and compliant with international standards.
- Water and tailings management are key risk areas, with further testing and monitoring recommended to reduce uncertainty.

1.11 Markets and contracts

Thor have an existing gold doré purchasing and refining contract with Metalor Technologies SA. All doré produced at Douta will be sold under the existing contract terms.

All gold sales are in US$.

1.12 Environmental and social summary

Under Senegal's Environmental Code (Law No. 2023-15), the Douta Project is classified as a "classified establishment," requiring a full Environmental & Social Impact Assessment (ESIA) and environmental authorisation. Comprehensive environmental and social baseline studies were completed for the Project between 2021 and 2025, covering dry and wet seasons and progressively expanding to include adjacent exploration licences. These studies supported the preparation of an ESIA, which was submitted in May 2025 and updated in August 2025 in response to comments from the Senegalese interministerial review committee. The ESIA integrates environmental, biological, and socio-economic baseline data and was developed concurrently with exploration drilling, resource modelling, and PFS technical studies, enabling early identification of environmental constraints and opportunities that informed the project layout, mine footprint, and LOM design. Given the timing of the submission of the ESIA (May 2025) the project layout concentrated on the Douta and Birima permit areas of Makosa, Makosa North, Makosa East and Makosa Tail only. The Baraka 3 area was not included in the ESIA. It has latterly been surveyed (Q4 2025) but is still to complete full seasonal environmental baseline surveys.

The Company's environmental and social framework complies with Senegalese mining, labour, and forestry legislation and aligns with International Finance Corporation (IFC) Environmental and Social Performance Standards, as adopted by the parent company, Thor. Provisional approval of the ESIA was granted in August 2025 following ministerial and regional consultations, with final approval received on 16 January 2026.

The ESIA currently covers Phase 1 operational components, consisting of oxide ore processing using a conventional metallurgical flowsheet. Phase 2, which will process fresh (refractory) ore through an additional recovery circuit, will be addressed in a future amendment following completion of metallurgical test work. The approved ESIA clearly specifies a Phase 2 ESIA project component will be submitted to incorporate the inevitable expansion of the project footprint (across the 3 Els) and design improvements regarding additional pits, the location of the water storage dam and the scale of the TSF.

Baseline studies for the ESIA indicate a tropical climate with distinct dry (November-April) and wet (May-October) seasons. Surface and groundwater quality is high, characterised by low mineralisation and neutral to slightly basic pH, with no evidence of mining-related contamination. Air quality is dominated by dust (PM_{10})[1], primarily along unpaved roads, while other pollutants and noise levels remain below regulatory thresholds. Biological surveys identified 227 plant species, including several International Union for Conservation of Nature (IUCN)-listed and nationally protected species, and confirmed the presence of chimpanzees in the southern permit area. However, in linking with an adjoining mine's environment protection area and with appropriate mitigation and biodiversity management measures, mining is considered feasible as outlined in the project ESIA.

The Douta exploration area in the ESIA spans three municipalities with a combined population exceeding 40,000. Stakeholder engagement identified strong community support linked to employment and infrastructure benefits, alongside concerns related to land access. No physical resettlement is anticipated; economic displacement will be managed through a Resettlement Action Plan and community development initiatives focused on local employment, water access, health, education, and support for local enterprises.

An Environmental and Social Management Plan (ESMP) has been developed in the ESIA to manage impacts on land, water, air, biodiversity, and communities, and includes provisions for monitoring, rehabilitation, waste management, and community engagement. Progressive closure and rehabilitation are planned throughout the mine life, with total closure costs (for the Phase 1 ESIA components) estimated at US$4.6 million (M), inclusive of engineering and contingency allowances.

1.13 Capital cost estimates

The initial Phase 1 CapEx cost is estimated at US$253.5M and incurred over an 18-month period. Phase 2 CapEx is estimated at US$60.1M and expected to occur in 2031. Sustaining CapEx is estimated at US$53.0M along with closure costs at US$10.2M giving a LOM total CapEx of US$376.8M. The LOM CapEx is summarised in Table 1.7.

Table 1.7 LOM CapEx summary

Category	Phase 1 CapEx (US$M)	Phase 2 CapEx (US$M)	Sustaining LOM CapEx (US$M)	Closure costs (US$M)	Total LOM CapEx (US$M)
Mining	19.1				19.1
Process plant	177.9	60.1			238.0
TSF and water storage	21.9		53.0		74.9
Other project costs	19.0				19.0
Owners' costs	15.6				15.6
Closure costs				10.2	10.2
Total	**253.5**	**60.1**	**53.0**	**10.2**	**376.8**

Sources: NORINCO, AMC, Knight Piésold, 2025.

Capital cost estimates presented in this section reflect total project costs from July 2026 to end of mine life. Mining activities are to be undertaken by a mining contractor with equipment costs contained within the mining operating costs.

The capital cost estimate was developed in collaboration with Northern International Cooperation Co., Ltd. (NORINCO), the Company's Engineering Procurement & Construction (EPC) turnkey partner at its

[1] PM_{10} refers to Particulate Matter 10, which are inhalable airborne particles with a diameter of 10 micrometres (µm).

Segilola project, using a methodology consistent with the proven approach adopted for that project. The estimate incorporates EPC turnkey components, providing strong cost definition and execution certainty. On a dollar-per-tonne basis, the projected capital intensity is highly competitive and broadly aligned with the benchmarks achieved at Segilola.

A contingency of 10% has been applied and included in all capital items above with the exception of the Processing Plant and TSF capex which have a 5% contingency allowance included.

1.14 Operating cost estimates

The LOM operating cost estimates are summarised in Table 1.8.

Table 1.8 LOM operating costs

Description	Phase 1 only			Phase 1 and 2		
	Total (US$M)	Cost ($/oz)	Cost ($/t ore)	Total (US$M)	Cost ($/oz)	Cost ($/t ore)
Mining	205	498	13.0	667	650	18.5
Processing	259	632	16.5	913	891	25.4
G&A	51	124	3.3	117	114	3.3
Refining	3	7	0.2	9	8	0.2
Cash operating cost	**518**	**1,262**	**33.0**	**1,706**	**1,664**	**47.4**
Royalties	72	175	4.6	180	175	5.0
Total Cash Cost	590	1,437	37.6	1,886	1,839	52.4
Sustaining capital	23	56	1.5	53	52	1.5
AISC	**613**	**1,493**	**39.1**	**1,939**	**1,890**	**53.9**

Source: NORINCO, Knight Piésold, Thor.

1.14.1 Mining costs

Mining operating costs were provided by a mining contractor and used across the LOM schedule as summarised in Table 1.9. The ore mining costs include cost estimates for the owner's technical services team and grade control costs. Surface haulage costs were based on contractor rates for Baraka material based on a 35 km haul distance and dewatering costs estimated pumping requirements for in-pit and ex-pit dewatering using diesel rates of US$1.00/L and electricity rates of US$0.21/kwh.

Table 1.9 Mining OpEx unit costs

Area	OpEx unit cost (US$/t mined)
Ore mining cost - oxide and transitional	4.75
Ore mining cost - fresh	4.90
Waste mining cost - oxide and transitional	2.75
Waste mining cost - fresh	2.90
Surface haulage cost (Baraka)	5.75
Dewatering costs	0.01
Rehandle costs	0.50

Source: NORINCO, AMC.

1.14.2 Processing costs

Processing operating costs were estimated for the different ore types to be treated during the Oxide Ore (Phase 1) and the Primary Ore (Phase 2) of the operation. These are summarised in Table 1.10. Tailings monitoring and management costs were estimated at ~US$880k per year and included in the processing costs, equating to an additional US$0.22/t ore processed.

Table 1.10 Processing OpEx unit costs by ore type and area

Area	Processing unit cost (US$/t ore)		
	Oxide	Transitional	Fresh
Makosa (Main, North, East, and Tails)	16.14	16.14	33.35
Baraka (East and West)	16.14	16.14	17.14

Source: NORINCO, 2025.

1.14.3 Other costs

Total fixed mine level general and administration (G&A) costs are estimated at US$3.25/t ore. Refining and transport costs were based on current contracts at Segilola. Royalties of 5% were applied to the gross revenue of gold produced.

1.15 Economic analysis

An economic evaluation of the Project has been completed using a detailed cash-flow model. The model is based on annual cash flows and incorporates processed tonnages and grades for the CIL feed, metallurgical recoveries, metal prices, operating costs, refining charges, royalties, and both initial and sustaining capital expenditures. Gold revenues are calculated using a payable factor of 99.90%. The analysis applies a base gold price of US$3,500 per ounce.

The Project has been assessed on a "100% equity" basis, with all debt and equity financing considerations excluded. Inflation has not been factored into the assessment. Discounting and Internal Rate of Return (IRR) calculations commence at the start of construction, using a 5% discount rate.

The Company notes that the Mining Convention for the Project is yet to be negotiated with the Government of Senegal. As a result, it is not currently possible to incorporate the expected fiscal incentives and tax exonerations that are typically granted under such agreements into the calculation of the post-tax net present value (NPV). For the purposes of this PFS, we have therefore applied a standard approach whereby exploration expenditure and project development capital are accumulated into a tax loss pool, against which future taxable income is fully offset, with the Senegalese statutory corporate tax rate of 30% applied thereafter. The Company anticipates an improved post-tax economic outcome once the Mining Convention is finalised, as such agreements customarily provide additional tax incentives that enhance project value.

The Company also notes that Senegalese mining legislation provides for a 10% State free-carried interest, which will be formally awarded to the State upon finalisation of the Mining Convention. This interest has not been incorporated into the current economic analysis, which is presented on a "100% equity" basis and will be reflected in future evaluations once the Mining Convention has been agreed.

The project is expected to have a LOM ranging from just under five years for Oxide Ore phase only (Phase 1) and a total of over 13 years when Oxide & Primary ore (Phase 1 and 2) is included. A summary of the mining physicals and economics for each phase is shown in Table 1.11. Phase 1 delivers an NPV[5] of US$449M from investment capital of US$254M. Additional capital for Phase 2 of US$60M is required to process the fresh material, extending the mine life to over 13 years and delivering an NPV[5] of US$908M. The IRR for each phase is 72% and 73% respectively. The payback period for Phase 1 is within 12 months of the start of mining.

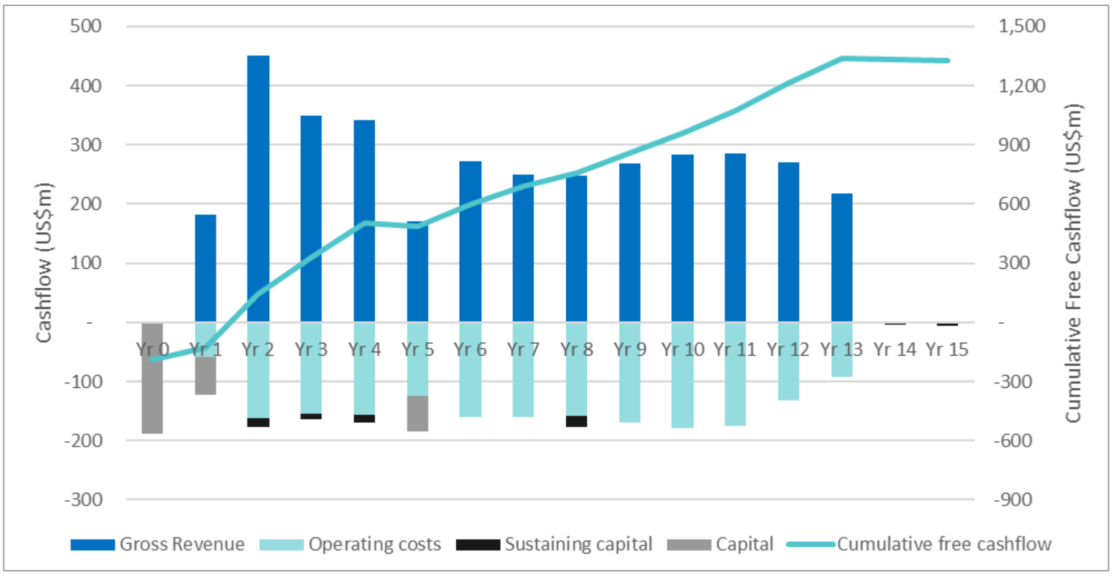
Table 1.11 Mining and processing physicals and financials summary

	Units	Oxide Ore Phase only	Oxide & Primary Ore Phases
Production			
Total ore mined	Mt	15.9	36.6
Total material mined	Mt	62.6	209.3
Strip ratio	x:x	2.9	4.7
Total ore processed	Mt	15.7	36.0
Head grade	g/t	0.9	1.0
Recovery	%	87.9%	84.7%
Gold recovered	koz	411	1,025.7
Production costs			
Mining cost	$/t mined	3.3	3.2
Cash operating cost	$/t ore processed	33.0	47.4
Cash operating cost	$/oz	1,262	1,664
AISC	$/oz	1,493	1,890
Capital costs			
Phase 1 capital	$M	253.5	253.5
Phase 2 capital	$M	-	60.1
Sustaining capital	$M	23.0	53.0
Closure costs	$M	10.0	10.2
Financial			
Gross Revenue	$M	1,437	3,590
Pre-tax NPV5	$M	449	908
Pre-tax IRR	%	72%	73%
Post-tax NPV5	$M	321	633
Post tax IRR	%	60%	61%

Source: AMC, 2025.

Cashflows for Phases 1 and 2 are shown in Figure 1.6. These show early payback within 12 months of the start of mining.

Figure 1.6 Cashflows for Phases 1 and 2



Source: AMC, 2026.

The project value was assessed by undertaking sensitivity analyses on the gold price, operating costs, and capital costs. This shows that the project is most sensitive to gold price and then operating costs. The results of sensitivity analyses related to gold price is presented below in Table 1.12.

Table 1.12 Pre-tax NPV gold price sensitivity

Gold price (US$/oz)	Discount multiple US$M		
	0.00%	5.00%	10.00%
3,000	840	561	379
3,500	1,327	908	634
4,000	1,815	1,255	888
4,500	2,302	1,603	1,143

Source: AMC, 2025.

1.16 Risks and opportunities

During the course of this study, a number of risks were identified:

- Mineralisation and grade variability may exist due to "nuggety" grade distributions at the local scale. This could impact the production grades when compared with the Mineral Resource block model. An infill resource definition drilling campaign and grade control programs are recommended to counter this risk.
- Bulk density calculations have been completed on a relatively small data set. Incorrect bulk density calculations may lead to actual mined tonnes being less than planned. Additional bulk density determinations are required in future drill campaigns.
- There is no visible indication of ore / waste contacts that can be used during grade control and mining. Ore mark outs will be based entirely on grade control assays. A robust and accurate grade control and ore mark out procedure and drill and blast practices will be required.
- The absence of any geotechnical or hydrogeological studies for the Baraka deposits increases the risk of incorrect wall angles being used for the optimisations and pit designs. These studies need to be completed to safely allow the extraction and future expansion of these pits.
- Dewatering costs for the entire project are based on limited hydrogeological information. Additional information is required for all deposits to allow for more accurate design and costing of dewatering programs.
- The pool of skilled labour in the area may be limited, particularly with the neighbouring mine already being in operation. Expatriate operators and trainers will be required, at least initially.
- Metallurgical variability is poorly understood for some regions and ore types. This could produce inconsistent and unexpected recovery fluctuations during processing. Additional metallurgical variability studies are required.

A formal Risk Response Plan will be developed as part of the next stage of work.

The following opportunities were identified during the study:

- With approximately 30 exploration targets within the exploration leases, there is potential for additional mineralisation to be discovered during the life of the operation.
- Inter-ramp and overall slope angles of the pits could potentially be steepened with additional geotechnical study, particularly for oxide and transitional material, and at Baraka 3.
- Improved selective mining unit / bench height studies could assist with dilution control.
- Alternative energy sources could be investigated including waste to liquid plants and solar arrays. A trade-off between capital cost and operating cost savings should be carried out.

1.17 Recommendations

During the course of this study, a number of recommendations have been proposed in various areas of the project. A summary of these recommendations is provided below with more detail in Section 26 and relevant individual sections.

1.17.1 Geotechnical recommendations

- Construct a three-dimensional structural model using fault and shear structures collected from diamond drilling and verified with mapping activities on implementation.
- Conduct additional uniaxial compressive strength (UCS), triaxial strength, and tensile strength tests on rock types where limited data is available.
- Carry out additional direct shear testing in future data collection programs on bedding, joint and shear structures in each rock type.
- Undertake additional geotechnical drilling so as to provide a more comprehensive understanding of the ground conditions and rock mass properties at all locations within the Project area, particularly Baraka 3, where no geotechnical information has been collected to date.
- Improve the understanding of likely slope behaviour through continued limit equilibrium analysis and kinematic analysis for all areas of the proposed pits.

1.17.2 Hydrogeological recommendations

- Determine hydraulic properties through in situ testing which can be done in conjunction with the geotechnical drilling program.
- Create a hydrogeological database to track monitoring and testing data.
- Generate a groundwater model, producing groundwater profiles for the latest pit slope designs.

1.17.3 Mining recommendations

- Consider rerunning optimisations at a higher gold price prior to the commencement of mining operations, pending current market conditions.
- Conduct representative grade control drilling on part of the Year 1 pits to test and reconcile model performance. This will assist in developing a robust grade control programme, including grade control modelling and reconciliation methods, ore demarcation, drill-blast and mining practices.
- Conduct additional sensitivity studies to assess the impact on the Mineral Reserve and open pit limits at various gold prices to ensure potential infrastructure does not restrict future expansion.
- Conduct a trade-off study between contract mining and owner-operator mining.
- Conduct a thorough dilution and bench height study.
- Move scheduling activities into purpose-built mine scheduling software such as Deswik.Sched or MineMax.
- Perform waste rock dump optimisations to optimise construction and rehabilitation opportunities.

1.17.4 Metallurgical recommendations

- Perform characterisation of the variability of carbon, sulphur and silicates across the orebodies.
- Conduct additional metallurgical variability test work to expand the knowledge on recovery performance for both transition and fresh ore types.
- Undertake characterisation of a variability suite of selected transition ore samples for their sulphides, organic carbon and silicate content, and test for recovery by CIL.

- Undertake characterisation of a variability suite of selected fresh ore samples for their sulphides, organic carbon and silicate content, and test for recovery by suspension roasting and CIL.
- Expand the testing sample set to include increased variability of S, As, and C contents in future test programs.
- Conduct additional study of the gold distribution within each ore type and parallel assessment of silicate and sulphide content to refine the understanding of the best process options for each type.

1.17.5 Processing recommendations

Review the processing flowsheet to determine benefits of parallel processing of the Phase 1 and Phase 2 material, such as utilising a second comminution circuit or reviewing dry grinding technologies as direct feed to the roasting plant.

1.17.6 Infrastructure

- Conduct a trade-off study on the location of the processing plant to determine if there is a better location further to the south of the current position.
- Undertake additional ground investigations at both the TSF and WSD locations such as:
  - Drilling and core logging to log the foundation and collect samples.
  - Standard Penetration Tests to confirm the suitability of embankment foundation locations.
  - Trial pitting to identify potential borrow zones and foundation geometry.
  - Geotechnical laboratory testing of construction materials.
- Undertake hydrogeological assessments of both the TSF and the WSD, which would include:
  - Installation of boreholes upstream and downstream of the TSF (baseline water quality and for monitoring water quality).
  - Permeability testing within boreholes; map groundwater levels and gradients seasonally.
  - Development of a seepage model.
- Undertake tailings rheological testing to inform stability analysis and dam breach assessments.
- Conduct a probabilistic seismic hazard assessment to inform stability analysis.

1.17.7 Environmental and social recommendations

- Update the Environment and Social Impact Assessment (ESIA) for the Phase 2 refractory gold process and expansion of the mine and project footprint across the 3 EL.
- Develop a biodiversity management plan, particularly for chimpanzees.
- Conduct additional test work to define Acid Rock Drainage (ARD) potential from waste rock and increase the test dataset from the oxide and transitional ores.
- Develop an arsenic control strategy for the roasters including a locally used laterite detoxification process.
- Develop and implement several specific management plans required under the umbrella of the ESMP.
- Establish community agreements for socio-economic benefits to local communities.
- Compile the Economic Displacement Management Plan, including a robust compensation procedure.

1.17.8 Capital and operating cost recommendations

- A number of capital estimates will require additional investigation in order to progress the Project to the next stage. Key items include:
  - Haul road construction.
  - Communications.
  - First fills.
  - Power supply, which can be adjusted now that power demand for the site is better understood.
- Several operating costs should also be progressed to a higher confidence level. These include:
  - Mining costs: Pending changes to the mining schedule based on the recommendations above, detailed quotes should be obtained from mining contractors on a bench by bench and pit by pit basis.
  - Processing costs and recovery rates: Following test work outlined in the above recommendations, the recovery formula may need to be adjusted and then re-test reagent consumption rates that flow through to the processing costs.
  - G&A costs: Currently this is based on a benchmark number so will require additional work to align it with Project specific information.
  - Treatment Charges and Refining Costs (TCRC): A specific quote for the Douta Project should be obtained, rather than relying on current costs at the Segilola mine.

Contents

Tables

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2 Introduction

Following the Request for Proposal (RFP) received from Thor Explorations Ltd. (Thor or Company), AMC Consultants (UK) Limited (AMC) is pleased to submit this Technical Report (Thor Douta Gold Project Technical Report, Technical Report, or Report) for the Douta Gold Project (Douta, Property, or Project) located in southeast, Senegal in the administrative region of Kédougou and prefecture of Saraya.

This Technical Report provides the results of a pre-feasibility study (PFS) for the Douta Project. The Technical Report has been produced in accordance with the Standards of Disclosure for Mineral Projects, as contained in National Instrument 43-101 (NI 43-101). Mineral Resources and Mineral Reserves are classified in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards – for Mineral Resources and Mineral Reserves May 2014 (the CIM Definition Standards).

The Douta Project consists of the Makosa, Makosa Tail, and Baraka 3 Mineral Resources which are situated approximately 800 kilometres (km) east-southeast of the capital Dakar. Thor holds a mining permit (Demande 11618) and two exploration permits (EL03709 and EL02254).

2.1 Issuer

Thor is listed on the Toronto Stock Exchange (TSX-V:THX) and on the AIM Market of the London Stock Exchange (AIM:THX). Douta is 100% owned by African Star Resources (ASR) a wholly owned subsidiary of Thor. Thor is a West African focused gold producer, currently operating the Segilola gold project in Nigeria and has exploration projects in Côte d'Ivoire as well as Senegal.

2.2 Qualified Persons' independence

The Qualified Persons (QPs) from AMC, Mr Dominic Claridge and Mr Robert Chesher, confirm that they are independent of Thor and the Douta Project. The QPs are Fellows of the Australasian Institute of Mining and Metallurgy (AusIMM) and are bound by the codes of ethics of the AusIMM whose charter includes the upholding of standards and developing and promoting professional best practice in the mining industry.

Mr Alfred Gillman is the General Manager (Exploration and Resources) of Thor and not classed as independent.

For the purpose of preparing the Technical Report, the QPs reviewed and analysed the data received, and drew their own conclusions, augmented by direct field observations and knowledge of the Project, and of the results of detailed communication with key Thor personnel. Specific documents referenced in this report are listed in Section 27.

The Technical Report is dated 26 January 2026 and has an effective date of 24 January 2026. Those conclusions may change in the future with changes in relevant metal prices, exploration, and other technical developments regarding the mineral assets and the market for mineral properties.

2.3 Qualifications of QPs

All of the persons noted as QPs meet the requirements of NI 43-101 for QPs. The qualifications and experience of the QPs and contributing specialists for this Technical Report and their area of contribution are listed in Table 2.1.

Table 2.1 Persons who prepared this Technical Report

Qualified Person	Position	Employer	Independent of Thor	Date of site visit	Professional designation	Sections of report
Qualified Persons responsible for the preparation and signing of this Technical Report						
Dominic Claridge	Principal Mining Engineer	AMC Consultants (UK) Limited	Yes	6-9 Jan 2025	FAusIMM	2, 3, 15, 16, 18-22, 24, parts of 1, 25-27
Robert Chesher	Senior Principal Consultant	AMC Consultants Pty Ltd	Yes	N/A	FAusIMM(CP MET), RPEQ	13, 17, parts of 1, 25-27
Alfred Gillman	General Manager (Exploration and Resources)	Thor Explorations Ltd.	No	Regular	FAusIMM(CP)	4-12, 14, 23
Other experts who assisted the Qualified Persons						
Expert	**Position**	**Employer**	**Independent of Thor**	**Visited site**	**Professional designation**	**Sections of report**
Louise Portous	Environmental and Social Manager	Thor Explorations Ltd.	No	Regular	MRTPI	20, Parts of 1,
Thomas Swanton	Geotechnical Engineer	Knight Piésold	Yes	6-9 Jan 2025	Registered with IOM3	Parts of 18, 21
Johnny Mercer-Royce	Principal Consultant	AMC Consultants (UK) Limited	Yes	N/A		Parts of 21, 22
Alex Gallagher	Director	Hydrotechnica Ltd	Yes	N/A		Parts of 16, 18
Zhai Qingxiang	Project Director	Northern International Cooperation Co., Ltd.	Yes	N/A		Parts of 13, 17, 21

2.4 Terms of reference

2.4.1 Purpose of the Technical Report

This Technical Report summarising the Douta PFS is to support Thor's development strategy and Senegalese Mining Convention permit application for the Project.

Thor owns mineral assets located in the south-eastern region of Senegal that are operated by its 100% owned subsidiary, ASR. The mineral assets consist of the Douta open pit gold project containing both oxide and sulphide resources.

- Phase 1 of the project will mine and treat the oxide resources at a capacity of 4.0 Mtpa. Gold recovery will consist of a carbon-in-leach (CIL) circuit. Oxide ore processing (Oxide Project) has an anticipated life of four years.
- Phase 2 will mine and process the sulphide ore underlying the oxidised cap. The sulphide ore will be roasted to expose the refractory gold particles prior to entering the cyanide leaching (CIL) circuit. The roaster will have a capacity of 2.4 Mtpa.

Mineral Resource and Mineral Reserve estimates classified and reported in accordance with the CIM Definition Standards, have been prepared for the Phase 1 Oxide Project and the Phase 2 Sulphide Project. NI 43-101 utilises the definitions and categories of Mineral Resources and Mineral Reserves as set out in the CIM Definition Standards.

Table 2.2 Summary of Douta properties

Project	Prospect	Number	Type	Area (km^2)	Permit holder	Interest (%)	Royalty (%)
Douta	Makosa	11618	Demande	58.15	Societe International Mining Company SARL	100	1.25
Douta	Douta West	3709	Exploration	106	Birima Gold Resources Consulting	70	0
Douta	Bousankhoba	2254	Exploration	374	Global Mining Resources	65	0

Source: Thor, 2025.

2.5 Site inspections

The QPs for Mineral Resources and Mineral Reserves have visited the Douta site for the purposes of preparing this Technical Report.

The QP for Mineral Resources, Alfred Gillman, visited Douta on the following dates: 29 July 2022 – 1 August 2022, 11-14 April 2025, and 5-6 June 2025. In addition to inspecting the Project site and the laboratory and reviewing a suite of representative drill core, the visits facilitated regular interactions with site professionals.

The QP for Metallurgy, Robert Chesher, has not visited Douta, as the ore processing plants that are proposed for the Oxide and Sulphide Phases, are still in the design phase.

The QP for Mineral Reserves, Dominic Claridge, visited Douta for three days from 6-9 January 2025. He undertook an inspection of the proposed open pit and ore processing locations to evaluate site conditions.

2.6 Sources of information

The information in this Technical Report was derived from numerous sources, principally studies conducted by and on behalf of Thor. The following consultants contributed to the completion of this Report:

- Knight Piésold — Tailing Storage and Water Storage Facilities
- Hydrotechnica — Hydrological Studies related to pit dewatering
- NORINCO — Processing Flowsheet and associated CapEx, OpEx

2.7 Conventions and abbreviations

Costs are expressed in United States dollars (US$ or $), unless otherwise specified. All references to ounces (oz) of gold (Au) are troy ounces (31.1035 grams (g)). Unless otherwise stated, quantities are in metric (SI) units and tonnes are dry tonnes.

Commonly used units, abbreviations, and terms are shown in Table 2.3 and Table 2.4.

Table 2.3 List of units

Unit	Description
%	percent
% (w/w)	weight percent
$, US$	United States Dollar
/	per
°	degrees
°C	degrees Celsius
C$	Canadian Dollar
cm	centimetres
g	grams
g/L	grams per litre
g/rev	Grams per revolution
g/t	grams per tonne
GWh	gigawatt hour
h	hour
ha	hectares
Hw	Hydraulic head
Hz	hertz
in	inches
kg	kilogram
kg/t	kilograms per tonne
km	kilometre
km/h	kilometres per hour
km^2	square kilometres
koz	thousand ounces
kPa	kilopascal
kt	thousand tonnes
kV	kilovolt
kWh	kilowatt hour
kWh/t	kilowatt hours per tonne
L	litre
L/min	litres per minute
L/s	litres per second
LOI	Loss on Ignition
M	million
m	metre
m/s	metres per second
m^2	square metre
m^3	cubic metre
m^3/h	cubic metre per hour
Ma	million years
mASL	metres above sea level
mg/L	milligrams per litre

Unit	Description
min	minute
mL	millilitres
mm	millimetres
Mm^3	million cubic metres
Mt	million tonnes
Mtpa	million tonnes per annum
MW	megawatts
MWh/a	Megawatt/hour per annum
oz	ounce
ppb	parts per billion
ppm	parts per million
RPM	revolutions per minute
rw	effective well / pit radius
t	tonnes (metric)
t/m^3	tonnes per cubic metre
tph	tonnes per hour
$\mu g/m^3$	micrograms per cubic metre
µm	micrometre
US$	United States Dollar
V	volt

Table 2.4 List of abbreviations and terms

Abbreviation & term	Definition
>	greater than
<	less than
3D	three-dimensional
Ag	Silver
Ai	Abrasion Index
AIM	London Alternative Investment Market
AISC	all-in sustaining cost
ALS	ALS Laboratories
AMC	AMC Consultants (UK) Limited
ANFO	ammonium nitrate fuel oil
ANSD	National Agency for Statistics and Demography
ARD	Acid Rock Drainage
As	arsenic
As_2O_3	arsenic trioxide
ASR	African Star Resources
Au	gold
AusIMM	Australian Institute of Mining and Metallurgy
Bassari	Bassari Resources Limited
Bi	Bismuth
BBWi	Bond Ball Mill Index
BWi	Ball Work Index

Abbreviation & term	Definition
C	carbon
CapEx	capital expenditure
CaO	lime
Cd	Bismuth
CDA	Canadian Dam Association
CFA	West African CFA Franc
CIL	carbon-in-leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	Carbon in Pulp
Co	Cobalt
COS	crushed ore stockpile
CP	Certified Professional
CP MET	Certified Professional - Metallurgy
CRM	Certified Reference Material also referred to as a Standard
CSA	Canadian Securities Administrators
CSR	Corporate Social Responsibility
Cu	copper
CuSO	Copper Sulphate
CV	coefficient of variation
CWi	Crushing Work Index
DD	Diamond Drill
DEFCCS	Directorate of Water, Forestry, Hunting and Soil Conservation (*Direction Des Eaux Et Forêts, Chasses Et Conservation Des Sols*)
DFS	Definitive Feasibility Study
DGPRE	Department of Water Resources Management and Planning (*Direction de la Gestion et de la Planification des Ressources en Eau*)
DH	Drillhole
DMG	Direction des Mines et la Geologilies
DO	Dissolved oxygen
Douta, Property, Project	Douta Gold Project
Douta West	Douta West Prospect
DWi	Drop Weight Index
E	east
EBITDA	Earnings Before Interest, Tax, Depreciation and Amortisation
E&S	Environmental and Social
E&S PS	Environment and Social Performance Standards
E, C & I	Electrical, Control and Instrumentation
EGL	Effective Grinding Length
EL	Exploration Licence
EN	critically endangered species
EPC	Engineering, Procurement, and Construction
ESIA	Environment and Social Impact Assessment
ESMP	Environmental and Social Management Plan
EW	electrowinning
FAusIMM	Fellow of the Australasian Institute of Mining and Metallurgy

Abbreviation & term	Definition
Fe	iron
FEL	front-end loader
FRS	Fresh (sulphic) ore
G&A	general and administration
GHG	greenhouse gases
GPS	global positioning system
H_2O	water
H_2O_2	Hydrogen peroxide
HCl	hydrochloric acid
HCN	hydrogen cyanide
HDPE	High-density polyethylene
HFO	heavy fuel oil
HIC	High intensity conditioning
HP	High-Pressure
HSSE	Health, Safety, Security and Environment
IBC	intermediate bulk container
ICPE	Installations Classées pour la Protection de l'Environnement (Classified Installations for Environmental Protection)
ICR	Intensive cyanidation reactor
ID	identification
ID^2	inversed distance squared
IDC	Investment Drilling Company
IFC	International Finance Corporation
IMC	International Mining Company
IMO	Independent Metallurgical Operations Pty. Ltd.
IOM^3	Institute of Materials, Minerals and Mining
IP	fully protected species
IRA	Inter-ramp slope angle
IRR	internal rate of return
ITCZ	Inter Tropical Convergence Zone
IUCN	International Union for Conservation of Nature
JC	Jinchan
KKI	Kedougou-Kenieba Inlier
LBMA	London Bullion Market Association
LNG	Liquefied natural gas
LOM	life-of-mine
LP	Low-Pressure
LV	Light Vehicle
MC1	Makosa Oxide
MC2	Makosa Tail Fresh
MC3	Makosa Fresh
Metalor	Metalor Technologies SA
MLA	Mineral Liberation Analysis
MOX	moderately oxidised
MRTPI	Member of the Royal Town Planning Institute

Abbreviation & term	Definition
MSW	Mandakhole Southwest
MTSZ	Main Transcurrent Shear Zone
N	north
N_2	nitroge
NaCN	Sodium cyanide
NE	north-east
NEU	Northeastern University
NI 43-101	National Instrument 43-101 (Standards of Disclosure for Mineral Projects), Form 43-101F1 and the Companion Policy Document 43-101CP
NKNP	Niokolo-Koba National Park
NNE	north-northeast
No.	number
NOG	Non-opaque gangue
NORINCO	Northern International Cooperation Co., Ltd.
NO_x	nitrogen oxide
NPV	net present value
NS	north-south
NW	north-west
OK	ordinary kriging
OpEx	Operating cost
OSA	Overall slope angles
Pb	lead
PAX	potassium amyl xanthate
PDC	Plan de Developement Communal
PFS	Preliminary Feasibility Study
pH	pH is a measure of hydrogen ion concentration; a measure of the acidity or alkalinity of a solution
Phase 1	Phase1 – Oxide Ore
Phase 2	Phase 2 – Primary Ore
POAS	Plans d'Occupation et d'Affection des Sols (Land Use Allocations Plan)
POI	Plan d'Operation Interne (Internal Emergency Response Plan)
PP	partially protected species
PPE	personal protective equipment
PRI	preg-robbing index
Project	Douta Gold Project
Property	Douta Gold Project
PSA	pressure swing adsorption
PVC	polyvinyl chloride
QAQC	Quality Assurance and Quality Control
QC	Quality control
QEMSCAN	Quantitative Evaluation of Minerals by Scanning Electron Microscopy
QP	Qualified Person
RAB	rotary air blast
RBF	radial bias function
RC	reverse circulation

Abbreviation & term	Definition
RF	Revenue factor
RFP	Request for Proposal
RMR	Rock Mass Rating
RO	reverse osmosis
ROM	run-of-mine
RPEQ	Registered Professional of Queensland
RQD	Rock Quality Designation
S	sulphur
SAG	Semi-Autogenous Grinding
Sb	Antimony
SCR	Selective catalytic reduction
SCSE	SAG Circuit Specific Energy
SD	Standard deviation
SEDAR+	System for Electronic Document Analysis and Retrieval
SEM	Scanning Electron Microscope
SG	specific gravity
SGL	Segilola Gold Limited
SI	International System of Units
SMBS	sodium metabisulphite
SMC	Semi-Autogenous Grinding Mill Comminution
SMU	Selective Mining Unit
SO_2	sulphur dioxide
SOX	strongly oxidised
SPA	Sale and purchase agreement
SR	suspension roasting
STE	State Technical Services
SPT	Standard Penetration Test
SW	south-west
TCRC	Treatment Charges and Refining Costs
TDS	Total Dissolved Solids
Technical Report, Report	Thor Douta Gold Project Technical Report
Thor, Company	Thor Explorations Ltd.
TSF	Tailings Storage Facility
TSS	Total Suspended Solids
TSX-V	Canadian TSX Venture Exchange
UCS	uniaxial compressive strength
UFG	ultrafine grinding
UK	United Kingdom
UNESCO	United Nations Educational, Scientific and Cultural Organization
UTM	Universal Trans Mercator
UV	ultra-violet
VA	village assembly
VAT	value added tax
VFD	Variable Frequency Drive

Abbreviation & term	Definition
VU	vulnerable species
W	west
WAD	weak acid dissociable
WOX	weakly oxidised
WSD	Water Storage Dam
WTP	Water Treatment Plant
XRD	X-ray diffraction
ZIC	Zone of Hunting Interest (*Zones d'Intérêt Cynégétique*)
Zn	zinc

3 Reliance on other experts

The QP has relied, in respect of legal aspects, upon the work of the Expert listed below. To the extent permitted under NI 43-101, the QP disclaims responsibility for the relevant section of the Technical Report.

- Expert: Thor.
- Report, opinion, or statement relied upon: Information on mineral tenure and status, title issues, royalty obligations, etc.
- Extent of reliance: Full reliance following a review by the QP.
- Portion of Technical Report to which disclaimer applies: Section 4.2 and Section 4.3.

The QP has relied, in respect of legal aspects upon the work of the Expert listed below. To the extent permitted under NI 43-101, the QP disclaims responsibility for the relevant section of this Technical Report:

- Expert: Thor.
- Report, opinion, or statement relied upon: Information on gold price, taxation, sales contracts.
- Extent of reliance: Full reliance following review by QP.
- Portion of Technical Report to which the disclaimer applies: Section 19 and Section 22.

The QP has relied, in respect of legal aspects upon the work of the Expert listed below. To the extent permitted under NI 43-101, the QP disclaims responsibility for the relevant section of this Technical Report:

- Expert: Thor.
- Report, opinion, or statement relied upon: Information on Environmental, Permitting, and Social Impact.
- Extent of reliance: Full reliance following review by QP.
- Portion of Technical Report to which the disclaimer applies: Section 20.

4 Property description and location

4.1 Property location

The Douta Project is located in eastern Senegal, approximately 620 km south-east of the capital city of Dakar and 80 km north of the town of Kedougou. The Douta tenure and associated land holdings are shown in Figure 4.1.

Figure 4.1 Project location map

Source: Thor, October 2025.

4.2 Land tenure

Under Article 3 of the Senegal Mining Code, all Mineral Resources belong to the State. Companies with mining titles may own extracted minerals, provided they comply with their licence terms and the law.

Excluding artisanal mining regulations, mineral exploration and development require two primary permits. The exploration permit (*permis de recherche*) authorises exploration for up to four years, with two possible three-year renewals. The exploitation permit (*permis d'exploitation*), available in two types according to project scale, authorises mining for up to five years on smaller projects and five to twenty years on larger ones. Small-scale permits can be renewed every three years without limit; large-scale permits can be renewed as needed until the resource is depleted.

To facilitate a smooth transition from exploration to mining, the Mining Code guarantees the holder of an exploration permit a priority right to obtain an exploitation permit over the area covered by their exploration activities, provided that all legal and technical conditions are met. During the transition from an exploration permit to an exploitation permit (mining lease), the permit is assigned an application (Demande) number.

To achieve either the granting of permits or renewals, a mining convention or agreement between the investor and the State forms the primary contract that details the legal, fiscal, administrative, and corporate obligations for the permitted operations.

Mining rights are transferable, subject to prior approval from the Ministry of Mines. The Mining Code outlines the procedure, the information required from both parties, and all the documents necessary for the transaction's purpose to facilitate the transfer.

Table 4.1 summarises the Company's tenure in Senegal by type, with outlines of each permit provided in Figure 4.2.

Table 4.1 Summary of tenure

Prospect	EL Number	Permit holder	Type	Interest (%)	Royalty (%)	Area (km^2)	Effective date	Expiration date	Comment
Douta West	03709	Birima Gold Resources Consulting	Exploration	100	1.25	106.02	20 Feb 2023	19 Feb 2027	
Bousankhoba	03720	GMRS (Global Mining & Resources SA)	Exploration	65		374	2 Feb 2023	1 Feb 2027	
Makosa	11618	IMC (International Mining Company SARL)	Demande	100	1.25	58.14	7 Jun 2024		Registered date as Mining Application)

Source: Thor, 2025.

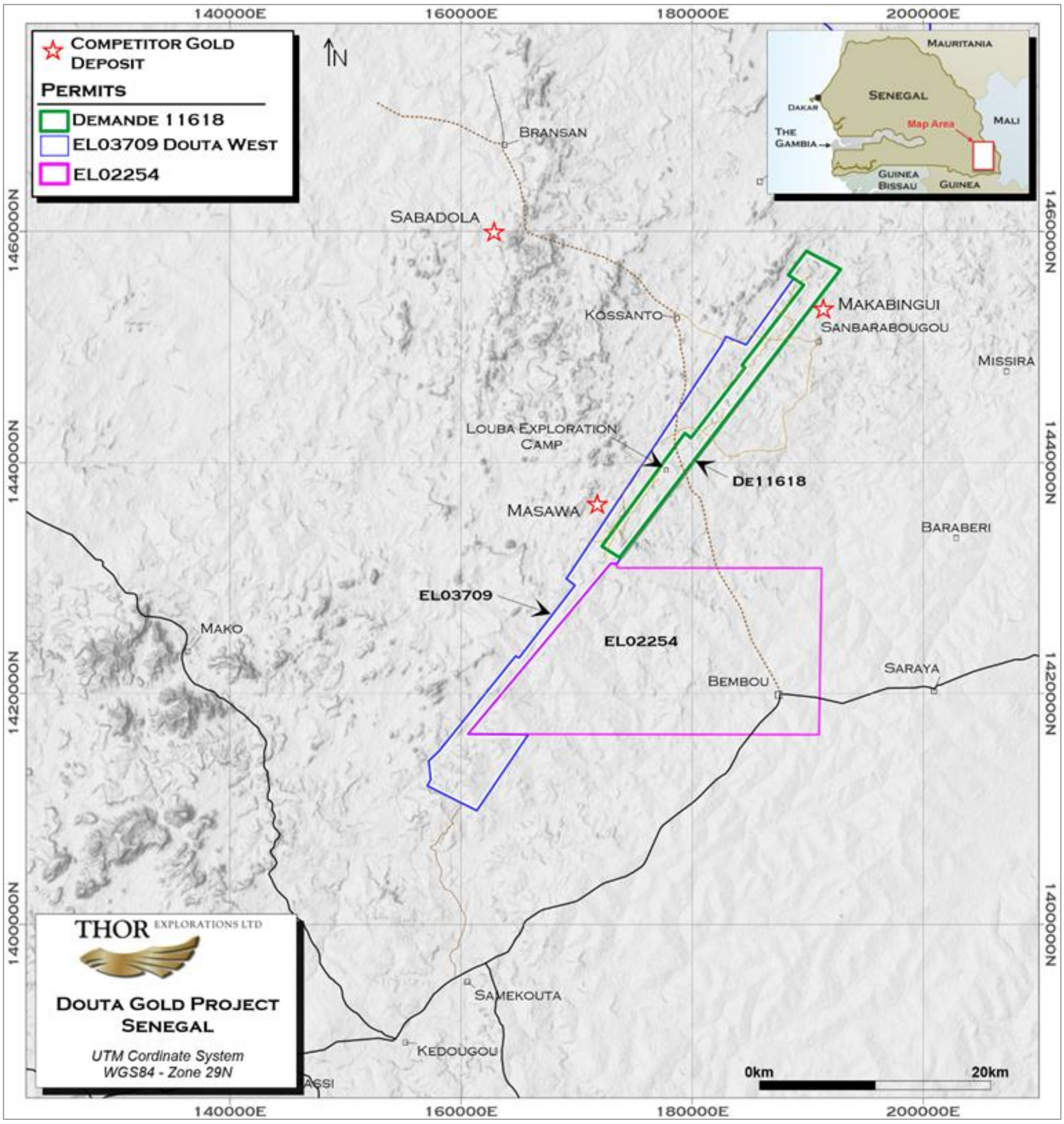
The Douta West and Bousankhoba Permits were issued on 20 February 2023, and 2 February 2023, respectively. Both permits are currently in their initial four-year validity period, which will conclude on 19 February 2027, for Douta West, and 1 February 2027, for Bousankhoba.

According to the Mining Law of 2016, specifically Articles 17 and 18, these exploration permits can be renewed twice, each time through a decree by the Minister responsible for mines. The renewal periods cannot exceed three years and are contingent upon compliance with the obligations outlined in the current mining code.

Furthermore, after the second renewal, Article 19 of the Mining Code provides that the holder of an exploration permit may request a retention period of up to two years, provided that a non-commercial deposit is both proven and recognised by the state. If the exploration permit is not exploited by the end of this retention period, the holder will forfeit all rights associated with it.

In summary, the holders of the Bousankhoba and Douta West Permits have a total of nine additional years available for exploration activities.

Figure 4.2 Douta Project permit location map



Source: Thor, October 2025.

4.2.1 Douta mining application

The Douta Mining Application, formerly exploration licence EL02038, centred at 13°2'54"N latitude, 11°55'39"W longitude, was originally granted to International Mining Company SARL (IMC) in 2009 pursuant to a Mining Agreement executed in 2009. The permit was referred to as the Douta Project.

In May 2011, IMC entered into an option agreement with ASR. The option period ran until November 2011 and was then extended for three months until February 2012. In November 2015 the permit was renewed for another three years, and a subsequent two-year extension in November 2018, running until November 2020.

An exploration permit is issued by order of the Minister of Mines for an initial period of four years and may be renewed twice, each for a further four-year period, by order of the Minister. With each renewal, the area covered by the permit is reduced by one-quarter. As part of this process, in 2015, the licence underwent a compulsory reduction to its current area of 58 km^2.

An application for an exploitation permit was submitted by IMC in September 2020 and was acknowledged by the Ministry of Mines the same month. The exploration licence, which expired in November 2020 after the exploitation permit application was submitted, benefits from automatic prorogation until the Ministry of Mines decides on the application for an exploitation permit.

The Company is party to an option agreement with IMC, pursuant to which, with effect from 24 February 2012, the Company exercised its option to acquire a 70% interest in the Douta Gold Project to be held through African Star Resources (ASR).

As consideration for the exercise of the option, the Company issued to IMC 11,646,663 Common Shares, based on a volume weighted average trading price for the 20 trading days preceding the option exercise date of C$0.2014 (or US$0.2018) per share, valued at C$2,678,732 based on the Company's closing share price on 24 February 2012. The share payment includes consideration paid to IMC for extending the period for exercising the option.

During September 2025, the Company signed a binding sale and purchase agreement (SPA) with IMC to acquire the remaining 30% minority equity interest in the Douta Project.

The acquisition is subject to the completion of certain conditions precedent, including final approval of the Minister of Mines.

The dimensions of the mining permit application (Demande) are approximately 31 km north-south by between 1.35 and 2 km east-west.

4.2.2 Douta regional tenure

In addition to Demande 11618, there are two contiguous exploration permits, EL03709 and EL02254, known as the Douta West Prospect (Douta West) and the Bousankhoba Prospect, respectively.

In February 2024, the Company acquired an initial 70% interest in the Douta-West Licence in a joint venture agreement with Birima Gold Resources. Douta West adjoins the western boundary of Demande 11618 and extends over a north-south distance of 57 km. In January 2026 Thor purchased the remaining 30% of Douta West bringing Douta West to 100% ownership.

In September 2025, the Company acquired an initial 65% interest in the Bousankhoba Exploration Permit EL02254, which is contiguous with the eastern boundary of the Douta West Prospect. The terms of the earn-in agreement include a minimum exploration program spanning 24 months and an earn-in payment of US$160,000, payable within the first six months of signing.

Thor Mineral Resources are confined to the 100% owned Douta and Douta West licence areas.

4.3 Environmental liabilities

The Douta Exploration Licence is not in a national park or a nationally designated environmentally sensitive area. The Niemenike Conservation area, which is of national significance, lies about 20 km from the project's south-western boundary. The Douta Exploration Licence covers a largely modified environment resulting from human activities, including harvesting forest plants and burning vegetation. These are the result of sporadic and unregulated historic artisanal mining. Some areas still have primary and secondary vegetation. Most streams within the exploration licence area are ephemeral.

ASR must follow the Senegal 2016 Mining Code. The Code requires annual contributions by mining title holders to a local development fund equal to 0.5% of sales, minus annual fees. The fund supports economic and social development in mining communities, including initiatives that promote women's empowerment. Mining projects also require a prior environmental impact assessment, which must be approved by the Directorate of the Environment and Classified Establishments.

To gain initial environmental baseline information within the Douta Exploration Licence, and to obtain data for inclusion in a project Environment and Social Impact Assessment (ESIA) a dry season ecology survey was undertaken in May 2021 by Synergie, a registered environmental consultancy in Senegal. In 2022, the wet season ecology survey was completed when the ephemeral streams were flowing. Additional biodiversity surveys were undertaken between 2023 and 2025 to include the full EL land areas including the Birima EL. Studies included specialist primate surveys. Further baseline seasonal surface and groundwater hydrological surveys were undertaken between 2023 and 2025 as outlined in Table 4.2. Gathering of specific environmental and social baseline information culminated in an ESIA being submitted to the Ministry of Environment in March 2025. Follow a series of public meetings (governmental and community) an updated ESIA was submitted in November 2025 with approval from the Ministry of Environment received on 19 January 2026. The QP of this section is not aware of any environmental liabilities on the property. The Company has all the required permits to conduct the proposed work on the property. The QP is not aware of any other significant factors or risks that may affect access, title, or the right or ability to perform the proposed work program on the property.

Table 4.2 Summary of ESIA studies

Survey	Area	Included in ESIA	Consultant	Date
Dry Season Aquatic and Terrestrial Ecology Surveys	Douta	Yes	Synergie	May 2021
Surface & Groundwater Baseline Surveys	Douta	Yes	AGTS	Nov 2022
Aquatic & Terrestrial Ecology Wet Season Survey, including Chimpanzee Survey	Douta	Yes	Synergie	May 2023
Air Quality & Noise Baseline Surveys	Douta	Yes	Synergie	Nov 2023
Biodiversity, Surface and Ground Water Surveys, Air Quality Surveys, Chimpanzee Survey	Douta, Baraka 3	Yes	Synergie	Apr 2025
Environmental Impact Assessment Phase 1	Douta,	Yes	Synergie	Mar 2025
Environmental Impact Assessment Phase 1 Updated	Douta	Yes 2025	Synergie	Jul and Nov 2025

Source: Thor, January 2026.

5 Accessibility, climate, local resources, infrastructure, and physiography

5.1 Accessibility

The Douta Gold Project is located approximately 620 km east of the Senegal capital, Dakar. Dakar is serviced daily by commercial flights from main cities of the world. Charter flights are available from Dakar to Kedougou, the regional center, located approximately 80 km from Douta.

The proposed main transport route for capital equipment would be from the port of Dakar, a journey of approximately 12 hours. The tarred road from Dakar to Kedougou and Moussala-Bamako is located south and east of the permit. The unsealed road from Bembou to Kossanto provides access to the site from both the north and south.

5.2 Climate

The climate at Douta is strongly influenced by the north and southward movement of the Inter Tropical Convergence Zone (ITCZ) which creates distinctive wet and dry seasons between June to September and October to May respectively. Despite some minor inconvenience during the wet season, the operating season is year-round.

The mean annual rainfall ranges between 1,084 millimetres (mm) and 1,184 mm per year, of which 90% falls in the four months of June to September.

The site is in the Sahelian Transition Zone between the Sahara Desert in the north and the tropical climate in the south. Temperatures range between approximately 13°C and 43°C (average 28°C) with the hottest months between March and June.

5.3 Local resources

Immediately to the north of the centrally located Louba exploration camp are the villages of Mandokholing and Lafia. These are both predominantly pastoral communities. About 100 people reside in Mandokholing.

The artisanal mining village of Khossanto, located 7.5 km north-west of the northern extent of the permit, comprises approximately 2,000 inhabitants, and is regarded as the informal capital of the rural community of an estimated 4,500 people.

5.4 Infrastructure

Local infrastructure is limited to small rural settlements connected by gravel roads and paths.

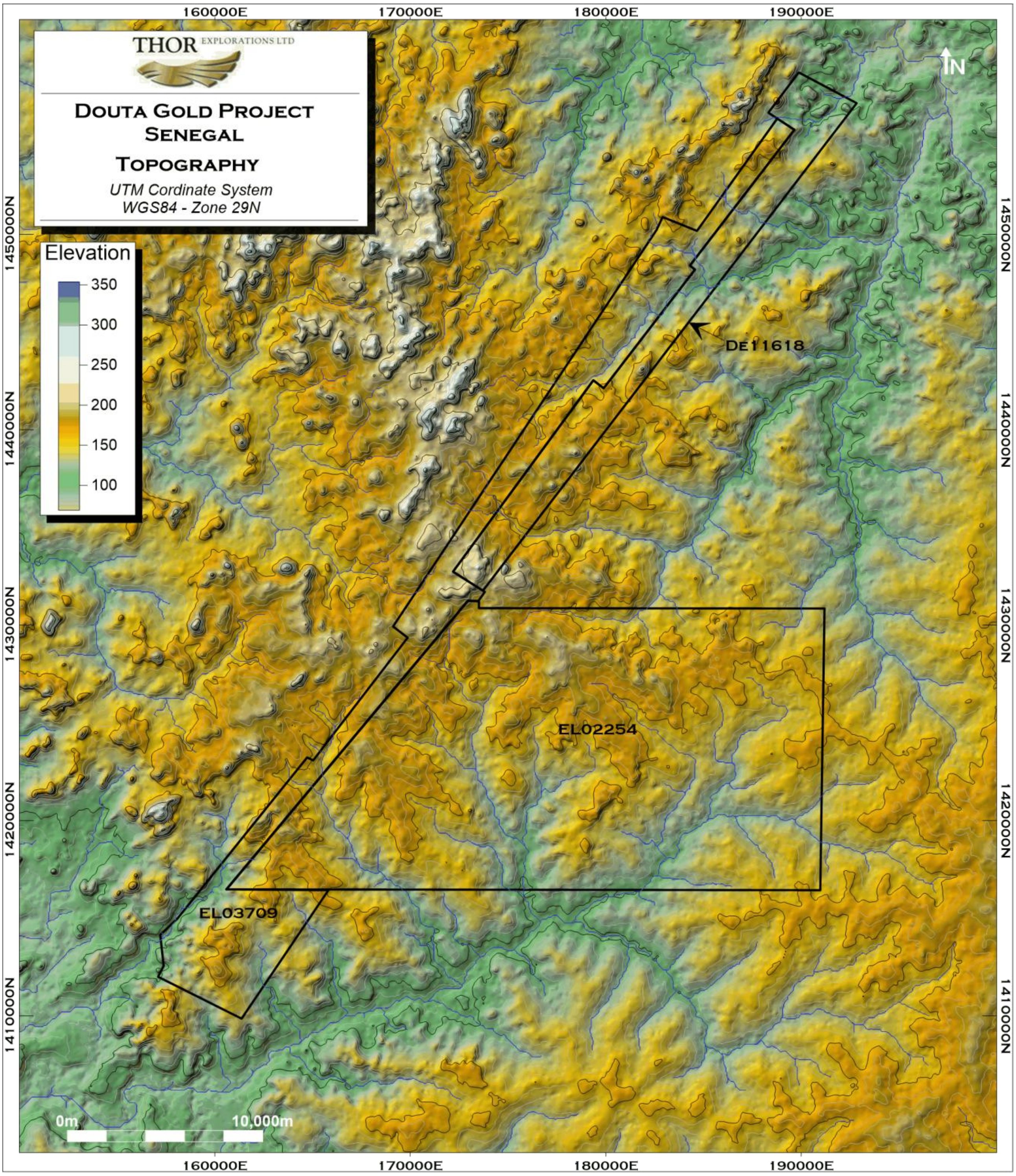
There is no national water network through the area. At the Louba Exploration Camp non-potable water is sourced from a borehole with drinking water supplied in bottled form.

There is no (or very limited) national power grid through the area. Presently, an on-site diesel generator, fuelled regularly by purchases from either the Bembou, Saraya, or Kedougou fuel stations, supplies electricity power to the camp.

5.5 Physiography

The topography of the area is generally undulating with elevations ranging from 115 m to a maximum of 240 m above mean sea level (Figure 5.1). The Faleme River is located approximately 30 km from the northern margin of the permit with the drainage characterised by a dense network of small seasonal streams. Vegetation ranges from savannah to thick bushes and large trees on hillsides. Watercourses are sporadically marked by palms.

Figure 5.1 Topographic map of the Douta Project area and surrounds



Source: Thor, 2025.

6 History

6.1 Previous ownership

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

Although artisanal mining has long been one of the primary activities of the Malinke people in the region, the Makosa deposit has not been extensively exploited.

The Douta permit was initially part of Randgold Resources's Kounemba permit. This land package comprising three licences (Kounemba, Kanoumering, and Tomboronkoto) was selected based on a mineralised structure that was interpreted from Landsat imagery to extend south from the Sabodala gold deposit and Niamia Permit in the North, where thick sequences of deformed volcaniclastic rocks including andesitic lithic tuff were found.

The late 2003 and early 2004 regional soil sampling program at 1,000 m by 100 m spacing, identified 11 high-priority targets, for detailed work. Due to the low tenor of the Massawa anomaly (initially coded as MSW, acronym for Mandakhole Southwest anomaly), it was originally selected as a secondary target. A follow-up detailed soil sampling grid program was completed in mid-2005, and identified a 3.5 km long, 100 m to 400 m wide soil anomaly at greater than 50 ppb gold in soil.

The area east of Massawa (the present Douta licence) was relinquished by Randgold Resources in 2007-2008.

In 2009, IMC was granted this area under the name of Douta permit with an original area of 103 km^2.

In February 2011, IMC entered into an option agreement with ASR. The option period ran until November 2011 and then extended for three months until February 2012.

In November 2015, the permit was renewed for another three years. In November 2018, the licence was granted a two-year extension running until November 2020.

An exploration permit is issued by order of the minister of mines for an initial period of four years. It is renewable twice for further four-year periods by order of the minister. Each renewal of an exploration permit results in the reduction by one-quarter of the area covered by the permit.

In 2015 the licence underwent a compulsory reduction to its current area of 58 km^2.

An exploration permit may be transferred, except during its first validity period. Transfer is subject to the approval of the minister of mines. The mining convention attached to the transferred exploration permit is subject to the registration formalities and the payment of capital gains tax with the tax authority.

An application for an exploitation permit has been submitted by IMC in September 2020 and was acknowledged by the Ministry of Mines the same month. The exploration licence, which expired in November 2020 after the exploitation permit application was submitted, benefits from automatic prorogation until the Ministry of Mines decides on the application for exploitation permit. The Company is party to an option agreement ("the Option Agreement") with IMC, pursuant to which, with effect from 24 February 2012, the Company exercised its option to acquire a 70% interest in the Douta Gold Project to be held through ASR.

Table 6.1 Licence acquisitions

Date	Event	Owner	Operator	Area (km^2)
Nov 2009	E02038 granted	IMC	IMC	103
Feb 2011	optioned to ASR	IMC	ASR	103
Nov 2011	option extended	IMC	ASR	103
Nov 2012	3 year renewal granted	IMC	ASR	77.92
Nov 2015	3 year renewal granted	IMC	ASR	58
Nov 2018	2 year extension granted	IMC	ASR	58
2020	Application of Douta Exploitation Licence	IMC	ASR	58
Jun 2024	Exploration licence converted to Mining application n11618	IMC	ASR	58
Apr 2024	Thor acquires 70% of Douta West Licence	Birima Gold Resources	ASR	106
Sep 2025	Thor acquires remaining 30% of Douta	Thor	ASR	58
Jan 2026	Thor acquires remaining 30% of Douta West	Thor	ASR	106

Source: Thor, 2026.

6.2 Past production

There has been no gold production from the permit area.

6.3 Exploration history

Historical generative exploration activities on the project have included geophysics, geological mapping, and soil sampling. Since acquiring the licence area, IMC with its partner ASR, carried out an extensive number of works summarised in Table 6.2.

Table 6.2 Summary of exploration activity for Douta completed by Thor

Year	Type	Quantity / No. holes / metres	Sample density / scale	No. assays	Company
2010-25	Soil sampling	6,790	400 m x 50 m	4,500	IMC
2011	Remote sensing (Aster / Landsat)		1: 40,000		ASR
2011	Trenching	8 (3,040 m)		2,700	ASR
2011	Ground magnetic survey		1,500 m x 200 m		ASR
2012	Base camp construction				ASR
2012	Fractioned sampling	992			ASR
2012-25	Termite mounds sampling,	11,693	200 m x 50 m		ASR
2012	Rotary air blast (RAB) drilling	184 (7,942 m)		3,678	ASR
2012	Diamond drilling	13 (1,531 m)		1,249	ASR
2012	Mapping				ASR
2015	Rock chip sampling	500			ASR

Source: Thor, October 2025.

6.4 Geochemical surveys

6.4.1 Douta permits

A soil geochemistry campaign was conducted by IMC in 2010. Soil samples were taken on a 400 m x 50 m grid along the entire length of the permit. In total, 4,500 samples were taken and analysed for gold.

The results of this work identified numerous soil anomalies that led to the signing of a partnership agreement with ASR.

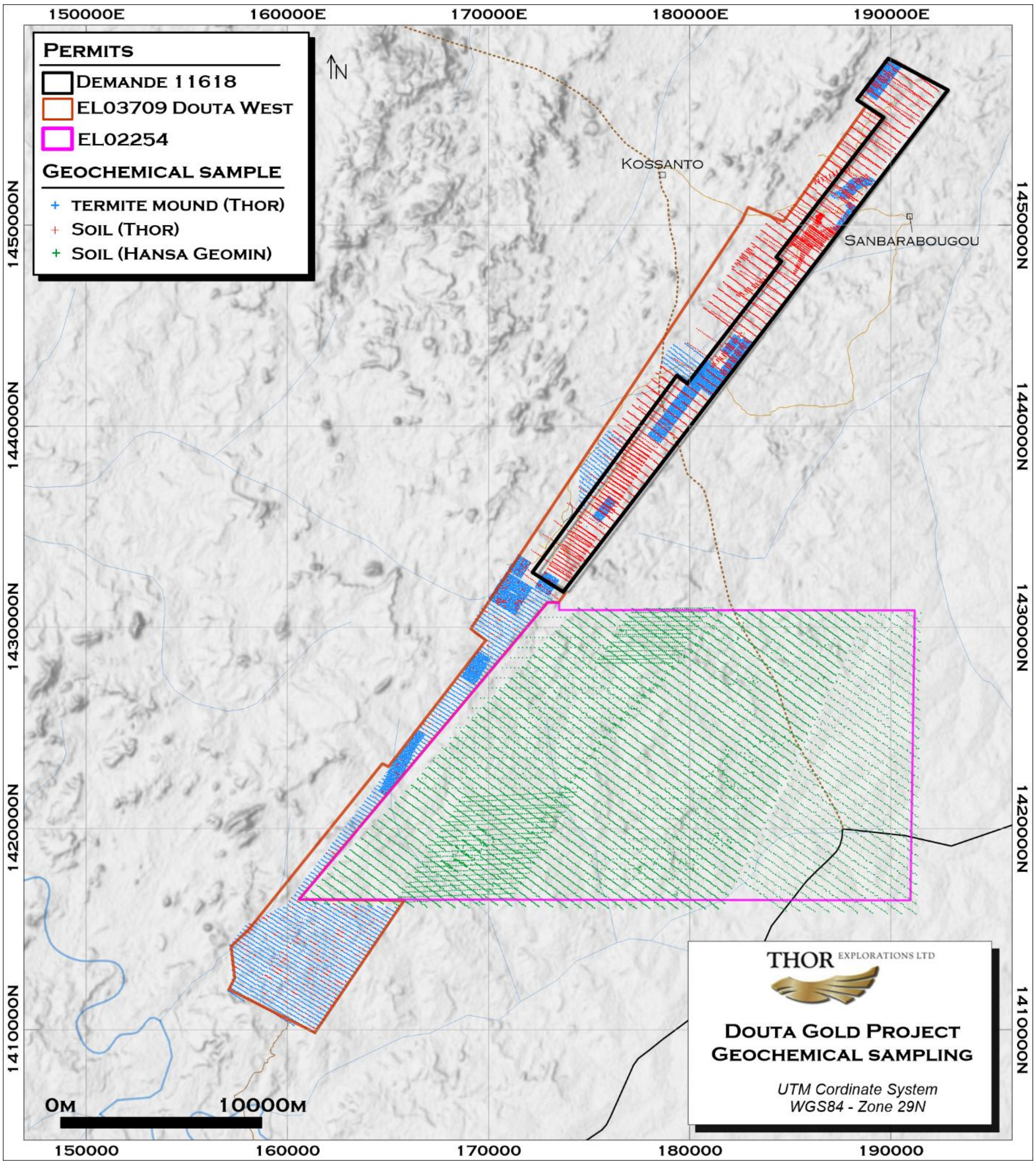
To date, a total of 18,483 surface geochemical samples has been obtained from within the Douta and Douta West permits (Figure 6.1). Most of this sampling has been either termite mound sampling (11,693) or surface soils (6,790) on a 200 m x 50 m grid.

6.4.2 Bousankhoba Permit (EL02254)

In the years 1998-2000, the German company Hansa Geomin undertook gold exploration work, primarily soil geochemistry using an average grid of 200 m x 100 m. A total of 20,243 samples were collected and assayed for gold. The results of this program revealed significant north-east trending geochemical anomalism over an 18 km strike length in the western portion of the permit.

In 2007, the permit was granted to Libah Investment, which, in a joint venture with the Australian company Bassari Resources Limited (Bassari), also undertook additional exploration work between 2009 and 2016, including termite mound and rock geochemistry (541 samples), regolith mapping, trenching.

Figure 6.1 Geochemical survey map of the Douta Project and associated permits



Source; Thor, October 2025.

7 Geological setting and mineralisation

7.1 Regional geology

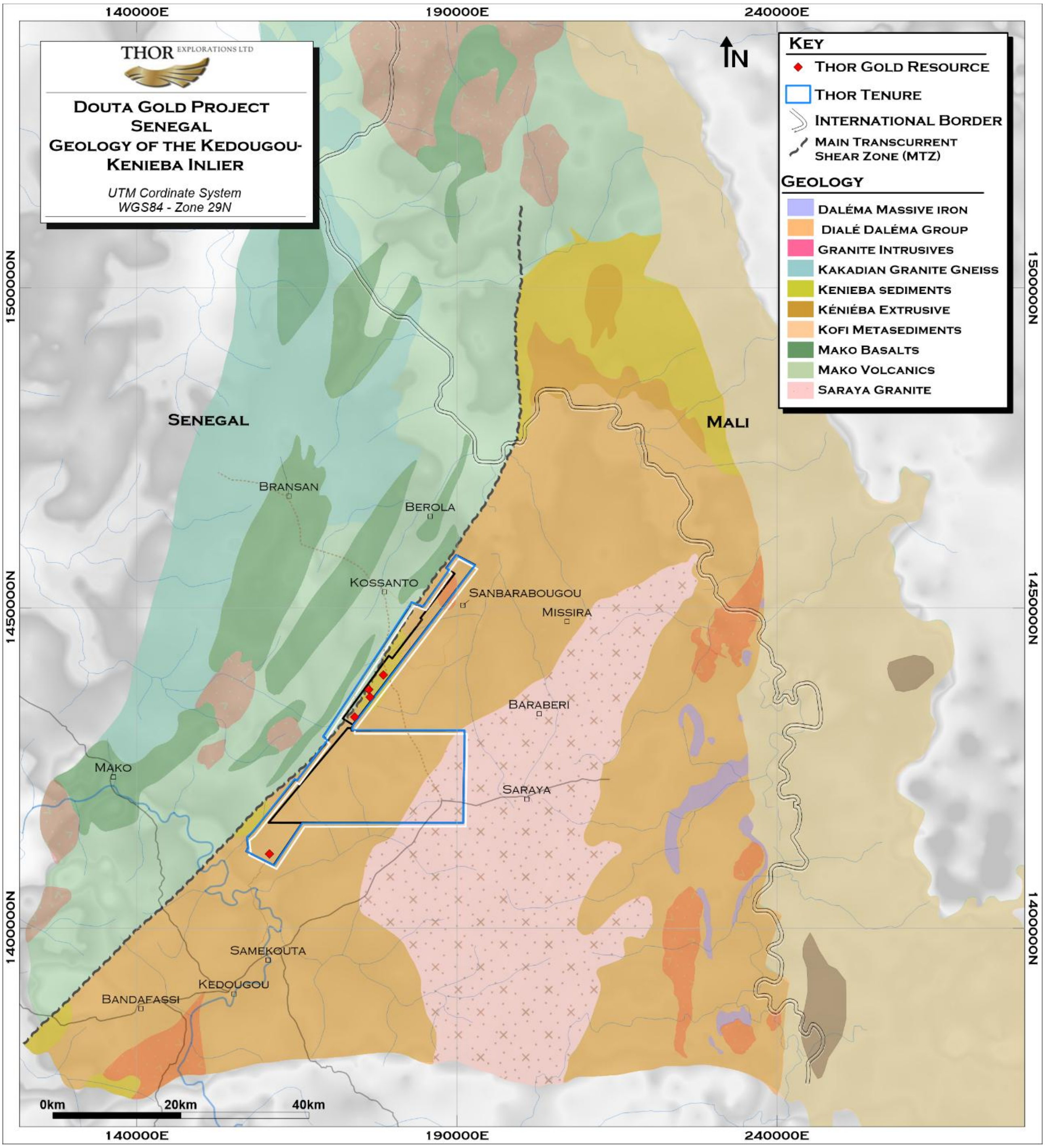
The West African Craton can be divided into three main regions that are exposed beneath the Phanerozoic cover. In the north, the Reguibat Rise extends over Mauritania and western Algeria and consists of an Archean terrane in the west and Paleoproterozoic (Birimian) terrane in the east. The southern Leo Rise covers a large area over southern Mali, Côte d'Ivoire, Burkina Faso, Niger, Ghana, and Guinea, and is separated from the Reguibat Rise by the Late Proterozoic to Phanerozoic sedimentary Taoudeni Basin. The western Archaean portion, known as the Man Shield, is separated from the eastern Birimian Supergroup of the Baoule Mossi domain by the Sassandra fault. Two Birimian inliers, the Kayes and Kedougou-Kenieba, suggest the continuity of the Proterozoic basement underneath the Taoudeni intra-cratonic basin. The Sabodala-Massawa Project is located within the 2,213 Ma to 2,198 Ma age Kedougou-Kenieba Inlier (KKI).

The KKI is divided into the Mako Belt to the west, and an overlying Dialé sedimentary basin to the east (Dia et al., 1997). The Mako Belt consists of greenstones and sedimentary rocks, dated between 2,160 Ma and 2,200 Ma, which are intruded by ultramafic to felsic plutons yielding ages of 2,070 Ma to 2,210 Ma (Dia et al., 1997; Gueye et al., 2007).

All rock types, excluding post-Birimian dykes, are metamorphosed to a lower green schist facies during the Eburnean orogeny. The belt basin margin, between the Mako and Dialé-Daléma series, is structurally controlled and marked by the regional-scale NE-trending, Main Transcurrent Shear Zone (MTSZ) (Ledru et al., 1991). The MTSZ runs along the length of the Douta Project permits (Figure 7.1).

7.2 Property geology and mineralisation

Figure 7.1 Geology of West Africa and Kedougou-Kenieba Inlier



Source: Thor, 2025.

7.2.1 Summary

The geology of the Douta permit is dominated to the east by the Dialé sedimentary formations and to the west by the mafic and volcaniclastics formations of the Mako Belt (Figure 7.2).

From south to north, the main structural feature of the exploration licence is the NNE-to-NE striking MTSZ.

A parallel structure located 2.5 km to the west of the MTSZ hosts the Massawa gold deposit and its satellite deposits owned by Endeavour Mining.

The properties comprising the Douta Project can be subdivided into two main resource areas both of which are hosted by sedimentary and volcano-sedimentary rocks of the Dialé Group:

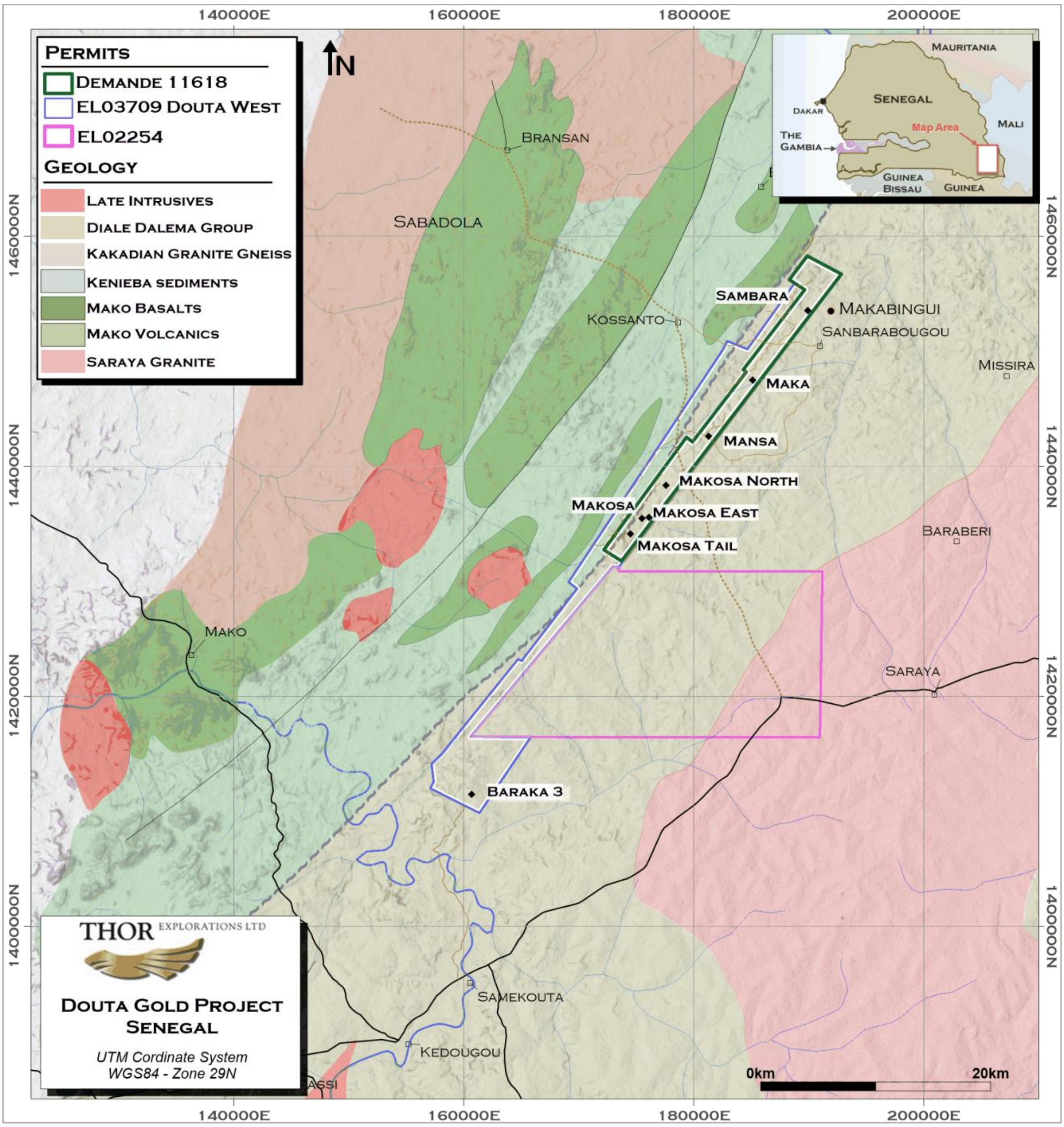
- Makosa, Makosa North, Makosa East, Makosa Tail
- Baraka 3

In addition, there are three exploration areas:

- Sambara
- Maka
- Mansa

Both resource areas present similar lithologies, which generally trend north-northeast with steep dips towards the north-west. The rocks have been metamorphosed to the lower greenschist facies. Turbidite sediments (greywacke), volcano sedimentary rocks, and graphitic shales dominate the sequence, which, in the vicinity of Makosa and Makosa Tail, is conformably intruded by a gabbro dyke. At Makosa East, granitic rocks have been intersected in the footwall sequence.

Figure 7.2 Prospect location and geology



Source: Thor, October 2025.

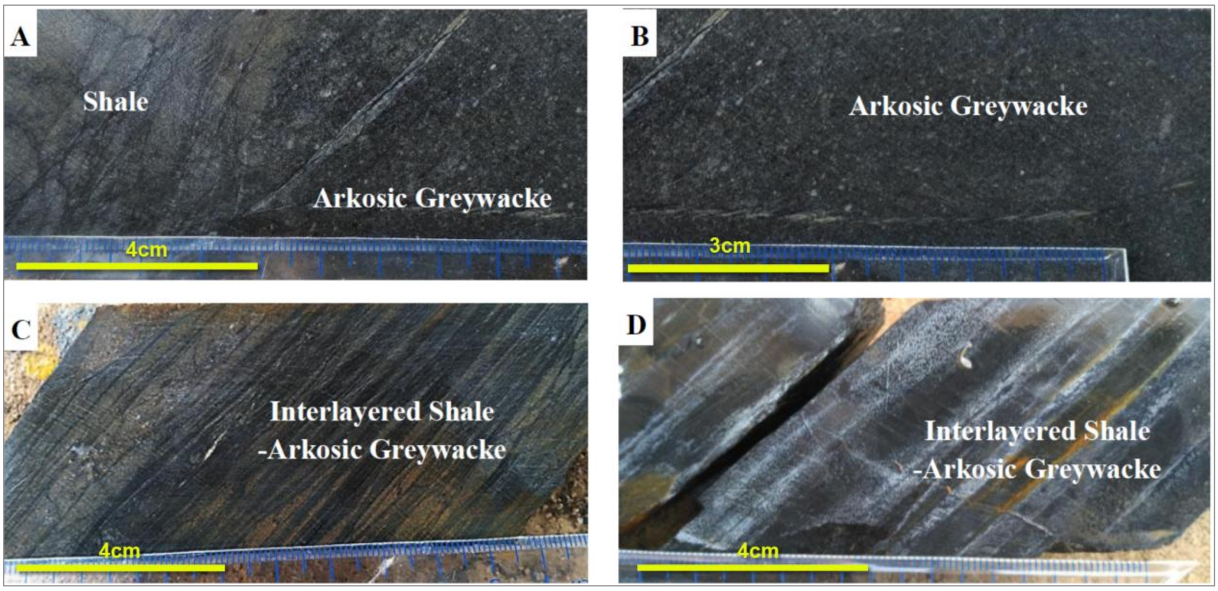
7.2.2 Lithology

7.2.2.1 The turbidite sedimentary sequence

The turbidite sequence constitutes the primary geological unit of the Makosa Prospect and Baraka 3. This rock is typically massive and locally cleaved, comprising alternating centimetre to metre-thick layers of coarse-grained sandstone, interpreted as arkosic greywacke, and fine-grained siltstone, interpreted as shale. Bedding generally strikes north-east and dips steeply at 75° to 80° toward the north-west. Bedding orientations are observable in both outcrop and oriented drill core and is typically aligned with the Makosa Fault systems.

Medium- to fine-grained arkosic greywacke serves as the principal host rock for gold mineralisation (Figure 7.3). It predominantly consists of monocrystalline, irregularly shaped albite and finely crystalline aggregates of quartz fragments or clasts, each a few millimetres in size, with interstitial sericite embedded in a dark, poorly sorted, metamorphosed groundmass. The shale is a fine-grained, dark-grey rock interbedded with the coarser arkosic greywacke. Contacts between these two lithologies range from sharp to gradational. The shale is locally graphitic and contains euhedral, possibly syngenetic, sulphide minerals (Dieng, 2018).

Figure 7.3 Photographs of typical turbidite rocks



Notes:
A Gradual contact fine-grained shale-coarse-grained arkosic greywacke.
B Medium to fine-grained arkosic greywacke consisting predominantly of irregularly-shaped albite and finely crystalline aggregates of quartz fragments / clasts, interstitial sericite embedded in a dark poorly-sorted metamorphosed groundmass.
C & D Finely bedded Shale-Arkosic greywacke.
Source: Dieng, 2018.

At Baraka 3, the shale is similar to that described in Makosa, with finely interlayered shale-greywackes Figure 7.4). The shale is less competent than the greywackes and shows ductile deformational features.

Figure 7.4 Photograph of interlayered shale and greywacke from Baraka 3

Source: Thor, 2025.

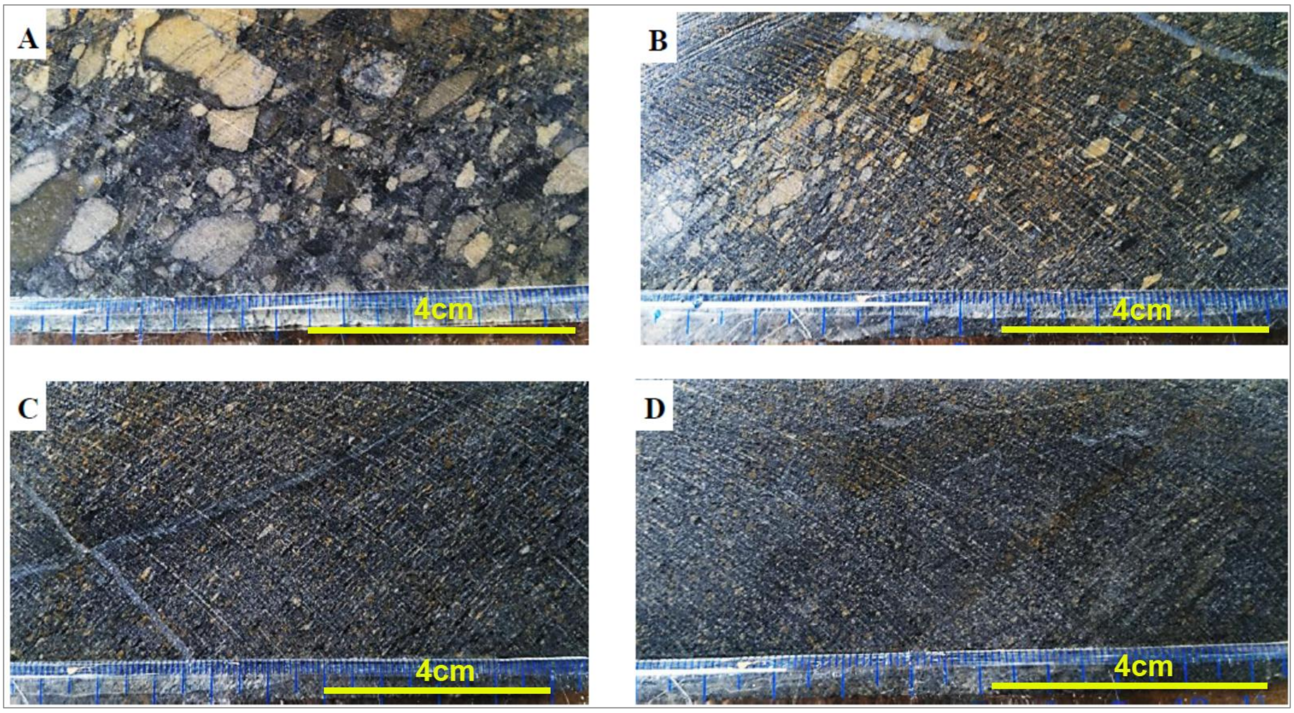
7.2.2.2 Volcano sedimentary rocks

Volcaniclastic rocks are locally interbedded with the turbidite sequence. These rocks consist of alternating layers of dark green tuffaceous material and pyroclastic deposits. The ash and lapilli layers exhibit well-sorted fragment size distributions which indicate a north-westward younging direction.

Tuffaceous assemblages comprise dense accumulations of well-crystallised, fine-grained material with uniform fragment size (Figure 7.5).

Pyroclastic assemblages are interlayered with tuffaceous rocks and are distinguished by a predominance of volcanic fragments of variable size and shape, ranging from sub-angular to rounded, and coloured pink to pale green. These fragments are embedded in a fine-grained, dark-green, sericite-rich matrix. The assemblages contain mafic volcanic rock fragments and thinly laminated tuffaceous material (Dieng, 2018).

Figure 7.5 Photographs of the volcanoclastic rocks



Notes:

A Pyroclastic rock composed of large fragments and clasts embedded in a fine-grained dark-green and sericite-rich matrix.

B Pyroclastic rock with predominance of preferentially-oriented rounded and variably-sized fragments of volcanic materials embedded in a fine-grained matrix.

C Greenish tuffaceous rocks with predominance of ash particles.

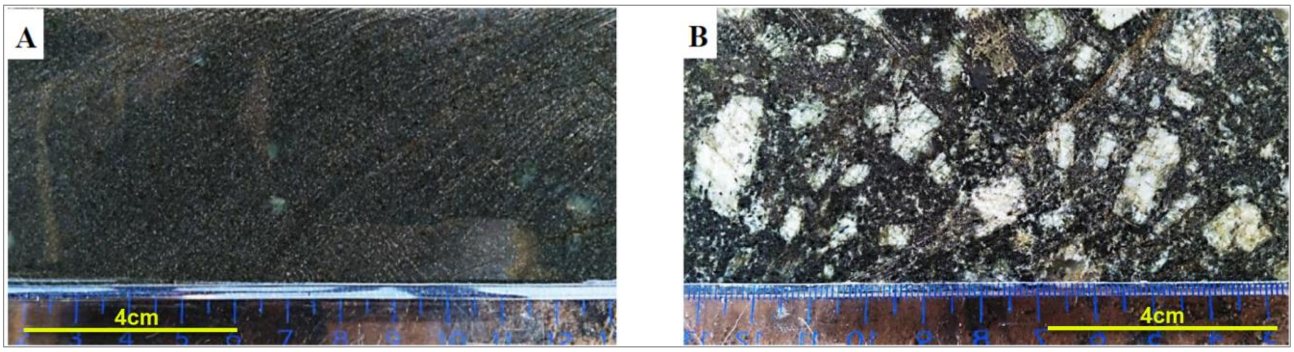
D Dense package of tuffaceous rock with well crystallised homogenous fine-grained materials.

Source: Dieng, 2018.

7.2.2.3 Gabbro dykes

The gabbroic rocks exhibit a dark greyish-green colour and display fine-grained and porphyritic textures (Figure 7.6). Porphyritic gabbros are mapped south of the deposit and transition to finer-grained varieties toward the north (Dieng, 2018). The dyke measures approximately 100 m in thickness at Makosa Tail and thins northward within the central Makosa deposit. The gabbro is concordant with the principal fault systems and the sedimentary host unit, which parallels the Main Transcurrent Fault. It is predominantly undeformed, although local weak deformation is observed. In certain areas, the gabbro is displaced by east-west to east-northeast-striking late faults. Gold mineralisation at Makosa Tail is primarily developed along the eastern margin of the dyke. In the northern part of Makosa Tail, mineralisation crosses the dyke and is situated along the western margin at the northern end (Figure 8.1 and Figure 8.2). At the southern end of the Makosa deposit, mineralisation occurs in both the footwall and hanging wall of the dyke, as well as within the dyke itself (Figure 8.2).

Figure 7.6 Photographs of the gabbro dyke



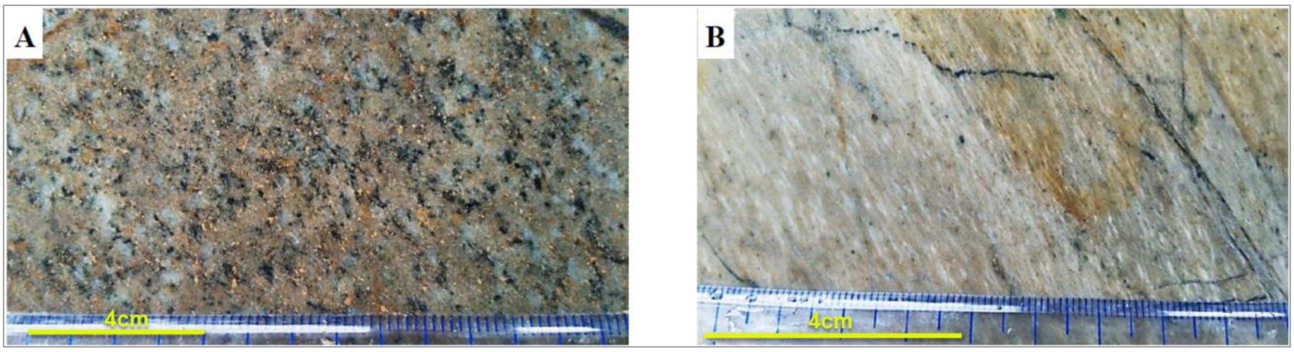
Notes:
A Fine-grained gabbro dyke.
B Porphyritic-textured gabbro.
Source: Dieng, 2018.

At Baraka 3, thin gabbroic dykes have been mapped to both the east and west of the mineralised zones. However, the mineralisation is not developed in proximity to the gabbro as is observed at Makosa.

7.2.2.4 Granitic rocks

The granitic rocks at Makosa are predominantly medium to fine-grained and exhibit equigranular to weakly porphyritic textures. These dykes are distinguished by pink K-feldspar phenocrysts within a gray, equigranular groundmass composed of subeuhedral plagioclase, anhedral quartz, biotite, and minor dark green hornblende. The granite is located in the footwall of the Makosa Fault, displays a sub-vertical and steep north-west dip, and is emplaced within discontinuous fracture systems. It is concordant with the main north-east-striking fault zone that controls gold mineralisation (Dieng, 2018). Tectonic stresses have resulted in overprinting by fractures, faults, and shears, which are likely associated with the Late Eburnean Event (Figure 7.7).

Figure 7.7 Photographs of the granite dyke



Notes:
A Undeformed granite.
B Sheared and fractured granite dyke.
Source: Dieng, 2018.

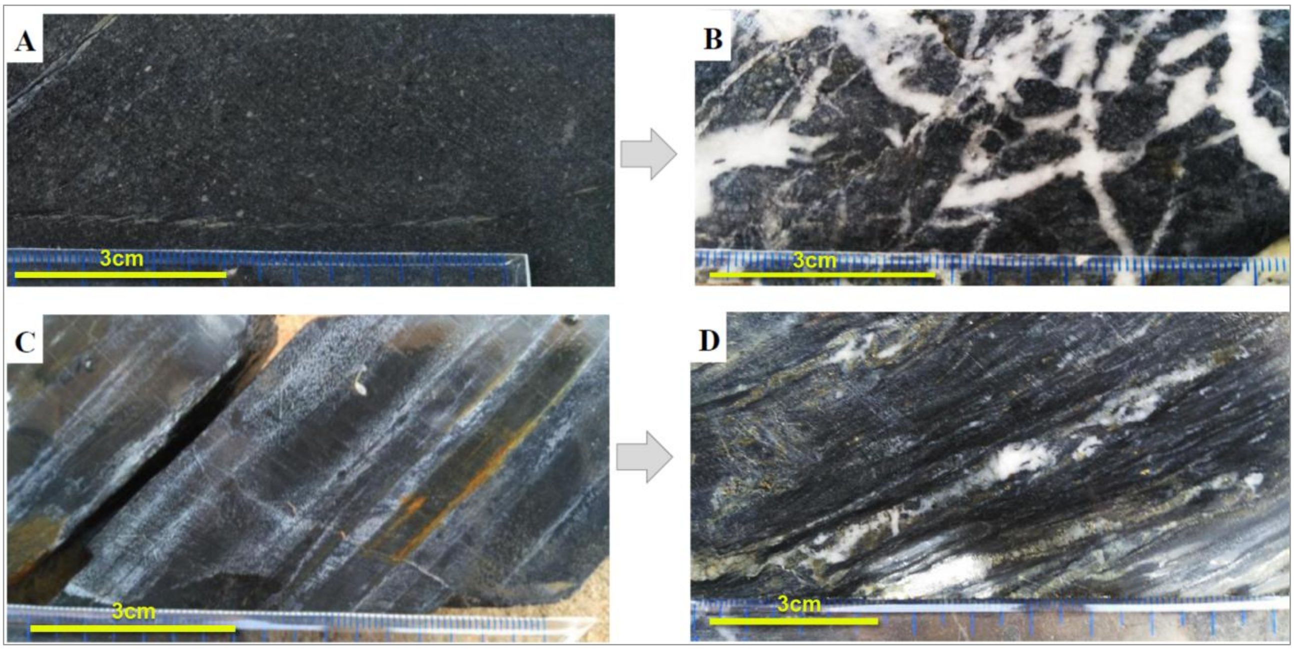
7.2.3 Structure

7.2.3.1 Summary

Gold mineralisation is associated with the north-east-striking, steeply north-west-dipping (75° to 80°) Makosa Shear Zone, which forms part of the MTSZ shear system. Rocks within the Makosa Shear Zone display a north-west-dipping foliation (S1) and a north-east-trending elongation lineation, defined by boudinaged clasts within siltstone layers, sedimentary breccia, and pyroclastic rocks (Figure 7.8).

The deformation fabric is heterogeneously developed in sedimentary rocks due to variations in rheological properties. Shear strain is concentrated in relatively weak lithologies such as siltstone, which exhibits a pronounced penetrative foliation, rotation, and boudinage structures (Dieng, 2018). In contrast, the more competent arkosic greywackes display brittle deformation features, including brecciation and veining. The brecciated rocks are hydrothermally altered, show evidence of fluid-rock interaction, and are locally invaded by a stockwork of quartz veins (Dieng, 2018).

Figure 7.8 Photographs of various deformation types in the Makosa Shear Zone

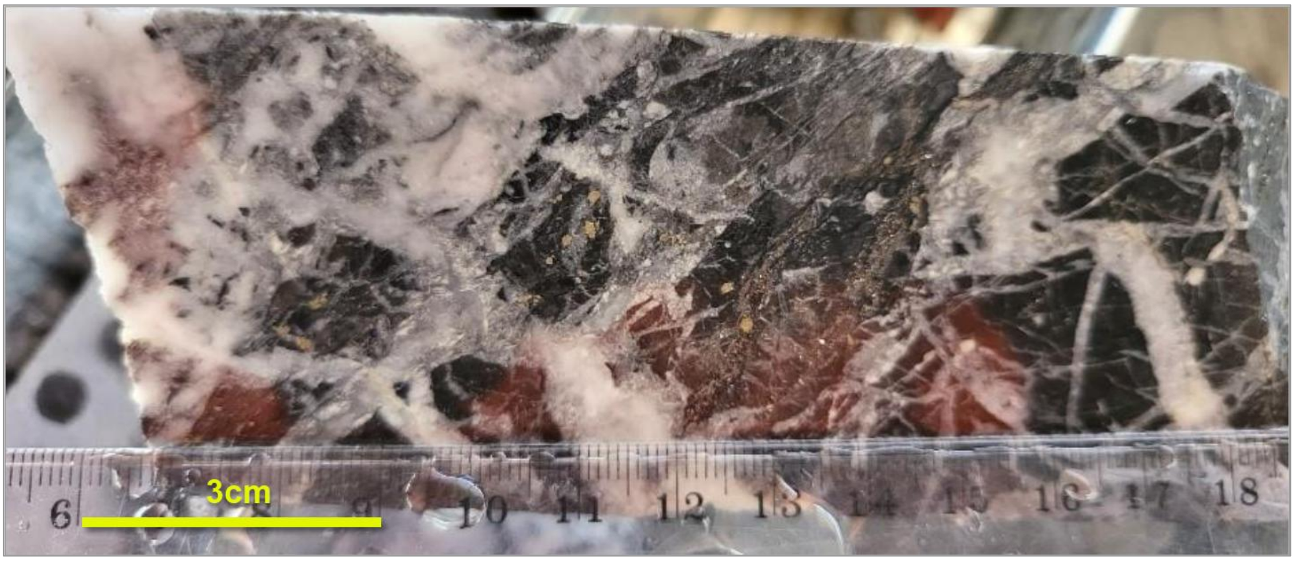


Notes:
A Undeformed arkosic greywackes.
B Arkosic greywackes displaying brittle features including brecciation and veining.
C Undeformed shale.
D Penetrative foliation and rotation and boudinage features in shale host rock.
Source: Dieng, 2018.

The Makosa deposit formed from left-stepping geometry, which is consistent with extensional fault systems in a sinistral-reverse compressional tectonism, similar to the D2 deformation of the late Birimian event that affected the West African Shield (Dieng, 2018).

At Baraka 3, gold mineralisation is associated with two main strike directions - either north-south (Baraka East) or north-east (Baraka West). The mineralised structures are steeply dipping and form part of the MTSZ shear system. The deformed greywackes display brittle deformation features, including brecciation and veining (Figure 7.9). The brecciated rocks are hydrothermally altered, indicating fluid-rock interaction.

Figure 7.9 Photograph of greywackes showing brittle features including brecciation and veining



Source: Thor, 2025.

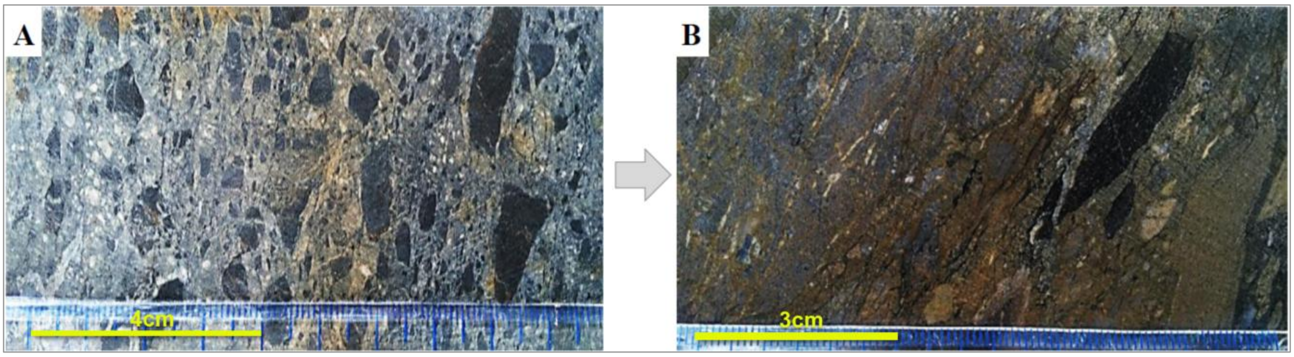
7.2.3.2 Structural history

Field investigations and analysis of drill-core structures show that the Makosa Shear Zone has experienced at least two major overlapping phases of compressional deformation. The first is an early ductile deformation, likely from the D1 deformation phase, which probably occurred deeper in the crust. The second is a later, more brittle-ductile phase that took place at a shallower crustal level and overprints the earlier ductile features.

7.2.3.3 Early D1 deformation phase

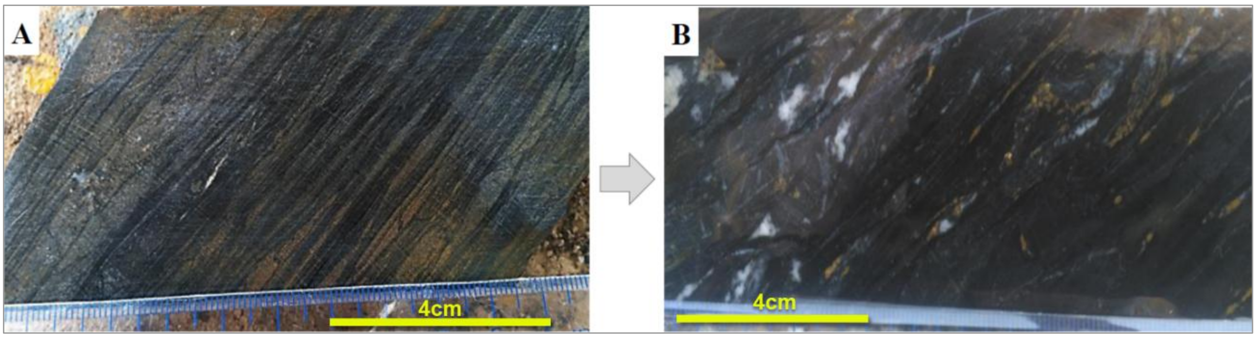
The ductile component of the Makosa Shear Zone represents the earliest identified deformation phase. This shear zone exhibits a north-west-dipping foliation and a north-east-trending elongation lineation. The lineation is characterised by elongated clasts within sedimentary breccias (Figure 7.10), pyroclastic rocks, and shale layers (Figure 7.11).

Figure 7.10 Photographs of the shear zone and typical ductile features in the sedimentary breccia in core



Notes:
A Undeformed sedimentary breccia.
B D1 Ductile deformation with penetrative foliation, boudinage and rotation and preferential mineral orientation affecting the sedimentary breccia.
Source: Dieng, 2018.

Figure 7.11 Photographs of the shear zones and ductile features in the shale layer in plan view section



Notes:
A Undeformed shale.
B D1 Ductile deformation with penetrative foliation, boudinage and rotation and preferential mineral orientation within the shale.
Source: Dieng, 2018.

7.2.3.4 D2 deformation phase

The brittle-ductile deformation within the Makosa Shear Zone overprints earlier ductile structures. Brittle-ductile and brittle features, including veins and breccias (Figure 7.11), are most prominent in arkosic greywacke sedimentary rocks. Planar and sigmoidal quartz-filled tension gashes are widespread along the Makosa Shear Zone and serve as reliable indicators of shear sense, consistently demonstrating sinistral lateral motion. These tension gashes, which are extensional fractures formed in the brittle-ductile regime, occur in en échelon arrays and represent the most distinctive structural features reflecting the orientation of the original stress field.

7.2.4 Mineralisation

7.2.4.1 Summary

Gold mineralisation is structurally controlled, hosted in faulted and sheared contact between sedimentary successions and occurs in zones of large north-east-striking, north-west-dipping structural corridors, that contain a complex network of extensional dilation fracture systems.

The style of gold mineralisation is similar along the 12 km shear zone within the Douta permit. The mineralisation is hosted by deformed sedimentary rocks near the contact with gabbro or volcaniclastics.

Gold mineralisation is associated with a low-temperature mineral assemblage, including quartz stockworks, sericite, chlorite, and calcite. Abundant disseminated fine pyrite and arsenopyrite occur in the wall rock.

Two generations of pyrite have been identified:

- Syngenetic (primary) massive, sometimes spherical, pyrite.
- Fine-grained pyrite associated with a hydrothermal gold mineralisation event.

Hematite and carbonate-chlorite alteration is widespread.

The gabbro is locally mineralised in places, suggesting two generations of gabbro in the area. Hydrothermal gold-bearing fluids are thought to have originated from metamorphic fluids generated from dehydration of water-rich minerals during thermal-tectonism or from hydrothermal fluids degassing from syn-orogenic calc-alkaline felsic intrusive magma that intruded the deposit (Dieng, 2018).

7.2.4.2 Alteration

The principal alteration minerals associated with gold mineralisation are chlorite, sericite, quartz, carbonate, pyrite, and arsenopyrite. Gold is disseminated within the wall rock or occurs in association with quartz veins.

Chlorite is present as light to dark green sheet-like aggregates, disseminated patches, veins, or pervasively distributed within the groundmass of the mineralised system. It likely replaced amphibole and biotite in the arkosic greywacke host rock. Chlorite commonly occurs with quartz and calcite in veins and is also associated with gold and arsenopyrite. Inclusions of chlorite are observed within pyrite. The association of chlorite and calcite indicates the presence of a syn-ore hydrothermal mineralising fluid at approximately 300°C during gold deposition.

Calcite is observed as irregular small patches within the groundmass of the host rock. It is also intergrown with quartz or chlorite in veins. Calcite typically shows partial to extensive replacement by chlorite. During hydrothermal alteration, plagioclase feldspar may be either partially or completely replaced by calcite in both the matrix and individual mineral grains of the host rock.

Quartz occurs as veinlets, fracture fillings, or as pervasive alteration of the wall rock. Quartz and chlorite are closely associated, with quartz overgrowing chlorite and forming rims around chlorite grains. Quartz is also present as stockwork that brecciates the host rock. The quartz veins frequently display syn-tectonic recrystallisation, which is indicative of deformation.

Sericite is present as light-yellow aggregates that replace albite in the arkosic greywacke host rock, likely formed during metamorphic retrogression. It is also disseminated as very fine grains within the groundmass of both arkosic greywacke and shale and is interpreted to be part of an alteration assemblage that predates mineralisation.

Pyrite is the most abundant sulphide mineral identified. Two types of pyrite are distinguished: anhedral and euhedral. Anhedral pyrite is temporally associated with the mineralising hydrothermal fluid and occurs as inclusions in quartz veins, stringer veinlets, blebs, patches, and fine grains disseminated within the mineralised rock. Euhedral pyrite is present in both undeformed and unmineralised host rock and the mineralised system.

Arsenopyrite displays a wide range of grain sizes from a few micrometres to several hundred micrometres. It occurs as finely disseminated euhedral crystals, aggregates of crystals, and relatively large euhedral porphyroblasts. Arsenopyrite frequently shows minor incipient oxidation and replacement by iron-rich oxidation products (Blake, 2012). A strong association exists between arsenopyrite, pyrite, and gold mineralisation in these ores.

Minor amounts of pyrrhotite, marcasite, covellite, and chalcopyrite are present in the sulphide assemblage. Pyrrhotite commonly occurs as an accessory mineral and is closely associated and forms inclusions with arsenopyrite. Covellite is present in trace amounts, also as inclusions within arsenopyrite. Marcasite aggregates display lath-like textures which are indicative of the replacement of former pyrrhotite grains as described by Blake (2012).

7.2.4.3 Gold mineralisation

Gold mineralisation is primarily controlled by structural features and occurs within an extensive zone of deformation and hydrothermal alteration characterised by quartz, calcite, chlorite, pyrite, and arsenopyrite. Gold grains are observed in association with pyrite and arsenopyrite. Many of the native gold grains occur as very fine-grained inclusions in the arsenopyrite. However, a significant portion occurs along the contact between the arsenopyrite and the abundant silicate rich inclusions (Blake, 2012).

Gold mineralisation is developed over a 6 km strike length in sheared and brecciated sedimentary rocks, specifically arkosic greywacke and shale, located in the footwall of the Makosa Shear Zone, which trends northeast and dips steeply to the north-west. Mineralisation is not visually discernible and varies from 2 m to over 20 m true width. Gold grades are evenly distributed throughout the deposit with a slight overall increase in grade towards the north.

7.2.5 Mineralogy

7.2.5.1 Summary

Dr Chris Blake, Mineralogy Consultant, completed petrographic analyses on gold-bearing chip and core samples from Makosa. Each polished and thin section was systematically examined using conventional reflected and transmitted light microscopy with the individual minerals being identified based on their optical properties. A series of photomicrographs were also prepared to illustrate important mineralogical and textural features.

Several polished sections were also examined using a Scanning Electron Microscope (SEM) with the individual minerals being identified based on their mineral chemistry as determined by qualitative energy dispersive microbeam analyses. A series of backscattered electron images were prepared to illustrate important mineralogical and textural features (Figure 7.12).

7.2.5.2 Gangue mineralogy

The gangue mineralogy of the samples consists predominantly of quartz, plagioclase, fine-grained muscovite, and chlorite together with minor amounts of calcite, clay minerals, and carbonaceous materials (Blake, 2012).

7.2.5.3 Ore / sulphide mineralogy

Pyrite / marcasite and arsenopyrite are the dominant sulphide minerals and account for less than 5% (w/w) of the samples as a whole. Accessory ore minerals observed in minor amounts include pyrrhotite, covellite, and lollingite.

7.2.5.4 Arsenopyrite

Arsenopyrite accounts for the bulk of the arsenic content and occurs as finely disseminated euhedral crystals, as aggregates of crystals and as relatively large euhedral porphyroblasts. The largest arsenopyrite grains exceed several hundred micrometres. The bulk of the arsenopyrite occurs as poikiloblastic euhedral crystals that contain abundant inclusions and intergrowths of transparent gangue minerals. The arsenopyrite often exhibits minor incipient oxidation and replacement by Fe-rich oxidation minerals. There is a strong association between arsenopyrite and the gold mineralisation of these ores (Blake, 2012).

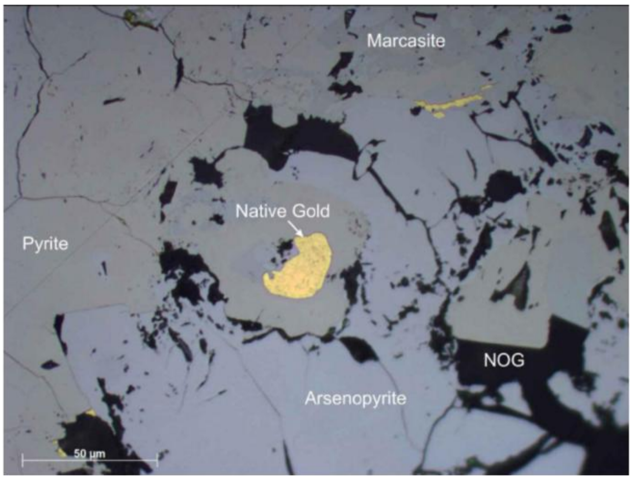
7.2.5.5 Pyrite / marcasite

Pyrite is common and occurs predominantly as euhedral porphyroblastic crystals that form overgrowths on and partially replace arsenopyrite. Discrete pyrite crystals commonly exceed several hundred micrometres in size. Marcasite is also common and typically exhibits lath-like textures that are characteristic of the replacement of pyrrhotite grains. The marcasite exhibits varying degrees of replacement by pyrite. Trace amounts of arsenic are present within some of the pyrite grains. Arsenic-zoned pyrite is a common host for microscopically invisible lattice-bound gold, and it is expected that a proportion of the gold content of these ores might be present in this form. The pyrite often exhibits minor incipient oxidation and replacement by Fe-rich oxyhydroxides. There is a close association between pyrite and gold mineralisation (Blake, 2012).

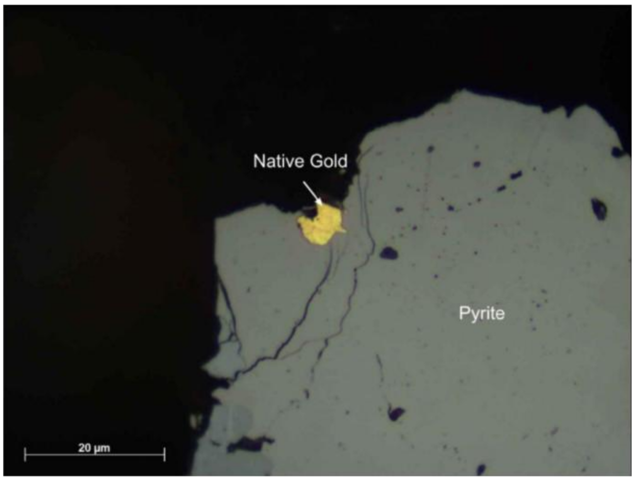
7.2.5.6 Gold associations

Many occurrences of microscopically visible native gold grains were observed with the most common association being fine-grained native gold with arsenopyrite (Figure 7.12). The largest grains identified in this association exhibited a mean circular diameter of less than 15 µm, although most grains were less than 5 µm. The second most common association observed during the examination of the heavy mineral concentrates was native gold with pyrite and marcasite. The native gold associated with pyrite / marcasite is also typically fine-grained. The largest observed grain exhibited a mean circular diameter of 20 µm. An Associations between native gold and non-opaque gangue (NOG) minerals are far less common than the gold-sulphide associations The native gold grains associated with NOG are typically fine-grained with the largest grain exhibiting a mean circular diameter of approximately 12 µm (Blake, 2012).

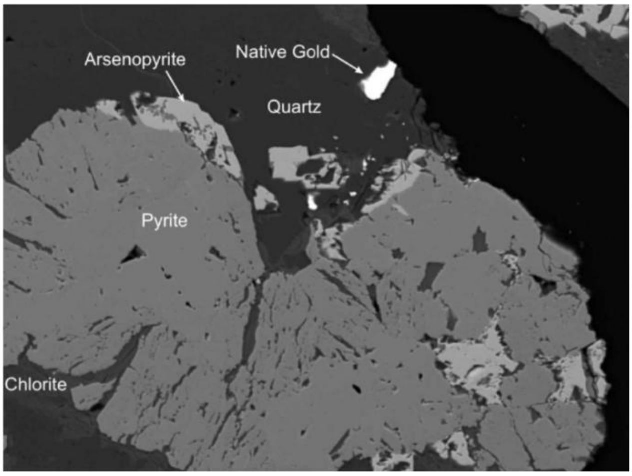
Figure 7.12 Photomicrographs showing gold distribution



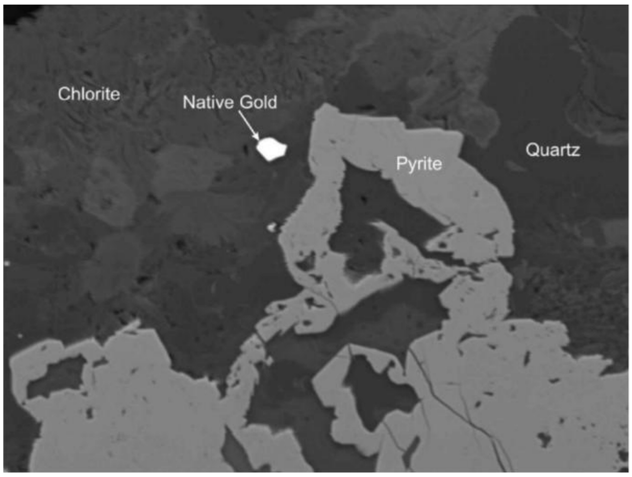
A reflected light photomicrograph of the same area illustrated. Marcasite (mottled grey shades) is also weakly resolved in the pyrite.



A reflected light photomicrograph illustrating a native gold grain on the margins of pyrite.



A backscattered electron image illustrating numerous native gold grains in quartz and chlorite.
Width of view approximately 300µm.



A back-scattered electron image showing native gold grain within an altered and compositionally zoned chlorite/clay aggregate. The chlorite is intergrown with quartz and pyrite.
Width of view approximately 160µm.

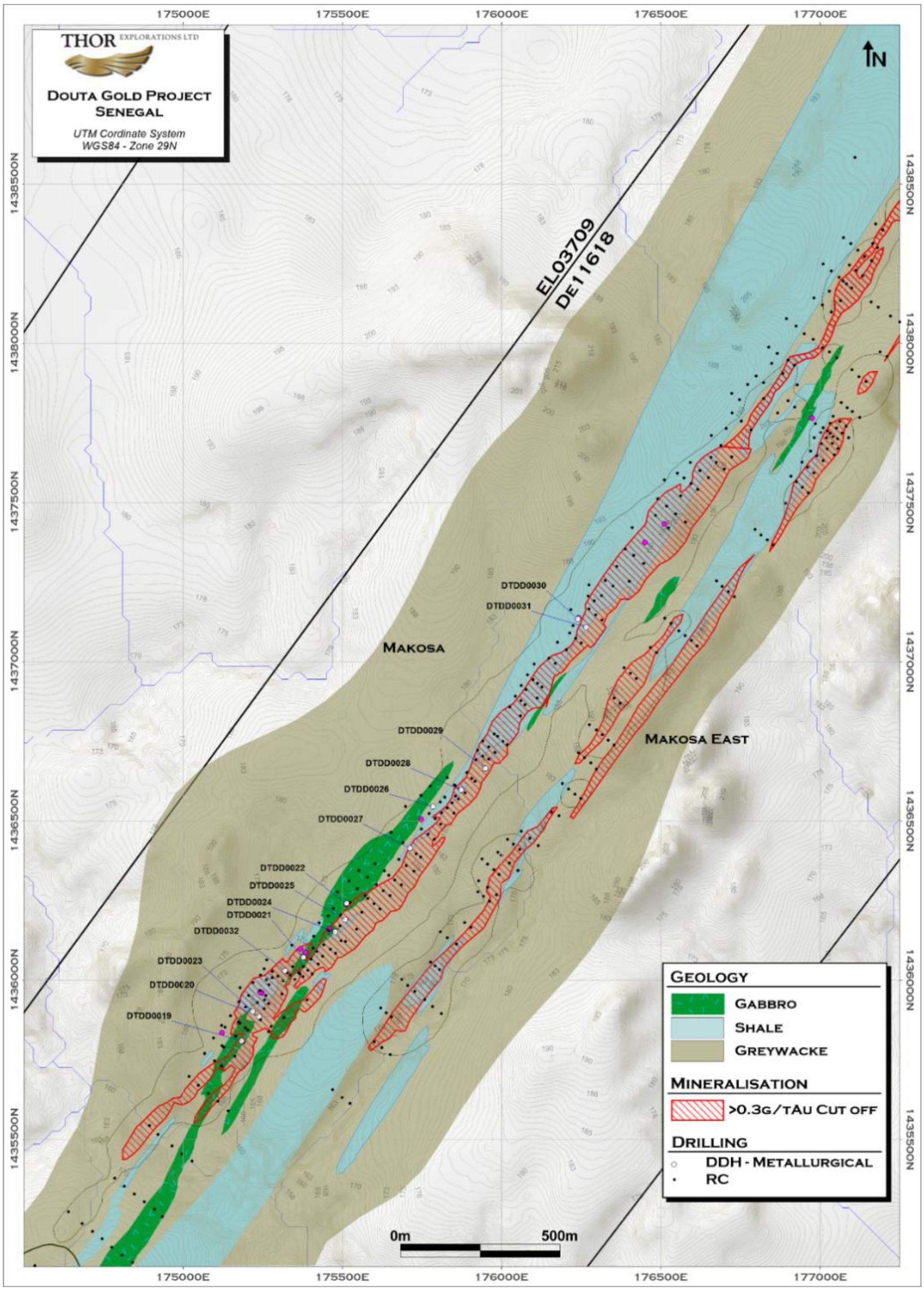
Source: After Blake, 2012.

8 Deposit types

The Makosa deposit is a typical shear-zone-controlled orogenic-type gold mineralisation hosted in a greenstone, folded, and faulted sedimentary sequence of turbidite successions near the contact with syn-tectonic gabbroic intrusive dykes (Dieng, 2018).

The main Makosa deposit (including the northern extensions known as Makosa North) is continuous over a NE-SW oriented (043°) strike length of 5.5 km (Figure 8.1). The southern extremities pinch out into weak stringer-like mineralisation in the Makosa Gap zone which covers 350 m of strike-length.

Figure 8.1 Geology of the Makosa Resource Area

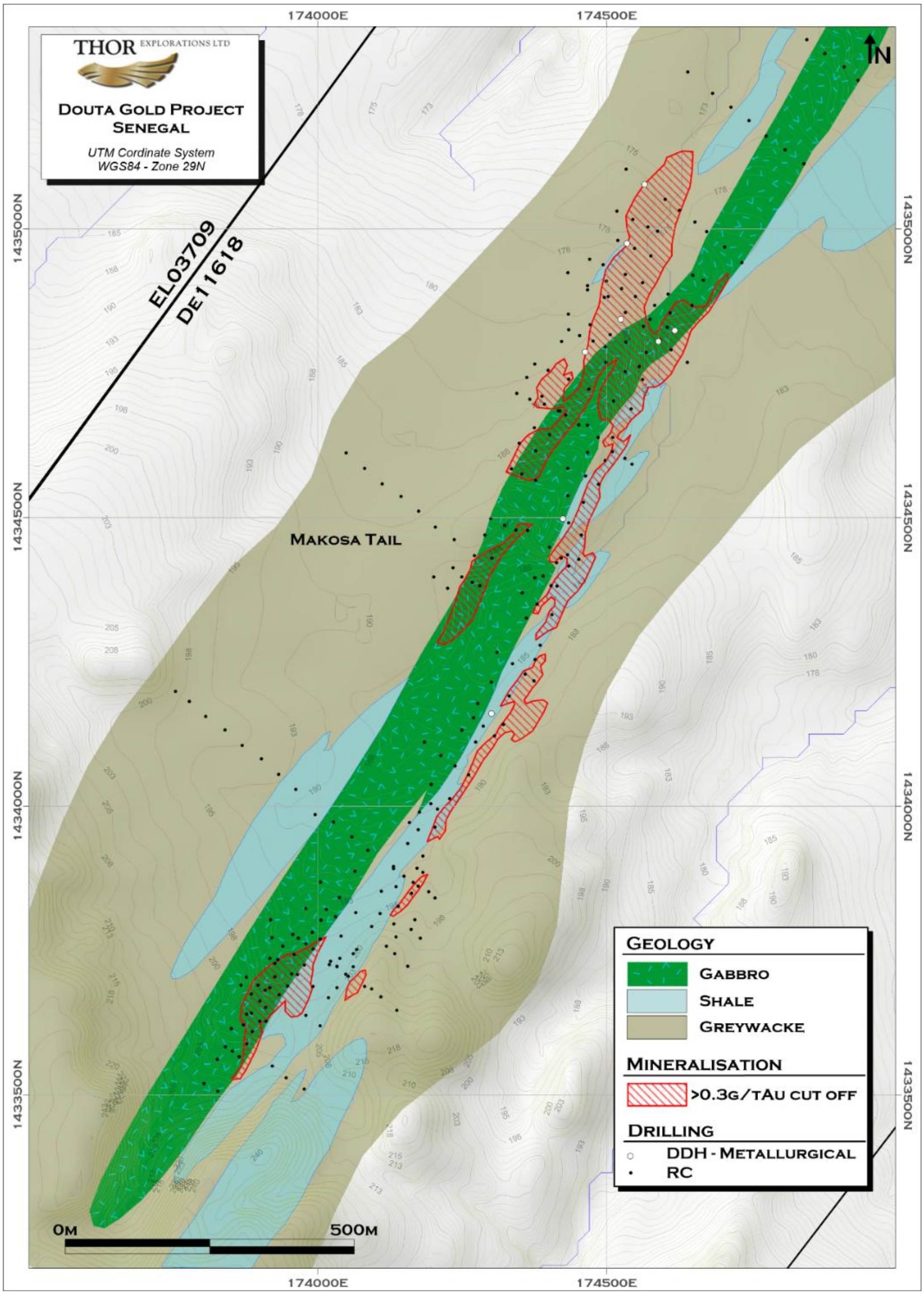


Source: Thor, October 2025.

The Makosa Tail deposit extends southwards from the southern extremity of the Makosa Bridge. Most gold mineralisation throughout the system is developed within sediments in close proximity to conformable gabbroic intrusive rocks (Figure 8.2).

The Baraka 3 deposit is developed over a strike length of 800 m along the NS to NNE structure, which appears to pinch out along the northerly trend but is open to the north along the NE trend.

Figure 8.2 Geology of the Makosa Tail Resource Area



Source: Thor, October 2025.

9 Exploration

9.1 Summary

The following data types have been collected and compiled:

- Airborne and ground geophysics are interpreted and integrated with field geology (regolith and outcrop mapping) to identify major prospective structures, lithologies, and alteration zones that will provide a project-scale regolith framework in which the context of any surface geochemistry can be evaluated.
- Surface geochemistry to delineate gold-bearing corridors and targets.
- Rotary air blast (RAB) drilling of prospective structures where extensive transported materials render surface sampling of low effectiveness.

Based on the compiled data and the knowledge base of the exploration team, targets are prioritised by best chance of hosting economic mineralisation that meets the main objective of increasing the Mineral Resource inventory within the permit areas.

The acquisition of both the Douta West and Bousankhoba permits has allowed for a regional-scale exploration strategy that is largely underpinned by a comprehensive geochemical database (Figure 9.1). It is evident that, particularly in the southern regions, there are numerous geochemical targets of which only a few have been drill tested (Figure 9.2).

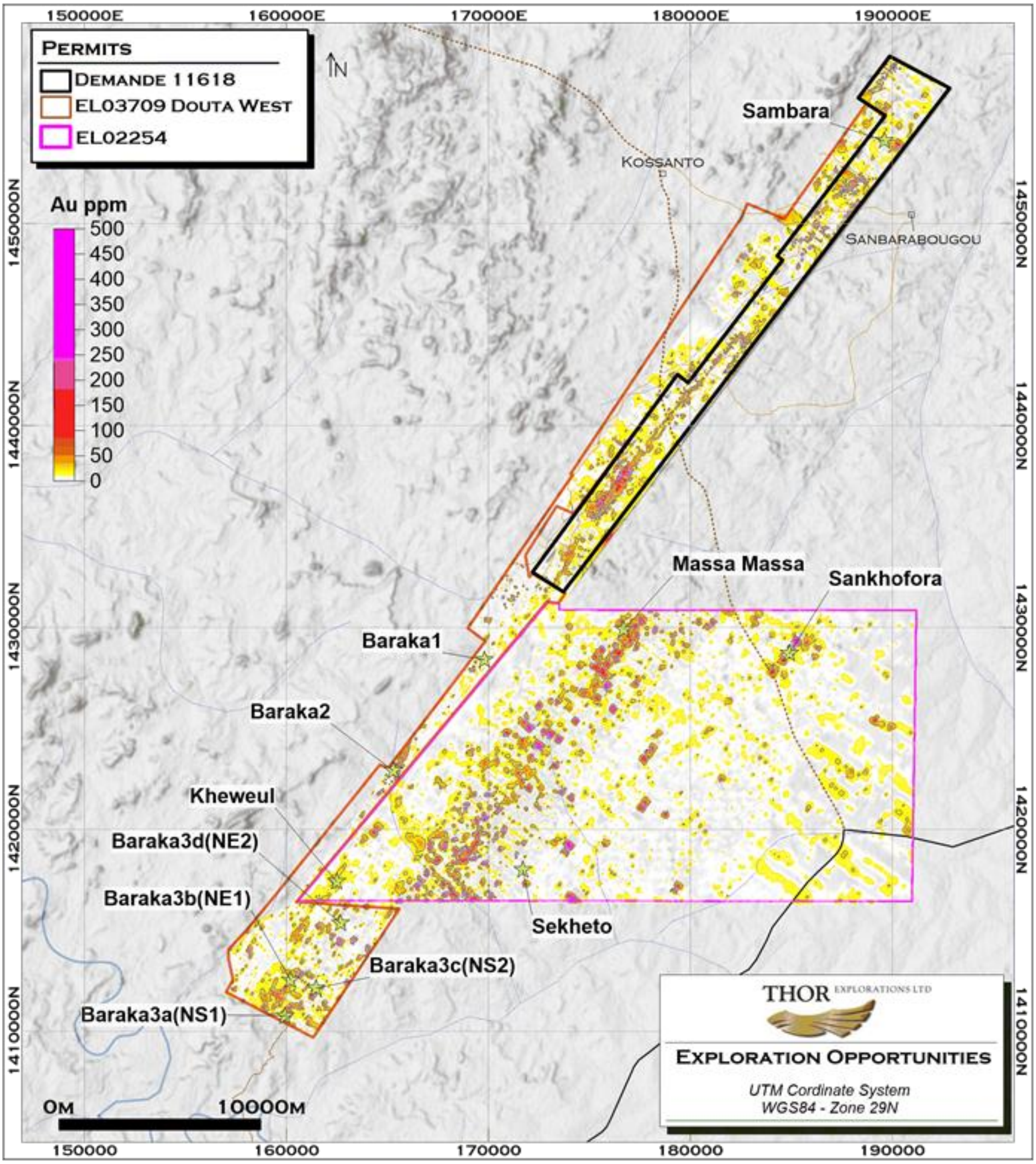
Extensive first stage exploration, completed over the Bousankhoba permit, has identified numerous geochemical targets associated with a major shear zone over an 18 km strike length. Two prospects, Massa Massa and Sekhoto, are located along this zone. An additional prospect known as the Sakhofara is located 7 km east of Massa Massa in the northern part of the permit and is defined by a 3.5 km north-east trending geochemical anomaly.

To date follow-up RAB drilling has been confined to the Sekhoto Prospect. The significant intersections obtained in the RAB drilling are yet to followed with systematic RC drilling.

The prospectivity of the permit is further enhanced by its location between Baraka 3 to the south and Basari Resource’s 1M ounce Makabingui deposit to the north. It is possible that the mineralised shear zone extends through all three gold occurrences.

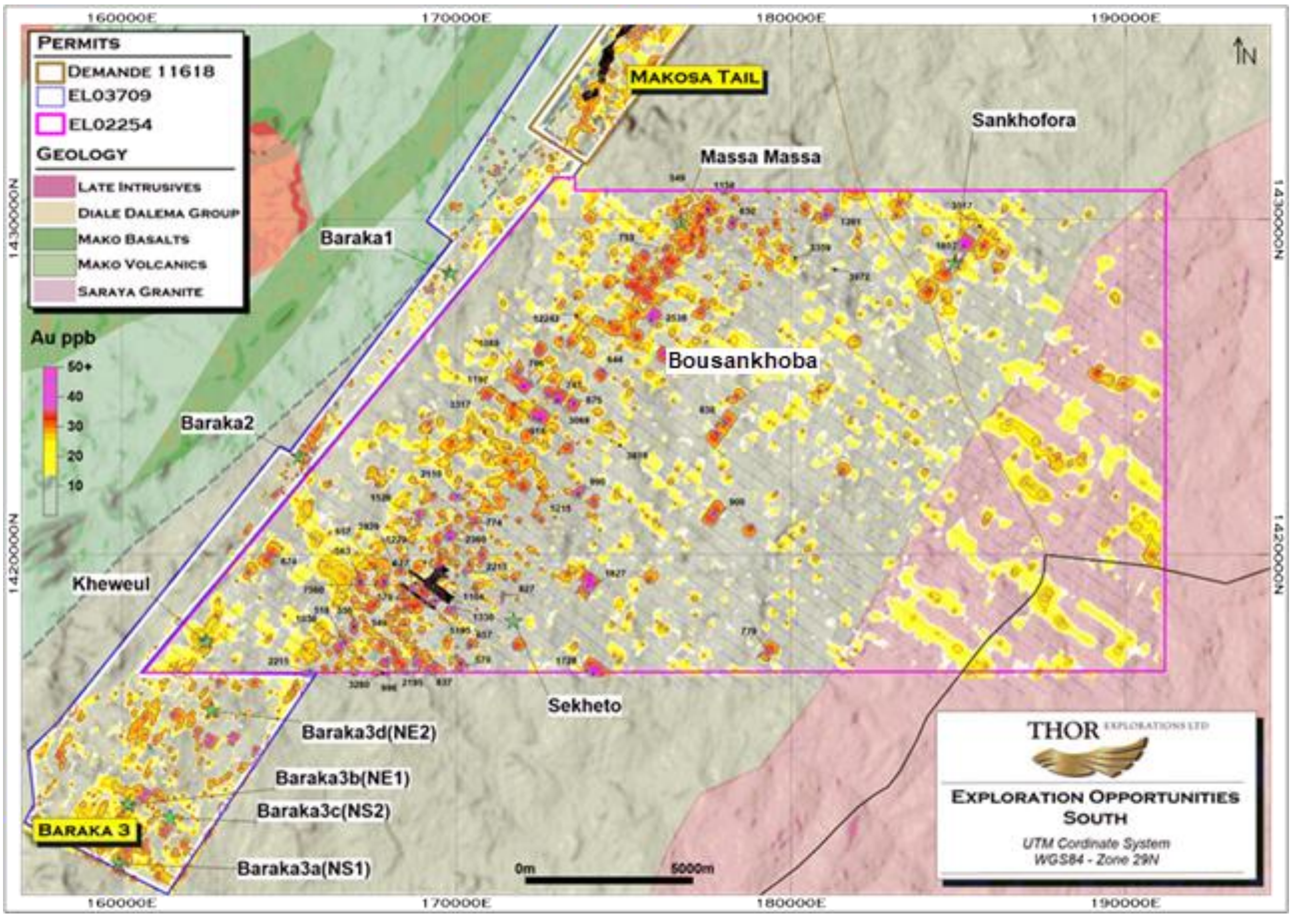
Thor considers both the Bousankhoba and Douta West permits to highly prospective with the potential to provide satellite resources that will complement the Douta Project.

Figure 9.1 Gold in soil geochemistry



Source: Thor, 2025.

Figure 9.2 Gold in soil geochemistry in the Baraka 3 and Bousankhoba Areas



Source: Thor, 2025.

9.2 Geophysics

9.2.1 Ground magnetic survey

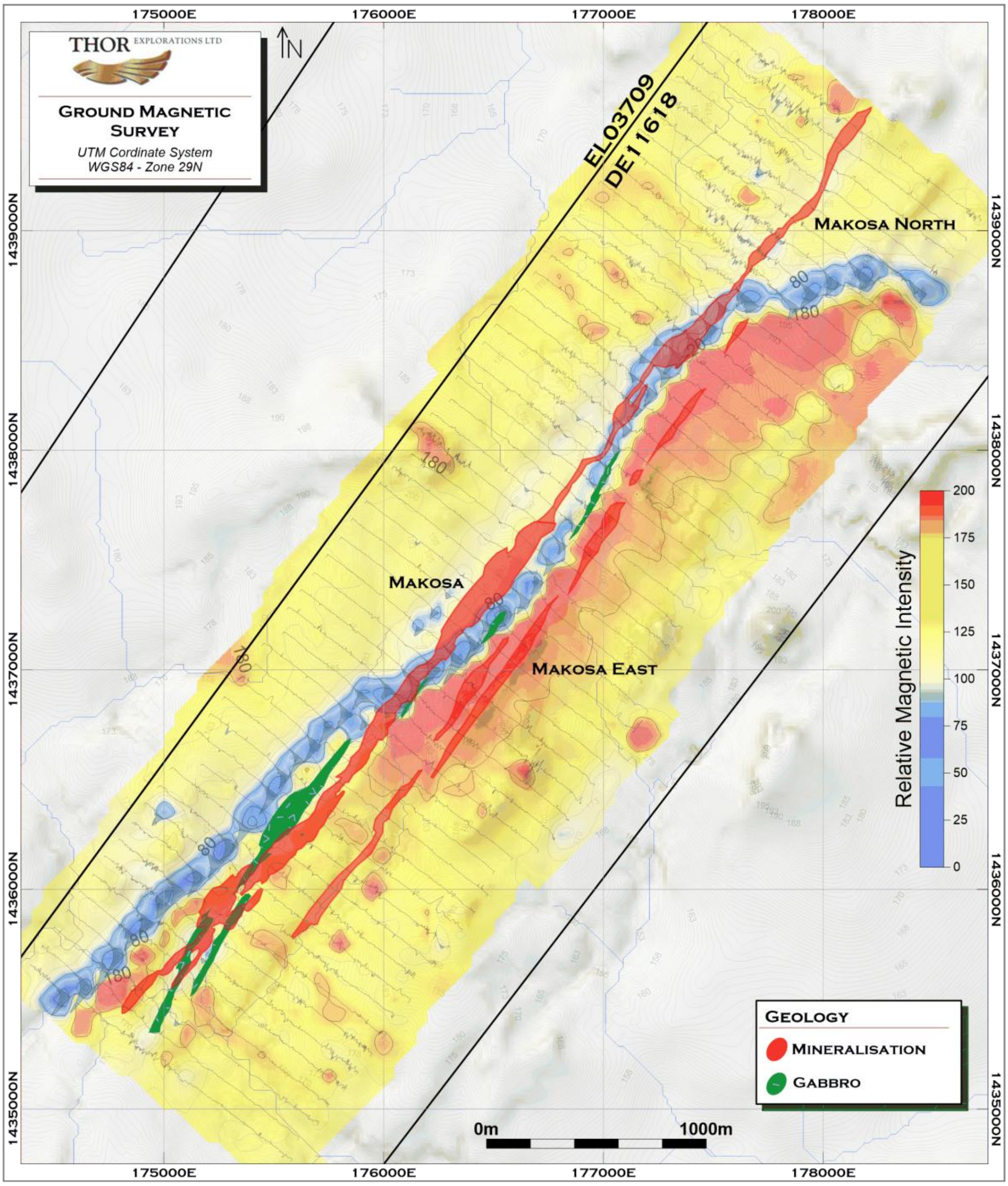
A ground magnetometer survey was carried out through the Makosa Prospect by ASR in July 2011.

Magnetic measurements were made on 1,500 m long by 200 m lines using a Scintrex G858 Caesium Vapor Magnetometer.

The measurements were continuously taken along the lines, with the magnetic sensor's location determined by a system connected to the global positioning system (GPS). The magnetometer cycles are done every second to give a measurement interval of 1 to 2 m along the profile, depending on the progress through the bush. Daytime magnetic field variations were measured using a magnetometer at intervals of 100 seconds.

The interpretation of the magnetic measurements in the Makosa Prospect allowed the identification of potential structures and mafic intrusions (Figure 9.3). In particular, the strong north-easterly trending magnetic high is coincident with the gabbro that is developed in the Makosa Tail and southern Makosa area.

Figure 9.3 Ground magnetics survey area



Source: Thor, 2025.

9.3 Geochemical surveys

9.3.1 Trenching

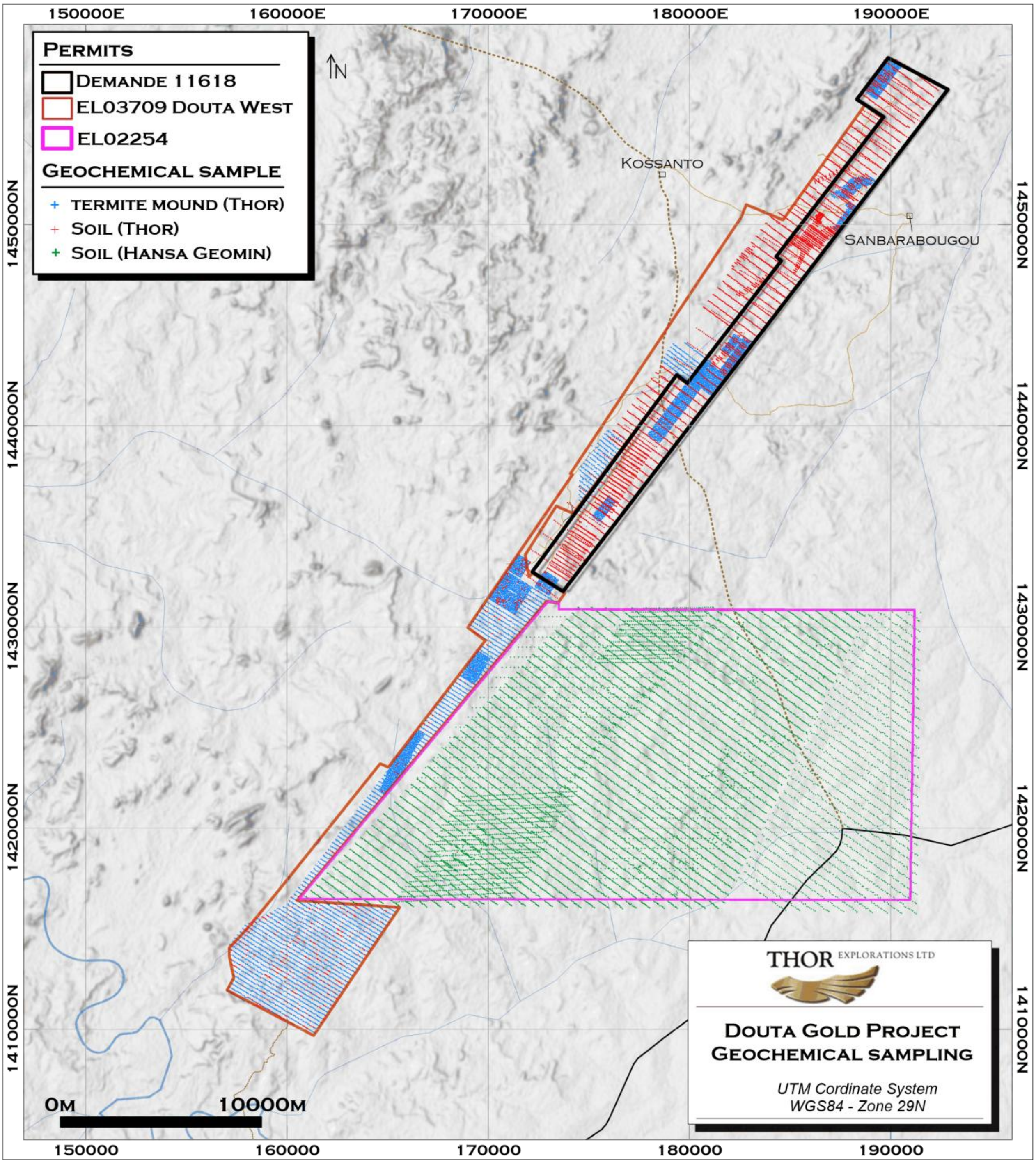
Eight trenches with a total length of 3,040 m were excavated by hand on the Makosa Prospect in 2011 to test geochemical anomalies. The results of the trench campaign confirmed that the gold anomaly from soil geochemistry is in situ and comes from the underlying saprolites and justified follow up drill testing.

9.3.2 Auger-assisted geochemical surveys and termite mound sampling

To date, a total of 18,483 surface geochemical samples has been obtained from within the Douta and Douta West permits (Figure 9.4). Most of this sampling has been either termite mounds (11,693) or surface soils (6,790) on a 200 m x 50 m grid. The areas in which the different sampling techniques were used are generally discrete so that each dataset i.e., termite mound or soil data, was interpreted separately. In the four areas that overlapping sampling methods occurred no bias between the two datasets was detected.

This survey was designed to better define gold anomalism that had been delineated during the historic surface-soil geochemical programs. The results of the auger program enable targeted reverse circulation (RC) drill testing of areas to the north and south of the Makosa deposit.

Figure 9.4 Geochemical survey map of the Douta Project and associated permits showing individual sample points



Source: Thor, October 2025.

10 Drilling

10.1 Summary

Exploration drilling at the Property has been a combination of diamond drill core (DD), RC, and RAB. Totals of 60 DD holes for 7,425 m, 1,699 RC holes for 119,718 m, and 184 RAB holes for 7,942 m have been drilled on the Douta Project permits (Table 10.1 and Table 10.2).

Table 10.1 Drilling statistics summarised by prospect

Deposit / prospect	Category	No. holes			No. metres			Total No.	Total metres
		RAB	RC	DD	RAB	RC	DD		
Makosa East	Resource		187			10,567		187	10,567
Makosa North	Resource		217	1		15,206	246	218	15,452
Makosa Tail	Resource		225	20		17,544	2,252	245	19,796
Baraka 3	Resource		329	3		20,409	438	332	20,847
Total	**Resource**		**958**	**24**		**63,726**	**2,936**	**982**	**66,662**
Maka	Exploration		92			7,254		92	7,254
Makosa	Exploration	184	314	36	7,942	29,142	4,489	534	41,573
Mansa	Exploration		86			5,214		86	5,214
Sambara	Exploration		88			5,604		88	5,604
Sambara NW	Exploration		38			2,520		38	2,520
Baraka 1	Exploration		34			1,804		34	1,804
Baraka 2	Exploration		89			4,454		89	4,454
Total	**Exploration**	**184**	**741**	**36**	**7,942**	**55,992**	**4,489**	**961**	**68,423**
Total Project		**184**	**1,699**	**60**	**7,942**	**119,718**	**7,425**	**1,943**	**135,085**

Source: Thor, 2025.

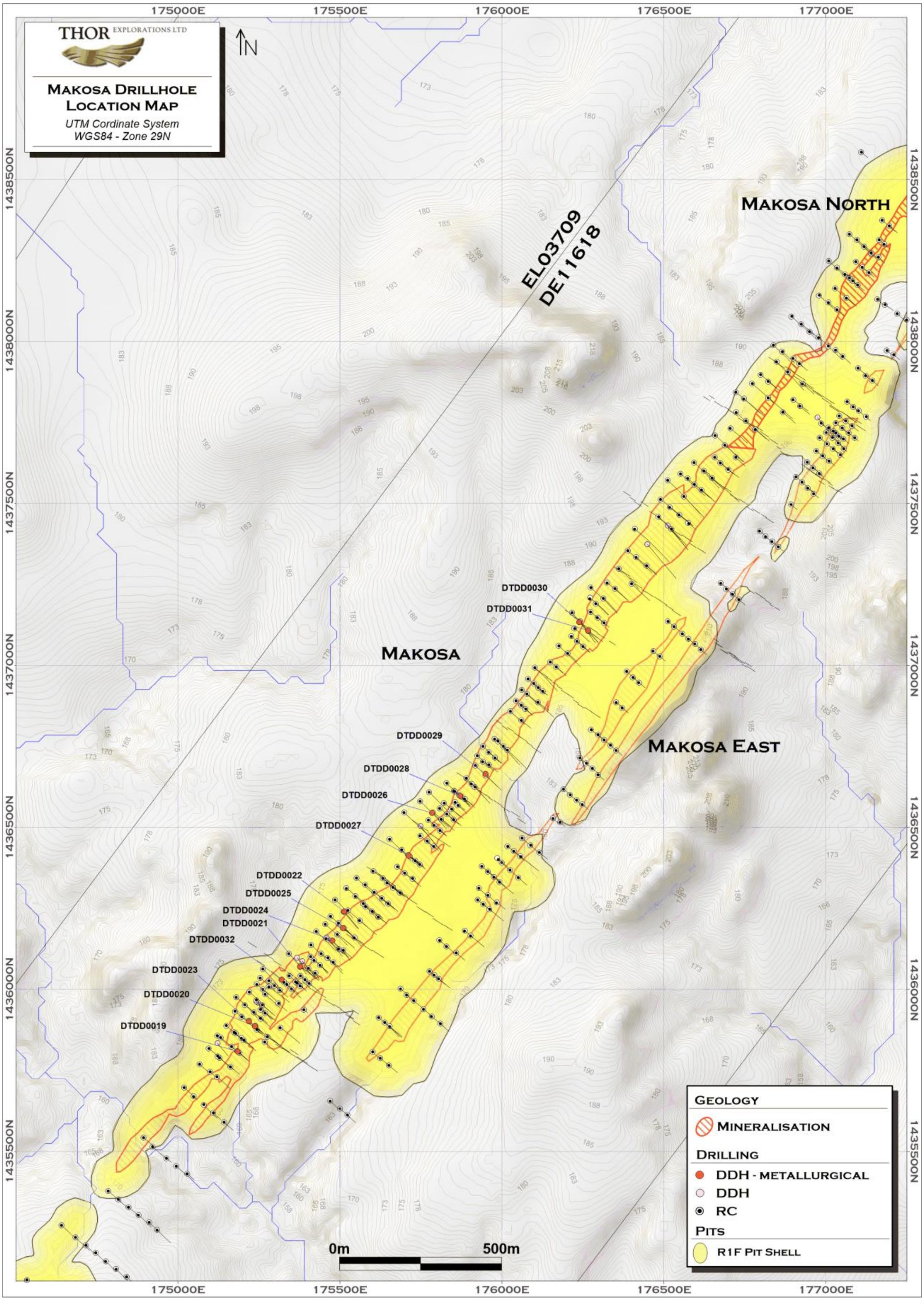
Table 10.2 Drilling statistics summarised by year

Year	RAB		RC		DD		No. holes	No. metres
	No. holes	No. metres	No. holes	No. metres	No. holes	No. metres		
2011					2	405	2	405
2012	184	7,942			13	1,532	197	9,474
2017			24	2,052			24	2,052
2018			72	9,014			72	9,014
2020			132	9,462			132	9,462
2021			215	15,204			215	15,204
2022			356	26,898			356	26,898
2023			181	15,153	37	3,891	218	19,044
2024			259	13,300	2	546	261	13,846
2025			460	28,635	6	1,050	466	29,685
Total	**184**	**7,942**	**1,699**	**119,718**	**60**	**7,425**	**1,943**	**135,085**

Source: Thor, 2025.

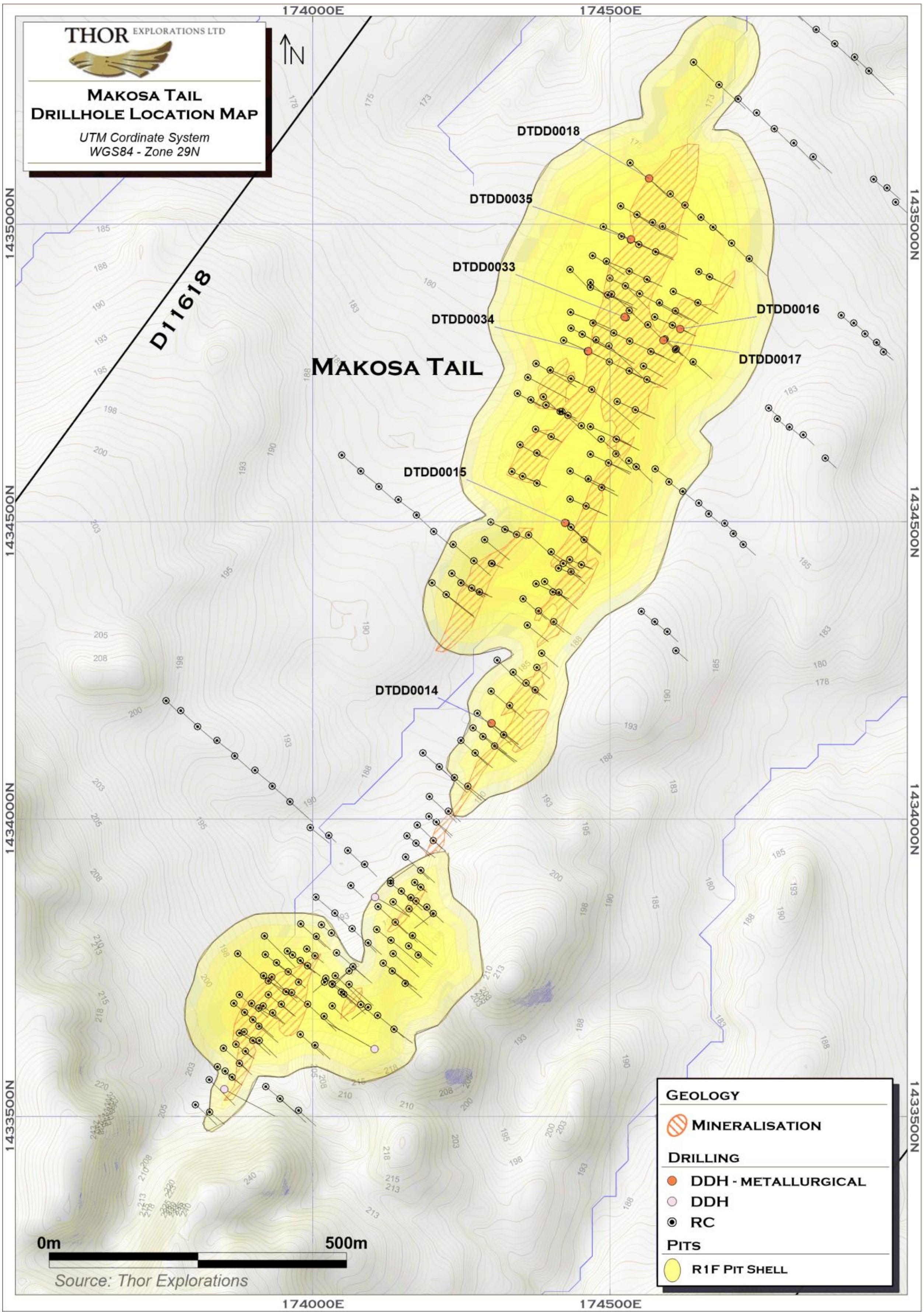
The distribution of drilling over the main resource areas is shown in Figure 10.1 and Figure 10.2.

Figure 10.1 Makosa drillhole location map



Source: Thor, October 2025.

Figure 10.2 Makosa Tail drillhole location map



Source: Thor, October 2025.

The Project uses the UTM Zone 29N datum WGS84 coordinate system. The boundary between Zone 28 and Zone 29 passes between Makosa and Makosa Tail. Survey data from Makosa Tail and Baraka 3, both located in UTM Zone 28, are converted into the same grid system as Makosa and the rest of the Project (UTM Zone 29N, datum WGS84).

10.2 Drilling methods

Thor has established and followed standard operating procedures for RAB, RC, and DD. Drillhole collars are surveyed using a Leica system 1200 differential GPS. Prior to March 2018, downhole surveys were conducted using either a Reflex EZ-Trac or Reflex Act2 Single Shot camera. Since March 2018, all drillholes have been surveyed with a Reflex EZ-Gyro, with measurements taken at 10 m intervals on both entry and exit of the hole. Only on-entry data are used.

10.3 Diamond drilling

Diamond drilling was used for RC-twinning, metallurgical sampling and geotechnical purposes. Wireline drilling was carried out using exclusively HQ diameter diamond bits and represents about 5% of the total number of drilling metres in the resource area with the remainder comprising RC and RAB drilling.

10.3.1 Diamond core sampling procedure

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 m and returned to 1 m intervals as soon as geologically sound.

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.

Thor conducted half-core sampling on HQ core.

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.

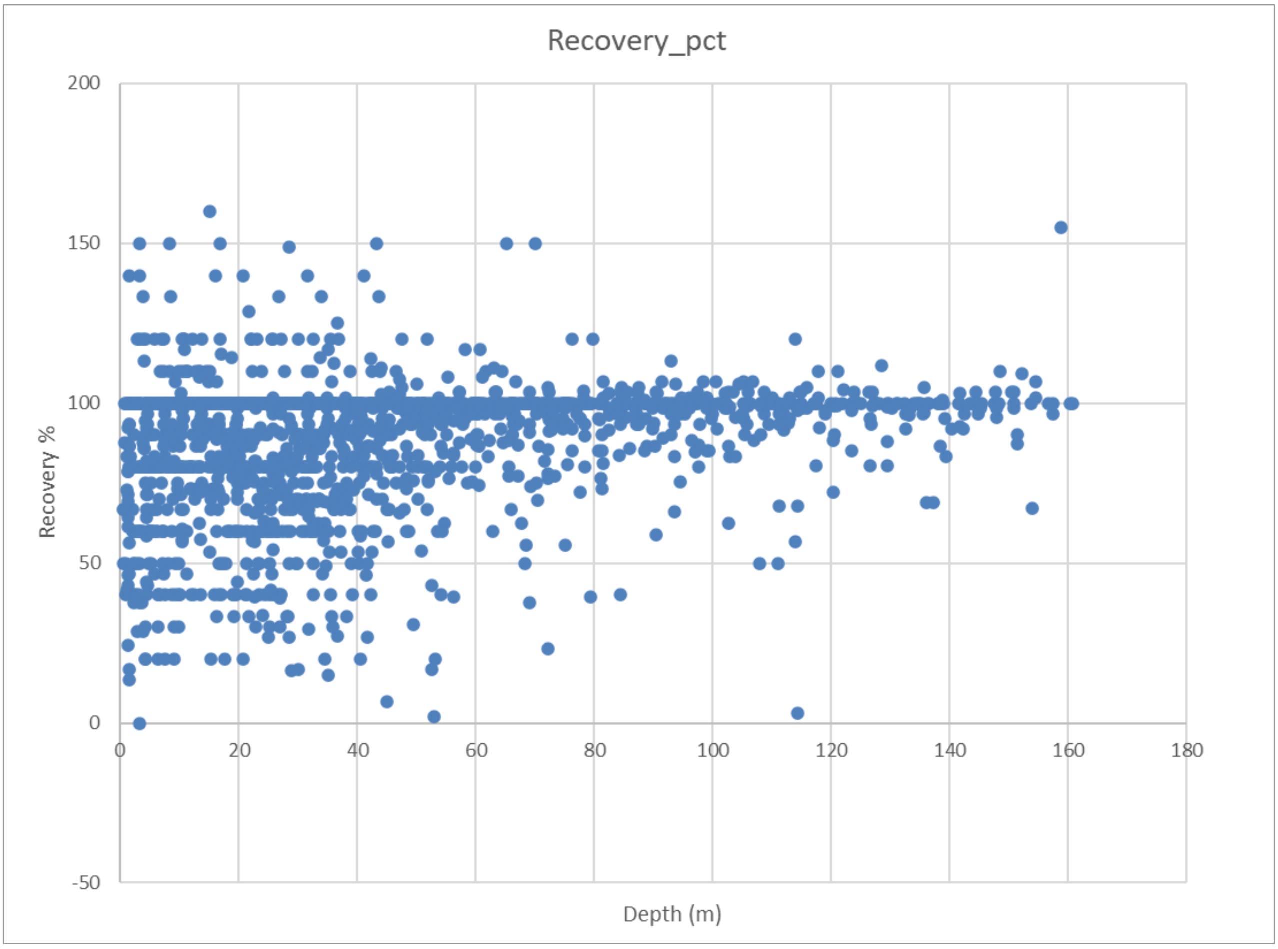
Before the core was cut, it was turned to ensure that the geological boundaries were cut at the optimum angle. The same orientation used was used throughout the program and remained consistent throughout each drillhole. The core was then cut down the orientation line.

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

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

A total of 1,903 core recovery calculations were carried out with an overall average of 88%. Figure shows the generally improving recoveries with depth reflecting the increasing rock competency as opposed to recoveries from the moderate to highly weathered saprolite and highly fractured and brecciated zones returned poor recoveries.

Figure 10.3 Diamond drilling core recovery



Source: Thor, 2025.

10.3.2 Diamond core logging

Core recovery is measured in the field and during detailed logging; core loss is marked out clearly. The DD core was oriented using ACTx. In cases where orientation is not possible, the core is assembled with previous runs to extend the orientation line from previous runs of orientated core, such that structural directions in the form of alpha and beta angles are documented. Logging for DD were completed on hard copy before being transcribed to the database.

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.
- Recovery and Rock Quality Designation (RQD) – 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 metadata such as hole ID, declination, azimuth, hole depth, core diameter, date, and water ingress were also recorded.

All core was photographed both wet and dry before being marked and cut for assaying.

10.4 Reverse circulation drilling

RC drilling is used at Douta for both exploratory and resource definition drilling. RC drillholes have been drilled at a 5.5 in / 14 cm diameter which produces approximately 20 kg of material per one-metre sample interval. The RC drill rods are 6 m in length.

The first RC drilling at Makosa was completed by IDC using KL900 rig in July 2017. The subsequent drilling was completed by Sendrill and Sengold drilling contractors using a Thor 5000 rig and booster combination. RC holes were drilled using either 16 cm or 17.4 cm diameter rods with a 5.5” face-sampling bit size.

10.4.1 Reverse circulation drilling procedures

A geologist is present during RC drilling operations. Prior to commencement of any drilling, cross sections for all planned drillholes are printed, indicating the expected geology and mineralisation to be intercepted. The geologist lays out the site to ensure that, where possible, the drilling and sampling operations will not interfere with each other.

Polyvinyl chloride (PVC) casing is used to collar the drillhole which helps prevent drillhole collapse and sample contamination. If the drillhole intersects the water table, an auxiliary booster is utilised to ensure that the samples are dry. After each rod change, air is blown down the hole to dry it out before drilling recommences. If a drillhole collapses whilst drilling, the drillhole is stopped and abandoned due to the risk of contamination.

10.4.2 Reverse circulation sampling procedures

RC chips are collected from the rig in standard one-metre intervals through a cyclone. RC samples are not composited at this stage, allowing for better definition and increased understanding of gold variability. On the rare occasion a wet sample is obtained, it is dried before being manually split. Auxiliary booster units were used as necessary to ensure that most of the samples collected were dry.

Gilson single-stage splitters are used to produce an average split weight of 2.3 kg after 3 or 4 stages of splitting.

The remaining sample was collected and stored in large plastic bags either at the drill site or laydown facility. To monitor the representativeness of the split samples, a field duplicate was taken every 20th sample. Results from the field duplicate samples illustrate there is no apparent bias. This is considered appropriate for the style of mineralisation and size of the RC sample over each 1 m interval.

10.4.3 Reverse circulation logging

All RC samples are logged on one-metre intervals as per the sample length received from the RC rig. Logging is completed on paper logs which were transferred to an Access database later. In 2025 the MaxGeo DataShed5 system was implemented. For each drillhole, the following are logged as a minimum: lithology, visible mineralisation, vein intensity, alteration, oxidisation, and depth of water table.

10.5 Rotary air blast drilling

RAB drilling is used for reconnaissance exploration drilling programs. RAB drillholes are typically drilled to blade refusal, which in most cases coincides with the top of the unoxidised bedrock. The maximum, practical drill depth for most rigs is around 60 to 80 m where the oxidation and overburden profile is very well developed. Drillholes are angled 60° to 70° to the surface. Collar survey data are recorded using handheld GPS units. No downhole surveys are performed.

Cuttings are recovered via a cyclone that is attached by a drill pipe to the top of the sealed drillhole collar. Unlike RC cuttings, the cuttings from a RAB hole are exposed to the wall rock as they ascend to the collar of the drillhole for collection.

Samples submitted for analysis are a composite of two individual one-metre samples. The 1 m sample is taken via a pipe inserted into the cuttings pile in two passes, forming a cross pattern. The composite weighs approximately 2.5 kg. The RAB cuttings are left in one metre piles on the ground near the drillhole collar. The subsample composite is collected in a clear plastic bag with the top folded twice and stapled over the fold with a paper sample-number tag inserted in the fold.

RAB drilling was used in the early stages of the project and was successful in identifying the Makosa mineralised structure. All subsequent drilling was carried out with either RC or diamond drilling. RAB drilling has not been used as part of the Mineral Resource estimate.

11 Sample preparation, analyses, and security

11.1 Introduction

Thor has established industry-standard operating procedures at Douta for sample preparation, analyses, security, and Quality Assurance and Quality Control (QAQC). The methodology for sample preparation and the analyses carried out are documented and regularly reviewed.

The QAQC procedures have been prepared to provide a specific system for limiting technical risk on the project by ensuring best practice at drilling stage, thereby ensuring maximum quality control on sample and assay data.

Responsibilities:

- Geology Managers: Review, approve, and support the operating procedure.
- Project Geologists: Support the procedure within their respective sections.
- Exploration Geologists: Implement, enforce, supervise, and communicate the procedure to the sampling team.
- Geology Technicians and Sampler Technicians: Compliance with the procedure.

The Mineral Resource estimate presented herein is based solely on RC and DD sampling data. No samples from RAB drilling are used for Mineral Resource estimation.

The data and sampling techniques are audited internally by the Company's QP. The QP believes that the sample collection, preparation, analysis, and security used at the Douta Gold Project performed in accordance with exploration best practices and industry standards and are suitable for use in Mineral Resource estimation.

11.2 Sample preparation

The Exploration Geologist is responsible for all sampling activities conducted by geological technicians and samplers, including sampling, sample bagging, numbering and tagging, sorting, transportation, security, completion of the analytical submission sheets, and QAQC program. The Project Geologist is responsible for the overall drilling and sampling programs.

The first stage on sample preparation is carried out on site. Core samples are cut in half with a diamond saw and broken into smaller rock pieces for packaging. The RC sampling is done systematically for each metre from the top to the end of the drillhole.

RC samples are split using single-stage Gibson splitters. A sample between 1.5 and 3 kg is collected for lab Assay and sample reference for further analysis. A duplicate QAQC sample is collected every 20 samples, and standards and blanks are inserted before sending samples to the Lab for gold analysis.

All samples are placed into sample bags with assigned sample numbers, then closed, sealed, and inserted into larger rice bags that are securely sealed. Samples that are sent for assay to the ALS Chemex laboratory in Bamako, Mali, are securely transported by the company / Lab trucks.

Laboratory sample preparation includes weighing and drying, and crushing to 75% passing 2 mm. A 250 g or 1,000 g split (for fire assay and bulk respectively) is then pulverised to 85% passing 75 µm.

11.3 Sample analysis

Thor used ALS Chemex (ALS) in Bamako, Mali, as its primary fire assay laboratory for gold and multi-element analysis. ALS is an ISO9001:2008 accredited laboratory. The QP has not audited the sample preparation for assaying laboratory in Mali. The laboratory is independent of Thor.

Thor samples were analysed by fire assay with an atomic absorption spectroscopy (AAS) finish (Au-AA26). 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 diluted hydrochloric acid. The final solution was analysed by AAS.

Since 2022, ALS opened a sample preparation facility in Kedougou. This facility crushes and pulverises the samples in order to send only the pulp to ALS Bamako or ALS Ouagadougou. Thor's senior geologists visited the Ketougou facility in 2022 and in 2023 to review the laboratory practices.

SGS laboratory, located in Bamako, Mali is used for umpire sample gold analysis with Fire Assay method. ALS Chemex in Perth, Australia is used for metallurgy test work.

11.4 Sample security

Samples are under the supervision of Thor staff from collection at rig, to processing at the site exploration camp, to delivery onto the laboratory transport.

Samples are bagged, sealed, and numbered and delivered to ALS Chemex in Bamako, Mali or since 2022 ALS Chemex, Kedougou. A hard copy of the sample submission form is sent with the samples and a digital copy along with a list of samples included in the submission is emailed to the laboratory.

RC samples are bagged and tied with custom tags before being weighed and documented. Samples are weighed and documented at the rig. The samples are stored in a secure facility at the Louba exploration camp until dispatched. DD samples are stored in core boxes (with the appropriate numbering and markings) at the core shed area.

Returned pulp samples are stored (under clean and dry conditions to avoid contamination) in the core yard area with dedicated space storage. Disposal of pulp sample varies from project to project. Thor generally store pulp samples until the area is mined out.

Samples at ALS are also kept in a secure samples yard. The laboratory discards samples after three months unless otherwise directed.

As samples are analysed at ALS Chemex laboratory in Bamako, Mali, Thor has put in place prompt, secure and direct shipping of samples to these laboratories, including a logistic partner, who transports the samples to Kedougou.

11.5 Quality Assurance and Quality Control

11.5.1 Douta 2024-2025

11.5.1.1 Summary

During this period, 76 batches from ALS Chemex laboratory in Bamako, Mali were reviewed, with analysis for gold (Au) (see Table 11.1 to Table 11.4). Assay information is downloaded to the DataShed 5 platform. The QAQC data were audited and reported by MaxGeo.

- No issues were identified for the laboratory's own inserted Blanks.
- There are a large number of field-inserted Certified Reference Material or Standards (CRM) that appear to have been mislabelled with the incorrect CRM ID. In light of this, Thor has reviewed its field sampling workflow and has implemented procedures to improve it. Thor will utilise the DataShed 5 QAQC Dashboard, which will assist in identifying standard swaps quickly.

A total of 685 field duplicates and 447 laboratory pulp checks have been reported.

Both field duplicate and laboratory pulp checks highlighted no major issues (MaxGeo, 2025).

Table 11.1 Laboratory summary

Laboratories	ALS_SENEGAL
No. of Batches	76
No. of DH Samples	11,620
No. of QC Samples	1,254
No. of CRM / Standard Samples	1,760

Source: Thor, 2025.

Table 11.2 QC category ratios

QC_Category	DH sample count	QC sample count	Ratio of QC samples to DH samples
Field duplicate	11,620	582	1:17
Lab Pulp Checks	11,620	569	1:20

Source: Thor, 2025.

Table 11.3 Standard type ratios

Standard type	DH sample count	Standard type count	Standard sample count	Ratio of QC standard to DH samples
BLANK	**11,620**	**1**	**344**	**1:34**
CRM	11,620	9	1,384	1:8
CRM_LAB	11,620	3	32	1:368

Source: Thor, 2025.

Table 11.4 Gold standards submitted with original assays

Au standard(s)					Calculated values			
Std code	**Method**	**Exp method**	**Exp value**	**Exp SD**	**Mean Au**	**SD**	**CV**	**Mean bias**
BLK	**FAOG_AAS**	**FAOG_AAS**	**0**	**-**	**0.0028**	**0.0102**	**0**	**0.00%**
OREAS_235b	FAOG_AAS	FAOG_AAS	1.63	0.053	1.6366	0.0751	0.0459	0.40%
OREAS_241	FAOG_AAS	FAOG_AAS	6.91	0.309	7.0851	0.3628	0.0512	2.53%
OREAS_253	FAOG_AAS	FAOG_AAS	1.22	0.044	1.1938	0.0345	0.0289	-2.15%
OREAS_253b	FAOG_AAS	FAOG_AAS	1.24	0.036	1.2122	0.0996	0.0822	-2.24%
OREAS_254b	FAOG_AAS	FAOG_AAS	2.53	0.061	2.4699	0.1211	0.0490	-2.38%
OREAS_254c	FAOG_AAS	FAOG_AAS	2.57	0.082	2.5252	0.0709	0.0281	-1.74%
OREAS_255c	FAOG_AAS	FAOG_AAS	4.08	0.087	4.0332	0.0931	0.0231	-1.15%
OREAS_256b	FAOG_AAS	FAOG_AAS	7.84	0.207	7.6346	0.1911	0.0250	-2.62%

Source: Thor, 2025.

11.5.1.2 Blanks performance

No issues observed. The majority of blank (+95%) results returned below the detection limit, as expected.

Certified Reference Material (CRM) Performance:

OREAS_235b

All but four results returned within three standard deviations. The outlying samples are likely mislabelled standards and are most likely to be OREAS_253b. After removing the outlier samples, no issues are observed (Figure 11.1 A).

OREAS_241

All but two results returned within three standard deviations (Figure 11.1 B). These are likely mislabelled standards and are most likely to be OREAS_256b.

OREAS_253

All results lie within three standard deviations, with a slight negative bias observed (Figure 11.1 C).

OREAS_253b

All but five results returned within three standard deviations (Figure 11.1 D). These are likely mislabelled standards. After removing the outlying samples, a very slight negative bias is observed.

OREAS_254b

All but two results returned within three standard deviations Figure 11.2 A). These are likely mislabelled standards and are most likely to be OREAS_235b. After removing the outlier samples, a negative bias is observed.

OREAS_254c

All results lie within three standard deviations, with a slight negative bias observed (Figure 11.2 B).

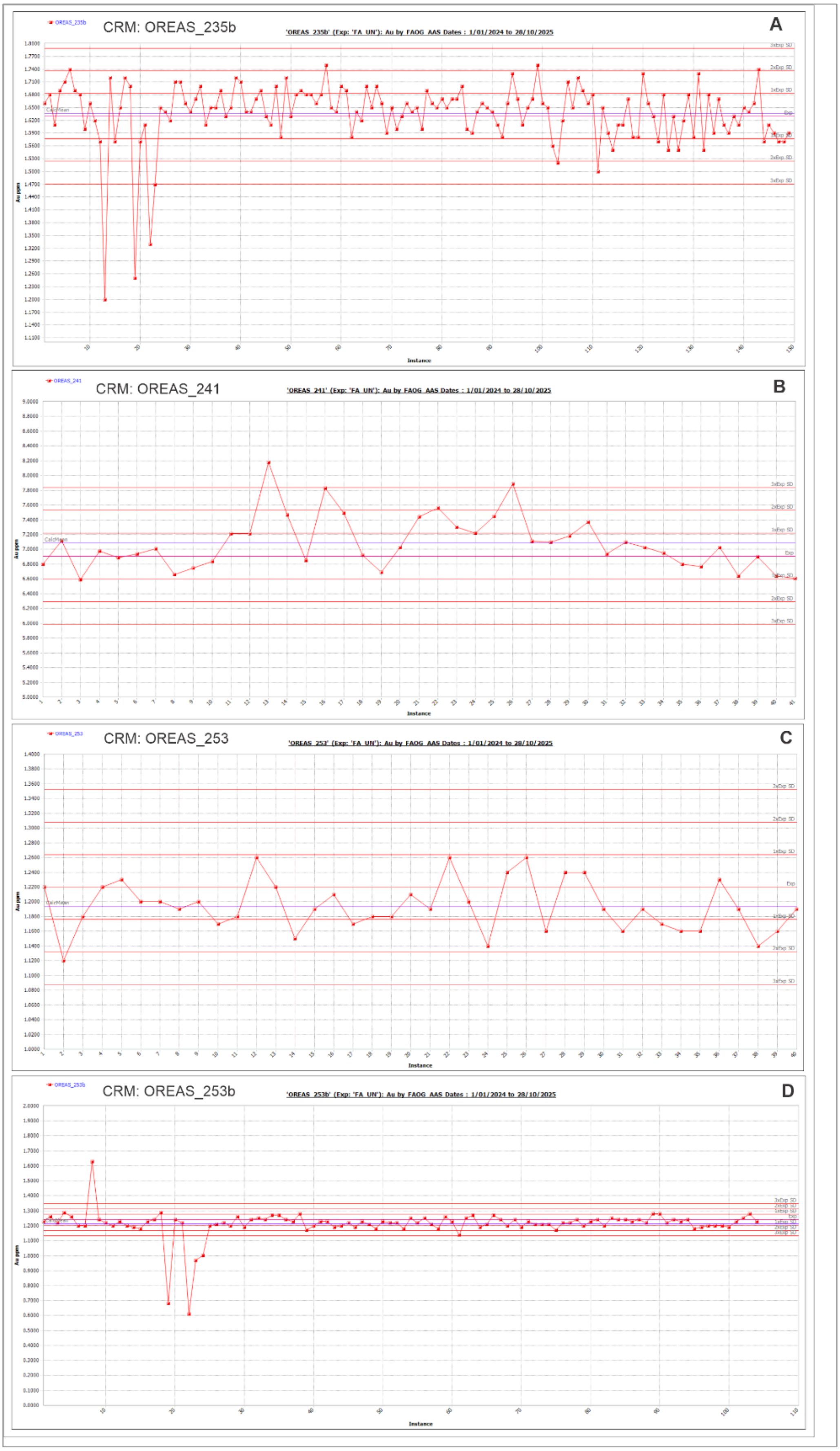
OREAS_255c

All but one result returned within three standard deviations (Figure 11.2 C). As the outlier is just above three standard deviations, it may not be a standard swap. Thor should review the affected batch.

OREAS_256b

All results lie within three standard deviations (Figure 11.2 D). A negative bias is observed, with most results falling below the expected value.

Figure 11.1 CRM performance charts 1



Source: MaxGeo, 2025.

Figure 11.2 CRM performance charts 2

Source: MaxGeo, 2025.

11.5.1.3 Drillhole physical original versus field duplicate repeats

A total of 582 field duplicate samples were analysed. The summary statistics (Table 11.5) indicate moderate repeatability at lower grades Figure 11.3 A).

Table 11.5 Drillhole physical original versus repeats: field duplicate repeats – all methods (matched methods)

Range (g/tAu)	No. samples	Mean Au1	Mean Au2	SD Au1	SD Au2	CV Au1	CV Au2	sRPHD (mean)
0.0 - 0.5	529	0.07	0.07	0.1	0.1	1.37	1.38	-0.21
0.5 – 2.5	48	1.06	1.05	0.47	0.49	0.45	0.46	0.45
>2.5	5	6.63	3.07	5.78	0.92	0.87	0.3	20.34
ALL	**582**	**0.21**	**0.18**	**0.86**	**0.43**	**4.17**	**2.41**	**0.02**

There are 529 repeat pairs at a grade range of 0 – 0.5 ppm Au. The repeats are moderate, with some scatter (Figure 11.3 B).

There are 48 repeat pairs at a grade range of 0.5 – 2.5 ppm Au. The repeats are moderate, with some scatter (Figure 11.3 C).

There are only five repeat pairs at a grade range of > 2.5 ppm Au (Figure 11.3 D).

11.5.1.4 Drillhole physical original versus laboratory pulp check repeats

A total of 447 the laboratory's own pulp check samples were analysed. The summary statistics, shown in Table 11.6 and Figure 11.4 A, indicate a generally good repeatability, with some scatter at very low grades.

Table 11.6 Scatter - drillhole (sample name): original vs lab pulp check repeats for Au ppm

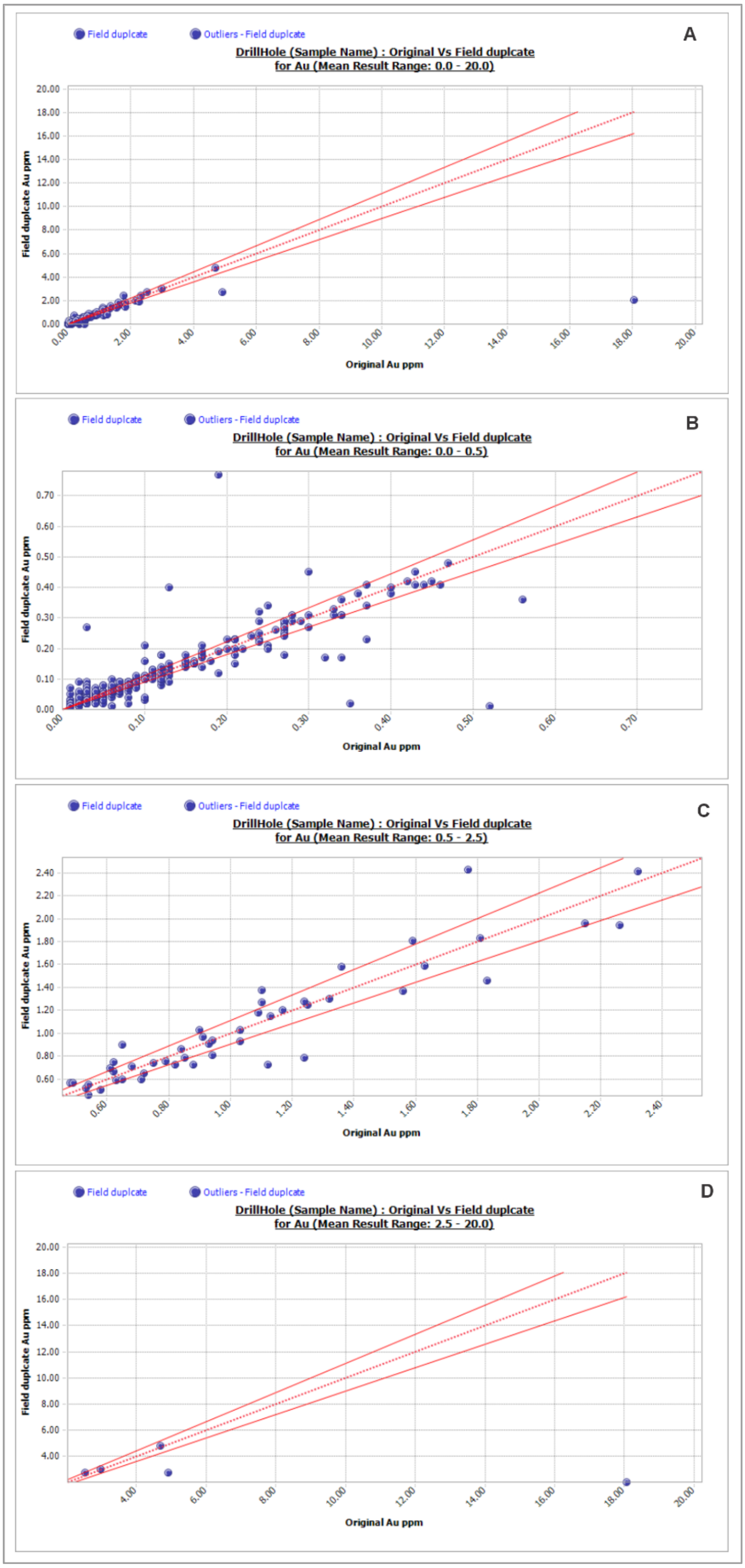
Range (ppm Au)	No. samples	Mean Au1 (ppm Au)	Mean Au2 (ppm Au)	SD Au1 (ppm Au)	SD Au2 (ppm Au)	CV Au1	CV Au2	sRPHD (mean)
0.0 - 0.5	406	0.07	0.06	0.09	0.09	1.39	1.41	2.06
0.5 – 2.5	36	1	1	0.41	0.41	0.41	0.41	0.07
>2.5	5	3.92	3.95	1.07	1.12	0.27	0.28	-0.15
ALL	**447**	**0.18**	**0.18**	**0.51**	**0.51**	**2.76**	**2.78**	**1.87**

There are 406 repeat pairs at a grade range of 0 – 0.5 ppm Au. The repeats are good, with some scatter at very low grades (Figure 11.4 B).

There are 36 repeat pairs at a grade range of 0.5 – 2.5 ppm Au. Repeatability is good in this grade range (Figure 11.4 C).

There are only 5 repeat pairs at a grade range of > 2.5 ppm Au (Figure 11.4 D). Repeatability is good.

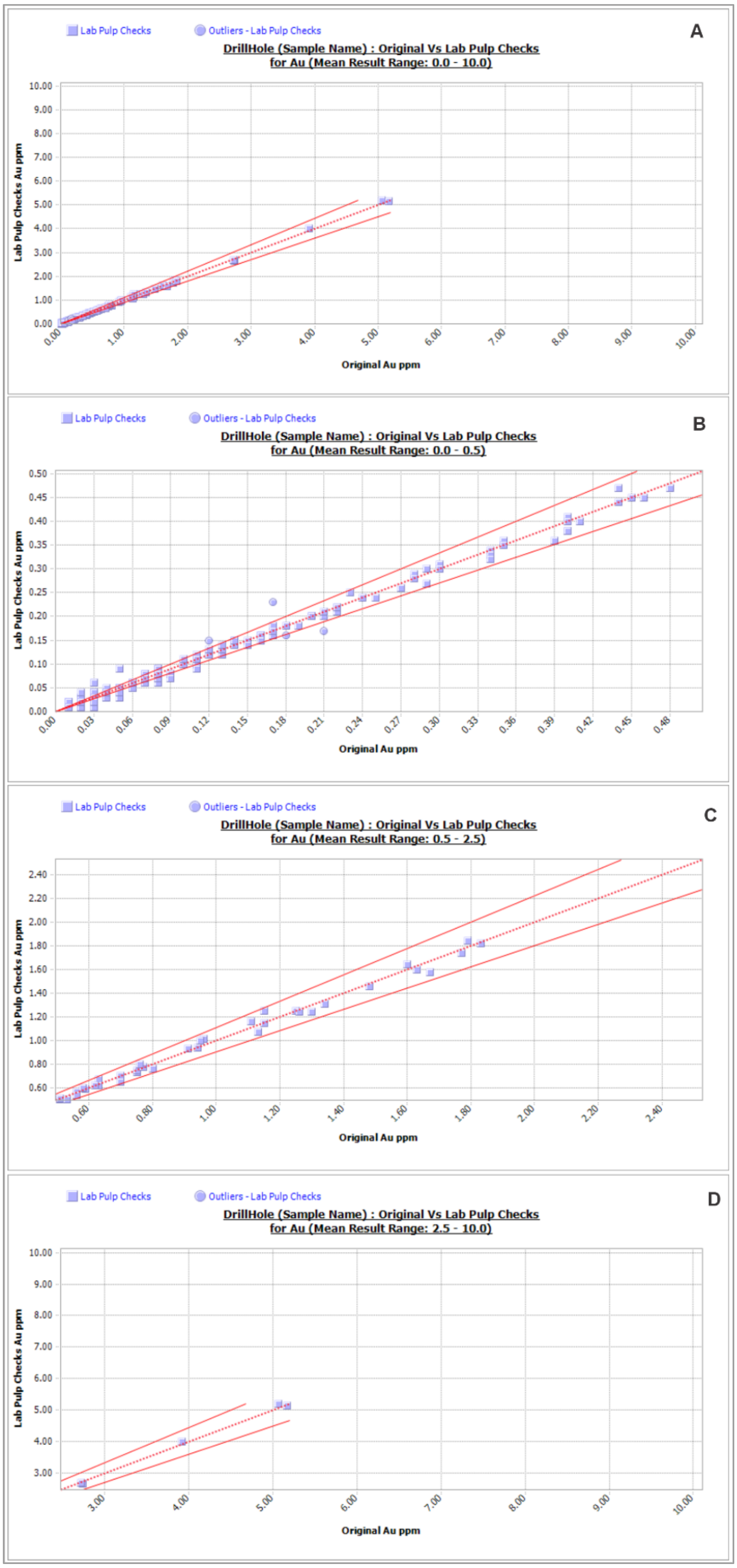
Figure 11.3 Drillhole physical original versus field duplicate repeats



Source: MaxGeo, 2025.

Figure 11.4 Drillhole physical original versus laboratory pulp

Check repeats



Source: MaxGeo, 2025.

11.5.2 Additional QAQC audits

In addition to the QAQC audit carried out on the Douta resource drilling for the period 2024-2025, MaxGeo carried out the following independent audits:

- Douta_QAQCReport_2022_2023
- DoutaWest_QAQCReport_2024_2025

Both studies concluded the following:

- Standard Reference Material:
  - No issues were identified for the inserted Blank samples.
  - Mislabelling of field-inserted CRMs is a concern. Numerous field-inserted standards appear to have been assigned an incorrect Standard ID.
  - Where Standard IDs appeared to be correct, no issues were identified. A small negative bias is observed through many of the standard.
- Repeats:
  - Both field duplicate and laboratory pulp checks highlighted no major issues.

11.6 Twinned holes

Fifteen RC intersections were twinned with diamond core drilling, typically within 3 m of the original RC drillholes. The comparative data (Table 11.7, Figure 11.5) show:

- Diamond core intersections display intercept lengths averaging 16% less than the respective RC intersections.
- Diamond core average intersection Au grades are 19% higher than the respective RC intersections.
- Diamond core total gram-metres for all fifteen intersections are 6% higher than the respective total RC total gram-metres.

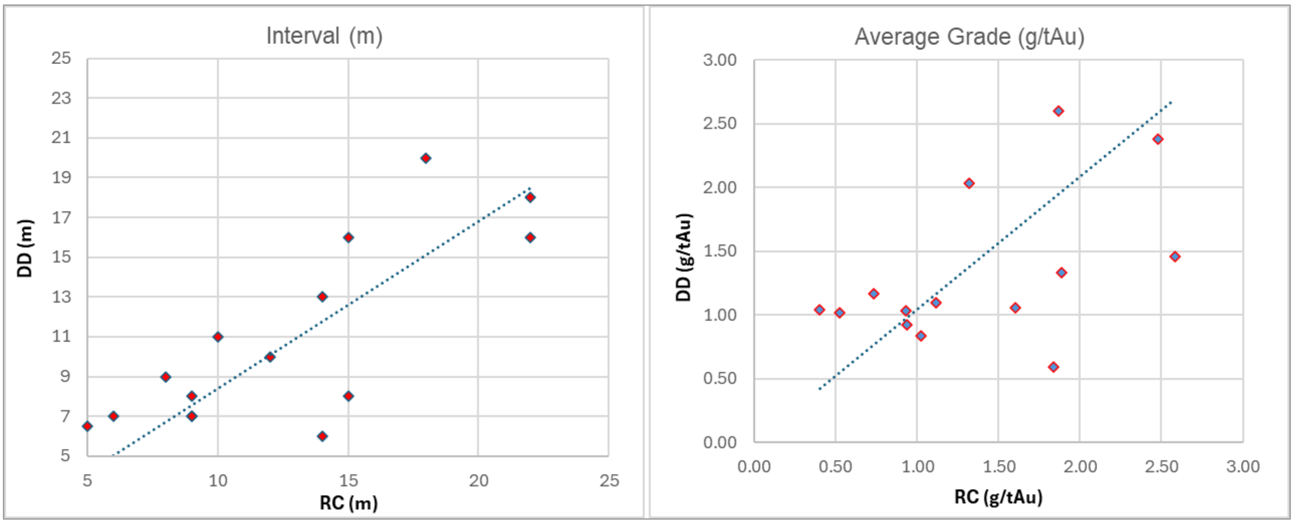
The variances reflect the more selective sampling procedures relating to diamond core. These data suggest there may be over-reporting of volume and tonnage, as the resource estimate is mostly based on RC drilling. With possible over-reporting of volume and tonnes, there is also a potential for under-reporting of grade. However, the reduced gram-metres in the RC data suggest a possible global under-reporting of contained gold.

Table 11.7 RC versus DD comparative data

Hole ID		Interval		Grade		Gram metres	
RC	**DD**	**RC (m)**	**DD (m)**	**RC (g/tAu)**	**DD (g/tAu)**	**RC GM**	**DD GM**
DTRC004	DTDD0021	22	16	1.96	4.95	43.1	79.2
DTRC004	DTDD0021	5	6.5	1.84	0.59	9.2	3.8
DTRC004	DTDD0021	9	7	0.40	1.04	3.6	7.3
DTRC011	DTDD0025	9	7	1.60	1.06	14.4	7.4
DTRC019	DTDD0026	14	6	0.73	1.17	10.3	7.0
DTRC019	DTDD0026	10	11	2.58	1.45	25.8	16.0
DTRC021	DTDD0027	9	8	0.52	1.02	4.7	8.1
DTRC021	DTDD0027	15	8	1.02	0.83	15.3	6.7
DTRC036	DTDD0029	15	16	1.32	2.03	19.8	32.5
DTRC046	DTDD0031	22	18	1.12	1.10	24.6	19.7
DTRC206	DTDD0015	14	13	1.88	1.33	26.4	17.3
DTRC234	DTDD0016	18	20	0.93	1.03	16.7	20.6
DTRC235	DTDD0017	8	9	2.48	2.38	19.8	21.4
DTRC271	DTDD0020	6	7	0.94	0.92	5.6	6.5
DTRC272	DTDD0023	12	10	1.87	2.60	22.4	26.0
Total		**188**	**162.5**			**261.8**	**279.6**
Weighted Average				1.39	1.72		
Variance			-25.5		0.33		-17.8
Variance %			-16%		19%		-6%

Source: Thor, 2025.

Figure 11.5 RC versus DD comparative data



Source: Thor, 2025.

11.7 General statement on sample preparation and analysis

In the QP's opinion, the sample preparation, analysis, and security procedures at the Project are adequate for use in the estimation of Mineral Resources. The QAQC programs, as designed and implemented by Thor, are consistent with industry standard practice, and the assay results in the database are suitable for use in a Mineral Resource estimate. The QP has not identified any issues that could materially affect the accuracy, reliability, or representativeness of the results.

12 Data verification

12.1 Introduction

The Douta Mineral Resource data are currently stored in a DataShed5 database. The QP has reviewed and verified the Douta database on an ongoing basis since 2018. Thor has used a set of protocols to ensure data integrity. The QP reviewed those procedures and results and conducted a series of checks to further verify the Douta resource data.

12.2 Verification work

Sampling, logging, and data entry have been carried out in the field by qualified geologists. Data is captured on paper logging sheets with the geologist later transcribing the data into a custom Microsoft Excel template before uploading to the DataShed5 platform that is administered by MaxGeo.

In addition to the inbuilt validation tools provided by DataShed5, Thor geologists further verify data using Leapfrog 2025.2.2.

Using Leapfrog 2025.2.2, the QP visually inspected the drillhole traces, performed basic database validation procedures, including the following.

- Intervals exceeding the total hole length.
- Inconsistent downhole survey records.
- Out-of-sequence and overlapping intervals.
- Invalid data formats and out-of-range values.
- Duplicate entries.

Drillhole traces were also reviewed in 3D, level plan, and in vertical sections and found no unreasonable geometries. In addition, standard data integrity checks were performed by various software programs on the drillhole database including:

Several issues such as incorrect azimuth and collar RLs were identified and corrected.

Approximately 100 assay values in the database were checked against the original assay certificates, and no errors were observed.

12.3 General statement

In the QP's opinion, based on the data verification procedures undertaken for the Project, the resource databases are considered to be adequate for the purpose of Mineral Resource estimation.

13 Mineral processing and metallurgical testing

13.1 Introduction

The orebody is comprised of three broad ore types representing oxide, transition, and sulphidic ores. Gold is distributed throughout the ore types with variable grades but is on average higher in the sulphidic ores. The oxide and transition ores are the result of weathering alteration and have been further classified as strongly oxidised (SOX), moderately oxidised (MOX), and weakly oxidised (WOX). SOX ores dominate the oxide ores progressing to lesser levels of oxidation through the transition ore types.

Gold is hosted in the general oxidised gangue lattice, in pyrite and arsenopyrite, and silicates.

SOX and MOX ores are amenable to whole ore cyanide leaching. WOX ores exhibit lower recoveries for cyanide leaching and the sulphidic ores exhibit poor whole ore cyanide leach recoveries. Alternate recovery processes have been investigated to improve gold recovery in the less oxidised and sulphidic ores.

The project will be developed in two phases. Phase 1 will process oxide and transition ores via a conventional CIL processing circuit. Phase 2 will add a suspension roasting circuit to treat sulphidic ores prior to CIL processing.

13.2 Phase 1 metallurgical test work

Test work reported here focusses on defining the response of the differing ore types to the Phase 1 CIL process. Test work has been undertaken by:

- ALS Limited, Perth, Australia in 2021 (ALS, 2021)
- Independent Metallurgical Operations Pty. Ltd. (IMO) in 2024 and 2025

13.2.1 ALS 2021

ALS completed work on 34 samples from Makosa and Makosa Tail prospects assessing amenability to cyanide leaching.

13.2.1.1 Samples

Samples were obtained from reverse circulation drillholes as in Table 13.1.

Table 13.1 Sample location and type

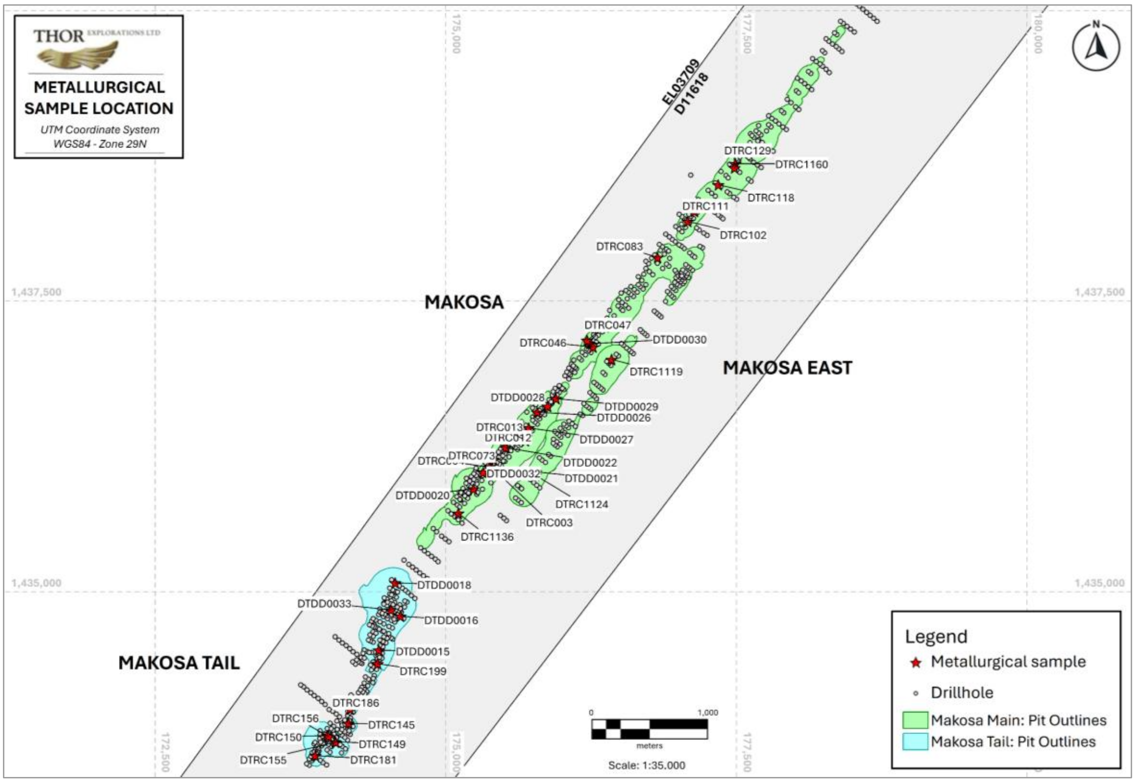
Sample ID	Head grade assay (g/t)	Hole-ID	From	To	Prospect	Zone	Lithology
DTMET0013	1.62	DTRC083	57	61	Makosa	Fresh	Graphitic shale
DTMET0014	2.62	DTRC102	33	37	Makosa	Oxide	Greywacke / felsic
DTMET0015	0.97	DTRC111	4	12	Makosa	Oxide	Greywacke
DTMET0016	1.25	DTRC118	14	20	Makosa	Oxide	Greywacke
DTMET0017	1.21	DTRC129	32	37	Makosa	Transitional	Greywacke / graphitic shale
DTMET0018	1.24	DTRC083	57	61	Makosa	Fresh	Graphitic shale
DTMET0019	2.03	DTRC102	33	37	Makosa	Oxide	Greywacke / felsic
DTMET0020	0.80	DTRC111	4	12	Makosa	Oxide	Greywacke
DTMET0021	0.98	DTRC118	14	20	Makosa	Oxide	Greywacke
DTMET0022	1.60	DTRC129	32	37	Makosa	Transitional	Greywacke / graphitic shale
DTMET0023	2.16	DTRC003	20	25	Makosa		
DTMET0024	1.63	DTRC003	25	32	Makosa		

Sample ID	Head grade assay (g/t)	Hole-ID	From	To	Prospect	Zone	Lithology
DTMET0025	0.83	DTRC004	37	42	Makosa		
DTMET0026	6.17	DTRC004	45	50	Makosa		
DTMET0027	1.99	DTRC004	50	56	Makosa		
DTMET0028	0.61	DTRC004	56	61	Makosa		
DTMET0029	1.75	DTRC004	61	66	Makosa	Fresh	Graphitic shale
DTMET0030	1.03	DTRC012	25	30	Makosa		
DTMET0031	1.34	DTRC012	30	35	Makosa	Fresh	Graphitic shale
DTMET0032	2.07	DTRC013	62	64	Makosa	Fresh	Graphitic shale
DTMET0033	1.16	DTRC046	17	21	Makosa	Oxide	Shale
DTMET0034	1.52	DTRC046	27	37	Makosa	Oxide	Shale
DTMET0035	2.53	DTRC047	116	122	Makosa	Fresh	Greywackes/graphitic
DTMET0036	1.84	DTRC073	106	111	Makosa	Fresh	Greywacke
DTMET0037	1.58	DTRC073	111	117	Makosa	Fresh	Graphitic shale
DTMET0038	5.90	DTRC199	24	29	Makosa Tail	Oxide	
DTMET0039	2.32	DTRC186	75	80	Makosa Tail	Fresh	
DTMET0040	16.48	DTRC155	17	19	Makosa Tail	Oxide	
DTMET0041	0.85	DTRC155	25	32	Makosa Tail	Oxide	
DTMET0042	10.46	DTRC156	7	12	Makosa Tail	Oxide	
DTMET0043	6.76	DTRC145	15	22	Makosa Tail	Oxide	
DTMET0044	13.15	DTRC149	36	41	Makosa Tail	Oxide	
DTMET0045	2.92	DTRC150	25	32	Makosa Tail	Oxide	
DTMET0046	3.03	DTRC181	4	9	Makosa Tail	Oxide	

Source: AMC, 2025.

Figure 13.1 shows the locations of all samples selected for metallurgical testing inclusive of Phase 2 test programs.

Figure 13.1 Selected metallurgical sample locations



Source: Thor, 2026.

13.2.1.2 Assays

Assay results for the metallurgical samples are in Table 13.2.

Table 13.2 Selected assay results

Sample ID	Ag (ppm)	As (ppm)	AuAve (g/t)	C-tot (%)	Fe (%)	S (%)
DTMET0013	<2	1,260	1.62	4.44	4.28	2.46
DTMET0014	<2	1,610	2.62	2.25	3.30	<0.02
DTMET0015	<2	320	0.97	0.15	5.38	<0.02
DTMET0016	<2	940	1.25	0.75	4.76	<0.02
DTMET0017	<2	600	1.21	2.28	4.42	0.94
DTMET0018	<2	1,200	1.24	3.96	4.36	2.14
DTMET0019	<2	1,190	2.03	2.25	3.38	<0.02
DTMET0020	<2	410	0.80	0.15	5.88	0.04
DTMET0021	<2	780	0.98	0.63	4.84	<0.02
DTMET0022	<2	620	1.60	2.01	4.42	0.76
DTMET0023	<2	970	2.16	0.90	4.50	0.06
DTMET0024	<2	990	1.63	1.17	4.18	0.08
DTMET0025	<2	1,050	0.83	2.04	3.92	1.42
DTMET0026	<2	4,530	6.17	3.27	2.72	1.04
DTMET0027	<2	1,560	1.99	1.02	4.42	1.74
DTMET0028	<2	2,040	0.61	1.26	5.42	2.56
DTMET0029	<2	2,640	1.75	1.35	4.48	1.74
DTMET0030	<2	1,720	1.03	0.96	4.58	0.02

Sample ID	Ag (ppm)	As (ppm)	AuAve (g/t)	C-tot (%)	Fe (%)	S (%)
DTMET0031	<2	1,590	1.34	1.08	5.60	<0.02
DTMET0032	<2	3,930	2.07	3.33	3.68	1.54
DTMET0033	<2	460	1.16	0.27	4.98	0.06
DTMET0034	<2	2,270	1.52	0.93	4.74	0.04
DTMET0035	<2	8,710	2.53	4.08	4.78	2.80
DTMET0036	<2	910	1.84	1.80	4.88	1.78
DTMET0037	<2	120	1.58	1.80	4.14	1.42
DTMET 0038	<2	1,010	5.90	1.50	6.82	<0.02
DTMET 0039	<2	3,530	2.32	2.94	3.10	1.08
DTMET 0040	<2	310	16.5	<0.03	4.74	0.02
DTMET 0041	<2	250	0.85	<0.03	5.06	<0.02
DTMET 0042	<2	400	10.5	<0.03	3.46	0.02
DTMET 0043	<2	2,050	6.76	<0.03	10.0	<0.02
DTMET 0044	<2	1,240	13.2	<0.03	6.06	<0.02
DTMET 0045	<2	730	2.92	0.33	3.52	0.14
DTMET 0046	<2	6,130	3.03	0.12	4.20	<0.02

Source: Thor, 2025.

Fresh ores are characterised by high sulphur grades, oxide ores by low or zero sulphur grades. Carbon is variable across each ore type.

13.2.1.3 Cyanide leaching

Cyanide leach tests were undertaken on all samples using the test conditions below.

Table 13.3 Cyanide leach test conditions

Parameter	Unit	Value
P_{80} particle size	µm	106
NaCN concentration – Initial / Maintain	ppm	3,330 / 2,000
Carbon	g/L	20
pH via lime addition	pH	10.5
Slurry density	%w/w	40
Dissolved oxygen (DO) – via O_2 sparging	mg/L	20-30
Duration / Sampling intervals	hours	0, 24, 48

Source: NORINCO, 2025.

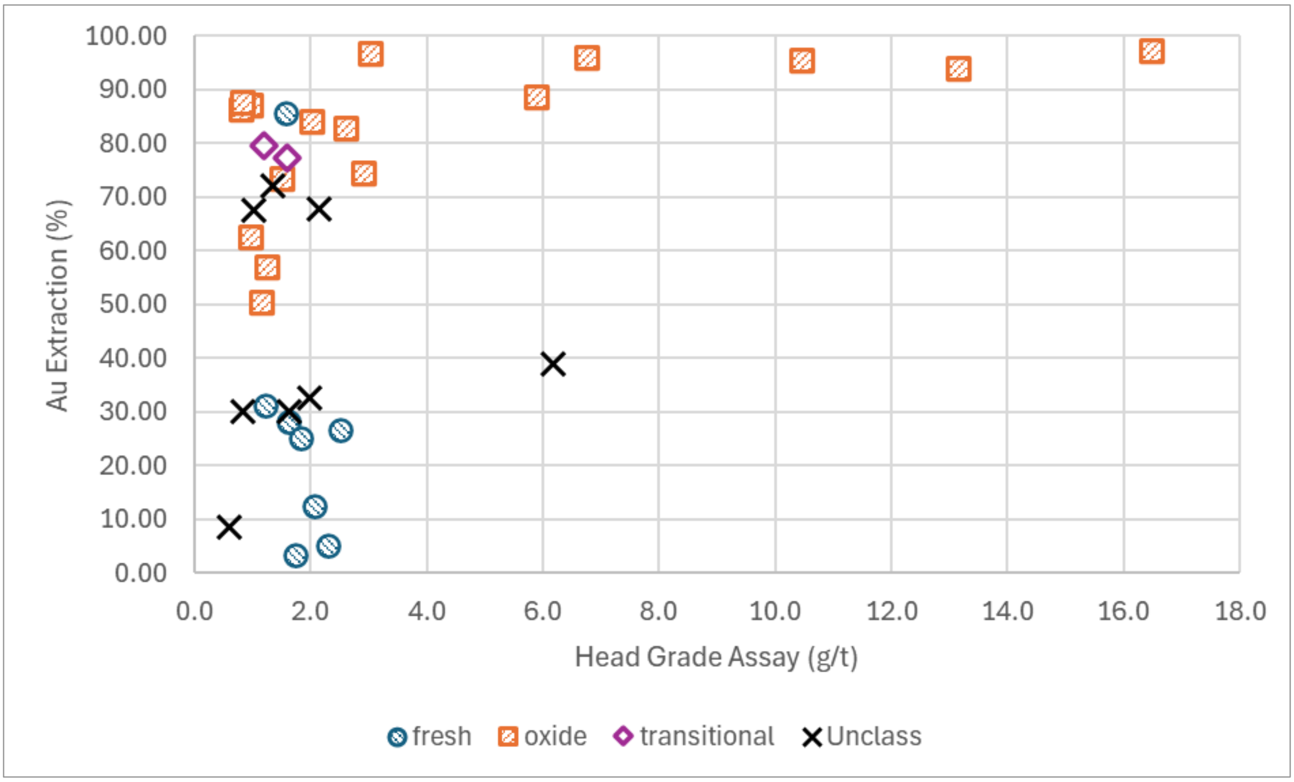
Results of leaching are shown in Table 13.4 and plotted in Figure 13.2. The results are grouped by zone. Oxide materials have the highest recoveries and are generally amenable to cyanide leaching. Lower recoveries are likely due to the double refractory nature of the ore, i.e., presence of sulphide ore and preg-robbing carbonaceous material. Later work by IMO demonstrated that significant portions of gold are distributed in silicates and are not recoverable using the proposed flowsheet. Carbon in variable proportions of organic and inorganic carbon is also present, and preg-robbing has been demonstrated in subsequent test work. The unclassified group splits into two groups - low sulphur grades with recoveries of approximately 70%, and high sulphur grades with recoveries of 30% and less. These materials are probably fresh ore types.

Table 13.4 Cyanide leach recoveries

Sample ID	Head grade assay (g/t)	Prospect	Zone	Lithology	Au extraction (%)
DTMET0013	1.62	Makosa	Fresh	Graphitic shale	28.07
DTMET0018	1.24	Makosa	Fresh	Graphitic shale	31.05
DTMET0029	1.75	Makosa	Fresh	Graphitic shale	3.37
DTMET0032	2.07	Makosa	Fresh	Graphitic shale	12.37
DTMET0035	2.53	Makosa	Fresh	Greywackes / graphitic	26.59
DTMET0036	1.84	Makosa	Fresh	Greywacke	25.07
DTMET0037	1.58	Makosa	Fresh	Graphitic shale	85.55
DTMET0039	2.32	Makosa Tail	Fresh		5.10
DTMET0014	2.62	Makosa	Oxide	Greywacke / felsic	82.59
DTMET0015	0.97	Makosa	Oxide	Greywacke	87.00
DTMET0016	1.25	Makosa	Oxide	Greywacke	56.99
DTMET0019	2.03	Makosa	Oxide	Greywacke / felsic	84.03
DTMET0020	0.80	Makosa	Oxide	Greywacke	86.19
DTMET0021	0.98	Makosa	Oxide	Greywacke	62.55
DTMET0033	1.16	Makosa	Oxide	Shale	50.43
DTMET0034	1.52	Makosa	Oxide	Shale	73.34
DTMET0038	5.90	Makosa Tail	Oxide		88.40
DTMET0040	16.48	Makosa Tail	Oxide		97.10
DTMET0041	0.85	Makosa Tail	Oxide		87.40
DTMET0042	10.46	Makosa Tail	Oxide		95.30
DTMET0043	6.76	Makosa Tail	Oxide		95.80
DTMET0044	13.15	Makosa Tail	Oxide		93.90
DTMET0045	2.92	Makosa Tail	Oxide		74.40
DTMET0046	3.03	Makosa Tail	Oxide		96.50
DTMET0017	1.21	Makosa	Transitional	Greywacke / graphitic shale	79.55
DTMET0022	1.60	Makosa	Transitional	Greywacke / graphitic shale	77.20
DTMET0023	2.16	Makosa	Unclass		67.85
DTMET0024	1.63	Makosa	Unclass		30.01
DTMET0025	0.83	Makosa	Unclass		30.06
DTMET0026	6.17	Makosa	Unclass		38.98
DTMET0027	1.99	Makosa	Unclass		32.53
DTMET0028	0.61	Makosa	Unclass		8.67
DTMET0030	1.03	Makosa	Unclass		67.52
DTMET0031	1.34	Makosa	Fresh	Graphitic shale	72.04

Source: Thor, 2025.

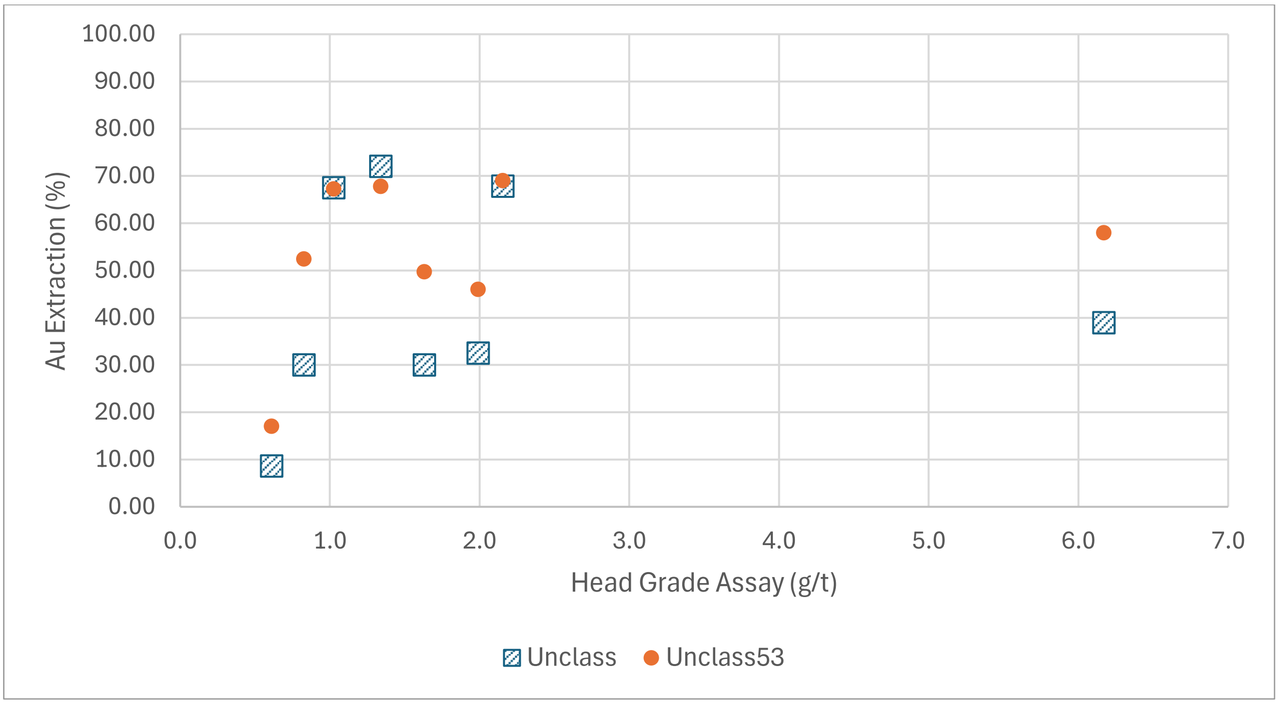
Figure 13.2 Cyanide leach recoveries



Source: ALS, 2021; Thor, 2026.

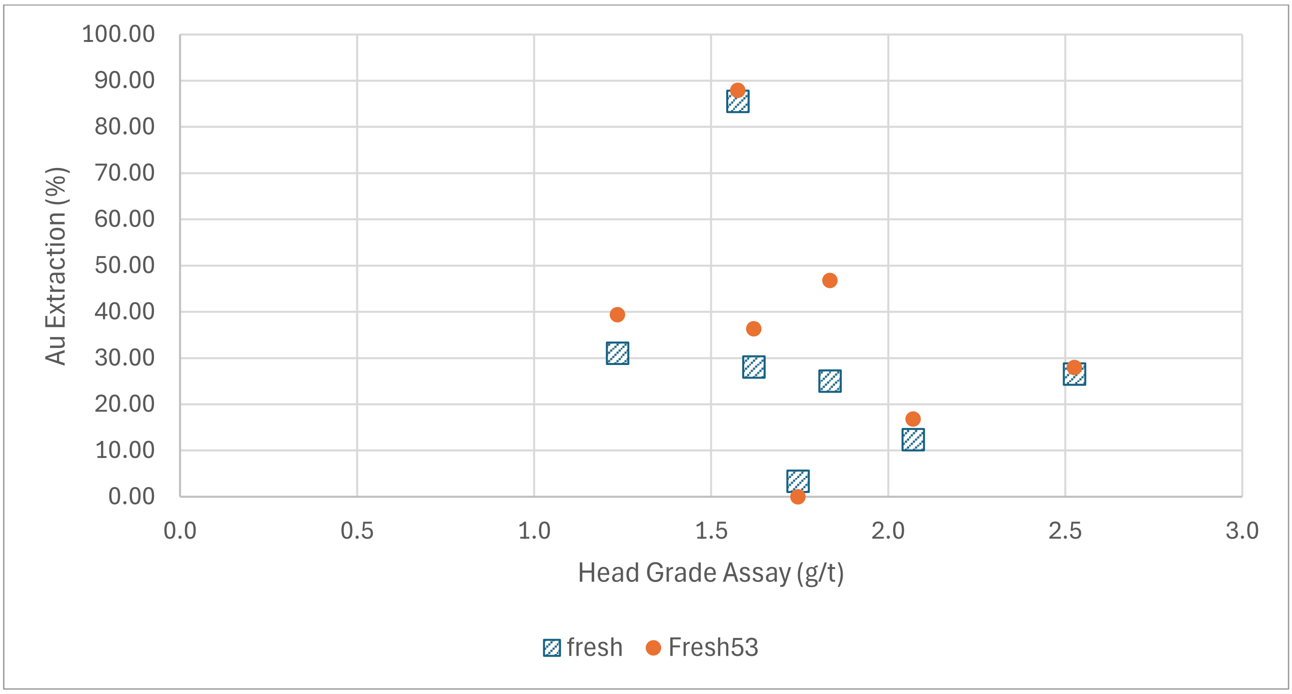
Repeat testing was undertaken on Makosa ores at reduced P_{80} grind size of 53 µm to assess if better liberation could improve recoveries. Increased recoveries were achieved for five of eight of the unclassified test samples and three of seven fresh test samples. This indicates that a proportion of the gold is finer and may also be distributed in refractory phases. The oxide test samples typically showed slightly lower recoveries suggesting that grind size is adequate for these materials. Figure 13.3, Figure 13.4, and Figure 13.5 are unclassified, fresh, and oxide samples respectively.

Figure 13.3 Unclassified sample Au recoveries at 106 and 53 µm P_{80}



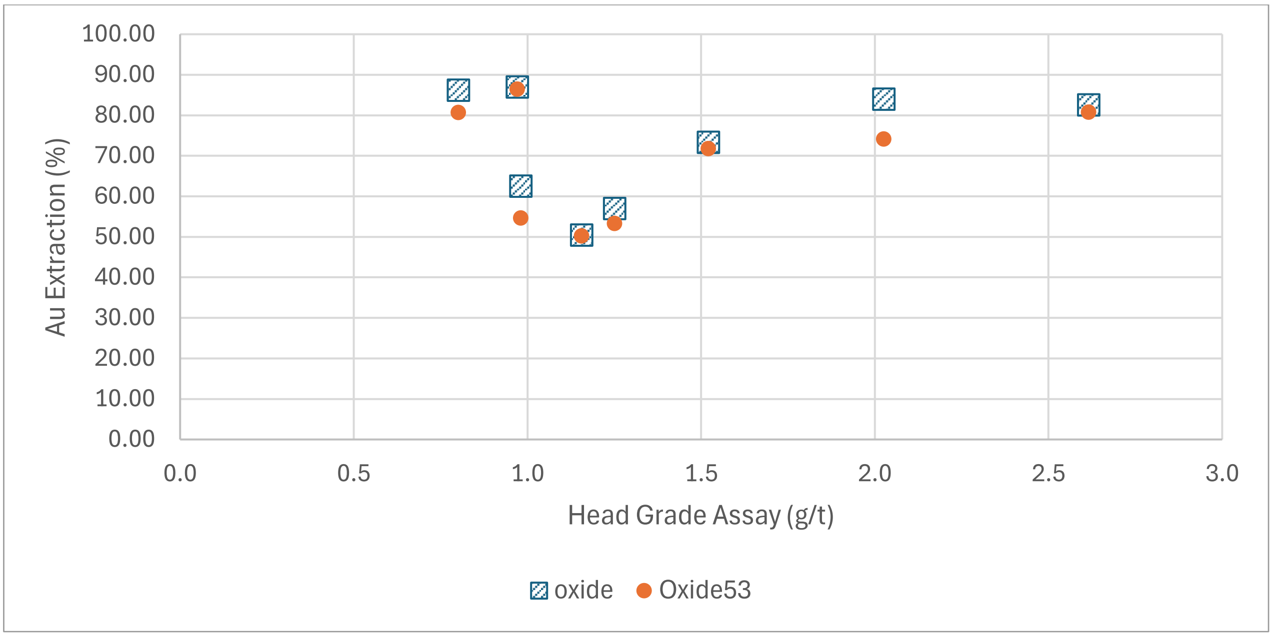
Source: ALS, 2021; Thor, 2026.

Figure 13.4 Fresh sample Au recoveries at 106 and 53 µm P_{80}



Source: ALS, 2021; Thor, 2026.

Figure 13.5 Oxide sample Au recoveries at 106 and 53 µm P_{80}



Source: ALS, 2021; Thor, 2026.

Based on this program of testing, additional samples were collected and submitted for extensive testing by IMO in Perth, Australia.

13.2.2 Independent Metallurgical Operations - 2024

IMO performed metallurgical tests in 2024 to assess the processing requirements for the oxide and fresh ore types, particularly focusing on the fresh ore due to its dominance in the resource and early indications of its refractory nature. The testwork program consisted of testing on three master composites and seven variability composites.

Master composite testwork (MC1, MC2, MC3):

- Comminution testing.
- Composite characterisation inclusive of head assay, mineralogy, and diagnostic leach work.
- Gravity gold concentration and cyanide leach optimisation testwork.
- Flotation optimisation testwork (conducted on fresh composites only).
- Bulk Flotation and concentrate processing, including oxidative leaching, roasting and bacterial oxidation (conducted on fresh composites only).

Variability composite testwork:

- Comminution testing.
- Composite characterisation.
- Gravity gold concentration and cyanide leach testwork (conducted on oxide and transitional composites only).
- Flotation testwork (conducted on transitional and fresh ore composites only).

The key outcomes of this work were as follows:

- MC1 Makosa Oxide:
  - The MC1 oxide composite is very amenable to conventional gravity separation and cyanide leaching, reporting a maximum gold recovery of 96%.

- MC2 Makosa Tail Fresh:
  - This composite achieved a maximum gold recovery of 61% when processed via flotation ultrafine grind / oxidative leach / CN leach. The remaining 39% unrecovered gold is likely locked in silicates at ultrafine particle sizes as supported by the detailed characterisation work (diagnostic analysis reported ~40% of the gold is silicate locked).
- MC3 Makosa Fresh:
  - This composite achieved a maximum gold recovery of 27% when processed via flotation / roast / CN leach. The remaining 73% unrecovered gold is likely locked in silicates at ultrafine particle sizes as supported by the detailed characterisation work (diagnostic analysis reported approximately 75% of the gold is silicate locked).

Variability samples:

- Oxide samples (VC1, VC3) are amenable to gravity recovery of gold.
- Transition and fresh ores generally had low gravity gold recoveries.
- Overall results indicate the oxide ore is readily leachable however the transitional ore varies from 86.6% (Makosa Tail, VC2) to 60.2% (Makosa, VC5) recovery.
- Variability composites of fresh ore (VC4, VC6, VC7) were not tested by leaching based on the low recoveries of 15% to 19% achieved with the fresh master composites (MC2, MC3).

Overall conclusions from this stage of metallurgical testing were as follows:

- Oxide ores can be processed by conventional gravity and CIL.
- The selection of CIL processing mitigates against preg-robbing for those ores with organic carbon.
- Transition ores can be processed by CIL but further work is necessary to understand the variations observed.
- Fresh ores are not amenable to use of CIL processes and achieved poor recoveries even after flotation concentration and oxidative or high temperature treatment of the concentrates followed by cyanide leaching.

13.2.2.1 Samples

IMO was supplied samples of ore recovered by RC drilling. Ore types tested were weathered (oxidised MC1), non-oxidised (Makosa Tail MC2), and non-oxidised (Makosa MC3) A total of three (3) Master and seven (7) Variability Composites were formed during the program (see Table 13.5).

Table 13.5 Douta samples

Composite type	Composite ID	Project zone	Ore type	Calculated Au grade g/t
Master Composites	MC1	Makosa	Oxide (SOX)	1.77
	MC2	Makosa Tail	Fresh	0.84
	MC3	Makosa	Fresh	1.26
Variability Composites	VC1	Makosa Tail	Oxide (SOX)	0.57
	VC2	Makosa Tail	Transitional (MOX)	0.80
	VC3	Makosa	Oxide (SOX)	0.71
	VC4	Makosa	Fresh	1.62
	VC5	Makosa	Transitional (MOX+WOX)	0.70
	VC6	Makosa	Fresh	2.60
	VC7	Makosa	Fresh	0.51

Source: Thor, 2025.

The source of samples is in Table 13.6 and locations are in Figure 13.1.

Table 13.6 Sample drillhole locations

Sample_ID	Drillhole ID	From	To	Au grade (g/t)
MC1	DTDD0020	0	5	1.77
	DTDD0028	1	8	
	DTDD0029	0	8	
MC2	DTDD0015	52	63	0.84
	DTDD0018	57	70	
	DTDD0033	37	58	
MC3	DTDD0020	27	35	1.26
	DTDD0021	45	53	
	DTDD0022	90	102	
	DTDD0026	75	94	
	DTDD0030	74	84	
VC1	DTDD0016	0	8	0.57
VC2	DTDD0016	8	23	0.8
VC3	DTDD0028	22	27	0.71
VC4	DTDD0027	38	56	1.62
VC5	DTDD0027	18	33	0.7
VC6	DTDD0021	55	76	2.6
VC7	DTDD0032	138	151	0.51

Source: Thor, 2025.

13.2.2.2 Comminution

Comminution testing was done on diamond drill core master composites and variability samples. The testing on Comp 7 and Comp 8 was completed in 2025 but is reported here for comparison.

SMC testing was conducted on the master composite from diamond drill cores, MC2 and MC3, and a sample from Douta West, Comp 8 only. MC1 was too friable to generate the required particle sizes for testing and a second Douta West sample was also too friable for testing. SMC testing results are presented in Table 13.7. Parameters determined are as follows:

- Dwi - Drop Weight Index, a measure of overall rock strength.
- Mi parameters - Comminution indices.
- SG - specific gravity.

Table 13.7 SMC test results

Composite		Dwi (kWh/m^3)	Dwi (%)	Mi Parameters (kWh/t)				SG
				Mia	Mib	Mih	Mic	
MC2	Makosa Tail, Fresh Ore	6.66	50	18.9	33.15	13.9	7.2	2.77
MC3	Makosa, Fresh Ore	4.01	18	12.9	26.43	8.6	4.5	2.7
Comp 8	Douta West, MOX	1.06	2.00	4.80	-	2.50	1.30	2.48

Source: Thor, 2025.

Derived parameters are shown in Table 13.8. Parameters determined are as follows:

- A, b, A*b – Parameters describing resistance to impact breakage.
- t_a – tumbling mill design parameter.
- SCSE – SAG Circuit Specific Energy, standardised energy parameter for SAG milling.

Table 13.8 Comminution derived parameters

Composite		A	b	A*b		ta		SCSE (kWh/t)	
				Value	%	Value	%	Value	%
MC2	Makosa Tail, Fresh Ore	66.0	0.63	41.6	58.5	0.39	59.0	9.83	58.1
MC3	Makosa, Fresh Ore	60.8	1.11	67.5	23.6	0.65	27.3	7.90	22
Comp 8	Douta West, MOX	60.40	3.88	234.40	2.00	2.45	2.00	5.51	1.90

Source: Thor, 2025.

On the basis of the testing results, the fresh sulphidic ores are characterised as “medium to moderately soft” with respect to impact breakage and the Douta West ore is characterised as “very soft”.

Bond Ball Mill Index (BBWi) tests were carried out on all Makosa and Makosa Tail ore samples except VC1 and VC3 due to insufficient samples. Comp 7 and Comp 8 from Douta West were tested in a follow-up program and were determined to be highly grindable with low BBWi values. BBWi test results are shown in Table 13.9.

Table 13.9 Master composite BBWi tests

Composite		Closing screen size (µm)	BBWi (kWh/t)	Grindability (g/rev)	80% Passing feed size (µm)	80% Passing product size (µm)	Mib (kWh/t)
MC1	Makosa, Oxide Ore	106	14.2	1.35	2,041	76	19.5
MC2	Makosa Tail, Fresh Ore	106	22.7	0.80	2,636	85	33.1
MC3	Makosa, Fresh Ore	106	18.7	1.01	2,414	84	26.4
VC1	Makosa Tail, Oxide Ore						
VC2	Makosa Tail, Transitional Ore	106	13.07	1.59	1,970	82	17.3
VC3	Makosa, Oxide Ore						
VC4	Makosa, Fresh Ore	106	14.30	1.42	2,359	85	19.0
VC5	Makosa, Transitional Ore	106	16.05	1.27	2,225	87	21.7
VC6	Makosa, Fresh Ore	106	20.94	0.90	2,277	85	30.2
VC7	Makosa, Fresh Ore	106	20.76	0.91	2,291	85	29.9
Comp 7	Douta West, MOX	106	3.9	8.74	1,442	102	3.76
Comp 8	Douta West, Fresh	106	8.7	2.30	1,260	64	11.54

Source: Thor, 2025.

The range of BBWi values from 3.9 kWh/t to 22.7 kWh/t indicates throughputs will vary significantly with significantly higher throughputs likely for oxide and transitional ores. The low grindability of different ores can be attributed to the presence of silicates and quartz in varying proportions.

13.2.2.3 Characterisation

Samples were characterised by elemental assay, preg-robbing index, quantitative mineralogy, and diagnostic leaching. Key assay data of the samples are given in Table 13.10.

Table 13.10 Selected assay data

Element	Unit	MC1	MC2	MC3	VC1	VC2	VC3	VC4	VC5	VC6	VC7
		Makosa, Oxide Ore	**Makosa Tail, Fresh Ore**	**Makosa, Fresh Ore**	**Makosa Tail, Oxide Ore**	**Makosa Tail, Transitional Ore**	**Makosa, Oxide Ore**	**Makosa, Fresh Ore**	**Makosa, Transitional Ore**	**Makosa, Fresh Ore**	**Makosa, Fresh Ore**
Estimated Au grade	**g/t**	**1.77**	**0.84**	**1.26**	**0.57**	**0.80**	**0.71**	**1.62**	**0.70**	**2.60**	**0.51**
Au average	**g/t**	**1.79**	**0.86**	**1.33**	**0.46**	**0.81**	**0.87**	**1.43**	**0.95**	**2.61**	**0.29**
Ag	ppm	3.06	0.18	0.55	0.56	2.12	0.07	0.48	0.35	0.46	0.14
As	ppm	883	1,371	1,204	1,919	1,091	1,135	2,209	427	2,315	208
Total Carbon	%	0.16	1.62	3.16	0.16	0.06	7.54	3.78	2.99	2.2	4.01
Organic Carbon	%	0.14	0.32	1.56	0.10	0.06	0.10	1.80	0.59	1.03	0.64
Fe	%	7.49	5.41	4.66	22.23	6.75	2.69	3.77	4.71	3.63	5.13
Sulphur	%	0.01	1.21	1.65	<0.01	0.03	0.01	1.79	0.03	1.74	0.79
Sulphide	%	0.001	1.19	1.61	<0.01	0.01	0.01	1.71	0.02	1.72	0.78

Source: Thor, 2025.

The results indicate the following:

- Arsenic grades range from 208 ppm to 2,315 ppm and are correlated with gold grades. Gold associated with arsenic may be refractory, challenging recovery from the ores.
- Sulphur is present as sulphides and associated with fresh ores.
- Carbon is present as carbonates and organic carbon. Organic carbon is strongly associated with fresh ores and to a lesser extent transitional suggesting preg-robbing will be an important negative issue for fresh ores.

Preg-robbing tests to determine Preg-robbing Index (PRI) were conducted on all ores. (see Table 13.11). Fresh ores were found to be strongly preg-robbing. The two transitional ores exhibited minor preg-robbing but the oxide ores were mixed. MC1 showed moderate preg-robbing even though its organic carbon was 0.14% marginally higher than VC1 and VC3 at 0.1% which showed no preg-robbing.

Table 13.11 Preg-robbing indices

Sample	Ore type	PRI
MC1	Makosa, Oxide Ore	31%
MC2	Makosa Tail, Fresh Ore	49%
MC3	Makosa, Fresh Ore	100%
VC1	Makosa Tail, Oxide Ore	-1%
VC2	Makosa Tail, Transitional Ore	11%
VC3	Makosa, Oxide Ore	0%
VC4	Makosa, Fresh Ore	77%
VC5	Makosa, Transitional Ore	8%
VC6	Makosa, Fresh Ore	100%
VC7	Makosa, Fresh Ore	100%

Source: NORINCO, 2025.

To investigate the factors underlying poorer gold recovery of the fresh ore master composites, Quantitative Evaluation of Minerals by Scanning Electron Microscopy (QEMSCAN) mineralogy was completed on ground samples separated with a Knelson Concentrator into a Knelson gravity concentrate and a gravity tail.

Main observations from this work were as follows:

- Pyrite is the dominant sulphide detected throughout the samples (average of 3% by mass), with over 50% of the mineral classified as 'well liberated'. The less liberated pyrite is mainly locked with silicates, and less commonly with oxides / carbonates.
- Minor arsenopyrite (about 0.3% by mass) is present in each sample. The arsenopyrite has a P_{80} of approximately 40 µm in the samples, and approximately 77% of it is classified as 'well liberated'. The less liberated arsenopyrite is mainly associated with silicates, and less commonly with oxides / carbonates and pyrite.
- Silicates are the major non-sulphide gangue detected in each sample. Quartz and feldspars are the major silicates, followed by micas, chlorite, amphiboles, and kaolinite.
- All gold grains >1 µm are either completely encapsulated in a host mineral or only partially exposed. Gold grains occur most commonly in pyrite and arsenopyrite, and much less commonly in silicates.
- No gold > 1 µm was detected in the Knelson tailings.

- Pyrite was estimated to contain approximately 15% of the total gold.
- UP to 45% of the total gold in MC2 was estimated to be hosted in arsenopyrite and up to 18% in MC3.

Diagnostic leach testing was undertaken on the three Master Composites to assess the deportment of gold throughout the ore.

Oxide ore (MC1) was amenable to direct cyanide leaching which recovered 90.6% of the gold. Neither MC2 or MC3 ores were amenable to cyanidation with less than 10% each recoverable by free and intensive cyanidation routes. Gold in MC2 was distributed as 40% silicate locked and 53% in sulphides / other refractory phases. Gold in MC3 was distributed as 75% silicate locked and 15% in sulphides / other refractory phases.

13.2.2.4 Whole of ore UFG and cyanide leaching

Master composites were tested by ultrafine grinding to P_{80} of 7 µm and cyanide leaching to assess early in the test program if increased liberation was achievable and could improve recoveries. Results are given in Table 13.12.

Table 13.12 Whole ore leaching - Ultrafine grind

Composite	Unit	MC1	MC2	MC3	MC2	MC3
		Makosa, Oxide	Makosa Tail, Fresh	Makosa, Fresh	Makosa Tail, Fresh	Makosa, Fresh
Leach Test	#	Round1-LT1	Round1-LT2	Round1-LT3	Round2-LT1	Round2-LT2
Carbon added	g	-	-	-	30	30
Calc'd Ore head grade	g/t	2.43	1.07	1.26	0.98	1.34
Assay Ore head grade	g/t	1.79	0.86	1.33	0.86	1.33
48 Hour Gold recovery	%	85.5	2.4	0.1	28.3	22.4

Source: NORINCO, 2025.

These results indicate that oxide ore is amenable to standard cyanide leaching. Fresh ores are refractory and also exhibit preg-robbing (therefore, double refractory). The Round 2 leach tests demonstrated that preg-robbing can be mitigated by carbon addition to the leach but the majority of the gold is refractory or locked in silicates and not recoverable to carbon under these conditions.

13.2.2.5 Gravity, leach

Amenability to gravity recovery of gold was tested using a Nelson concentrator. Results of recovery followed by intensive leaching for each master composite is shown in Table 13.13. Recovery was highly variable, ranging from 53.3% to 4.76%.

Table 13.13 Master composite gravity gold recovery

Composite	Unit	MC1	MC2	MC3
Gravity Gold recovery	%	53.3	6.7	4.76

Intensive leach tails were recombined with gravity tails in preparation for testing of the effect of grind size and reagent dose on recovery by CIL. Table 13.14 shows results for trials at P_{80} sizes of 75 µm, 106 µm, and 150 µm.

Table 13.14 Master composite recoveries

	MC1 – Makosa Oxide		MC2 – Makosa Tail Fresh		MC3 – Makosa Fresh	
Grind size P_{80} (µm)	Gravity recovery (%)	Total recovery (%)	Gravity recovery (%)	Total recovery (%)	Gravity recovery (%)	Total recovery (%)
75	53.8	95.7	6.7	18.8	4.8	14.9
106	53.7	95.3	6.5	18.4	4.7	14.6
150	50.0	95.1	6.9	17.8	4.8	16.1

Source: NORINCO, 2025.

Gravity recovery was effective for MC1 (oxide) but low for the two fresh ore master composites. Subsequent leaching of the gravity tails was poor (10 to 12% recovery increase) for the fresh ore composites and not impacted by the grind size.

Leach reagent optimisation tests were only performed for MC1 (oxide), with results shown in Table 13.15. Fresh ore composites have been deemed unsuitable for processing by CIL and alternative processes have been assessed as described later.

Table 13.15 MC1 CIL reagent optimisation

MC1 – Makosa, Oxide	Unit	Leach test		
Leach Number	#	3	10	11
Grind size P_{80}	µm	150	150	150
Maintained pH 8	pH	8.5-9.0	9.5	8.5-9.0
Initial / Maintained NaCN	ppm	500 / 300	500 / 300	400 / 200
Calc'd Ore head grade	g/t	3.53	3.25	3.49
Assay Ore head grade	g/t	1.79	1.79	1.79
Gravity Gold recovery	%	50.00	55.70	53.40
Overall Gold recovery	%	95.10	94.80	95.30

Source: NORINCO, 2025.

Results showed that the oxide ore can be leached at higher pH (9.5) and lower cyanide concentration (400 ppm / 200 ppm) with comparable results.

Testing of variability composites was restricted to oxide (VC1, VC3) and transitional (VC2, VC5) ores due to the low gravity and CIL recoveries from the fresh ore master composites. The results are shown in Table 13.16.

Table 13.16 Variability composite gravity and CIL recoveries

Variability composite	Unit	MC1	VC1	VC2	VC3	VC5
		Makosa, Oxide	Makosa Tail, Oxide	Makosa Tail, Trans	Makosa, Oxide	Makosa, Trans
Leach Test	#	LT11	LT-VC1	LT-VC2	LT-VC3	LT-VC5
Calc'd Ore head grade	g/t	3.53	0.6	0.83	0.92	0.81
Assay Ore head grade	g/t	1.79	0.46	0.81	0.87	0.95
Gravity Gold recovery	%	50.00	8.50	5.90	1.00	11.00
Overall Gold recovery	%	95.10	88.00	86.60	93.10	60.20

Source: NORINCO, 2025.

The variability samples all demonstrated low gravity recoveries likely due to fine dissemination of the gold in the ore. CIL recoveries were generally high with the exception of VC5 Makosa transition ore indicating refractory, silicate locked gold comprises a significant proportion of the gold.

13.2.2.6 Pre-concentration flotation, oxidative leaching, and roasting

Pre-concentration flotation

The fresh ores were not amenable to CIL leaching. Consequently, testing to determine whether reasonable recovery of a flotation concentrate could be achieved, that could then be oxidatively leached or roasted to release the gold. The work was completed in two Phases. Phase 1 was optimisation of flotation conditions for the master composites, and the transition and fresh variability composites. Phase 2 was bulk flotation of MC2 and MC3 fresh ore master composites to produce sufficient concentrate for subsequent oxidative leaching or roasting testing. In addition, samples were prepared for bacterial oxidation leach testing in China.

All work was conducted in a standard 4 L Agitair bench flotation cell using Perth tap water. Final optimised flotation parameters for the testwork are shown in Table 13.17. The master composite recoveries were strongly influenced by reagent dosage, recovery increasing with increases in both $CuSO_4$ addition and PAX.

Table 13.17 Optimised flotation parameters

Parameter	Unit	Value
Grind size P_{80}	mm	75
Slurry density	%w/w	30
Total duration	minutes	13
pH	-	Natural
$CuSO_4$ addition	g/t	500
PAX	g/t	200

Source: NORINCO, 2025.

The optimum recoveries for Makosa Tail (MC2) and Makosa (MC3) master composites are shown in Table 13.18.

Table 13.18 Optimum flotation recoveries MC2 and MC3

	Unit	MC2	MC3
		Makosa Tail, Fresh Ore	Makosa, Fresh Ore
Au average	g/t	0.86	1.33
Conc Au grade	g/t	11.0	12.6
Au recovery	%	82.3	62.8

Source: NORINCO, 2025.

High intensity conditioning (HIC) was also tested to assess potential improvements in recovery. The HIC process increased the solids density from 40 wt% to 50 wt% and the agitation rate from 900 RPM to 1,100 RPM. Flotation was performed at the previously determined optimum conditions. HIC resulted in an increase in concentrate gold grade of approximately 25% but reduced recovery for MC3 by 9.9% to 52.0%. Considering that the overall objective was to maximise recovery it was decided to discontinue further HIC testing.

The transition and fresh ore variability composites were tested using the optimised flotation conditions determined for the master composites. The results are shown in Table 13.19 with the master composites for comparison.

Table 13.19 Flotation results for variability composites

	Unit	MC2	MC3	VC2	VC4	VC5	VC6	VC7
		Makosa Tail, Fresh Ore	Makosa, Fresh Ore	Makosa Tail, Transitional Ore	Makosa, Fresh Ore	Makosa, Transitional Ore	Makosa, Fresh Ore	Makosa, Fresh Ore
Au average	g/t	0.86	1.33	0.81	1.43	0.95	2.61	0.29
Conc Au grade	g/t	11.0	12.6	3.2	11.5	9.5	24.6	3.4
Au recovery	%	82.3	62.8	23.0	85.5	54.1	47.7	88.8

Source: NORINCO, 2025.

The recovery of arsenic and sulphur for all samples displays a close correlation with gold recovery. This trend indicates that the gold throughout the ore is associated with sulphide species including arsenopyrite / pyrite.

To further develop flowsheet options bulk flotation of Makosa Tail (MC2) and Makosa (MC3) fresh ores were undertaken to generate sufficient material to test three pre-treatment options. Flotation conditions were the same as previously established. Tests were conducted with 10 kg samples in a 24 L Agitair bench flotation cell. Results for two rounds each of bulk flotation compared with batch test results are given in Table 13.20.

Table 13.20 Bulk flotation of fresh ores

	Unit	Makosa Tail			Makosa		
		Batch	Round 1 Bulk	Round 2 Bulk	Batch	Round 1 Bulk	Round 2 Bulk
Mass recovery	%	6.3	7.9	7.2	6.6	7.4	7.0
Concentrate Au grade	g/t	11.01	10.9	10.61	12.61	11.85	11.60
Concentrate Au recovery	%	78.9	90.6	88.5	61.9	69.5	69.0

Source: NORINCO, 2025.

Using larger scale equipment has resulted in increased recoveries when compared with the small batch flotation unit. Arsenic and sulphur recoveries also increased.

IMO undertook oxidative leaching and two-stage roast test work to establish the potential of either route to process fresh ore.

Oxidative leaching

Both MC2 and MC3 samples were subjected to test work with the conditions shown in Table 13.21.

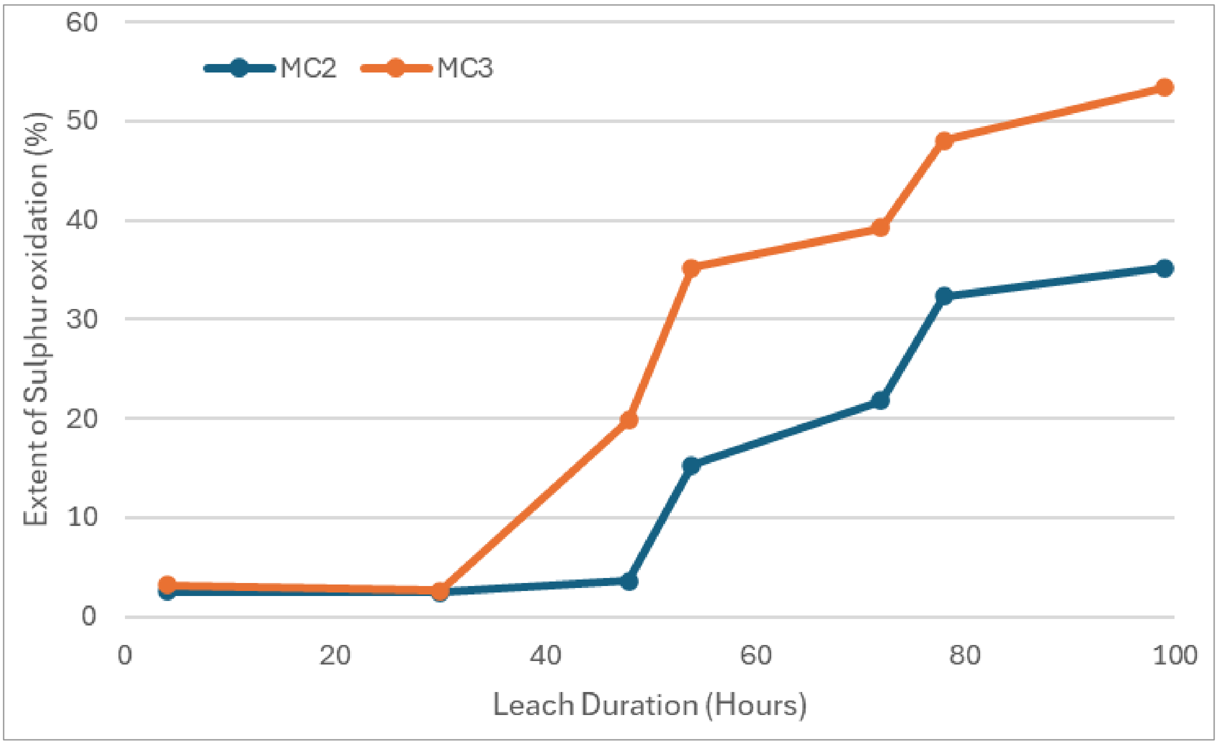
Table 13.21 Oxidative leach parameters

Parameter	Unit	Value
Sample mass	g	800
Sizing P_{80}	µm	10
Temperature	°C	95
pH via lime addition	pH	5-5.5
Slurry density	%w/w	20
Dissolved oxygen (DO) – via O_2 sparging	mg/L	>15
Duration / sampling intervals	hours	4, 30, 48, 54, 72, 78, 99

Source: NORINCO, 2025.

Samples were withdrawn at the nominated intervals and assayed for the extent of sulphur oxidation. Figure 13.6 shows the kinetic leach profiles generated for MC2 and MC3. The extent of sulphur oxidation is low for both ores reaching a maximum of 53.4% for Makosa ore (MC3) and 36.0% for MC2.

Figure 13.6 Kinetic leach profiles



Source: IMO, 2024; Thor, 2026.

The final oxidative leach products were intensively cyanide leached under the test conditions shown in Table 13.22.

Table 13.22 Cyanide leach parameters

Parameter	Unit	Value
NaCN initial / maintain	ppm	2,000 / 1,500
Carbon	g/L	10
pH via lime addition	pH	10.5
Slurry density	%w/w	40
Dissolved oxygen (DO) – via O_2 sparging	mg/L	10-15
Duration / sampling intervals	hours	2, 4, 8, 24

Source: NORINCO, 2025.

The combined results for these tests are given in Table 13.23. Makosa tail (MC2) had the highest flotation recovery and leach recovery combined of 61%. Makosa (MC3) had much lower flotation recovery and very low leach recovery and a combined overall recovery of 15.3%.

Table 13.23 Combined results flotation, oxidative leach, CIL

Sample	Units	Flotation	Oxidative leach	CN leach	Overall
MC2	Rec (%)	90.6	100	67.3	61.0
MC3	Rec (%)	69.5	100	22.0	15.3

Source: NORINCO, 2025.

For Makosa Tail the low gold recovery of 67.3% by CN leaching following oxidative leaching may be a result of the gold being locked in silicates. The high flotation recovery is not consistent with earlier diagnostic leaching that reported MC2 contained approximately 40% of the gold to be silicate locked, as this would be unlikely to be recovered by flotation. However, MC2 also had the lowest oxidation of sulphur during oxidative leach which likely resulted in lower release of gold available for cyanide leaching.

For Makosa this composite shows a lower flotation recovery more consistent with a higher level of silicate gold locking reported by diagnostic leaching. But the fact that a significant 78% of the gold was unrecoverable in the cyanide leach following oxidative leaching suggests that even though 53% of the sulphur was oxidised it did not result in effective gold release. For both samples and ore types, the effect of gold deportment between silicates and sulphides is clearly important but testing has not demonstrated a clear quantitative understanding of the mechanism of gold release for each material type. Overall, the combination of flotation followed by oxidative leaching and CIL gives relatively poor results.

2-Stage roast

Both MC2 and MC3 samples were subjected to test work with the conditions shown in Table 13.24.

Table 13.24 Two-stage roast parameters

Parameter	Unit	Value
Sample mass	g	500
Stage 1 temperature	°C	450
Stage 1 time	minutes	90
Stage 2 temperature	°C	650
Stage 2 time	minutes	90

Source: NORINCO, 2025.

Following the two-stage roast, the samples were cyanide leached using the same leach parameters shown in Table 13.22. Summarised results are shown in Table 13.25. Roasting was more effective for Makosa ore (MC3) than oxidative leaching but less effective for Makosa Tail ore (MC2). Overall recoveries were low for these processing routes.

Table 13.25 Combined results flotation, two-stage roast, CIL

Sample	Units	Flotation	Two-stage roast	CN leach	Overall
MC2	Rec (%)	90.6	100	52.9	48.0
MC3	Rec (%)	69.5	100	38.9	27.1

Source: NORINCO, 2025.

13.2.3 IMO 2025 metallurgical testing

In 2025 IMO was commissioned to extend test work to additional areas of the orebody not previously tested, with the work focused on oxide and transition ore types. Work was also undertaken to improve knowledge of preg-robbing behaviour and to characterise the response of transition ores to CIL recovery of gold. The focus of the 2025 program was to:

- Confirm CIL processing can mitigate or eliminate preg-robbing behaviour.
- Confirm CIL processing as preferred method for oxide ores.
- Confirm CIL processing as suitable for transition ores.

13.2.3.1 Samples

Two batches of samples were provided to IMO for testing. The first batch of samples was from the Douta West lease area and was comprised of seven composites from RC drillholes and two samples from diamond drillholes (Table 13.26). These were a mix of MOX and sulphidic (Fresh) ore types.

Table 13.26 Douta West Makosa MOX and sulphidic samples

Composite type	Composite ID	Project Zone	Ore type	Calculated Au grade (g/t)
RC Composite	Comp 1	Makosa South Shallow	Oxide (MOX)	2.47
	Comp 2	Makosa South Deep	Fresh	2.31
	Comp 3	Makosa Central Shallow	Oxide (MOX)	4.12
	Comp 4	Makosa South Deep	Oxide (MOX)	2.98
	Comp 5	Makosa North Shallow	Oxide (MOX)	9.33
	Comp 6	Makosa North Deep	Oxide (MOX)	1.36
DD Composite	Comp 7	Makosa	Oxide (MOX)	2.39
	Comp 8	Makosa	Fresh	1.22

Source: IMO, 2025.

The second batch of samples comprising four RC composites were also from Douta West lease area specifically targeting transition ores for CIL leach test work.

Table 13.27 Douta West Makosa transition composites

Composite type	Composite ID	Project zone	Ore type	Calculated Au grade (g/t)
RC Composite	MKS1	Makosa East	Transition	1.27
	MKS2	Makosa	Transition	1.32
	MKS3	Makosa North	Transition	1.41
	MKS4	Makosa East	Transition	1.87

Source: IMO, 2025.

The source of samples for this work is shown in Table 13.28 and in Figure 13.1.

Table 13.28 Source of metallurgical testwork samples

Sample ID	Drillhole ID	From	To	Au grade (g/t)
Comp 1	DWRC129	29	48	2.43
Comp 2	DWRC130	106	131	1.93
Comp 3	DWRC140	16	36	5.92
Comp 4	DWRC154	62	69	3.65
Comp 5	DWRC163	13	27	6.12
Comp 6	DWRC163	36	51	4.48
Comp 7	DWDD001	40	62	2.78
Comp 8	DWDD002	102	129	1.22
Comp 8	DWDD002	174	178	1.22
MKS1	DTRC1119	29	40	1.27
MKS2	DTRC1136	21	26	1.32
MKS3	DTRC1160	26	34	1.41
MKS4	DTRC1124	18	25	1.87

Source: IMO, 2025.

13.2.3.2 Comminution

Comminution testing was only done on the DD samples from Douta West Comp 7 and Comp 8. The results are reported in Section 0.

13.2.3.3 Characterisation

Samples were characterised by elemental assay and PRI with results shown in Table 13.29, Table 13.30, and Table 13.31. Quantitative mineralogy and diagnostic leaching test work is ongoing.

Table 13.29 Assays Douta West Makosa MOX and sulphidic samples

Element	Unit	Comp 1	Comp 2	Comp 3	Comp 4	Comp 5	Comp 6	Comp 7	Comp 8
		RC MOX	RC FRS	RC MOX	RC MOX	RC MOX	RC MOX	DDMOX	DD FRS
Au average	ppm	2.43	1.93	5.92	3.65	6.12	4.48	2.78	1.22
Ag	ppm	0.52	0.62	1.83	1.20	0.52	0.39	1.82	0.68
As	ppm	1,062	2,903	812	1,632	745	565	1,064	2,237
Total Carbon	%	0.03	0.8	0.05	0.06	0.05	0.05	0.08	1.16
Organic Carbon	%	0.03	0.17	0.04	0.04	0.04	0.05	0.07	0.18
Cu	ppm	216	97	202	219	175	102	182	145
Fe	%	3.58	7.32	4.40	5.44	5.72	3.13	6.33	8.33
Total Sulphur	%	<0.01	3.31	0.02	0.02	0.01	0.01	0.01	3.01
Sulphide Sulphur	%	<0.01	2.66	0.02	<0.01	<0.01	<0.01	<0.01	2.56

Source: IMO, 2025.

MOX samples have low carbon and sulphur content indicating that they will have recovery properties similar to SOX ore types. Fresh samples have high sulphur content of which greater than 80% is sulphidic sulphur.

Table 13.30 Assays Douta West Makosa transition composites

Element	Unit	MKS1	MKS2	MKS3	MKS4
Au average	g/t	1.19	1.1	1.355	1.59
Ag	ppm	<0.5	0.6	<0.5	<0.5
As	ppm	550	1099	1623	656
Total Carbon	%	0.05	0.56	1.78	0.78
Organic Carbon	%	0.05	0.51	0.21	0.39
Cu	ppm	37	89	47	117
Fe	%	2.54	5.79	7.57	4.76
Total Sulphur	%	0.05	0.04	0.42	0.19
Sulphide Sulphur	%	0.04	0.01	0.3	<0.01

Source: IMO, 2025.

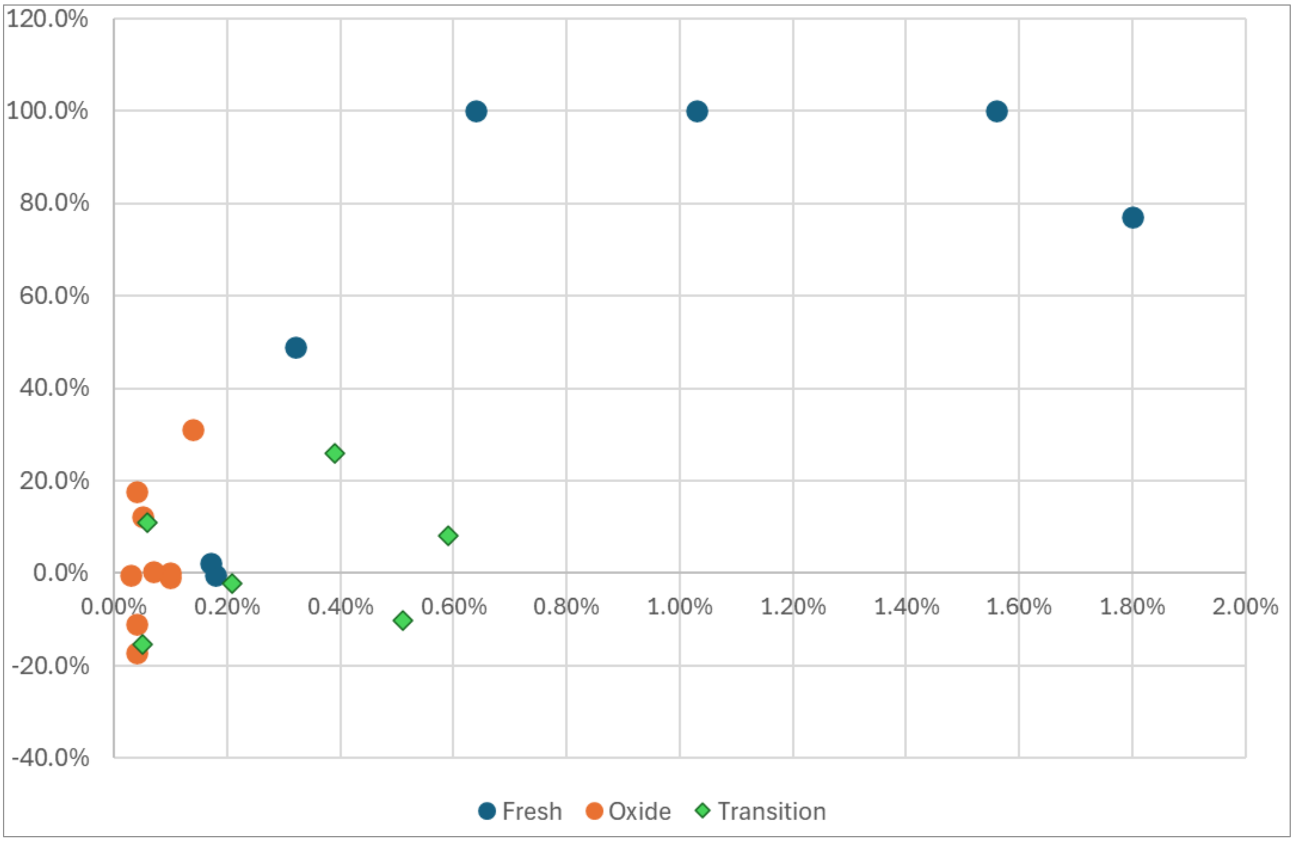
Preg-robbing tests gave inconsistent results when compared with the 2024 tests. None had high PRI and particularly the fresh ore type samples with slightly elevated but low organic carbon content tested as not preg-robbing. Figure 13.7 combines all preg-robbing data from both test work programs.

Table 13.31 Preg-robbing index - 2025 test work

Sample	Project Zone	Ore type	Preg-robbing index
Comp 1	Makosa South Shallow	Oxide (MOX)	-0.5%
Comp 2	Makosa South Deep	Fresh	2.1%
Comp 3	Makosa Central Shallow	Oxide (MOX)	17.8%
Comp 4	Makosa South Deep	Oxide (MOX)	-11.1%
Comp 5	Makosa North Shallow	Oxide (MOX)	-17.1%
Comp 6	Makosa North Deep	Oxide (MOX)	12.3%
Comp 7	Makosa	Oxide (MOX)	0.3%
Comp 8	Makosa	Fresh	-0.4%
MKS1	Makosa East	Transition	-15.3%
MKS2	Makosa	Transition	-10.2%
MKS3	Makosa North	Transition	-2.3%
MKS4	Makosa East	Transition	26.1%

Source: IMO, 2025.

Figure 13.7 Preg-robbing index (PRI) - all ore types



Source: IMO, 2025; Thor, 2026.

Oxide ore types have low organic carbon content but still exhibit low PRI for some samples. Transition ore types have a wider range of organic carbon but exhibit similar PRI to the oxides with both none and low indices. Fresh ores also exhibit zero PRI for two low carbon samples (<0.2% C) but all others above 0.3% C have high PRI.

13.2.3.4 Gravity

Amenability to gravity recovery of gold was tested using a Knelson concentrator with gold recovery by intensive CN leaching of gravity concentrate. Results for the oxide and fresh composites tested are shown in Table 13.32 and those for transitional ore samples tested are in Table 13.33.

Table 13.32 Douta West oxide and fresh gravity recovery

	Comp 1	Comp 2	Comp 3	Comp 4	Comp 5	Comp 6	Comp 7	Comp 8
	RC MOX	RC FRS	RC MOX	RC MOX	RC MOX	RC MOX	DD MOX	DD FRS
Concentrate mass recovery (%)	0.53	0.68	0.54	0.80	0.51	0.49	0.43	0.53
Gravity Gold recovery (%)	40.5	24.9	30.0	37.0	42.0	41.4	20.0	12.1

Source: IMO, 2025.

Table 13.33 Douta West transitional ore gravity recovery

	MKS1	MKS2	MKS3	MKS4
Concentrate mass recovery (%)	0.52	0.47	0.53	1.05
Gravity Gold recovery (%)	4.5	8.0	3.9	9.2

Source: IMO, 2025.

The moderate gravity recovery of oxide ore types between 20% and 42.0% is slightly lower but consistent with the 2024 test work. Similarly, transitional ores exhibited low recoveries consistent with 2024 test work. The fresh ores exhibited increased gravity recoveries when compared with 2024 but were still lower than oxide ore recoveries.

13.2.3.5 Leach testing

The focus of the 2025 program was to:

- Confirm CIL processing can mitigate or eliminate preg-robbing behaviour.
- Confirm CIL processing as preferred method for oxide ores.
- Confirm CIL processing as suitable for transition ores.

Douta West composites Comp 1 to Comp 8 were tested using baseline conditions established during the 2024 test work on sample MC1. The influence of grind size was determined at P_{80} of 150, 106, 75, and 53 µm. Initial / maintain NaCN concentrations were 500 / 300 ppm. One test was completed at NaCN concentrations of 700 / 500 ppm and found to have minimal impact. Recovery increased as the P_{80} was reduced. P_{80} of 75 µm was chosen as the preferred target size but a reduction to 53 µm could result in increased gold recovery. It is recommended that a trade-off study be undertaken to establish if recovery can be increased sufficiently to exceed the increased cost of increased grinding energy to achieve a finer grind size.

Results for the testing at P_{80} of 75 µm and 500 / 300 ppm NaCN are in Table 13.34.

Table 13.34 Gravity CIL recoveries for Douta West composites

	Comp 1	Comp 2	Comp 3	Comp 4	Comp 5	Comp 6	Comp 7	Comp 8
	RC MOX	RC FRS	RC MOX	RC MOX	RC MOX	RC MOX	DD MOX	DD FRS
Gravity Gold recovery (%)	41.3	27.8	28.3	37.0	42.6	42.7	20.2	11.3
CIL recovery (%)	52.5	51.9	62.0	55.2	51.3	50.4	66.3	65.5
Total Gold recovery (%)	93.8	79.7	90.3	92.2	93.9	93.1	88.5	76.8

Source: IMO, 2025.

Lowest recoveries of 79.7% and 76.8% were attained from fresh ores. All remaining oxide samples gave recoveries between 88.5% and 93.9%.

Douta West transition ores were tested similarly to the oxide samples. No effect was observed due to grind size P_{80}. Additional carbon (20 g/L) was added to counter preg-robbing and resulted in recovery increases of between 7% and 43%. Optimised results are given in Table 13.35.

Table 13.35 Recoveries for transition ores

	MKS1	MKS2	MKS3	MKS4
	Transition	**Transition**	**Transition**	**Transition**
Gravity Gold recovery (%)	4.5	7.6	3.6	8.3
CIL recovery (%)	82.6	74.1	80.0	50.2
Total Gold recovery (%)	87.1	81.7	83.6	58.5

Source: IMO, 2025.

13.3 Phase 2 metallurgical testwork

Test work has been undertaken by:

- Mineral Processing and Powder Technology Institute of Northeastern University (NEU)

NEU was commissioned by Northern International Cooperation Co., Ltd. (NORINCO) to conduct pre-feasibility metallurgical testwork for the Douta Gold Project. This followed detailed testwork performed by IMO that indicated fresh gold ores from the Makosa and Makosa Tail zones are challenging to process using conventional methods, including flotation, ultrafine grinding (UFG), cyanide leaching, oxidative leaching, and their various combinations. The primary difficulties are attributable to carbon preg-robbing, refractory gold in pyrite and arsenopyrite, and silicate locking of gold. In response to these challenges, NEU investigated a novel process for the Douta Gold Project: suspension oxidation roasting followed by cyanide leaching.

Testwork comprised the following:

- Sample preparation.
- Ore characterisation including assays, mineralogy, gold particle size characteristics and gold deportment.
- Crushing–suspension oxidation roasting–cyanide leaching.
- Flotation pre-enrichment–oxidation roasting–cyanide leaching.

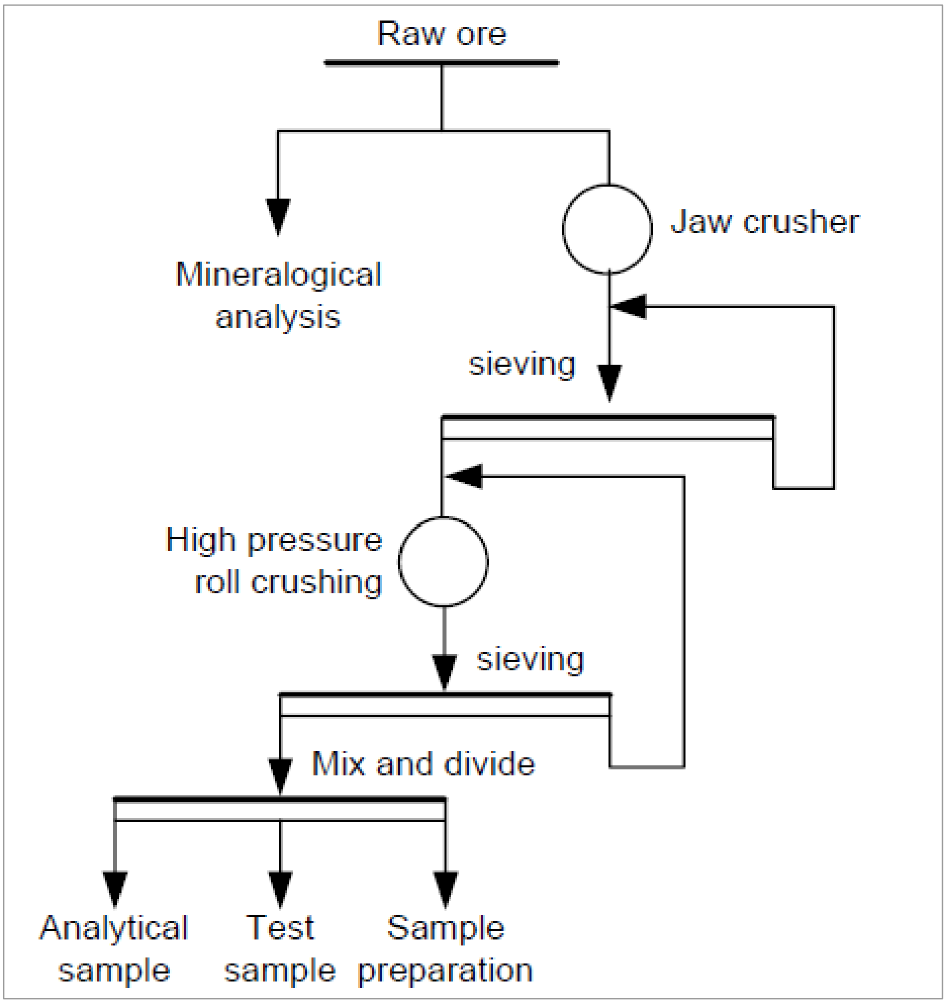
13.3.1 Samples

NEU tested samples prepared from Makosa zone (MC3) RC drilling samples and prepared from Makosa Tail (MC2) RC drilling samples. The sample masses received were 350 kg and 200 kg respectively. Samples were fresh (sulphidic) ores.

Each sample was mixed, crushed with jaw followed by double roll crusher and screened to -1 mm. 50 kg was subsampled from each primary sample and the remainder retained as back up for future work.

For roasting test work, the raw ore samples were mixed and divided into 100 g per bag, with 100 g taken for each oxidation roasting test. After the roasted product cooled to room temperature, it was thoroughly mixed and divided to obtain 50 g for leaching tests; another 30 g of the roasted product was used to determine the Au grade, and the remaining roasted product was kept as the backup sample. After the leaching tests were completed, the leach residue was washed, filtered, and dried, then thoroughly mixed and divided again, with 30 g taken for Au grade analysis and the rest kept as the backup sample. The raw ore sample processing flow is shown in Figure 13.8.

Figure 13.8 Sample preparation process



Source: NEU.

Flotation pre-concentration samples (500 g) were prepared by combining and blending 100 g samples.

NEU also tested two master composites (MC2, MC3) and the seven variability samples (VC1 to VC7) previously tested by IMO. Testwork involved direct cyanide leaching, oxidation roasting–cyanide leaching, and flotation preconcentration–roasting–cyanide leaching evaluating the response of additional samples to the process conditions determined in the initial testing.

13.3.2 Mineralogy

A single sample of each ore was tested for chemical composition, carbon speciation, mineral content by X-ray diffraction (XRD), and mineral content by Mineral Liberation Analysis (MLA). Gold occurrence was determined by a mineral particle search for gold grains. Assays are shown in Table 13.36.

Table 13.36 Makosa and Makosa Tail ore assays

Sample	SiO_2	Al_2O_3	MgO	CaO	TFe	Na	S	Sb	P	K	Au	Ag	C	As	Te	LOI
Makosa Tail	55.85	14.36	3.46	4.85	5.51	1.98	1.77	<0.01	0.07	1.35	1.93	3.52	1.3	0.21	<0.01	5.80
Makosa	54.88	13.56	2.91	4.00	5.08	1.28	1.75	<0.01	0.056	1.55	1.35	3.49	3.09	0.148	<0.01	8.88

Source: NEU.

The results of carbon speciation analysis are shown in Table 13.37. The data indicate that carbon in raw ore is mainly present as graphite, organic carbon and carbonate, with respective contents of 0.166%, 0.031%, and 1.103%. Graphitic and organic carbon are responsible for preg-robbing in the leach process and must be managed. Carbon in the Makosa sample is higher when compared with Makosa Tail ore and is primarily present as graphite, organic carbon, and carbonate, with respective contents of 0.511%, 0.033%, and 2.546%. The total content of graphite and organic carbon is 0.544%, which exhibits a pronounced preg-robbing effect during cyanide leaching.

Table 13.37 Carbon speciation test results

Sample	Graphite (%)	Organic Carbon (%)	Carbonate (%)	Total (%)
Makosa Tail	0.156	0.031	1.103	1.300
Makosa	0.511	0.033	2.546	3.090

Source: NEU.

The mineralogy of the ores was determined by XRD and scanning electron microscopy using an MLA. XRD is sensitive to mineral grades above 1 to 2% (typically) and therefore only identifies major minerals. MLA was used for a full quantitative analysis. XRD identified quartz, albite, muscovite, and kaolinite as the major minerals in both ore types.

MLA analysis identified the following minerals in each ore type (>1%).

Table 13.38 Quantitative mineral analysis

Mineral	Makosa Tail (%)	Makosa (%)
Quartz	37.5	30.8
Albite	12.9	20.0 (plagioclase)
Muscovite	18.2	24.8
Pyrite	5.2	3.9
Dolomite	3.9 (Ferroan)	8.4
Microcline	2.4	-
Ferrochlorite	2.9	-
Diopside	2.8	
Chamosite	1.1	4.2
Calcite		1.4
Biotite		1.2

Source: NEU.

Pyrite is accompanied by arsenopyrite at levels of less than 0.5%.

Gold occurs as a fine-grained, visible gold encapsulated in pyrite and silicate hosts, and intergranular between pyrite grains and other mineral pairs. Differences between Makosa Tail and Makosa ore are shown in Table 13.39.

Table 13.39 Gold occurrence

Occurrence	Makosa Tail	Makosa
Encapsulated	79%	56%
Intergranular	21%	44%

Source: NEU.

The differences in occurrence will result in differences in recovery due to the combined impact of potential liberation and proportion encapsulated.

13.3.3 Roast – leach testing

Suspended oxidation roasting–cyanide leaching tests and flotation pre-concentration–suspended oxidation roasting–leaching tests were carried out. Differences in ore properties between Makosa tail ore and Makosa ore such as a higher C content resulted in slightly different test conditions as documented below in this section.

Testing was conducted to optimise roasting conditions using a standard leaching protocol. Once optimal roast conditions were established, testing to re-optimise the leach conditions was completed.

The initial leaching test conditions were: grind size P_{95} of 38 µm, pulp concentration 30%, Jinchan (JC) reagent dosage 2.5 kg/t, CaO used to adjust pH to 11.5, leaching time 24 hours, and leaching temperature 25°C.

The roasting test conditions are shown in Table 13.40. Gas flow rate was held constant for all tests. Feed ore sample mass was 100 g. Grind size was investigated for Makosa ore only.

Table 13.40 Experimental conditions - roast-leach testing

Parameter	Range
Temperature	500, 550, 600, 650, 700, 750, 800 (°C)
Roast time	0.25, 0.5, 1, 2, 4, 8 (hr)
Oxygen Concentration	5, 10, 21, 30, 40 (%)
Grind size (% less than 74 µm) (Makosa Only)	50, 60, 70, 80, 90, 99

Source: NEU.

Standard parameters for the roasting optimisation tests are show in Table 13.41.

Table 13.41 Optimisation parameters - roasting studies

Parameter	Value
Feed particle size	-1 mm / -0.074 mm (Makosa)
O_2 conc	10%
Roast time	2 hr
Temperature	650°C
Gas flow rate	2 L/min

Source: NEU.

Makosa Tail key observations from these tests were as follows:

- Recovery increased with roasting temperature to 650°C and then stabilised between 88.9% and 87.5%. At 800°C the recovery fell to 84% possibly indicating ore sintering and secondary encapsulation of gold.
- Recovery increased quickly to 87.1% at 0.5 hours roasting time then increased minimally to 88.7% after 4 hours.
- Recovery increased to 89.7% at 21% O_2 then fluctuated between 91.0% and 89.0% at higher O_2 concentrations.

The optimal test conditions were determined to be feed particle size -1 mm, roasting temperature 650°C, oxygen concentration 21%, roasting time 0.5 hours, total gas flow rate 2 L/min.

Makosa key observations from these tests were as follows:

- Recovery increased with roasting temperature to 650°C maximising at 70.7%. At 800°C the recovery fell to 57.4% possibly indicating ore sintering and secondary encapsulation of gold.
- Recovery increased to 77.0% at 6 hours roasting time then decreased at longer times. At 2 hours roast time, recovery was 66.3%.
- Recovery increased to 70.8% at 21% O_2 and 71.3% at 30% O_2.
- Reducing the feed particle size to -0.074 mm increased leach recovery to 82.3%.

Makosa ore requires a finer grind and longer roasting time to achieve similar Au recovery to Makosa Tail ore.

The optimal test conditions were determined to be feed particle size -0.074 mm, roasting temperature 650°C, oxygen concentration 21%, roasting time 2 hours, total gas flow rate 2 L/min.

Leaching test conditions are shown in Table 13.42. Samples were ground prior to cyanide leaching to improve leach performance. 50 g of ground sample was used for the leach tests.

Table 13.42 Cyanide leaching test parameters

Parameter	Range
Grind size (% less than 38 μm)	55, 65, 75, 85, 95,
JC reagent	0.5, 1.0, 2.5, 5, 10
Leach pH	9.5, 10.5, 11.5, 12.5
Leach time (hr)	8, 16, 24, 32
Pulp concentration (%w/w)	20, 30, 40, 50
Leach temperature (°C)	25, 30, 35, 40

Source: NEU.

Standard leach conditions for the optimisation are shown in Table 13.43.

Table 13.43 Standard leach conditions for optimisation studies

Condition	Value
Leach time	24 hr
JC reagent	2.5 kg/t
Pulp conc.	30%
Leach pH	11.5
Grind size P_{75}	38 μm

Source: NEU.

Makosa Tail key observations from these tests were as follows:

- Recovery increased as the proportion of feed less than 38 μm increased to 75% (P_{75}) and then stabilised between 87.0% and 87.6%.
- Recovery increased quickly to 90.4% at a reagent dosage of 2.5 kg/t then decreased minimally to 89.3% at 10 kg/t.
- Recovery increased to 90.4% at pH of 11.5 with a slight decrease at higher pH.
- Recovery maximised between 16 and 24 hours leach time. 24 hours leach time does not increase recovery but has been selected as the convenient process period for design purposes.

- Recovery maximised at a pulp concentration of 40% w/w.
- Recovery was highest at 25°C temperature, decreased slightly to 35°C with a further significant decrease at 40°C.

For Makosa Tail, the overall optimal test conditions were as follows:

- Roasting test: feed particle size -1 mm, roasting temperature 650°C, oxygen concentration 21%, roasting time 0.5 hours, total gas flow rate 2 L/min.
- Roasted ore leaching test: grinding fineness P_{75} of 38 µm, pulp concentration 40%, JC reagent dosage 2.5 kg/t, CaO to adjust pH to 11.5, leaching time 24 hours, leaching temperature 25°C.

For Makosa ore, key observations from these tests were as follows:

- Maximum recovery was 81.5% at a reagent dosage of 5 kg/t. Only a small reduction in recovery (minimum = 79.6%) was observed for other reagent dosages.
- Recovery was 80.3% at pH of 11.5 with a decrease to below 70% recovery at higher pH.
- Recovery maximised at 16 hours leach time. 24 hours leach time reduced recovery by approximately 3% but has been selected as the convenient process period for design purposes.
- Recovery maximises 81.4% at a pulp concentration of 40% w/w.
- Recovery is highest (83.6%) at 35°C temperature and 79.4% at 25°C.

For Makosa, the overall selected test conditions were as follows:

- Roasting test: feed particle size -0.074 mm, roasting temperature 650°C, oxygen concentration 21%, roasting time 2 hours, total gas flow rate 2 L/min.
- Roasted ore leaching test: grinding fineness P_{95} of 38 µm, pulp concentration 40%, JC reagent dosage 2.5 kg/t, CaO to adjust pH to 11.5, leaching time 24 hours, leaching temperature 25°C.

Makosa ore requires longer roasting time and finer grind than Makosa Tail ore. Higher oxygen concentrations are also beneficial.

The JC reagent is a sodium cyanide substitute for testing leach performance. Two additional tests were performed on Makosa and Makosa Tail roasted ore products to validate the experimental recovery. Recoveries are in Table 13.44.

Table 13.44 Cyanide test - Makosa and Makosa Tail

	Makosa	Makosa Tail
CN Test 1	81.3%	86.7%
CN Test 2	81.7%	88.2%

Source: NEU.

13.3.4 Flotation pre-concentration – roasting – leaching

Ore pre-concentration by flotation test work was undertaken to optimise gold recovery from ores (sulphidic) not amenable to the gravity / CIL process. The concentrate produced by flotation was roasted and leached using parameters determined for the whole-ore roast and leach process. Overall gold recoveries were determined using the parameters established for leaching of whole ore.

13.3.4.1 Flotation optimisation

Initial flotation test arrangement to establish feed particle size was single rougher followed by two scavenger stages. Pulp conditions and reagents were natural pH (7.82), butyl xanthate 100 g/t (collector), No.2 oil 60 g/t (pine oil frother), and rougher flotation time of five minutes.

Flotation test conditions are shown in Table 13.45. Reagent tests were conducted with a single rougher, dual scavenger configuration. Physical parameter tests were conducted with a single rougher, single cleaner configuration. Concentrates were ground prior to cyanide leaching to improve leach performance. 50 g of ground sample was used for the leach tests.

Table 13.45 Flotation optimisation test parameters

Parameter	Range
Grind size (% less than 74 µm)	60, 70, 80, 90
pH control – Na_2CO_3 dosage (g/t)	0.0, 500, 1000, 1500, 2000
Collector dosage (g/t)	50, 75, 100, 125, 150
Frother dosage (g/t)	20, 30, 40, 50, 60
Flotation time (min)	2, 3, 4, 5, 6

Source: NEU.

Standard flotation conditions for the optimisation tests are shown in Table 13.46.

Table 13.46 Flotation standard test parameters

Parameter	Value
Sample mass	500 g
Na_2CO_3	1,500 g/t
Collector (Butyl xanthate)	75 / 100 g/t
Temperature:	25°C
Frother (No.2 Pine Oil)	50 g/t
Grind size P_{80}	74 µm
Configuration	1x Rgh, 2x Scav.

Source: NEU.

Key observations from these tests on Makosa Tail ore were as follows:

- Recovery increased as the proportion of feed less than 74 µm increased to 90% (P_{90}) but Au grade reduced significantly between P_{70} of 74 µm and P_{80} of 74 µm. P_{80} of 74 µm was selected as optimal in this circumstance.
- Recovery maximised at Na_2CO_3 dosage of 1,500 g/t then decreased minimally at 2,000 g/t.
- Au recovery increased up to a collector dosage of 70 g/t then declined for all higher dosages.
- Au recovery increased up to a frother dosage of 50 g/t then stabilised for all higher dosages.
- Au recovery maximised at a rougher flotation time of 4 minutes. Increased flotation time did further increase recovery.

The final optimised flotation parameters were sample mass 500 g, Na_2CO_3 1,500 g/t, collector (Butyl xanthate) 75 g/t, temperature: 25°C, frother (No.2 Pine Oil) 50 g/t, grind size P_{80} of 74 µm, and flotation time 4 minutes.

Key observations from the tests on Makosa ore were as follows:

- Recovery increases as the proportion of feed less than 74 µm increased to 90% (P_{90}) but Au grade reduced significantly between P_{70} of 74 µm and P_{80} of 74 µm. P_{80} of 74 µm was selected as optimal in this instance.
- Recovery maximised at Na_2CO_3 dosage of 1,500 g/t then decreased minimally at 2,000 g/t.

- Au recovery plateaued between a collector dosage of 100 g/t and 125 g/t then declined for all higher dosages.
- Au recovery increased up to a frother dosage of 60 g/t with only a small increase from 50 g/t to 60 g/t.
- Au recovery maximises at a rougher flotation time of 10 minutes.

The final optimised flotation parameters were sample mass 500 g, Na_2CO_3 1,500 g/t, collector (Butyl xanthate) 100 g/t, temperature: 25°C, frother (No.2 Pine Oil) 50 g/t, grind size P_{80} of 74 µm, and flotation time eight minutes.

Further testing of a rougher, scavenger, cleaner circuit in both open circuit and closed-circuit configurations was done. Flotation parameters as above were used for open circuit testing and for the rougher stage of the closed-circuit test. The closed-circuit scavenger conditions were modified to butyl xanthate at 50 g/t, No.2 oil at 30 g/t, and a flotation time of 3 minutes for Makosa Tail ore, and for Makosa ore of 5 and 4 minutes for scavenger 1 and 2 respectively.

The results of the open and closed-circuit testing are shown in Table 13.47.

Makosa ore demonstrated higher flotation recovery when compared with Makosa Tail ore.

Table 13.47 Open and closed-circuit flotation tests

	Recovery (%)	Tails Loss (%)
Makosa Tail – Open Circuit	58.0	18.0
Makosa Tail – Closed-Circuit	80.85	18.15
Makosa – Open Circuit	65.4	15.6
Makosa – Closed-Circuit	84.2	15.8

Source: NEU.

13.3.4.2 Suspension roasting of flotation concentrate

The optimised flotation protocol produced concentrate with a gold grade of 7.13 g/t (Makosa Tail) and 6.4 g/t (Makosa). The elemental composition of the concentrates is in Table 13.48.

Table 13.48 Flotation concentrate analysis

Sample	SiO_2 (%)	Al_2O_3 (%)	MgO (%)	CaO (%)	TFe (%)	Na (%)	S (%)	Sb (%)	P (%)	K (%)	Au (g/t)	Ag (g/t)	C (%)	As (%)	Te (%)	LOI (%)
Makosa Tail	44.3	13.4	2.96	3.34	11.27	1.68	8.13	<0.01	0.061	1.64	7.13	1.46	2.1	0.82	<0.01	10.42
Makosa	39.7	14.81	1.75	1.45	11.35	0.67	10.0	<0.01	0.049	2.13	6.39	2.44	6.21	0.71	<0.01	16.77

Source: NEU.

Significant upgrading of Fe, S, As and Au was obtained. Minor upgrade of C also occurred. The increased carbon and sulphur content caused intense exothermic oxidation, impacted temperature control and caused sintering and secondary encapsulation of gold reducing leach recovery. To mitigate this response, additional roasting test work was completed at lower oxygen concentrations. All other optimal roast and leach parameters were maintained for these tests. Results are shown in Table 13.49 for Makosa Tail and Table 13.50 for Makosa ore.

Table 13.49 Makosa Tail Au recoveries for reduced oxygen concentration roasting

O_2 Concentration (%)	Product	Yield (%)	Au (g/t)	Au recovery (%)
10	Roasted conc.	100	7.82	82.6
	Leach residue	85.88	1.36	
15	Roasted conc.	100	7.93	80.83
	Leach residue	87.9	1.52	
21	Roasted conc.	100	7.78	76.6
	Leach residue	83.32	1.82	

Source: NEU.

Table 13.50 Makosa Au recoveries for reduced oxygen concentration roasting

O_2 Concentration (%)	Product	Yield (%)	Au (g/t)	Au recovery (%)
10	Roasted conc.	100	5.16	77.13
	Leach residue	90.83	1.18	
15	Roasted conc.	100	5.31	80.22
	Leach residue	93.35	1.05	
21	Roasted conc.	100	5.55	70.99
	Leach residue	91.28	1.61	

Source: NEU.

The optimal oxygen concentrations were determined to be 10% (Makosa Tail) and 15% (Makosa) for the roasting of flotation concentrates.

13.3.5 Beneficiation summary

Gold recoveries for each ore type and process are given in Table 13.51.

Table 13.51 Gold recoveries by ore and process option

Process option	Makosa Tail Fresh	Makosa Fresh
Suspension oxidation roasting cyanide leaching	88.90%	82.26%
Flotation pre-concentration, suspension oxidation roasting cyanide leaching	66.78%	67.53%

Source: NEU.

13.3.6 Supplementary sample tests -IMO samples

Testing was conducted according to ore type as shown in Table 13.52.

Table 13.52 IMO Sample test matrix

Sample	Ore type*	Whole ore Leach	Roast Leach	Flot. Roast Leach
MC2	MT Fresh		X	
MC3	M Fresh		X	
VC1	MT Oxide (SOX)	X		
VC2	MT Transitional (MOX)		X	
VC3	M Oxide (SOX)	X		
VC4	M Fresh		X	X
VC5	M Transitional (MOX+WOX)		X	
VC6	M Fresh		X	X
VC7	M Fresh		X	X

Note: *Ore type M= Makosa, MT = Makosa Tail.
Source: NEU.

Results of the tests are shown in Table 13.53.

Table 13.53 Final test results - IMO variability samples

Sample	Ore type*	Whole ore Leach		Roast Leach		Flot Roast Leach	
		Feed Au (g/t)	Rec (%)	Feed Au (g/t)	Rec (%)	Feed Au (g/t)	Fl_Rec|L_Rec (%)
MC2	MT Fresh			0.84	79.5		
MC3	M Fresh			1.26	74.1		
VC1	MT Oxide (SOX)	0.47	85.1				
VC2	MT Transitional (MOX)			0.9	90.0		
VC3	M Oxide (SOX)	0.78	84.6				
VC4	M Fresh			1.09	71.6	1.01	69.3|67.7
VC5	M Transitional (MOX+WOX)			0.84	86.9		
VC6	M Fresh			2.01	88.1	1.95	32.6|92.9
VC7	M Fresh			0.31	74.2	0.3	91.1|58.3

Note: *Ore type M = Makosa, MT = Makosa Tail.
Source: NEU.

Whole ore suspension oxidation roasting and leaching gave the highest gold recoveries for both ore types. Mineralogical analyses suggest that gold losses are primarily due to encapsulation in silicates preventing dissolution in the leaching stage. Roasting is only partially effective in unlocking this gold.

Flotation pre-concentration followed by roasting and leaching of the concentrate was less effective due to additional gold losses to flotation tails. Roasting stage recovery was similar to whole ore but when combined with flotation losses, overall recoveries were reduced to 66% to 68%.

13.3.7 Leaching, activated carbon adsorption, and loading

A sample of Makosa ore was tested for carbon adsorption performance. The sample was roasted and assayed for gold and carbon. The roasted ore was leached under standard conditions for 24 hours. Two samples each of residue and leach solution were assayed for gold. 100 mL of the leach solution was stirred with 20 g activated carbon for 30 minutes. The loaded carbon and barren leach solution were assayed in duplicate for gold. Results are in Table 13.54.

Table 13.54 Cyanide leaching–activated carbon adsorption test results

Test	Au (g/t) in roasted Makosa ore	Au (g/t) in Leach residue	Gold leach recovery	NaCN Consumption (kg/t)
1	1.2	0.19	84.17	0.3
2	1.2	0.19	84.17	0.3
	Au (gm^{-3}) in pregnant solution	**Au (gm^{-3}) in barren solution**	**Gold Loaded Carbon Au (g/t)**	**Gold adsorption (%)**
1	0.77	0.08	33.16	89.61
2	0.75	0.07	32.01	90.67

Source: NEU.

The results indicate that the two parallel tests had good repeatability, with a gold leaching rate of 84.17%, NaCN consumption of 0.30 kg/t, and gold adsorption rate of approximately 90%.

13.4 Ongoing test work

The following test work is ongoing at the time of reporting:

- Residue and leachate analysis for As, Cu, Pb, Zn, Cd, Fe, Bi, Sb from the CIL process.
- Determination of free and weak acid dissociable (WAD) cyanide in CIL tails.
- Acid Rock Drainage (ARD) testing on feed ores.
- Mineralogy on Douta West composites.

13.5 Recovery estimates

The recoveries determined from test work are set out in Table 13.55 for the differing ore types.

Table 13.55 Collated recoveries by ore type

Ore type	Orebody	Process	Recovery (%)
Oxide	All	CIL	92.5
Transitional	Makosa (Excluding East)	CIL	82.65
Transitional	Makosa East	CIL	72.8
Fresh	Makosa	Suspension roast - CIL	81.0
Fresh	Makosa Tail	Suspension roast - CIL	88.0

Source: NEU; IMO, 2025.

The recoveries were developed from test work completed at IMO and NEU. IMO tested samples supplied by Thor from selected drillholes. Samples were selected by ore types classified as oxide, transition, and fresh and by orebody location as in Table 13.56. Sub samples of two master composites of fresh ore type (MC2, MC3) were delivered to NEU, China for testing of the suspension roasting process.

IMO test work optimised CIL process parameters and recoveries for oxide and transition ore types. Oxide ores showed recoveries consistently more than 90% with low variation by orebody. All results were averaged to determine the final recovery. Transition ore types exhibited variation between Makosa and Makosa East locations. Recoveries were averaged for each of the two locations. IMO testing of fresh ore types gave low and variable results by CIL and these were excluded from the recovery data.

Fresh ores from Makosa and Makosa Tail were tested by suspension roasting and CIL extraction of the calcined product by NEU. Results were consistently higher than methods tested by IMO and have been utilised as recoveries for fresh ores in pit optimisations.

13.6 Sample representativity

Selection of samples for metallurgical testing was conducted based on geological domains and ore type (oxide, transition, fresh).

Table 13.56 Metallurgical samples - source and type

Composite type	Composite ID	Project zone	Ore type	Au grade (g/t)	Org C (%)	Sulphide (%)	As (ppm)
Master Composites	MC1	Makosa	Oxide (SOX)	1.77	0.14	0.001	883
	MC2	Makosa Tail	Fresh	0.84	0.32	1.19	1,371
	MC3	Makosa	Fresh	1.26	1.56	1.61	1,204
Variability Composites	VC1	Makosa Tail	Oxide (SOX)	0.57	0.1	0	1,919
	VC2	Makosa Tail	Transitional (MOX)	0.80	0.06	0.01	1,091
	VC3	Makosa	Oxide (SOX)	0.71	0.1	0.01	1,135
	VC4	Makosa	Fresh	1.62	1.8	1.71	2,209
	VC5	Makosa	Transitional (MOX+WOX)	0.70	0.59	0.02	427
	VC6	Makosa	Fresh	2.60	1.03	1.72	2,315
	VC7	Makosa	Fresh	0.51	0.64	0.78	208
RC Composite	Comp 1	Makosa South Shallow	Oxide (MOX)	2.47	0.03	0	1062
	Comp 2	Makosa South Deep	Fresh	2.31	0.17	2.66	2903
	Comp 3	Makosa Central Shallow	Oxide (MOX)	4.12	0.04	0.02	812
	Comp 4	Makosa South Deep	Oxide (MOX)	2.98	0.04	0	1632
	Comp 5	Makosa North Shallow	Oxide (MOX)	9.33	0.04	0	745
	Comp 6	Makosa North Deep	Oxide (MOX)	1.36	0.05	0	565
DD Composite	Comp 7	Makosa	Oxide (MOX)	2.39	0.07	0	1064
	Comp 8	Makosa	Fresh	1.22	0.18	2.56	2237
RC Composite	MKS1	Makosa East	Transition	1.27	0.05	0.04	550
	MKS2	Makosa	Transition	1.32	0.51	0.01	1099
	MKS3	Makosa North	Transition	1.41	0.21	0.3	1623
	MKS4	Makosa East	Transition	1.87	0.39	0	656

Source: Thor.

The majority of the samples were from the Makosa orebody. Few samples were selected from the Makosa Tail orebody and there were no samples from Baraka.

The master composites MC1, MC2, and MC3 were the only multiple drillhole composites designed to provide sufficient mass for extensive testing of process options. All other samples were prepared from single interval drillholes with the exception of Comp 8 which combined two separate intervals from a single drillhole.

The single drillhole samples were each selected to represent a single ore type.

Gold, organic carbon, arsenic and sulphide sulphur are the key elements influencing metallurgical behaviour. The observed atypical differences in metallurgical response within an ore type indicate that insufficient variations in the content of these elements have been included in the metallurgical sample suite.

The selection of an expanded testing sample set including increased variability of S, As, and C contents is recommended for future test programs.

13.7 Deleterious elements

The main deleterious element is arsenic. The grade ranges up to 3,000 ppm across all ore types due to the presence of arsenopyrite. Grades are lower in strongly oxidised ores and higher in fresh (sulphidic) ores.

The deportment of arsenic is controlled by the treatment process. Arsenic in oxide and transition ores treated by conventional CIL deports primarily to the residue and will be deposited in the Tailings Storage Facility (TSF) after ferric oxidation to As^{5+}. As^{5+} is stable with low leachability and is suitable for long term storage.

Fresh ores and remnant transition ores will be processed by thermal treatment and oxidation in a suspension roasting furnace. Arsenopyrite is oxidised and arsenic volatilised as As_2O_3 in the gas stream. Gases are scrubbed in a wet scrubbing system followed by electrostatic precipitation to capture remaining particulate matter. Scrubbing systems suitable for toxic gas removal are in use globally meeting emission standards for a range of gaseous emissions including arsenic emissions from copper smelters. The Douta process plant will implement a system to achieve emissions equivalent to or lower than comparable benchmarked operations.

13.8 Summary

The major outcomes of the metallurgical testing programs are summarised here:

- Oxide ores can be processed by conventional gravity and CIL which also mitigates against preg-robbing for those ores with organic carbon.
- Transition ores can be processed by CIL but further work is necessary to understand the variations observed.
- Fresh ores are not amenable to use of CIL processes and achieved poor recoveries even after flotation concentration and oxidative or high temperature treatment of the concentrates followed by cyanide leaching.
- Fresh ores can be treated by suspension roasting and CIL for gold recovery.

Gold distribution and fineness is variable between and within ore types as demonstrated by recovery responses. Oxidised ores have a higher proportion of coarser gold amenable to gravity recovery. Generally other ore types do not and display poorer gravity responses with the exception of two fresh ore samples which gave 12 and 20% recoveries. Transition ores have a more consistent response of approximately 5% gravity recoveries but show generally high CIL recoveries with one exception indicating that the gold is not locked in silicates or sulphides. The poor leaching recoveries of fresh ores which is improved by suspension roasting indicates that a proportion of the locked gold is both fine and recoverable.

13.9 Recommendations and further work

The following recommendations are made for further metallurgical work to advance the study to feasibility level.

- The variability in organic carbon content and silicates in the orebody and by ore types has not been quantified. Test work results demonstrate that variations in both occur in transition ores and fresh ores. Characterisation of the variability of carbon and silicates across the orebody is recommended.
- Additional metallurgical variability test work is required to expand the knowledge on recovery performance for both transition and fresh ore types. The recovery is impacted by gold distribution in arsenopyrite and silicates, and by the level of organic carbon.

- It is recommended that a variability suite of selected transition ore samples is characterised for their sulphides, organic carbon and silicate contents and tested for recovery by CIL.
- It is recommended that a variability suite of selected fresh ore samples is characterised for their sulphides, organic carbon, and silicate contents, and tested for recovery by suspension roasting and CIL.
- The selection of an expanded testing sample set including increased variability of S, As, and C contents is recommended for future test programs.

Gold distribution within each ore type and parallel assessment of silicate and sulphide content is recommended to refine the understanding of the best process options for each type. The observation of differences by ore location should also be further investigated as an input to mining plan and process schedule.

14 Mineral Resource estimates

14.1 Project summary

Mineral Resources for the Douta Project are estimated for five gold deposits and prospects located on the Douta Demande and the Douta West exploration permit (Figure 14.1). Separate blocks models cover the Makosa, Makosa Tail, and Baraka 3 deposits. The Makosa block model encompasses the Makosa, Makosa North, and Makosa East deposits. Mineral Resources are reported inclusive of Mineral Reserves at an effective date of 24 January 2026 (Table 14.1).

The methods, parameters, assumptions, and support data used for the Douta block models, which date back to 2023, were reviewed to ensure they remain current. Models have been updated as required to either include new information or revised cost assumptions such as gold price and operation costs.

The same overall approach was used for each model whereby block grade and density estimates are constrained by domains representing the mineralisation, lithology, and weathering surfaces. Mineral Resources are reported within pit shells generated by AMC. Only classified blocks greater than or equal to the open pit cut-off grades and within the open pit shells are reported.

QP for the Mineral Resource estimates is Alfred Gillman, General Manager (Exploration and Resources) for Thor and is not independent. Mr Gillman is a QP in accordance with NI 43-101.

Table 14.1 Douta Project Mineral Resources as of October 2025

Deposit	Classification	Volume (Mm^3)	Average density	Tonnage (Mt)	Grade (g/t Au)	Ounces MAu
Makosa North	Indicated	3.67	2.69	9.90	1.08	0.34
	Inferred	1.73	2.79	4.80	1.02	0.16
	Ind + Inf	5.4	2.74	14.70	1.05	0.5
Makosa	Indicated	7.62	2.71	20.60	1.06	0.7
	Inferred	0.55	2.72	1.50	0.95	0.05
	Ind + Inf	8.17	2.72	22.10	1.00	0.75
Makosa East	Indicated	3.11	2.67	8.30	0.92	0.25
	Inferred	0.50	2.68	1.30	0.87	0.04
	Ind + Inf	3.61	2.68	9.60	0.89	0.29
Makosa Tail	Indicated	3.92	2.71	10.60	1.03	0.35
	Inferred	0.52	2.67	1.40	0.57	0.03
	Ind + Inf	4.44	2.69	12.00	0.80	0.38
Baraka 3	Indicated	0.54	2.11	1.10	1.43	0.05
	Inferred	0.09	2.02	0.20	0.99	0.01
	Ind + Inf	0.63	2.07	1.30	1.21	0.06
Total	**Indicated**	**18.86**	**2.58**	**50.50**	**1.10**	**0.34**
	Inferred	**3.39**	**2.58**	**9.20**	**0.88**	**0.06**

Notes:
- CIM Definition Standards were followed for Mineral Resources.
- Mineral Resources are inclusive of Mineral Reserves.
- Mineral Resources have been constrained by optimised pit shells based on a gold price of US$4,000/oz.
- Resources are reported inclusive of Reserve.
- Calculated breakeven cut-off grades based on mining costs, metallurgical recovery, milling costs, and G&A costs range between 0.21 g/t to 0.33 g/t across the various deposits.
- Open pit Mineral Resources are reported in situ at a cut-off grade of 0.30 g/t Au which is the average of the individual calculated breakeven cut-offs.
- The Mineral Resource is considered to have reasonable prospects for economic extraction by open pit mining methods above a 0.30 g/t Au and within an optimised pit shell.

- High grade assays were capped at 10 g/t Au.
- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
- Bulk density is assigned according to weathering profile with a weighted average of 2.71.
- Effective date for the Mineral Resource is 24 January 2026.
- Sum of individual amounts may not equal due to rounding.

Source: Thor, Oct 2025.

Table 14.2 Douta Project Mineral Resource estimate, tonnage by weathering zone – Indicated classification

Classification	Deposit	Tonnage (M)				
		SOX	MOX	WOX	FRS	Total
Indicated	Makosa	1.24	6.93	5.79	24.87	38.82
Indicated	Makosa Tail	0.06	1.68	1.32	7.56	10.62
Indicated	Baraka 3 East	0.01	0.22	0.09	0.07	0.40
Indicated	Baraka 3 West	0.03	0.52	0.12	0.09	0.75
Indicated	**Total**	**1.34**	**9.35**	**7.31**	**32.59**	**50.59**
Percentage		3%	18%	14%	64%	

Source: Thor, October 2025.

Table 14.3 Douta Project Mineral Resource estimate, ounces gold by weathering zone – Indicated classification

Classification	Deposit	Ounces (x1000)				
		SOX	MOX	WOX	FRS	Total
Indicated	Makosa	43.3	226.7	188.3	833.7	1,292.0
Indicated	Makosa Tail	1.4	47.5	39.5	264.9	353.3
Indicated	Baraka 3 East	1.1	14.3	5.4	4.3	25.1
Indicated	Baraka 3 West	1.0	19.6	3.7	3.3	27.6
Indicated	**Total**	**46.9**	**308.0**	**236.9**	**1,106.2**	**1,698.0**
Percentage		3%	18%	14%	65%	

Source: Thor, October 2025.

Table 14.4 Douta Project Mineral Resource estimate, tonnage by weathering zone – Inferred classification

Classification	Deposit	Tonnage (M)				
		SOX	MOX	WOX	FRS	Total
Inferred	Makosa	0.06	0.69	0.39	6.54	7.68
Inferred	Makosa Tail	0.03	0.31	0.23	0.82	1.39
Inferred	Baraka 3 East	0.00	0.00	0.00	0.00	0.00
Inferred	Baraka 3 West	0.01	0.16	0.01	0.00	0.19
Inferred	**Total**	**0.10**	**1.17**	**0.63**	**7.36**	**9.26**
Percentage		1%	13%	7%	79%	

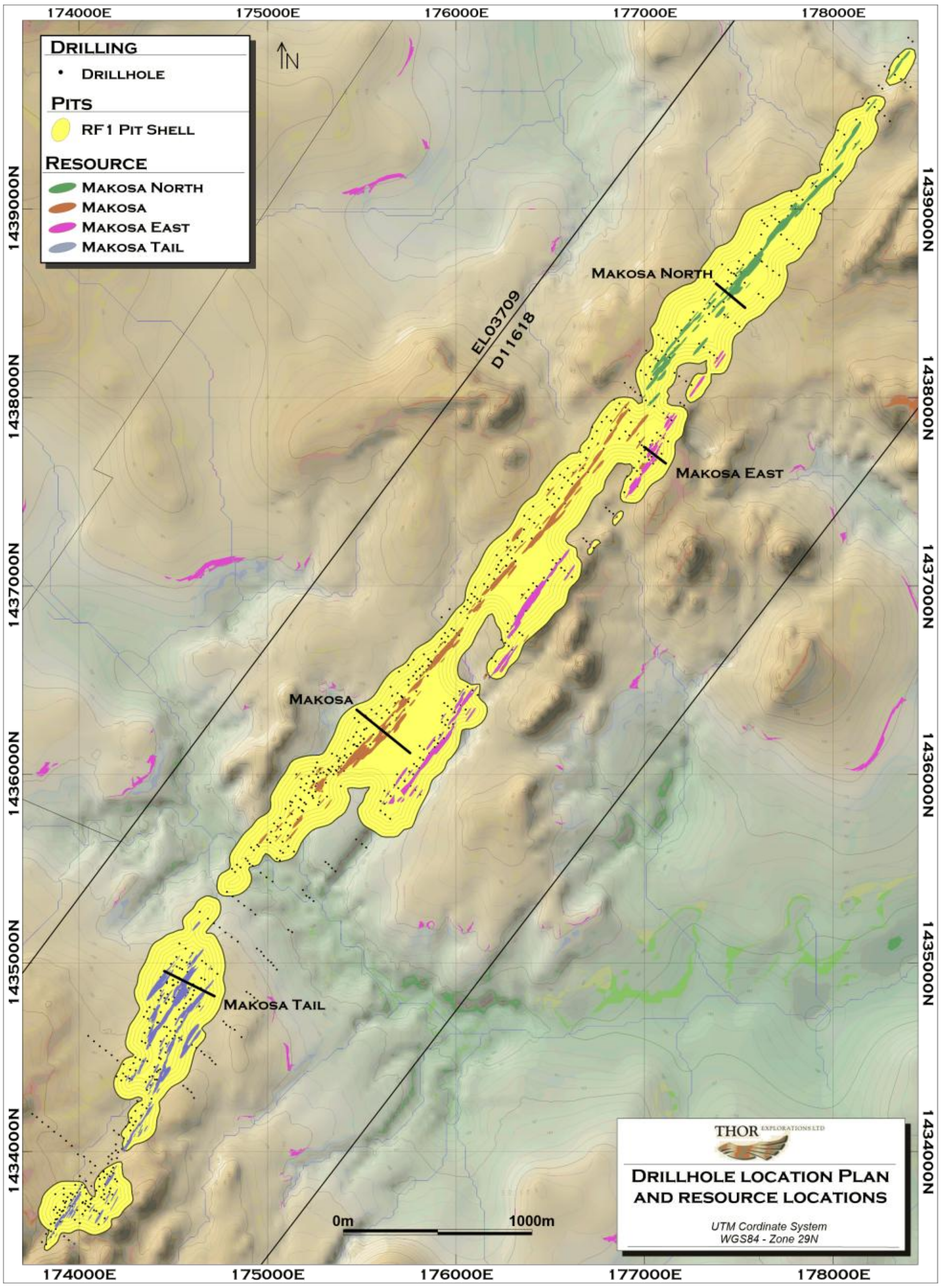
Source: Thor, October 2025.

Table 14.5 Douta Project Mineral Resource estimate, ounces gold by weathering zone – Inferred classification

Classification	Deposit	Ounces (x1000)				
		SOX	MOX	WOX	FRS	Total
Inferred	Makosa	1.1	15.2	9.6	215.5	241.4
Inferred	Makosa Tail	0.4	5.1	4.6	15.3	25.4
Inferred	Baraka 3 East	0.0	0.0	0.0	0.1	0.1
Inferred	Baraka 3 West	0.5	5.1	0.3	0.0	5.9
Inferred	**Total**	**2.0**	**25.4**	**14.5**	**231.0**	**272.9**
Percentage		1%	9%	5%	85%	

Source: Thor, October 2025.

Figure 14.1 Location of deposits containing Mineral Resources



Source: Thor, October 2025.

14.2 Resource database

For each deposit, a sub-set of the master drillhole database was created including tables for: collar coordinates, downhole azimuth and dip direction, lithology, assays, and density data. Table 14.6 summarises the total drilling in individual resource databases by drillhole type.

The block models used mostly reverse circulation drilling to estimate block grades.

Table 14.6 Summary of drillhole database by drillhole type

Deposit / prospect	Category	No. holes		No. metres		Total No.	Total metres
		RC	DD	RC	DD		
Makosa East	Resource	187		10,567		187	10,567
Makosa North	Resource	217	1	15,206	246	218	15,452
Makosa Tail	Resource	225	20	17,544	2,252	245	19,796
Baraka 3	Resource	329	3	20,409	438	332	20,847
Total	**Resource**	**958**	**24**	**63,726**	**2,936**	**982**	**66,662**

Source: Thor, October 2025.

14.3 Bulk density measurements

Bulk density was determined by Thor technicians on core samples using the water-immersion method. Porous or absorbent samples were coated with wax after obtaining an initial weight in air, then immersed in water and weighed again.

Bulk density control samples were used as QC checks for sample density determinations. Densities were measured on a control sample before the first and after the last sample density measurement of each hole.

The bulk density (tonnage factor) assignments were based on a total of 1,119 water-immersion -half core measurements (Table 14.7).

Table 14.7 Bulk density statistics

Weathering zone	Makosa Area		Baraka 3 Area	
	Number	Average	Number	Average
SOX	6	2.40	-	-
MOX	14	2.50	13	2.00
WOX	218	2.59	20	2.18
FRS	769	2.78	79	2.71
Total	**1,007**		**112**	

Source: Thor, October 2025.

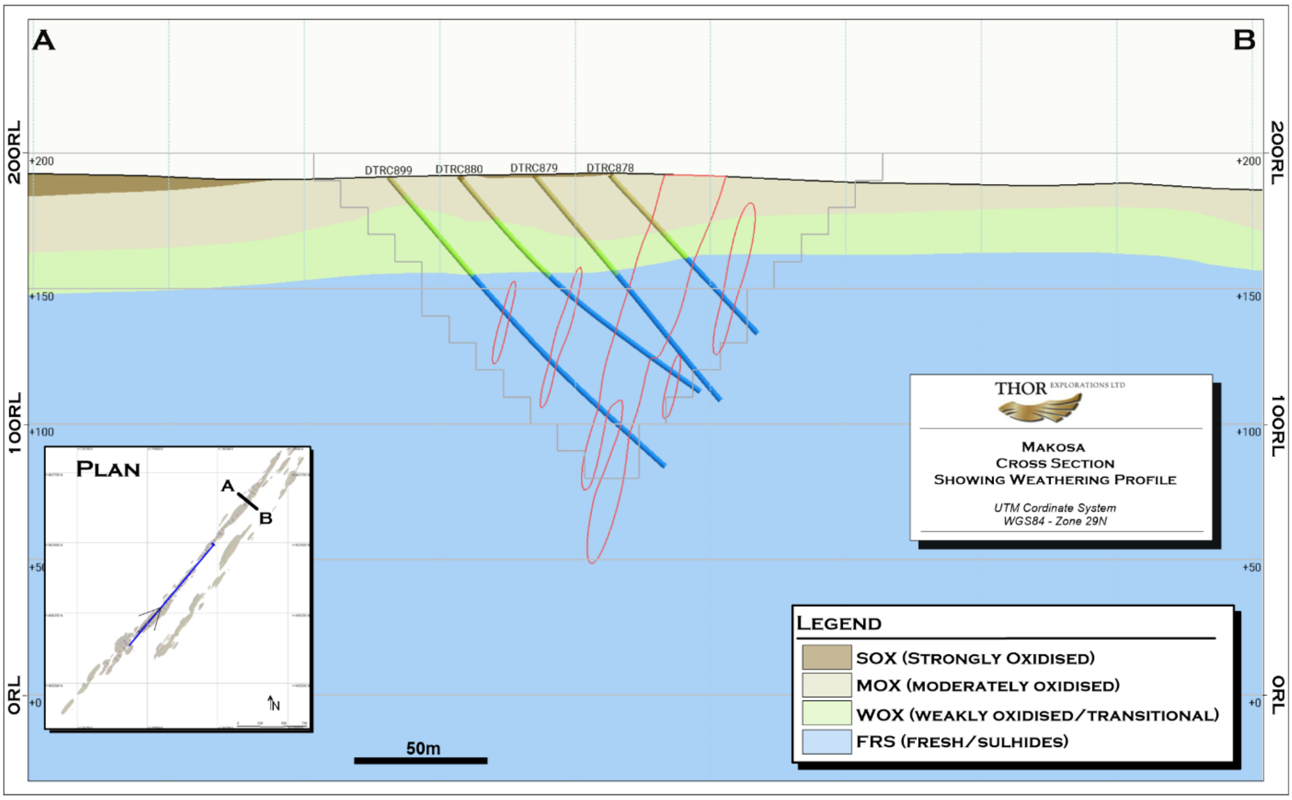
A validation exercise was run to verify measurements obtained on site on core samples. Eighteen samples were submitted for comparative measurements to ALS Johannesburg using the OA-GRA08a method. The variances between the two data sets are negligible (Table 14.8).

Table 14.8 Bulk density comparisons

Hole ID	From (m)	To (m)	Interval (m)	Zone	ALS	Thor	Variance	Variance %
DTDD0001	45.78	46.00	0.22	Fresh	2.70	2.69	0.01	0.20
DTDD0001	56.89	57.06	0.17	Fresh	2.75	2.73	0.02	0.80
DTDD0001	71.73	71.87	0.14	Fresh	2.73	2.71	0.02	0.60
DTDD0001	74.05	74.17	0.12	Fresh	2.73	2.71	0.02	0.70
DTDD0001	82.15	82.32	0.17	Fresh	2.70	2.72	-0.02	-0.60
DTDD0001	83.73	83.85	0.12	Fresh	2.76	2.53	0.23	8.40
DTDD0002	78.70	78.87	0.17	Fresh	2.66	2.73	-0.07	-2.60
DTDD0002	81.22	81.37	0.15	Fresh	2.60	2.37	0.23	9.00
DTDD0002	97.50	97.70	0.2	Fresh	2.62	2.61	0.01	0.50
DTDD0002	106.55	106.69	0.14	Fresh	2.65	2.54	0.11	4.10
DTDD0002	110.64	110.80	0.16	Fresh	2.68	2.74	-0.06	-2.20
DTDD0002	139.80	139.93	0.13	Fresh	2.75	2.82	-0.07	-2.60
DTDD0006	126.34	126.55	0.21	Fresh	2.71	2.74	-0.03	-1.00
DTDD0011	81.09	81.20	0.11	Fresh	2.78	2.81	-0.03	-1.00
DTDD0011	99.60	99.73	0.13	Fresh	2.47	2.35	0.12	4.90
DTDD0011	100.19	100.43	0.24	Fresh	2.72	2.73	-0.01	-0.50
DTDD0012	46.51	46.66	0.15	Fresh	2.66	2.74	-0.08	-3.00
DTDD0012	104.00	104.12	0.12	Fresh	2.65	2.56	0.09	3.50
Average					**2.68**	**2.66**	**0.02**	**0.60**

Source: Thor, October 2025.

Figure 14.2 Cross section through Makosa showing weathering profile (looking NE)



Source: Thor, October 2025.

14.4 Geological and mineralisation models

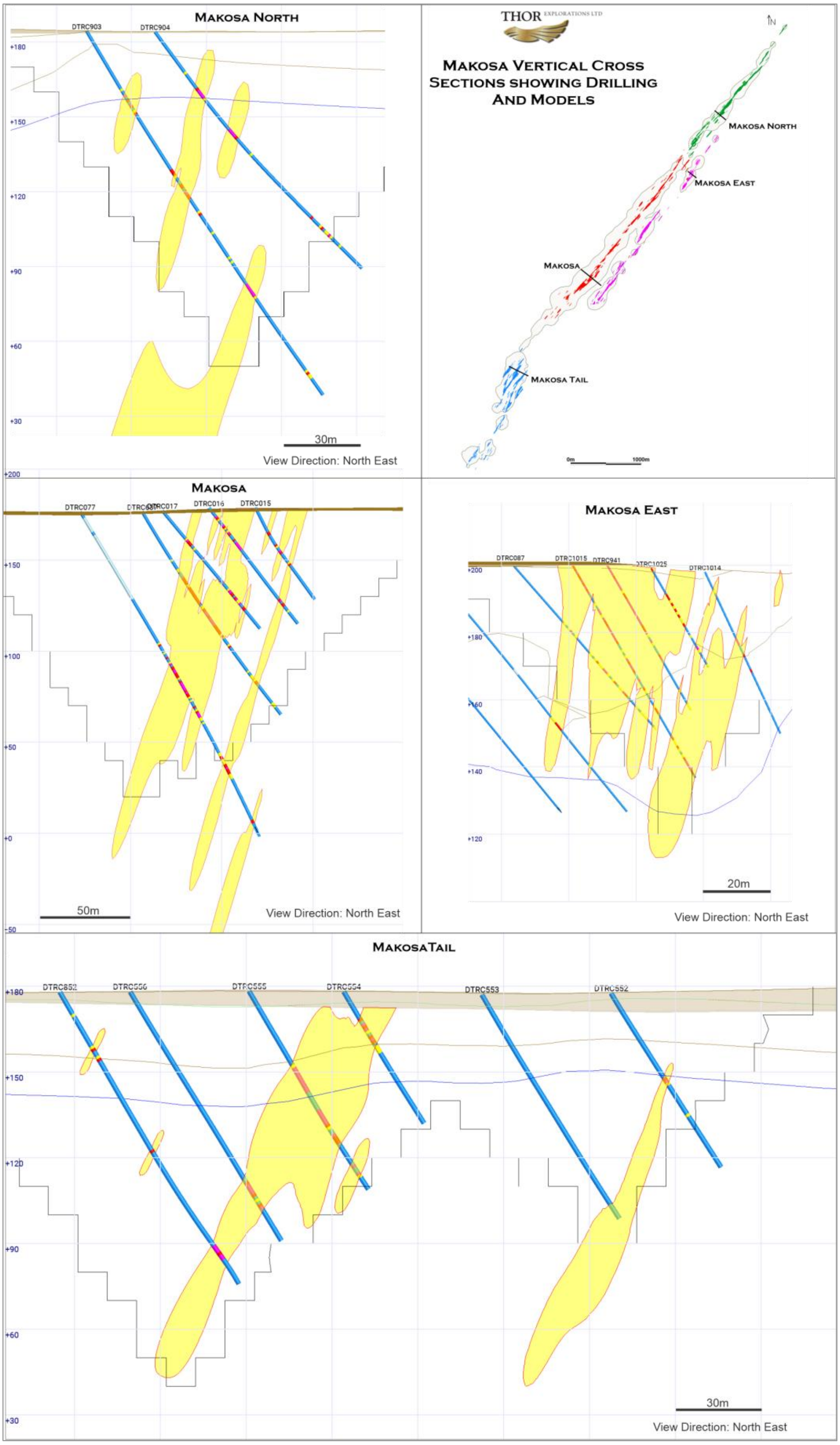
Mineralisation wireframes were interpreted and generated in Seequent Leapfrog. The geological models, including the mineralisation wireframes, were generated by the QP.

14.4.1 Makosa

A topographic surface was generated from a LIDAR survey that was undertaken by Thor in 2024. Drillhole collars were pressed to this surface. Lithology models were generated for the gabbro, greywacke, and shale units. An oxidation surface was constructed from points representing the base of the weathered rock profile in each drillhole. Oxide and “fresh” (unoxidised) rock solids were generated.

Grades are constrained by discrete domains that are derived using Leapfrog’s indicator radial basis function (Indicator RBF). A steep north-westerly trend was applied (Figure 14.3).

Figure 14.3 Cross section through Makosa showing resource wireframes (looking NE)



Source: Thor, October 2025.

14.4.2 Baraka 3

A topographic surface was generated from a LIDAR survey that was undertaken by Thor in 2025. Drillhole collars were pressed to this surface. An oxidation surface was constructed from points representing the base of the weathered rock profile in each drillhole. Oxide and "fresh" (unoxidised) rock solids were generated.

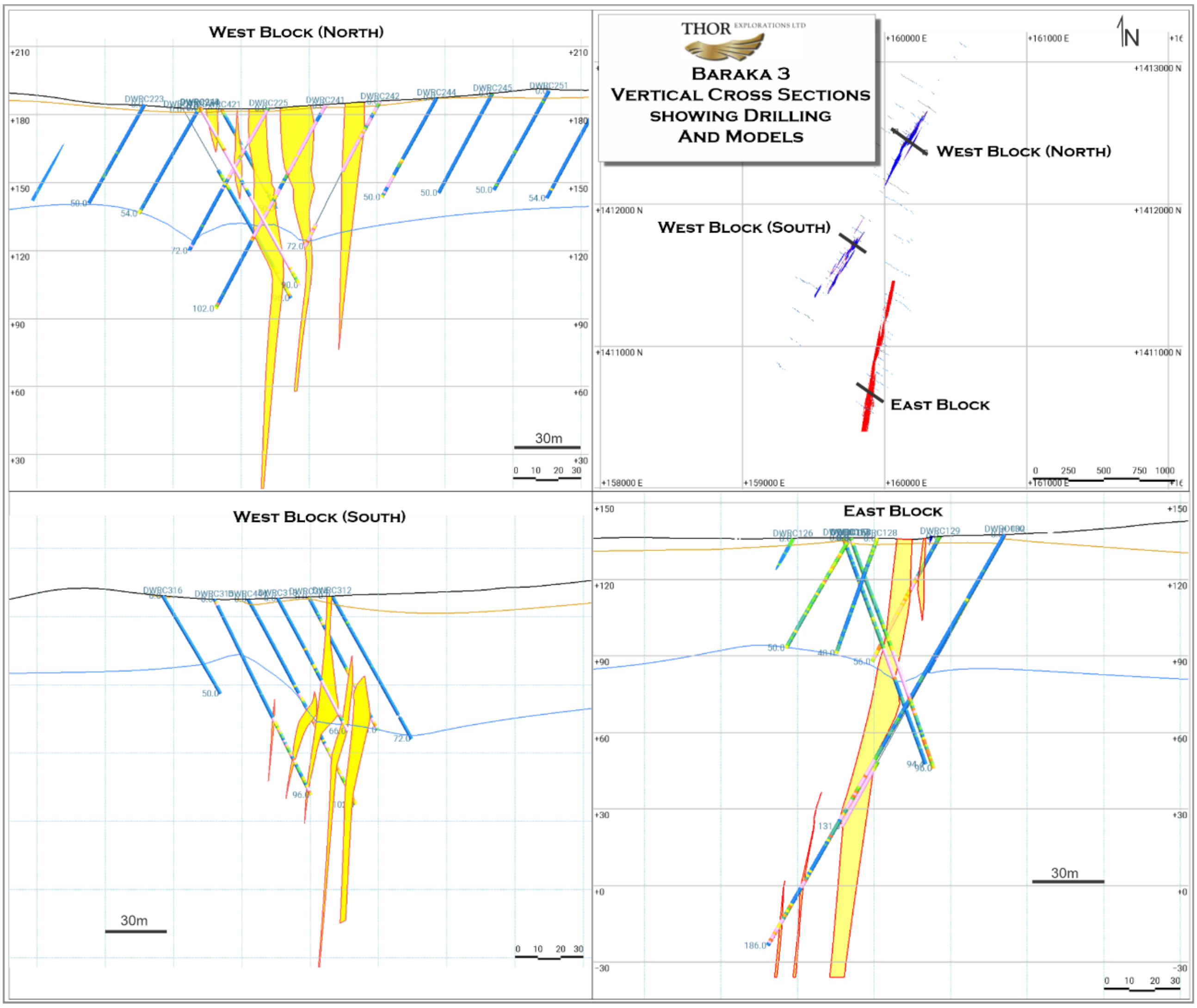
Grades are constrained by discrete domains that are derived using Leapfrog's Indicator RBF. A steep north-westerly trend was applied (Figure 14.4).

Baraka 3 is characterised by two main trends:

- A northerly trend through the East Block.
- A north-easterly trend through the West Block.

Gold mineralisation in both blocks is developed in steep-dipping to vertical parallel lodes. Gold is associated with stockwork-style quartz veining. Drilling to date has not closed off the lodes at depth.

Figure 14.4 Cross section through Barak 3 showing resource wireframes



Source: Thor, October 2025.

14.5 Assay statistics

Descriptive statistics for the raw gold assays within the mineralisation domains were generated for each deposit. The results are presented in Table 14.9.

Table 14.9 Assay statistics

Parameter	Makosa	Makosa East	Makosa North	Makosa Tail	Baraka 3
Count	28,819	12,753	14,057	21,500	14,772
Maximum	64.6	20.9	17.6	59.1	73.2
Minimum	0	0	0	0	0.01
Mean	0.22	0.15	0.16	0.18	0.15
Q1	0.01	0.01	0.01	0.01	0.01
Q3	0.15	0.09	0.1	0.07	0.04
SD	0.89	0.47	0.48	1.08	0.88
CV	4.12	3.12	3.03	5.98	5.85

Source: Thor, October 2025.

14.6 Managing high grade samples

Where the distribution of assay grades is skewed positively or approaches log-normal, erratic high-grade assay values can have a disproportionate effect on the average grade of a deposit. To reduce the effect of these outliers on the average grade, their influence was reduced by cutting or capping at a specific grade level.

For most deposits, capping levels were determined and applied to each mineralisation domain prior to compositing. Composites were capped, instead of raw assays.

A capping level was established using a combination of histograms and probability plots.

A capping level (top cut) of 10 g/t Au is applied for all the Makosa domains. An 8 g/t Au top cut was applied to the Baraka 3 grade estimates. The number of composites affected by the application of top cuts is listed in Table 14.10 and Table 14.11.

Table 14.10 Number of composites affected by the application of top cutting – Makosa

Area	Makosa			Makosa North	Makosa East			Makosa Tail				
Domain	MK01	MK02	MK03	MKN	MKE01	MKE02	MKE03	MT01	MT02	MT03	MT04	MT05
Top Cut	10	10	10	10	10	10	10	10	10	10	10	10
No. Comps Cut	2	18	3	2	2	0	0	4	7	1	0	8

Source: Thor, October 2025.

Table 14.11 Number of composites affected by the application of top cutting – Baraka 3

Area	Makosa East						Makosa Tail			
Domain	100	200	300	400	500	600	N001	N002	N003	N004-N010
Top Cut	8	8	8	8	8	8	8	8	8	8
No. Comps Cut	0	5	7	1	1	0	3	0	2	0

Source: Thor, October 2025.

14.7 Composite samples

Composites were extracted from within the domain boundaries. In cases where the distance between domain boundaries produced unequal composite lengths, the actual composite lengths for the respective drillhole intersection were distributed equally and thus resulted in no residuals. Composite statistics for the deposits that contain Mineral Reserves are listed in Table 14.12 and Table 14.13.

Table 14.12 Composite statistics - Makosa

Area	Makosa			Makosa North	Makosa East			Makosa Tail				
Parameter	**MK01**	**MK02**	**MK03**	**MKN**	**MKE01**	**MKE02**	**MKE03**	**MT01**	**MT02**	**MT03**	**MT04**	**MT05**
Count	766	2,009	1,038	1,299	479	432	561	1,383	455	879	280	168
Maximum	15.8	48.4	20.2	14.9	20.4	5.9	7.6	21.0	50.6	13.8	4.7	51.1
Minimum	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.1
Mean	1.1	1.3	1.0	1.1	1.0	0.9	0.9	0.9	1.5	0.9	0.7	2.3
Q1	0.5	0.5	0.5	0.5	0.4	0.4	0.4	0.3	0.4	0.3	0.2	0.3
Q3	1.2	1.4	1.2	1.3	1.1	1.1	1.1	1.0	1.3	1.1	0.8	1.2
CV	1.1	1.8	1.1	1.0	1.5	0.9	0.9	1.6	2.8	1.3	1.2	2.6
Variance	1.3	5.1	1.3	1.2	2.1	0.6	0.7	2.3	17.5	1.5	0.7	35.5
Length	0.91	0.94	0.93	0.94	0.94	0.96	0.93	0.93	0.92	0.93	0.89	0.89

Source: Thor, October 2025.

Table 14.13 Composite Statistics - Baraka 3

Block	East					West								
Parameter	**100**	**200**	**300**	**400**	**500**	**N001**	**N002**	**N003**	**N004**	**N005**	**N006**	**N007**	**N008**	**N009**
Count	128	215	134	63	6	385	89	66	76	23	23	35	11	18
Maximum	7.1	16.2	73.2	9.6	9.6	11.6	7.1	15.9	3.1	8.4	2.0	6.6	2.7	2.2
Minimum	0.0	0.0	0.0	0.1	0.3	0.0	0.0	0.0	0.1	0.1	0.1	0.1	0.1	0.1
Mean	1.3	1.9	2.6	1.5	2.1	1.0	0.9	1.1	0.7	1.3	0.8	1.4	1.1	0.8
Q1	0.3	0.3	0.4	0.4	0.3	0.5	0.3	0.3	0.3	0.3	0.3	0.3	0.4	0.4
Q3	1.9	2.8	2.7	2.0	1.0	1.2	1.0	1.0	0.8	1.1	1.2	2.6	1.7	1.1
SD	1.3	2.4	6.6	1.9	3.7	1.1	1.1	2.2	0.7	1.9	0.6	1.7	0.9	0.5
CV	1.0	1.3	2.6	1.2	1.7	1.1	1.3	2.0	0.9	1.5	0.7	1.2	0.8	0.7
Variance	1.8	6.0	44.1	3.7	13.6	1.3	1.3	5.0	0.4	3.6	0.3	2.8	0.8	0.3

Source: Thor, October 2025.

14.8 Block model parameters

Mineral Resources for the Douta Project are reported from four individual sub-blocked block models built using Leapfrog Edge. The methodology for each model is identical in that block grade and density estimates are constrained using domains representing the mineralisation, lithology, and weathering surfaces.

The block models are rotated to better align with the general mineralisation trend directions. The block model that covers Makosa, Makosa North, and Makosa East was constructed along a north-east (040°) orientation. The Makosa Tail block was constructed along a north north-east (030°) orientation (Figure 14.5).

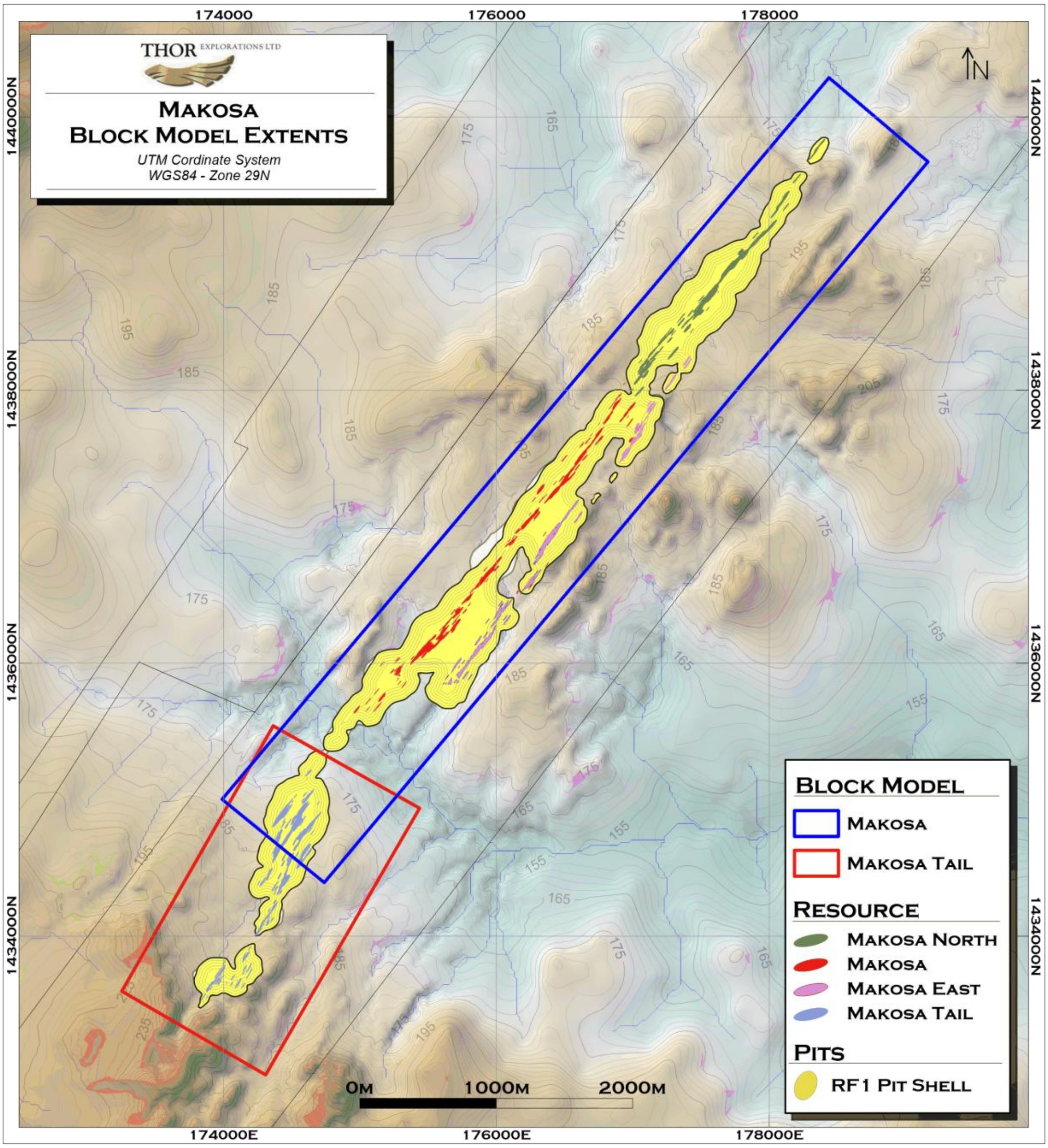
At Baraka 3, two block models were constructed: an Eastern model that is oriented north-south (010°) and a Western model oriented at north-east (030°; Figure 14.6). Block model parameters are listed in Table 14.14.

Table 14.14 Block model parameters (WGM 84 UTM Zone 29N)

Deposit	Block model origin (m)			Boundary size (m)			Azimuth	Block size (m)					
								Parent block			Sub-block		
	X	Y	Z	X	Y	Z		X	Y	Z	X	Y	Z
Makosa	174,000	1,435,000	300	950	6,900	402	40	5	20	6	1.25	5	1.5
Makosa North	174,000	1,435,000	300	950	6,900	402	40	5	20	6	1.25	5	1.5
Makosa East	174,000	1,435,000	300	950	6,900	402	40	5	20	6	1.25	5	1.5
Makosa Tail	173,250	1,433,600	300	1,230	2,160	402	30	5	20	6	1.25	5	1.5
Baraka 3 East	159,670	1,411,305	200	365	1,760	237	10	2.5	5	3	0.625	1.25	0.75
Baraka 3 West	159,370	1,410,350	200	320	995	225	30	2.5	5	3	0.625	1.25	0.75

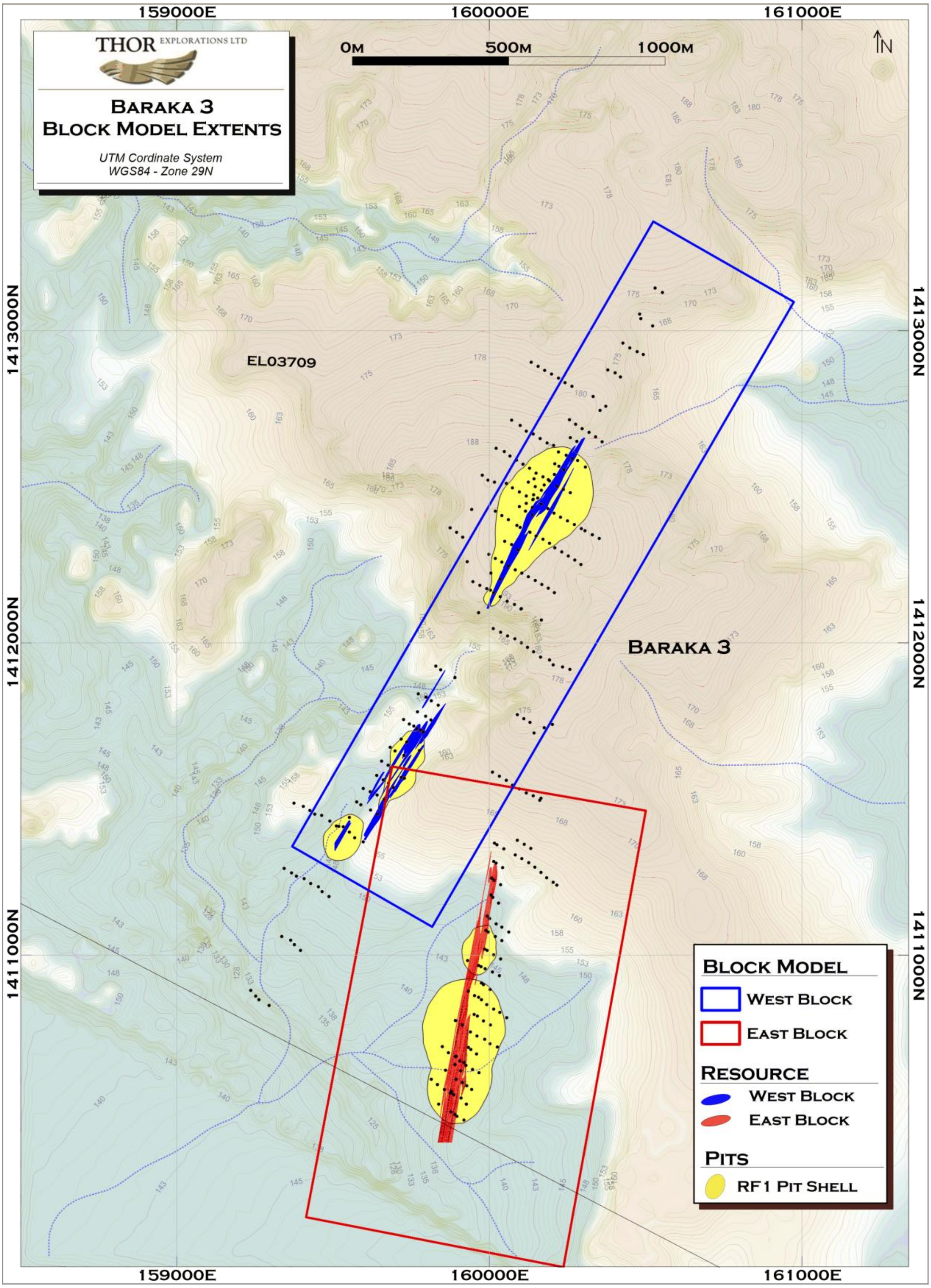
Source: Thor, October 2025.

Figure 14.5 Makosa block model extents



Source: Thor, October 2025.

Figure 14.6 Baraka 3 block model extents



Source: Thor, October 2025.

14.9 Grade estimation

Block grade estimation methods and parameters are summarised in Table 14.15. All estimates used a single-pass approach with sufficiently large search ellipses to ensure all blocks within the domains were estimated with gold grades. Only diamond core and reverse circulation drilling data was used to estimate block grades. All estimations used the inversed distance squared (ID^2) interpolation method. The Ordinary Kriging (OK) interpolation was used to compare with the ID^2 method. Variogram models were generated for use in domains estimated using the OK method. Model parameters are summarised in Table 14.15.

Table 14.15 Grade estimation methods and parameters

Deposit	Domain	Method	Ellipsoid ranges			Ellipsoid directions			Number of samples	
			Max.	Inter.	Min.	Dip	Dip Azi.	Pitch	Minimum	Maximum
Makosa	MK01	ID^2	100	160	20	72	310	90	4	8
	MK02	ID^2	100	160	20	75	309	90	4	8
	MK03	ID^2	100	160	20	75	309	90	4	8
Makosa North	MKN	ID^2	100	160	20	75	309	90	4	8
Makosa East	MKE01	ID^2	100	160	20	70	306	90	4	8
	MKE02	ID^2	100	160	20	70	306	90	4	8
	MKE03	ID^2	100	160	20	70	306	90	4	8
Makosa Tail	MT01	ID^2	100	60	20	75	300	90	4	8
	MT02	ID^2	100	60	20	75	300	90	4	8
	MT03	ID^2	100	60	20	75	300	90	4	8
	MT04	ID^2	100	60	20	75	300	90	4	8
	MT05	ID^2	100	60	20	75	300	90	4	8
Baraka 3 East	100	ID^2	100	100	10	85	280	90	4	8
	200	ID^2	100	100	10	85	280	90	4	8
	300	ID^2	100	100	10	85	280	90	4	8
	400	ID^2	100	100	10	85	280	90	4	8
Baraka 3 West	N001	ID^2	100	100	10	88	90	90	2	8
	N002	ID^2	100	100	10	88	90	90	2	8
	N003	ID^2	100	100	10	88	90	90	2	8
	N004	ID^2	100	100	10	88	90	90	2	8
	N005	ID^2	100	100	10	88	90	90	2	8
	N006	ID^2	100	100	10	88	90	90	2	8
	N007	ID^2	100	100	10	88	90	90	2	8
	N008	ID^2	100	100	10	88	90	90	2	8
	N009	ID^2	100	100	10	88	90	90	2	8

Source: Thor, October 2025.

14.10 Bulk density models

Weathering surfaces and solids were constructed for the Makosa and Baraka 3 areas by creating a stratigraphic sequence model using the database weathering codes. The weathering surfaces / solids were used as hard boundaries with respect to density assignments. The weathering model was evaluated onto the respective block models.

Each weathering zone was assigned a uniform bulk density for both mineralised and unmineralised resource portions, irrespective of lithology (see Table 14.16).

Table 14.16 Density assignments

Weathering zone	Makosa area	Baraka 3 area
SOX	2.40	2.00
MOX	2.50	2.00
WOX	2.60	2.20
FRS	2.80	2.70

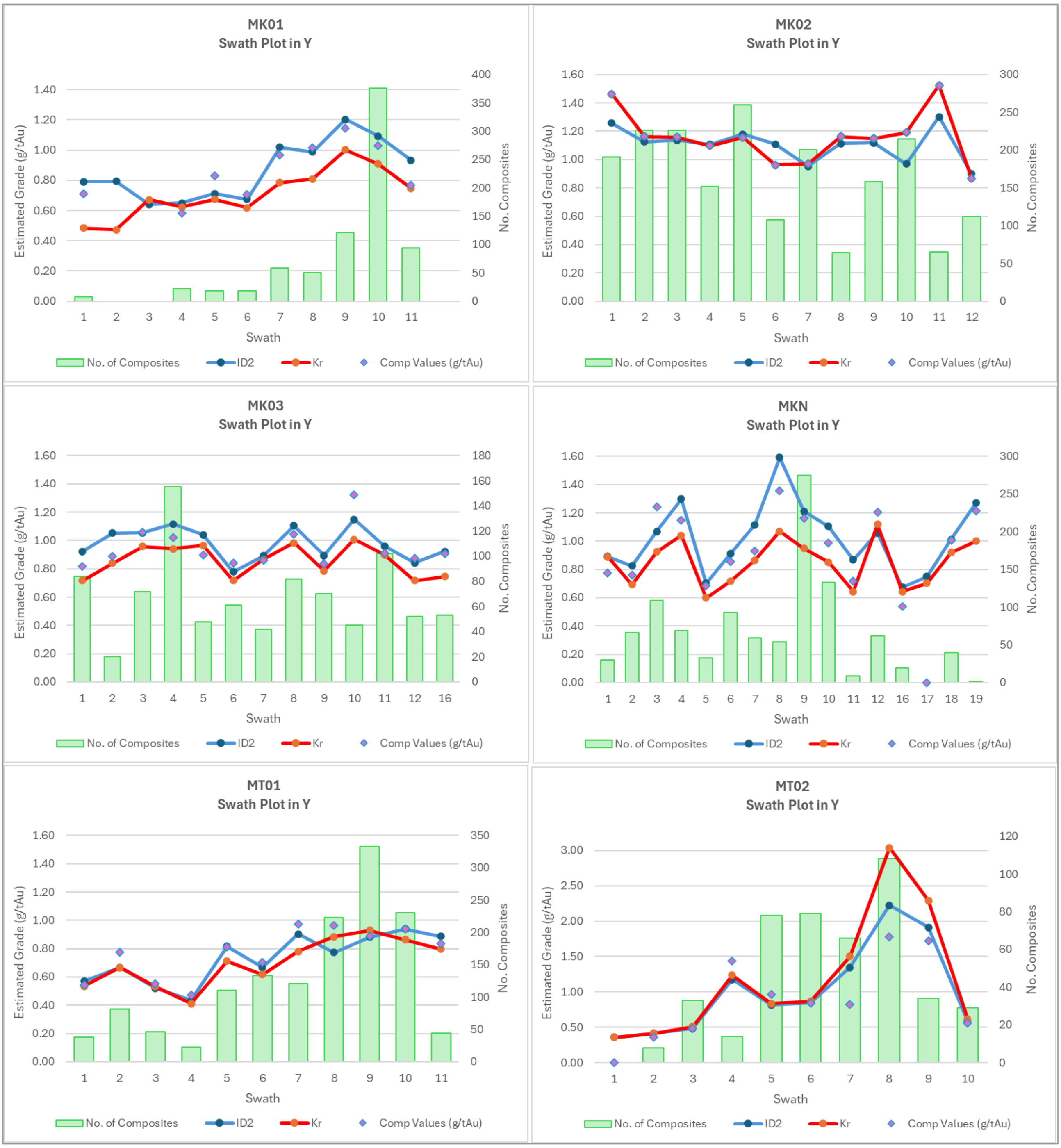
Source: Thor, October 2025.

14.11 Block grade validation

Block model grade validation consisted of a visual validation and a comparison of the average block grade to the average composite grade by domain. Visual validation comparing assay and composite grades to block grade estimates showed reasonable correlation with no significant overestimation or overextended influence of high grades.

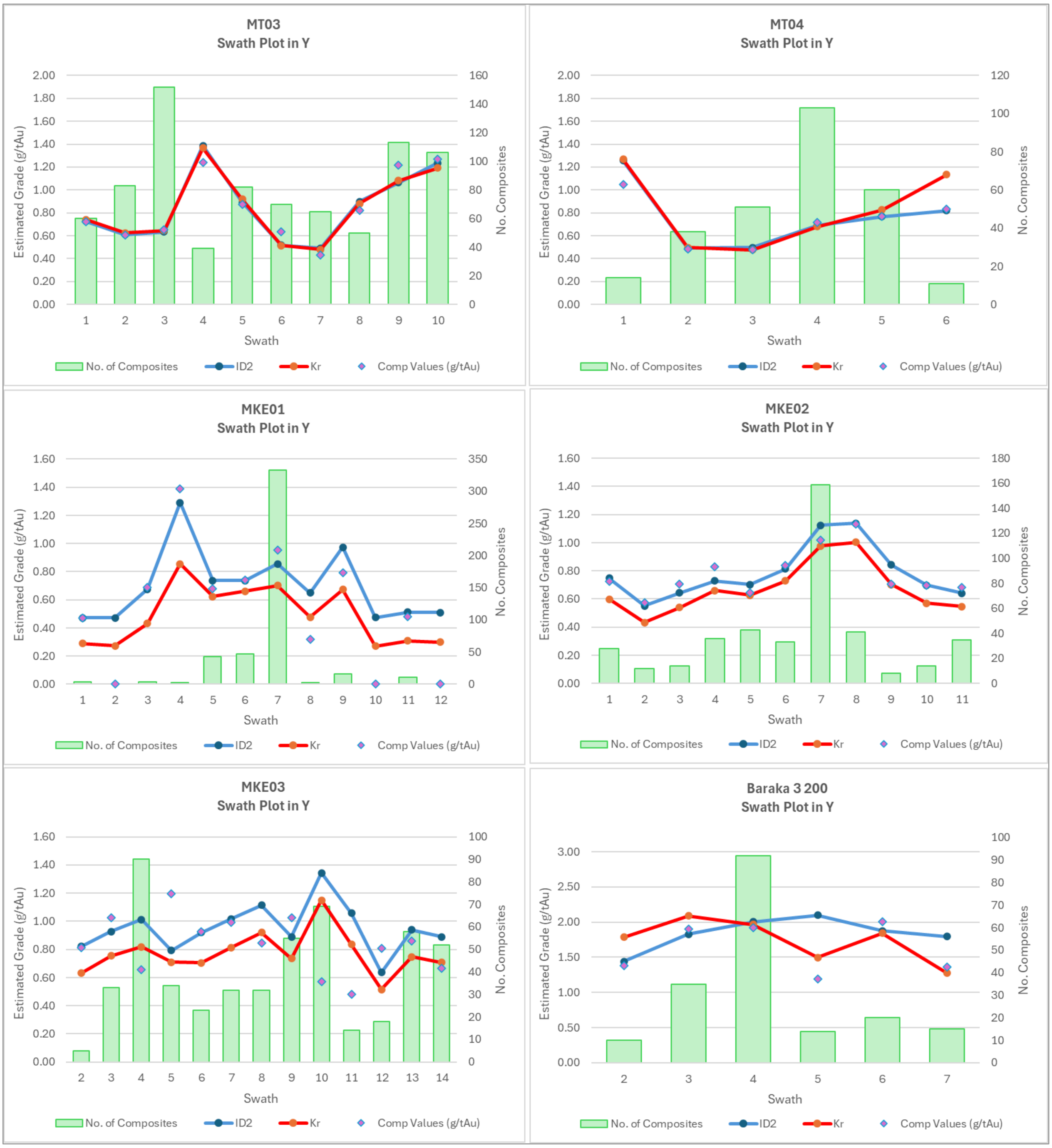
Swath plots in the north-south 'Y' axis were generated for most deposits to compare average composite grades to average block grades. Local average composites are generally more variable than average block grades. However, the swath plots demonstrate a reasonable correlation between the composite grades and block grade estimates. Comparisons were also made using ordinary kriged interpolation. The correlations returned by the alternate estimate are variable, ranging from positive to negative. However, the overall correlation is considered reasonable (Figure 14.7 and Figure 14.8).

Figure 14.7 Validation swath plots for Makosa and Makosa Tail



Source: Thor, October 2025.

Figure 14.8 Validation swath plots for Makosa Tail, Makosa East, and Baraka 3



Source: Thor, October 2025.

14.12 Resource classification

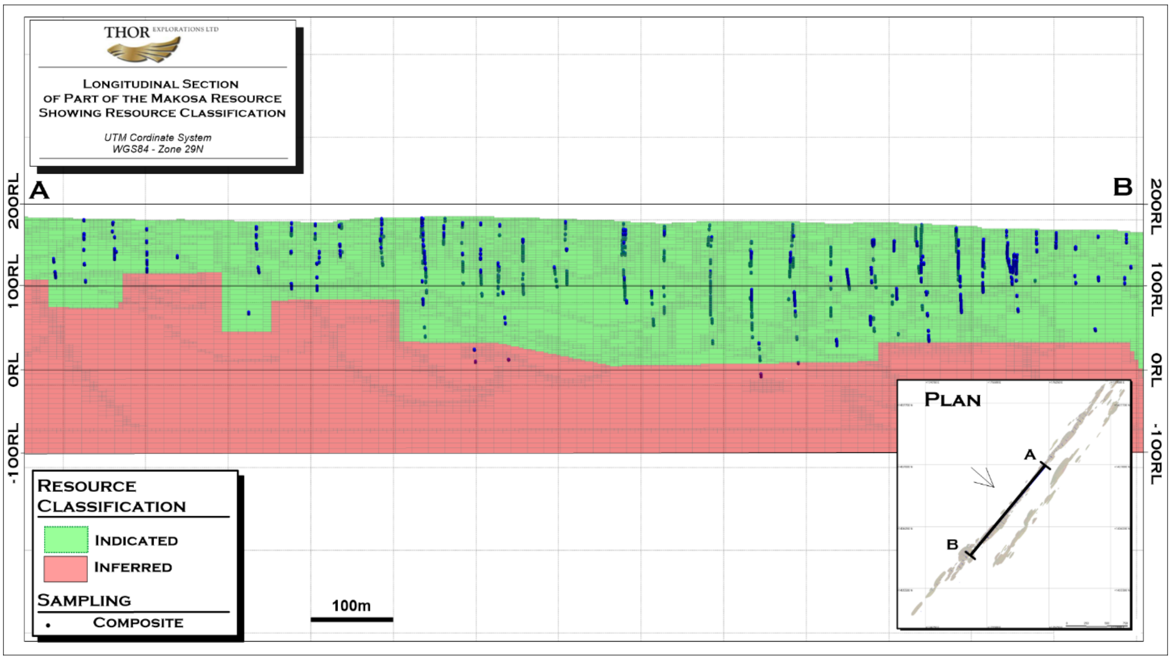
Mineral Resource classification complies with the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014).

The Mineral Resources are inclusive of the Mineral Reserves reported in Section 15; the Mineral Reserve estimate is a subset of the Indicated Mineral Resources.

Mineral Resource classification is primarily based on drillhole spacing and continuity of grade and is manually assigned using resource classification wireframes.

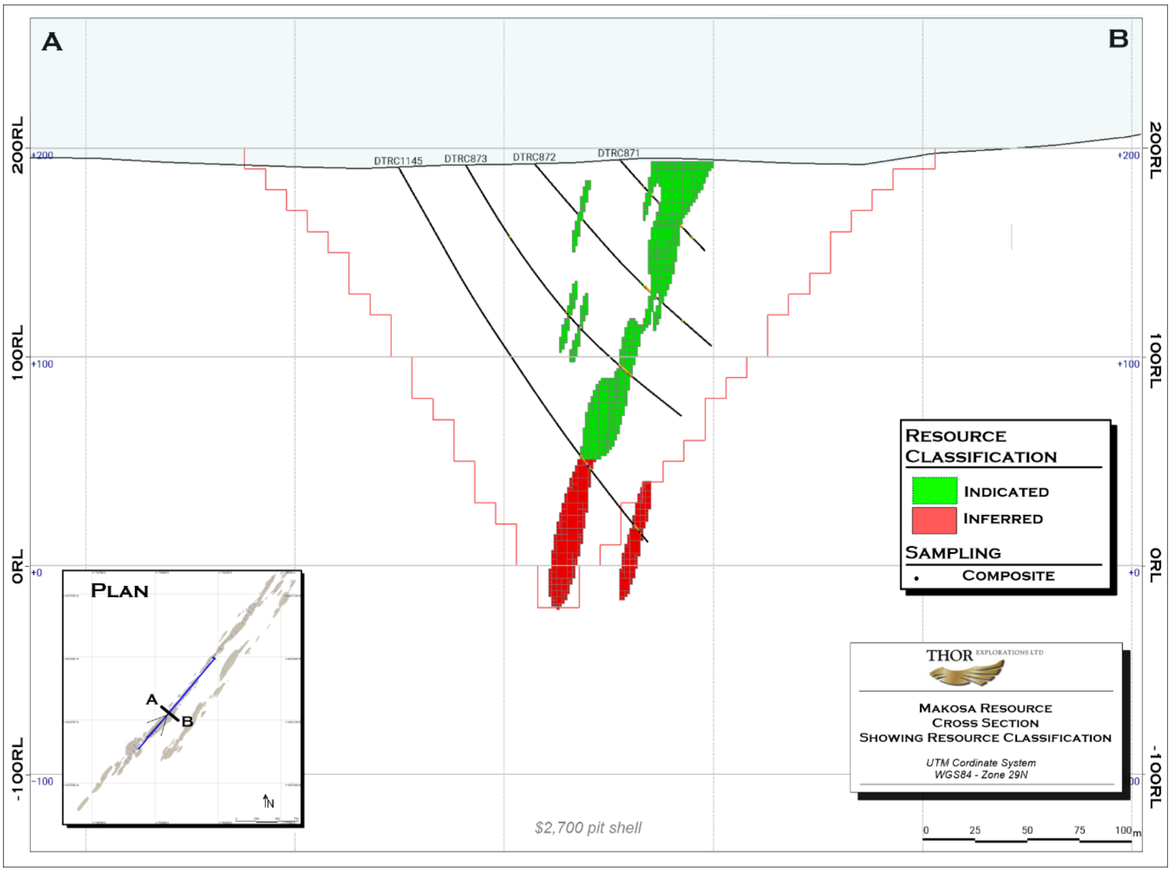
Portions of the resources that are drilled at between 25 m and 50 m spaced sections are classified as Indicated. Areas where drill spacing exceeds 50 m are classified as Inferred Mineral Resource (Figure 14.9 and Figure 14.10).

Figure 14.9 Longitudinal section of the resource classification at Makosa (looking SW)



Source: Thor, October 2025.

Figure 14.10 Vertical cross section of the resource classification at Makosa



Source: Thor, October 2025.

15 Mineral Reserve estimates

15.1 Statement of Reserves

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

> *"A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility level as appropriate that include the application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.*
>
> *Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.*
>
> *The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported."*[2]

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

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

This section states the estimated Mineral Reserves for the Makosa and Baraka 3 deposits on Thor's Douta Project. Baraka 3 is treated as a single deposit, while Makosa is divided into two separate areas: Makosa Main, which includes Makosa, Makosa North, and Makosa East, and Makosa Tail.

There are no existing stockpiles contained within the project area as this is a Greenfields project.

Table 15.1 shows a summary of the Douta Project Mineral Reserves, which have been reported in accordance with CIM Definition Standards, and considering all relevant Modifying Factors. All units are in the metric system with material quantities specified in metric tonnes (or a derivative thereof) and all grades are specified in grams per tonne.

Mineral Reserves are defined at the point where the ore is delivered to the processing plant. As such, mining recovery and dilution factors have been incorporated into these numbers.

The Mineral Resource does not contain any Measured Mineral Resource material for any of the deposit areas; therefore, no Proven Mineral Reserves have been declared.

Inferred Resources are explicitly excluded from the Mineral Reserves.

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

Table 15.1 Summary of Mineral Reserve Estimate for the Douta Project

Area	Classification	Oxide		Transitional		Fresh		Total	
		Tonnes (Mt)	Au grade (g/t)	Tonnes (Mt)	Au grade (g/t)	Tonnes (Mt)	Au grade (g/t)	Tonnes (Mt)	Au grade (g/t)
Makosa Main	Probable	8.7	0.88	5.6	0.91	14.1	1.13	28.4	1.01
Makosa Tail	Probable	1.7	0.82	1.2	0.89	4.4	1.25	7.3	1.09
Total Makosa	Probable	10.4	0.87	6.8	0.91	18.5	1.16	35.6	1.03
Baraka 3	Probable	0.8	1.13	0.2	0.98	0.0	1.46	1.0	1.11
Douta total	**Probable**	**11.1**	**0.89**	**7.0**	**0.91**	**18.5**	**1.16**	**36.6**	**1.03**

Notes:
- CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM 2014) were used for reporting of Mineral Reserves.
- Mineral Reserves are estimated using a long-term gold price of US$3,000 per troy oz for all mining areas.
- Mineral Reserves are stated in terms of delivered tonnes and grade before processing recovery.
- Mineral Reserves are defined by pit optimisation and are based on viable breakeven cut-offs as generated by material type, process destination and metallurgical recoveries.
- Metal recoveries are variable dependent on material type and mining area.
- Open pit dilution and geological ore loss are applied through the regularisation of the Mineral Resource models to an appropriate selective mining unit size.
- The QP responsible for this item of the Technical Report is not aware of any mining, metallurgical, infrastructure, permitting or other relevant factors that could materially affect the Mineral Reserve estimates.
- Effective date of Mineral Reserves is 24 January 2026.
- Tonnage and grade measurements are in metric units.
- Totals may not compute exactly due to rounding.

Cut-off grades vary for each deposit area and are also dependent on the oxidation state of the material. This is due to the changing processing costs and metallurgical recoveries associated with the different geological units. The Baraka 3 deposit also incurs an ore haulage cost due to its distance from the processing plant.

Cut-off grades range from 0.28 g/t to 0.56 g/t for Makosa and 0.35 g/t to 0.43 g/t for Baraka 3.

The Mineral Reserve estimate was undertaken using the Deswik mine planning software (Version 2025.2) and demonstrated that mining of the Makosa and Baraka 3 deposits at the Douta Project is practical and economically viable.

The QP is not aware of any known environmental, permitting, legal, taxation, socio-economic, marketing, political or other relevant factors which could materially affect the stated Mineral Reserves. The project is situated in a region conditioned to mining, both large scale and artisanal. The permitting process is underway with the Senegalese government and no issues are expected. Socio-political factors are unlikely to affect the Project and will be managed through the Company's Environmental and Social Management System.

15.2 Modifying Factors used in conversion to Mineral Reserve

15.2.1 Introduction

"Modifying Factors" are those factors that need to be considered to convert a Mineral Resource to a Mineral Reserve under the CIM standards. These factors cover aspects such as mining, processing, geotechnical, environmental, and marketing elements.

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

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

15.2.2 Mining

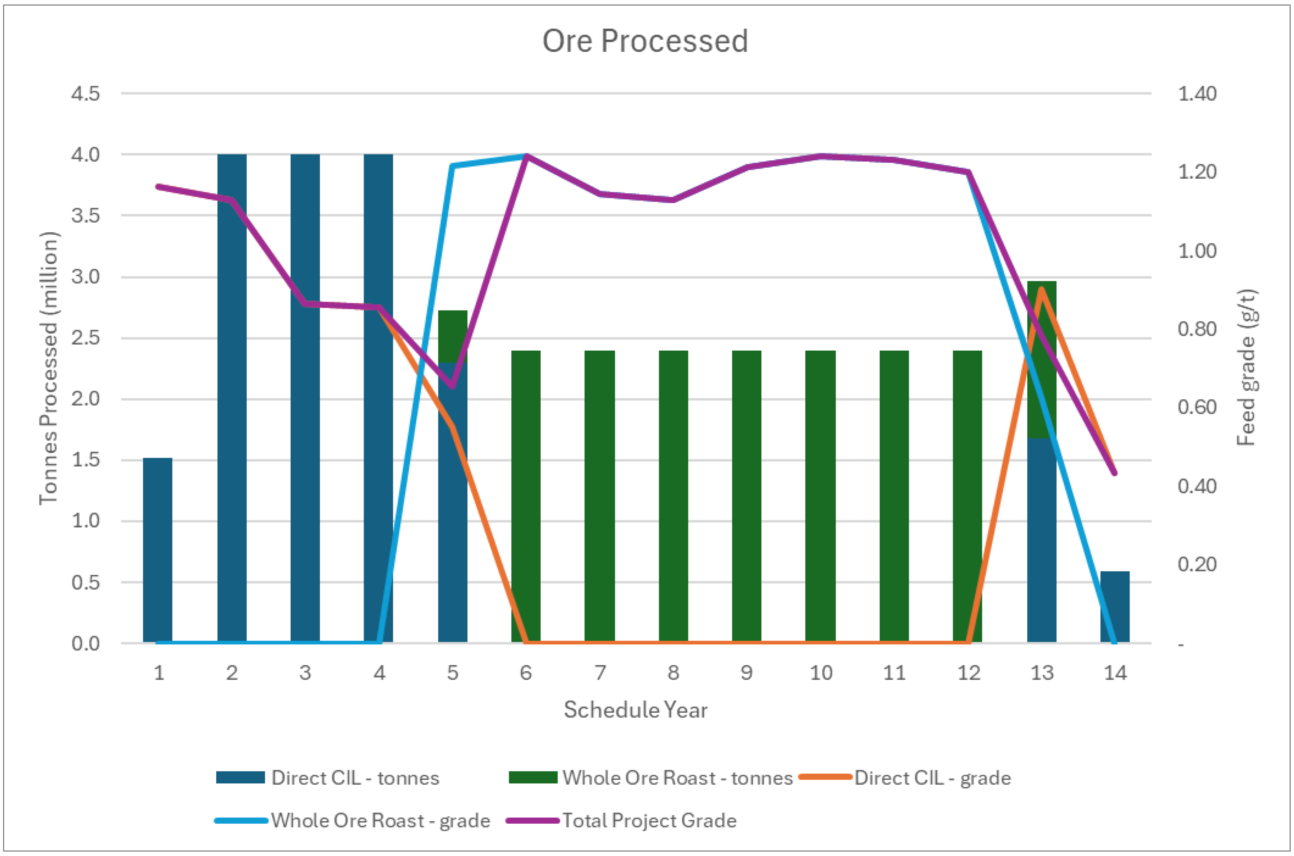
The deposits at the Douta Project will be mined by conventional truck / shovel open pit operations. Pit optimisations were conducted using the Mineral Resource geological models and other factors discussed in this document. This exercise produced the optimal economic pit shells under the given parameters.

Engineered pit designs were constructed based on the RF=1.0 shell for each deposit. These designs incorporated geotechnical and practical aspects such as ramp access.

Mining costs were provided by Thor based on existing contracts within Thor's other operation and broadly align with expected values for similar West African operations. Costs varied for different oxidation states with additional costs applied to ore mining.

A mining and processing schedule was developed to demonstrate that the Mineral Reserve material could be successfully mined both practically and economically. The schedule indicates a project life of over 13 years from the commencement of operation. Figure 15.1 shows the ore processed on an annual basis over the course of the mining schedule.

Figure 15.1 Ore processed during the life of the project



Source: AMC, January 2026.

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

15.2.3 Processing

Processing parameters such as metallurgical recovery and processing costs have been developed for different deposits and oxidation states. These parameters were applied to both the pit optimisation process and the mining schedule / cost model.

Ore haulage costs were applied to mill feed delivered from Baraka 3, some 35 km to the south of the proposed processing plant location.

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

15.2.4 Geotechnical / hydrogeological

A geotechnical study was undertaken by MINEMET in December 2023 where a series of pit slope angles and configurations were provided. These were used by AMC to inform the optimisation process and aid in the development of pit designs.

A subsequent review by AMC of the geotechnical data showed that there were some gaps in the MINEMET study, indicating that further geotechnical work should be undertaken prior to the development of the pits. However, the geotechnical study is adequate for the purposes of determining the Mineral Reserve.

Several hydrogeological studies have been conducted on the project in recent years. These include a piezometric analysis of drillhole information in 2023, a geophysical survey to identify potential water bore locations, also in 2023, and a water quality study conducted in 2023 and 2024. AMC also identified areas for improvement with the hydrogeological knowledge but concluded that these gaps do not prevent Mineral Reserves from being declared.

Sections 16 and 20 contain more information on the geotechnical and hydrogeological aspects of the project.

15.2.5 Environmental

The project is situated primarily in the Sabodala District, between the communes of Missirah Sirimana and Khossanto, with approximately one-sixth of the EL extending into the Bembou District. The Falémé River, forming the border with Mali, lies 15 km north-east of the permit. To the south-west, the Niokolo-Koba River - an important tributary of the Gambia River - flows through the Niokolo-Koba National Park (NKNP), a UNESCO World Heritage Site located 14 km from the project boundary.

The ELs are within a government designated Zone of Hunting Interest (ZIC or *Zones d'Intérêt Cynégétique*). ZICs have been established by the national government to regulate and promote hunting tourism. These zones are distinct from protected areas such as national parks, where hunting is totally prohibited. Licences and restrictions of fauna able to be hunted are designated for specific ZIC corridors.

An ESIA covering Phase 1 of the project for Makosa and Makosa Tail was submitted to the Senegal government in March 2025. This was revised in August 2025 following an interministerial government review. Regional public hearings held in November 2025 confirmed strong community support. Final approval was received in January 2026. Additional environmental surveys undertakne inQ4 2025 at the Baraka 3 area are not included in this PFS as they are not yet complete. The ESIA for Baraka 3 and Phase 2, specifically relating to the Phase 2 processing methodology, expansion of the mine footprint and improved design layouts to the water storage dam, tailings storage facility, camp and waste rock dumps, is expected to be completed in H1 2026. Additional information is presented in Sections 16, 20, and 26.

Several specific management plans required under the umbrella of the ESMP are currently absent or in development. These will be completed and implemented prior to the commencement of operations. More information is available in Section 20.

15.2.6 Location and infrastructure

The Douta Project is located in eastern Senegal, approximately 620 km south-east of the capital city of Dakar and 80 km north of the town of Kedougou.

Dakar is serviced daily by commercial flights from main cities of the world. Charter flights are available from Dakar to Kedougou, the regional centre, located approximately 80 km from Douta.

The proposed main transport route for capital equipment would be from the port of Dakar, a journey of approximately 12 hours. The tarred road from Dakar to Kedougou and Moussala-Bamako is located south and east of the permit. The unsealed road from Bembou to Kossanto provides access to the site from both the north and south.

Local infrastructure is limited to small rural settlements connected by gravel roads and paths.

There is no national water network through the area. At the Project Exploration Camp non-potable water is sourced from a borehole with drinking water supplied in bottled form.

There is no (or very limited) national power grid through the area. Presently, an on-site diesel generator, fuelled regularly by purchases from either the Bembou, Saraya, or Kedougou fuel stations, supplies electricity power to the camp.

Transportation costs of spares and consumables have been considered as part of the costing of the project. More information can be found in Sections 4, 5, and 18.

15.2.7 Marketing elements and factors

The Mineral Reserve estimation has assumed a gold price of US$3,000/oz for the optimisation and pit design work and a price of US$3,500/oz for financial analysis.

A government royalty of 5% has been applied, along with fixed TCRC costs of US$15/oz and variable TCRC costs of 0.4% gold value.

All prices quoted are in US dollars.

Sections 16 contains more details on this topic.

15.2.8 Legal elements

The Company is party to an option agreement with IMC, pursuant to which, with effect from 24 February 2012, the Company exercised its option to acquire a 70% interest in the Douta Project to be held through ASR.

During September 2025, the Company signed a binding sale and purchase agreement (SPA) with IMC to acquire the remaining 30% minority equity interest in the Douta Project. The acquisition is subject to the completion of certain conditions precedent, including final approval of the Minister of Mines.

An Exploitation Permit (DE11618) is currently in place for the lease covering the Makosa Main and Makosa Tail deposits. An Exploration Permit (EL03709) is in place for the Baraka 3 deposits and includes some of the project infrastructure. A second adjacent Exploration Permit (EL02254) is held by the company, allowing further exploration activities to continue.

Section 4 contains additional information on the legal and permitting framework.

15.2.9 General costs and revenue elements

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

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

Section 22 contains information on this economic analysis.

15.2.10 Social elements

The Project spans the communes of Missirah Sirimana, Bembou, and Khossanto. No settlements lie within the mining footprint, but Sambarabougou and Mandankholi are within 0.2-1.3 km of the project boundary. Communities are familiar with mining impacts—both positive (employment, procurement, market creation) and negative (dust, blasting, land disturbance, competition for resources, influx of job seekers).

The 2023 Census, conducted by Senegal's National Agency for Statistics and Demography, shows that the Kédougou Region is one of Senegal's fastest growing. Mining is the region's main economic driver, particularly artisanal mining. Approximately 43% of households are engaged in mining in some form.

The project has considered the impact of mining and processing activities on the communities and waste dumps, processing facilities and other structures have been situated within the lease boundaries accordingly.

More information on the social elements of the project can be found in Section 20.

16 Mining methods

16.1 Geotechnical considerations

16.1.1 Introduction

The Douta Gold Project is situated in the Kéniéba inlier of eastern Senegal. Preliminary mine planning and economic shell analyses suggest that the Douta pits could reach a depth of up to 170 m. The upper 40 m will consist of various regolith soils, while the lower slopes will be in foliated rock.

The orebody trends north-east and dips steeply to the north-west. The footwall is on the East (south-east) and the hangingwall is on the West (north-west) side. The rock domain includes meta-sediments and basic intrusive rocks, with the Makosa orebody and host rock foliation striking north 40 degrees east and dipping 70 degrees north-west. Foliation is prominent in the sedimentary rocks (greywackes and shales), while the gabbros are more massive and less foliated.

AMC conducted a review of the existing geotechnical data provided. This data included:

- Lithology models
- Regional structural information
- Local joint-set data from drilling
- Rock mass property data
- Rock strength data
- Hydrogeological information
- Geotechnical Site Investigation Factual Report and Open Pit Slope Design for the Douta Project, MINENET, December 2023

It should be noted that this review considered only the data presented in the documents listed above. If data exists elsewhere, it may not have been considered when evaluating data confidence.

16.1.2 Review outcomes

The geotechnical review is primarily based on the Geotechnical Site Investigation Factual Report and Open Pit Slope Design prepared by MINENET with the focus on Makosa and Makosa Tail zones. AMC has reviewed this report and concurs that the data is generally of a standard that meets the requirements of a PFS as outlined by Reed and Stacey (2009), referred to as Level 2. Where AMC has assessed that a Level 2 standard of data confidence has not been achieved, this is noted as a deficiency and further work is recommended to improve data confidence.

Table 16.1 details the outcomes of the geotechnical review, along with recommendations for further action.

Table 16.1 Outcomes of geotechnical review

Review item	Data confidence	Comments and recommendations
Geological Model	At Level 2	A reasonable geological model has been generated around the mineralised zone.
Structural Model – Major structures	Below Level 2	No structural model or fault wireframes were provided, however structural data from recent diamond drilling was provided, included some logged shear structures. AMC recommends that a three-dimensional structural model be constructed using fault and shear structures collected from the diamond drilling and verified with mapping activities on implementation.
Structural Model – Fabric	At Level 2	A structural database has been established based on the geotechnical logging of 13 oriented drillholes. Structural data has been recorded, and an initial stereographic assessment of the structural fabric has been undertaken.
Hydrogeological Model	Below Level 2	Hydrogeological information at the site was limited during the geotechnical investigation program. To achieve the target Level 2 rating, the following items are required: • Determine hydraulic properties through in situ testing which can be done in conjunction with the geotechnical drilling program. • Create a hydrogeological database to track monitoring and testing data. • Generate a groundwater model, producing groundwater profiles for the latest pit slope designs.
Intact Rock Strength	Below Level 2	While a number of samples have been sent to the laboratory for tests including classification, triaxial strength testing, tensile testing and direct shear testing of discontinuities, it is AMC's view that additional testing is required to gain the target level of confidence. AMC recommends that additional uniaxial compressive strength (UCS), triaxial strength and tensile strength tests should be completed on the rock types where limited data is available. A total number of seven UCS, triaxial and tensile strength tests should be targeted for each rock type, including the existing tests.
Strength of Structural Defects	Below Level 2	Although some direct shear testing has been conducted as part of the geotechnical investigation program, the level of testing is insufficient to fully characterise the behaviour of the rock mass or the variation in defect strength parameters across the different rock types at the site. AMC recommends that additional direct shear testing be carried out in future data collection programs on bedding, joint and shear structures in each rock type.
Rock Mass Classification	At Level 2	AMC considers the use of the RMR89 and NGI_Q' rock mass classification systems to be standard practice in the mining industry and an appropriate approach for assigning geotechnical conditions. These classification systems have been implemented at Douta to determine Rock Mass Classification.

Source: AMC, 2025.

16.1.3 Slope design parameters

The stability of the regolith slopes was analysed using strength parameters derived from laboratory test results and data from comparable sites with similar conditions using the limit equilibrium method in the Rocscience SLIDE program. The analyses were conducted for slope angles of 44° and 40°, with Factors of Safety ranging from 1.07 to 1.19, respectively.

Kinematic analyses were conducted to assess the rock slope stability for the two slope sectors, including the hangingwall and footwall. The hangingwall stability assessment indicated no instances of kinematic instability for the inter-ramp slope angles up to 60°.

The stability of the footwall is significantly affected by kinematic instability due to the foliation dip of approximately 65°, which limits both the allowable bench angle and overall slope. In Makosa Tail, the foliation dip is shallower, averaging 48°, which requires separate design considerations for this zone compared to Makosa.

MINENET proposed the following slope design parameters for Makosa and Makosa Tail zones:

- **Saprolite – All Walls – Selective Depressurisation**
  - Bench height: 10 m
  - Bench angle (Batter): 60°
  - Berm width: 5 m
  - Inter-ramp slope angle: 42.9°
- **Saprock / Weathered_Rocks – All Sectors**
  Average 3 benches with loosening blasts
  - Bench angle (Batter): 70°
  - Single lift: 10 m
  - Berm width: 6 m
  - Inter-ramp angle: 46.1°
- **Rocks – Makosa and Makosa Tail – Hangingwall**
  Twin Benching with Buffer-Trims
  - Drilling lift: 10 m
  - Bench / drilling angle: 80°
  - Operational offset: 2 m
  - Berm width: 6.5 m
  - Inter-ramp slope angle: 59.0°
- **Rocks – Makosa Sector – Footwall**
  Twin Benching with Buffer-Trims
  - Drilling lift: 10 m
  - Bench / drilling angle: 70°
  - Operational offset: 2 m
  - Twin-bench face angle: 65° – average dip of foliation controlling angle
  - Berm width (alternate): 6.5 m
  - Inter-ramp slope angle: 51.7°
- **Rocks – Makosa Tail – Footwall (controlled by ~50 deg foliation dip)**
  Smooth Blast Buffer & Stab Hole Footwall-follow
  - Drilling lift: 10 m
  - Bench angle: 55°
  - Operating offset width: 2 m (minimum achievable stand-off)
  - Inter-ramp slope angle: 48.0°

These parameters were subsequently revised to match the 6 m block height in the Resource model, while still honouring the inter-ramp slope angles.

There was no limit equilibrium analysis result provided in the MINENET report (2023) for the final pit design with saprolite bench angle of 60°. It appears that these design parameters rely on assumptions based on comparable sites with similar conditions and the implementation of systematic dewatering measures. Additionally, no analysis results were provided for slope angles in saprock or weathered rock to support recommended slope design parameters.

The kinematic analyses of rock slopes provided were limited to assessing wedge sliding failure and were conducted only for a 60° slope angle in the hangingwall and a 50° slope angle in the footwall.

These gaps highlight the need for additional geotechnical analyses to validate the proposed slope designs and ensure their stability under site-specific conditions.

No geotechnical studies have been conducted for the Baraka 3 deposits. The Makosa Tail geotechnical assumptions have been used for the Baraka 3 pits. Given the shallow nature of the pits and the low tonnage that these contribute to the overall inventory and mining schedule, AMC feels that this approach is appropriate for this PFS. However, a geotechnical study needs to be conducted at Baraka 3 before development of the deposits can advance.

16.1.4 Summary

While the geotechnical information is generally of a standard to support a PFS, AMC notes that there are a number of areas where additional work is required. In particular, AMC notes the following:

- There was no structural model or fault wireframes provided to AMC. It is recommended that a 3D structural model should be constructed using Regional-scale structural information in combination with mapping activities and fault structures collected from the diamond drilling.
- Shearing along geological contacts and within the weathered rock is common, which can significantly influence slope stability. To better understand the potential impacts, additional investigations are required to accurately characterise the spacing, orientation, and shear strength of these features.
- Hydrogeological information at the site was limited during the geotechnical investigation program. It is recommended that a comprehensive hydrogeological study be undertaken as part of future geotechnical investigations to support slope design and stability assessments. The study should include the following components:
  - Determine hydraulic properties through in situ testing which can be done in conjunction with the geotechnical drilling program.
  - Create a hydrogeological database to track monitoring and testing data.
  - Generate a groundwater model, producing groundwater profiles for the latest pit slope designs.
- There are limited Uniaxial and Deformation Modulus and tensile strength tests in some of the rock types.

  Additional Uniaxial compressive strength, triaxial strength, and tensile strength tests should be completed on the rock types where limited data is available. The total number of seven UCS, triaxial, & tensile strength tests should be targeted, including the existing tests for each rock type.
- AMC recommends that additional direct shear testing be carried out in future data collection programs on bedding joint and shear structures in each rock type.
- The geotechnical study for the Makosa and Makosa Tail zones, which span approximately 4.5 km, was based on data from 13 geotechnical drillholes This limited dataset is insufficient to adequately understand the ground conditions and rock mass variability across such a large area.

  It is recommended that additional geotechnical drillholes be drilled and logged in future investigation programs to provide a more comprehensive understanding of the ground conditions and rock mass properties.

- AMC recommends that the geotechnical assessment and rock mass classification should be updated when additional drilling and mapping data is available for future study.
- AMC recommends that geotechnical studies are conducted at Baraka 3 before any further development of the assets in that area continues.

16.2 Hydrogeology

A number of hydrogeological activities have been undertaken as part of the exploration of the Douta Project and have been used to support this study. These activities were undertaken between 2022 and 2024 and include:

- Analysis of the hydrogeological context and piezometry of the site.
- Geophysical survey of the local area for the purpose of identifying suitable bore hole locations.
- Completion of two boreholes.
- A water quality study.

As part of the geotechnical review, the hydrogeological information was examined. Hydrogeological information at the site is limited. As a result, the potential influence of groundwater on the slope stability of the future pits was assessed by experience gained from adjacent properties.

To improve the confidence in the hydrogeology, the following items are required:

- Determine hydraulic properties through in situ testing which can be done in conjunction with the geotechnical drilling program.
- Create a hydrogeological database to track monitoring and testing data.
- Generate a groundwater model, producing groundwater profiles for the latest pit slope designs.

As with the geotechnical information, hydrogeological investigations at Baraka 3 have not been undertaken. These will need to be completed before further development of the Baraka 3 assets is undertaken.

The planned dewatering strategy is discussed in Section 18.8.

16.3 Mining method selection

Mining at Douta will be undertaken by conventional contractor-operated truck and excavator open-pit mining. Mining will be conducted by the excavators on 6 m benches with two 3 m flitches per bench.

The upper oxide layers will be a mixture of free-dig and light blasting, while the lower transitional and fresh zones will require 100% drill and blast activities.

Ore will be categorised by material and grade through in-pit grade control. Ore from Makosa Main and Makosa Tail will be hauled by the haul truck fleet to the ROM pad adjacent to the crusher. Ore from Baraka 3 will be transported by the haul truck fleet to a ROM stockpile at Baraka 3. It will then be rehandled into road trucks and hauled 35 km to the ROM pad adjacent to the crusher. Waste from all pits will be hauled to the nearest available waste dump by the haul truck fleet.

Mining has been divided into two phases. Phase 1 focusses on the shallow oxide and transitional material that can be treated with a conventional CIL process. Phase 2 expands the mining to include fresh material which will be treated through a suspension roasting process prior to the CIL stage.

16.4 Mine optimisation

16.4.1 Optimisation iterations

During the course of 2025, several rounds of pit optimisations were conducted, with variations to processing cost, metallurgical recovery, throughput and geological model updates being the main differences. The final pit optimisation work was conducted in October and November 2025.

16.4.2 Resource block models

The Douta Mineral Reserves are comprised of four open-pit mining areas as follows:

- Makosa (comprising of Makosa Main, Makosa North, and Makosa East)
- Makosa Tail
- Baraka 3 East
- Baraka 3 West

Each area was represented by a single geological model. The following block models were used as the basis for the Mineral Reserve:

- Makosa: Block Model Makosa.mdl
- Makosa Tail: Block Model Makosa Tail.mdl
- Baraka 3 East: Block Model East Block.dm
- Baraka 3 West: Block Model West Block.dm

All models were provided as rotated models to align with the strike of the target orebodies.

As the Douta Project is a Greenfields project, there are no existing ore stockpiles to be included in the Mineral Reserves or mine plan.

16.4.3 Input data for optimisations

16.4.3.1 Mining costs

Mining costs were obtained from NORINCO and Thor. Base mining costs were provided as a fixed rate per tonne of material mined. The Contractor did not provide fixed and variable mining cost components. The mining costs are based on a fuel price of $1.30/L. Table 16.2 shows the base unit mining costs used in this project.

Table 16.2 Unit mining costs

Material	Unit mining cost
Oxide and Transitional	$2.75/t
Fresh	$2.90/t

Additional costs for an owner's technical services team, grade control drilling and assaying costs have been estimated by Thor to be $2.00/t ore. These costs were applied to ore mining costs on top of the unit mining costs shown in Table 16.2. This price includes RC Grade Control drilling and assaying costs.

Mining costs have been compared to similar operations in West Africa, including Thor's current operation in Nigeria, and align with expectations, given a reasonable margin of error.

16.4.3.2 Ore haulage costs

The Baraka 3 deposits are approximately 35 km to the south of the current location for the ROM pad. Ore will be hauled from the ore stockpile at Baraka 3 to the ROM pad by conventional on-highway trucks.

A fixed loading cost of $0.50/t has been assumed, along with a unit rate of $0.15/t/km, resulting in an overall ore haulage cost of $5.75/t ore.

Separate ore haulage is not required for ore sourced from the Makosa Main or Makosa Tail pits.

16.4.3.3 Processing costs

Processing costs are defined for several material types based on oxidation and domain (Table 16.3). These costs are based on the flowsheets described in Section 17.

Table 16.3 Processing costs

Material	Processing cost ($/t processed)
Makosa and Makosa Tail Oxide	17.50
Makosa and Makosa Tail Transitional	17.50
Makosa Tail Fresh	35.76
Makosa Fresh	35.76
Baraka 3 Oxide	17.50
Baraka 3 Transitional	17.50
Baraka 3 Fresh	18.50

Source: NORINCO, 2025.

A General and Administration cost of $3.25/t was also applied to processed tonnes.

16.4.3.4 Metallurgical recoveries

Metallurgical recoveries are defined for the different material types (Table 16.4). These figures are supported by metallurgical test work conducted by Thor and reviewed by AMC.

Table 16.4 Metallurgical recoveries

Ore type	Metallurgical recovery
Makosa and Makosa Tail Oxide	92.50%
Makosa Transitional (excluding Makosa East WOX)	82.65%
Makosa East WOX Transitional	72.80%
Makosa Tail Transitional	86.60%
Makosa Fresh	81.00%
Makosa Tail Fresh	88.00%
Baraka 3 Oxide	92.50%
Baraka 3 Transitional	90.00%
Baraka 3 Fresh	76.00%

Source: NORINCO, 2025.

16.4.3.5 Geotechnical parameters

Pit slope configurations and Inter-ramp slope angles (IRA) were provided by MINEMET in the 2023 report (see Section 16.1). Oxide and transitional material were assigned one slope configuration each, while the fresh rock at both Makosa and Makosa Tail were further divided into hanging wall and footwall zones.

Overall slope angles (OSA) were then developed from these IRA by assessing expected slope height and likely ramp configuration, based on earlier optimisations carried out as part of this study. OSAs in some materials are considered conservative, and this provides an opportunity for improved design and economics once additional geotechnical investigation is undertaken.

No geotechnical investigation has been undertaken at Baraka 3. As these pits are shallow, mainly targeting oxide and transitional material, the Makosa Tail geotechnical parameters were used. As discussed in Section 16.1, further geotechnical investigation needs to be undertaken at Baraka 3.

Table 16.5 shows the Inter-ramp and OSAs used in the optimisations.

Table 16.5 Pit slope angles

Geotechnical domain	Inter-ramp slope angle	Overall slope angle
Oxide (SOX and MOX)	42.9°	34°
Transitional (WOX)	46.1°	36°
Fresh Rock – Makosa Hanging Wall	59°	51°
Fresh Rock – Makosa Footwall	51.7°	44°
Fresh Rock – Makosa Tail Hanging Wall	59°	51°
Fresh Rock – Makosa Tail Footwall	48°	39°
Fresh Rock – Baraka 3 Hanging Wall	59°	51°
Fresh Rock – Baraka 3 Footwall	48°	39°

Source: AMC, 2025.

16.4.3.6 Gold price

The gold price for the Mineral Reserve was set at US$3,000/oz while the Mineral Resource price was set at US$4,000/oz. These prices were provided by Thor and reflect Thor's philosophy and position on future gold trends.

16.4.3.7 Royalties and selling costs

A government royalty of 5% is applied to the revenue.

Selling costs include a fixed $15/oz charge and a variable charge of 0.4% of value.

Discounting was not applied to the optimisations.

16.4.4 Cut-off grade determination

Breakeven cut-off grades were calculated for each domain, utilising the information above. Table 16.6 shows the breakeven cut-off grades calculated and used in this study.

Table 16.6 Breakeven cut-off grades @ US$3,000/oz

Ore type	Cut-off grade (g/t)
Makosa and Makosa Tail Oxide	0.28
Makosa Transitional (excluding Makosa East WOX)	0.30
Makosa East WOX Transitional	0.34
Makosa Tail Transitional	0.29
Makosa Fresh	0.56
Makosa Tail Fresh	0.51
Baraka 3 Oxide	0.35
Baraka 3 Transitional	0.35
Baraka 3 Fresh	0.43

Source: AMC, 2025.

16.4.5 Application of dilution

Several different dilution methods were considered, including:

- Regularisation
- Deswik.SO
- Deswik.DO
- AMC internal dilution process (drilldil.mac)

Regularisation was ultimately chosen as it provided reasonable results and was the simplest and fastest process to complete on the four rotated models.

All models were regularised to a standard cell size that represented the likely Selective Mining Unit (SMU) with cell dimensions of:

- X (East) = 5 m
- Y (North) = 10 m
- Z (Vertical) = 6 m

The other model framework parameters were kept constant during the regularisation process, including the respective model rotation angles.

Table 16.7 shows the results of the regularisation process. Overall tonnage and metal dilution is reasonable for Makosa and Makosa Tail, where lodes are wider and can support the SMU. Tonnage and metal loss at the Baraka 3 deposits is very high. This is due to the fact that the original Baraka 3 models have parent cell sizes half that of the SMU size, and many lodes are only a single cell wide. These lodes are diluted during the process to the point where they no longer have the grade to be considered ore. While the Baraka 3 deposits may benefit from a smaller SMU size, two of the pits are larger than several Makosa pits. This suggests that the pits will all be mined with the same equipment and methodology, hence resulting in similar dilution and ore loss patterns.

Table 16.7 Results of dilution process

Parameter	Makosa	Makosa Tail	Baraka 3 East	Baraka 3 West
Tonnage dilution	14.0%	13.5%	71.6%	34.9%
Tonnage losses	-13.6%	-20.4%	-403.4%	-489.9%
Tonnage net change	0.4%	-6.9%	-331.8%	-455.0%
Metal loss	-9.2%	-11.4%	-343.1%	-402.2%

Source: AMC, 2025.

16.4.6 Optimisation results

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

For the development of the pit designs and subsequent mining schedule, two separate optimisations were undertaken. The first, run at a gold price of $3,000/oz, included all Indicated material along with the various input parameters discussed previously. The resultant shells would be used to guide the ultimate pit designs and support the Mineral Reserve.

The second, also run at a gold price of $3,000/oz, excluded all fresh rock and only considered oxide and transitional material. This represented the expected feed sources for Phase 1. These shells were then used to guide the Phase 1 pit design process, ensuring that Phase 1 was economic as a standalone project.

In both cases and for all pits, the Revenue Factor 1 pit shell was chosen to form the basis of design. Table 16.8 shows the results of the optimisations of the four models considering all Indicated material (Phase 1 and Phase 2) while Table 16.9 shows the results of the optimisations when fresh rock is excluded (Phase 1 only).

The Baraka 3 East optimisation was limited by the southern boundary of the exploration lease EL03709, although this only caused a minimal reduction.

Figures 16.1 to 16.4 show the results of the optimisations while Figures 16.5 to 16.7 show the Revenue Factor 1 pit shells for each of the optimisations.

Table 16.8 Reserve optimisation results (Phase 1 and Phase 2)

	Makosa Main	Makosa Tail	Baraka 3 East	Baraka 3 West	Total
Oxide Ore Tonnes (kt)	8,703	1,619	327	497	11,145
Oxide Ore Grade (g/t)	0.90	0.85	1.30	1.04	0.91
Oxide Waste Tonnes (kt)	53,123	16,632	4,117	3,271	77,144
Transitional Ore Tonnes (kt)	5,645	1,155	134	74	7,007
Transitional Ore Grade (g/t)	0.93	0.94	1.07	0.94	0.93
Transitional Waste Tonnes (kt)	20,699	6,628	402	272	28,002
Fresh Ore Tonnes (kt)	14,983	4,426	41	22	19,471
Fresh Ore Grade (g/t)	1.15	1.30	163	1.23	1.19
Fresh Waste Tonnes (kt)	41,013	22,832	35	11	63,890
Total Ore Tonnes (kt)	29,331	7,200	501	592	37,624
Total Ore Grade (g/t)	1.03	1.14	1.26	1.03	1.06
Total Waste Tonnes (kt)	114,835	46,092	4,554	3.554	169,036
Total Tonnes Mined (kt)	144,166	53,292	5,055	4,146	206,659
Contained Ounces (koz)	974	263	20	20	1,278
Recovered Ounces (koz)	817	233	18	18	1,087
Revenue (US$M)	2,308	659	52	51	3,070
Value (US$M)	962	264	24	22	1,272

Source: AMC, 2025.

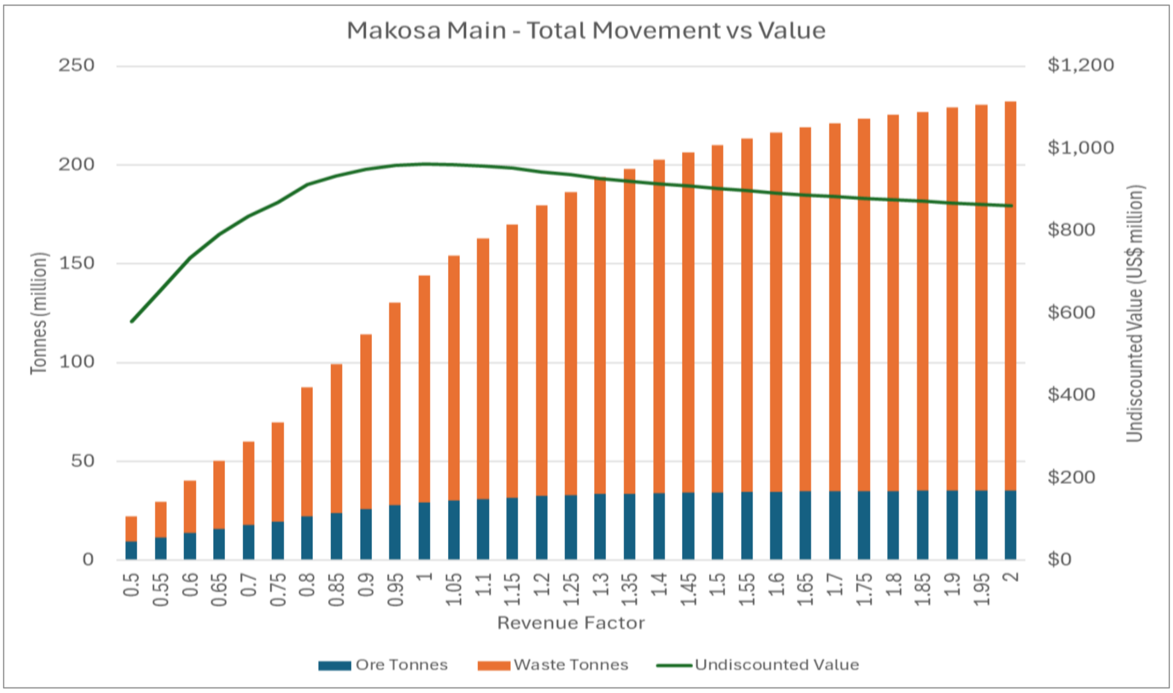
Table 16.9 Phase 1 optimisation results

	Makosa Main	Makosa Tail	Baraka 3 East	Baraka 3 West	Total
Oxide Ore Tonnes (kt)	8,639	1,547	318	490	10,994
Oxide Ore Grade (g/t)	0.90	0.86	1.32	1.04	0.91
Oxide Waste Tonnes (kt)	26,599	6,174	2,626	2,768	38,166
Transitional Ore Tonnes (kt)	5,023	961	77	46	6,108
Transitional Ore Grade (g/t)	0.97	1.01	1.18	1.03	0.98
Transitional Waste Tonnes (kt)	3,376	439	129	272	4,215
Fresh Ore Tonnes (kt)	0	0	0	0	0
Fresh Ore Grade (g/t)	0	0	0	0	0
Fresh Waste Tonnes (kt)	182	0	0	0	0
Total Ore Tonnes (kt)	13,661	2,509	396	536	17,102
Total Ore Grade (g/t)	0.92	0.92	1.29	1.04	0.94
Total Waste Tonnes (kt)	30,157	6,613	2,755	2,863	42,387
Total Tonnes Mined (kt)	43,818	9,121	3,150	3,399	59,489

	Makosa Main	Makosa Tail	Baraka 3 East	Baraka 3 West	Total
Contained Ounces (koz)	406	74	16	18	514
Recovered Ounces (koz)	357	67	15	17	455
Revenue (US$M)	1,008	189	43	47	1,286
Value (US$M)	577	106	23	22	728

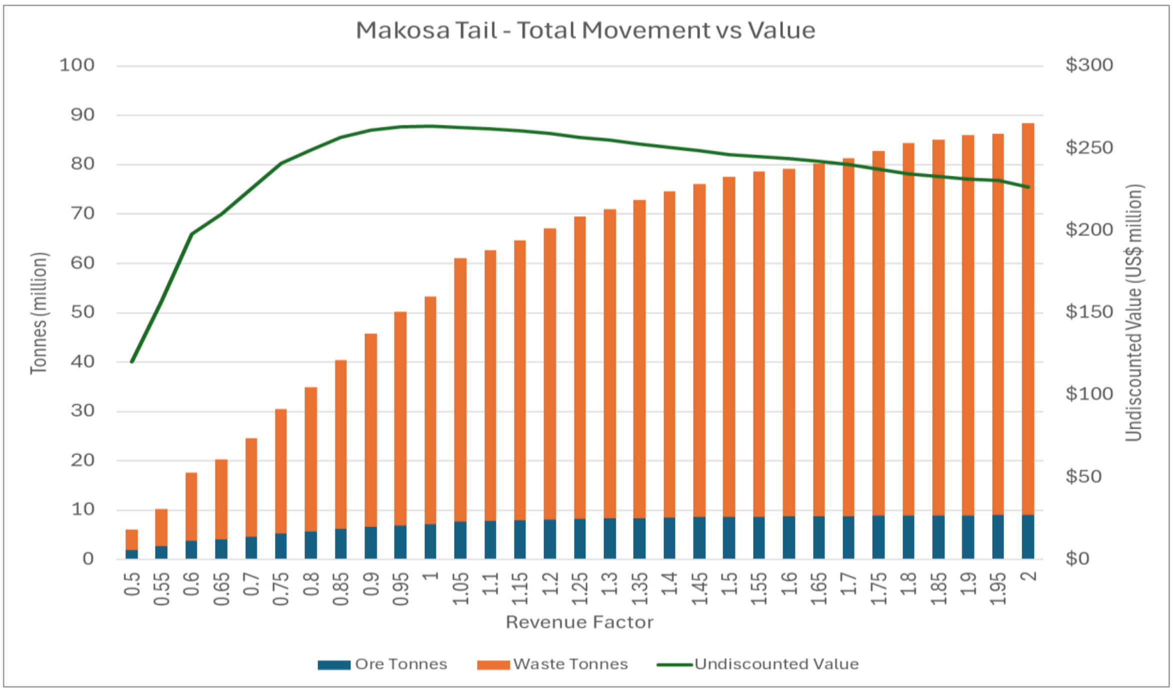
Source: AMC, 2025.

Figure 16.1 Makosa Main Phase 1 & 2 Optimisation



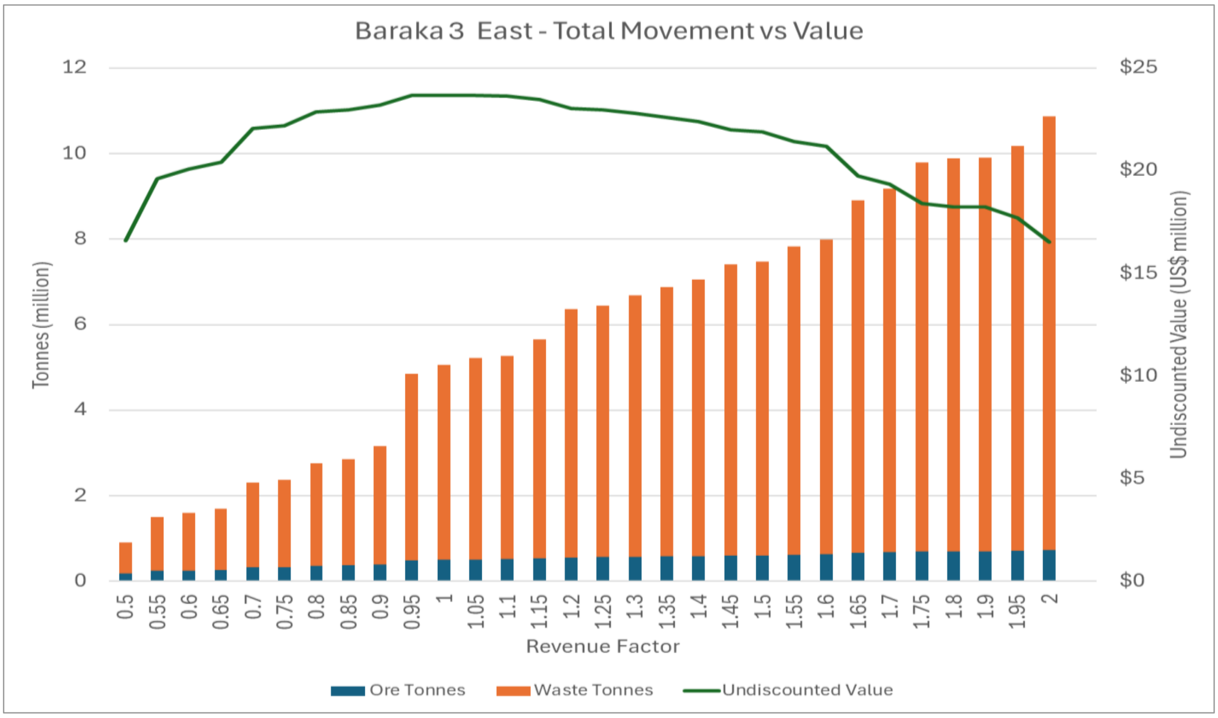
Source: AMC, January 2026.

Figure 16.2 Makosa Tail Phase 1 & 2 Optimisation



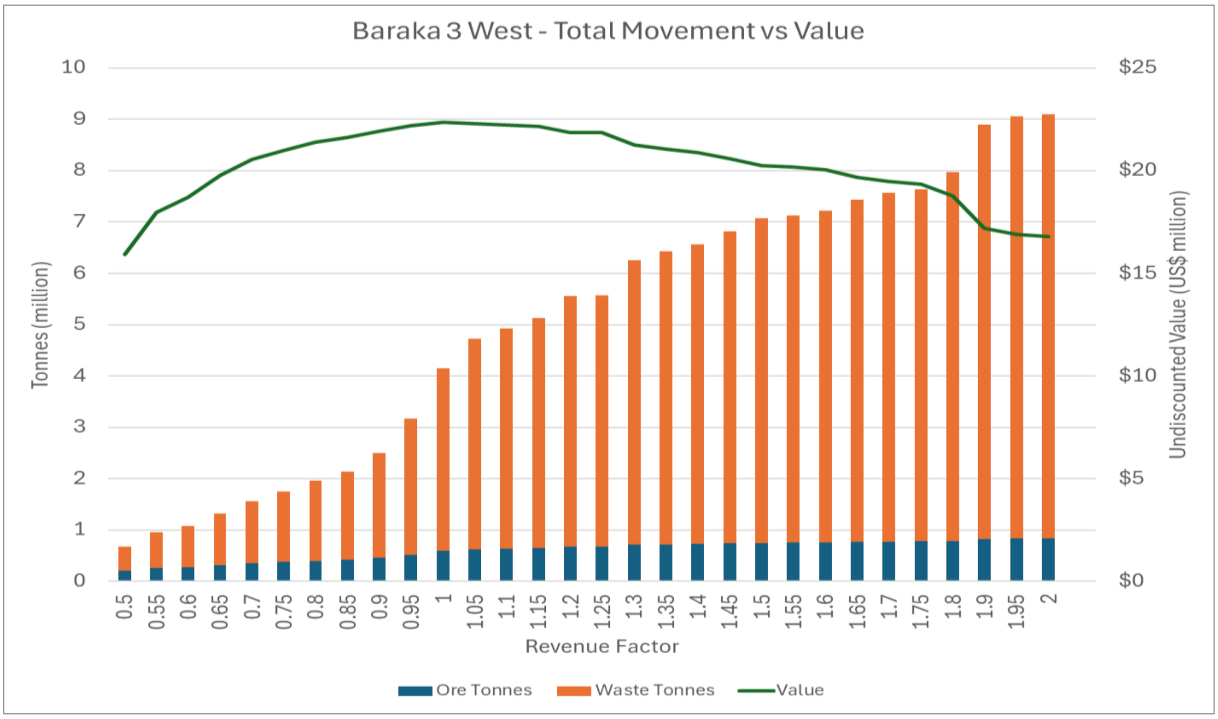
Source: AMC, January 2026.

Figure 16.3 Baraka 3 East Phase 1 & 2 Optimisation



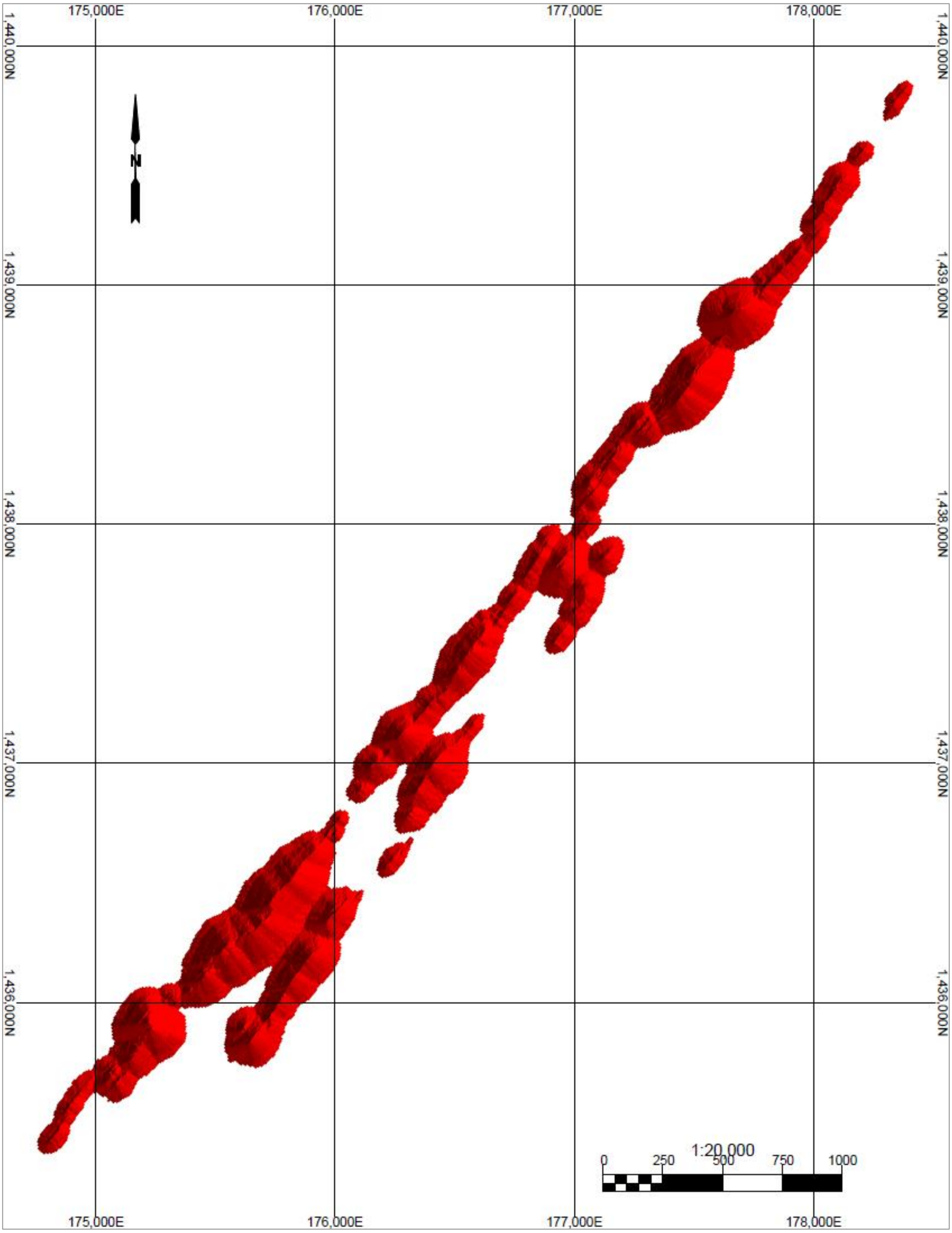
Source: AMC, January 2026.

Figure 16.4 Baraka 3 West Phase 1 & 2 Optimisation



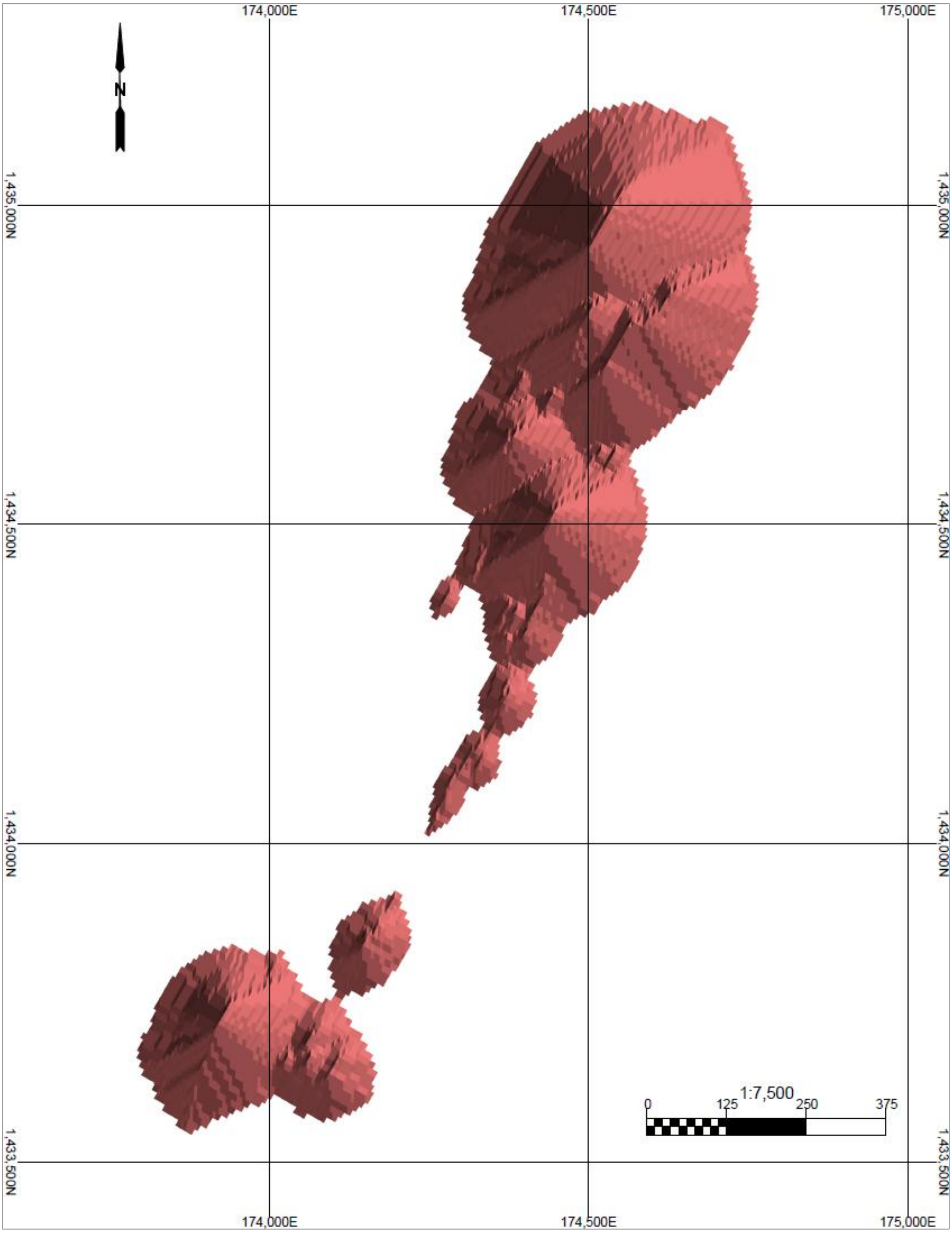
Source: AMC, January 2026.

Figure 16.5 Makosa Main Phase 1 & 2 Optimisation RF 1 Shell



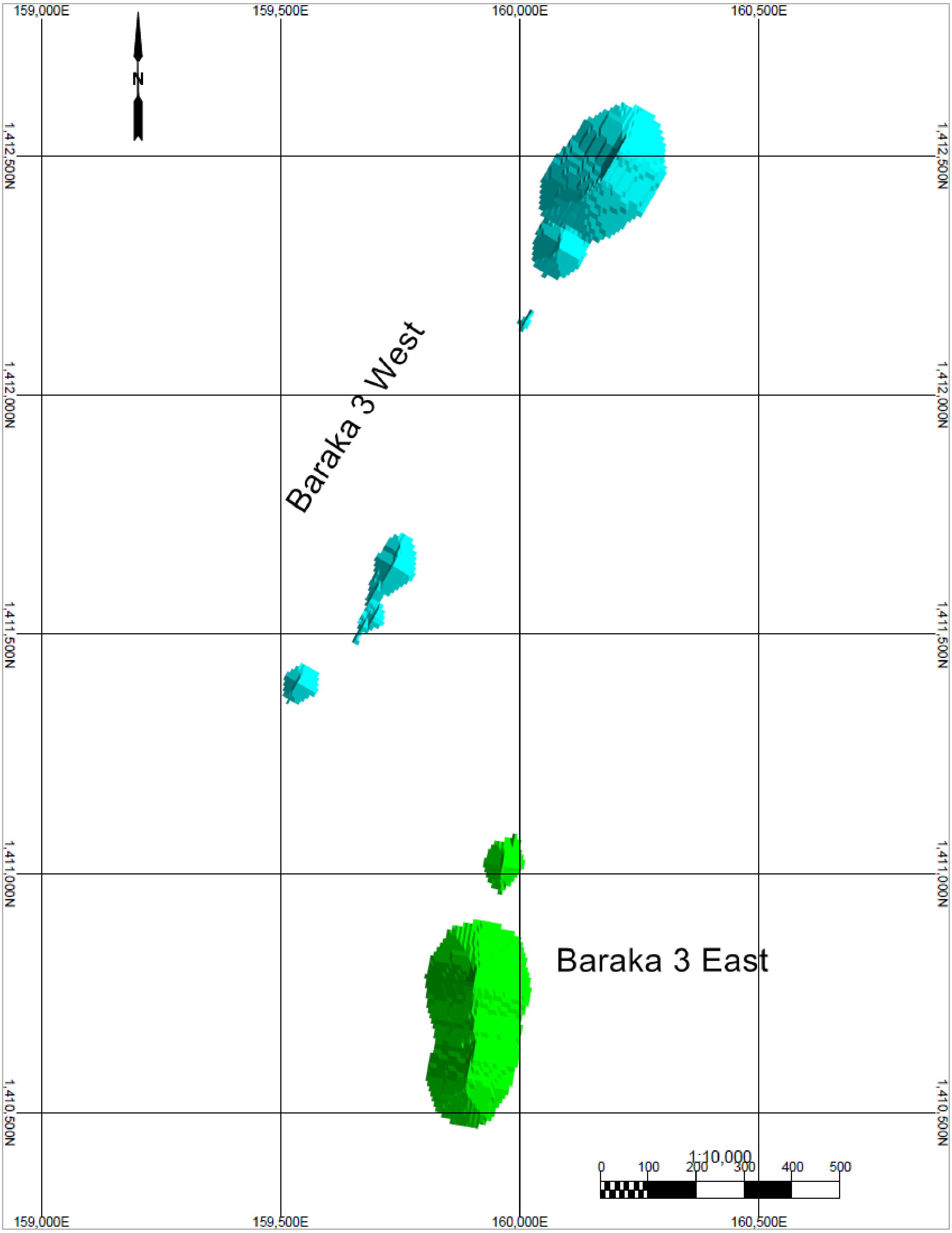
Source: AMC, January 2026.

Figure 16.6 Makosa Tail Phase 1 & 2 Optimisation RF 1 Shell



Source: AMC, January 2026.

Figure 16.7 Baraka 3 Phase 1 & 2 Optimisation RF 1 Shells



Source: AMC, January 2026.

16.5 Preliminary scheduling

Along with the various optimisation exercises throughout 2025, preliminary scheduling was undertaken to determine likely ore supply and cash flow obtainable from the various optimisation scenarios. This scheduling also helped to inform decisions relating to subsequent optimisation inputs.

Mine schedules were created which were based on the optimisation shells for the various scenarios under consideration including the implementation of a separate roasting phase for the fresh rock and splitting of the pits into Phase 1 and Phase 2.

16.6 Mine design

16.6.1 Mine design parameters

Using the selected optimisation shells as references, open pit designs were developed to allow a more realistic mining scenario. Ramps were including in the designs and ramp widths were varied between single- and double-lane to reflect likely mining practice, with single-lane ramps being employed at the bottom of the pits and where haul truck traffic was expected to be light.

Batter, berm and wall angle configurations varied for the different material types based on the geotechnical assessment. Where practical, a “goodbye cut” was designed at the base of each pit to maximise extraction of crusher feed. Ramps were not included in these goodbye cuts as it was assumed that the mining equipment would be able to top load the limited tonnages in these final benches.

Based on the assumed mining equipment and operating practices, a bench height of 6 m was used, although geotechnical conditions allow for up to three benches to be excavated between safety berms. In most cases, a 2 m operational standoff was maintained between benches where no berm was present, to represent operational drill restrictions.

It is expected that each bench will be drilled and blasted to a depth of 6 m, but that load and haul activities will occur on 3 m flitches. Grade control and blast control practices would be implemented to support this approach.

As the project is divided into Phase 1 (oxide / transitional) and Phase 2 (fresh rock) the pits were also divided along these lines. Oxide pits were first developed, based on the oxide-only optimisation shells. Once these pits were designed, they were used in conjunction with the Mineral Reserve optimisation shells to produce a second set of pit designs for Phase 2. This approach allowed for consideration of minimum mining widths, ensuring that the mining fleet would have room to bring down the Phase 2 behind the oxide pit void. Access issues around the oxide pit voids were also addressed in the Phase 2 pit designs.

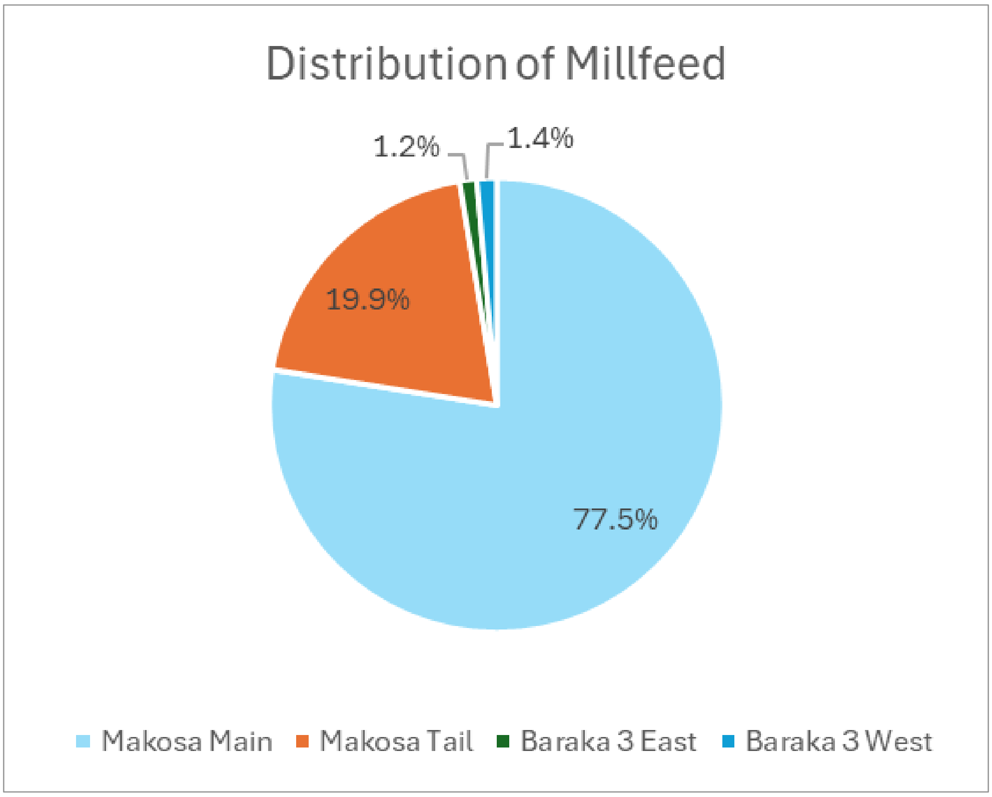
16.6.2 Pit statistics

The project area has been divided into four key regions in terms of pit designs and align with the four geological models. These regions are:

- Makosa Main
- Makosa Tail
- Baraka 3 East
- Baraka 3 West

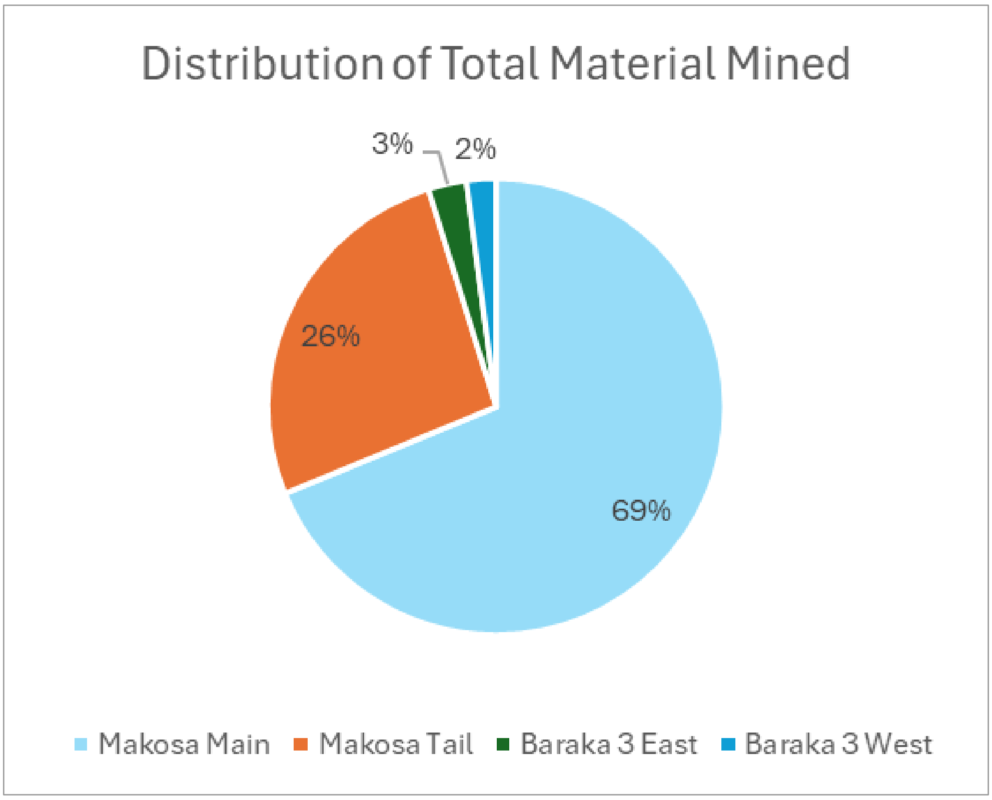
Figure 16.8 shows a graphical breakdown of mill feed between regions while Figure 16.9 shows the breakdown of total material. The total pit inventory by region is shown in Table 16.10, while Table 16.11 shows the pit inventories by region and phase.

Figure 16.8 Distribution of mill feed within the pit designs by region



Source: AMC, January 2026.

Figure 16.9 Distribution of total material within the pit designs by region



Source: AMC, January 2026.

Table 16.10 Pit inventory by region

Region	Oxide Material			Transitional Material			Fresh Material			Total Material			
	Ore tonnes (kt)	Ore grade (g/t)	Waste tonnes (kt)	Ore tonnes (kt)	Ore grade (g/t)	Waste tonnes (kt)	Ore tonnes (kt)	Ore grade (g/t)	Waste tonnes (kt)	Ore tonnes (kt)	Ore grade (g/t)	Waste tonnes (kt)	Total tonnes (kt)
Makosa Main	8,703.9	0.88	51,683.3	5,599.5	0.91	21,980.0	14,061.3	1.13	41,966.8	28,364.7	1.01	115,630.1	143,994.8
Makosa Tail	1,665.6	0.82	16,781.2	1,165.2	0.89	7,070.6	4,435.8	1.25	24,118.3	7,266.5	1.09	47,970.1	55,236.6
Baraka 3 East	303.2	1.29	4,667.6	126.5	1.00	554.4	10.9	1.53	26.6	440.5	1.21	5,248.5	5,689.1
Baraka 3 West	459.7	1.02	3,575.0	59.4	0.93	280.1	2.1	1.11	6.0	521.2	1.01	3,871.1	4,392.3
Total	**11,132.4**	**0.89**	**76,717.0**	**6,950.5**	**0.91**	**29,885.1**	**18,510.1**	**1.16**	**66.117.7**	**36,592.9**	**1.03**	**172,719.8**	**209,312.8**

Source: AMC, 2025.

Table 16.11 Pit inventory by phase

Phase	Oxide Material			Transitional Material			Fresh Material			Total Material			
	Ore tonnes (kt)	Ore grade (g/t)	Waste tonnes (kt)	Ore tonnes (kt)	Ore grade (g/t)	Waste tonnes (kt)	Ore tonnes (kt)	Ore grade (g/t)	Waste tonnes (kt)	Ore tonnes (kt)	Ore grade (g/t)	Waste tonnes (kt)	Total tonnes (kt)
Phase 1													
Makosa Main	8,370.8	0.89	25,343.7	4,055.8	0.94	5,770.6	168.5	1.06	455.6	12,595.1	0.91	31,569.9	44,165.1
Makosa Tail	1,532.9	0.86	5,445.5	776.3	0.97	610.5	24.9	1.58	11.6	2,334.0	0.90	6,067.5	8,401.5
Baraka 3 East	303.2	1.29	4,667.6	126.4	1.00	554.4	10.9	1.53	23.6	440.5	1.21	5,248.5	5,689.1
Baraka 3 West	459.7	1.02	3,585.0	59.4	0.93	280.1	2.1	1.11	6.0	521.2	1.01	3,871.1	4,392.3
Total Phase 1	**10,666.5**	**0.90**	**39,041.7**	**5,017.9**	**0.94**	**7,215.6**	**206.5**	**1.15**	**499.7**	**15,890.9**	**0.92**	**46,757.0**	**62,647.9**
Phase 2													
Makosa Main	333.2	0.64	26,339.6	1,543.6	0.84	16,209.4	13,892.7	1.13	41,511.2	15,769.6	1.09	84,060.2	99,829.7
Makosa Tail	132.7	0.42	11,335.7	388.9	0.74	6,460.1	4,410.9	1.25	24,106.7	4,932.5	1.19	41,902.6	46,835.1
Total Phase 2	**465.9**	**0.58**	**37,675.3**	**1,932.6**	**0.82**	**22,669.5**	**18,303.6**	**1.16**	**65,618.0**	**20,702.1**	**1.12**	**125,962.8**	**146,664.9**
Total	**11,132.4**	**0.89**	**76,717.0**	**6,950.5**	**0.91**	**29,885.1**	**18,510.1**	**1.16**	**66,117.7**	**36,593.0**	**1.03**	**172,719.8**	**209,312.8**

Source: AMC, 2025.

Makosa Main is the largest contributor in terms of ore (78%) and total movement (67%). This region is divided into 5 distinct pit zones. Phase 1 consists of 12 pits within these zones as shown in Figure 16.10. Phase 2 consists of 10 pits, all of which are extensions of the Phase 1 pits (Figure 16.11). Two Phase 1 pits, namely Pit 2.2 and Pit 5.3, have no economic material associated with Phase 2 within their designs and are therefore not included in the Phase 2 mine plan, having been fully depleted during Phase 1. Pit 11.1 is identical to Pit 1.1 therefore it also does not contribute to the Phase 2 schedule.

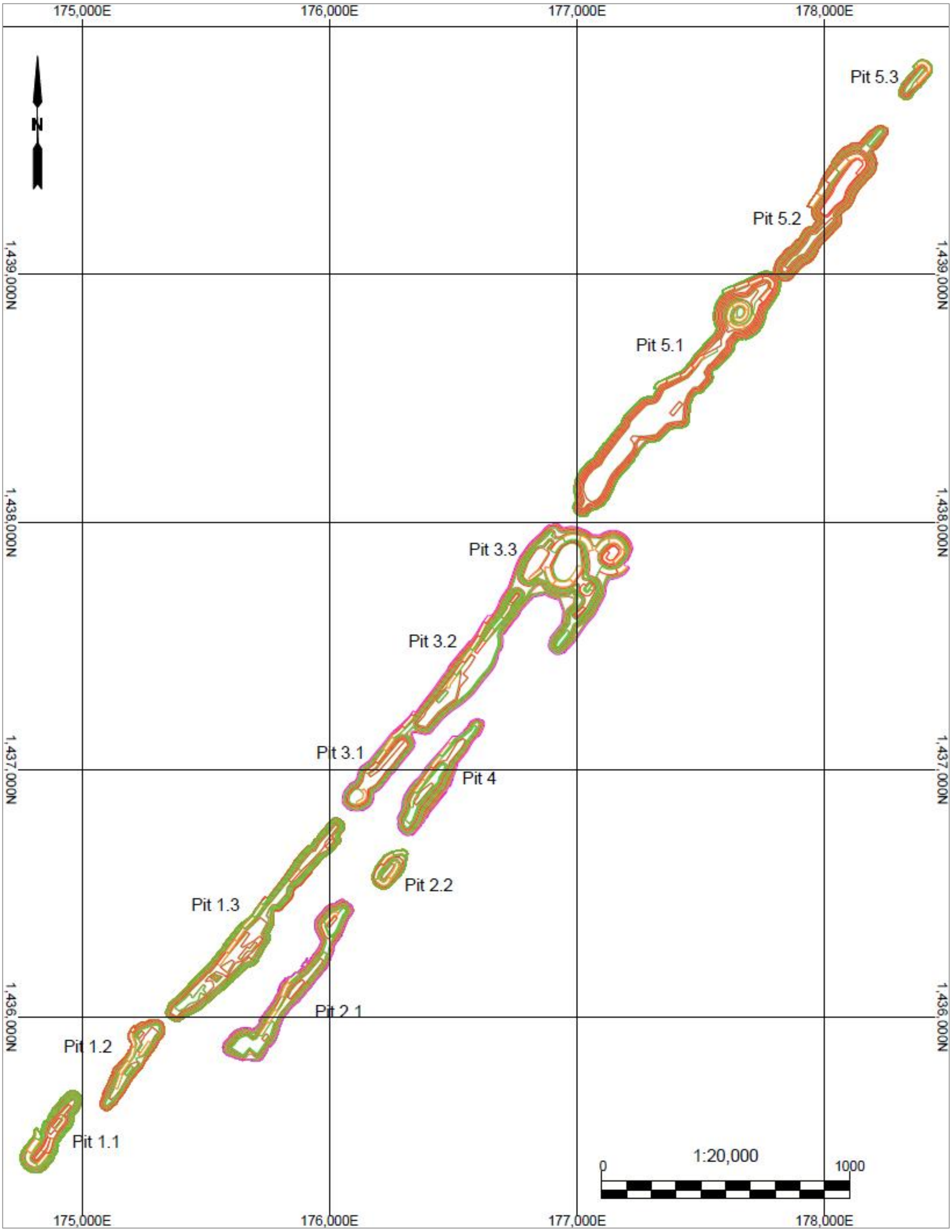
Where pit designs overlap, pit inventories were developed with a proposed extraction sequence in mind, and the solid associated with the pit to be mined second was depleted by the solid of the pit to be mined first. One example of this is with Pit 13.1 and Pit 14.

Makosa Tail contributes significant amounts of ore (20%) and total movement (26%) to the mine plan. Phase 1 at Makosa Tail consists of four distinct pits (Figure 16.12) while Phase 2 involves the expansion and deepening of the two larger pits (Figure 16.13). The two central pits do not form part of the Phase 2 mine plan.

Baraka 3 was divided into two regions, based on the different geological models provided. Baraka 3 East consists of two distinct pits while Baraka 3 West contains three distinct pits. In each case, one pit is significantly larger than the others. Pit 42 at Baraka 3 East was ultimately discarded from the Mineral Reserve as it was not economic once the ramps and berms had been added to the design. Baraka 3 does not contain any Phase 2 material, and all pits are excavated as part of the Phase 1 operations. Figure 16.14 shows the Baraka 3 East and West pit designs.

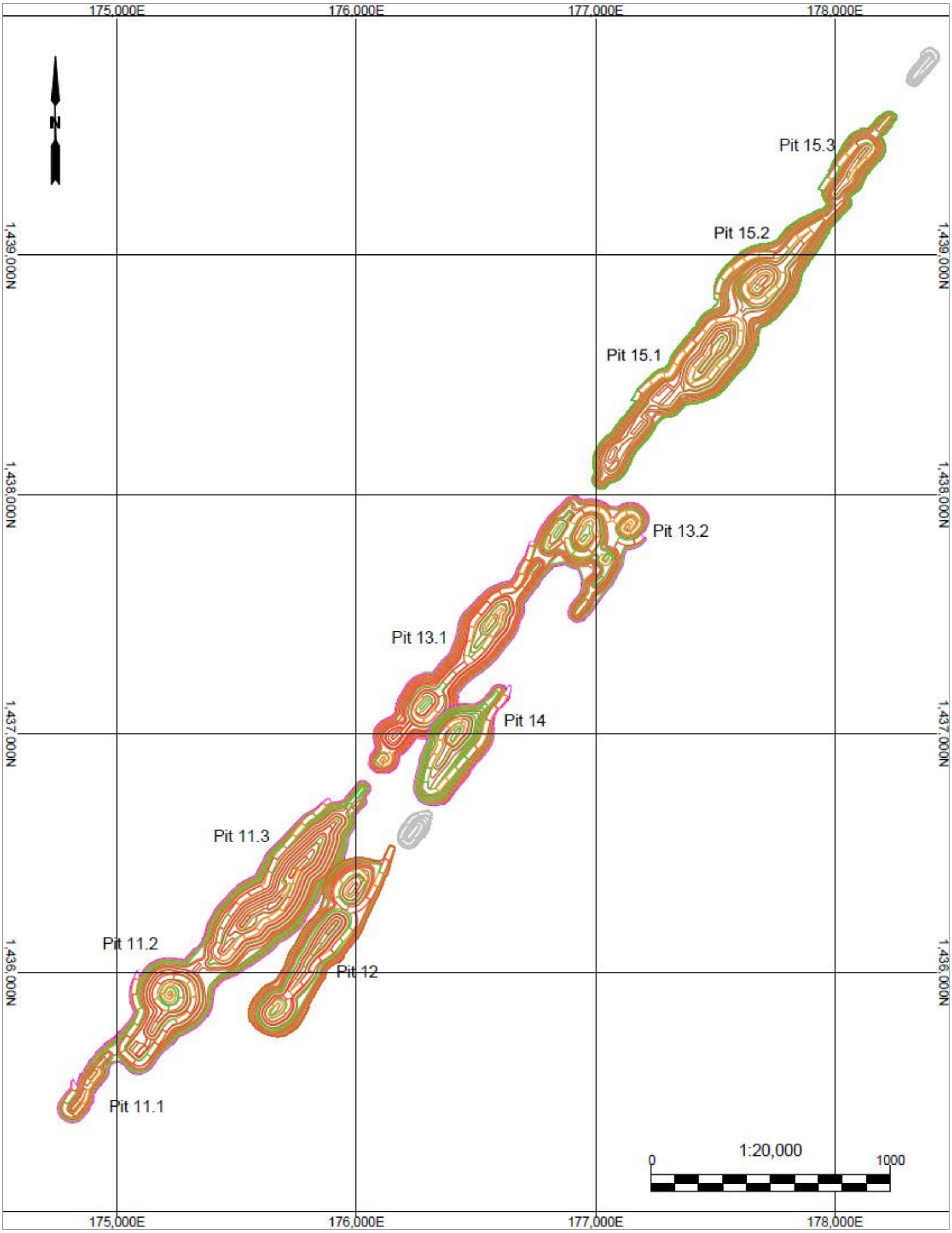
The Baraka 3 East Pit 41was limited in the south by the southern lease boundary of the EL0370 lease.

Figure 16.10 Makosa Main Phase 1 Pits (Oxide)



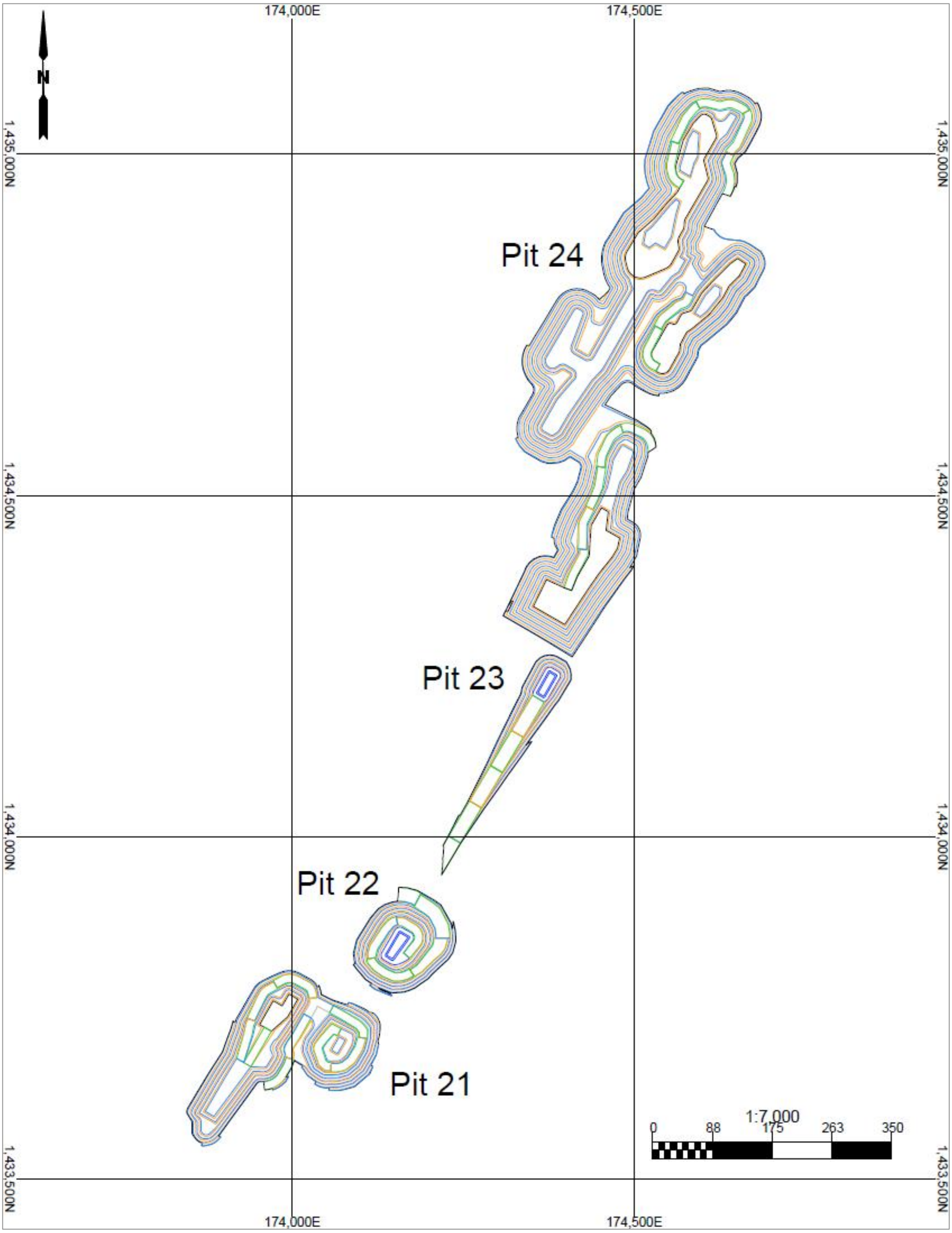
Source: AMC, January 2026.

Figure 16.11 Makosa Main Phase 2 Pits (Primary)



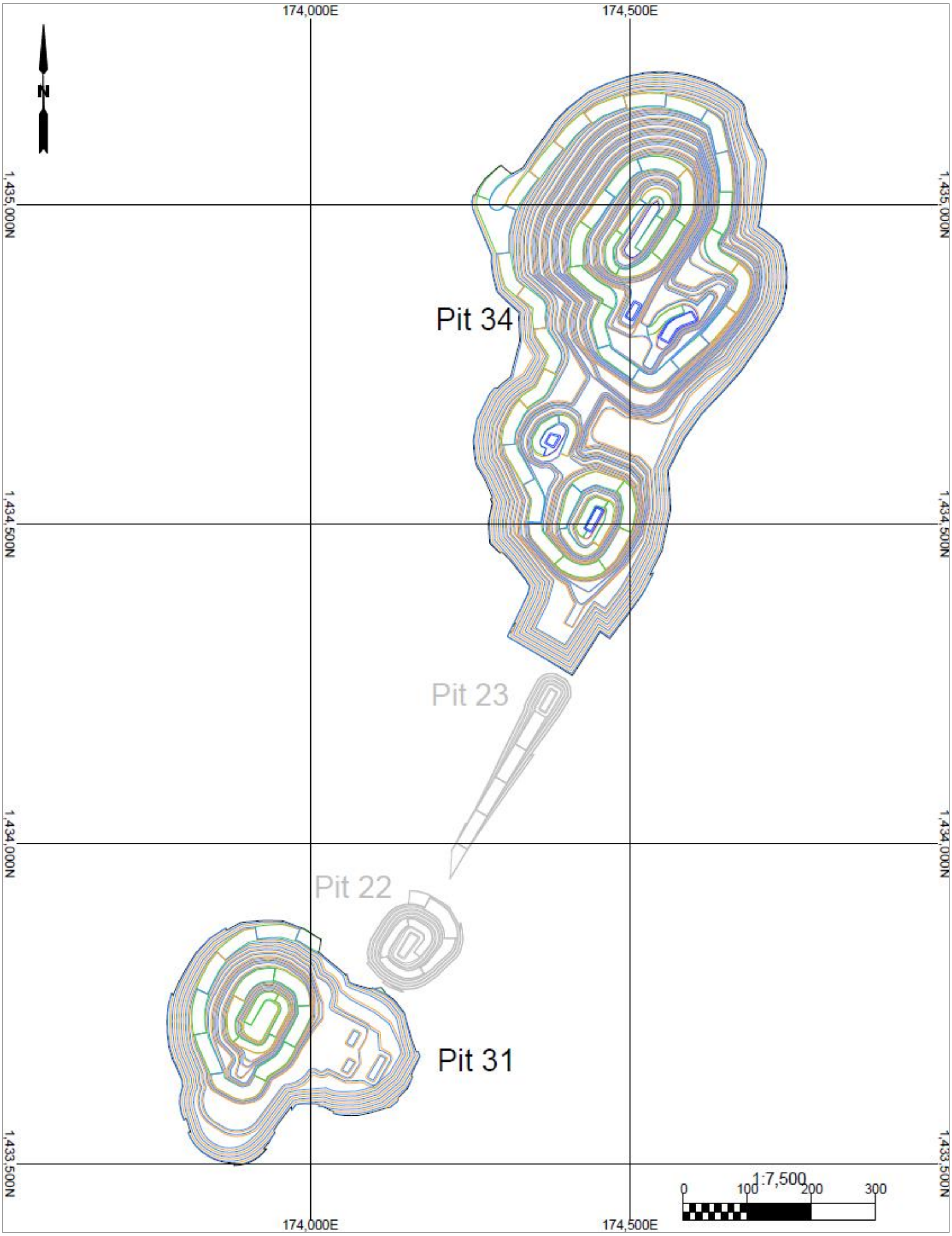
Source: AMC, January 2026.

Figure 16.12 Makosa Tail Phase 1 Pits (Oxide)



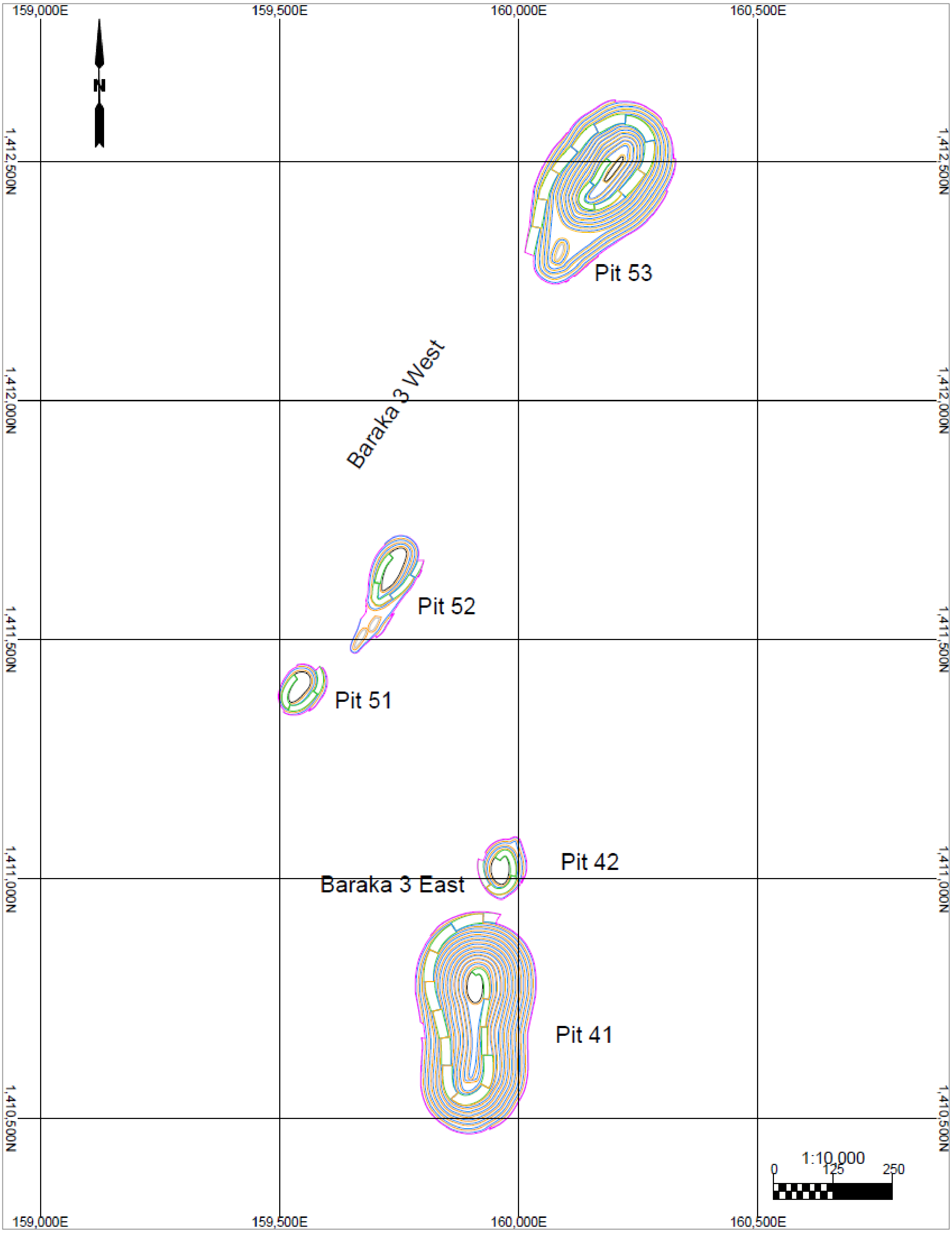
Source: AMC, January 2026.

Figure 16.13 Makosa Tail Phase 2 Pits (Primary)



Source: AMC, January 2026.

Figure 16.14 Baraka 3 East and West Phase 1 Pits (Oxide)



Source: AMC, January 2026.

16.6.3 Waste rock dumps, haul roads, and ROM pads

Waste rock dumps were designed for each of the mining regions. Given the current land use, there is little restriction on waste dump capacity. However, the presence of the eastern boundary of DE11618 did cause some restrictions on placement and expansion of two of the Makosa Main waste rock dumps.

Table 16.12 shows the total available waste dump capacity. The capacity is sufficient to accommodate the waste from all pits, with relevant local dump capacities also suitable for nearby pits.

Table 16.12 Waste dump capacity

Waste dump	Capacity (m^3)
Makosa Main North	16.1 million
Makosa Main Central	18.2 million
Makosa Main South	28.9 million
Makosa Tail	25.8 million
Baraka 3 North	2.3 million
Baraka 3 South	3.1 million

Source: AMC, 2025.

A run-of-mine (ROM) ore pad location has been designated adjacent to the processing plant location. Allowance has also been made for two low grade stockpiles in the vicinity of the ROM pad, one for oxide and one for fresh material.

A ROM transfer pad has been designated at Baraka 3 to allow haul trucks to deposit the ore from the Baraka 3 pits where it will then be rehandled into road trucks and hauled back to the Makosa ROM pad along an internal access road.

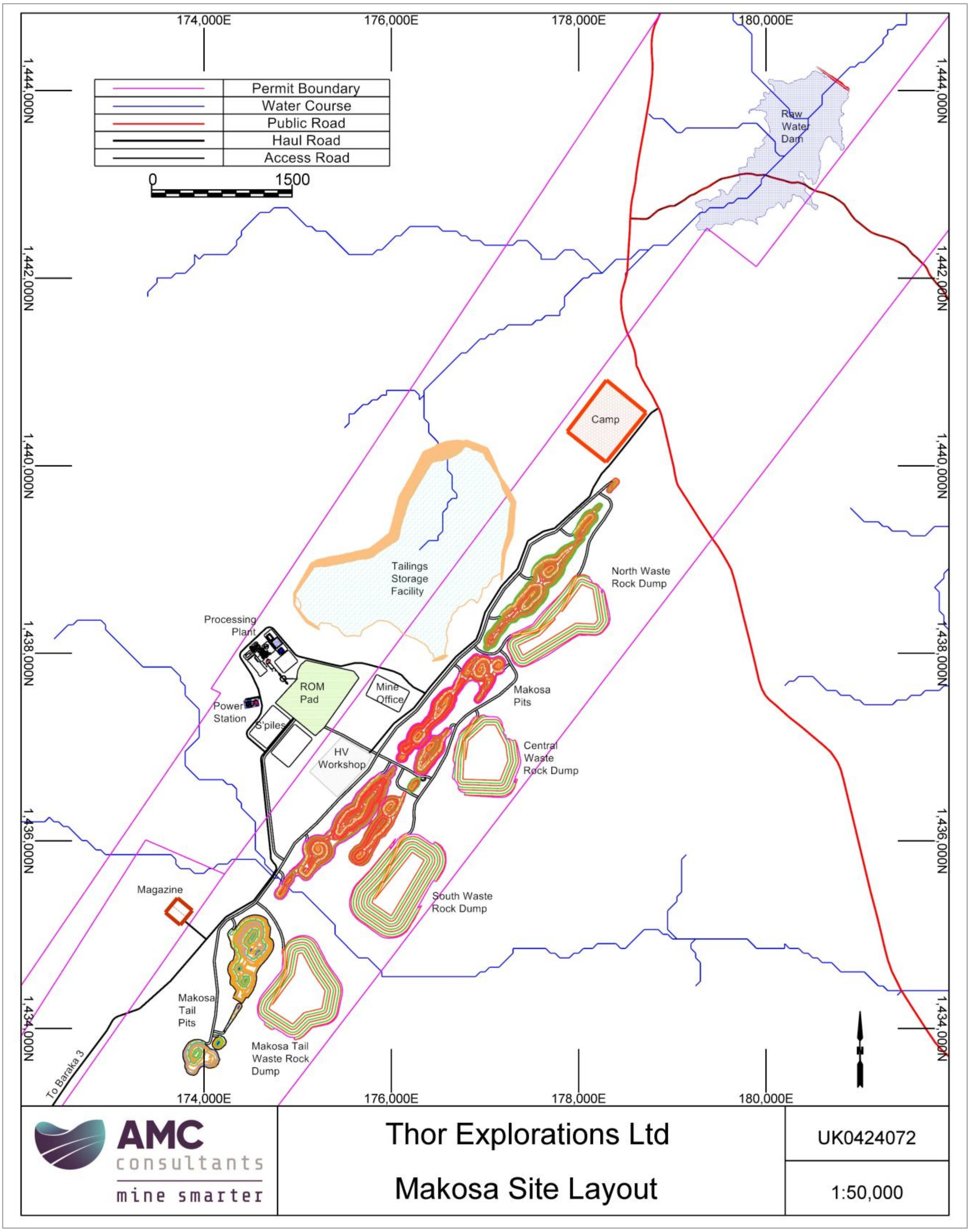
Haul road paths have been designed for all pits, providing access to the relevant waste dumps and ROM pads. Haul roads have been designed with a running width of 20 m.

Access road paths have been developed for general site access, including to provide access from the public road to the processing plant and mine workshops, to provide access to the explosives magazine and to provide a direct link to the Baraka 3 deposits from the processing plant at Makosa. Access roads have been assigned a nominal running width of 8 m.

A detailed cut and fill analysis has not been conducted for either the surface haul roads or the access roads but is not required for this level of study.

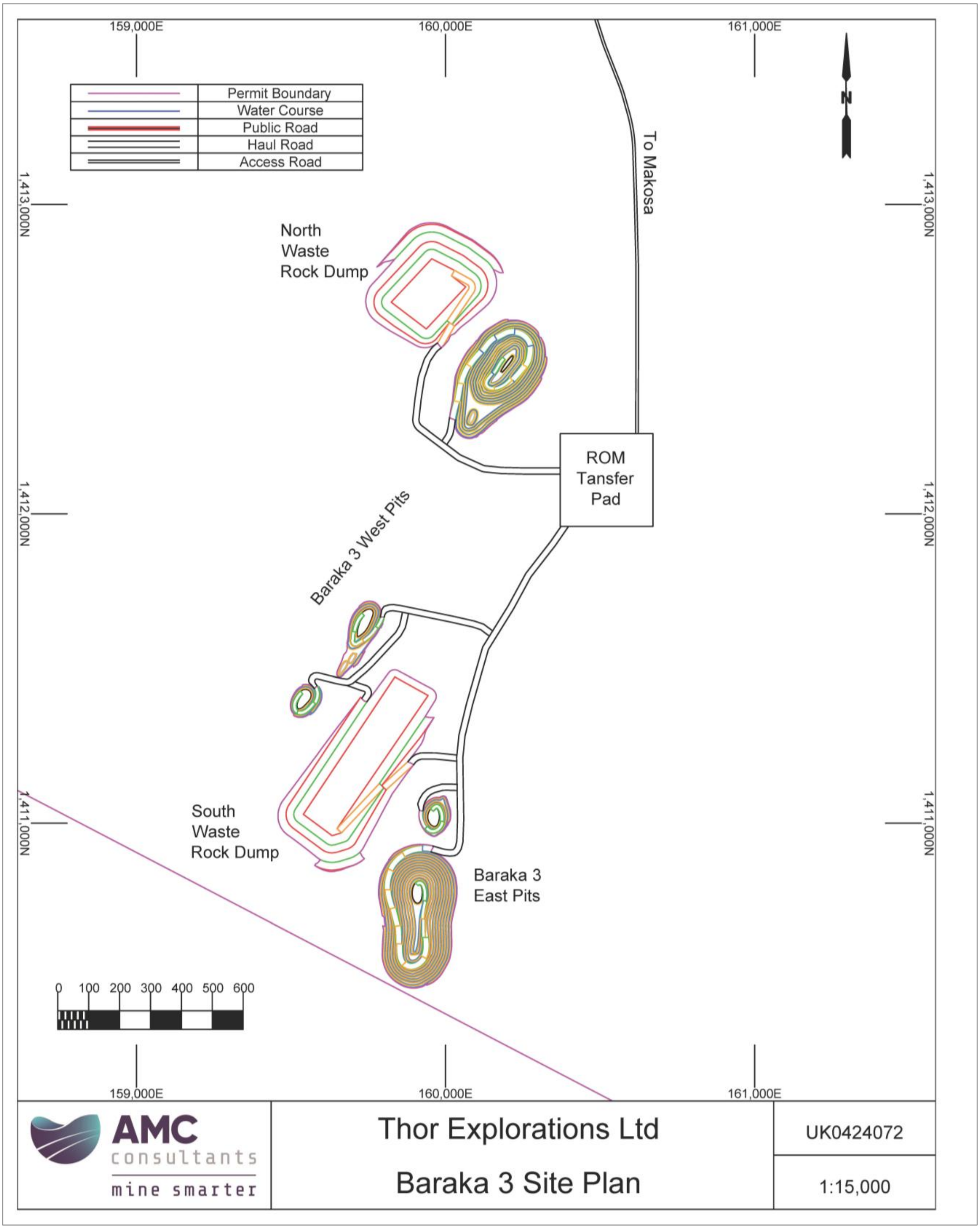
Figure 16.15 shows the location of waste dumps, roads and other surface infrastructure at the Makosa site while Figure 16.16 shows the same information for the Baraka 3 site.

Figure 16.15 General arrangement for the Makosa site



Source: AMC, January 2026.

Figure 16.16 General arrangement for the Baraka 3 site



Source: AMC, January 2026.

16.7 Production schedule

Production is divided into two distinct Phases. Phase 1 processes oxide and transitional material through a standard CIL leach processing flowsheet while Phase 2 treats primary fresh ore through a suspension roasting component prior to the CIL process. This approach allows for the deferment of capital associated with the construction of the roasters by focusing on oxide ore from all pits prior to commencing the mining of fresh ore.

The target throughput for Phase 1 is 4 million tonnes per annum through the processing plant while the target for Phase 2 is 2.4 million tonnes per annum, being comprised of 1.2 million tonnes through each of the two roasting units.

The ultimate pits were designed while honouring these two phases. The Phase 1 pits, derived from optimised shells for oxide and transitional material, were mined first. Once these pits were exhausted, the larger, Phase 2 pits were mined.

Ore and waste mining commences in Month 5 of the first year of the schedule, at a reduced rate of 500,000 t. In Month 6, it increases to 750,000 tonnes and then increases again to 1,500,000 tonnes in Month 12. For the next 7 years, the mining rate is kept consistent at 1.5 million tonnes per month, giving an annual rate of 18 million tonnes. To meet ore requirements in the tail end of the mine life, the mining rate is increased to 2 million tonnes per month for 28 months before reducing to zero over the final 12 months of operation.

This increase in total mining rate late in the mine life is noted as an area where improvements could be made, to assist with mining contract requirements, but does keep the overall cost of mining stable for the first 8 years of the operation.

Processing commences in Month 7 of the production schedule at a reduced rate of 1 million tonnes per annum. This reduced rate is maintained for three months, before increasing to the target rate of 4 million tonnes per annum for Phase 1 Oxide. In Year 5, the oxide and transitional material from the Phase 1 pits is exhausted and sufficient Primary mill feed material is available to commence Phase 2 processing. Phase 2 processing throughput is also assigned a 3 month ramp up period, before increasing to the target production rate of 2.4 million tonnes per year.

Processing of the primary ore through the Phase 2 roasting circuit continues until July of Year 12 when the primary mill feed is exhausted. The roasting circuit is then deactivated, and a straight CIL circuit is re-established to enable processing of the 2.3 million tonnes of predominantly transitional mill feed that has been mined and stockpiled during the excavation of the Phase 2 pits.

In Years 5 and 13, both Phase 1 and Phase 2 mill feed is processed. Processing tonnages have been calculated to ensure that the combined process feed does not exceed the capacity for each phase, pro-rated on months of operation in the year for the two phases. Year 5 also sees a sharp increase in strip ratio as Phase 1 stockpiles are processed while waste from Phase 2 is stripped.

Due to metallurgical differences, the transitional material at Makosa was divided into two separate domains: Makosa Main and Makosa East. All other material types at Makosa and all material types in the other regions consisted of one metallurgical domain only.

The production schedule yields 1,026 koz recovered ounces from a total mill feed tonnage of 36.6 million tonnes at a grade of 1.03 g/t.

Table 16.13 shows the ramp-up approach for the first year of operation. Tables 16.14 to 16.17 show the mining schedule while Table 16.18 shows the processing schedule for the life of the mine.

Figure 16.17 shows the annual ex-pit movement over the course of the mining schedule while Figure 16.18 shows the ex-pit movement by scheduling period (month). Figure 16.19 shows the annual processing schedule and Table 16.19 shows the pit progression by year.

Table 16.13 Mining and processing ramp-up in Year 1

Month	Total material mined target (t)	Material processed target (t)
1	-	-
2	-	-
3	-	-
4	-	-
5	500,000	-
6	750,000	-
7	750,000	168,478
8	750,000	168,478
9	750,000	163,043
10	750,000	339,726
11	750,000	339,726
12	1,500,000	339,276

Source: AMC, 2025.

Table 16.14 Makosa Main mining schedule

	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Total
Oxide Ore Tonnes(kt)	1,345.1	2,564.2	986.9	641.7	19.2	13.8	13.4	4.0	19.4	45.2	15.3	-	-	-	5,668.1
Oxide Ore Grade	1.19	1.14	1.07	0.92	0.84	0.76	0.86	0.75	1.40	0.96	1.27	-	-	-	1.12
MM Trans Ore Tonnes (kt)	167.8	736.9	774.0	613.1	153.9	163.9	96.4	1.6	90.4	103.1	140.1	2.3	-	-	3,043.4
MM Trans Ore Grade	1.08	1.22	1.17	1.05	1.22	1.20	0.96	0.93	1.21	0.97	1.06	1.22	-	-	1.14
ME Trans Ore Tonnes (kt)	-	445.5	16.2	200.7	3.4	-	3.4	124.7	-	92.3	0.0	-	-	-	886.1
ME Trans Ore Grade	-	1.01	1.05	0.90	1.30	-	0.76	0.93	-	1.05	0.64	-	-	-	0.98
Fresh Ore Tonnes (kt)	-	25.6	20.4	94.4	435.5	2,576.7	2,588.5	2,462.2	4,085.4	1,148.1	775.1	1,230.3	-	-	13,442.3
Fresh Ore Grade	-	0.97	1.03	1.14	1.18	1.24	1.13	1.12	1.21	1.13	1.02	1.14	-	-	1.16
Total Ore Tonnes (kt)	1,512.9	3,772.2	1,797.5	1,549.9	612.0	2,754.3	2,701.7	2,592.5	2,195.2	1,388.6	930.5	1,232.5	-	-	23,039.8
Total Ore Grade	1.18	1.14	1.11	0.98	1.18	1.24	1.12	1.11	1.21	1.11	1.03	1.14	-	-	1.14
Oxide LG Tonnes (kt)	613.5	1,219.7	588.4	407.8	53.5	41.4	35.9	17.6	9.1	27.2	21.7	-	-	-	3,035.9
Oxide LG Grade	0.44	0.44	0.44	0.44	0.41	0.41	0.41	0.43	0.44	0.43	0.40	-	-	-	0.44
MM Trans LG Tonnes (kt)	76.5	239.8	295.0	232.2	108.7	84.2	68.2	23.6	35.7	50.4	81.4	0.8	-	-	1,296.7
MM Trans LG Grade	0.46	0.45	0.46	0.46	0.46	0.44	0.43	0.39	0.46	0.44	0.43	0.39	-	-	0.45
ME Trans LG Tonnes (kt)	-	127.8	6.9	98.9	1.4	-	6.0	90.7	-	41.7	0.1	-	-	-	373.3
ME Trans LG Grade	-	0.49	0.45	0.48	0.46	-	0.45	0.46	-	0.47	0.38	-	-	-	0.47
Fresh LG Tonnes (kt)	-	1.4	2.0	4.5	24.6	113.4	130.2	131.4	91.9	33.7	30.4	55.4	-	-	619.0
Fresh LG Grade	-	0.57	0.58	0.58	0.58	0.58	0.58	0.58	0.58	0.58	0.58	0.58	-	-	0.58
Total LG Tonnes (kt)	690.1	1,588.7	892.3	743.4	188.2	239.0	240.3	263.4	136.7	153.0	133.6	56.2	-	-	5,324.9
Total LG Grade	0.44	0.45	0.45	0.45	0.46	0.50	0.51	0.51	0.54	0.48	0.46	0.58	-	-	0.46
Waste Tonnes (kt)	4,297.1	10,779.4	9,274.0	7,596.3	14,723.3	15,006.7	15,058.0	9,798.8	8,085.5	10,458.4	6,741.1	3,811.6	-	-	115,630.1
Total Tonnes Mined (kt)	6,500.0	16,140.3	11,963.7	9,889.6	15,523.5	18,000.0	18,000.0	12,654.8	10,417.4	11,906.2	7,805.1	5,100.4	-	-	143,901.0
Ore Strip Ratio	3.30	3.28	5.66	5.38	24.36	5.54	5.66	3.88	3.75	7.64	7.39	3.14	-	-	5.25
Ore+LG Strip Ratio	1.95	2.01	3.45	3.31	18.40	5.01	5.12	3.43	3.47	6.78	6.34	2.96	-	-	4.08

Notes:
- All tonnages are displayed as kilotonnes
- MM = Makosa Main
- ME = Makosa East
- LG = Low Grade material

Source: AMC, 2025.

Table 16.15 Makosa Tail mining schedule

	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Total
Oxide Ore Tonnes (kt)	-	104.5	441.6	260.7	-	-	-	4.6	3.0	-	5.6	-	-	-	820.0
Oxide Ore Grade	-	1.39	1.24	1.18	-	-	-	0.70	0.72	-	1.04	-	-	-	1.23
Trans Ore Tonnes (kt)	-	0.6	84.8	392.6	-	-	-	1.9	122.2	15.4	22.3	-	-	-	639.8
Trans Ore Grade	-	1.16	1.41	1.27	-	-	-	0.82	1.19	1.19	1.28	-	-	-	1.27
Fresh Ore Tonnes (kt)	-	-	13.1	9.4	-	-	-	-	257.3	1,222.6	1,662.5	767.3	-	-	3,932.1
Fresh Ore Grade	-	-	1.72	1.66	-	-	-	-	1.29	1.33	1.32	1.40	-	-	1.34
Total Ore Tonnes (kt)	-	105.1	539.5	662.6	-	-	-	6.6	382.4	1,238.0	1,690.4	767.3	-	-	5,391.8
Total Ore Grade	-	1.38	1.28	1.24	-	-	-	0.73	1.25	1.33	1.32	1.40	-	-	1.31
Oxide LG Tonnes (kt)	-	95.0	426.7	204.3	-	-	-	63.7	42.8	0.0	13.0	-	-	-	845.6
Oxide LG Grade	-	0.42	0.43	0.43	-	-	-	0.38	0.37	0.58	0.37	-	-	-	0.42
Trans LG Tonnes (kt)	-	0.4	72.7	225.2	-	-	-	3.5	195.9	8.7	19.1	-	-	-	525.4
Trans LG Grade	-	0.49	0.43	0.45	-	-	-	0.38	0.41	0.42	0.41	-	-	-	0.43
Fresh LG Tonnes (kt)	-	-	1.8	0.6	-	-	-	-	45.9	184.6	184.2	86.6	-	-	503.7
Fresh LG Grade	-	-	0.56	0.52	-	-	-	-	0.55	0.55	0.56	0.56	-	-	0.56
Total LG Tonnes (kt)	-	95.4	501.2	430.2	-	-	-	67.2	284.6	193.2	216.3	86.6	-	-	1,874.7
Total LG Grade	-	0.42	0.43	0.44	-	-	-	0.38	0.43	0.55	0.53	0.56	-	-	0.46
Waste Tonnes (kt)	-	1,659.2	3,390.7	1,017.5	-	-	-	5,271.5	0,415.5	0,568.8	3,288.3	2,358.6	-	-	47,970.1
Total Tonnes Mined (kt)	-	1,859.7	4,431.4	2,110.4	-	-	-	5,345.2	1,082.6	1,895.4	5,194.9	3,212.4	-	-	55,132.1
Ore Strip Ratio	-	16.70	7.21	2.18	-	-	-	812.91	27.98	8.69	7.99	3.19	-	-	9.24
Ore+LG Strip Ratio	-	8.28	3.26	0.93	-	-	-	71.46	15.61	7.38	6.97	2.76	-	-	6.60

Notes: All tonnages are displayed as kilotonnes. LG = Low Grade material.
Source: AMC, 2025.

Table 16.16 Baraka 3 mining schedule

	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Total
Oxide Ore Tonnes (kt)	-	-	73.0	362.2	111.6	-	-	-	-	-	-	-	-	-	546.8
Oxide Ore Grade	-	-	1.36	1.37	1.50	-	-	-	-	-	-	-	-	-	1.39
Trans Ore Tonnes (kt)	-	-	-	36.7	96.8	-	-	-	-	-	-	-	-	-	133.5
Trans Ore Grade	-	-	-	1.22	1.16	-	-	-	-	-	-	-	-	-	1.18
Fresh Ore Tonnes (kt)	-	-	-	2.1	10.9	-	-	-	-	-	-	-	-	-	13.0
Fresh Ore Grade	-	-	-	1.11	1.53	-	-	-	-	-	-	-	-	-	1.46
Total Ore Tonnes (kt)	-	-	73.0	401.0	219.3	-	-	-	-	-	-	-	-	-	693.3
Total Ore Grade	-	-	1.36	1.35	1.35	-	-	-	-	-	-	-	-	-	1.35
Oxide LG Tonnes (kt)	-	-	38.3	130.4	47.3	-	-	-	-	-	-	-	-	-	216.1
Oxide LG Grade	-	-	0.47	0.46	0.46	-	-	-	-	-	-	-	-	-	0.46
Trans LG Tonnes (kt)	-	-	-	22.7	29.7	-	-	-	-	-	-	-	-	-	52.3
Trans LG Grade	-	-	-	0.47	0.46	-	-	-	-	-	-	-	-	-	0.46
Fresh LG Tonnes (kt)	-	-	-	-	0.0	-	-	-	-	-	-	-	-	-	0.0
Fresh LG Grade	-	-	-	-	0.47	-	-	-	-	-	-	-	-	-	0.47
Total LG Tonnes (kt)	-	-	38.3	153.1	77.0	-	-	-	-	-	-	-	-	-	268.4
Total LG Grade	-	-	0.47	0.46	0.46	-	-	-	-	-	-	-	-	-	0.46
Waste Tonnes (kt)	-	-	1,493.5	5,445.9	2,180.2	-	-	-	-	-	-	-	-	-	9,119.6
Total Tonnes Mined (kt)	-	-	1,604.8	6,000.0	2,476.5	-	-	-	-	-	-	-	-	-	10,081.3
Ore Strip Ratio	-	-	20.99	13.96	10.29	-	-	-	-	-	-	-	-	-	13.54
Ore+LG Strip Ratio	-	-	13.42	9.83	7.36	-	-	-	-	-	-	-	-	-	9.48

Notes: All tonnages are displayed as kilotonnes. LG = Low Grade material.
Source: AMC, 2025.

Table 16.17 Combined mining schedule

	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Total
Oxide Ore Tonnes (kt)	1,345.1	2,668.7	1,501.5	1,264.6	130.8	13.8	13.4	8.6	22.3	45.2	20.9	-	-	-	7,034.9
Oxide Ore Grade	1.19	1.15	1.13	1.11	1.40	0.76	0.86	0.72	1.31	0.96	1.21	-	-	-	1.15
Trans Ore Tonnes (kt)	167.8	1,182.9	875.0	1,243.0	254.1	163.9	99.8	128.3	212.6	210.7	162.3	2.3	-	-	4,702.7
Trans Ore Grade	1.08	1.14	1.20	1.10	1.20	1.20	0.96	0.92	1.20	1.02	1.09	1.22	-	-	1.13
Fresh Ore Tonnes (kt)	-	25.6	33.5	105.9	446.5	2,576.7	2,588.5	2,462.2	2,342.7	2,370.7	2,437.6	1,997.5	-	-	17,387.4
Fresh Ore Grade	-	0.97	1.30	1.18	1.19	1.24	1.13	1.12	1.22	1.23	1.22	1.24	-	-	1.20
Total Ore Tonnes (kt)	1,512.9	3,877.3	2,410.0	2,613.6	831.4	2,754.3	2,701.7	2,599.1	2,577.6	2,626.6	2,620.8	1,999.8	-	-	29,124.9
Total Ore Grade	1.18	1.15	1.16	1.11	1.23	1.24	1.12	1.11	1.22	1.21	1.21	1.24	-	-	1.18
Oxide LG Tonnes (kt)	613.5	1,314.7	1,053.5	742.6	100.8	41.4	35.9	81.3	51.9	27.2	34.7	-	-	-	4,097.5
Oxide LG Grade	0.44	0.44	0.44	0.44	0.44	0.41	0.41	0.39	0.38	0.43	0.39	-	-	-	0.44
Trans LG Tonnes (kt)	76.5	368.0	374.5	579.0	139.8	84.2	74.2	117.8	231.6	100.8	100.5	0.8	-	-	2,247.8
Trans LG Grade	0.46	0.46	0.45	0.46	0.46	0.44	0.44	0.45	0.42	0.45	0.42	0.39	-	-	0.45
Fresh LG Tonnes (kt)	-	1.4	3.8	5.2	24.6	113.4	130.2	131.4	137.8	218.3	214.6	142.0	-	-	1,122.7
Fresh LG Grade	-	0.57	0.57	0.57	0.58	0.58	0.58	0.58	0.57	0.56	0.56	0.57	-	-	0.57
Total LG Tonnes (kt)	690.1	1,684.1	1,431.8	1,326.7	265.2	239.0	240.3	330.6	421.3	346.3	349.8	142.8	-	-	7,468.0
Total LG Grade	0.44	0.44	0.44	0.45	0.46	0.50	0.51	0.49	0.46	0.52	0.50	0.57	-	-	0.46
Waste Tonnes (kt)	4,297.1	12,438.6	14,158.2	14,059.7	16,903.5	15,006.7	15,058.0	15,070.3	18,501.1	21,027.1	20,029.3	6,170.1	-	-	172,719.8
Total Tonnes Mined (kt)	6,500.0	18,000.0	18,000.0	18,000.0	18,000.0	18,000.0	18,000.0	18,000.0	21,500.0	24,000.0	23,000.0	8,312.8	-	-	209,312.8
Ore Strip Ratio	3.30	3.64	6.47	5.89	20.65	5.54	5.66	5.93	7.34	8.14	7.78	3.16	-	-	6.19
Ore+LG Strip Ratio	1.95	2.24	3.69	3.57	15.42	5.01	5.12	5.14	6.17	7.07	6.74	2.88	-	-	4.72

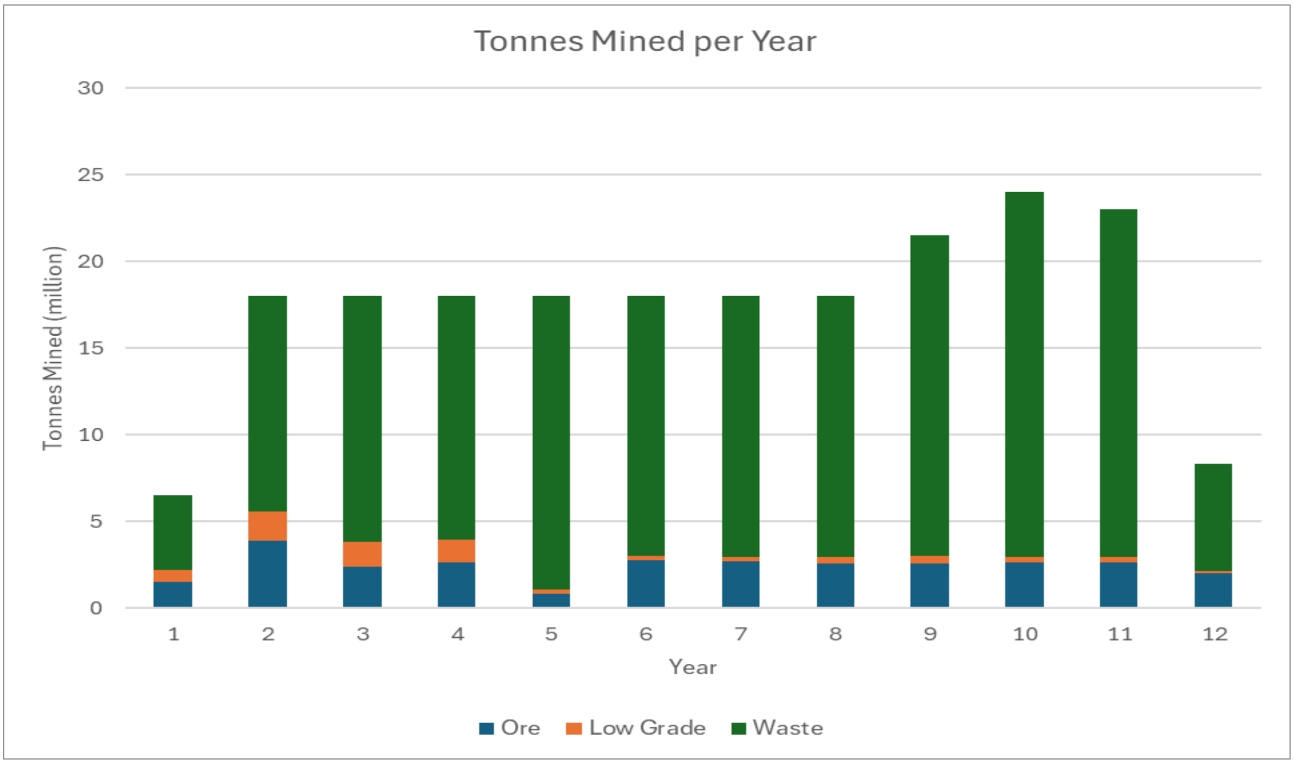
Notes: All tonnages are displayed as kilotonnes. LG = Low Grade material.
Source: AMC, 2025.

Table 16.18 Processing schedule

	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14 Q1	Total
Phase 1															
Processed Tonnes (kt)	1,519.2	4,000.0	4,000.0	4,000.0	2,296.5	-	-	-	-	-	-	-	1,676.7	590.5	18,082.9
Processed Grade (g/t)	1.16	1.13	0.86	0.86	0.55	-	-	-	-	-	-	-	0.90	0.43	0.90
Contained Metal (oz)	56,770	145,391	111,227	110,057	40,704	-	-	-	-	-	-	-	48,677	8,232	521,058
Recovered Metal (oz)	52,068	128,652	99,657	97,625	35,091	-	-	-	-	-	-	-	40,577	6,718	460,387
Phase 2															
Processed Tonnes (kt)	-	-	-	-	425.0	2,400.0	2,400.0	2,400.0	2,400.0	2,400.0	2,400.0	2,400.0	1,285.1	-	18,510.1
Processed Grade (g/t)	-	-	-	-	1.22	1.24	1.14	1.13	1.21	1.24	1.23	1.20	0.63	-	1.16
Contained Metal (oz)	-	-	-	-	16,631	95,772	88,319	87,209	93,627	95,829	94,930	92,671	25,884	-	690,870
Recovered Metal (oz)	-	-	-	-	13,061	77,575	71,538	70,639	76,579	81,276	81,827	77,485	21,600	-	571,581
Total Schedule															
Processed Tonnes (kt)	1,519.2	4,000.0	4,000.0	4,000.0	2,721.5	2,400.0	2,400.0	2,400.0	2,400.0	2,400.0	2,400.0	2,400.0	2,961.8	590.5	36,593.0
Processed Grade (g/t)	1.16	1.13	0.86	0.86	0.66	1.24	1.14	1.13	1.21	1.24	1.23	1.20	0.78	0.43	1.03
Contained Metal (oz)	56,770	145,391	111,227	110,057	57,335	95,772	88,319	87,209	93,627	95,829	94,930	92,671	74,560	8,232	1,211,928
Recovered Metal (oz)	52,068	128,652	99,657	97,625	48,152	77,575	71,538	70,639	76,579	81,276	81,827	77,485	62,177	6,718	1,031,968

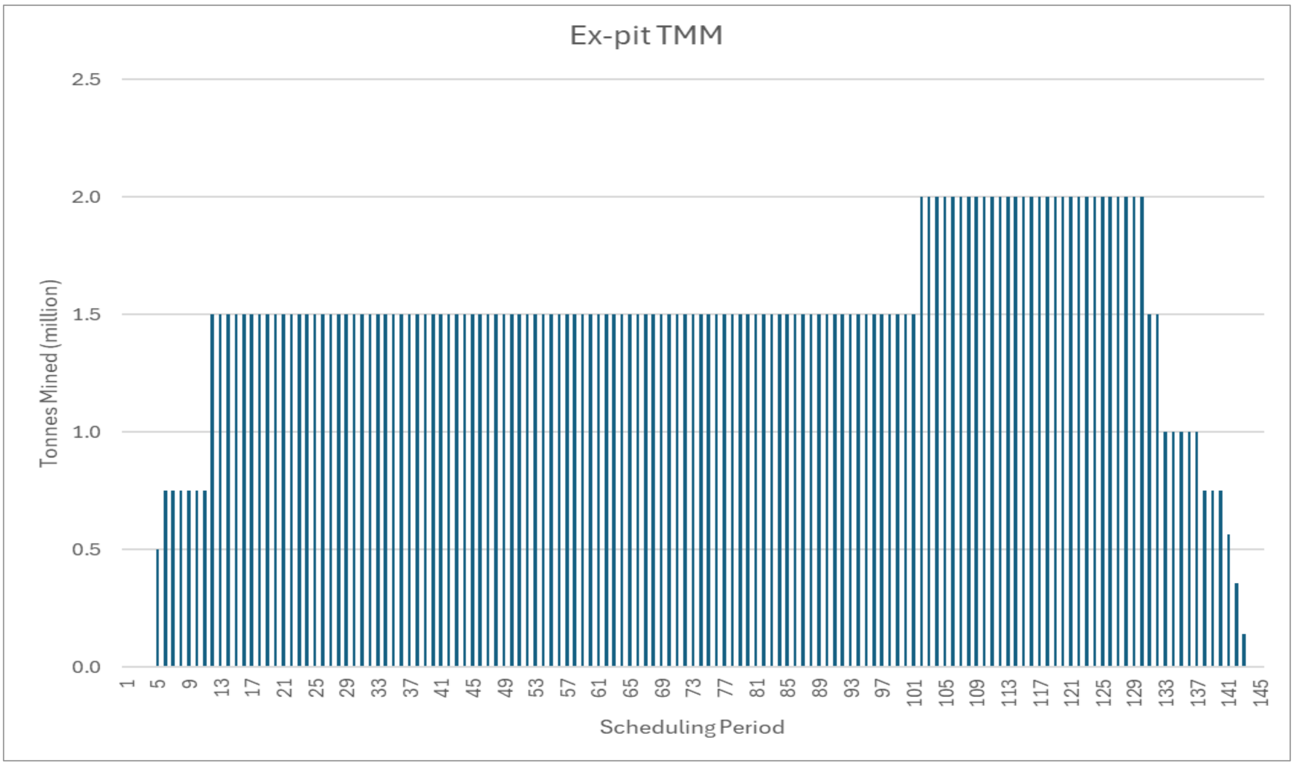
Notes: All tonnages are displayed as kilotonnes.
Source: AMC, 2025.

Figure 16.17 Annual ex-pit movement



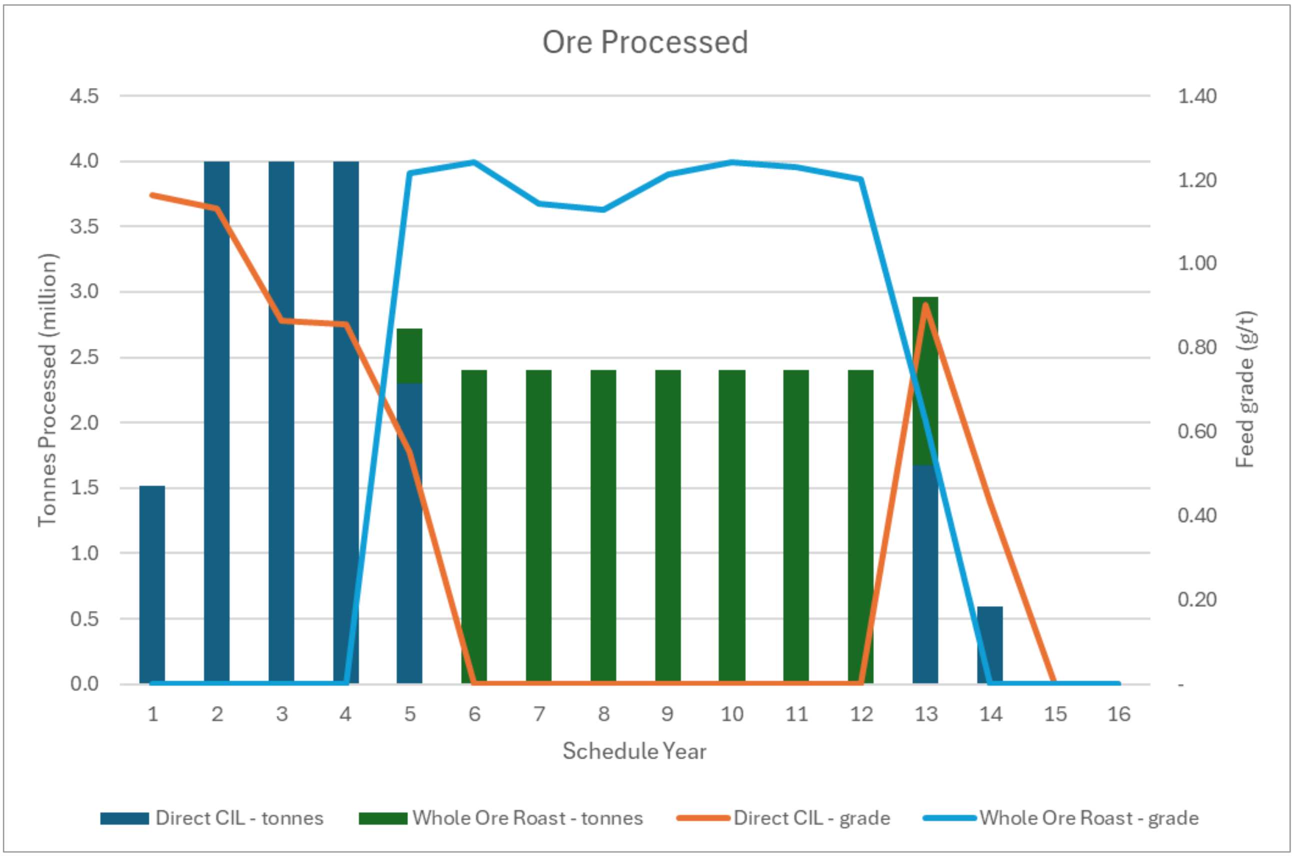
Source: AMC, January 2026.

Figure 16.18 Monthly ex-pit movement



Source: AMC, January 2026.

Figure 16.19 Annual processing



Source: AMC, January 2026.

Table 16.19 Pit progression

Bench	MM Pit 1-1	MM Pit 1-2	MM Pit 1-3	MM Pit 2-1	MM Pit 2-2	MM Pit 3-1	MM Pit 3-2	MM Pit 3-3	MM Pit 4	MM Pit 5-1	MM Pit 5-2	MM Pit 5-3	MT Pit 21	MT Pit 22	MT Pit 23	MT Pit 24
1	Yr 4	Yr 1	Yr 1	Yr 2	Yr 2	Yr 2	Yr 1	Yr 3	Yr 1	Yr 2	Yr 2	Yr 2	Yr 2	Yr 3	Yr 3	Yr 2
2	Yr 4	Yr 1	Yr 1	Yr 2	Yr 2	Yr 2	Yr 1	Yr 3	Yr 1	Yr 2	Yr 2	Yr 2	Yr 2	Yr 3	Yr 3	Yr 2
3	Yr 4	Yr 1	Yr 1	Yr 2	Yr 2	Yr 2	Yr 1	Yr 3	Yr 1	Yr 2	Yr 3	Yr 2	Yr 2	Yr 3	Yr 3	Yr 2
4	Yr 4	Yr 1	Yr 1	Yr 2	Yr 2	Yr 2	Yr 1	Yr 3	Yr 1	Yr 2	Yr 3	Yr 2	Yr 2	Yr 3	Yr 3	Yr 3
5	Yr 4	Yr 1	Yr 2	Yr 2	Yr 3	Yr 2	Yr 2	Yr 4	Yr 2	Yr 3	Yr 3	Yr 2	Yr 2	Yr 3	Yr 3	Yr 3
6	Yr 4	Yr 1	Yr 2	Yr 2	Yr 3	Yr 2	Yr 2	Yr 4	Yr 2	Yr 3	Yr 3		Yr 3	Yr 3	Yr 3	Yr 4
7	Yr 4	Yr 1	Yr 2	Yr 2	Yr 3	Yr 2	Yr 2	Yr 4	Yr 2	Yr 3	Yr 3		Yr 3	Yr 3		Yr 4
8	Yr 4		Yr 2	Yr 2			Yr 2	Yr 4	Yr 2	Yr 4	Yr 4		Yr 3	Yr 3		Yr 4
9	Yr 4						Yr 2	Yr 4	Yr 2	Yr 4	Yr 4		Yr 3	Yr 3		Yr 4
10							Yr 2	Yr 4	Yr 2	Yr 4						
11								Yr 4		Yr 4						
12										Yr 4						
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Bench	B3E Pit 41	B3W Pit 51	B3W Pit 52	B3W Pit 53	MM Pit 11-2	MM Pit 11-3	MM Pit 12	MM Pit 13-1	MM Pit 13-2	MM Pit 14	MM Pit 15-1	MM Pit 15-2	MM Pit 15-3	MT Pit 31	MT Pit 34
1	Yr 3	Yr 3	Yr 3	Yr 3	Yr 6	Yr 4	Yr 7	Yr 10		Yr 10	Yr 5	Yr 9		Yr 11	Yr 8
2	Yr 3	Yr 3	Yr 3	Yr 3	Yr 6	Yr 4	Yr 7	Yr 10		Yr 10	Yr 5	Yr 9		Yr 11	Yr 8
3	Yr 4		Yr 3	Yr 3	Yr 6	Yr 5	Yr 7	Yr 10		Yr 10	Yr 5	Yr 9		Yr 11	Yr 8
4	Yr 4		Yr 3	Yr 4	Yr 6	Yr 5	Yr 7	Yr 10		Yr 10	Yr 5	Yr 9		Yr 11	Yr 8
5	Yr 4		Yr 4	Yr 4	Yr 6	Yr 5	Yr 7	Yr 11		Yr 10	Yr 5	Yr 9		Yr 11	Yr 8
6	Yr 5			Yr 4	Yr 6	Yr 5	Yr 7	Yr 11		Yr 10	Yr 5	Yr 9		Yr 11	Yr 8
7	Yr 5			Yr 4	Yr 7	Yr 5	Yr 8	Yr 11		Yr 10	Yr 5	Yr 9		Yr 11	Yr 8
8	Yr 5			Yr 4	Yr 7	Yr 5	Yr 8	Yr 11		Yr 10	Yr 6	YR 10		Yr 11	Yr 9
9	Yr 5			Yr 4	Yr 7	Yr 6	Yr 8	Yr 11		Yr 10	Yr 6	YR 10		Yr 11	Yr 9
10	Yr 5			Yr 4	Yr 7	Yr 6	Yr 8	Yr 12		Yr 10	Yr 6	YR 10	Yr 5	Yr 11	Yr 9
11	Yr 5			Yr 4	Yr 7	Yr 6	Yr 8	Yr 12		Yr 10	Yr 8	YR 10	Yr 5	Yr 11	Yr 9
12	Yr 5			Yr 4	Yr 7	Yr 6	Yr 8	Yr 12	Yr 9	Yr 10	Yr 9	YR 10	Yr 5	Yr 12	Yr 10
13	Yr 5			Yr 4	Yr 7	Yr 6	Yr 8	Yr 12	Yr 9	Yr 11	Yr 9	YR 10		Yr 12	Yr 10
14				Yr 4	Yr 8	Yr 6	Yr 8	Yr 12	Yr 9	Yr 11	Yr 9	YR 10		Yr 12	Yr 10
15					Yr 8	Yr 7	Yr 9	Yr 12	Yr 9	Yr 11	Yr 9	YR 10		Yr 12	Yr 10
16					Yr 8	Yr 7	Yr 9	Yr 12	Yr 9	Yr 11	Yr 9	YR 10		Yr 12	Yr 10
17					Yr 8	Yr 7	Yr 9	Yr 12	Yr 10	Yr 11	Yr 9	YR 10		Yr 12	Yr 11
18					Yr 8	Yr 7	Yr 9	Yr 12	Yr 10	Yr 12	Yr 9	YR 10		Yr 12	Yr 11
19					Yr 8	Yr 7	Yr 9	Yr 12		Yr 12	Yr 9	YR 10		Yr 12	Yr 11
20					Yr 8	Yr 7	Yr 9	Yr 12		Yr 12	Yr 9	YR 10		Yr 12	Yr 11
21					Yr 8	Yr 7				Yr 12	Yr 9	YR 10			Yr 11
22					Yr 8	Yr 8					Yr 10				Yr 11
23					Yr 8	Yr 8									Yr 11
24					Yr 8	Yr 8									Yr 11
25					Yr 8	Yr 8									Yr 11
26						Yr 8									Yr 11
27															Yr 12
28															Yr 12
29															Yr 12
30															Yr 12
31															Yr 12
32															Yr 12
33															Yr 12
34															Yr 12

Source: AMC, 2025.

16.8 Mining equipment

A detailed equipment study has not been conducted. It is assumed that a mining contractor will be employed for all mining activities and the ultimate fleet selection will be the contractor's choice.

Although each scheduling period has been restricted to 3-4 working areas, the geographic extent of the project will lend itself to multiple faces / pits in operation in any one period, with regular relocation of loading equipment between periods. Therefore, loading equipment that can be transferred quickly over long distances will be of benefit.

Contrary to the above, the orebody will be quite selective in many areas. Despite being highly mobile, a front-end loader will most likely not have the accuracy required to keep dilution and ore loss to a minimum. AMC believes that excavators will be the best option, coupled with a suitable sized float to quickly transport the loading units between pits.

It is expected that the mining fleet will be comprised of on-road haul trucks or small rigid body haul trucks. It is unlikely that the mining fleet will consist of trucks larger than those with a tray capacity of 90 t, similar to a Caterpillar 777.

Three to four excavators with operating weights of 100 t to 120 t will likely be required for mining operations when matched with rigid body haul trucks of 75 to 100 t capacity. Using smaller road trucks will require six to seven 50 tonne excavators. Given the size and configuration of the pits along with the variability of the orebody, it is likely that a mixed fleet may be the most economic and provide the best trade-off between productivity and selectivity.

Suitable ancillary equipment will be required typical of any open pit operation. AMC expects that additional graders and water carts will be required, given the geographic extent of the operations.

16.9 Operating strategy

The project has been developed based on the use of a mining contractor, on a cost per tonne basis. All costs, including explosives and fuel, are included in this cost per tonne rate, although the supply of one or both of these may be through Thor or under a separate contract.

Thor personnel will be responsible for all geological, geotechnical, and mine engineering activities. Thor's mining team will likely include:

- A Mine Manager and Alternate Mine Manager
- Geologists (resource and grade control) and geology assistants
- Mining Engineers (scheduling will be undertaken by the principal to a weekly level)
- Geotechnical and Hydrogeological staff
- Surveyors
- Contract Management Personnel (including supervisors)

Thor will not undertake any mining activities directly. Therefore, all mobile maintenance will be the responsibility of the contractor. This includes waste oil and other waste management from such maintenance activities.

It is likely that the haulage of mill feed from Baraka 3 will be performed under a second haulage contract and provides additional opportunity for local businesses. This will be investigated further prior to the commencement of Baraka 3 operations.

17 Recovery methods

17.1 Introduction

The process plant design for the Project is based on a robust metallurgical flowsheet designed for optimum recovery with minimum operating costs. A two-stage process has been designed. Phase 1 focuses on the recovery of cyanide-soluble gold and Phase 2 focuses on the recovery of gold hosted in sulphides and silicates (refractory gold). The gold amenable to cyanide leaching is primarily hosted in oxidised ore horizons with high leaching recoveries. The Phase 1 plant is based on well proven CIL process technology. Increased proportions of refractory gold occur as the transition is made from oxide to transition and then to fresh (sulphidic / silicate locked). Carbon in both organic and inorganic forms exists in the deposit making some ores preg-robbing. Phase 2 adopted a plant design based on the upgrading of iron ores by suspension roasting (SR). The design criteria of the Douta process facility were devised based on metallurgical test work results obtained and some assumptions, where necessary. The plant design is based on the treatment of ore from the Douta pit over the life-of-mine (LOM).

17.2 Process design philosophy

The design philosophy is based on separation of the processing of ore types. Oxidised ore types are amenable to CIL processes. Fresh ore types demonstrate lower, variable recoveries by direct CIL due to an increased refractory component. Gold is hosted in pyrite and arsenopyrite and also in silicates. The ore with gold-bearing sulphides and silicates is subjected to roasting in a heavy fuel oil (HFO) suspension roaster designed to remove the organic carbon minerals, oxidise sulphides and thermally crack silicates and sulphides for contact by lixiviants. The roasting product is reground to further liberate gold minerals. Regrinding is followed by a conventional CIL circuit.

17.2.1 Ore characteristics

The comminution test results, together with the specific gravity determinations obtained from test work, were used to characterise the ore. The test work results derived from variability testing on the various composites were analysed and the comminution parameters characterising the orebodies were obtained and recommended by IMO.

17.2.2 Plant operating schedule

The plant has been designed for a throughput of 4 Mtpa for Phase 1 and 2.4 Mtpa for Phase 2. The average LOM milling rates by ore type are 28.68% oxide, 14.47% transitional, and 56.84% fresh as given in the Project mining schedule. Oxide, transitional, and fresh ore crushing will be via a primary jaw crusher, with the overall utilisation of crushing assumed at 75% based on operating experience from similar plants. The overall utilisation of the milling plant will be assumed at 90.4% based on 330 operating days per year.

17.2.3 Plant recovery

The laboratory dissolution test results that were obtained from metallurgical tests on composites and variability samples have been used to determine the recoveries in the design basis of the Project.

17.3 Process description

The processing circuit is to be constructed in two phases as follows:

- Phase 1 will treat oxide and transitional ores via a conventional CIL circuit.
- Phase 2 will treat fresh ore via suspension roasting. The roasted product will be reground and treated in a reduced capacity CIL circuit (cf. Phase 1).

Design throughput for Phase 1 is 4 Mtpa of ore to comminution. The proposed process plant design for Phase 1 is based on established gravity / CIL technology, which consists of crushing, milling, gravity recovery of free gold followed by leaching / adsorption of gravity tailings, elution and gold smelting to recover gold from loaded carbon, cyanide destruction in plant tailings and tailings disposal. Services to the process plant will include reagent mixing, storage and distribution, power and water supply, oxygen generation and compressed air services. The plant will operate in the Phase 1 configuration until oxide ore is depleted which is nominally planned to be during Year 5 of the Project.

The plant will treat 2.4 Mtpa of fresh, sulphide ore in Phase 2. A roasting circuit treating the comminution product stream will be added to the flowsheet for Phase 2. The suspension roasting process will expose refractory gold particles prior to cyanide leaching. The circuit will consist of a pre-roasting dewatering stage, pre-roasting product storage silo, suspension roasting, and calcine repulping and regrinding.

The overall material flow commences with primary crushing of ROM ore using a jaw crusher and stockpiled ahead of the plant.

Milling will consist of a primary semi-autogenous grinding (SAG) mill, secondary ball mill, and a pebble crusher. The ball mill will be in closed circuit with a hydrocyclone cluster. Pebbles generated from the SAG mill will be conveyed to a pebble crusher where they will be further reduced in size and recirculated to the SAG mill. The discharge from both mills will be combined in a cyclone feed sump and will be pumped to the cyclone cluster for classification.

A proportion of the cyclone underflow will be bled to the gravity circuit for recovery of gravity gold, with the balance gravitating to the ball mill for further size reduction. Gold will be recovered from the gravity concentrates through a combination of intensive cyanidation and electrowinning facilities. The gravity recovery tailings will be transferred back to the mill feed for further liberation. Gold that is not gravity recoverable is recovered through the CIL process.

The overflow from the cyclone cluster will be pumped to the CIL circuit when the oxide / transitional ore is treated and to the pre-roasting dewatering circuit when fresh ore is treated. Slurry from cyclone overflow will feed a ten-stage CIL circuit when treating oxide / transitional, where gold will be dissolved and adsorbed onto activated carbon. Cyanide content in plant tailings will be reduced using the INCO Sulphur Dioxide / Air Process. The resultant CIL tailings will then be pumped to the tailing thickener prior to being pumped to the TSFs. When treating fresh ore, slurry from cyclone overflow will feed the pre-roasting dewatering thickener and filter press followed by suspension roasting, where gold contained minerals will be calcine roasted. The roaster product will be reground before reporting to the CIL circuit.

Loaded carbon from the CIL circuit will be acid-washed prior to elution, followed by reactivation of the eluted carbon. The solution from the elution circuit will be subjected to electrowinning, where gold will be deposited onto cathodes as sludge. Periodically, the sludge will be washed off the cathodes and dried. The dried gold “sludge” will then be smelted to produce gold doré, which will be shipped to a precious metals refinery.

An overall process flow diagram of the Phase 1 process plant is shown in Figure 17.1. The design of the plant consists of the following basic sections:

- Primary crusher, crushed ore stockpile (COS), stockpile reclaim system
- Milling
- Gravity and intensive cyanidation
- CIL
- Cyanide destruction

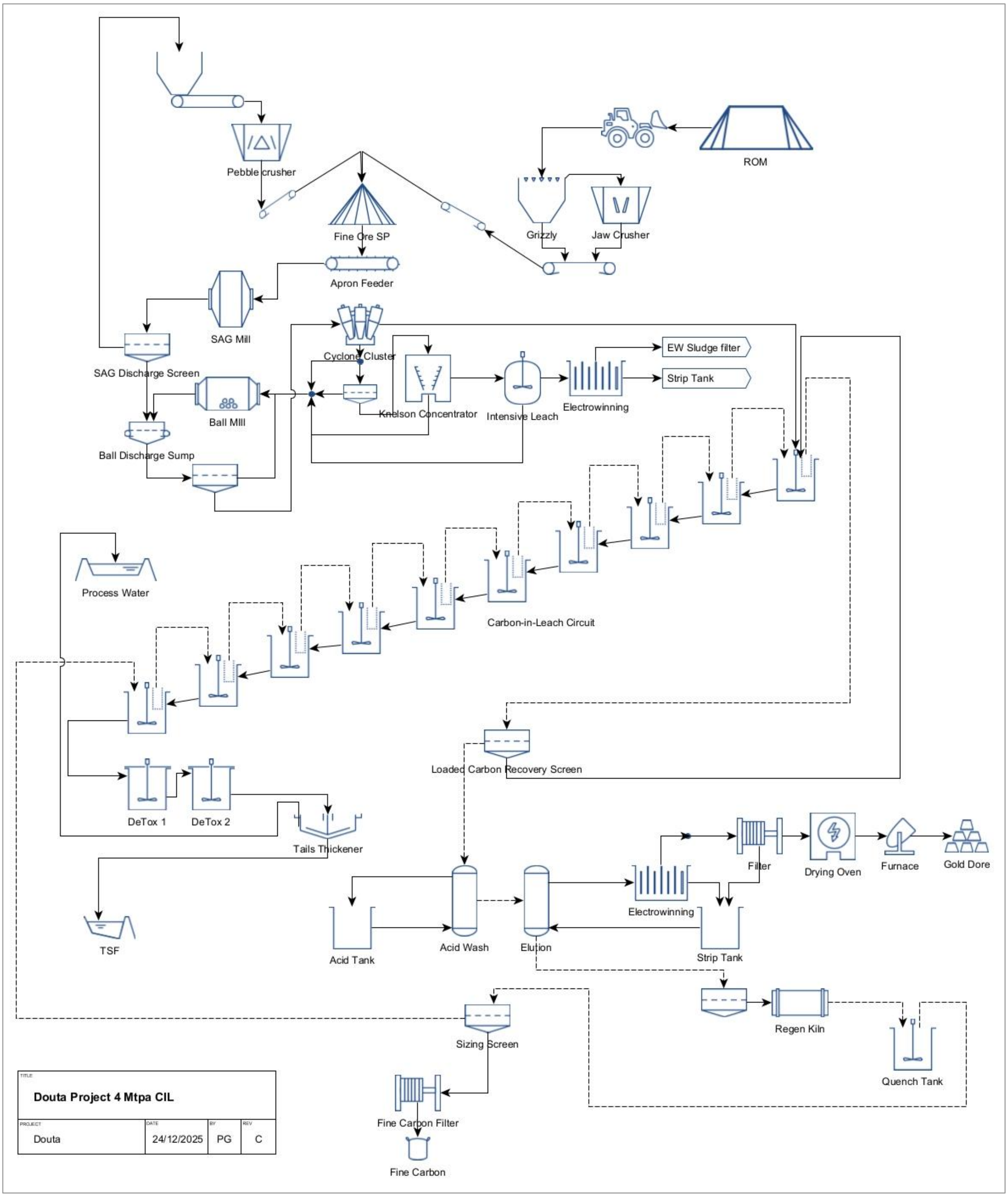
- Tailings thickener, TSF
- Acid wash
- Elution
- Electrowinning and gold room
- Carbon regeneration
- Gold room
- Reagents services
- Air services
- Oxygen plant
- Plant water services

Phase 2 will incorporate the following additional sections:

- Pre-roasting dewatering circuit – thickener, press filter
- Pre-roasting product storage silo
- Suspension roasting
- Repulping and Regrinding circuit

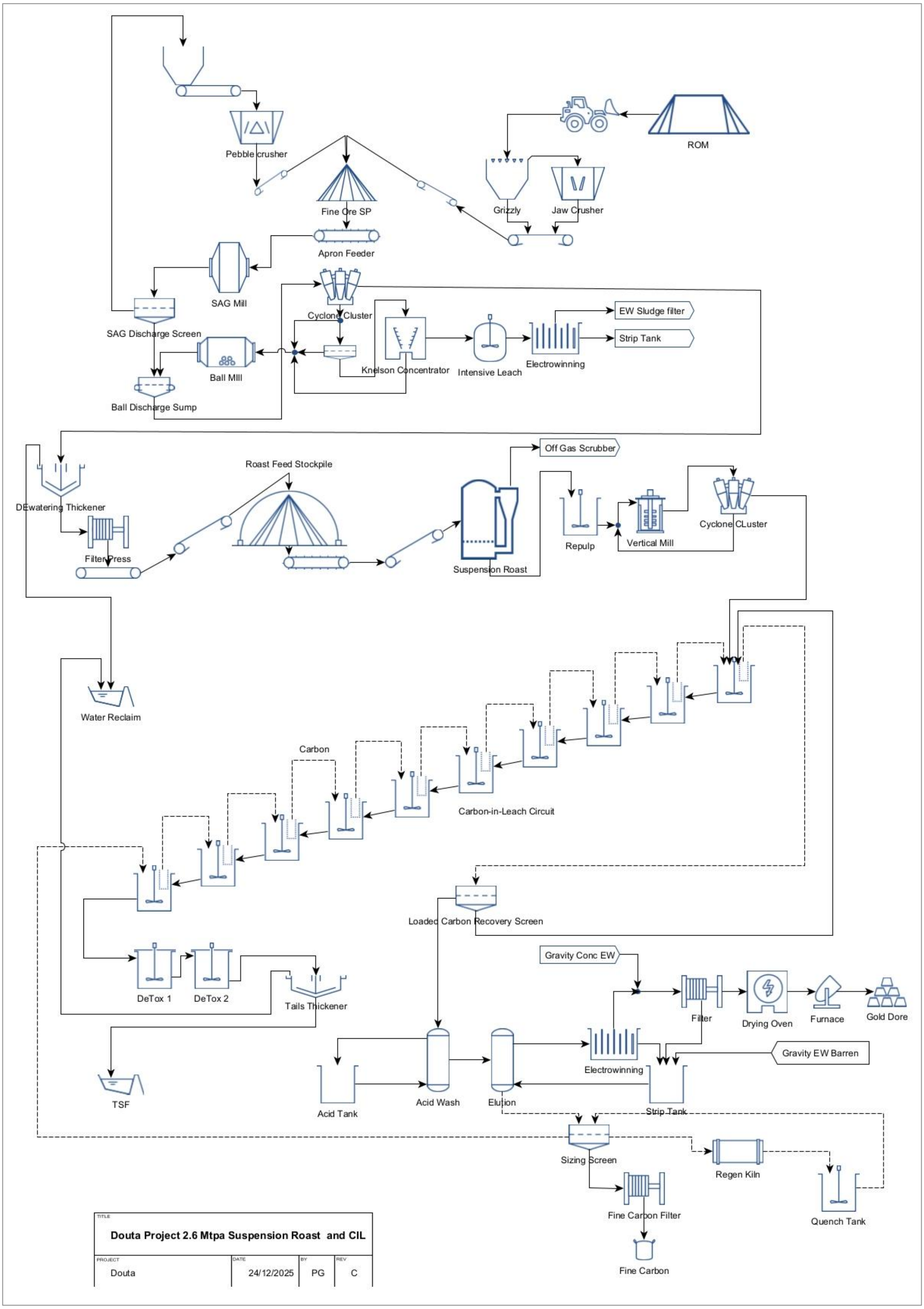
An overall process flow diagram of the Phase 2 process plant is shown in Figure 17.2.

Figure 17.1 Douta flowsheet Phase 1 CIL



Source: Thor, NORINCO.

Figure 17.2 Douta flowsheet Phase 2 SR-CIL



Source: Thor, NORINCO.

17.3.1 Primary crushing and stockpile

The design of the crushing and milling circuit was based on NORINCO experience with previous projects when treating material with similar characteristics, together with recommendations obtained from IMO.

Oxide, transitional and fresh ores are treated separately. Year 1 to Year 5 of the LOM will process high oxide / transitional ore. Fresh ore will feed the plant from Year 5. There will be a single primary crusher station to treat oxide / transitional and fresh ore from Douta.

Ore from the pits will be direct dumped into the crusher by trucks. Front-end loader (FEL) operation will also be provided. A fixed rock breaker will be utilised to break oversize rocks.

Ore will be crushed by the jaw crusher and withdrawn from the discharge pocket by the variable speed primary crusher apron feeder. This apron feeder will discharge ore onto the stockpile feed conveyor. A weightometer will indicate the feed tonnage and be used to control the apron feeder speed. The stockpile feed conveyor will discharge ore onto the crushed ore stockpile. The stockpile will have a live capacity of 24 hours of mill feed at 4 Mtpa. Ore will be withdrawn from the stockpile by four variable speed apron feeders. These feeders will discharge ore onto the SAG mill feed conveyor.

The crushing circuit will be serviced by a dust collection system.

An emergency ore bin is planned when the stockpile conveyor is offline. Loaders will withdraw ore from the crushed ore stockpile, feed the bin, and feed the SAG mill feed conveyor to maintain production. Mill area spillage slurry or ores can be fed to this bin.

17.3.2 Comminution circuit

The comminution circuit is a conventional SABC (SAG mill, ball mill, pebble crusher) circuit with a portion of the cyclone underflow feeding a Knelson concentrator. The SAG mill is 8.53 m by 4.5 m EGL (Effective Grinding Length) with 9 MW VFD (variable frequency drive) single drive motor. The ball mill is 7.32 m by 11.0 m EGL with 9 MW VFD single drive motor.

Crushed ore will be fed directly into the SAG mill feed chute. Process water addition to the SAG mill feed will be in ratio to the ore feed rate in order to maintain a relatively constant mill feed slurry density to optimise grinding efficiency.

Lime, used for pH control in the leach circuit, will be metered directly onto the mill feed conveyor using a variable speed rotary feeder in Phase 1. The speed of the feeder will be varied according to the mill feed tonnage. Lime milk will be prepared and dosed to the leaching circuit in Phase 2.

The SAG mill design is a variable speed grate discharge design which will allow slurry and small pebbles to pass out of the mill. The SAG mill product will flow to a single deck vibrating screen for removal of pebbles. Screen undersize will be combined with ball mill discharge in the ball mill discharge sump. After dilution with process water, the slurry will pass to a single deck screen where undersize will be pumped to the cyclone cluster for classification. Screen oversize will be recycled to the ball mill feed chute. Process water addition to the ball mill discharge sump will be automatically controlled to maintain a constant cyclone feed density.

Slurry from the cyclone underflow will be split, one portion will feed the gravity circuit and the other portion will be returned to the ball mill feed chute. Gravity tails will be pumped to the ball mill feed chute.

The ball mill trommel screen undersize will gravitate to the mill discharge hopper.

The SAG mill and ball mill grinding media will be added to maintain the required mill load.

The grinding area will be serviced by dedicated vertical sump pumps which will allow spillage and clean up to be returned to the circuit via the cyclone feed sump.

Oversize from the SAG mill discharge screen will be conveyed to the pebble crusher feed bin via a series of belt conveyors. A self-cleaning belt magnet will be positioned at the head chute of the first conveyor to remove any scrap metal and steel media which could potentially damage the pebble crusher. A pebble crusher feed weightometer will be installed prior to the pebble crusher feed bin. The feed bin will provide surge capacity ahead of the pebble crusher and allow a controlled feed to be presented to the crusher.

Downstream of the cross-belt magnet, the pebbles will pass under a metal detector prior to discharging into the pebble crusher feed bin. Should the pebble crusher not be operational, a diverter gate ahead of the pebble crusher feed bin will allow pebbles to bypass the pebble bin and crusher and feed directly to a reject storage bin. Similarly, should the metal detector detect tramp metal (not removed by the cross-belt magnet), the diverter gate ahead of the pebble crusher feed bin will automatically allow pebbles to bypass the pebble bin and crusher and feed directly to the reject storage bin.

Pebbles will be withdrawn from the pebble crusher feed bin by a variable speed vibrating feeder. The pebble crusher will discharge crushed pebbles directly onto the pebble crusher discharge conveyor which will return the crushed pebbles to the SAG mill feed conveyor.

The pebble crusher discharge conveyor will be fitted with a weightometer for process control purposes.

A vibrating trash screen will be included in the design prior to the leach to remove trash material. The trash screen will be configured such that the undersize from the trash screen will gravitate to a sump and will be pumped to the CIL.

17.3.3 Gravity and intensive cyanidation

The results of the emulating GRG batch laboratory tests indicated that free gold is available mainly in oxide and transitional ore. In addition, intensive cyanidation of the resultant gravity concentrates from most of the ore types showed favourable leach kinetics and gold extraction. A gravity circuit consisting of a centrifugal concentrator, intensive cyanidation reactor and a dedicated electrowinning cell for handling the pregnant leach solution will be implemented to recover free gold from a portion of the cyclone underflow. Simulation results from the GRG tests and the concentrate leach extraction parameters obtained from the tests were used as the basis of the design.

Feed for the gravity circuit will be taken as a bleed from the cyclone underflow. The circuit will consist of a scalping screen, a centrifugal concentrator (Knelson) and an Acacia reactor (in-line intensive cyanide leach reactor) located in the milling area, and dedicated electrowinning cell located in the gold room.

The concentrator will be operated on a semi-batch basis with the periodic discharge of the coarse, high specific gravity material (gravity concentrate) via a sealed and protected pipeline to the Acacia concentrate storage hopper. The tails slurry from the centrifugal concentrator will gravitate to the mill feed chute for further processing.

Gravity concentrates from the centrifugal concentrator will be processed in batches through the Acacia. The concentrates will be leached under intensive cyanidation conditions in the reactor. The pregnant liquor will be pumped to a small storage tank located in the gold room prior to the solution being electrowon in a dedicated electrowinning cell.

The cathodes from the gravity electrowinning cell may be treated separately to assist in metallurgical accounting. Spent electrolyte will be recycled to the CIL circuit.

17.3.4 CIL

Cyanidation tests conducted on oxide and fresh gravity tailings samples indicate reasonable gold dissolution within a leaching time of 24 h for both oxide and fresh feed.

Trash screen underflow from the grinding circuit will gravitate the CIL circuit. The CIL circuit will consist of ten CIL tanks. The tanks will be interconnected with launders and slurry will flow by gravity through the tank train. Each tank will be fitted with a dual impeller, mechanical agitator to ensure uniform mixing and a mechanically swept woven wire inter-tank screen to retain the carbon. All tanks will be fitted with bypass facilities to allow any tank to be removed from service for agitator or screen maintenance.

Lime added directly to the mill feed conveyor in Phase 1 and lime milk added to the CIL tanks in Phase 2 will ensure that the slurry pH is suitable for cyanidation.

Fresh / regenerated carbon will be returned to the circuit at CIL Tank 10 and advanced counter current to the slurry flow by pumping slurry and carbon using open, recessed impeller pumps from CIL Tank 10 to CIL Tank 9 and so on. An open, recessed impeller pump will be used to transfer slurry from CIL Tank 1 to the loaded carbon recovery screen. The carbon will be washed and dewatered on the loaded carbon recovery screen prior to reporting to the acid wash column. The associated slurry and wash water will return to the leach feed distribution box.

Slurry from the last CIL tank (CIL tails) will gravitate to the vibrating carbon safety screen to recover any carbon leaking from worn screens or overflowing tanks. Screen underflow will gravitate to tailing circuit. Screen oversize (recovered carbon) will be collected in the fine carbon bin for potential return to the circuit.

Barren carbon returning to the adsorption circuit from the carbon regeneration kiln will be screened on the sizing screen to remove fine carbon and quench water. The sized and regenerated carbon will report directly to CIL Tank 10.

A spare inter-tank screen will be provided to allow maintenance on screens to be conducted in a timely manner without the need to bypass CIL tanks for extended periods.

A cyanide analyser will be provided to measure and optimise free cyanide levels in the first CIL tank where the bulk on the cyanide solution is added and in two other CIL tanks down the CIL train. Allowance is also made for caustic addition for pH adjustment.

A hydrogen cyanide (HCN) gas monitor will also be installed in the leach area.

Oxygen is injected into the first five CIL tanks by means of spargers. Oxygen will be fed from a dedicated oxygen plant. A flowmeter will be installed on each line feeding oxygen, and the flow will be manually set with hand valves. Monitoring of the dissolved oxygen in each tank will be conducted manually.

An event pond will be provided to accommodate the necessary volume according to the industry standards. The event pond will have a submersible pump for recovery of slurry and / or liquor from the pond.

The design of the CIL area will also incorporate a tower crane which will be used to facilitate cleaning of the inter-stage screens and general maintenance.

17.3.5 Acid wash, elution

17.3.5.1 Acid wash

The loaded carbon batch from the first CIL tank is washed with clean spray water on the loaded carbon screen and is discharged directly into the acid wash column. Once a batch has been accumulated, the carbon is washed with dilute hydrochloric acid to remove scale that builds up on the carbon in the CIL circuit.

The acid wash process is carried out in three steps: elutriation, acid wash and rinse. At the end of the acid wash process, a batch of loaded carbon is ready for transfer into elution.

The carbon batch is washed with raw water to remove any light trash such as slimes, plastic and organic fibrous material captured in the bed. Blowing small amounts of air through the acid wash column often assists in lifting trapped trash during the elutriation process. The trash is flushed to the tails screen feed box via a gravity pipeline.

The dilute Hydrochloric acid (HCl) solution is made up in the acid wash tank using concentrated HCl and raw water; Dilute acid is pumped through the loaded carbon bed in the acid wash column using the acid wash pump.

After being washed with HCl, the carbon is rinsed with raw water and the rinse effluent is directed to the tails sump.

The acid-washed carbon batch is hydraulically transferred into the elution column.

A spillage pump is provided in the acid wash section. Spillage is pumped to the tails screen feed box. Two safety showers are provided in this area. Each safety shower is activated by a foot pedal and is equipped with an eye bath.

17.3.5.2 Elution

Acid washed carbon is eluted by pumping a hot caustic cyanide solution (eluate) through the column at 140°C and a maximum operational pressure of 600 kPa. Gold absorbed onto loaded carbon is eluted off the carbon and recovered in the eluate solution.

Before introduction into the elution column, the eluate is heated to 140°C through a heating system consisting of a carbon steel flame tube style direct eluate heater. Eluate is pumped directly into contact with the outside of these tubes under pressure from the eluate pump. This is coupled with a regenerative heat exchanger which acts to preheat the eluate by recovering thermal energy from the eluate stream leaving the top of the elution column and cooling the eluate stream from 140°C to around 95°C prior to entering the electrowinning cells.

17.3.6 Electrowinning

The electrowinning (EW) circuit located in the gold room consists of two dedicated circuits:

- CIL gold EW circuit
- Gravity gold EW circuit

17.3.6.1 CIL electrowinning

The pregnant electrolyte from elution is directed to the electrowinning cell feed tank. This allows de-aeration of the electrolyte to the cell and equally distributes the solution among the cells. Excess electrolyte from the cell feed tank overflows with the return electrolyte from the cells.

Sludging-type mesh cathodes are utilised to electrowin gold from the pregnant electrolyte. An electric current is applied across the cell electrodes and gold is deposited as fine sludge, loosely adhering to the pad of stainless-steel knit mesh contained in the cathode basket. Samples of the electrowinning tails (barren electrolyte) are taken at regular intervals during the electrowinning process. These are analyzed for gold, caustic, and cyanide concentrations in solution. Electrowinning is complete once the gold tenor in the barren electrolyte reduces to the required level. When an electrowinning cycle is complete, the barren electrolyte is sent to the eluant tank.

17.3.6.2 Gravity electrowinning

The gravity EW circuit is similar to the CIL EW circuit. Feed is from the intensive cyanidation circuit. The electrolyte tails from the gravity electrowinning cell are recycled back to the gravity pregnant liquor tank. When an electrowinning cycle is complete, the barren electrolyte is pumped to the CIL feed box.

Cathodes are periodically lifted from the EW cells (gravity and CIL) using a hoist and washed of deposited gold sludge in the cathode wash tank. The cathode wash pump provides high-pressure water spray to remove the sludge adhering to the cathode mesh. The sludge accumulated on the floors of the electrowinning cells is washed into the cell sludge tank with the sludge from the cathode wash tank. The sludge is then pumped to the sludge filter press and dewatered. The filter cake is placed in trays and taken to the drying oven to remove surface moisture. A fume extraction system on the electrowinning cells extracts potentially poisonous and explosive gases that evolve during the electrowinning process. A fresh air fan is installed to force air into the gold room to improve ventilation inside the building.

17.3.7 Carbon regeneration

Barren carbon is transferred hydraulically from the elution column to the kiln feed hopper. Excess water and carbon fines drain through strainers fitted at the bottom and at the overflow of the hopper. Excess water and carbon fines discharge into the carbon transfer water tank.

Carbon is fed from the kiln feed hopper to the regeneration kiln using a variable speed screw feeder. The feed rate is set manually.

The regenerated carbon exiting the kiln is immediately quenched with water in the quench pan to prevent oxidation reactions with atmospheric oxygen. The quenched carbon is pumped to the carbon sizing screen using the carbon transfer pump. The screen oversize (regenerated, sized carbon) will be returned to the CIL circuit.

Fresh carbon make-up can also be added with regenerated carbon to the last CIL tank or to CIL tank 6 if the last CIL tank is offline.

17.3.8 Gold room

The gold room is a secure facility which houses pregnant liquor tanks for the Acacia intensive leach and from the elution circuit, and EW cells for each pregnant liquor feed. Filtered gold sludge from cathode stripping is loaded onto drying trays and the trays are loaded into the cathode sludge drying oven. The dried sludge is cooled and mixed with smelting fluxes at the required ratios. The fluxed, dried gold sludge is loaded into the smelting crucible, which is fitted into the smelting furnace.

The diesel-fired smelting furnace operates at temperatures between 1,200°C and 1,400°C. The furnace is fitted with a temperature control system and has a hydraulic tilting system for use during gold pour. The smelting furnace is covered by a fume hood with a flue duct that is vented outside the gold room.

At the completion of a smelt, the furnace firing system is switched off, and the molten contents of the crucible are poured into doré molds mounted on a cascade trolley. The doré collects in the first mold with any excess collected in the second mold, while slag overflows and collects in a slag collection crucible on the last cascade.

The heavy metallic phase sinks to the bottom of the molds whilst the light slag phase floats on top of the metallic phase. When both phases cool down and solidify, the glassy slag phase is easily broken away from the metallic phase, and the gold bar remains.

The doré bar is further cleaned by chipping off and wire-brushing the slag adhering to the surface of the bar.

The gold room will be equipped with a self-contained ventilation system. A safety shower is provided in the elution area. The safety shower is activated by a foot pedal and is equipped with an eye bath.

17.3.9 Cyanide destruction

The INCO cyanide destruction process utilises SO_2 and air in the presence of a soluble copper catalyst to oxidise cyanide to the less toxic compound cyanate (OCN^-). Sodium metabisulphite (SMBS) is used as the SO_2 source and is dosed into the cyanide destruction feed box as a 20% w/v solution. The detoxification process requires the presence of soluble copper to act as a catalyst and to ensure that any free cyanide present is bound to copper as a weak acid dissociated (WAD) complex. The oxygen required in the reaction will be supplied by sparging blower air into the cyanide detoxification tank. The reaction is carried out at a pH of 8.5 which is maintained by controlled lime addition to the cyanide destruction feed box.

CIL tailings will report to a carbon safety screen before reporting to the two cascading cyanide destruction tanks. Fine carbon from the screen oversize will be collected in a bag for recovery of residual gold by ashing in a carbon fines furnace and smelting of the inorganic calcine, while the screen undersize will flow to the detox tanks. Other streams such as spent acid from the acid rinse step, and sump pump discharges from the acid mixing area and tailings area will also be routed to the detox tanks.

An automatic two-stage slurry sampler will be installed on the CIL tails line to the carbon safety screen and will collect representative sample for plant control and metallurgical accounting purposes.

A standby carbon safety screen will be implemented.

17.3.10 Tailings thickener and disposal

The final tailings slurry in the detox tanks will be transferred to the tailings thickener and compacted to the target 50% (w/w) slurry density prior to pumping to the TSF.

Flocculant will be diluted in a tailings thickener flocculant static mixer and then added to the thickener to aid particle settling. Thickener overflow will gravitate to the process water pond to be reused as plant process water. Thickener underflow will be pumped to the TSF.

17.4 Process description – Phase 2 roast – CIL

Phase 2 development includes the construction of a roasting system and associated infrastructure to treat fresh sulphidic ores. The roasters will process the double refractory ore, i.e., Au in silicate and Au in sulphide phases, and oxidise carbon in the ore feed with low to mediate sulphur / arsenic contents. The roasting process has been demonstrated to release the majority of encapsulated gold for leaching.

17.4.1 Pre-roasting dewatering circuit

The primary cyclone overflow will be processed in a two-stage dewatering circuit. The feed will be deaerated in a high-rate thickener feed box prior to entry into the thickener. Pre-diluted flocculant will be added in the feed launder and feed well.

Thickener overflow will gravitate to the process water pond to be reused as plant process water. Thickener underflow at around 50% solids w/w will be pumped to the pre-roasting filter press feed tank and then to the filters.

The pre-roasting press filter feed tank distributor will distribute the slurry to two buffer tanks, with 2 hours of residence time each. The press filter feed pumps will be cross connected to the two buffer tanks. Four pumps will be dedicated to supply one of four press filters.

The four pre-roasting press filters will dewater slurry to produce filtered cake with a targeted moisture content of 20%.

The slurry withdrawn from the buffer tanks will be pumped under pressure into the recessed plates lined with filter cloths during the filtration cycle. Solids will be retained as cake and the filtrate will be collected in a filtrate tank and recycled to the pre-roasting thickener feed box.

Once the filtration cycle is complete, drying air (from dedicated drying air compressors and drying air receivers for the pre-roasting slurry handling area) will be introduced to displace any remaining moisture content from the filtered cake and cloth, enhancing the dryness. A core blow phase (by injecting compressed air though the central feed ports) will be in operation to expel any residual slurry from the internal cavities, that will minimise contamination between cycles and reduce mechanical wear. A cyclone vent will also be incorporated to vent displaced air and vapor during pressurisation and drying.

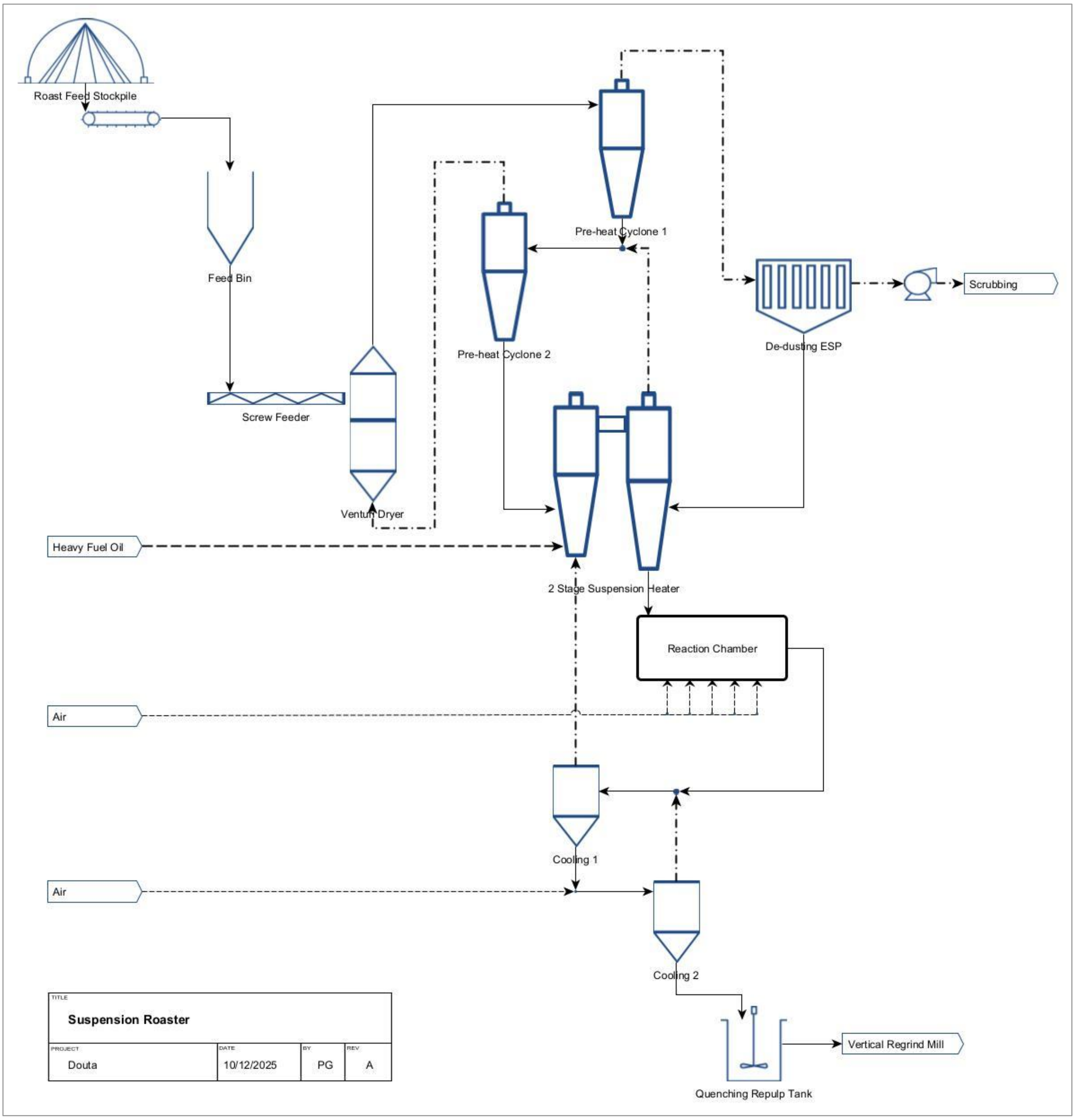
Dried cake from each filter will be discharged by gravity onto dedicated conveyors. The dried cakes from four conveyors will discharge onto a combined pre-roasting conveyor which will transfer the cake to two pre-roasting storage silos. Silos will be in a roofed enclosure. Product will be recovered via one vibrating feeder per silo for transfer to the suspension roaster. Storage capacity of each silo is five hours.

After the filtration and cake drying phases, low pressure process water may be introduced through a dedicated cloth rinsing and core wash tank and pumping systems to rinse the filter cloths, that will remove (rinsing) fine particles and residual slurry, preventing blocking of the pores and central feed ports. A high-pressure cloth wash water tank may also be provided to pump treated water (as wash water) to clean any residual solid particles from the filter cloths after each cycle. The rinsing / washing step(s) will prevent blinding and maintain good permeability of filter cloths for the next filtration cycle.

17.4.2 Suspension roasting

Two suspension roasters of patented design will operate in parallel to achieve 2.4 Mtpa throughput. The basic suspension roaster flowsheet is shown in Figure 17.3. HFO will be the main energy source to operate the roasters. While the process reaction is exothermic, oxidation of the sulphur contained in the ore cannot maintain the minimum temperatures required for stable operation.

Figure 17.3 Suspension roasting flowsheet



Source: Thor; NORINCO.

Roasting feed transfers to a metered feeder installed underneath for metal accounting purpose. Feed is screw fed to a Venturi drier followed by 2-stage cyclone separators used to pre-heat the feed using off gas from the 2-stage suspension heater. The cyclone separator gas exhaust is extracted to the baghouse dust collector for dust collection and then further treated by the gas scrubber. The cyclone separator underflow flows into the 2-stage suspension heaters to heat the feed to the target 650°C in a controlled, increasing gradient. The residence time prior to the heaters is between three and five minutes.

Following heating the feed transfers under gravity to the reaction chamber for approximately 60 minutes where organic carbon is oxidised. Oxidised product is exhausted to a two-stage cooling system to reduce it to the target of 80°C. The final roast calcine is collected into one quenching / repulping tank.

Gaseous nitrogen oxides (NOx) – contaminants from the combustion process are converted to nitrogen and water using selective catalytic reduction (SCR). Vaporised urea reagent is injected directed into flue gas and reacts with NOx resulting in nitrogen (N_2) and water (H_2O) formation. The catalyst is made from titanium, vanadium and tungsten.

Sulphur dioxide (SO_2) is removed from the flue gas by passing it through a chamber that exposes the flue gas to a slurry of finely ground limestone. The slurry absorbs SO_2 from the flue gas and the calcium in the limestone reacts with the SO_2 to form a mixture of calcium sulphite and calcium sulphate. The slurry is dewatered in a 2-stage process involving a hydrocyclone and vacuum filter system to produce a gypsum cake for disposal or sale.

Arsenopyrite will decompose during the heating process. Highly toxic arsenic trioxide (As_2O_3) can be generated should reaction parameters not be controlled to prevent its formation. The level of As_2O_3 is partially controlled by the reaction temperature and oxygen concentration. Heating in the 2-stage heaters is under oxygen deficient conditions to minimise As_2O_3 formation. Over 600°C, the arsenic will form ferric and calcium arsenates ($FeAsO_4$, $Ca_3(AsO_4)_2$) solids and will deport with the calcined product to CIL.

17.4.3 Repulping and regrinding

Roaster calcine drops into one conditioning tank for repulping and quenching. The slurry is transferred to two cascading buffer tanks each with a two-hr buffer time to stabilise the slurry. The buffer contents will be pumped to one regrind mill feed distributor to ensure evenly feeding the tower mills.

The regrind circuit is comprised of one regrind cyclone and three tower mills where the roaster product particle size is reduced to P_{95} of 38 µm. The final regrind cyclone overflow will feed the CIL circuit for leaching while the cyclone underflow will gravitate to the tower mills for regrinding. Grinding media loading unit and rig cranes will be equipped for ease of maintenance.

One on-line particle size analyser will monitor the regrind mills product size distribution.

Media will be charged from bulk bags into a feed hopper located at the topmost level of each mill installation. A rig crane will be provided for lifting the media bags to the hopper. Kibbles (container suitable for hoisting) will be provided to collect the media charge when the mill is dismantled and the rotors removed for maintenance. The kibbles will be used to refill the media charge before restart using the rig hoist.

Either mill can be bypassed and the circuit operated as a single stage regrind if needed though capacity and product particle size will be limited by the reduced available regrind mill power.

17.5 Mass and water balances

The mass and water balance are listed in Table 17.1 and Table 17.2.

Table 17.1 Mass and water balance - Phase 1 – oxide / transitional

Stream	Solid flowrate	Solid concentration	Slurry density	Slurry flowrate	Slurry flowrate	Water flowrate
Unit	**tph**	**%**	**t/m³**	**m³/h**	**tph**	**m³/h**
SAG mill new feed	505.05	95.00	2.45	217.17	531.63	26.58
Process water to SAG mill sprout						141.77
SAG mill discharge	505.05	75.00	1.88	358.94	673.40	168.35
Ball mill feed	1,262.38	68.66	1.00	1,830.83	1,838.69	576.32
Process water to ball mill sprout (spray)						17.74
Ball mill discharge	1,262.38	68.0	1.73	1,070.43	1,856.44	594.06
Mill sump dilution water						465.80
Cyclones feed	1,767.43	59.0	1.58	1,895.17	2,995.64	1,228.21
Cyclones U/F	1,262.63	76.6	1.91	862.92	1,649.09	386.46
Cyclones O/F	505.05	37.5	1.30	1,032.34	1,346.80	841.75
Gravity screen spray water						120.00
Gravity screen feed	252.53	56.1	1.54	292.58	449.82	197.29
Gravity screen pass	202.02	53.5	1.50	251.57	377.36	175.34
Gravity screen reject	50.51	69.7	1.77	41.01	72.46	21.95
Gravity feed - Knelson	202.02	53.5	1.50	251.57	377.36	175.34
Gravity water consumption						70.00
Gravity Concentrate	0.25	63.3	1.65	0.24	0.40	0.15
Gravity tailing	201.77	45.1	1.70	262.92	446.96	245.19
Lime addition	0.35	20	1.07	1.65	1.77	1.41
Cyanide addition	0.30	20	1.08	1.40	1.52	1.21
Water spray to trash screen						50.00
CIL feed	505.05	36.1	1.14	1,032.34	1,399.43	894.38
Tailings thickener feed	505.05	36.1	1.14	1,032.34	1,399.43	894.38
Tailings thickener U/F	505.05	49.8	1.45	698.81	1,013.28	508.23
Tailings thickener O/F						386.15
Final tails						508.23
Water input						894.52
Water output						894.38
Water balance						0.15
TSF Return						254.11
Raw water for the plant						254.26

Source: NORINCO.

Table 17.2 Mass and water balance - Phase 2 – fresh

Stream	Solid flowrate	Solid concentration	Slurry density	Slurry flowrate	Slurry flowrate	Water flowrate
Unit	**tph**	**%**	**t/m³**	**m³/h**	**tph**	**m³/h**
SAG mill new feed	303.03	95.00	2.45	130.30	318.98	15.95
Process water to SAG mill sprout						85.06
SAG mill discharge	303.03	75.00	1.88	215.36	404.04	101.01
Ball mill feed	757.33	0.64	1.00	1,174.27	1,178.99	421.66
Process water to ball mill sprout (spray)						4.33
Ball mill discharge	757.33	64.00	1.66	711.78	1,183.32	426.00
Mill sump dilution water						209.85
Cyclones feed	1,060.36	59.00	1.58	1,136.99	1,797.21	736.86
Cyclones U/F	757.58	76.57	1.91	517.68	989.38	231.81
Cyclones O/F	303.03	37.50	1.30	619.40	808.08	505.05
Gravity screen spray water						120.00
Gravity screen feed	151.52	47.66	1.42	223.54	317.88	166.36
Gravity screen pass	121.21	53.54	1.50	150.94	226.42	105.20
Gravity screen reject	30.30	33.13	1.26	72.59	91.46	61.16
Gravity feed - Knelson	121.21	53.54	1.50	150.94	226.42	105.20
Gravity water consumption						70.00
Gravity Concentrate	0.25	63.27	1.65	0.24	0.40	0.15
Gravity tailing	120.96	40.86	1.70	174.13	296.02	175.06
Dewatering thickener feed	303.03	37.50	1.30	619.40	808.08	505.05
Dewatering thickener O/F						200.11
Dewatering thickener U/F	303.03	49.84	1.45	419.29	607.97	304.94
Dewatering filter press feed	303.03	49.84	1.45	419.29	607.97	304.94
Dewatering filter press filtrate						228.70
Dewatering filter press cake	303.03	79.90	1.99	190.59	379.26	76.23
Suspension roasting feed	303.03	79.90	1.99	190.59	379.26	76.23
Suspension roasting emissions or dust	15.15					76.23
Suspension roasting product	287.88				287.88	0.00
Suspension roasting product reslurry make-up water						479.80
Suspension roasting product	287.88	37.50	1.30	588.43	767.68	479.80
Cyclones feed	1,151.52	46.05	1.40	1,783.40	2,500.38	1,348.87
Regrind Cyclones U/F	863.64	49.84	1.45	1,194.97	1,732.70	869.07
Cyclones O/F	287.88	37.50	1.30	588.43	767.68	479.80
Regrind mill feed	863.64	49.84	1.45	1,194.97	1,732.70	869.07
Regrind mill product	863.64	49.84	1.45	1,194.97	1,732.70	869.07
Lime addition	0.21	20.00	1.07	0.99	1.06	0.85
Cyanide addition	0.18	20.00	1.08	0.84	0.91	0.73
Water spray to trash screen						50.00
CIL feed	287.88	35.14	1.13	619.40	819.25	531.37
CIL discharge	287.88	35.14	1.13	619.40	819.25	531.37
Detox. Feed	287.88	35.14	1.13	619.40	819.25	531.37
Detox. Discharge	287.88	35.14	1.13	619.40	819.25	531.37

Stream	Solid flowrate	Solid concentration	Slurry density	Slurry flowrate	Slurry flowrate	Water flowrate
Unit	**tph**	**%**	**t/m³**	**m³/h**	**tph**	**m³/h**
Tailings thickener feed	287.88	35.14	1.13	619.40	819.25	531.37
Tailings thickener U/F	287.88	49.84	1.45	398.32	577.57	289.69
Tailings thickener O/F						241.68
Final tails						289.69
Water input						1,036.57
Water output						1,036.42
Water balance						0.15
Raw water for the plant						221.22

Source: NORINCO.

17.5.1 Process design criteria

The key process design criteria are listed in Table 17.3.

Table 17.3 Summary of key process design criteria – Phase 1

Process design parameter	Units	Values	Sources
Head grade – design	g/t Au	1.03	1
Design bond ball mill work index (bwi)	kWh/t	14.2	2
Run-of-mine (ROM) material size, F_{80}	mm	415	1
Primary crushed product size, P_{80}	mm	150	4
Grinding circuit product size, P_{80}	mm	74	2
Operating schedule			
Processed ore			
Annual design tonnage	Dry t/year	4,000,000	1
Crushing operation			
Operating hours per day	hrs	24	3
Operating days per year	day	365	3
Crushing plant availability	%	75	5
Operating hours per year	hrs	6,570	3
Feed rate	dry tph	608	3
Milling operation			
Operating hours per day	hrs	24	3
Operating days per year	day	365	3
Milling plant availability	%	90.4	5
Operating hours per year	hrs	7,919	3
Feed rate	dry tph	505	3
Gravity operation			
Operating hours per day	hrs	24	3
Operating days per year	day	365	3
Milling plant availability	%	90.4	5
Operating hours per year	hrs	7,919	3
Feed rate	dry tph	252.53	3
Sag mill			
Effective grinding length	m	4.5	4
Diameter	m	8.53	4
Power	kW	9,000	4

Process design parameter	Units	Values	Sources
Ball mill			
Effective grinding length	m	11	4
Diameter	m	7.32	4
Power	kW	9,000	4
Design gravity gold recovery			
Strongly and medium oxidised	%	50	2
Weakly oxidised	%	50	2
Overall gold recovery			
Strongly and medium oxidised	%	90	2
Weakly oxidised	%	70	2
Ounces recovered			
Strongly and medium oxidised			
Gravity	oz/year	66,238	3
CIL	oz/year	52,990	3
Overall – gravity + CIL	oz/year	119,228	3
Weakly oxidised			
Gravity	oz/year	66,238	3
CIL	oz/year	26,495	3
Overall – gravity + CIL	oz/year	92,733	3
Leaching circuit residence time	hrs	24	2
Leach slurry density	% solids w/w	36.1	3
Number of CIL Tanks	#	10	3
Cil tank volume	m^3	2,755	4
Elution circuit type	-	Pressure Zadra	4
Elution circuit column size	Carbon content (t)	12	5
Frequency of Elution	Strips / week	7	5

Sources column:
1 - ASR (Owner)
2 - Metallurgical test work
3 - Calculated data
4 - Vendor data or recommendation
5 - Industry standard or practice

Table 17.4 Summary of process design criteria – Phase 2

Process design parameter	Units	Values	Sources
Head grade – design	g/t Au	1.03	1
Filtration plant availability	%	80	1
Design bond ball mill work index (BWi)			
Makosa tail	kWh/t	22.7	2
Makosa	kWh/t	18.7	2
A x b			
Makosa Tail	-	41.6	2
Makosa	-	67.5	2
Run-of-mine (ROM) material size, F_{80}	mm	415	1
Primary crushed product size, P_{80}	mm	150	4
Primary grinding circuit product size, P_{80}	µm	74	2
Operating schedule			

Process design parameter	Units	Values	Sources
Processed Ore			
Annual design tonnage	dry t/year	2,400,000.00	1
Crushing operation			
Operating hours per day	hrs	24	3
Operating days per year	Day	365	3
Crushing plant availability	%	75	5
Operating hours per year	hrs	6,570	3
Feed rate	dry tph	365.30	3
Milling operation			
Operating hours per day	hrs	24	3
Operating days per year	Day	365	3
Milling plant availability	%	90.4	5
Operating hours per year	hrs	7,919	3
Feed rate	dry tph	303.07	3
Gravity operation			
Operating hours per day	hrs	24	3
Operating days per year	day	365	3
Milling plant availability	%	90.4	5
Operating hours per year	hrs	7,919	3
Feed rate	dry tph	151.52	3
Roasting operation			
Operating hours per day	hrs	24	3
Operating days per year	day	365	3
Milling plant availability	%	90.4	5
Operating hours per year	hrs	7,919	3
Feed rate	dry tph	303.07	3
SAG mill			
Effective grinding length	m	4.5	4
Diameter	m	8.53	4
Power	kW	9,000	4
Ball mill			
Effective grinding length	m	11	4
Diameter	m	7.32	4
Power	kW	9,000	4
Design gravity gold recovery			
Makosa tail	%	5	2
Makosa	%	5	2
Overall gold recovery			
Makosa tail	%	88.00	2
Makosa	%	82.00	2
Ounces recovered			
Makosa tail			
Gravity	oz/year	3,974	3
CIL	oz/year	65,178	3
Overall – gravity + CIL	oz/year	69,152	3

Process design parameter	Units	Values	Sources
Makosa			
Gravity	oz/year	3,974	3
CIL	oz/year	61,204	3
Overall – gravity + CIL	oz/year	65,178	3
Regrinding circuit product size, P_{95}	µm	38	2
Pre-roasting filtered cake target moisture	%solids w/w	20	2
Roasting time			
Makosa Tail	mins	60	2
Makosa	mins	60	2
leaching circuit residence time	hrs	24	2
leach slurry density	% solids w/w	36.1	3
Number of CIL tanks	#	10	3
CIL tank volume	m^3	2,755	4
Elution circuit type	-	Pressure Zadra	4
Elution circuit column size	Carbon content (t)	12	5
Frequency of elution	Strips / week	7	5

Sources column:
1 - ASR (Owner)
2 - Metallurgical test work
3 - Calculated data
4 - Vendor data or recommendation
5 - NORINCO data base
6 - Industry standard or practice

17.6 Reagents

Reagents and consumables required for the operation include grinding media, cyanide, caustic, lime, flocculant, hydrogen peroxide, diesel (for plant use only), hydrochloric acid (HCl), SMBS, urea, copper sulphate and smelting fluxes. Reagents will be delivered to site in bags, drums (HCl) or intermediate bulk containers and stored until required.

Mixing and storage of solutions will be conducted on site using dedicated mixing, storage and delivery systems.

The reagent consumptions obtained during the test work for oxide and fresh composite were used to estimate the size of the equipment associated with mixing, storage, and distribution. The storage area will allow for three-month stock holding capacity for grinding media, carbon, caustic, flocculant, hydrogen peroxide, hydrochloric acid, and smelting fluxes.

All reagents are transported on pallets with forklifts. All materials are lifted using hoist and lifting frame to dosing positions when being mixed and stored for use.

A summary of the annual reagent consumption is shown in Table 17.5.

Table 17.5 Annual reagents consumption

Reagents	Unit consumption (kg/t milled)	Annual consumption (t/year or m^3/year)
Phase 1		
Hydrated lime	0.3	1,200
Sodium cyanide	1.2	4,800
Hydrochloric acid (32%)	-	400
Caustic	-	200
Borax	-	32
Sodium nitrate	-	16
Soda ash	-	16
Flocculant	0.03	120
Activated carbon	0.08	320
Copper sulphate	0.125	500
SMBS	0.21	840
120 mm forged grinding ball	0.6	2,400
50 mm forged grinding ball	1	4,000
Phase 2		
Hydrated lime	0.3	20
Sodium cyanide	1.2	2,880
Hydrochloric acid (32%)	-	240
Caustic	-	120
Borax	-	19.2
Sodium nitrate	-	9.6
Soda ash	-	9.6
Flocculant	0.06	144
Activated carbon	0.08	192
Copper sulphate	0.125	300
SMBS	0.21	504
Urea	-	175
120 mm forged grinding ball	0.6	1,440
50 mm forged grinding ball	1	2,400
20 mm forged grinding ball	0.3	720

Source: NORINCO.

17.6.1 Lime

Lime is added to the SAG mill feed conveyor for pH control in the CIL circuit. The lime circuit includes a lime storage silo and dosing system with capacity of one shift. An allowance has been made for a stock holding capacity of six months for bagged lime.

17.6.2 Cyanide

Cyanide make-up and dosing facilities are designed to account for the total cyanide usage in CIL, elution and intensive cyanidation and allows for at least two make-ups per day. A minimum stock holding capacity of three months for bagged sodium cyanide is expected.

The cyanide make-up tank is equipped with a cyanide mixer. The made-up cyanide solution is pumped to the cyanide dosing tank using the standby / duty cyanide transfer pump.

Two dosing pumps are used to dose cyanide to intensive cyanidation, the CIL tanks and elution. Any cyanide spillage occurring during the make-up process is immediately hosed down with hosing water and reports to the cyanide spillage sump and is pumped back to the cyanide make-up tank or to the leach feed splitter box in the CIL section.

A safety shower is located close to the cyanide make-up and dosing tanks. It is activated by a foot pedal and is equipped with an eye bath.

17.6.3 Caustic

Caustic usage will be on a batch basis and thus the make-up tank will also be a dosing and storage tank. Storage tank volume will have sufficient capacity for one day for elution and intensive leach. To avoid production of HCN gas, allowance is also made to add caustic directly to the CIL circuit should any problem with the lime dosing system arise.

The caustic dosing pump runs only for the time required to deliver the various batch quantities of the reagent to the various distribution points: CIL, acid wash, elution, and intensive cyanidation.

A safety shower is located close to the caustic make-up tank. It is activated by a foot pedal and is equipped with an eye bath.

17.6.4 SMBS

SMBS usage will be on a batch basis and thus the make-up tank will also be a dosing and storage tank. The SMBS storage volume will have sufficient capacity for one day of operation.

A safety shower is located close to the SMBS make-up tank. It is activated by a foot pedal and is equipped with an eye bath.

17.6.5 Urea

Urea is injected to the off-gas scrubbing system as a vaporised stream to provide reducing agent in the catalytic reduction of nitrogen oxides. A minimum stock holding of three months of bagged urea is allowed for.

17.6.6 Copper sulphate

Copper Sulphate usage will be on a batch basis and thus the make-up tank will also be a dosing and storage tank. The Copper Sulphate storage volume will have sufficient capacity for one day of operation.

A safety shower is located close to the Copper Sulphate make-up tank. It is activated by a foot pedal and is equipped with an eye bath.

The tanks shall be made of stainless steel for corrosion protection.

17.6.7 Hydrochloric acid

HCl is used for acid washing of loaded carbon. The HCl will be delivered in drums. Transfer pumps will transfer the solution to the dedicated acid dosing tank. Dosing will be conducted by a set of dedicated dosing pumps (running and standby).

17.6.8 Hydrogen peroxide

Hydrogen peroxide (H_2O_2) solution will be used for short periods only, as backup oxygenation to the CIL tanks when the oxygen plant is offline. It can also be used as a source of oxygen supply for the intensive leach reactor in the gravity circuit.

Hydrogen peroxide solution will be supplied from a dedicated intermediate bulk container (IBC). Dosing will be done by a set of dedicated dosing pumps directly from the IBC. The hydrogen peroxide is only used as back up in case of problems in the oxygen unit.

Dedicated spillage pumps will handle the reagent and sludge spillages. Spillages will either be disposed of or returned to the treatment plant depending on the type of spill.

17.6.9 Activated carbon

Granulated activated carbon will be delivered in bulk bags and used as an absorbent for the leached (dissolved) gold in the CIL process. Carbon will be regenerated on a cyclic basis. The consumption rate of carbon is associated with losses arising from breakage, spillage or degeneration based on standard industry practice.

17.6.10 Flocculant

The initial flocculant consumption for the thickener is based on similar industry practice. Operational monitoring of consumption will be used to determine long term consumption requirements.

17.6.11 Smelting fluxes

The consumptions of smelting fluxes: borax, sodium carbonate, and silica were estimated as per standard industry practice with similar systems.

17.6.12 Grinding media

Grinding media consists of 120 mm, 50 mm, and 20 mm forged steel balls. The SAG and ball mill grinding media consumptions were based on similar industry practice.

17.7 Services

17.7.1 Oxygen services

A pressure swing adsorption (PSA) oxygen plant will provide oxygen for the CIL plant. A modularised approach to the oxygen plant will allow for additional capacity to be easily included during the operation, if needed.

17.7.2 Water

This plant will be equipped with raw and process water supply systems. The raw water from the mine wide water storage dam will be treated for process use, including reagent mixing, gland sealing, and feed to the potable water treatment plant. The thickener overflow streams and press filter filtrate will be the main feed to the process water system. Raw water will be used as make up water to the process water system.

17.7.2.1 Raw and treated water supply

Plant raw water will be pumped from the mine wide water storage dam to the raw water pond at the plant. The average plant raw water requirement is 126 m^3/h for Phase 1 and 149 m^3/h for Phase 2. An allowance has been made for the raw water from the plant raw water pond to be diverted to the process water pond for top-up and during plant commissioning.

Raw water is used for the gravity concentrator, carbon transfer, water treatment unit, and fire control systems. Each system is supplied by dedicated pumps, depending on the duty.

The fire water system is part of a vendor package.

Raw water will be treated in a sand filtration plant before being supplied to all users requiring clean water with low suspended solids, such as:

- Gland seal water for slurry pumps (cyclone feed, CIL feed, tailings, process water, tailings filter feed, and tailings transfer pumps).
- Treated water for reagents mixing.
- Supply water to the potable water treatment plant.
- Treated water for gravity concentrator fluidisation water back up, SAG and ball mill seals, cloth wash, samplers / analyzers wash, cathode wash, and make up to the strip solution tank and carbon transfer water tank.

The raw water pond will also reserve a portion of the volume for fire water. Fire water will be supplied and distributed to fire hydrants and hose reels within the plant via a ring main distribution system.

17.7.2.2 Process water supply

The process water pond supplies the process water requirements of the plant.

The raw water pond overflow will report to the process water pond. Tailings thickener overflow water streams and potable water treatment plant backwash effluent will also be sent to this pond.

TSF return water is pumped directly to the process water pond. Raw water can also be pumped directly from the raw water supply line to the process water pond if needed for top-up or during plant start-up and commissioning, and if the tailings thickening and filtration return water is unavailable.

The High-Pressure (HP) process water pumps will supply water to screen sprays in the milling and CIL circuits, mainly to carbon sizing, carbon safety, gravity scalping, loaded carbon recovery, ball mill trommel, SAG mill discharge, and trash screens.

The Low-Pressure (LP) process water pumps will deliver dilution and flushing water to different plant areas, mainly to the mill discharge hopper, SAG and ball mill feeding, tailings filter feed pumps, tailings thickener underflow and flocculant static mixers.

Process water is also used as service water for cleaning applications.

17.7.2.3 Potable water supply

Treated raw water will supply a potable water treatment plant to be further treated and sterilised via reverse osmosis (RO) filtration, chlorination, and ultra-violet (UV) sterilisation.

Plant potable water will be supplied to the power station, administrative / plant offices, laboratories, ablutions, and safety showers (including eyewash stations).

17.7.2.4 Ponds and volumes

One mine-wide raw water storage dam is required. Pumps will be installed in the raw water storage dam to enable pumping of water to the process and raw water ponds located in the plant. The sizes of the raw and process water ponds are shown in Table 17.6 and Table 17.7.

Table 17.6 Plant raw water pond

Item	Unit	Value
Type of Reservoir		Raw water
Source of water		Mine wide raw water storage dam
Plant raw water reservoir volume	m^3	3,000

Source: NORINCO.

Table 17.7 Plant process water pond

Item	Unit	Value
Type of reservoir		Process water
Source of water		Return water and raw water top-up
Selected process water storage	m^3	15,000

Source: NORINCO.

17.7.3 Air

This plant will be equipped with a vendor-supplied rotary screw air compressor and dryer system to meet plant air demands.

The HP compressors, one duty and one standby, supply the HP air requirements for mills and instrumentation.

The compressed air will be dried and fed to a plant air receiver from where it will be distributed to the required plant areas, via dedicated air receivers servicing the crushing, milling, reclaim, elution, and CIL areas respectively.

The instrument air is distributed to all the air-operated instruments throughout the plant.

17.7.4 Fuel

A diesel-supply system will also be installed to support fuel requirements. Diesel will be supplied from a dedicated diesel header tank to the strip solution heater, carbon reactivation kiln and smelting furnace systems.

17.8 Energy requirements

The estimated annual power consumption is summarised in Table 17.8 based on processing fresh materials.

Table 17.8 Annual power usage

Area	**Annual power usage**
	MWh/a
Feed preparation	2,468
Milling	154,679
Gravity and CIL leaching	6,881
Roasting (Phase 2)	12,676
Regrind circuit (Phase 2)	8,000
Elution and Goldroom	512
Tailings handling	1,070
Reagents	87
Water services	3,019
Air services	6,378
Fuels	18
Raw water supply	538
Plant services	132
Plant buildings	2,638
Total	**199,096**

Source: NORINCO.

18 Project infrastructure

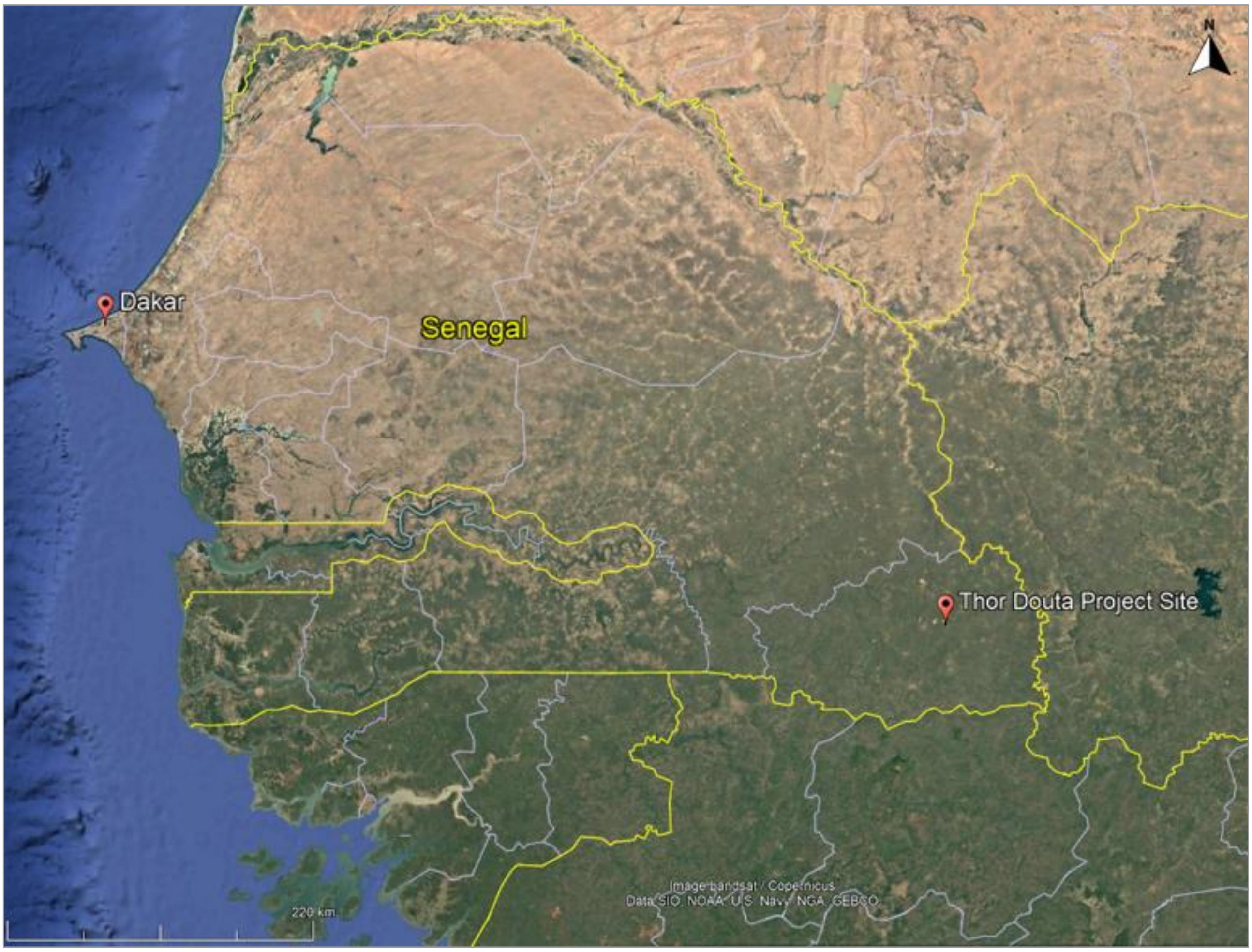
18.1 Supporting infrastructure

18.1.1 Location

The Thor Douta Property is located in Eastern Senegal, approximately 620 km east of Dakar. The Property is located north of the smaller town of Kedougou. This location is in close proximity to other mining activities including Sabadola-Massawa project owned by Endeavour Mining. The area is a more rural and less commercially developed area of Senegal, as opposed to the more developed areas surrounding the major city of Dakar.

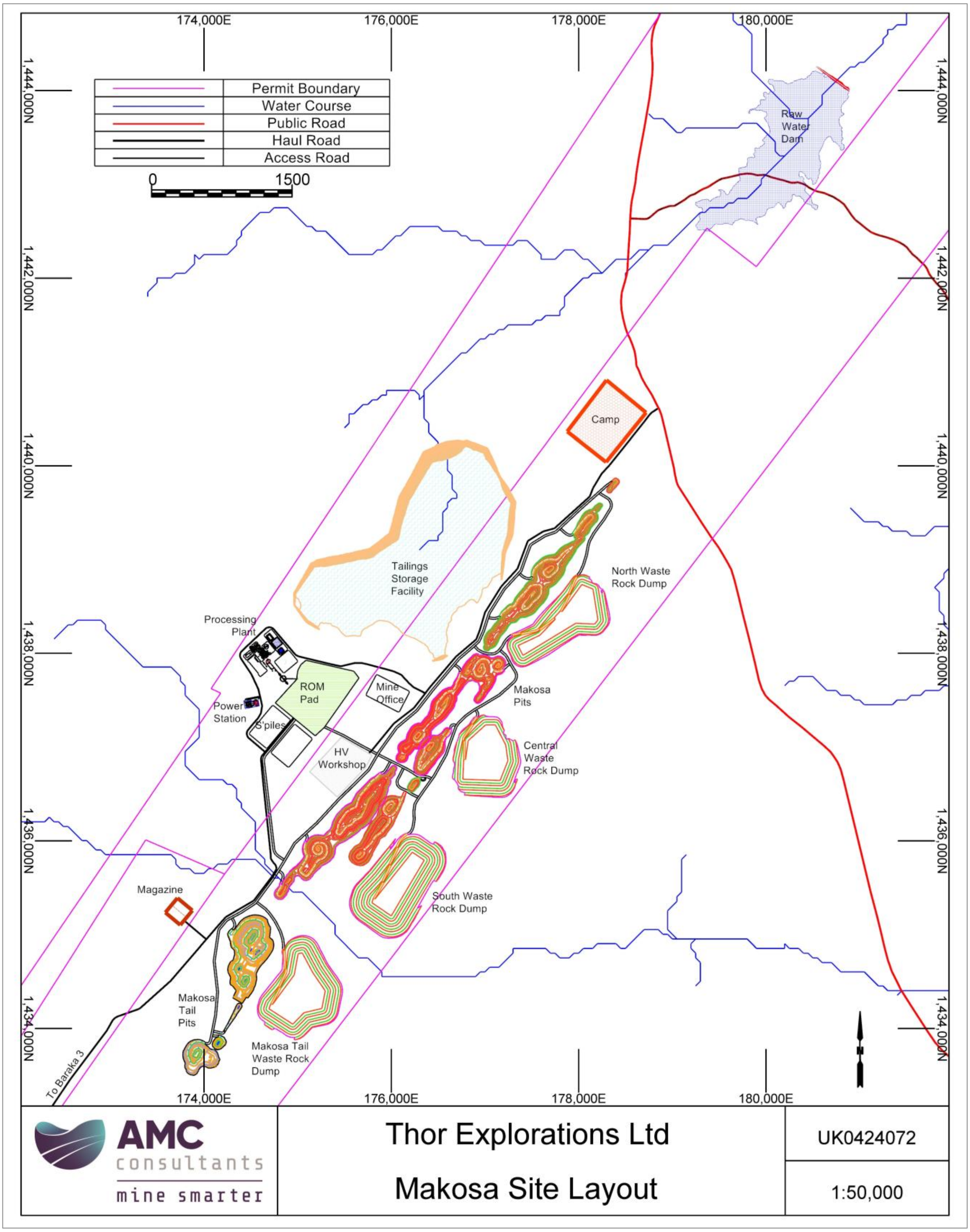
Figure 18.1 shows the location of the project within Senegal, while Figure 18.2 shows the general arrangement for the Makosa site, where most of the Project infrastructure is located. Figure 18.3 shows the general arrangement of the Baraka 3 site, located 35 km to the south of Makosa, and consisting solely of mining operations.

Figure 18.1 Thor Douta site location



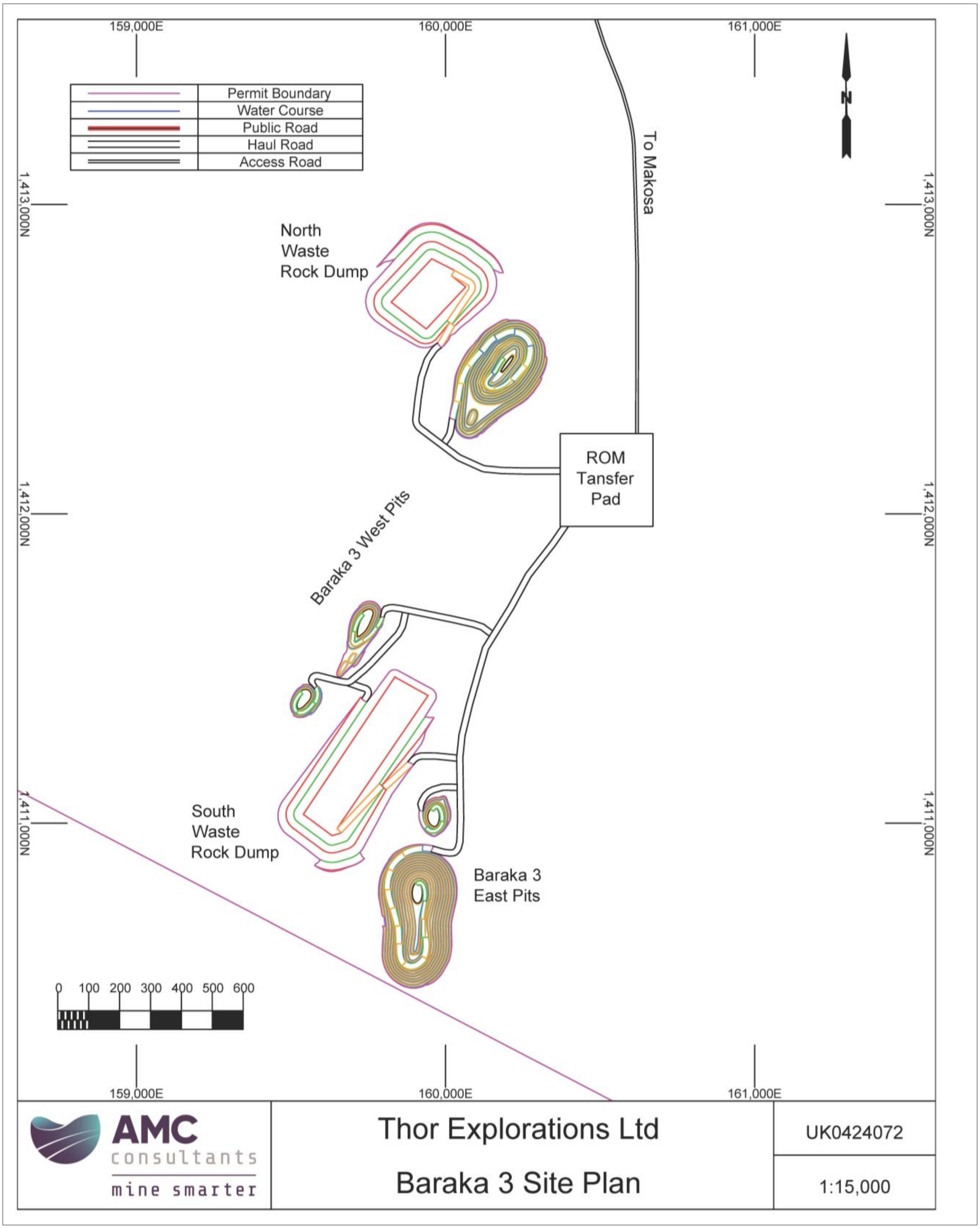
Source: Google Earth Pro. (2025). Map showing city of Dakar the location of the Thor Douta Mining site, Senegal. Google Earth. Available at: https://earth.google.com. Accessed 1 December 2025.

Figure 18.2 General arrangement for the Makosa site



Source: AMC, January 2026.

Figure 18.3 General arrangement for the Baraka 3 site



Source: AMC, January 2026.

18.1.2 Roads

Senegal's highway network includes a primary national road network with primary roads N1 through to N7. These roads are in a fairly good condition and offer a reliable means of transporting goods to and from the mine site through connections to major cities like Dakar. There are also ongoing expansions to increase the network length and major projects planned to connect to other major cities.

The primary highway route that connects to the mine site leads from Dakar to site through various towns and mostly consists of travel on major highways.

The approximate total distance of this trip is about 720 km from the city of Dakar, including the final stretch of dirt road. Shown below is the road to site, where the N1 leads from Dakar to the town of Tembacounda, from where the N7 then joins the towns of Tembacounda and Kedougou. From Kedougou a secondary road which is also tarred, links Kedougou to the small town of Bembou, where the final stretch is a 20 km dirt road to site.

Figure 18.4 Major roads leading to site

Source: Google Earth Pro. (2025). Map showing city of Dakar and major roadways to smaller town and the location of the Thor Douta Mining site, Senegal. Google Earth. Available at: https://earth.google.com. Accessed 1 December 2025.

18.1.3 Railway

Senegal has limited rail network coverage, especially in the more rural regions where the mine site is located. There is central railway network that runs across Senegal, this line is referred to as the Dakar-Bamako metre-gauge line and ultimately connects the city of Dakar with the landlocked city of Bamako in Mali. The railway does face challenges with regards to operability and maintenance, and ongoing rehabilitation aims to restore the line to full operational capacity.

The closest that this railway gets to the mining site is in the town of Tembacounda. Thus, if the railway is envisaged to be utilised for goods transportation, then trans-shipment will have to be down at the town of Tembacounda, whereafter the N7, and rest of the road network will have to be utilised further to get to site.

Due to ongoing rehabilitation work happening on the railway and uncertainty around the infrastructure in Tembacounda to facilitate freight handling to and from the railway, it is unsure if at this stage the project can depend on any rail bound logistics and should rather focus on the road network as primary means of people and goods transportation. Although future opportunities could be investigated to see if utilising some of the railway network could be advantageous to the project.

Figure 18.5 Senegal main rail network line

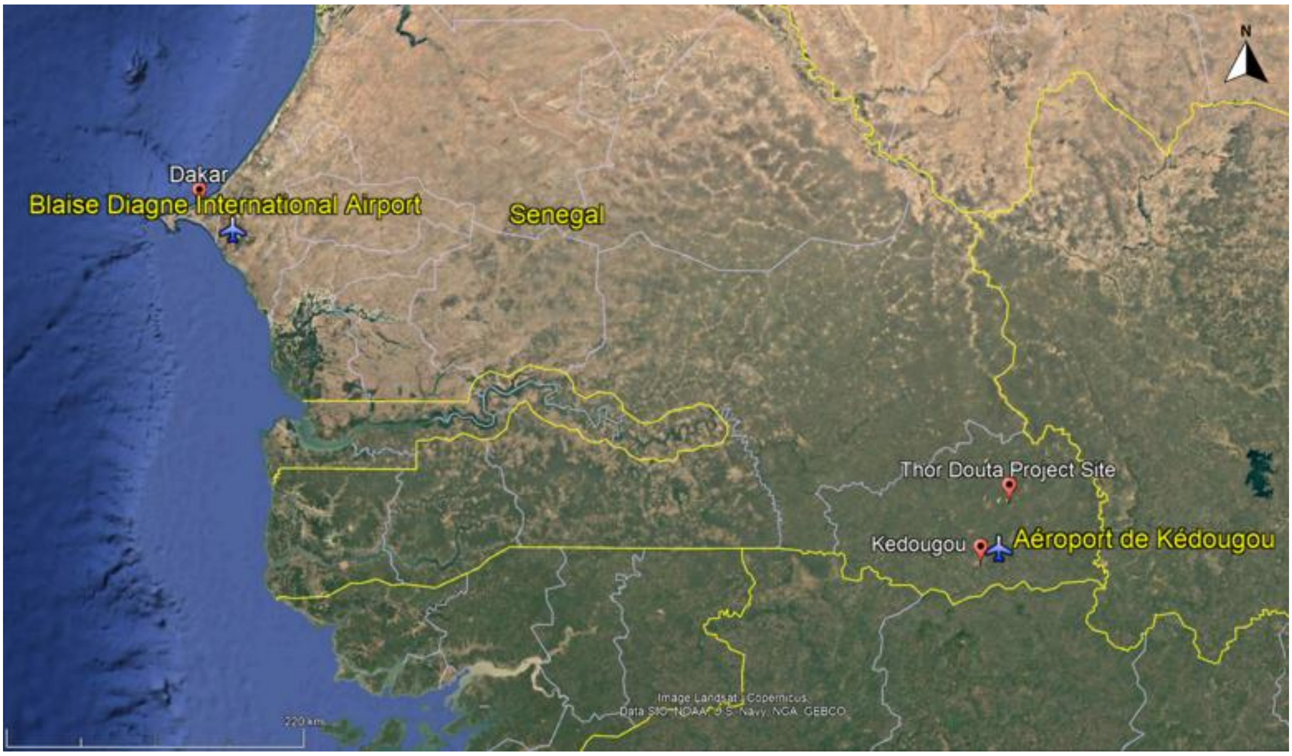
Source: Google Earth Pro. (2025). Map showing city of Dakar and major railways. Also showing the N7 major road to smaller towns and the location of the Thor Douta Mining site, Senegal. Google Earth. Available at: https://earth.google.com. Accessed 1 December 2025.

18.1.4 Airports

The main airport serving Senegal for international people and freight transportation is located just outside the city of Dakar, Blaise Diagne International Airport. This would be the primary means of air freight or passenger transportation in and out of Senegal to support the mine.

Located near the town of Kedougou is a smaller domestic airport called Aéroport de Kédougou, which can offer domestic air travel for passengers as a faster travel alternative to site. This airport is, however, not suited for air freight transport.

Figure 18.6 Senegal airports to support mine



Source: Google Earth Pro. (2025). Map showing city of Dakar and airports in Senegal in relation to the Thor Douta Mining site, Senegal. Google Earth. Available at: https://earth.google.com. Accessed 1 December 2025.

18.1.5 Port facilities

The most suitable port to be used for sea freight is the Port of Dakar. Senegal does have a few minor ports, but for logistical needs to support a mining operation and project of this scale, the Port of Dakar is best suited.

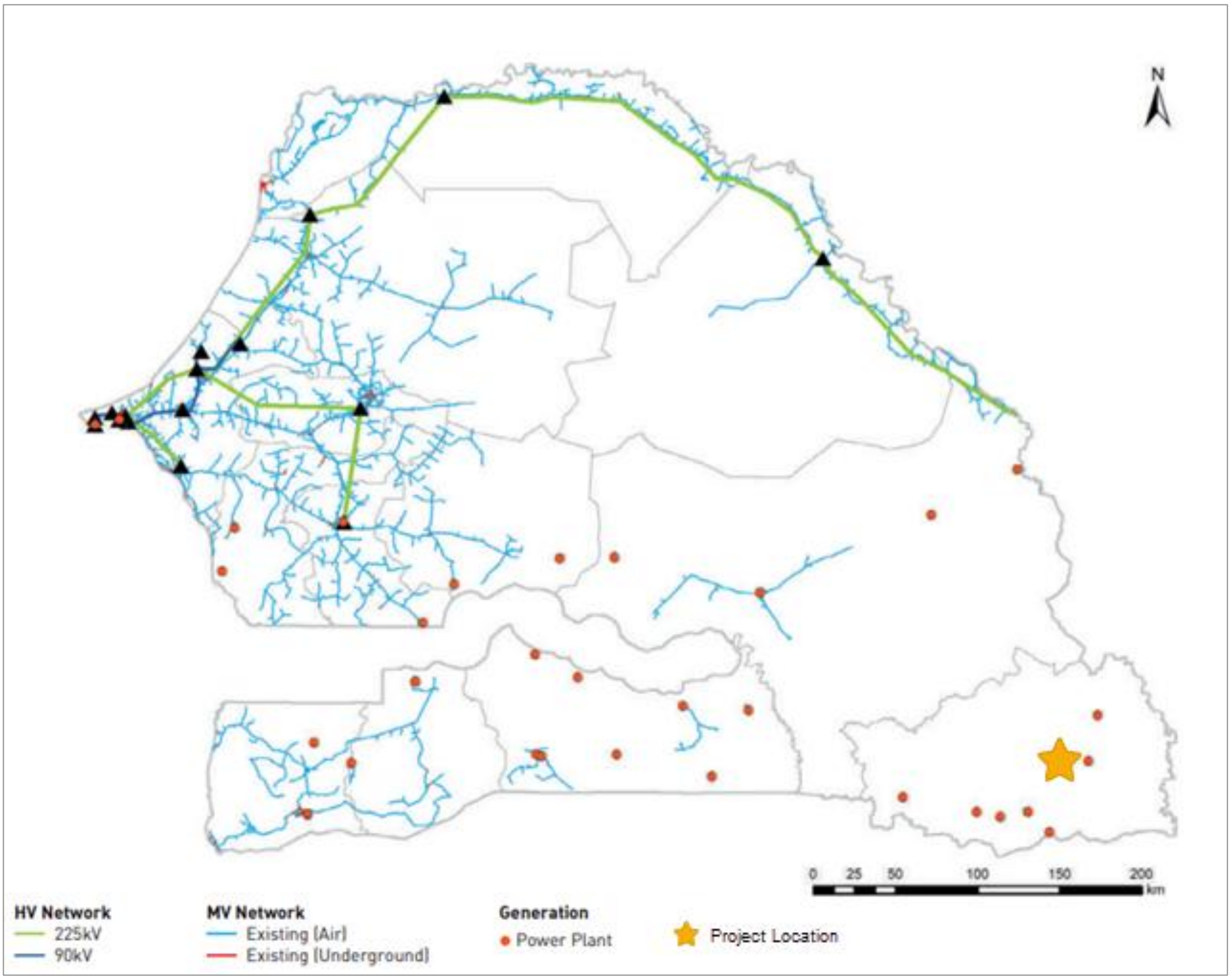
18.2 Power supply and local grid

The national power producer in Senegal is Senelec, who is responsible for production, transmission, and distribution of electricity. The generation capacity is mostly through thermal power stations based on HFO and supplied via independent power producers. There is also a mix of hydroelectric and solar power stations that supplement supply.

Senegal's national grid is isolated to the northern and western parts of the country, servicing the larger cities and towns and commercial hubs. Where Senelec operates a 225 kV and 90 kV network. The more rural areas towards the south, where the mine site is located, are not covered by a grid, and power generation for towns is done locally. Thus, the mine does not have access to a national grid to tie into for power.

Senelec do have future projects in the pipeline to expand the 225 kV to more rural areas in the south and establish grid infrastructure to more rural areas, thus there is future potential for grid power tie-in, however there is no guarantee on when and if enough capacity would be available. Thus, for the mine site it is recommended to rely on own generation through a local power plant.

Figure 18.7 Senegal national grid and power plant locations



Source: ScienceDirect, Least-cost electrification pathways for Senegal by 2030, Available at: https://www.sciencedirect.com, Accessed 1 December 2025.

18.2.1 HFO power plant

Due to the lack of grid power in the region of the mine, allowance has been made for the project to establish a 16 x 2 MW HFO / diesel power plant locally at the mine. Thus, the site will have access to a total installed generator capacity of 32 MW (16 off 2 MW HFO generators).

The generation sets will work in parallel and produce power at 10.5 kV to feed to mining operations. Allowance has also been made for water tanks to provide cooling water to the power plant.

A tank farm to support the power plant has been allowed to provide a 20-day operational capacity buffer for fuel storage. The tank farm includes:

- 2 x 1,300 m^3 HFO storage tank, each with a dimension of 13 m in diameter and 10.5 m in height and overall storage capacity of 1,400 t.
- 2 x 250 m^3 diesel storage tank, each with a dimension of 7 m in diameter and 7 m in height and overall storage capacity of 240 t.
- 1 x 500 m^3 HFO buffer tank, each with a dimension of 9 m in diameter and 8.7 m in height and overall storage capacity of 550 t.
- 1 x 200 m^3 treated HFO tank, each with a dimension of 6.8 m in diameter and 6 m in height and over all storage capacity of 220 t.

- 4 x 50 m^3 HFO day tank, each with a dimension of 3.8 m in diameter and 4.5 m in height and overall storage capacity of 55 t.
- 2 x 20 m^3 diesel day tank, each with a dimension of 2.4 m in diameter and 5 m in height and overall storage capacity of 55 t.

The engines will use a combination of diesel and heavy fuel oil, and annual power generation is estimated at around 190-200 GWh, depending heavily on the redundancy allowance for maintenance that is followed. If a redundancy allowance for maintenance can be relaxed to a N+2 philosophy, then there exists an opportunity to achieve closer to around 245 GWh of produced power annually. Further optimisation of the power plant and required power to support all facilities on site is advised.

It is currently assumed that ex-pit dewatering boreholes and camp power could be handled by utilising more generator sets in the current power plant design. However, an alternative that should also be considered is to power ex-pit dewatering boreholes via more localised generators due to lengthy cable reticulation that would be required if powered from the process plant MCC’s located far away from some of the pit boundaries.

18.3 Tailings Storage Facility

The analysis and design of the TSF was undertaken by Knight Piésold as part of this PFS. Below is a summary of the findings in relation to the TSF at Douta.

Tailings storage will occur through the pumping of tailings from the process plant to a TSF indicated in Figure 18.2. The proposed TSF footprint is about 270 Ha, and the site is situated within a gently sloping valley that drains toward the northeast. Surface conditions comprise moderate to sparse vegetation, overlaying a 1-5 m laterite cuirasse layer and ~20 m thick Saprolite layer.

The objective of the TSF is to provide storage for the tailings received from the process plant during the 14-year LOM. The design of the TSF meets the international stability standards and will minimise seepage.

The processing plant will produce tailings in two stages: Phase 1 (Oxide) – CIL during Years 1-5 and Phase 2 (Fresh) – CIL / ROAST tailings during Years 5-14. Both streams will be converted to a slurry and deposited via spigots into the TSF impoundment. Phase 1 assumes ore production of 4 Mtpa while Phase 2 assumes 2.4 Mtpa. Tailings storage requirements are estimated at 12.7 Mm^3 and 15.2 Mm^3 for Phases 1 and 2 respectively.

The facility will be a conventional TSF which involves the piping and deposition of tailings slurry to a lined area. The footprint of the TSF will be lined in two stages. The first stage will provide storage capacity for Years 1 and 2, while the second stage will complete the lining of the entire TSF footprint.

The TSF foundation will be cleared of all vegetation, and the topsoil will be removed to an average depth of 150 mm. The topsoil will then be stored within 2 km of the TSF. Where needed, an allowance has been made for the excavation of hard and soft laterite to provide a suitable surface for liner installation. The foundation sub-soils will be reworked at the base of the facility to form a suitable subgrade for liner installation.

A needle-punched unwoven protection geotextile, selected based on the protection required for the HDPE liner, will be placed on the prepared impoundment subgrade to prevent puncture from the reworked in situ soils. A 1.5 mm thick HDPE geomembrane liner shall be installed upon the protection geotextile.

The TSF construction has been divided into four stages to reduce upfront CAPEX costs by extending liner installation and embankment construction into the operational period. The Starter Dam will provide capacity for Year 1 (~3.5 Mt). Raise 1 of the TSF will involve an embankment raise of 7 m and increase the capacity to ~18 Mt. This Raise will provide capacity for the remainder of the Phase 1 LOM schedule (Years 2-5). Construction of TSF Raise 2 will increase the embankment crest elevation by 3 m, increasing the capacity to ~26 Mt. This will be sufficient to contain the Phase 2 tailings produced during Years 5-8. Raise 3 construction will raise the embankment crest a further 4 m, increasing the TSF capacity to ~38 Mt, enough to contain the tailings produced for the remainder of the mine life.

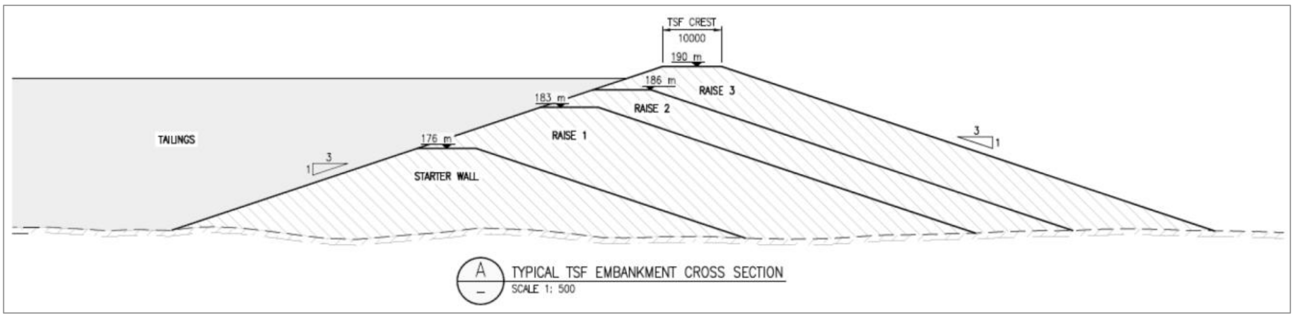
The TSF design geometries and embankment cross section are presented in Table 18.1 and Figure 18.8.

Table 18.1 TSF design geometry and material quantities

Parameter	Value	
Crest Width	10 m	
Starter Dam Crest Elevation	176 mASL	
Raise 1 Crest Elevation	183 mASL	
Raise 2 Crest Elevation	186 mASL	
Raise 3 Crest Elevation	190 mASL	
Upstream Slope	1V:3H	
Downstream Slope	1V:3H	
HDPE Geomembrane	Starter Dam	646,639 m^2
	Raise 1	1,081,922 m^2
	Raise 2	344,456 m^2
	Raise 3	627,787 m^2
Protection Geotextile	Starter Dam	646,639 m^2
	Raise 1	1,081,922 m^2
	Raise 2	344,456 m^2
	Raise 3	627,787 m^2
Embankment Fill	Starter Dam	318,405 m^3
	Raise 1	843,826 m^3
	Raise 2	634,693 m^3
	Raise 3	1,232,643 m^3

Source: Knight Piésold, 2026.

Figure 18.8 TSF embankment final cross section



Source: Knight Piésold, 2026.

A monthly excel water balance over the LOM was developed for the Project. Two different scenarios were considered: A base case representing design conditions at the TSF and defined based on the Project's design criteria and a Sensitivity Analysis on the main loss parameters to account for the uncertainty and impact of these parameters on the water balance results.

The main results from the analysis can be summarised as follows:

- The peak TSF inflows estimated for the Base Case scenario increase during LOM, from 670,000 m^3/month at the beginning of the operation to 850,000 m^3/month towards the end of the operation as the TSF footprint increase and the contribution of beach runoff and direct pond precipitation to the inflows increases.
- Entrainment and evaporation are the main losses in the TSF as the facility will be lined and the water from the tailing's seepage will be recirculated to the TSF pond from the seepage collection pond.
- The TSF pond volume increases during the operation as the TSF footprint increases. The average TSF pond volume varies between 200,000 m^3 (dry season) and 1.4 Mm^3 (wet season) during LOM for the Base Case scenario.
- The peak average pond volumes for the sensitivity analysis scenarios range between 1.36 Mm^3 (40% Entrained Water) and 1.98 Mm^3 (30% Entrained Water) and are reached at the end of the wet season (October). The TSF pond is reduced to the minimum operational volume during the dry season. The difference between sensitivity analysis scenarios increases towards the end of the LOM as the TSF footprint increases.
- The results are sensitive to loss parameters, especially to the entrainment rate at the TSF.
- Given the uncertainty on the input parameters, it is recommended that further testing is carried out on the tailing's material and site-specific climate monitoring (precipitation, evaporation, and temperature) be implemented to inform and update the water balance during the following design phases.

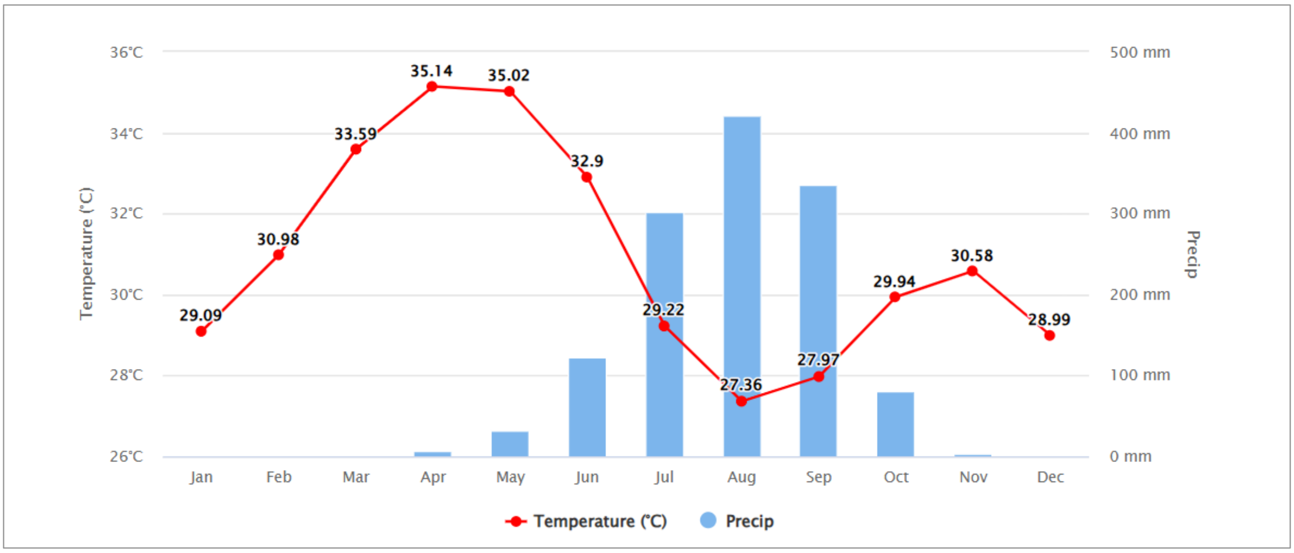
18.4 Water supply

The Process Plant will require a minimum raw water volume of 254.26 m^3/h for Phase 1 and 221.22 m^3/h for Phase 2.

There are no municipal water supplies that can provide the required water for mining operations. Thus, to supply water to mining operations a Water Storage Dam (WSD) is planned to be constructed to dam up an existing stream within the mining licence boundary and take benefit of the higher rainfall period from June to September for rainwater catchment.

Rainfall totals about 1,300 mm of rain throughout the year, averaging 108 mm per month, however six months of the year see less than 1 mm of rain per month (Figure 18.9). Thus, sufficient buffer storage in the dam is required to bridge the dryer months of little to no rainfall.

Figure 18.9 Senegal, Kedougou temperature and annual average rainfall



Source: Weather and Climate, Senegal Climate showing annual average temperature and rainfall figures, Kedougou Available at: https://weatherandclimate.com/, Accessed 1 December 2025.

Due to no municipal available sources of water, the process dam will be the main supply for process water, potable water and any mining related service water requirements. To provide adequate water, a process water pump pontoon has been allowed for along with piped reticulation to the process plant and camp sites to provide water from the dam.

Another potential source of water is through pit dewatering activities, where pit dewatering will be directed to the water dam for storage to be used.

An analysis and design of the WSD was undertaken by Knight Piésold as part of this PFS. Below is a summary of the findings in relation to the WSD at Douta.

The WSD design footprint is approximately 1,220,000 m^2 and is situated along a gently sloping valley with a ~3 m deep eroded channel along its base which drains North-East towards the town of Sanbarabougou, located ~15 km North-East (Figure 18.2). Surface conditions comprise moderate to sparce vegetation, overlaying a 1-5 m Laterite cuirasse layer and ~20 m thick Saprolite layer.

The WSD has been designed in accordance with the CDA Guideline FoS Values (CDA, 2019). The WSD will be constructed prior to the commissioning of the mine.

The WSD foundation will be cleared of all vegetation and the topsoil will be removed to an average depth of 150 mm. The topsoil will then be stored either in the same location used for the TSF topsoil storage or nearby.

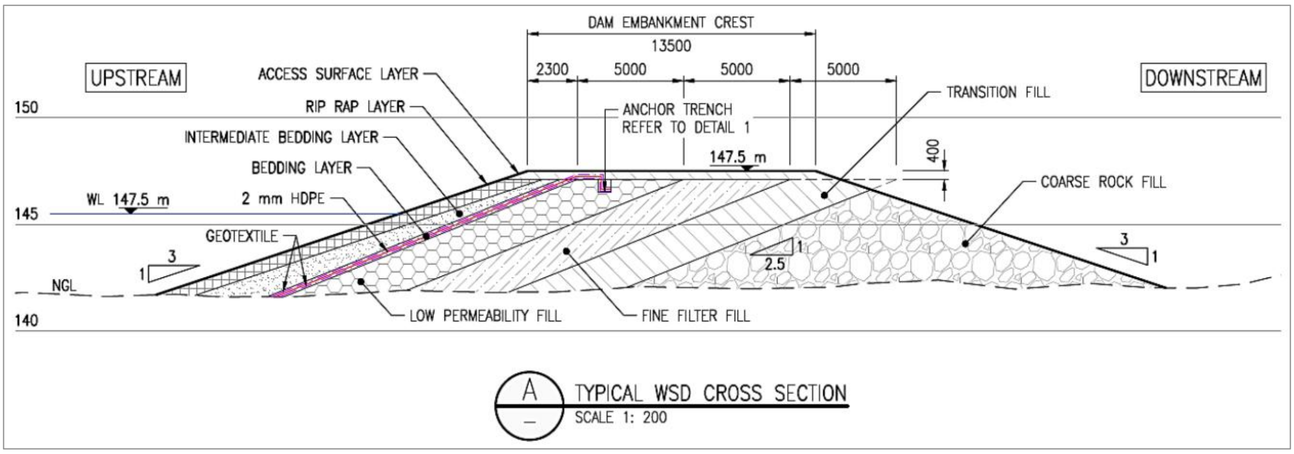
The WSD will be constructed using several different materials including rockfill, transition fill, fine filter, fine fill, bedding material, riprap, selected gravel and HDPE geomembrane. The total embankment fill will be ~30,000 m^3. Section dimensions of the WSD embankment are presented in Table 18.2 and Figure 18.10.

Table 18.2 WSD embankment design parameters and material quantities

Embankment	Unit	Parameter
Construction Methodology		Downstream
Crest Elevation	mASL	147.5
Maximum Height	m	~5.5
Crest Width	m	13.5
Upstream Slope	V:H	1:3.0
Downstream Slope	V:H	1:3.0
Rockfill	m^3	5,800
Transition Fill	m^3	5,900
Fine Filter	m^3	5,900
Fine Fill	m^3	5,900
Bedding Layer	m^3	950
Inter-Bedding Layer	m^3	3,150
Riprap	m^3	2,400
Liner Area	m^2	3,500

Source: Knight Piésold, 2025.

Figure 18.10 WSD embankment typical section



Source: Knight Piésold, 2026.

After completing a water balance study for the WSD, Knight Piésold noted the following:

- For the Base Case scenario, representing design conditions at the WSD and defined based on the Projects design criteria, the monthly average WSD inflows range between 0 m^3/month during the dry season and 4.0 Mm^3/month during the wet season.
- The WSD pond varies between 1.8 Mm^3 (dry season) and 4.2 Mm^3 (wet season) over the LOM for the Base Case scenario.
- The Minimum WSD pond volumes for the sensitivity analysis scenarios range between 1.1 Mm^3 and 2,3 Mm^3 and are reached at the end of the dry season. The results are especially sensitive to the hydraulic conductivity of the natural ground, with the hydraulic conductivity scenarios showing the widest range of pond volume results.
- The minimum raw water requirement at the Process Plant is met for the Base Case and all the evaluated sensitivity analysis scenarios.
- Based on the available information at the time of this study the WSD has enough capacity to provide raw water to the Process Plant through the LOM.

- Given the uncertainty on the input parameters and the sensitivity of the results to the hydraulic conductivity, it is recommended that a ground investigation be carried out in the WSD area as well as site specific climate monitoring (precipitation, evaporation and temperature) to inform and update the water balance during the following design phases.

Potable water shall be provided to site via a potable water treatment plant that can treat the sourced water from the process water dam to produce an adequate potable water source for mining operations. The mining camp is estimated to house around 1,100 people based on the camp layout, and assuming an average potable water allowance of 275 L per person per day, the potable water requirement is estimated to be around 302.5 m^3 per day. Allowing for drinking water, ablution, food preparation, laundry, and other miscellaneous uses.

18.5 Site supporting infrastructure

Site supporting infrastructure will include facilities that support mining and processing operations and allowances are made in the design and costing for various supporting infrastructure.

18.5.1 Access roads

Roads have been allowed for in the design to connect the various infrastructure and mining locations and include roads linking to following critical areas.

Roads within main mining footprint and Makosa pits include the following:

- Entrance roads
- Roads to stores depot
- Roads to pits and dumps
- Roads to TSF
- Roads to WSD

The combined length of these roads is approximately 11-12 km according to the preliminary site layout. An allowance for these roads will be made for a dual lane gravel road with multi layered engineered fill and compaction. The roads should allow for a stable base course and wearing surface of crushed stone. Layers should be properly compacted and graded. The final road design will be subject to detailed civil design, although an allowance shall be made for the study for an adequate quality gravel road.

The Baraka pits are located approximately 35 km away from the primary site, and an allowance shall be made for a similar specification gravel road from the main site to access these pits.

Other:

- Gates – Allowance should be made for gates leading into the mining site.
- Car parking to be allowed for as follows:
  - Process plant area (3 off 30 m x 60 m parking lots).
  - Camp Area 1 (1 off 100 m x 45 m).
  - Camp Area 2 (1 off 55 m x 40 m).

18.5.2 Office and administrative buildings

Various administrative and office buildings will be required to support the mining operations including:

- Reception and Administration
- Stores offices and stores shed

- Ablution Facilities - showers / change facilities
- Training Facility / Muster and Meeting Rooms
- Canteen
- Mine Operations facilities / supervisor's office / dispatch
- Core Shed and Exploration facilities

18.5.3 Workshops and other facilities

The following workshops and other facilities are required to support mining operations and fleet have been allowed for in the design:

- Mechanical (Repairs and Maintenance)
- Electrical
- Vehicle Wash pad
- Drilling Contractors workshops and office
- Wastewater and Toilets at each
- Site wide wastewater management (run off water and sewage)

18.5.4 Camp / accommodation village

An accommodation village has been included in the capital estimate. The amount and style of accommodation have been based on the number of employees and visitors anticipated on site at peak manning times. The following buildings and facilities have been allowed for in the estimate:

- Camp gatehouse and security
- Kitchen and mess hall
- Office and storage
- Laundry
- Recreation facilities
- Mosque
- Multi-denominational prayer room
- Accommodation units to suit both senior and junior employees

18.5.5 Explosive magazines

An explosive magazine area has been allowed for in the layout, however due to safety requirements, it is located away from the main processing and mining activities. The planned location for the magazine is just west of the process water dam, within the mining lease area.

Hazardous explosive materials will be transported and stored within the magazine area and will have to adhere to Senegalese explosive transport and storage regulations.

18.6 Ancillary mobile fleet requirements

Ancillary mobile fleet will be required to provide supporting services to the operation and maintenance of the camp and process plant areas and it is estimated to include:

- Process plant (ROM Loader, light vehicles (LVs), Bobcat, forklift)
- Technical Services (LVs)
- Administration (LVs)
- Stores (forklift, crane, LVs)
- Fire truck

- Ambulance
- Rescue Vehicle
- Mining (LVs, grader, bulldozer, loader)

Table 18.3 Ancillary mobile fleet numbers and specifications

Description	Specification	QTY
ROM Loader	CAT Wheel Loader (FEL), CAT 992	2
LVs	Toyota Hilux	20
Bobcat	Bobcat S650 Skid Steer Loader	5
Forklift	Bobcat B50X-7 Electrical Forklift, 5ton capacity	5
Mobile Crane	Mobile cranes allowed for	2
Fire Truck	N/A -Allowance Made	1
Ambulance	N/A -Allowance Made	2
Rescue Vehicle	N/A -Allowance Made	1
Grader	Allowance for CAT 18 Motor Grader	1
Bulldozer	Allowance for CAT D9 large Dozer	1

18.7 Communication

18.7.1 In-pit communication

For in-pit communication it is advised to allow for a wireless communications network that allows for communication with fleet and other equipment in operation in the pits. The system depends on line of sight between permanent and mobile repeater towers that feed back to a main tower located at the plant or an office that houses the main server infrastructure. The combination of these towers ensures that equipment in pit is always in range to ensure connectivity at all times. For the purposes of this study a view has been taken on a potential network layout that could offer this level of connectivity with permanent repeater towers on pit edges and a fleet of mobile towers in pit. Allowances have been made for communication infrastructure.

18.7.2 Telecommunications

For telecommunication connectivity it is advised for a cost-effective solution to allow for a grid of Starlink satellite terminals to create a high-speed link at the mining site for internet connectivity for email and VoIP communications. This solution is both cost effective and reliable and has been utilised by smaller mining operations around the globe.

It is reported by multiple new sources that under a government initiative that Starlink is to be launched in Senegal in 2025 / 2026, and the Starlink Official website coverage map reports coverage coming online in 2026. Thus, this option should prove viable as a preferred fit for purpose and lower cost telecommunications solution.

18.8 Pit dewatering

The pits are sub-divided into the following areas for the dewatering assessment.

Group 1, pit dewatering areas located near the process water dam:

- Makosa Main South
- Makosa Main North
- Makosa Tail 1
- Makosa Tail 2

Group 2, pit dewatering areas located approximately 35 km from the main mining area:

- Baraka West 1 – North
- Baraka West 2 – Central
- Baraka West 3 – South
- Baraka East 1
- Baraka East 2

Pit dewatering is required to manage rainwater inflows during the wet season, and groundwater ingress into pits year-round:

- Groundwater control is proposed via a combination of ex-pit dewatering boreholes and pumping of residual inflows from in pit sumps.
- Rainwater is collected in sumps.
- Water pumped from pit-sumps is considered contact water and typically requires discharge via settling ponds for environmental compliance. Water pumped from boreholes is considered non-contact water and can be discharged directly to surface environment or to the process water dam.

Water from Group 1 dewatering boreholes will be suitable for use in the process circuit and should be directed to the process water dam. Water levels in the process water dam will need control either via a T-piece and valve on the dewatering pipeline or overflow from the process water dam. Unused water will be discharged to the WSD and used for dust suppression on site.

18.8.1 Pit floor pumping

Skid or trailer mounted diesel driven pumps are recommended in pit to remove rainfall run-off and residual groundwater reporting to sumps. Due to criticality of dewatering activities and their impact on safety and production, it is advised to have backup pumps that can be mobilised to replace any failed pumps. Therefore, the approach will be to try and standardise pump selections for commonality of equipment.

Based on analysis of the historical rainfall record, the following daily pumping requirements are estimated for each of the pits.

Table 18.4 Pit dewatering daily and hourly duties

Pit name	Daily estimated pumping requirement (m^3/day)	Estimated utilisation %	Pump sizing, hourly flow rate (m^3/h)
Makosa Main South	8,733	90	404
Makosa Main North	17,625	90	816
Makosa Tail 1	4,356	90	202
Makosa Tail 2	1,519	90	70
Baraka West 1 – North	1,025	90	47
Baraka West 2 – Central	241	90	11
Baraka West 3 – South	98	90	5
Baraka East 1	1,093	90	51
Baraka East 2	139	90	6

Source: AMC, 2025.

Pump sizing will allow for a buffer capacity of 20% to ensure that surges can be accommodated.

Table 18.5 Estimate of pump power requirement

Pump location	Pump type	Duty	Standby	Flow pump (L/s)	Total head (m)	Motor size (kW)	Voltage (V@Hz)	Total power (kW)
Makosa Main South	Diesel Pump -Centrifugal	2	1	60	77	90	525 V @ 50 Hz	180
Makosa Main North	Diesel Pump -Centrifugal	4	0	60	68	75	525 V @ 50 Hz	300
Makosa Tail 1	Diesel Pump -Centrifugal	1	0	60	87	90	525 V @ 50 Hz	90
Makosa Tail 2	Diesel Pump -Centrifugal	1	1	20	78	30	525 V @ 50 Hz	30
Baraka West 1 – North	Diesel Pump -Centrifugal	1	0	20	94	37	525 V @ 50 Hz	37
Baraka West 2 – Central	Diesel Pump -Centrifugal	1	1	5	96	11	525 V @ 50 Hz	11
Baraka West 3 – South	Diesel Pump -Centrifugal	1	0	5	131	15	525 V @ 50 Hz	15
Baraka East 1	Diesel Pump -Centrifugal	1	0	20	99	37	525 V @ 50 Hz	37
Baraka East 2	Diesel Pump -Centrifugal	1	0	5	102	11	525 V @ 50 Hz	11
Totals		**13**	**3**					**711**

Source: AMC, 2025.

18.8.2 Ex-pit dewatering system

Achieving and optimising design slope angles in West Africa requires depressurisation of saprolite using dewatering boreholes. Dewatering boreholes are drilled into the top of fresh rock, typically 80 – 120 m below ground depending on the thickness of the saprolite and the depth of fracturing in fresh rock. The boreholes are installed with PVC, with slotted (screened) PVC located below the saprolite and adjacent to the transition zone and fractured fresh rock to allow water to enter but sealed (isolated) from the saprolite to prevent the ingress of clay material that might damage pumps. The saprolite is then 'drained' by pumping water from the transition zone and rock below.

For the purpose of the PFS, data available from a historical hydrogeological study is combined with the pit designs to estimate future groundwater inflows:

- For the majority circular open pits, the Dupuit-Theim analytical equation for groundwater flow to a well is used (Table 18.6).
- For the linear pits of Makosa Main (north), to a well Darcy's equation for groundwater flow to a trench is used. Flow to Makosa Main (north) is estimated by calculating flow parallel and perpendicular to the orebody because the shear zone has higher permeability (Table 18.7).

The results are used to estimate the number of boreholes required to stop groundwater entering the open pits and depressurise the saprolite.

18.8.3 Dewatering borehole CapEx

Eighty-eight boreholes are estimated to be required based on an average borehole productivity of 3 L/s. Each borehole will cost approximately US$60,000, which includes a pump, the pump control system, rising main, electrical connections, borehole headworks and an allowance for surface pipe of approximately $5,000.

The total CapEx for ex-pit dewatering and depressurisation of the saprolite is US$5,280,000 to be profiled according to mine development.

Table 18.6 Estimate of groundwater inflow and the number of dewatering boreholes required for each pit (excluding Makosa Main north pit)

Pit	Baraka West 1- north	Baraka West 2 - central	Baraka West 3 - south	Baraka East 1 - south	Baraka East 2 - north	Makosa Tail 1	Makosa Tail 2	Makosa Main south
Area (m^2)	85,389	20,101	8,177	91,099	11,602	363,015	126,545	727,790
Effective radius (rw, m)	212.3	107.6	53.3	189.2	63.8	471.7	236.3	761.9
Pit crest	185	155	150	145	140	175	200	160
Water table	175	145	140	135	130	165	190	150
Base pit	110	125	125	120	60	0	105	30
Main aquifer thickness	60	60	60	60	60	60	60	60
Aquifer thickness at pit	60	20	15	15	60	60	60	60
Base Aquifer – Drawdown Level at pit (Hw, mRL)	115	85	80	75	70	105	130	90
R0 (input)	2,432	2,327	2,273	2,409	2,284	2,692	2,456	2,982
Q Steady state (m^3/day)	2,319	204	94	139	1,581	3,248	2,416	4,145
Q Steady state (L/s)	**26.8**	**2.4**	**1.1**	**1.6**	**18.3**	**37.6**	**28.0**	**48.0**
Number of dewatering bores	**9.0**	**7.0**	**5.0**	**8.0**	**6.0**	**11.0**	**8.0**	**14.0**

Source: AMC, 2025.

Table 18.7 Estimate of groundwater inflows and the number of boreholes required for Makosa Main north

	Flow parallel to shears	Flow perpendicular to shears
K	2	0.03
Deepest part of pit (aquifer)	90	90
SWL	150	150
R0	1,000	1,000
Hydraulic gradient	0.06	0.06
Aquifer width	300	3,800
Aquifer thickness	60	60
Flow (m^3/day)	2,160	422
Flow (L/s)	25	5
Total predicted inflows (L/s)		
Strike parallel	50	**Number of dewatering boreholes**
Strike perpendicular	10	**20**
Total groundwater flow	**60**	

Source: AMC, 2025.

19 Market studies and contracts

Thor operates the Segilola gold mine in Nigeria and has a contract with Metalor Technologies SA (Metalor) for the purchase and refining of gold doré produced at Segilola.

Thor has confirmed with Metalor that the gold produced at Douta would be subject to the existing contract terms.

All gold produced at Douta will be in the form of doré.

The contract with Metalor, stipulates that the responsibility of transportation from the Douta gold room to the refinery in Switzerland, will be the responsibility of Metalor, including security, transportation, and insurance.

There are two small airstrips within close proximity to the Douta project. The commercial Kedougou airport is approximately 60 km south-west and Endeavours Sabodala gold mine airstrip is approximately 30 km north-west. All gold shipments would then be transhipped via the international airport in Dakar.

Final gold sales price shall be based on either the London Bullion Market Association (LBMA) Gold PM Price (less a fixed fee per oz), or the Zurich spot price at Thor's election. The sales are in US$.

The Mineral Reserves were calculated at price of US$3000/oz. The QP is of the opinion that this price is reasonable.

20 Environmental studies, permitting, and social or community impact

20.1 Introduction

The Douta Gold Project is in eastern Senegal within the administrative region of Kédougou and the Saraya prefecture - a well-established gold-mining region that has expanded rapidly over the past two decades. Between 2021 and 2025, the Company completed progressive environmental and social (E&S) baseline studies covering ecology, hydrology, air quality, noise, community structure, and land use across both the Douta Exploration Licence and an adjacent Birima EL acquired more recently. The Baraka 3 area of the Birima EL is still to complete seasonal environmental baseline surveys. Elsewhere in the EL's full dry and wet season datasets were compiled during this period.

An ESIA, incorporating biodiversity, physical and social baselines was submitted in May 2025. This was revised in August 2025 following an interministerial government review and was then pre-validated in September 2025. Regional public hearings held in November 2025 confirmed strong community support. Final Ministry of Environment approval was attained on 16 January 2026. Additional environmental surveys undertaken in Q4 2025 at the Baraka 3 area are not included in this PFS as they are not yet complete. The approved ESIA defines a project area within the ELs and focuses on Phase 1 development (oxidised gold) in Makosa, Makosa North, Makosa East and Makosa Tail. Phase 2 of the ESIA will address the refractory gold processes and the expansion of the mine footprint and improved design layouts to the water storage dam, TSF, camp, waste rock dumps etc. The Phase 2 ESIA, expected in Q4 2026/Q1 2027, will outline the additional refractory process technology to be installed and confirm emission abatement systems. Other adjustments for tailings, water use, chemical use and storage will similarly be reassessed.

The ESIA currently covers only Phase 1. Approvals for Baraka 3 and Phase 2 roaster are pending and may materially affect the Mineral Reserves.

20.2 Douta project environment and social setting

20.2.1 Regional environmental context

The project is situated primarily in the Sabodala District, between the communes of Missirah Sirimana and Khossanto, with approximately one-sixth of the Douta EL extending into the Bembou District. The Falémé River, forming the border with Mali, lies 15 km north-east of the permit. To the southwest, the Niokolo-Koba River - an important tributary of the Gambia River - flows through the NKNP, a UNESCO World Heritage Site located 14 km from the project boundary.

The permit is bordered by the Kounemba Permit (Endeavour Mining) to the west and the Sambarabougou Permit (owned by Bassari) to the east. Several commercial mines operate in the broader region, including the Sabodala–Massawa complex (Endeavour), Sengold's Moura operation, as well as multiple exploration permits not yet developed.

The ELs are within a government designated Zone of Hunting Interest (ZIC or *Zones d'Intérêt Cynégétique*). ZICs have been established by the national government to regulate and promote hunting tourism. These zones are distinct from protected areas such as national parks, where hunting is totally prohibited. Licences and restrictions of fauna able to be hunted are designated for specific ZIC corridors.

Regional access is via major national routes (RN1 and N7) connecting Dakar, Tambacounda, and Kédougou, and secondary roads linking Saraya, Bembou, and Mali (via Moussala). The Project is also accessed via gravel roads from Mako, Kanouméring, Tinkoto, and Mandinkholé.

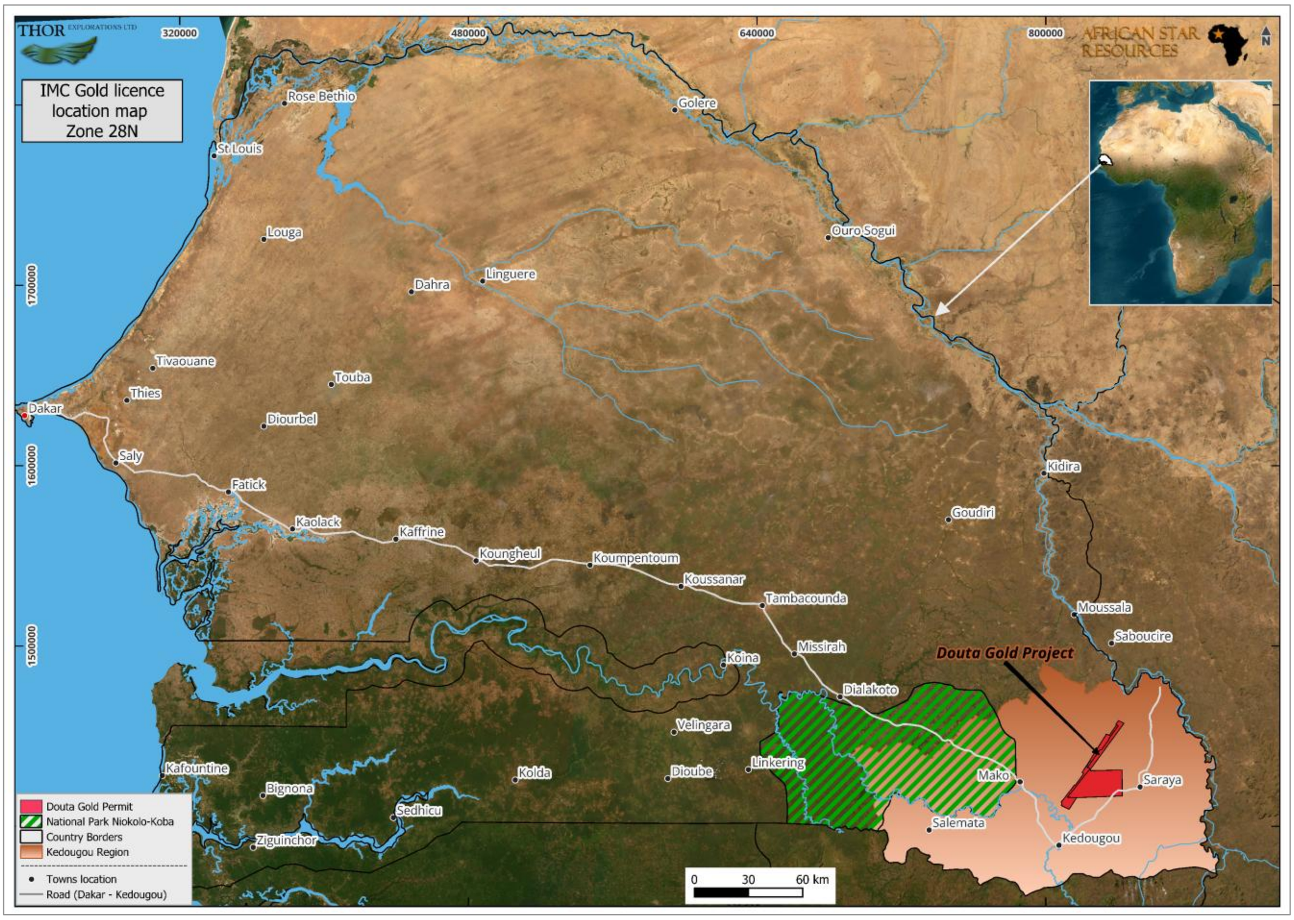
The Project climate is Sudano-Sahelian:

- Six-month wet season (May – October), peak rainfall July–September
- ~3,000 hours of annual sunshine
- Moderate winds (2.5-3.4 m/s)
- Dry-season humidity <35%

These conditions influence erosion potential, water availability, vegetation cycles, and air quality sensitivities.

Figure 20.1 highlights the regional aspects of the Douta Project location.

Figure 20.1 Regional location of Douta permit



Note: General location of the Project Permit.
Source: Thor, 2026.

20.2.1.1 Regional socio-economic setting

The ELs span the communes of Missirah Sirimana, Bembou, and Khossanto. Regional socioeconomic information presented in the Douta ESIA is based on Senegal's National Agency for Statistics and Demography (ANSD) 2023 census and survey data. This data shows the Kédougou Region is one of Senegal's fastest-growing (3.3 – 3.5% annually). The study-area population reached ~40,300 in 2023, with low density (5 – 9 persons/km^2) and a youthful demographic (46% under 15). Eighty-two percent are Senegalese; the remainder are migrants from neighbouring countries working in artisanal mining. Ethnic groups include Malinké, Diakhanké, Mandinka, Fulani, and Soninké.

The 2023 Census data also showed that land use is dominated by wooded and tree savannah (~97%). Agriculture (maize, sorghum, rice, groundnut) supports 34% of households but is constrained by limited equipment, poor storage, and weak market access. Livestock keeping (cattle, goats, sheep) is widespread but challenged by disease and inadequate veterinary support.

Mining - especially artisanal gold panning ("dioura") - is the region's main economic driver, engaging 43% of households. Issues include chemical use (mercury), unsafe deep pits, security risks, conflicts with industrial operators, and environmental degradation.

Access to public services remains limited. Health facilities consist of a small hospital and one clinic; prevalent diseases include malaria and gastrointestinal infections. Education levels are low (46% with no formal schooling). Water access is via wells and boreholes with variable yield; sanitation infrastructure is weak, with many uncovered pit latrines and widespread open dumping.

Energy is primarily solar and firewood. Transport relies heavily on lateritic roads that become unreliable in the wet season. Telecom coverage is intermittent.

No settlements lie within the mining footprint, but Sambarabougou and Mandankholi are within 0.2 – 1.3 km of the project boundary. Communities are familiar with mining impacts—both positive (employment, procurement, market creation) and negative (dust, blasting, land disturbance, competition for resources, influx of job seekers).

20.3 Environment and social baseline surveys

20.3.1 Overview of baseline studies

Between 2021 and 2025, Thor commissioned a series of dry and wet season environmental surveys covering ecology, hydrology, air quality, noise, and workshops and semi-structured interviews on socioeconomics factors. These studies provide a comprehensive multi-year baseline dataset across all seasons and project concept phases.

Table 20.1 Summary of environmental and social studies (2021–2025)

Month & year	Surveys	Area	Included in ESIA	Consultancy
May 2021	Dry Season Aquatic and Terrestrial Ecology Surveys	Douta	Yes	Synergie
Nov 2022	Surface & Ground water Baseline Surveys	Douta	Yes	AGTS
May 2023	Aquatic & Terrestrial Ecology Dry Season Survey including Chimpanzee Survey	Douta	Yes	Synergie
Oct 2023	Aquatic & Terrestrial Ecology Wet Season Survey including Chimpanzee Survey	Douta	Yes	Synergie
Nov 2023	Air Quality & Noise Baseline Surveys	Douta	Yes	Synergie
Nov 2023	Surface & Ground water Baseline Surveys	Douta	Yes	Synergie
Apr 2025	Biodiversity, Surface and Ground Water Surveys, Air Quality Surveys; Chimpanzee Survey	Douta and Birima	Yes	Synergie
Feb 2025	Consultation of stakeholders (Communities, regional technical services and central services)	Douta, Birima, and Baraka 3	Yes	Synergie

Month & year	Surveys	Area	Included in ESIA	Consultancy
Mar 2025	Environmental Impact Assessment Phase 1 submitted	Douta, Birima	ESIA Pre-validated 4 June 2025.	Synergie
Aug 2025	Environmental Impact Assessment Phase 1 Updated	Douta, Birima	Updated version responds to Govt responses - awaiting final approval Dec 2025	Synergie
Nov 2025	Environmental Impact Assessment Phase 1 Community Consultation	Douta, Birima	Public Consultation – community and regional authorities undertaken on Updated ESIA - Final Approval attained in January 2026	Synergie

Source: Thor, 2025.

All surveys were undertaken by accredited Senegalese firms (Synergie and AGTS) and follow national and international standards. Survey locations of these surveys are outlined in Figure 20.2.

For context, the ESIA defined three primary zones of analysis:

1. **Restricted study area:** The immediate project footprint where mining and processing infrastructure will be developed.
2. **Direct influence area (500 m radius):** Encompasses surrounding biological, physical, and socio-economic sensitivities, including flora, fauna, natural heritage, settlements, and cultural sites.
3. **Extended influence area:** The wider municipal and watershed context where indirect environmental and socio-economic effects may occur.

These zones align with IFC E&S PS and Senegal's national environmental assessment requirements.

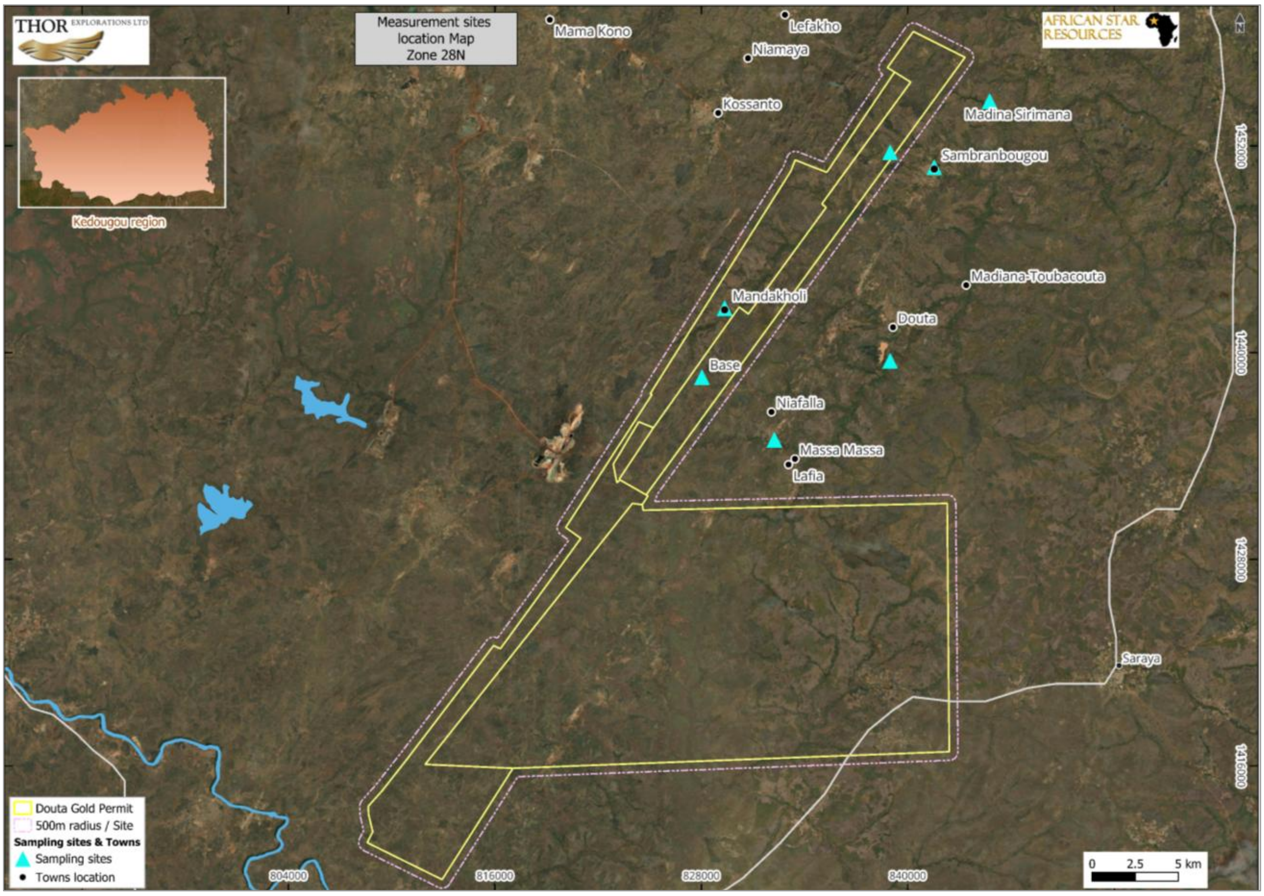
20.3.2 Environmental baseline

Information in the subsections below is summarised from the Douta ESIA (Updated November 2025) and includes information from baseline surveys outlined in Table 20.1.

20.3.2.1 Physical environment

Survey locations of physical environment aspects (NO_2, SO_2, noise, PM_{10}, $PM_{2.5}$, VOC) are outlined in Figure 20.2.

Figure 20.2 Physical environment sample points and local villages



Note: Location of measurement sites.
Source: ESIA, August 2025.

Regarding air quality the baseline surveys showed:

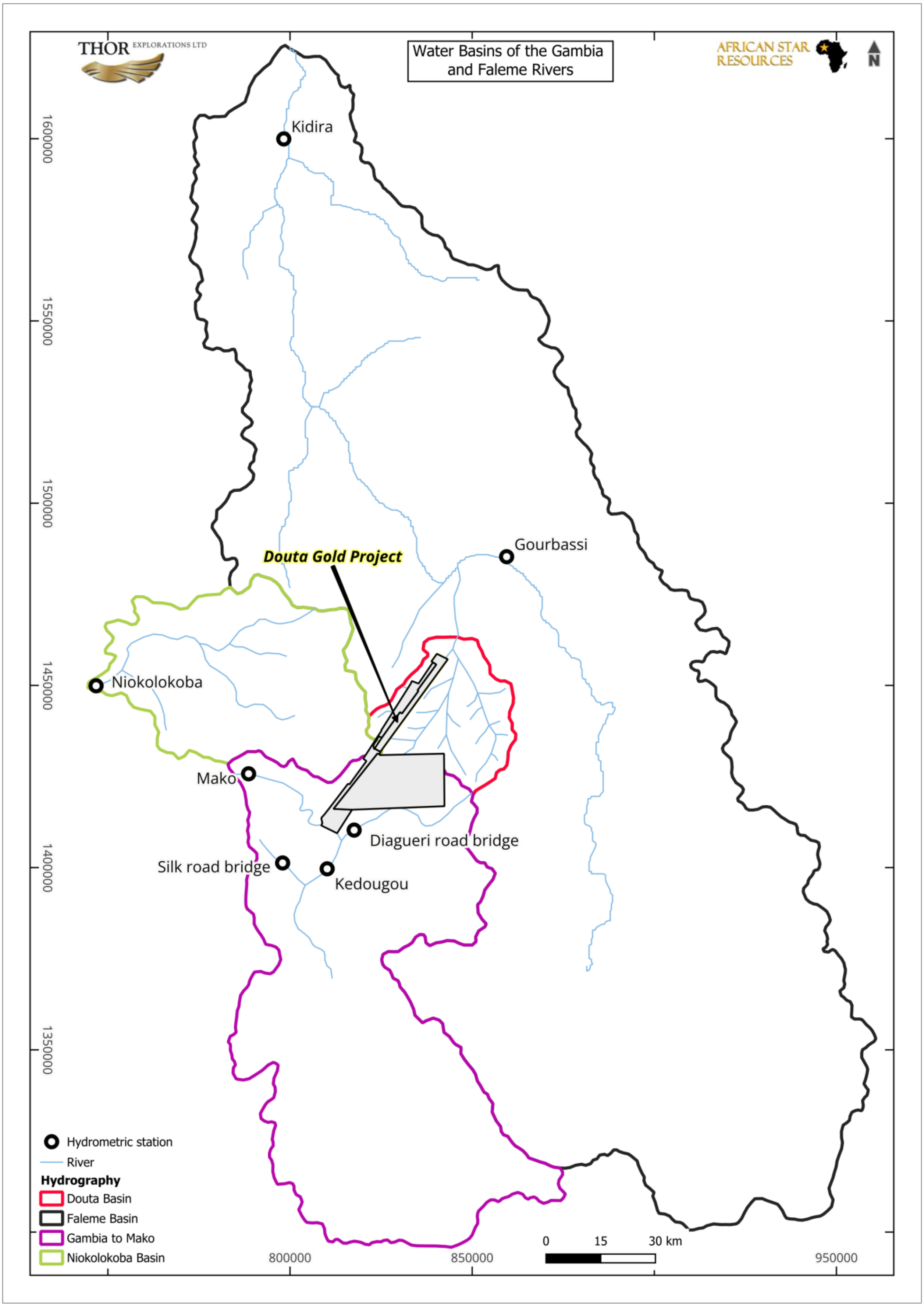
- PM_{10} concentrations exceeded 450 µg/m^3 in villages such as Sambranbougou.
- Primary drivers include unpaved laterite roads, motorcycle traffic, and local truck movement.
- Other potential pollutants ($PM_{2.5}$, SO_2, NO_2) were below Senegalese regulatory limits (NS 05-062, 2018).

As expected, given the predominately rural location, noise levels were within regulatory norms during both day and night.

With regard to surface water and groundwater, the hydrology regime is influenced by presence of two major seasons characteristic of the area's tropical domain: a dry season without any rainfall (from November to April), and a wet season with rainfall that extends from May to October. Temperatures are high throughout the year; but fall from June to September when the humidity levels rise sharply. It is also during this period that intense and continuous rainfall occurs, resulting in runoff in intermittent tributaries in the project area. This precipitation, combined with relative humidity, minimises evaporation and evapotranspiration values.

The surface water within the Douta permit lies within the Gambia and Senegal River basins and includes several intermittent tributaries that drain toward the Falémé River, a major regional watercourse (see Figure 20.3).

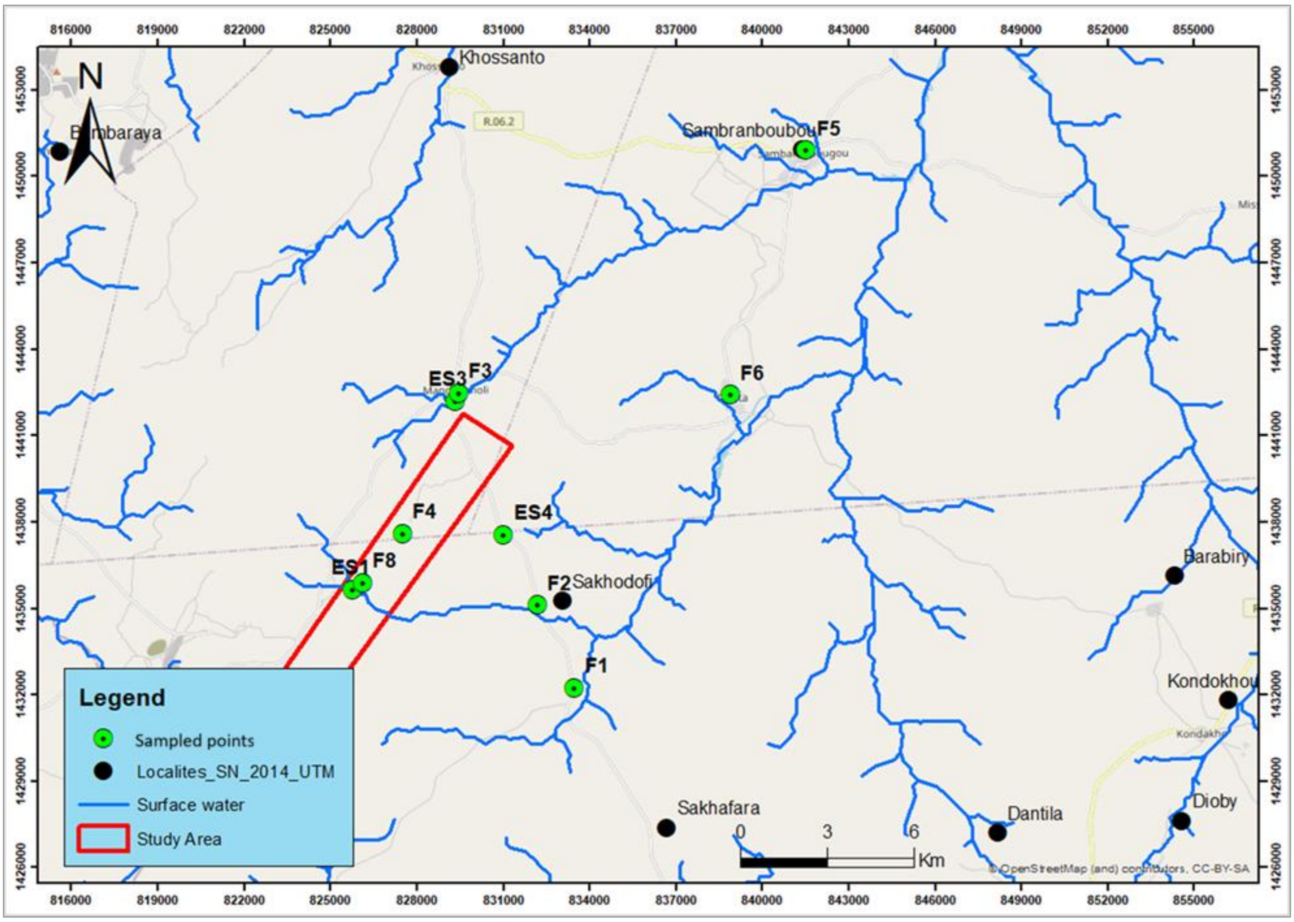
Figure 20.3 Location of Douta Project in the water basins of the Gambia and Falémé Rivers



Source: Douta Project ESIA, August 2025.

Water monitoring for surface and ground water was completed to establish a reference baseline for water quality within the proposed operating permit area. The assessment focused on the physicochemical characteristics, heavy metal concentrations, and bacteriological quality of both surface water and groundwater given current activities in the area including artisanal mining. Figure 20.4 below identifies the water monitoring locations upstream, midstream and downstream of the Douta mine project location.

Figure 20.4 Map of sampled points



Source: Map of Sampled Points Douta ESIA, August 2025.

Overall, the waters exhibited low mineralisation, with a slightly basic pH in groundwater and near neutral pH in surface waters. The ionic composition is dominated by calcium (Ca^{2+}) and bicarbonate (HCO_3^-) ions, typical of fresh waters. Geochemical interpretation suggests that silicate weathering is the predominant process governing mineralisation. No parameters exceeded regulatory or guideline limits, indicating good overall water quality. Bacteriological contamination has been noted with the presence of faecal coliforms in groundwater near the existing exploration camp.

For surface water the key survey findings were:

- Surface-water flow is restricted to wet season.
- Runoff intensity corresponds closely with rainfall.
- Several swamps and small tributaries cross or border the project area.
- No perennial streams occur within the permit boundary.

The proposed TSF, pits, and water-dam designs incorporate these seasonal hydrological patterns - refer Section 18 Project infrastructure.

Regarding groundwater baseline sampling revealed:

- **Low mineralisation** characteristic of fresh Ca–HCO_3 waters.
- **pH:** Groundwater: slightly basic.
- **No exceedances** of heavy metals or major ions against regulatory standards.
- **No evidence** of contamination from historical or artisanal mining.
- **Localised bacteriological contamination** (faecal coliforms) near the exploration camp, attributed to human activity rather than environmental conditions.

Groundwater is shallow and linked to weathered formations; deeper aquifers are not expected to interact with mining pits at initial oxide depths but should be monitored as operations progress.

In summary, given the mining context, special attention was directed toward heavy metal concentrations. Analytical results revealed no evidence of contamination, with values near or below detection limits—confirming the absence of heavy metal pollution that could eventuate from illegal mining or historic mining activities in the area.

With regard to acid rock drainage (ARD), it is recognised that Douta ore contains sulphur as metal sulphides primarily pyrite and arsenopyrite. The sulphur is generally oxidised in the oxide zones, partially oxidised in the transition zones, and highest in the fresh ore zones. Sulphur grades range up to approximately 3%. Mine waste rock will be stockpiled in mine waste dumps over the LOM. As such IMO recently completed preliminary ARD testing on Douta West samples. The results indicate that fresh sulphidic mine waste rock will require mitigation strategies to prevent or contain any ARD. The other materials classify as uncertain with regard to acid generation potential and a sampling and testing program testing is underway to develop mitigation strategies. At this PFS stage key ARD mitigation factors have been included in the design of the mining project – HDPE liner for the TSF (protection against seepage into groundwater) and a water deposit pond for water pumped out of the pit and from drainage for the waste rock dumps which will allow for water treatment prior to release to external water courses. Additionally, waste rock dump design will include layering and co-deposition of acid consuming waste rock from the pit and surrounding sources to further reduce the potential for acidic drainage.

Further detailed information on hydrogeology, watersheds and infrastructure (TSF, WSD, etc.) is outlined in Sections 4, 5, and 18 in this Technical Report.

20.3.2.2 Biological environment

With reference to flora surveys a total of 227 species were inventoried in 164 genera belonging to 50 families. The Fabaceae family is the most diverse with 59 species distributed in 33 genera. It is followed by the Poaceae (Gramineae) with 30 species and 23 genera, then by the Malvaceae which has 17 species and 12 genera. The genera Crotalaria and Indigofera (belonging to the Fabaceae) are the most diverse with 8 and 7 species, respectively.

In the woody layer, the most diverse genera are Acacia (5 species), Combretum (5 species), Ficus (4 species), and Grewia (4 species).

From the point of view of morphotypes, the flora of the permit area is composed of:

- 137 herbaceous plants
- 54 woody shrubby plants (maximum height 7 m to 10 m)

- 31 woody tree plants (minimum height of 10 m)
- 3 vines
- 1 palm tree
- 1 parasitic plant

Among the 227 species inventoried during the two seasons, four are classified in the IUCN's threatened species categories. These are three vulnerable species (VU): *Afzelia Africana, Khaya senegalensis* and *Vitellaria paradox* and a critically endangered species (EN): *Pterocarpus erinaceus*.

According to the Senegalese Forest Code, protected species are listed in the Douta project permit area. Among these are:

- Two fully protected species (IP): *Vitellaria paradox And Diospyros mespiliformis*.
- Nine partially protected species (PP): *Adansonia digitata, Afzelia africana, Borassus aethiopum, Cordyla pinnata, Khaya senegalensis, Pterocarpus erinaceus, Sclerocarya birrea, Tamarindus indica And Ziziphus Mauritian*.

All other species are of Least Concern or not listed under IUCN protection categories, and they also do not benefit from special protection status under Senegal's Forest Code.

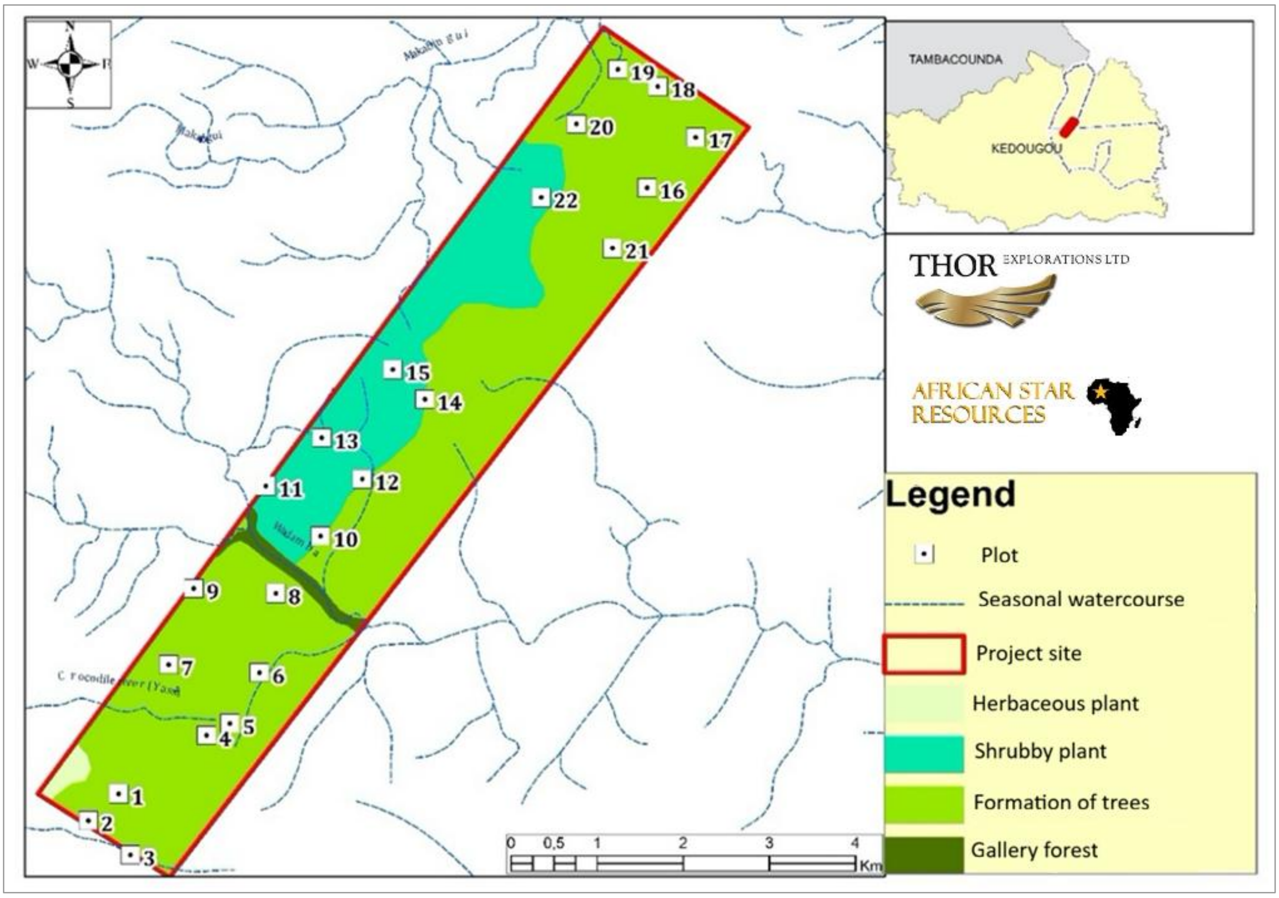
Information obtained from local populations and documentation established a list of plants most used by local populations. Apart from their regulatory and cultural services, plants are used in food (human and animal), traditional pharmacopoeia and in domestic uses such as timber and firewood. Several categories of plants can be distinguished:

- Multi-purpose plants: *Vitellaria pardoxa* (Shea tree), *Adansonia digitata* (Baobab), *Tamarindus indica* (Tamarind), *Cordyla pinnata* (Cayor pear), *Borassus aethiopum* (Rune palm), *Pterocarpus erinaceus* (Senegal rosewood), *Khaya senegalensis* (Caïlcédrat, Senegal Mahogany), etc.
- Medium-purpose plants: *Acacia seyal, Acacia sieberiana, Cassia sieberiana, Combretum glutinosum, etc*.
- Limited use plants: *Saba senegalensis, Vitex madiensis*.

Figure 20.5 shows the locations of plots inventoried in the ecology survey. In summary key flora findings are:

- Several species are protected under the Senegal Forest Code.
- Four species are listed by the IUCN (three Vulnerable, one Critically Endangered).
- Multi-use species (e.g., Vitellaria paradoxa, Adansonia digitata) are economically and culturally important.
- Vegetation is typical of Sudanian woodland savannahs with localised gallery forest.

Figure 20.5 Location of plots inventoried in the flora survey



Note: Map showing the location of the plots inventoried.
Source: ESIA, August 2025.

The inventory work relating to wild fauna has made it possible to find, directly or indirectly (through clues):

- 10 mammal species.
- 90 bird species.
- 2 reptile species.
- 2 bird species (Savannah Bateleur and Great Abyssinian Hornbill) are IUCN-listed as threatened.
- While chimpanzees were not directly observed, indirect evidence (tracks, nests, droppings) confirms the species' presence in the wider landscape.

For mammals, the red monkey (Patas), the green monkey, and the bushbuck have been well observed. For other mammals, including chimpanzees, indirect evidence (footprints, tracks, droppings) has been noted.

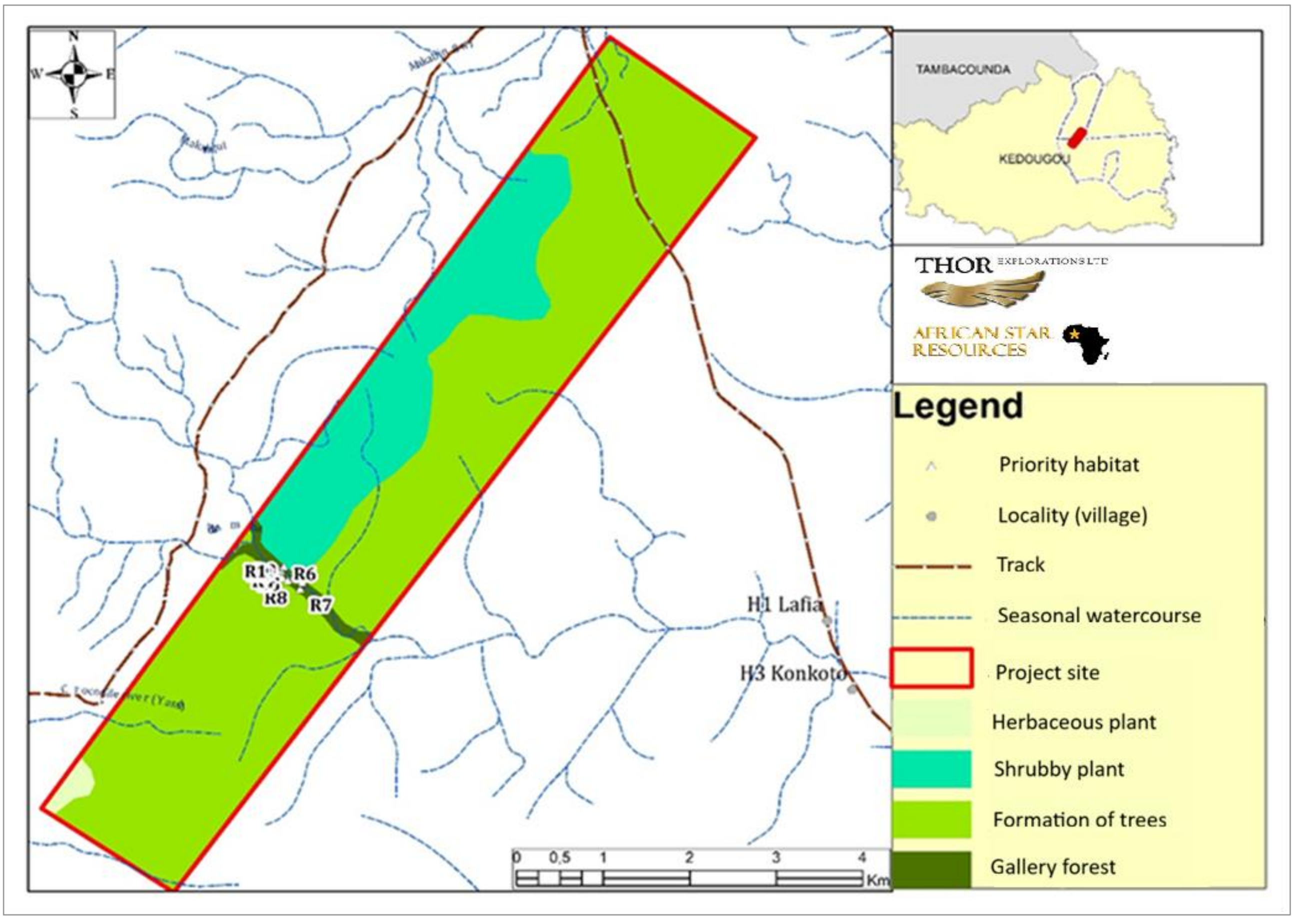
It is noted that the Kédougou region is home to 80-90% of Senegal's chimpanzees. Mining companies in the region have undertaken rigorous ESIAs to ensure biodiversity conservation through sustainable practices. The Douta ELs are adjacent to the Massawa and Sabodala mining permits and near Niokolo Koba National Park, a UNESCO World Heritage Site. Surveys on part of the Douta ELs were undertaken in 2021 and specialist surveys in 2023 and 2025. Evidence of 113 chimpanzee nests—all in the southern zone of the permit near Yassé confirmed continued chimpanzee presence, though mostly through older nests (95.6%). Nests were primarily built in tree species of Pterocarpus erinaceus, Parkia biglobosa, and Anogeissus leiocarpus. Other protected species identified included lions, bushbucks, green monkeys, and baboons.

The chimpanzee (and biodiversity) surveys also identified human activities—gold panning, deforestation, poaching, transhumance, and Borassus palm exploitation—pose major threats to wildlife. Recommendations to protect chimpanzee habitat included in the survey reports included the need to develop a biodiversity management plan to prevent species loss, foster collaboration between Thor and Endeavour Mining for joint biodiversity strategies (initial meetings have already commenced) and to implement measures to curb current human activities which are triggering deforestation, poaching, and gold panning impacts, and to mitigate potential impacts in wider zone of influence related to the Gambia River water quality and NKNP ecosystems. The biodiversity reports concluded that mining could proceed in the new Douta Project area due to the absence of chimpanzees, but urgent biodiversity management is essential to mitigate human pressures and safeguard wildlife in this ecologically sensitive zone which also borders a threatened World Heritage Site – the NKNP. This principle is included in the ESIA approved in January 2026.

The observed avian diversity is highly representative of the bird species reported in the study area. Of the 90 species belonging to 44 families, two species (the Savannah Bateleur and the Great Abyssinian Hornbill) are considered threatened according to the IUCN Red List. These highly mobile birds can move to other locations if significant changes are made. They remain very good indicators of environmental changes. Thus, they must be monitored at all stages of the process, even during the exploitation phase.

Figure 20.6 below outlines the key fauna area identified in the ecology surveys.

Figure 20.6 Priority habitats for conservation



Note: Priority habitats for conservation.
Source: Douta ESIA, August 2025.

In summary the fauna surveys indicated that:

- The study area has habitats suitable for hosting and conserving wildlife. However, the development of uncontrolled economic activities in this area will certainly reduce natural habitats and affect their resources. In the event of a major disturbance, animals that cannot adapt will move to areas where pressures are reduced or non-existent.
- Attention must be paid to endangered animal species by setting up a program for monitoring and regular analysis of their trends and the state of the resources they use in these environments.

The ESIA, in summarising the biophysical environmental aspects of the site (see Table 20.2), identified those requiring focus in environmental mitigation and management measures.

Table 20.2 Assessment of the sensitivity of the site (restricted study area) of the biophysical environment

Sensitive elements of the environment	Results of the analysis	Sensitivity	
		Not very sensitive	Sensitive
Natural environment			
Climate change	Rainfall, temperature, etc.	X	
Pedology	Most of the soils present in the area have either an agricultural or pastoral vocation		X
Geomorphology	Change in relief due to mining activities (Hill area)		X
Geology	The metamorphic base (depth 40 m) / depth of the deposit 15 m maximum	X	
Wildlife and plant resources			
Proximity to protected areas	PNNK (38 km)	X	
	NCR Nemenike (17 km)	X	
	ZIC Falémé		X
Presence of wildlife	Fauna under protection of Senegalese law and the IUCN		X
Water resources			
Aquifer	Gold mines are pits		X
Watercourses	Presence of the Niokolo and its tributaries the Yassé River		X

Source: Evaluation of the sensitivity of the site (restricted study area) of the biophysical environment, ESIA, August 2025.

20.3.3 Climate change and greenhouse gas emissions

The ESIA does not directly address climate change factors nor Green House Emissions. However, under Thor Company's Environment Policy the company states that, inter alia:

> *"We recognize that climate change is one of the most significant challenges facing the world today. We therefore aim to minimize our contribution to greenhouse gases (GHG), including by ensuring efficient use of energy and resources. We will also monitor and report on our GHG emissions progress so that project activities can be adjusted and form part of the Corporation's commitment to GHG reductions on a continual basis. We also ensure we will actively consider and plan for the risks of climate change to our business, and work with our host communities and other stakeholders on climate change related challenges"*.

The impacts of climate change have also been addressed in Sections 4, 5, and 18 on water resources hydrology. Further, Section 18 outlines fuel use options being explored to reduce GHG. As part of progressing the ESMP for the Douta Project a climate change risk analysis and GHG tool will be devised specifically for the project. The Douta Health Safety and Environment team are currently reporting on GHG emissions even at exploration stage.

20.3.4 Social baseline

Public consultation was undertaken as a core component of the ESIA for the Douta mining project. The exercises aimed to assess stakeholder acceptance, identify socio-economic and environmental risks, and integrate community concerns into project design.

20.3.4.1 Objectives and approach of the public consultation

The consultation sought to:

- Disseminate accurate information on the planned mining project.
- Assess perceived economic, social, and environmental impacts.
- Identify potential sources of conflict between stakeholders, particularly around land, forest resources, and gold-panning (dioura).
- Document expectations, recommendations, and concerns.
- Build consensus to support project acceptability and reduce future grievances.

A qualitative methodology was adopted, using semi-structured interviews and village assemblies (VA). Meetings included:

- Administrative and territorial authorities
- State Technical Services (STE)
- Local councils
- Communities of Massamassa, Sambrambougou, and Mandankholi

Each session included a project presentation followed by dialogue on stakeholder perceptions, fears, expectations, and recommendations. A final debrief validated stakeholder positions.

20.3.4.2 Summary of stakeholder inputs

Table 20.3 summarises the discussions and recommendations made by meeting attendees.

Table 20.3 Summary of stakeholder opinions and concerns, recommendations

Stakeholder group	Opinions & concerns	Recommendations
Administrative & Territorial Services	• Project seen as an economic opportunity but risks social conflict if forest use, land rights, and traditional gold-panning are not respected. • Past mining activities displaced communities and restricted access to fields, diouras, and grazing land. • Dust, traffic, and safety risks around villages. • Expect better communication after past conflicts (e.g., Sambrambougou incident requiring gendarmerie intervention).	• Inform authorities and comply with all legal procedures. • Integrate land-use plans (PDC, POAS). • Create dioura corridors. • Manage dust and traffic risks. • Prioritise local employment and youth training. • Support community investments (education, health, drinking water, women's gardening).
State Technical Services (STE)	**Natural Environment:** Heavy biodiversity pressure from mining and logging; lion corridors and a Hunting Interest Zone overlap with the permit. Loss of forest ecosystem services to villages.	• Complete all regulatory approvals (DMG, DEFCCS, DGPRE). • Demarcate permit boundaries to prevent land conflict. • Conduct forest inventory, pay logging fees, identify ecologically sensitive zones.

Stakeholder group	Opinions & concerns	Recommendations
	Hydrology: Poor site development may disrupt runoff feeding Falémé, Gambia, Senegal rivers and Niokolo-Koba ponds. **Regulation:** Project requires water extraction permits, compliance with classified facility laws, and formalised workforce registration. **Impacts:** Dust, noise, safety risks, groundwater competition, soil contamination risks, waste generation, restricted dioura access.	• Store topsoil for rehabilitation; establish baseline soil, water, air quality. • Install piezometers; develop proper TSF. • Implement HSSE plan, POI, waste management plan. • Collaborate with local communities on compensation and CSR. • Plan progressive rehabilitation and reforestation with forestry services.
Local Populations (Massamassa, Sambrambougou, Mandankholi)	• Communities do not oppose the mine and some have conceded fields, but fear eviction and loss of dioura access. • Gold panning is cultural, vital, and non-negotiable. • Mistrust created by past conflicts, unfulfilled promises and forced evictions. • Expect real benefits: employment, infrastructure, electrification, water supply.	• Maintain dialogue; avoid forceful evictions. • Return fertile dioura corridors. • Prioritise local youth employment and training. • Support women's vegetable gardens and mills. • Build classrooms, health posts, and rural access roads.

Source: Chapter 3 ESIA, August 2025.

From these meetings four cross-cutting themes were identified:

1. Economic Opportunities vs. Local Livelihood Risks: All stakeholder groups acknowledge the economic potential of gold mining for local and national development. However, communities fear:
   - Reduced access to ecosystem services (forest fruits, grazing, firewood, water points).
   - Loss of agricultural land.
   - Erosion of the traditional gold-panning economy.

   Unemployment remains a major issue despite numerous mining companies operating nearby:
2. Land and Forest Governance: Villages depend heavily on surrounding forests for subsistence. Because the project area is in a ZIC with wildlife corridors, strict regulatory coordination is needed. Villagers also request formal recognition and allocation of gold panning corridors.
3. Water and Hydrology Concerns: Stakeholders emphasise the risk of:
   - Disrupted runoff leading to drying of critical ponds in NKNP.
   - Groundwater competition between the mine and local households, livestock, and agriculture.
   - Potential contamination from hydrocarbons and processing chemicals.
4. Social Stability and Grievances: Past conflicts with other mining companies (evictions by gendarmerie) intensified mistrust. Stakeholders insist on a participatory grievance mechanism and transparent communication.

Stakeholders broadly support the Project if it delivers:

- Respectful coexistence with traditional gold-panning.
- Effective compensation and CSR.
- Environmental protection.
- Tangible improvements to local living conditions.

Despite high expectations, scepticism persists due to past unfulfilled commitments by other mining companies. Authorities insist that mining development must directly benefit surrounding communities, who face declining access to forest services, land, and gold-panning sites. Robust environmental management, transparent communication, community participation, and progressive rehabilitation are viewed as essential to preventing conflict and ensuring equitable development.

20.4 Land ownership

Land tenure in Senegal is governed by a combination of statutory law, customary land rights, and administrative land allocation mechanisms overseen by rural councils (Communautés Rurales) and the State. These overlapping systems shape how land is accessed, used, allocated, and compensated in mining project areas.

20.4.1 Statutory and customary tenure

The Douta Project area falls under the “Domaine National,” a classification that includes most rural lands. Under this system:

- The State retains ownership of the land.
- Local rural councils administer land allocations for agricultural, pastoral, residential, and community activities.
- Customary chiefs (“chefs de village”) and lineage heads (families or concessions) hold significant informal authority over land-use decisions.

Customary rights remain strong across all three communes intersecting the project area. Access for cultivation, grazing, and artisanal mining is typically negotiated at the village level, even though statutory approvals come from the commune and State.

20.4.2 Implications for project development

The ELs and Project layout do not include dwellings and structures within the two communities near the site – hence resettlement (and compensation) will be for economic displacement not physical displacement. Once the full mine project layout is finalised a definitive compensation boundary will be established for land required to be cleared for the project. It is expected that this will also be less than the Mining Licence boundary to minimise land already in agricultural use and maintain vegetation cover assisting in erosion control and maintain surface water quality.

Once the final layout is approved land access, clearing, compensation, and future livelihood restoration will require:

- A full Land Acquisition and Compensation Framework, consistent with IFC Performance Standard 5 and Senegal’s laws.
- Detailed mapping of farms, orchards, dioura corridors, sacred sites, grazing areas, and seasonal wetlands.
- Negotiation with both formal authorities (communes, prefectural administration) and informal authorities (village chiefs, elders, religious leaders).
- A grievance mechanism accessible to all users, including non-resident panners.

This framework will be finalised and disclosed before construction.

20.5 Project elements’ environment and social factors

20.5.1 Phases and timelines

The Douta Project is planned as a conventional open-pit gold mining operation supported by a CIL processing plant, a TSF, a raw water storage dam, power generation systems, fuel storage, workshops, an accommodation camp, and all associated environmental and administrative infrastructure. The submitted ESIA covers Phase 1 of the project, which focuses on the extraction and processing of oxide ore. A second phase (Phase 2), which will process fresh refractory ore, is intended for later development and will require the addition of a dedicated recovery circuit between the grinding and CIL stages.

20.5.2 Mining operations

20.5.2.1 Open pit design

Open pit mining at the Douta Project will be undertaken using conventional drilling, blasting, loading and hauling techniques, supported by hydraulic excavators and a standard truck fleet. Several pits will be mined along the orebody strike, beginning with Phase 1 extraction of the oxidised and highly weathered zone to depths of approximately 60 m. Phase 2 mining extends the pits down to about 200 m in fresh rock.

The pit design incorporates 6 m high benches with variable berms, and overall wall angles vary to reflect geotechnical conditions. Stormwater cut-off drains will be constructed adjacent to the perimeter haul roads to divert surface runoff away from the pits, thereby reducing flooding risk and improving geotechnical stability. Depressurisation measures will include pit sumps, vertical dewatering wells located behind pit crests, and installation of sub-horizontal drain holes in areas where pore pressure reduction is required. These measures are designed to collectively provide a safe working environment throughout the mine life.

During the construction phase, vegetation clearing and topsoil stripping will be completed across the pit footprint, and the topsoil will be stockpiled for use in future reclamation. Timber will be salvaged for community use. Pre-stripping will be performed using hydraulic excavators and haul trucks to expose the initial ore horizons. During operations, mining will proceed continuously, with waste and ore hauled to designated dumps or the ROM pad. All blasting operations are planned to remain more than 500 m from communities; however, should unforeseen geological conditions require blasting within this buffer, specialised controls such as modified charge design, blast mats, and short-duration community relocation will be implemented. At closure, dewatering activities will cease and the pits will be allowed to fill naturally unless underground mining requires an alternative strategy.

20.5.2.2 Waste disposal design

Waste rock management will be achieved by using engineered dumps established on prepared foundations following vegetation removal and topsoil recovery. Waste rock dumps will be constructed in 10 m lifts with 5 m berms, to ensure long-term structural integrity and reduce erosion. For Phase 1, the waste dumps have been sited to align with existing topography, ensuring that their crest does not exceed natural ridgelines and does not interfere with drainage or sterilise potential future resources.

Construction activities include progressive clearing, grading and establishment of access routes. Waste rock placement during operations will be controlled to ensure that potential mineralised areas remain accessible, natural watercourses are not obstructed, and progressive rehabilitation can occur once individual dump heights are achieved. Topsoil stored during clearing will be used to rehabilitate the outer batter slopes. Runoff from dump surfaces will be collected in toe drains, routed to sumps for sedimentation, and then pumped to the site water dam. As the mine approaches closure, the dump crests will be shaped with a gentle positive gradient to maintain drainage away from the centre and facilitate long-term stability.

20.5.2.3 ROM pad and ore stockpiles

Construction of the ROM pad and stockpile area will involve site clearing, grading and profiling to create a stable, trafficable surface for ore handling operations. Vegetation and topsoil will be removed and stored, and any salvageable timber will be available to local communities. During operations, ore will be delivered to the ROM pad by haul trucks and organised into separate stockpiles based on grade and material type to maintain consistent feed to the processing plant. Front-end loaders will manage stockpile movement, and the layout will accommodate efficient segregation of oxide, transition, and fresh ore when applicable.

20.5.2.4 Process plant

Phase 1 process oxide ore

The Douta process plant is designed to treat 4.0 Mtpa of ore using a conventional, industry proven flowsheet that emphasises robustness, high availability and cost-effective operation. Phase 1 focuses on relatively simple oxide ore, while space has been reserved for the addition of a fresh ore recovery circuit required in Phase 2. The plant layout allows good access for maintenance, and the overall design philosophy prioritises reliability and operability while avoiding unnecessary complexity. The facility will run 24 hours per day for 365 days per year with an availability target of 90.4%, corresponding to a nominal feed rate of approximately 505 tph.

Crushing and Stockpile Management: ROM ore will be delivered to a 200 t live-capacity bin fitted with a static grizzly to screen out oversize rocks (>415 mm). A rock breaker will be used to clear any bridging. A single-stage jaw crusher will operate at approximately 609 tph to reduce the ore to a size appropriate for SAG milling (150 mm). The crushed ore stockpile will provide buffer storage of 12,000 t, ensuring 24 hours of mill feeding capacity. Dust control infrastructure, including enclosures, skirting and water spray systems, will be employed at all major transfer points.

Grinding and Classification: Crushed ore will be reclaimed using dual apron feeders and transferred to a SAG mill equipped with a variable-speed drive to accommodate ore hardness variability. The circuit includes a ball mill in closed circuit with hydrocyclones, targeting a final product P_{80} of approximately 74 µm. SAG discharge will be screened by a trommel, with oversize “scats” diverted to a bunker. A portion of cyclone underflow—about 20%—feeds the gravity circuit, while the remainder returns to the ball mill. The system is fully instrumented and uses weightometers, flow metres and automated dilution control to maintain stable operating conditions.

Gravity and Intensive Leach Circuits: The gravity circuit includes a scalping screen to remove coarse material (>2 mm), followed by a Knelson concentrator. Gravity concentrate will be collected in a cone and processed daily in an intensive cyanidation reactor (ICR). Leach solutions containing cyanide and caustic will be heated and circulated to maximise gold dissolution. The resulting pregnant solution is recovered in a dedicated electrowinning cell, and the barren solution returns to the CIL circuit.

CIL Leaching and Adsorption: Cyclone overflow feeds a vibrating trash screen, after which the slurry enters the leach tanks. Ten leach tanks provide approximately 24 hours retention time, where activated carbon adsorbs dissolved gold. Air sparging, lime addition, cyanide dosing and agitation are controlled to maintain optimal leaching conditions. Carbon transfer between tanks will be performed using air-lift pumps, and interstage screens maintain carbon within the appropriate stages.

Elution, Electrowinning and Smelting: Loaded carbon is screened and transferred to the elution area, where it is washed and stripped at high temperature and pressure using caustic soda solution (without cyanide). Gold-enriched eluate is passed through an electrowinning cell fitted with carbon fibre cathodes, where gold is plated out. Collected sludge is washed, treated with nitric acid to remove impurities and smelted into bullion bars within a secure gold room.

Carbon Regeneration and Detoxification: Stripped carbon will undergo acid washing and thermal regeneration at temperatures exceeding 700°C to restore adsorption capacity. Tailings from the CIL circuit will be detoxified using the SO_2 / air process, converting cyanide to cyanate under controlled pH conditions, with copper sulphate serving as a catalyst. Detoxified slurry complying with emissions standards will then be pumped to the TSF.

Reagent handling systems include bulk cyanide storage and mixing, lime slaking and distribution, caustic and acid mixing facilities, activated carbon storage and diesel supply infrastructure.

Management of Sodium Cyanide and other hazardous reagents are planned to be in compliance with applicable regulatory requirements and aligned with internationally accepted best practices. Storage, handling, and use of cyanide will occur within engineered facilities incorporating secondary containment and controlled access, with reagent storage areas located away from surface water features. Standard operating procedures, personnel training, and emergency response measures will form part of the site's environmental and health and safety management systems. Process solutions and effluents are planned to be treated to reduce cyanide concentrations prior to discharge or reuse, with environmental monitoring programs implemented to confirm regulatory compliance.

All other materials used in processing and mining operations will be stored and managed in accordance with regulatory requirements.

Phase 2 refractory process for fresh ore

The Phase 2 suspension roasting plant is designed to treat 2.4 Mtpa of fresh ores that are refractory. The ores contain both sulphides as pyrite and arsenopyrite, carbon and silicates. Gold is locked in sulphides and in silicates. Roasting followed by oxidation of the ore oxidises sulphides and carbon and increases degradation of the silicates to release gold for subsequent recovery by CIL processing.

Ore feed is dried and pre-heated in off gas cyclone separators before dropping via gravity to be heated to 650°C in the 2-stage heaters. Hot feed transfers via gravity into the oxidation reaction chamber maintained at 650°C for 60 minutes to complete the oxidation process. Calcined product transfers to two cooling cyclone separators before quenching and repulping ahead of processing in the CIL circuit.

With regard to emissions the cyclone separator exhaust gases are extracted to a baghouse dust collection system comprised of electrostatic precipitators. Dust is recycled to the ore feed bin and gases passed to the gas scrubbing circuit.

Hot exhaust gases contain nitrogen oxides (NO_x), sulphur dioxide (SO_2), and may contain low levels of arsenic trioxide (As_2O_3).

The level of As_2O_3 is controlled by the temperature and oxygen concentration. Heating in the two stage heaters under oxygen deficient conditions minimises As_2O_3. Over 600°C, the arsenic will form ferric and calcium arsenates ($FeAsO_4$, $Ca_3(AsO_4)_2$) solids and will deport with the calcined product to CIL. The heaters operate at 650°C so arsenic will be retained as arsenates in the calcined product.

The remaining gases are treated with two stage scrubbing.

Selective catalytic reduction (SCR) will be used to reduce the level of NOx in the presence of a catalyst. The process introduces a reducing agent, usually ammonia or urea, over the catalyst where the nitrogen oxides are reduced to nitrogen gas (N_2) and water.

Wet flue gas desulfurisation method will be used where limestone is used to remove the SO_2 from the flue gas by passing the flue gas through a chamber that exposes the flue gas to a slurry of finely ground limestone. The calcium in the limestone slurry combines with the SO_2 to form calcium sulphate or gypsum which collects as a solid. The gypsum is inert and can be safely disposed in the TSF.

20.5.2.5 Tailings Storage Facility (TSF)

The TSF has been designed by Knight Piésold as a no-release downstream-constructed embankment system engineered to safely store approximately 25 Mt of tailings over an initial 10-year LOM. The design incorporates multiple engineered layers to ensure long-term stability, including an HDPE liner, compacted fine soil barrier, filter chimney drains, and successive layers of compacted soil and rockfill. Waste rock sourced from the open pit, subject to confirmation of non-acid-generating properties, will be used for future embankment raises.

During construction, vegetation and topsoil will be removed, with salvageable material stored for later use in reclaiming the embankment slopes. Embankment construction will use excavators, haul trucks, graders and compactors, drawing materials from within the basin and nearby waste rock sources. Tailings will be thickened to around 40% solids and pumped via a HDPE pipeline to the TSF, where controlled deposition through spigots will form a beach promoting drainage and consolidation.

Water management is central to TSF performance. A decant system will capture supernatant water for return to the process plant, providing 40% of the operation’s total water requirement. Stormwater diversions and drainage controls will prevent erosion and maintain embankment integrity. A TSF Operations Manual will be prepared inclusive of requiring continuous monitoring—including piezometer readings, stability assessments, groundwater and surface water sampling, and routine pipeline inspections to ensure safe operation.

Closure activities will include regrading the surface, capping with topsoil, establishing vegetation, and sealing the decant mechanism. A rock-lined emergency spillway will remain in place to manage extreme rainfall events. Long-term monitoring will continue into the post closure period.

20.5.2.6 Civil construction

The supporting civil works for the project include site access roads, haul roads, the processing plant platform, mine workshops, warehouses, fuel and explosive storage areas, sewage and waste facilities, power supply and communications infrastructure. Construction activities include land clearing, earthworks and the use of timber for construction where appropriate, with surplus timber distributed to local communities. Waterproof bunds (using laterite or other liners) will also be provided around relevant facilities to trap spillages facilitating cleanup and disposal – either back through the process plant (or STP) or transported offsite to a licenced facility. During operations, workshops will provide maintenance services for vehicles and equipment. At closure, all structures will be dismantled, and the land reprofiled and revegetated unless repurposed by local communities.

Explosives Magazine: A dedicated explosives magazine will be located approximately 3 km from the main mining area. It will store ANFO, emulsions, boosters, and non-electric detonators in compliance with Senegalese explosives regulations. Transport will occur in approved vehicles following strict safety protocols.

Site Water Management: Process operations require approximately 4 Mm^3 of water per year, supplied by reclaimed water from the TSF and supplemented by a planned 5.4 Mm^3 raw water dam located west of the mineralised zone. The site layout (to be reflected in the subsequent Water Management Plan) emphasises separation of clean and dirty water, efficient recycling, minimisation of raw water consumption, regular inspection of drainage systems, and monitoring of both surface and groundwater quality. Seasonal water balance variations will be addressed by operating the TSF pond at low levels during the dry season, ensuring adequate storage capacity for wet-season inflows.

Power Supply: A 32 MW onsite power plant will be required for project operations. While a HFO / diesel fuelled plant is currently planned, the company is evaluating a hybrid energy solution combining

LNG-fuelled generation with solar power, aiming to reduce greenhouse gas emissions and operating costs. LNG would be transported by road from Dakar, drawing on experience from the company's operations in Nigeria.

20.5.2.7 Transport

Access to the project area will be via the established Dakar–Kédougou highway and a sealed road already used by nearby mining operations. Heavy machinery, fuel, reagents and consumables will be transported along this route. Increased traffic during construction and operations will necessitate routine road maintenance and periodic road safety management.

20.5.2.8 Employment

Approximately a thousand personnel will be employed during the construction phase. During the operations phase approximately 650 personnel will be employed across mining, processing, administration and support functions. The employment strategy prioritises local labour, followed by national recruitment, with expatriate staff limited to specialist roles where local expertise is unavailable. Comprehensive training programmes will be implemented for semi-skilled positions, including equipment operation and plant technician roles. The approach mirrors the successful staffing model used at the company's Segilola mine in Nigeria and emphasises community engagement and long-term skills transfer.

20.6 Permitting and regulatory compliance

20.6.1 Senegalese requirements

The project is a Classified Installation (ICPE) and must comply with:

- Environmental code
- Mining code
- Forestry code
- Labour code
- Water and sanitation regulations
- Cultural heritage protection requirements

20.6.2 International requirements

Thor's Environment Policy explicitly states that the company and its subsidiaries will, inter alia, be:

> *"Complying with all applicable environmental laws, regulations, and other environmental obligations in countries in which we operate. This includes attaining relevant permits and licenses as required and monitoring their implementation.*
>
> *Maintaining environmental systems, plans and procedures to identify, monitor and control the environmental aspects of our activities.*
>
> *Applying well-researched and tested environmental management practices to prevent pollution and mitigate impacts.*
>
> *Complying with environmental emissions standards as set out in national legislation and where relevant international guidelines."*

As relevant the company would comply with the International Finance Corporation's Performance Standards on Environmental and Social Sustainability as required by international lenders. Other relevant IFC PS guidelines include Environmental, Health and Safety Guidelines for Mining and General Environmental, Health, and Safety General Guidelines which address:

- Environmental standards including air emissions, wastewater and water quality, hazardous materials management.
- Occupational health and safety such as PPE, physical and chemicals hazards.
- Community health and safety including traffic safety and transportation of hazardous materials, life and fire safety.
- Construction and decommissioning including environment and community health and safety.

Management plans required for the above would be prepared prior to construction and may be in addition to those mandated by Senegalese ESIA regulations.

20.6.3 Environmental impact assessment status summary

- Phase 1 ESIA submitted: May 2025.
- Interministerial / Regional review completed: August 2025.
- ESIA pre-validated approval issued: September 2025.
- Community and Public Hearing November 2025 – full endorsement by community.
- Approval attained: 16 January 2026.
- Phase 2 ESIA to be prepared and submitted in line with the DFS process – expected ESIA submission in Q4 2026 / Q1 2027.

20.7 Impact assessment and mitigation measures

20.7.1 Environmental risks and mitigation measures

The ESIA identified potential environmental impacts on land, water, air, biodiversity, and social systems. Land disturbance and vegetation clearing will result from pit excavation, road construction, and infrastructure development. Airborne dust and emissions may affect local air quality, while improper waste or water management could risk contamination.

Social impacts include temporary displacement of land users and reduced access to grazing and forest areas. However, positive impacts—such as employment, training, and infrastructure improvements—are expected to outweigh negative effects when mitigation is applied.

Hazard analysis identified risks including diesel fires, explosive process and storage accidents, and mechanical failures. Though their likelihood is low, their potential severity requires proactive management. The Company plans to establish a dedicated Health and Safety Committee, enforce fire permit systems for hot works, and conduct biannual emergency drills.

A summary of the key risks and mitigations proposed is outlined in Table 20.4.

Table 20.4 Summary of environmental risks and mitigation

Topic	Risks	Mitigation
Soil & land disturbance	Vegetation loss; erosion; compaction; contamination	Biodiversity Management Plan, Topsoil stripping & reuse; stormwater drainage and erosion control; progressive rehabilitation; hazardous-material controls
Air quality	Dust from haul roads, blasting, crushing; vehicle emissions; process-plant emissions; STP / WTP/ camp / workshop emissions	Watering; ≤30 km/h speed limits; covered loads; dust -suppression systems; embedded plant / process mitigation; compliance with national / international emissions standards; emissions monitoring (NS 05-062)
Water resources	Hydrocarbon spills; process-water leaks; TSF seepage; altered runoff; STP / WTP leaks or malfunctions	Lined storage / ponds; diversion channels; oil-water separators; fuel handling controls; routine water monitoring; bunds; compliance with surface / groundwater emissions standards
Biodiversity	Habitat loss; wildlife disturbance; protected-species risks; onsite / downstream pollution; hunting; illegal gold panning; noise disturbance	Biodiversity Action Plan; native species nurseries; rehabilitation; speed limits; antipoaching policy; fauna-safe waste management
Landscape & visual	Altered topography; dust deposition; temporary vegetation loss	Dust suppression; progressive rehabilitation; regrading and revegetation
Traffic	Traffic accidents; heavy-machinery interactions; in-migration	Traffic plan; community awareness; local hiring preference; coordination with authorities; human-rights–aligned security
Hazards & emergency response	Diesel / tank fires and explosions; explosive-magazine / ANFO incidents; conveyor-belt fires; hydrocarbon spills	Internal Emergency Response Plan (POI); fire-permit system; trained emergency-response team; evacuation drills; storm-detection systems for blasting; safe explosives transport; explosives depot >500 m from pits; traffic controls; chemicals and hazardous chemical plans and operating procedures

Source: Summary of actions for long-term environmental and social management of the areas concerned, ESIA, August 2025 and Thor recommendations.

20.7.2 Community development commitments

Based on economic plans already compiled at regional level and as discussed in the community meetings held as part of the ESIA process, Thor proposes to support:

- Electrification improvements.
- School and health-post enhancements.
- Road / bridge upgrades.
- Municipal service improvements.
- Local employment, local procurement and training initiatives.
- Support for existing government designated gold panning corridors (note these are located outside of the Project Exploration License areas).

20.7.3 Environnemental and Social Management Plan (ESMP)

The ESIA defined key impact and mitigation measures which have been outlined in an ESMP. The ESMP will be updated once the Phase 2 ESIA is completed and as informed by a DFS as the next stage of Douta Project's technical design and costing process. The ESMP consolidates all commitments, procedures, and monitoring requirements.

Key management plans are outlined in Table 20.5 below.

Table 20.5 Key topic specific Environment and Social Management Plans

Category	Plans / actions
Environmental	Soil protection, air-quality management, water-quality management, biodiversity conservation, waste & hazardous materials management, environmental monitoring.
Social	Community health / safety, compensation processes, livelihood restoration, community development commitments.
Monitoring & Compliance	Regular reporting to authorities / communities, inspection schedules, performance indicators.
Cross-cutting Plans	Cyanide management, noise / vibration plan, cultural-heritage chance-finds procedure, human resources and training; procurement, waste management; transport management; and security.

Source: Summary of actions for long-term environmental and social management of the areas concerned, ESIA, August 2025.

Additional Environment and Social Management Plans will also be prepared (if required) outside of those mandated in the ESIA to address IFC E&S PS requirements. These would include:

- Stakeholder engagement plan and transparent grievance process.
- Community health, safety, and security plan.
- Human resources and training management plan.
- Hazardous chemicals management plan.
- Waste and hazardous waste management plan.
- Carbon management and greenhouse gas management plan.
- Cultural heritage management plan and chance find procedures.
- Site specific health, safety, and environment management plan.
- Livelihood restoration plan and compensation procedure.

Other environmental and social commitments in the ESIA ESMP included:

- Obtaining all permits pre-construction.
- Demarcating the mine perimeter.
- Ecological inventories & carbon assessments.
- Wildlife-corridor protection.
- TSF, waste system, traffic plan compliance with standards.
- Air / noise / water monitoring stations.
- Topsoil conservation and reuse.
- Progressive reforestation (economically valuable species).

Health, Safety & Labour assessment and permits including:

- Enforcing PPE, HSSE procedures
- Registering workers
- Operational Emergency Plan (POI)
- Vehicle maintenance and safety
- Regular training and inspections

The ESMP will be updated before construction and will remain active throughout operations, closure, and post-closure.

20.8 Mine decommissioning and closure plan

20.8.1 Overview

The closure objectives for the Douta Gold Mine Project are described in Table 20.6 for each mining component.

Table 20.6 Closure objectives for each project component

Mine component	Closure objectives	Start	Finish
Open pits	Dewatering will cease on closure of the mine and the open pits will be allowed to flood. The pit slopes will be stable and groundwater quality will not be compromised. Reassignment to local community for water storage / use. A pounded protection bund will be formed around the pit to discourage entry.	Phased implementation of decommissioned pits commencing 2 years before End of Project	End of Project
Waste rock dump	Final slopes of the waste rock dump will be stable and any re-vegetation carried out.	Phased implementation of stockpiles once maximum height is achieved commencing 2 years before End of Project	5 years post closure
TSF	Profiling of the dam surface and layering with soil to encourage revegetation 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. Re-profiling and re-vegetation of the site.	End of Project	5 years post closure
Raw and process water storage tanks	Evaporation of remnant water, removal of contaminated solids to a waste disposal location offsite.	End of Project	5 years post closure
Raw water dam	Reassignment of responsibility to local community.	End of Project	End of Project

Source: Thor, 2025.

These are preliminary objectives which will be revised throughout the LOM. The removal of site infrastructure will be carried out unless public demand facilitates the use of the buildings etc. after mine closure.

The Project will implement a mine site reclamation plan at least 12 months prior to closure. The plan will focus on the reclamation of the TSF, ROM Pad, ore stockpile areas, process plant, and workshop. The main objectives of the plan will be to:

- Promote alternative economic activities in the area that are sustainable in the future.
- Ensure the safety of surrounding communities through public consultation and the erection of warning signs.
- Return the land to conditions capable of supporting the former land use (woodland and agriculture), or where this is not practical, or feasible, an alternative sustainable land use.
- Prevent potential significant adverse effects on adjacent water resources, namely groundwater and surface water contamination.

20.8.2 Closure activities

Mine dewatering will cease and the pit will flood to a depth approximating the natural water table. The likely use is that the water from the pit will be available for agricultural irrigation.

The surface waste rock dumps will be re-profiled to provide safe stable slopes and will be rehabilitated with some topsoil replacement and re-vegetation.

All ore on the ROM Pad and transient ore stockpiles will be processed. The area will be reprofiled to establish the natural drainage pattern. The following plant and equipment dismantling and disposal practices will be applied to the crusher plant, mill, process plant and workshops, provided there are no requirements for them from the local businesses, other mining companies, or government agencies:

- Removal of all brick buildings.
- Breaking out and removal of all concrete foundations; other than grinding mill and crusher foundations.
- Removal of steel frames.
- Demolish reinforced concrete structures and dispose of on site.
- Remove HDPE liners and backfill all process ponds.
- Remove electrical equipment, pumps, motors, and other fixed equipment.
- Remove all fuel storage tanks.
- Cut up and remove all steel tanks and vessels.
- Remove all pipelines.
- Dig up and remove all below-ground electricity cables.
- Remove conveyor-belting.
- Remove all mechanical equipment.
- Materials handling areas will be cleared of all raw materials.
- General site clean-up.
- Site levelling and profiling to re-establish the natural drainage pattern across the site; and Re-distribution of the stockpiled soils and re-vegetation of the site with indigenous grasses and trees.

Concrete foundations will be retained for use as foundations for future buildings if required. Septic tanks will be emptied, and the sludge will be treated to render it harmless, and it may be used in the re-vegetation process. Scrap metals and equipment will be sorted and sold. The company will remove, or otherwise dispose of, all equipment and materials that cannot be reused, recycled or sold, to an approved non-hazardous disposal site.

At closure, the upper surface of TSF will be re-profiled and re-vegetated. Stockpiled organic matter and soil from the initial site clearance will be spread over the storage facility to promote the growth of indigenous trees, shrubs and grasses transplanted from the mine nursery. Advice will be sought from a competent person regarding the species and diversity of vegetation to be established on the TSF. The decant for the tailings dam will be sealed. The TSF outer slope angles of approximately 18° will be resistant to erosion when re-vegetated. Test work carried out indicates that the tailings have very low net acid-producing potential and low sulphide content and as a result the tailings have been classified as low risk concerning ARD. A full review of the stability of the tailings dam and additional measures will be undertaken in order to close the tailings dam safely, will be carried out by competent external person(s) and a detailed tailings closure plan will be developed by the company a year before the operation is scheduled to cease.

Transport infrastructure such as site access roads, bridges and drainage channels will be removed, ripped and re-vegetated if the local community do not want them to be retained. This will be evaluated at the appropriate pre-closure time through the company's ongoing consultation program.

After cessation of mine operations, all pond silt will be tested for contamination. The results of this test work will indicate the measures needed to stabilise the silt to avoid additional contamination of the surface and groundwater. The ponds will be re-profiled with the addition of waste rock, previously stripped topsoil and organic matter, and re-vegetated.

For the purposes of re-vegetation, stripped soil and organic material will be stockpiled for future use during site preparation. A nursery of young trees will be established on site within three years of operations for the concurrent and progressive re-vegetation of closed stable areas. The nursery will consist of species endemic to the local area and some coloniser species which are tolerant to the dry conditions likely to be experienced at the tailings dam. In general, the stages of re-vegetation will be:

- Development of the Nursery.
- Site stabilisation and profiling.
- Site contamination assessment to evaluate conditions needed for re-vegetation, addition of stripped soil, fertiliser, and organic material.
- Planting or transplanting of seedlings from the nursery (under the supervision of a competent forester).
- Post-planting care (watering, and fertilising).
- Final site inspection to clarify if the re-vegetation is successful.

The areas to be re-vegetated during the demobilisation will include but not limited to:

- Waste rock dump
- Tailings dam
- Ore stockpiles
- Settling ponds
- Drainage channels
- Closed areas of the plant
- Roads and areas underlying removed infrastructure

The timetable for successful re-vegetation of all relevant areas of the project area is five years post closure.

A soil survey will be conducted at closure to identify any areas of inorganic and / or organic soil contamination. The soil survey will involve a series of test pitting to a depth of 500 mm, soil sampling and analysis. The number and location of test pits will be based on a site walkover / inspection at closure to identify potentially contaminated soils. A deeper soil inspection may be necessary at specific hotspots (pollution sources) depending on the findings of the near-surface soil survey.

Localised soil contamination resulting from the accidental spill of diesel and oil will be treated by the removal of contaminated soil from affected areas to an appropriate disposal facility. Soil contaminated with chemicals, reagents or oils will be removed to an approved hazardous waste disposal site.

The ROM pad, ore stockpiles and process plant site will be re-vegetated following the removal of any remaining ore, process-plant dismantling, removal from site of all equipment and materials, treatment or removal of contaminated soil (if any) and re-profiling of the area to reestablish natural drainage patterns. A soil improvement program will be carried out using stockpiled organic matter and topsoil, prepared organic mulches and fertiliser. Indigenous plants, shrubs and trees will be transplanted from a nursery area.

The dismantling and removal from site of all buildings, sewage systems, workshops, fuel storage facilities, electrical and mechanical equipment and materials will be carried out, unless they can be put towards a sustainable use. The mine drainage sedimentation ponds will be cleaned (if necessary) and backfilled.

All hazardous areas will be signposted and fenced off if necessary and the public informed of the associated dangers of inadvertent mine site access after closure through the company's ongoing public consultation program. Abandonment bund walls to be constructed around the pit to prevent inadvertent access.

20.8.3 Post-closure mine site inspection, environmental monitoring, and reporting

The Project will implement post-closure environmental inspection and monitoring to assess the success of mine reclamation and verify that the various components of the closed mine are not adversely impacting adjacent watercourses and groundwater, and that they do not pose a potential health risk and / or danger to the public. An independent consultant will conduct the site inspection and environmental monitoring.

The Company proposes that post-closure environmental inspection and monitoring be conducted bi-annually for the first two years to establish seasonal variations. Bi-annual site visits will be made in October (before the rains) and in April (at the end of the rains). Final inspection and monitoring will be conducted five years after mine closure. The findings of this inspection will determine whether any further post-closure site inspection is necessary.

20.8.4 Post-closure environmental inspection

Post closure environmental inspections will focus on:

- Pit wall instability with respect to encroachment on abandonment bund walls.
- Erosion on the waste rock dump sidewalls and upper surfaces.
- Erosion at the TSF.
- Success of establishing an indigenous vegetation cover on the TSF, process plant site, ROM pad and ore stockpile areas.
- Any activity by the public or persons unknown that may adversely affect the stability of disused mine structures, pose a danger to the community or result in environmental degradation.
- The condition of site access roads, bridges and culverts.

Consultations will be held with local community leaders to discuss any issues of concern pertaining to the closed mine site.

20.8.5 Post-closure environmental monitoring

Post-closure environmental monitoring will include the following tasks:

- Surface water sampling across the mine site
- Groundwater sampling at the plant area, TSF, and workshop

Surface water samples will be collected for analysis from the following sites:

- Drainage from the former process plant area
- Drainage from the former ROM Pad
- The flooded open pit
- Drainage from the waste rock dump

- Drainage from the TSF
- Drainage from the workshop and former ore stockpile area

The surface water samples will be submitted to an independent accredited laboratory and analysed for the key parameters including pH, EC, TSS, TDS, SO_4, Cu, and Co.

Groundwater samples will be collected from the six monitoring boreholes (piezometer) located at the disused TSF, former process plant and workshop. The groundwater samples will be submitted to an independent accredited laboratory and analysed for the key parameters pH, EC, TDS, SO_4, Cu, and Co.

20.8.6 Post-closure environmental reporting

The Company (using a Government registered Environmental Consultancy) will produce an annual post-closure environmental monitoring report at the end of Years 2 and 4 and a final post-closure environmental report at the start of Year 5. These post-closure environmental reports will present the findings of the mine site inspections / walkovers and the results of the environmental monitoring. Where reclamation activities have not obtained the desired result, the consultant will make recommendations on what additional reclamation work is required to achieve full reclamation. Any areas of concern will be highlighted. The reports will include a post-closure photographic record of mine reclamation.

No significant post-closure environmental issues are anticipated. Environmental inspections and monitoring should cease in Year 5, subject to applicable approvals.

20.8.7 Closure costs

The Company will maintain sufficient financial guarantees to support the cost of activities. Table 20.7 below presents the estimated costs for the closure and rehabilitation of the Project site relevant to the Phase 1 ESIA. Costs are presented for the various components of the closure and rehabilitation plan, with the exception of the TSF closure. The total cost is estimated at US$4.61M. TSF closure costs are accounted for within the capital estimate and financial model set out in Section 21.

Table 20.7 Estimate of closure and rehabilitation costs

Activities	**Cost (CFA)**	**Cost (US$)**
Revegetation of the tailings park	300,000,000	536,700
Revegetation of waste rock piles	280,000,000	500,920
Revegetation of the factory and mining camp	240,000,000	429,360
Upgrading production routes before transfer to the region	320,000,000	572,480
Securing pits (bunds)	400,000,000	715,600
Dismantling and disposal of waste	70,000,000	125,230
Soil characterisation	40,000,000	71,560
Management of hazardous waste and contaminated soil	100,000,000	178,900
Demobilisation of staff (social rehabilitation)	100,000,000	178,900
Information / awareness component	20,000,000	35,780
Training	40,000,000	71,560
Operations monitoring	150,000,000	268,350
Subtotal	2,060,000,000	3,685,340
Engineering (15%)	309,000,000	552,801
Unforeseen (10%)	206,000,000	368,534
Total excl. VAT	**2,575,000,000**	**4,606,675**

Source: ESIA Douta Project, August 2025.

This budget will be reviewed periodically to ensure adequate financial coverage.

20.9 Conclusion and future actions

The environmental and social studies conducted for the Douta Gold Project between 2021 and 2025 constitute a comprehensive baseline and impact assessment consistent with Senegalese law and international standards.

The ESIA received formal approval January 2026, following rigorous review and full public consultation. The assessment shows that:

- Baseline environmental and social conditions are well understood.
- Key risks have been identified.
- Engineering design incorporates appropriate avoidance, mitigation, and monitoring measures.
- The project can operate within acceptable environmental and social parameters.
- Long-term closure planning will return land to stable and productive uses.

The Project's ESMS, supported by thematic management plans and extensive community engagement commitments, provides the framework for responsible and transparent operation. Residual impacts are manageable and offset by significant socio-economic benefits, including employment, investment, and regional development.

The Douta Project can therefore proceed in compliance with regulatory requirements and in alignment with international best practice for responsible mining.

Over the next 12 to 18 months additional activities and surveys to address, assess and confirm environmental and social impacts and mitigation measures will include:

- Phase 2 ESIA for refractory gold processes, mine footprint expansion and amendments to site infrastructure, as well as emissions management, tailings and waste management and mitigating potential impacts on water use and chemical quantities.
- Development of a biodiversity management plan particularly for chimpanzees and linking with ecology protection zone in the adjoining Massawa Mining Licence.
- Technical studies and analysis for potential ARD impacts.
- Undertaking the topic specific management plans under the umbrella of the ESMP.
- Progressing community agreements for socio-economic benefits to local communities including employment, procurement, training and seed funding local businesses.
- Compiling the Economic Displacement Management Plan inclusive of a compensation procedure.
- Incorporating and progressing the environment and social factors above into the future iterations of the Douta Project.

21 Capital and operating costs

The capital cost (CapEx) and operating cost (OpEx) estimates for the Project were prepared by the following parties:

- The mining CapEx and OpEx were estimated by NORINCO and AMC.
- The process plant CapEx and OpEx were estimated by NORINCO.
- The TSF and water dam CapEx and OpEx were estimated by Knight Piésold.
- Other project costs were estimated by NORINCO, AMC, and Thor.
- The owners' costs were estimated by Thor.

All costs in this section are presented in US$.

21.1 CapEx estimate

The initial CapEx cost is estimated at US$253.5M and incurred over an 18-month period. Phase 2 CapEx is estimated at US$60.1M and expected to occur in 2031. Sustaining CapEx is estimated at US$63.2M giving a LOM total CapEx of US$376.8M. The LOM CapEx is summarised in Table 21.1.

Table 21.1 LOM CapEx summary

Category	Phase 1 CapEx (US$M)	Phase 2 CapEx (US$M)	Sustaining LOM CapEx (US$M)	Total LOM CapEx (US$M)
Mining	19.1			19.1
Process plant	177.9	60.1		238.0
TSF and water storage	21.9		53.0	74.9
Other project costs	19.0			19.0
Owners' costs	15.6			15.6
Closure costs			10.2	10.2
Total	**253.5**	**60.1**	**63.2**	**376.8**

Sources: NORINCO, AMC, Knight Piésold, 2025.

CapEx estimates presented in this section reflect total project costs from July 2026 to end of mine life. Mining activities are to be undertaken by a mining contractor with equipment costs contained within the mining operating costs.

Initial CapEx is defined as costs incurred up to start of processing in January 2028.

21.1.1 Mining CapEx

The open-pit mining CapEx was estimated by NORINCO and AMC and totals US$19.3M LOM based on a combination of budgetary quotes and estimates based on recent quotes obtained in West Africa. The mining CapEx is summarised in Table 21.2.

Table 21.2 Open-pit mining CapEx

Category	Total LOM CapEx (US$k)
Pit preparation and pre-production mining	4.6
Explosives	6.5
Mobile crusher	2.2
Dewatering	5.8
Total	**19.1**

Source: NORINCO and AMC, 2025.

Pit preparation costs are derived from current contractor OpEx quotations applied to establishment of mining areas prior to mining along with estimated rates for clearing and stripping topsoil prior to mining.

Dewatering costs include drilling of boreholes and purchase and installation of borehole and in-pit dewatering pumps.

Mine haul-road construction costs included in the mining cost model were re-assigned to Other Project capital costs (Section 21.1.4).

21.1.2 Process plant CapEx

The process equipment initial capital costs were estimated by NORINCO and total US$218.8M. These costs were estimated based on detailed design and costing of mechanical equipment from a combination of budgetary quotes from equipment vendors or estimates from similar projects in West Africa. These costs are split into two phases where Phase 1 is designed to treat oxide material prior to additional CapEx in Phase 2 to help treat Fresh ore. The Phase 1 and 2 capital costs are summarised in Table 21.3.

Table 21.3 Process plant CapEx summary

Category	Phase 1 CapEx (US$M)	Phase 2 CapEx (US$M)	Total CapEx (US$M)
Preliminary and general	40.6	-	40.6
Engineering works	5.7	3.0	8.7
Bulk earthworks	3.4	-	3.4
Civil works	21.4	1.4	22.9
Steel structure	11.8	0.7	12.5
Plateworks	3.7	-	3.7
Mechanical	50.1	50.6	100.7
Pipe & valves	13.7	-	13.7
E,C&I	17.8	1.5	19.3
Commissioning	1.2	-	-1.2
Contingency	8.5	2.9	11.3
Total	**177.9**	**60.1**	**238.0**

Source: NORINCO, 2025.

21.1.3 TSF CapEx

The TSF CapEx is based on the TSF design produced by Knight Piésold which is presented in Section 8. This estimate was based on detailed design quantities and rates for construction from similar projects in Africa. The TSF CapEx is summarised by phase in Table 21.4.

Table 21.4 TSF CapEx by phase

Category	Initial CapEx (US$M)	Lift 1 (Yr1) CapEx (US$M)	Lift 2 (Yr4) CapEx (US$M)	Lift 3 (Yr8) CapEx (US$M)	Total CapEx (US$M)
Water dam	4.8	-	-	0.0	4.8
TSF	17.1	23.0	12.6	17.4	70.2
Total	**21.9**	**23.0**	**12.6**	**17.4**	**74.9**

Source: Knight Piésold, 2025.

The TSF will be raised in four phases using primarily mine waste material.

TSF management sustaining CapEx is included with the process plant operating costs.

21.1.4 Other project costs and Owners' costs

Other projects costs and Owners' costs related CapEx are summarised in Table 21.5 and were provided by NORINCO, AMC, and Thor.

Table 21.5 Other project costs and Owners' costs CapEx

Category	Initial CapEx (US$M)
Haul road construction	1.0
Power supply	0.6
Mine water treatment plant	0.8
Laboratories	1.5
Workshops and facilities	2.4
Airstrip	
Office, admin buildings and camp construction	9.5
First fill	3.3
Ancillary mobile fleet	-
Communications	-
Owners' costs	15.6
Total	**34.6**

Source: AMC, NORINCO, Thor.

21.1.5 Contingency

A contingency of 10% has been applied and included in all capital items above with the exception of the Processing plant and TSF CapEx which have a 5% contingency allowance included.

21.2 OpEx estimate

The LOM OpEx estimates are summarised in Table 21.6.

Table 21.6 Summary of LOM OpEx

Area	Total OpEx (US$M)	OpEx unit cost (US$/oz)	OpEx (US$/t ore processed)
Mining	667	650	18.5
Processing	913	891	25.4
G&A	117	114	3.3
Refining	9	8	0.2
Cash operating cost	**1,706**	**1,664**	**47.4**
Royalties	180	175	5.0
Total Cash Cost	**1,886**	**1,839**	**52.4**
Sustaining capital	53	52	1.5
AISC	**1,939**	**1,890**	**53.9**

Source: Thor, 2025.

21.2.1 Mining OpEx

Mining operating costs were provided by a mining contractor-based spread over the LOM schedule as summarised in Table 21.7. Surface haulage costs were based on contractor rates for Baraka material based on a 35 km haul distance and dewatering costs were estimated pumping requirements for in-pit and ex-pit dewatering using diesel rates of US$1.00/L and electricity rates of US$0.21/kwh.

Table 21.7 Mining OpEx unit costs

Area	OpEx unit cost (US$/t mined)
Ore mining cost - oxide and transitional	4.75
Ore mining cost - fresh	4.90
Waste mining cost - oxide and transitional	2.75
Waste mining cost - fresh	2.90
Surface haulage cost (Baraka)	5.75
Dewatering costs	0.01

Source: AMC, 2025.

21.2.2 Processing OpEx

Processing operating costs were estimated by NORINCO for the different ore types to be treated during Phase 1 and Phase 2 of the operation. These are summarised in Table 21.8. Tailings monitoring and management costs were estimated by Knight Piésold at ~US$880k per year and included in addition to the processing costs equating to an additional US$0.22/t ore processed.

Table 21.8 Processing OpEx unit costs by ore type and area

Area	Processing unit cost (US$/t ore)		
	Oxide	Transitional	Fresh
Makosa (Main, North, East, and Tails)	16.14	16.14	33.35
Baraka (East and West)	16.14	16.14	17.14

Source: AMC, 2025.

21.2.3 General and administration OpEx

The G&A OpEx estimate is based on Thor's experience of operating in similar jurisdictions. G&A costs were estimated at $3.25/t ore.

21.2.4 Refining and transport cost

Refining and transport costs were based on current contracts at Thor's Segilola operations. Royalties of 5% were applied to the gross revenue of gold produced.

22 Economic analysis

22.1 Introduction

An economic evaluation of the Project has been completed using a detailed cash-flow model. The model is based on annual cash flows and incorporates processed tonnages and grades for the CIL feed, metallurgical recoveries, metal prices, operating costs, refining charges, royalties, and both initial and sustaining capital expenditures. Gold revenues are calculated using a payable factor of 99.90%. The analysis applies a base gold price of US$3,500 per ounce.

The Project has been assessed on a "100% equity" basis, with all debt and equity financing considerations excluded. Inflation has not been factored into the assessment. Discounting and IRR calculations commence at the start of construction, using a 5% discount rate.

The Company notes that the Mining Convention for the Project is yet to be negotiated with the Government of Senegal. As a result, it is not currently possible to incorporate the expected fiscal incentives and tax exonerations that are typically granted under such agreements into the calculation of the post-tax NPV. For the purposes of this PFS, we have therefore applied a standard approach whereby exploration expenditure and project development capital are accumulated into a tax loss pool, against which future taxable income is fully offset, with the Senegalese statutory corporate tax rate of 30% applied thereafter. The Company anticipates an improved post-tax economic outcome once the Mining Convention is finalised, as such agreements customarily provide additional tax incentives that enhance project value.

The Company also notes that Senegalese mining legislation provides for a 10% State free-carried interest, which will be formally awarded to the State upon finalisation of the Mining Convention. This interest has not been incorporated into the current economic analysis, which is presented on a "100% equity" basis and will be reflected in future evaluations once the Mining Convention has been agreed.

Input data was provided from a variety of sources, including the various consultants' contributions to this Technical Report and pricing obtained from external suppliers and contractors. The assessment was based upon:

- Capital cost estimates were prepared by a variety of consultants including NORINCO, AMC, and Knight Piésold as summarised in Section 21.
- The mining and processing schedule was developed by AMC for the Technical Report.
- Mining costs were provided from contractor estimates as well as AMC for dewatering and grade control.
- Processing operating costs were estimated by NORINCO.
- G&A operating costs were estimated by Thor.
- Royalty and discount rates were provided by Thor.
- Gold price assumptions were provided by Thor. Gold price is assumed at US$3,500/oz for the LOM.
- Metallurgical performance.
- Closure cost estimates were provided by Knight Piésold for the tailings facilities and by Thor for other closure costs.
- The cashflow analysis excludes any effects due to inflation and all dollars are expressed in real US$ as of 1 January 2026.
- No financing or free carry assumptions are included in this economic analysis.
- A discount rate of 5% was used in this analysis.

The analysis has been split into two stages to focus on a Phase 1 only analysis along with the 2-phase approach.

The Project is expected to have a LOM ranging from just over four years for the oxide ore phase only (Phase 1) and a total of 13 years when oxide & primary ore (Phase 1 and 2) is included. A summary of the mining physicals and economics for each phase is shown in Table 22.1. Phase 1 delivers an NPV[5] of US$449M from investment capital of US$254M. Additional capital for Phase 2 of $60M is required to process the fresh material, extending the mine life to 13 years and delivering an NPV[5] of US$908M. The IRR for each phase is 72% and 73% respectively. The payback period for Phase 1 is within 12 months of the start of mining.

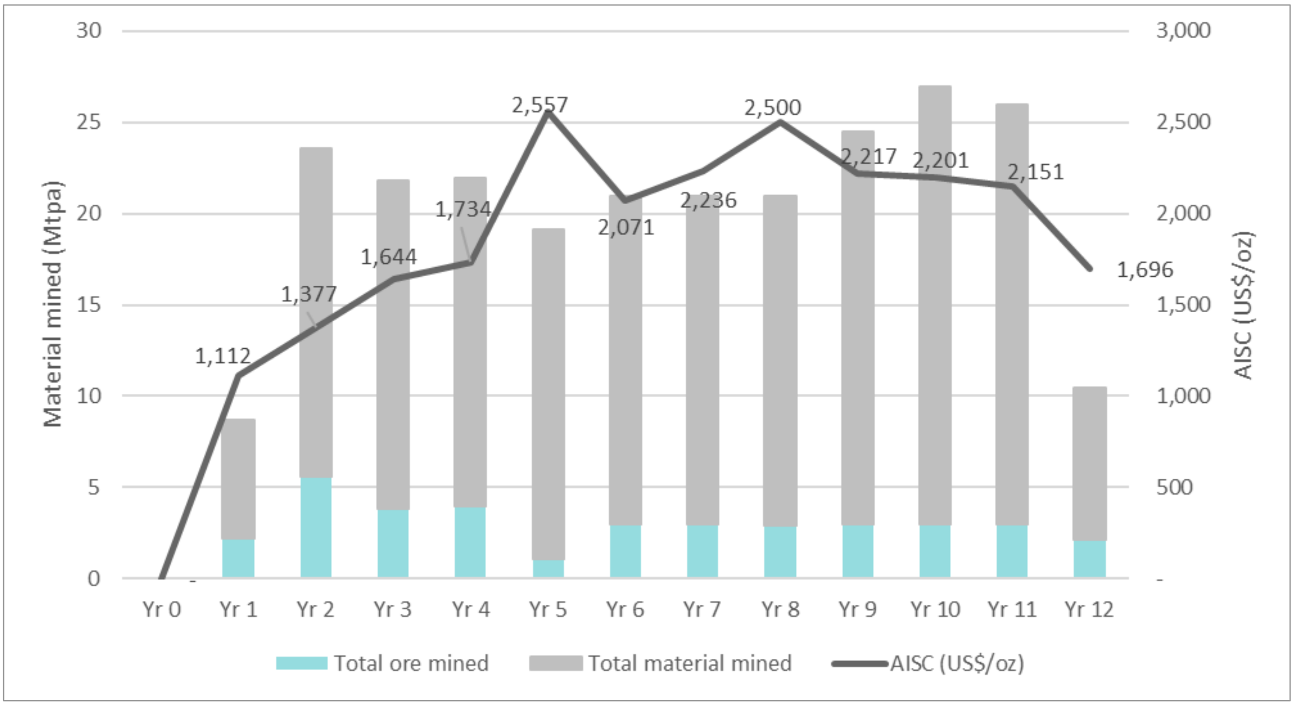
Table 22.1 Mining and processing physicals summary used in economic analysis

	Units	Oxide Ore Phase only	Oxide & Primary Ore Phases
Production			
Total ore mined	Mt	15.9	36.6
Total material mined	Mt	62.6	209.3
Strip ratio	x:x	2.9	4.7
Total ore processed	Mt	15.7	36.0
Head grade	g/t	0.9	1.0
Recovery	%	87.9%	84.7%
Gold recovered	koz	411	1,025.7
Production costs			
Mining cost	$/t mined	3.3	3.2
Cash operating cost	$/t ore processed	33.0	47.4
Cash operating cost	$/oz	1,262	1,664
AISC	$/oz	1,493	1,890
Capital costs			
Phase 1 capital	$M	253.5	253.5
Phase 2 capital	$M	-	60.1
Sustaining capital	$M	23.0	53.0
Closure costs	$M	10.0	10.2
Financial			
Gross Revenue	$M	1,437	3,590
Pre-tax NPV5	$M	449	908
Pre-tax IRR	%	72%	73%
Post-tax NPV5	$m	321	633
Post tax IRR	%	60%	61%

Note: A small amount of low-grade ore will be processed in Q1 of year 14, which has been excluded from the Economic Analysis.
Source: AMC, 2025.

Phase 1 and Phase 2 all-in sustaining costs (AISC) are shown against the mining production physicals in Figure 22.1. This shows early AISC during Phase 1 ranging from US$1,112/oz up to US$1,734/oz before increased processing costs for fresh material along with increased strip ratios increase the AISC to around US$2,000-2,500/oz during Phase 2.

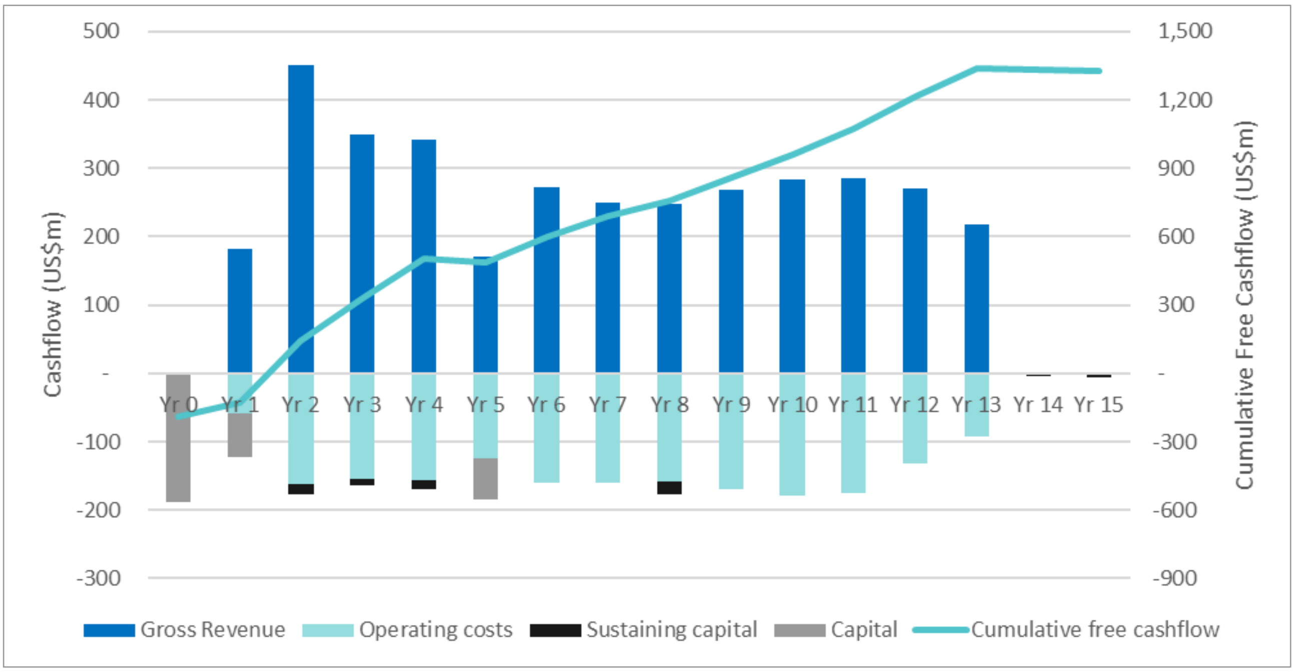
Figure 22.1 Mining physicals for Phases 1 and 2 against AISC



Source: AMC, 2025.

Cashflows for Phases 1 and 2 are shown in Figure 22.2. These show early payback within 12 months of the start of mining.

Figure 22.2 Cashflows for Phases 1 and 2



Source: AMC, 2025.

The breakdown of operating and sustaining costs for each phase are shown in Table 22.2 with the detailed cashflows for Phase 1 only and Phases 1 and 2 shown in Table 22.3 and Table 22.4.

The QP has reviewed the costs and other inputs to the economic model and believes that they are suitable for this level of study. It is also the opinion of the QP that the economic model is fit for purpose and addresses the requirements of this report.

Table 22.2 Cash costs and AISC for Phases 1 and 2

Description	Phase 1 only			Phase 1 and 2		
	Total (US$M)	Cost ($/oz)	Cost ($/t ore)	Total (US$M)	Cost ($/oz)	Cost ($/t ore)
Mining	205	498	13.0	667	650	18.5
Processing	259	632	16.5	913	891	25.4
G&A	51	124	3.3	117	114	3.3
Refining	3	7	0.2	9	8	0.2
Cash operating cost	**518**	**1,262**	**33.0**	**1,706**	**1,664**	**47.4**
Royalties	72	175	4.6	180	175	5.0
Total Cash Cost	**590**	**1,437**	**37.6**	**1,886**	**1,839**	**52.4**
Sustaining capital	23	56	1.5	53	52	1.5
Corporate G&A	-	-	-	-	-	-
AISC	**613**	**1,493**	**39.1**	**1,939**	**1,890**	**53.9**

Source: AMC, 2025.

Table 22.3 Phase 1 pre-tax cashflow

	Units	Total	Yr 0	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15
Physicals																		
Total ore mined	Mt	15.9	-	2.2	5.6	3.8	3.9	0.3	-	-	-	-	-	-	-	-	-	-
Total material mined	Mt	62.6	-	6.5	18.0	18.0	17.5	2.7	-	-	-	-	-	-	-	-	-	-
Strip ratio	x:x	2.9	-	2.0	2.2	3.7	3.4	6.8	-	-	-	-	-	-	-	-	-	-
Total ore processed	Mt	15.7	-	1.5	4.0	4.0	4.0	2.2	-	-	-	-	-	-	-	-	-	-
Head grade	g/t	0.9	-	1.16	1.13	0.86	0.86	0.54	-	-	-	-	-	-	-	-	-	-
Recovery	%	88%	0%	92%	88%	90%	87%	77%	-	-	-	-	-	-	-	-	-	-
Gold recovered	koz	411	-	52	129	100	98	33	-	-	-	-	-	-	-	-	-	-
Cashflow summary																		
Gross revenue	**US$M**	**1,437**	**-**	**182**	**450**	**349**	**342**	**114**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Royalties	US$M	(72)	-	(9)	(23)	(17)	(17)	(6)	-	-	-	-	-	-	-	-	-	-
Cash operating costs	**US$M**	**(518)**	**-**	**(49)**	**(140)**	**(138)**	**(138)**	**(53)**	**-**	**-**	-	-	-	-	-	-	-	-
Mining	US$M	(205)	-	(18)	(61)	(58)	(58)	(9)	-	-	-	-	-	-	-	-	-	-
Processing	US$M	(259)	-	(25)	(66)	(66)	(66)	(36)	-	-	-	-	-	-	-	-	-	-
G&A	US$M	(51)	-	(5)	(13)	(13)	(13)	(7)	-	-	-	-	-	-	-	-	-	-
Refining	US$M	(3)	-	(0)	(1)	(1)	(1)	(0)	-	-	-	-	-	-	-	-	-	-
Operating EBITDA	**US$M**	**847**	**-**	**124**	**287**	**194**	**187**	**55**	**-**	**-**	-	-	-	-	-	-	-	-
Sustaining capital	US$M	(23)	-	-	(14)	(9)	-	-	-	-	-	-	-	-	-	-	-	-
Mine cashflows	**US$M**	**824**	-	**124**	**273**	**185**	**187**	**55**	**-**	**-**	-	-	-	-	-	-	-	-
Capital costs	US$M	(253)	(189)	(64)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Closure costs	US$M	(10)	-	-	-	-	-	(2)	(5)	(4)	-	-	-	-	-	-	-	-
Free cashflow	**US$M**	**561**	**(189)**	**60**	**273**	**185**	**187**	**54**	**(5)**	**(4)**	-	-	-	-	-	-	-	-

Source: AMC, 2025.

Table 22.4 Phases 1 and 2 pre-tax cashflow

	Units	Total	Yr 0	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15
Physicals																		
Total ore mined	Mt	36.6	-	2.2	5.6	3.8	3.9	1.1	3.0	2.9	2.9	3.0	3.0	3.0	2.1	-	-	-
Total material mined	Mt	209.3	-	6.5	18.0	18.0	18.0	18.0	18.0	18.0	18.0	21.5	24.0	23.0	8.3	-	-	-
Strip ratio	x:x	4.7	-	2.0	2.2	3.7	3.6	15.4	5.0	5.1	5.1	6.2	7.1	6.7	2.9	-	-	-
Total ore processed	Mt	36.0	-	1.5	4.0	4.0	4.0	2.7	2.4	2.4	2.4	2.4	2.4	2.4	2.4	3.0	-	-
Head grade	g/t	1.0	-	1.16	1.13	0.86	0.86	0.66	1.24	1.14	1.13	1.21	1.24	1.23	1.20	0.78	-	-
Recovery	%	85%	0%	92%	88%	90%	87%	78%	81%	81%	81%	82%	85%	86%	84%	83%	0%	0%
Gold recovered	koz	1,026	-	52	129	100	98	49	78	72	71	77	81	82	77	62	-	-
Cashflow summary																		
Gross revenue	**US$M**	**3,590**	**-**	**182**	**450**	**349**	**342**	**170**	**272**	**250**	**247**	**268**	**284**	**286**	**271**	**218**	**-**	**-**
Royalties	US$M	(180)	-	(9)	(23)	(17)	(17)	(9)	(14)	(13)	(12)	(13)	(14)	(14)	(14)	(11)	-	-
Cash operating costs	**US$M**	**(1,706)**	**-**	**(49)**	**(140)**	**(138)**	**(140)**	**(116)**	**(147)**	**(147)**	**(147)**	**(156)**	**(165)**	**(162)**	**(118)**	**(82)**	**-**	**-**
Mining	US$M	(667)	-	(18)	(61)	(58)	(60)	(54)	(58)	(58)	(57)	(67)	(75)	(72)	(28)	-	-	-
Processing	US$M	(913)	-	(25)	(66)	(66)	(66)	(53)	(81)	(81)	(81)	(81)	(81)	(81)	(81)	(72)	-	-
G&A	US$M	(117)	-	(5)	(13)	(13)	(13)	(9)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(10)	-	-
Refining	US$M	(9)	-	(0)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	-	-
Operating EBITDA	**US$M**	**1,704**	**-**	**124**	**287**	**194**	**185**	**46**	**111**	**90**	**88**	**98**	**106**	**110**	**140**	**124**	**-**	**-**
Sustaining capital	US$M	(53)	-	-	(14)	(9)	(13)	-	-	-	(17)	-	-	-	-	-	-	-
Mine cashflows	**US$M**	**1,651**	-	**124**	**273**	**185**	**172**	**46**	**111**	**90**	**71**	**98**	106	**110**	140	**124**	**-**	**-**
Capital costs	US$M	(314)	(189)	(64)	-	-	-	(60)	-	-	-	-	-	-	-	-	-	-
Closure costs	US$M	(10)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(4)	(6)
Free cashflow	**US$M**	**1,327**	**(189)**	**60**	**273**	**185**	**172**	**(14)**	**111**	**90**	**71**	**98**	**106**	**110**	**140**	**124**	**(4)**	**(6)**

Source: AMC, 2025.

22.2 Sensitivity analysis

The project value was assessed by undertaking sensitivity analyses on the gold price, operating costs, and capital costs. This shows that the project is most sensitive to gold price and then operating costs. The results of sensitivity analyses are presented below for pre-tax results in Table 22.5 to Table 22.7 and for post-tax results in Table 22.8 to Table 22.10.

Table 22.5 Pre-tax NPV gold price sensitivity

Gold price (US$/oz)	Discount multiple US$M		
	0.00%	**5.00%**	**10.00%**
3,000	840	561	379
3,500	1,327	908	634
4,000	1,815	1,255	888
4,500	2,302	1,603	1,143

Source: AMC, 2025.

Table 22.6 Pre-tax NPV CapEx sensitivity

CapEx Flex	Discount multiple US$M		
	0.00%	**5.00%**	**10.00%**
20%	1,252	841	572
10%	1,290	875	603
0%	1,327	908	634
-10%	1,365	942	664
-20%	1,403	976	695

Source: AMC, 2025.

Table 22.7 Pre-tax NPV OpEx sensitivity

OpEx Flex	Discount multiple US$M		
	0.00%	**5.00%**	**10.00%**
20%	988	673	466
10%	1,158	791	550
0%	1,327	908	634
-10%	1,497	1,026	718
-20%	1,667	1,144	802

Source: AMC, 2025.

Table 22.8 Post-tax NPV gold price sensitivity

Gold price (US$/oz)	Discount multiple US$M		
	0.00%	**5.00%**	**10.00%**
3,000	584	387	257
3,500	928	633	438
4,000	1,271	878	618
4,500	1,615	1,123	798

Source: AMC, 2025.

Table 22.9 Post-tax NPV CapEx sensitivity

OpEx Flex	Discount multiple US$M		
	0.00%	**5.00%**	**10.00%**
20%	871	581	390
10%	899	607	414
0%	928	633	438
-10%	956	658	462
-20%	985	684	485

Source: AMC, 2025.

Table 22.10 Post-tax NPV OpEx sensitivity

OpEx Flex	Discount multiple US$M		
	0.00%	**5.00%**	**10.00%**
20%	687	465	318
10%	808	549	378
0%	928	633	438
-10%	1,048	716	498
-20%	1,168	800	557

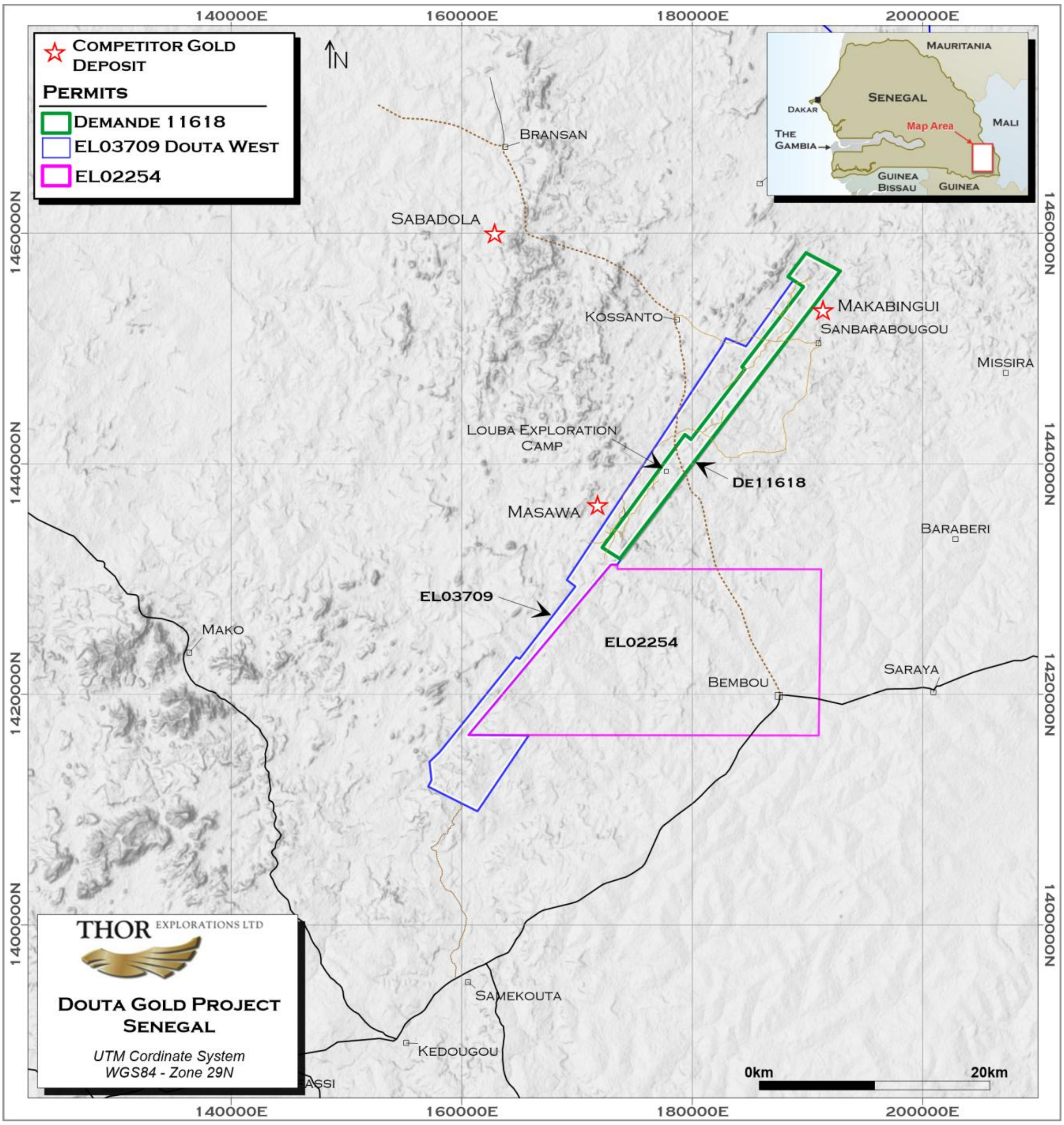
Source: AMC, 2025.

23 Adjacent properties

23.1 Massawa Gold Deposit

The Massawa Gold Mine, which is located approximately 5 km to the west of Makosa, forms part of the Sabodala-Massawa Mine which is owned by Endeavour Mining (Figure 23.1). The Sabodala-Massawa Mineral Resource comprises 80.4 Mt at 2.01 g/t Au for a contained 5,186 koz Au in the Measured and Indicated category together with 20.4 Mt grading 2.01 g/t Au for 1,322 koz Au in the Inferred category (Endeavour Mining, 2025).

Figure 23.1 Project location map showing competitor projects



Source: Thor, 2025.

Within these resources is a combined Proven and Probable reserve of 50.7 Mt grading 2.00 g/t Au for 3,260 koz Au has been reported (Endeavour Mining, 2025).

Regionally, Massawa is located on the over 150 km long NE trending MTSZ, which is a significant trans-crustal dislocation between the Mako Belt (basaltic flow rocks, minor intercalated volcaniclastics, and ultramafic sub-volcanic intrusions) and the Dialé-Dalema Basin (volcano-sedimentary to sedimentary rocks) within the Paleoproterozoic (Birimian) Kedougou-Kenieba inlier. Mineralisation is present within various lithologies but is structurally controlled within anastomosing shears which converge to the north.

The QP has been unable to verify the information and that the information is not necessarily indicative of mineralisation on the property that is subject of the Technical Report.

23.2 Makabingui Gold Deposit

The Makabingui exploration project, owned by Bassari, is located approximately 25 km NE of Massawa (Figure 23.1).

The Makabingui Mineral Resource comprises 2.6 Mt at 4.0 g/t Au for a contained 336,000 oz Au in the Indicated category together with 9.3 Mt grading 2.2 g/t Au for 669,000 oz Au in the Inferred category (Bassari, 2025). Within these resources a Probable Ore Reserve of 860,000 t at 5.7 g/t Au for 158 koz Au has been declared (Bassari, 2025).

Regionally, Makabingui is located in the Dialé-Dalema sedimentary basin to the east of the MTSZ which hosts Massawa. The deposit is hosted in gabbro in a pressure shadow along the southern margin of the Sambarabougou Granite. Exploration is also focused on a NE trending structural zone termed the Lafia Shear Zone which is situated both to the NE and SW of Makabingui.

The QP has been unable to verify the information and that the information is not necessarily indicative of mineralisation on the property that is subject of the Technical Report.

24 Other relevant data and information

24.1 Exploration opportunities

The Douta Project hosts a large but underexplored regional-scale gold system with strong potential to expand Mineral Resources. Existing mineralisation, prospects, and targets remain open along strike and at depth within major structural corridors and the extensive area between them.

The Project comprises permits covering approximately 538 km^2 within the highly prospective Birimian Dialé metasediments. Key deposits are located along the MTSZ (hosting the Makosa Tail, Makosa, Makosa North, and Makosa East deposits). Approximately 30 of the MTSZ lie within the Project area.

Additional mineralisation potential exists within and adjacent to these structures, where early-stage exploration has identified numerous prospects along secondary and tertiary structural zones.

Extensive datasets support ongoing target generation, including drilling, soil, and termite mound sampling. Aeromagnetic and electromagnetic surveys are planned to generate further exploration targets. Beyond the defined Mineral Resources, there are numerous prospects that have yet to be drill tested. A phased, property-wide exploration program is ongoing, with data review, target evaluation, and drill prioritisation.

There is a succession of targets in the "pipeline”, and it will be important to continue to rank and upgrade these. There is potential to delineate additional mineralisation with the current exploration program.

25 Interpretation and conclusions

25.1 General

This study has demonstrated that the Douta Project, as described in this report, is viable from both technical and economic aspects.

The Project yields 1,026 koz recovered ounces from a total mill feed tonnage of 36.6 million tonnes at a gold grade of 1.03 g/t. The project schedule is divided into two Phases. Phase 1 treats the oxide / transitional mill feed through a conventional CIL process. Phase 2 treats primary fresh mill feed through a whole ore suspension roasting circuit prior to the CIL circuit.

25.2 Geology

The Douta Project hosts a large but underexplored regional-scale gold system. Existing mineralisation, prospects, and targets remain open along strike and at depth within major structural corridors and the extensive area between them.

Eighteen percent of the Mineral Resources tonnes and 16% of the contained ounces are currently in the Inferred category. Additional drilling is reasonably expected to upgrade these resources.

The Mineral Resources for the Douta project were undertaken to adhere to CIM standards. The resource models were based on extensive drilling within the project boundaries and show good continuity. The gold price of $4000/oz has been used, which is below current trading prices. The QP is of the opinion that the Mineral Resources are up to the standard of a PFS level of study.

25.3 Mining

The extraction of the Mineral Reserve at Douta has been shown to be technically and economically feasible through conventional open pit mining methods. Pit optimisations show that there is economically viable material at Makosa, Makosa Tail, Baraka 3 East, and Baraka 3 West. Pit designs and mine scheduling confirm that the material can be safely and practically extracted using the assumed mining methods.

A total of 36.6 million tonnes of crusher feed at a gold grade of 1.03 g/t is contained within 20 Phase 1 pits and 12 Phase 2 pits. Makosa Main contributes 78% of the mill feed and 67% of the total material movement. Makosa Tail contributes 20% of the mill feed and 26% of the total movement with the remainder coming from the Baraka 3 pits.

A combined mining and processing schedule demonstrates that a processing rate of 4 Mtpa can be sustained for four years for Phase 1 and a rate of 2.4 Mtpa can be sustained for seven years. With a project ramp up, transition from Phase 1 to Phase 2, and treatment of oxide and transitional Phase 2 material at the end of the project, a project life of over 13 years is attained.

There is scope for larger pits with increased Indicated Resources, improved geotechnical or financial conditions, and in the case of Baraka 3, a change or addition to the southern lease boundary.

25.4 Mineral processing and recovery methods

The orebody is comprised of three broad ore types representing oxide, transitional and sulphidic ores. The oxide ores are amenable to whole ore cyanide leaching. Transitional ores exhibit lower recoveries for cyanide leaching and the sulphidic ores exhibit poor whole ore cyanide leach recoveries. Alternate recovery processes have been investigated to improve gold recovery in the less oxidised and sulphidic ores.

After various test work was conducted, it was determined that sulphidic ores are amenable to suspension roasting and CIL for gold recovery. The suspension roasting process will expose refractory gold particles prior to cyanide leaching.

To accommodate this approach, the processing circuit will be constructed in two phases as follows:

- Phase 1 will treat oxide and transitional ores via a conventional CIL circuit.
- Phase 2 will treat fresh ore via suspension roasting. The roasted product will be reground and treated with the CIL circuit to recover the gold.

While suspension roasting of gold ores is relatively new, it is underpinned by established technologies used to upgrade iron ores. The test work conducted to date demonstrates favourable results when this process is applied to the sulphidic gold ore found at the Douta project.

25.5 Major infrastructure

25.5.1 Power supply

Given the lack of coverage by Senegal's national grid, which is limited to the northern and western regions of the country, self-generation of power is the most practical option with the least risk associated with it.

The installation of a local 32 MW HFO / Diesel power plant will provide sufficient electricity for the operation. Coupled with HFO / Diesel storage for a 20-day operation buffer, this configuration provides a cost effective and low risk option for project power supply.

25.5.2 Tailings Storage Facility

A single TSF comprised of several lifts will be constructed on site. This conventional slurry tailings facility is capable of storing the tailings from both Phase 1 and Phase 2. The construction in stages will improve overall economics by deferring construction costs to later stages of the mine life while the fully lined base will reduce environmental risks of the structure.

The location chosen for the TSF makes good use of natural topography and is close to the currently proposed location of the processing plant, further controlling operating costs. Further tailings testing and site-specific climate monitoring will assist in refining the design of the facility.

25.5.3 Water supply

Given the lack of a municipal water supply, or any relevant national infrastructure, a WSD will be constructed within the mining licences area to supply all raw, process, potable, and service water requirements. The dam will allow the project to harvest and store water during the higher rainfall period from June to September and allow for the storage of sufficient quantities of water to meet the annual site-wide requirements when coupled with reclamation of process water from the TSF.

25.6 Environmental

An ESIA covering Phase 1 of the Project was submitted to the Senegalese government, and approval of the ESIA was received on 16 January 2026. Phase 2 will be addressed in a future amendment following completion of additional metallurgical test work. The project layout assessed in the ESIA accommodates both development stages and includes open pits, waste rock dumps, a tailings storage facility, water storage dam, processing plant, power supply, accommodation camp, workshops, chemical storage, and water and wastewater treatment systems.

Despite there being some environmental considerations within the Project area, such as several nationally protected plant species and chimpanzees present in the southern permit area, the region is used to mining activities and the ESMP is expected to adequately address these matters.

Stakeholder engagement has identified strong community support linked to employment and infrastructure benefits, alongside concerns related to land access. No physical resettlement is anticipated, and economic displacement will be managed through a Resettlement Action Plan and community development initiatives focussed on local employment, water access, health, education and support for local enterprises.

The ESMP has been developed to manage impacts on land, water, air, biodiversity, and communities, and includes provisions for monitoring, rehabilitation, waste management, and community engagement. By linking with an adjoining mine's environment protection area, mining is considered feasible as outlined in this PFS report, particularly with appropriate mitigation and biodiversity management measures in place.

25.7 Economic analysis

The project is expected to have a LOM ranging from just over four years for oxide ore phase only (Phase 1) and a total of over 13 years when oxide & primary ore (Phase 1 and 2) is included. A summary of the mining physicals and economics for each phase is shown in Table 1.11. Phase 1 delivers an NPV[5] of US$449M from investment capital of US$254M. Additional capital for Phase 2 of US$60M is required to process the fresh material, extending the mine life to over 13 years and delivering an NPV[5] of US$908M. The IRR for each phase is 72% and 73% respectively. The payback period for Phase 1 is within 12 months of the start of mining.

A Mining Convention for the Project is yet to be negotiated with the Government of Senegal. It is expected that the post-tax economic outcome will be improved once the Mining Convention is finalised, providing additional upside to the economic evaluation.

25.8 Risks and opportunities

25.8.1 Risks

During the course of this study, a number of risks were identified:

- Mineralisation and grade variability may exist due to "nuggety" grade distributions at the local scale. This could impact the production grades when compared with the Mineral Resource block model. An infill resource definition drilling campaign and grade control programs are recommended to counter this risk.
- Bulk density calculations have been completed on a relatively small data set. Incorrect bulk density calculations may lead to actual mined tonnes being less than planned. Additional bulk density determinations are required in future drill campaigns.
- There is no visible indication of ore / waste contacts that can be used during grade control and mining. Ore mark outs will be based entirely on grade control assays. A robust and accurate grade control and ore mark out procedure will be required.
- The absence of any geotechnical or hydrogeological studies for the Baraka deposits increase the risk of incorrect wall angles being used for the optimisations and pit designs. These studies need to be completed to safely allow the extraction and future expansion of these pits.
- Dewatering costs for the entire project are based on limited hydrogeological information. Additional information is required for all deposits to allow for more accurate design and costing of dewatering programs.

- The pool of skilled labour in the area may be limited, particularly with the neighbouring mine already being in operation. Expatriate operators and trainers will be required, at least initially.
- Metallurgical variability is poorly understood for some regions and ore types. This could produce inconsistent and unexpected recovery fluctuations during processing. Additional metallurgical variability studies are required.
- No stability analysis results were provided for slope angles in the saprock or weathered rock to support the recommended slope design parameters.
- A number of specific management plans required under the umbrella of the ESMP are absent. These should be developed and implemented.
- A structural model should be developed for use in pit design studies.

A formal Risk Response Plan will be developed as part of the next stage of work.

25.8.2 Opportunities

The following opportunities were identified during the study:

- With approximately 30 exploration targets within the exploration leases, there is potential for additional mineralisation to be discovered during the life of the operation.
- Inter-ramp and overall slope angles of the pits could be steepened with additional geotechnical study, particularly for oxide and transitional material, and at Baraka 3.
- Improved selective mining unit / bench height studies could assist with dilution control.
- Alternative energy sources could be investigated including waste to liquid plants and solar arrays. A trade-off between capital cost and operating cost savings should be carried out.
- Alternate configurations should be investigated to allow for both Phase 1 and Phase 2 mill feed to be processed in parallel. This will allow for the CIL circuit to continue to operate at a higher capacity once Phase 2 mill feed is introduced and remove the need to stockpile approximately 2.3 million tonnes of Phase 1 material at the end of the mine life.

26 Recommendations

26.1 Geotechnical

26.1.1 Geotechnical review

There does not appear to be a structural model or fault wireframes available, although structural data from recent diamond drilling was provided for review. It is recommended that a three-dimensional structural model be constructed using fault and shear structures collected from the diamond drilling and verified with mapping activities on implementation.

While a number of samples have been sent to the laboratory for tests including classification, triaxial strength testing, tensile testing, and direct shear testing of discontinuities, additional testing is required to gain the target level of confidence. It is recommended that additional uniaxial compressive strength (UCS), triaxial strength, and tensile strength tests should be completed on the rock types where limited data is available. A total number of seven valid UCS, triaxial, and tensile strength tests should be targeted for each rock type and weathering domain, including the existing tests.

Although some direct shear testing has been conducted as part of the geotechnical investigation program, the level of testing is insufficient to fully characterise the behaviour of the rock mass or the variation in defect strength parameters across the different rock types at the site. It is recommended that additional direct shear testing is carried out in future data collection programs on bedding, joint and shear structures in each rock type.

While the Rock Mass Classification systems employed at Douta are well recognised classification systems, it is recommended that additional geotechnical drilling is undertaken so as to provide a more comprehensive understanding of the ground conditions and rock mass properties at all locations within the Project area, particularly Baraka 3, where no geotechnical information has been collected to date.

26.1.2 Slope design parameters

The stability of the footwall is significantly affected by kinematic instability due to the foliation dip of approximately 65°, which limits both the allowable bench angle and overall slope. In Makosa Tail, the foliation dip is shallower, averaging 48°, which requires separate design considerations for this zone compared to Makosa.

There were no limit equilibrium stability analysis results provided in the MINENET report (2023) for the final pit design with a saprolite bench angle of 60°. It appears these design parameters have used assumptions based on comparable sites with similar conditions and the implementation of systematic dewatering measures. Additionally, no stability analysis results were provided for slope angles in the saprock or weathered rock to support the recommended slope design parameters.

Kinematic analyses of the rock slopes to assess potential structural failure were limited to wedge sliding failure analysis and were conducted for a 60° slope angle in the hangingwall and a 50° slope angle in the footwall only based on limited defect strength parameters.

These gaps highlight the need for additional geotechnical analyses in future studies to validate the proposed slope designs and ensure their stability under site-specific conditions.

26.2 Hydrogeological

As part of the geotechnical review, the hydrogeological information was examined. Hydrogeological information at the site is limited. As a result, the potential influence of groundwater on the slope stability of the future pits was assessed by experience gained from adjacent properties. To improve the confidence in the hydrogeology, the following items are required:

- Determine hydraulic properties through in situ testing which can be done in conjunction with the geotechnical drilling program.
- Create a hydrogeological database to track monitoring and testing data.
- Generate a groundwater model, producing groundwater profiles for the latest pit slope designs.

As with the geotechnical information, hydrogeological investigations at Baraka 3 have not been undertaken. These will need to be completed before further development of the Baraka 3 assets is undertaken.

26.3 Mining

- The Mineral Resource has been conducted at US$4,000/oz while the Mineral Reserve was conducted at a gold price of US$3,000/oz. Given the unprecedented increase in gold price over the last 12 months, it is recommended that the pit optimisations are rerun at a higher gold price prior to the commencement of mining operations.
- Additional sensitivity studies to assess the impact on the Mineral Reserve and open pit limits at various gold prices would allow Thor to ensure there is no restriction on pit expansion due to infrastructure locations. These should be run with Inferred material both considered and excluded, to allow the development of an understanding of the effects on the pit dimensions of converting Inferred material to Indicated material.
- The current assumption is that mining will be undertaken by a contractor. As the mine has a current operating life of over 13 years, the first 7 of which are consistent in total material mined, Thor should consider conducting a trade-off study between contract mining and owner mining. This suggestion is further supported by observations from neighbouring operations which are moving away from contract mining operations.
- A more detailed dilution and bench height study should be conducted, particularly for the Baraka 3 deposits, where the extremely narrow lodes result in very large dilution numbers after regularisation. This study would lead to an equipment selection study based on the ultimate SMU size chosen.
- Mine scheduling should be moved from Microsoft Excel to a mine scheduling package such as MineMax or Deswik.Sched. This will allow for more scheduling scenarios to be analysed quickly and will also make haulage and dump construction analysis much easier.
- Optimising waste dump construction is a proven method of reducing operating costs and assisting in progressive rehabilitation of waste dumps. Conducting a waste dump optimisation study as part of a larger mine scheduling exercise would be beneficial.

26.4 Metallurgical

- The variability in organic carbon content, sulphide content and silicates in the orebody and by ore types has not been quantified. Test work results demonstrate that variations in both occur in transition ores and fresh ores. Characterisation of the variability of carbon, sulphur, and silicates across the orebodies is recommended.
- Additional metallurgical variability test work is required to expand the knowledge on recovery performance for both transition and fresh ore types. The recovery is impacted by gold distribution in arsenopyrite and silicates, and by the level or organic carbon.

- It is recommended that a variability suite of selected transition ore samples is characterised for their sulphides, organic carbon and silicate content and tested for recovery by CIL.
- It is recommended that a variability suite of selected fresh ore samples is characterised for their sulphides, organic carbon and silicate contents and tested for recovery by suspension roasting and CIL.
- The selection of an expanded testing sample set including increased variability of S, As, and C contents is recommended for future test programs.
- Further study is recommended to refine the understanding of the best process options for each type. The observation of differences by ore location should also be further investigated as an input to the mining plan and process schedule.

26.5 Processing

Currently the Phase 2 suspension roasters and associated equipment are inserted into the flowsheet after the hydrocyclone cluster and before the leach tanks. This flowsheet design prevents Phase 1 ore from being conventionally treated once Phase 2 is commissioned. It will also mean a longer shutdown is required as the flowsheet is modified. Alternate configurations should be investigated to allow for both Phase 1 and Phase 2 mill feed to be processed in parallel. This will allow for the CIL circuit to continue to operate at a higher capacity once Phase 2 mill feed is introduced and remove the need to stockpile approximately 2.3 million tonnes of Phase 1 material at the end of the mine life. These changes will have the effect of bringing ounce production forward in the mine life, improving the overall economics.

Two possible options are:

- Adding a second comminution circuit.
- Reviewing dry grinding technologies as direct feed to the roasting plant.

26.6 Infrastructure

26.6.1 Processing plant

The location of the processing plant has been determined based on the assumption that Makosa and Makosa Tail are the dominant sources of mill feed. Prior to the construction of the processing plant, Thor should consider additional drilling that has been undertaken since this report that may suggest larger or more numerous deposits to the south and east.

26.6.2 Tailings Storage Facility

- Prior to the final design and construction of the TSF, the following ground investigations and laboratory tests should be conducted:
  - Drilling and core logging should be undertaken at the main embankment locations to log the foundation and collect samples.
  - Standard Penetration Tests should be undertaken on the embankment foundation locations.
  - Extensive trial pitting should be undertaken across the extent of the TSF impoundment to identify potential borrow zones and foundation geometry.
  - Geotechnical laboratory testing of construction materials should be undertaken to determine material engineering parameters (foundation indicator test work, triaxial testing, permeability, compaction, etc.).

- A hydrogeological assessment should also be undertaken, which would include:
  - Installation of boreholes upstream and downstream of the TSF (baseline water quality and for monitoring water quality).
  - Permeability testing within boreholes; map groundwater levels and gradients seasonally.
  - Development of a seepage model.
- Tailings rheological testing should be undertaken to inform stability analysis and dam breach assessments.
- A probabilistic seismic hazard assessment should be conducted to inform stability analysis.

26.6.3 Water storage dam

- Prior to the final design and construction of the WSD, the following ground investigations and laboratory tests should be conducted:
  - Drilling and core logging should be undertaken at the embankment location to log the foundation and collect samples.
  - Standard Penetration tests should be undertaken on the embankment foundation locations.
  - Extensive trial pitting should be undertaken across the extent of the WSD impoundment to identify potential borrow zones and foundation geometry.
  - Geotechnical laboratory testing of construction materials should be undertaken to determine material engineering parameters (foundation indicator test work, triaxial testing, permeability, compaction, etc.).
- A hydrogeological assessment should also be undertaken, which would include:
  - Installation of boreholes upstream and downstream of the WSD (baseline water quality and for monitoring water quality).
  - Permeability testing within boreholes; map groundwater levels and gradients seasonally.
  - Development of a seepage model.

26.7 Environmental and social

- The approved ESIA covers Phase 1 operations. Thor will update the ESIA for the Phase 2 refractory gold process, including emissions management, tailings and waste management and develop mitigation strategies for potential impacts on water use and chemical quantities.
- A biodiversity management plan should be developed, particularly for chimpanzees and link this with the ecology protection zone in the adjoining Massawa Mining Licence.
- Additional test work is required to define the ARD potential from waste rock and increase the test dataset from the oxide and transitional ores.
- A laterite detoxification process will need to be developed. Test work should be initiated to assess the performance of laterite reactive barriers on Douta contaminated mine water. Thor should consider possible collaboration with the Douta Project neighbours to potentially develop a more regional approach.
- Arsenic is present in some of the ores at the Project. An arsenic control strategy for the roasters will need to be tested and developed.
- A number of specific management plans required under the umbrella of the ESMP are absent. These should be developed and implemented.
- Community agreements for socio-economic benefits to local communities should be progressed. These will include topics such as employment, procurement, training and seed funding of local businesses.
- The Economic Displacement Management Plan will need to be compiled, including a robust compensation procedure.

26.8 Capital and operating costs

- A number of capital estimates will require additional investigation in order to progress the Project to the next stage. Key items include:
  - Haul road construction.
  - Communications.
  - First fills.
  - Power supply, which can be adjusted now that power demand for the site is better understood.
- Several operating costs should also be progressed to a higher confidence level. These include:
  - Mining costs: Pending changes to the mining schedule based on the recommendations above, detailed quotes should be obtained from mining contractors on a bench by bench and pit by pit basis.
  - Processing costs and recovery rates: Following test work outlined in the above recommendations, the recovery formula may need to be adjusted and then re-test reagent consumption rates that flow through to the processing costs.
  - G&A costs: Currently this is based on a benchmark number so will require additional work to align it with Project specific information.
  - TCRC Costs: A specific quote for the Douta Project should be obtained, rather than relying on current costs at the Segilola mine.

An estimate of cost to undertake recommendations is shown in Table 26.1.

Table 26.1 Recommendations cost estimate

Estimated costs to complete recommendations	**US$**
Geotechnical Drilling & Studies	150,000.00
Hydrogeological Drilling & Studies	100,000.00
Met Test Work - Variability Studies	100,000.00
TSF - Geotechnical testing	25,000.00
WSD - Geotechnical testing	25,000.00
Baraka environmental base line studies	20,000.00
Mining trade-off studies	150,000.00
Processing trade-off studies	75,000.00
Acid Rock Drainage testing work.	15,000.00
Total cost	**660,000.00**

Source: AMC, 2026.

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28 QP Certificates

QUALIFIED PERSON CERTIFICATE

I, Dominic Claridge, FAusIMM(CP), of Reading, United Kingdom, do hereby certify that:

1 I am currently employed as a Principal Mining Engineer with AMC Consultants (UK) Limited, with an office at Office 336a, Davidson House, Forbury Square, Reading, Berkshire RG1 3EU, United Kingdom.

2 This certificate applies to the technical report titled "Thor Douta Gold Project PFS" with an effective date of 24 January 2026 (the "Technical Report") prepared for Thor Explorations Ltd. ("the Issuer").

3 I am a graduate of the University of Sydney in Sydney, Australia (Bachelor of Engineering in Mining in 1988). I am a Fellow in good standing of the Australasian Institute of Mining and Metallurgy (Fellow #101409). I have more than 30 years of experience within the mineral industry working in roles including experience in mining operations, senior and corporate mine management, project planning and execution, technical due diligence, and Mineral Reserve estimation. I have experience in gold operations with both shaft and decline access. My experience extends to reopening historic mines in Australia and Africa.

5 I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4 I have visited the Project site between 6-9 January 2026, for 3 days.

5 I am responsible for Sections 2, 3, 15, 16, 18-22, and 24, and parts of Sections 1 and 25-27 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101.

7 I have not had prior involvement with the property that is the subject of the Technical Report.

8 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 24 January 2026
Signing Date: 5 March 2026

Original signed by
"Dominic Claridge"

Dominic Claridge, FAusIMM(CP)
Principal Mining Engineer
AMC Consultants (UK) Limited

QUALIFIED PERSON CERTIFICATE

I, Robert Chesher, FAusIMM (CPMET), of Brisbane, Australia, do hereby certify that:

1 I am currently employed as a Senior Principal Consultant with AMC Consultants Pty Ltd, with an office at Level 15, 100 Creek Street, Brisbane Qld 4000, Australia.

2 This certificate applies to the technical report titled "Thor Douta Gold Project PFS" with an effective date of 24 January 2026 (the "Technical Report") prepared for Thor Explorations Ltd. ("the Issuer").

3 I am a graduate of University of Queensland in St Lucia, Australia (BSc(Hons) in Metallurgy in 1977). I am a Fellow in good standing of the Australian Institute of Mining and Metallurgy (AusIMM) and am accredited as a Chartered Professional of the AusIMM in the discipline of Metallurgy (License #311429). I am a Registered Professional Engineer of Queensland (RPEQ #24758). I have practiced my profession continuously since 1977. My expertise is in corporate and technical (metallurgical) consulting, focusing on operational and performance reviews, improvements, and optimisation.

6 I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4 I have not visited the Project site.

5 I am responsible for Sections 13 and 17, and parts of Sections 1 and 25-27 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101.

7 I have not had prior involvement with the property that is the subject of the Technical Report.

8 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 24 January 2026
Signing Date: 6 March 2026

Original signed by
"Robert Chesher"

Robert Chesher, FAusIMM (CPMET)
Senior Principal Consultant
AMC Consultants Pty Ltd

QUALIFIED PERSON CERTIFICATE

I, Alfred Gillman, FAusIMM(CP), of Perth, Western Australia, do hereby certify that:

1 I am currently employed as General Manager (Exploration and Resources) with Thor Explorations Ltd. with an office at 4th Floor, 32 Wigmore Steet, West End London, UK.

2 This certificate applies to the technical report titled "Thor Douta Gold Project PFS" with an effective date of 24 January 2026 (the "Technical Report") prepared for Thor Explorations Ltd. ("the Issuer").

3 I am a graduate of University of Western Australia (Bachelor Science/Honours). I am a Fellow and Chartered Professional in good standing of the Australian Institute of Mining and Metallurgy.

7 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.

4 I have visited the Project site on the following dates 29 July – 1 August 2022, 11 April – 14 April 2025, 5 June – 6 June 2025.

5 am responsible for Sections 4-12, 14, and 23 of the Technical Report.

6 I am not independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101, as I am an employee of the Issuer.

7 have not had prior involvement with the property that is the subject of the Technical Report.

8 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 24 January 2026
Signing Date: 9 March 2026

Original signed by
"Alfred Gillman"

Alfred Gillman, FAusIMM(CP)
General Manager (Exploration and Resources)
Thor Explorations Ltd.

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