Feasibility Study
NI 43-101 Technical Report
Oko West Project

Prepared for:

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OKO GOLD MINES

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Effective Date: April 28, 2025
Issue Date: June 06, 2025

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Feasibility Study NI 43-101 Technical Report – Oko West Project

Revision #

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1. SUMMARY

1.1 Introduction

G Mining Ventures Corp. ("GMIN" or the "Company") mandated G Mining Services Inc. ("GMS") as lead consultant along with other engineering consultants to prepare a Feasibility Study ("FS") under the supervision of the QPs for the Oko West Project ("GYOW" or "Project"), located in Cuyuni-Mazaruni Mining Districts, Guyana.

GMIN is a gold mining company focusing on acquisition, exploration, and development of precious metal projects. Its flagship property is Tocantinzinho Gold Project located in the State of Pará, Brazil. The Company's common shares trade on the Toronto Stock Exchange (TSX: GMIN) and the OTC markets (OTCQX: GMINF).

This Technical Report is prepared in accordance with the guidelines of the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101") and Form 43-101F1. The objective of this report and the FS is the evaluation of the technical feasibility and economic viability of the Project, notably the development of an open pit and underground mine, including processing facilities and related infrastructures. This Report provides operating and capital costs estimations and an economic analysis of the Project.

The MRE was prepared following the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (2014), and in accordance with CIM Guidelines (2019) for Estimation of Mineral Resources and Reserves. The effective date of the mineral resource estimation is September 15, 2024. It is based, for the most part, on a drilling database and supporting geological information.

The mine design and Mineral Reserve estimate were completed to a level appropriate for feasibility studies. The Mineral Reserves were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (November 29, 2019) and CIM Definition Standards for Mineral Resources and Reserves (May 10, 2014).

As such, the Mineral Reserves are based on Measured and Indicated Mineral Resources and do not include any Inferred Mineral Resources. The Inferred Mineral Resources contained within the mine design are treated as waste at a grade of 0 g/t Au.

The qualified persons ("QP") of this Technical Report are the following:

- Mr. Christian Beaulieu, P.Geo., Mineral Resource Consultant for Minéralis Services-Conseils Inc.
- Dr. Kevin Leahy, B. Sc. (Hons), PhD, C.Geol., SiLC, at Environmental Resources Management (ERM).
- Mr. Neil Lincoln, P.Eng., Consulting Metallurgist for Lincoln Metallurgical Inc.
- Mr. Alexandre Burelle, P.Eng., Mine Planning and Financial Analysis Consultant for Evomine Consulting Inc.
- Mr. Sebastien Guido, P.Eng., M. Sc. Senior Engineer, Rock Mechanics at Alius Mine Consulting Inc.
- Mr. Paul Murphy, P.Eng., Project Manager as an independent consultant.
- Mr. Paulo Ricardo Behrens da Franca, P.Eng, Director at F&Z Consultoria e Projetos.
- Mr. Christian Beaulieu visited the Project site between April 18 and April 19, 2023, and between March 11 and 12, 2025.
- Mr. Alexandre Burelle visited the Project site between March 11 to March 12, 2025.
- Mr. Paul Murphy visited the Project site between March 31 to April 7, 2025.
- Mr. Sebastien Guido visited the Project site between April 12 to April 18, 2025.
- Mr. Paulo Ricardo Behrens da Franca visited the Project site between February 18 to February 20, 2025.
- Mr. Neil Lincoln, P.Eng., did not visit the site.
- Dr. Kevin Leahy did not visit the site.

1.2 Terms of Reference

The units of measure presented in this Technical Report, unless noted otherwise are in the metric system. Currency is expressed in United States dollars ("USD"), unless stated otherwise.

1.3 Reliance on Other Experts

This Technical Report has been prepared by GMS, under the supervision of the QPs for GMIN. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to GMS at the time of the preparation of this Report.

- Assumptions, conditions, and qualifications as set forth in this Report.
- Data, reports, and opinions supplied by GMIN and other third-party sources.

The QPs of this Technical Report believe that the basic assumptions contained in the information indicated above are factual and accurate and that the interpretations are reasonable. The QPs of this Technical Report have, to the extent applicable, relied on this data and have no reason to believe that any material facts have been withheld. The QPs of this Technical Report have taken all appropriate steps, in their professional judgement, to ensure that the work, information, or advice from the above information is sound and the QPs do not disclaim any responsibility for this Technical Report.

The QPs have not performed an independent verification of the land title and tenure information, as summarized in Section 4 of this Technical Report, nor have they verified the legality of any underlying agreement(s) that may exist concerning the permits or other agreement(s) between third parties, as summarized in Section 4 of this technical report. For this topic, the QPs of this Report have relied on information provided by the legal department of GMIN. The QPs have relied on GMIN for guidance on cost allocation and applicable taxes, royalties, and other government levies or interests, applicable to revenue or expenditures from the Oko West Project, as described in Sections 4, 19, 20 and 21.

The authors wish to emphasize that they are QPs only in respect of the areas in this Technical Report identified in their “Certificates of Qualified Persons” submitted with this Technical Report to the Canadian Securities Administrators. Except for the purposes legislated under Canadian provincial and territorial securities law, any other use of this Technical Report by any third party is at the party’s sole risk.

1.4 Property Description and Location

The Oko West Project straddles the Cuyuni-Mazaruni Mining Districts (administrative Region 7) in north central Guyana, South America. The Project is located approximately 120 km southwest of Georgetown, the capital city of the Co-operative Republic of Guyana (hereinafter, “Guyana”) and approximately 50 km from Bartica, the capital city of Region 7. Bartica is accessible by a 20-minute direct flight from the Ogle airport in Georgetown or by road and boat from Parika on the Essequibo River. There are regular boat services between Bartica and Parika. The Project is accessible by the Puruni and Aremu laterite roads from the town of Itaballi at the confluence of the Cuyuni and Mazaruni rivers.

The Project comprises one Prospecting Licence (“PL 004/2022” or “GMIN’s PL”) issued to GMIN Ventures Guyana Inc. (previously known as Reunion Gold Inc. and hereinafter, “GMIN Guyana”), GMIN’s 100%-indirectly owned Guyanese subsidiary, on September 23, 2022. GMIN’s PL is valid for three (3) years and is renewable twice for a period of one (1) year each time. The PL has a surface area of

approximately 10,890 acres (4,407 hectares). In 2024, additional land has been added to the Project as medium scale permits that will be converted to a Mining Licence. The Guyana Government holds the surface rights to the Prospecting Licence area. The PL allows GMIN to occupy the area.

The Government of Guyana imposes a royalty of 8% on gold produced from large-scale mines within the country. The Mineral Agreement signed as of April 17, 2024, between the Government of Guyana, the GGMC and GMIN Guyana establishes an 8% of Net Smelter Returns royalty applicable to open pit gold production and a 3% of Net Smelter Returns royalty from underground gold production.

1.5 Accessibility, Climate, Local Resources, Infrastructure & Physiography

The Project can be accessed via numerous methods:

- **Helicopter:** A direct flight from Ogle Airport to the helicopter pad at the campsite,
- **Fixed-Wing Plane and Car:** wing flight from Ogle Airport to the Bartica airstrip, followed by a car ride to Itaballi, crossing the Mazaruni River by ferry and a 1.5-hour car ride to site.
- **Car and Speedboat:** A 1-hour drive from Georgetown to Parika (at the mouth of the Essequibo River), followed by a 1-hour river boat ride to Itaballi, and concluding with a 1.5-hour car ride to the project site.
- **Four-Wheel Drive Vehicle:** An 8-hour journey from Georgetown to the project site, involving a paved drive to Linden, followed by laterite roads to Itaballi, including two ferry crossings.

The climate is equatorial and humid, with two (2) dry seasons, one from approximately March to mid-April and the other from August to November. The dry seasons' onset and duration vary from year to year. The heaviest precipitation is expected in May and June. The average yearly temperature is about 26°C; in the interior regions of Guyana, one can expect typical daily highs around 34°C to 36°C and typical nighttime lows of 16°C to 18°C.

The Oko West Project has operated throughout the year without any interruptions related to the weather. However, laterite road conditions deteriorate significantly during the rainy seasons and might cause transportation delays.

The region's infrastructure is in development, lacking power, roads, communications, and general services.

The city of Bartica (population of about 10,000), at the Essequibo, Mazaruni and Cuyuni rivers' confluence, is the primary hub for artisanal mining activity in northwest Guyana. It houses the government administrative offices for Region 7 and has basic commercial, transportation and manufacturing facilities. It is linked to

Parika by regular river speed boats transporting people and cargo. It has a hospital, an elementary-level school, and cell phone communication.

The town of Itaballi, at the left of the Mazaruni river, is the gateway for the road transportation of goods and services to all the artisanal mining operations not reachable by a river. It is also where wood logs harvested in the region get loaded into barges for transportation to the sawmills in Parika. The Cuyuni landing is the logical location for installing docking facilities to service the mining operation at Oko West.

The existing infrastructure at the Oko West Project includes campsite facilities designed to support mineral exploration activities and environmental studies on the property. The campsite has been upgraded to accommodate more people and provide better facilities.

There is no available grid electrical power in the region. The entire Guyana power system currently runs on heavy fuel or diesel thermal plants installed along the coast and at Linden and Bartica. There are no power lines or substations in the project vicinity.

1.6 History

GMIN Ventures Guyana Inc. is a Guyanese subsidiary wholly owned by G Mining Ventures Corp (GMIN) following the merger with Reunion Gold Corporation (RGD).

G Mining Ventures Corp. (“GMIN”) (TSX: GMIN) (OTCQX: GMINF) and Reunion Gold Corporation (“RGD”) (TSXV:RGD) (OTCQX:RGDFF) announced the entering into a definitive agreement to combine the two companies on April 22, 2024. The merger via a plan of arrangement was overwhelmingly approved by GMIN shareholders as well as Reunion Gold securityholders at their respective annual general and special meetings held on July 9, 2024, and was subsequently approved by the Ontario Superior Court of Justice (Commercial List) on July 11, 2024.

G Mining Ventures Guyana Inc. currently holds a 100% interest in the Oko West Project and Prospecting Licence (PL). This PL was issued on September 23, 2022, by the Guyana Geology and Mines Commission (GGMC), following the relinquishment of 11 medium-scale mining permits held by two (2) Guyanese entrepreneurs with whom Reunion had signed option agreements (see Section 4 for more details). These entrepreneurs had acquired some of these permits from other local parties and registered them to their names. The previous permit holders produced an unknown quantity of gold in an artisanal basis, primarily by alluvial mining.

Gold was discovered in the region in the late 19th century by artisanal miners. In 1935, Dr. Grantham of the British Geological Survey described mining activities in the area. British Guiana Goldfields Ltd. recognized the area's potential, extended the Kartabu-Puruni road to Bird Page on the Oko River, and established a trading post in 1938. In 1939, they drilled over 500 Banka holes in the Oko and Kairuni River areas but eventually shifted focus to the more profitable Aremu mine area.

In the mid-1960s, Aero Mineral Surveys Ltd., funded by the United Nations, conducted a regional magnetic and radiometric survey. Between 1966 and 1979, the British Geological Survey produced geological maps and interpretations based on regional fieldwork and geophysical anomalies.

In 2002, the Guyana Geology and Mines Commission (GGMC) carried out a geochemical survey that identified gold and molybdenum anomalies. From 2010 to 2015, extensive artisanal mining targeted gold-rich quartz veins, especially at Crusher Hill, north of the current Oko West Project. Artisanal activity continues today.

In 2016, Sandy Lake Gold Inc. collected high-grade grab samples at Crusher Hill, leading to an option agreement with Michael Vieira in 2018. The company was renamed G2 Goldfields Inc. in 2019 and continues to explore the Oko Main zone.

There is no record of the amount of gold production from artisanal mining within the Oko West Project area.

In 2018, local permit holder Mr. Bryan Stephens introduced Reunion Inc. to several of his mining claims, leading to a site visit at an active mining pit. During the visit, grab samples (BS-02, BS-03 and BS-04) were collected, returning gold assay results of 2.40, 6.00 and 45.84 g/t Au, respectively. Encouraged by these results and the favorable local geology, Reunion entered into an option agreement for two medium-scale permit blocks: Oko West and Oko East. The Oko East block was later dropped, and Reunion continued exploration on the Oko West property.

There is no record of historical drilling within the project area prior to Reunion's involvement.

1.7 Geological Setting and Mineralization

The Project is located within the Guiana Shield, which corresponds to the northeastern portion of the Amazonian Craton. With a total area of 900,000 km^2, it covers eastern Venezuela, Guyana, Suriname, French Guiana, the northern end of Brazil and easternmost Colombia.

The Trans-Amazonian Province is composed of large Rhyacian (2.20 - 2.05 Ga) granite-greenstone belts, including volcano-sedimentary rocks, metamorphosed to greenschist facies, intrusive granitoids, and TTG (tonalite-trondhjemite-granodiorite) gneisses (Vanderhaeghe et al., 1998; Santos et al., 2000; Tedeschi et al. 2018, Tedeschi et al., 2020). In Guyana, the greenstone belts are described from deepest to shallowest as basalt ± ultramafic rocks, intermediate to felsic volcanic rocks, and finally, tuffs and turbiditic sedimentary rocks (Gibbs, 1980; Tedeschi et al., 2020). They host multiple gold deposits; however, little is known about the relationship between gold mineralization, magmatism, and deformation (Tedeschi et al., 2020).

Two (2) major tectonic events have affected the Trans-Amazonian Province: a D_1 event involving into a N-S convergence of the Archean African and Amazonian cratons (2.18 to 2.13 billion years ago), followed by the closure of the volcanic arc basins, defined as the D_2 event (2.11 to 2.06 billion years ago), and marked by granitic magmatism, minor mafic intrusions, and regional greenschist metamorphism, as well as folding of the volcano-sedimentary formations. At Oko West, the Oko, Aremu and Puruni plutons are most likely the result of the D_1 tectonic event. These plutons caused deformation of the Barama-Mazaruni Supergroup volcano-sedimentary rocks (2.12 billion years old), leading to the formation of gold occurrences within local structures.

Gold mineralization at Oko West straddles the north-south striking contact between Barama-Mazaruni Supergroup greenstone belt rocks to the west and a granitoid pluton to the east (the Oko pluton). Locally, the Barama-Mazaruni Supergroup sequence comprises mafic volcanics, volcaniclastics, and siliciclastic and carbonaceous sediments and is the main host to mineralization at Oko West.

Lacroix and Hainque (2024) and Hainque et al. (2024) propose a geological evolution for the Oko West mineralization, based on field observations and structural analyses:

- **D0**: Deposition of volcano-sedimentary sediments, mafic volcanic rocks, and granitoid intrusions, leading to early potassic veins and metasomatism.
- **D1a**: A compressional event caused folding (F1) and the formation of bedding-parallel veins that may have contained minor gold.
- **D1b**: Tight folding continued, creating a penetrative foliation (S1) and transposing early veins (EV1) into S1, resulting in locally dismembered and sheared veins.
- **D2**: N-S folding developed S2 crenulation and late-stage quartz-Sulphide veins (EV2). Most of the gold was deposited during this phase along fractures related to F2 and S1.
- **D3**: Formation of two (2) sets of conjugated fractures or faults, potentially with significant displacement.

Long chemical weathering typical of humid equatorial paleoclimate produced a lateritic profile up to 100 m thick from the surface. This profile is typically composed of a veneer of pisolitic colluvium or latosol overlaying a massive clay zone, which pass into a mottled zone and then saprolite / saprock before reaching unweathered rocks at depth.

1.8 Deposit Types

The relationship between gold mineralization, magmatism, and structural deformation in the Guiana Shield remains poorly understood despite the presence of several major gold deposits. In French Guiana, orogenic gold deposits are mainly linked to the regional D2 tectono-metamorphic event (2.1–2.0 Ga), with mineralization typically occurring in quartz veins along shear zones within greenstone belts, associated with granitic intrusions.

Recent studies of the Karouni deposit in Guyana confirm gold mineralization around 2.084 Ga, hosted in shear-related quartz-carbonate-chlorite ± tourmaline-pyrite-gold veins. These veins form in dilatational bends due to late-stage dextral strike-slip movements, and mineralization is influenced by contrasting rock types: brittle units like dolerite and granodiorite host richer veins than more ductile basalts.

Similar structural controls are seen in other deposits across the region. For example, at Omai (Wenot Lake), mineralization occurs along shear zones between sedimentary and volcanic rocks. At the Rosebel Mine in Suriname, gold is found along shear zones at lithological contacts and to a lesser extent in fold-related axial cleavages. The Royal Hill Deposit also exhibits similar mineralization: gold is hosted in bedding-parallel veins along lithological boundaries. These examples highlight the strong influence of structural and lithological contrasts on gold deposition across the Guiana Shield.

1.9 Exploration

Modern exploration of the Oko West Project comprises geophysics, reconnaissance stream-sediment geochemistry, soil geochemistry, trenching and drilling.

Gold mineralization at Oko West was first identified to the north of the Project, and after some initial reconnaissance, a stream sediment survey was conducted using Bulk Leach Extractable Gold (BLEG) techniques for gold analysis. Although this survey did not cover the current known extents of gold mineralization, a soil geochemical survey was completed over the east of the Project and defined a gold anomaly straddling the contact between the Oko pluton to the east and the volcano-sedimentary sequence to the west, with a strike length of approximately 6 km. Trenching was subsequently undertaken over the anomaly and intersected 5.98 g/t Au over 69.0 m (trench 44) until the program was interrupted due to the

Covid-19 pandemic. The trenching program successfully validated the soil geochemical anomalies and confirmed the presence of significant in-situ gold mineralization in a sequence of sediments striking north-south and at the contact with the Oko pluton granitoid.

In August 2019, the Canadian company Terraquest covered the project area with an airborne geophysical survey of about 690-line km at a 200 m line spacing. In March 2024, an induced polarization geophysics program was completed. Five (5) areas of interest were surveyed and used for exploration targeting across the Oko West property using chargeability and resistivity data. Two (2) pole dipole (PDP) lines were done in Block 4 and three (3) in the East Takatu area. Ground Magnetics survey has been ongoing on the Oko West property since early 2023 with GMIN owned equipment and personnel.

In 2024, trenching on the Oko West Project targeted soil and stream sediment anomalies, focusing primarily on the western part of the property. Additional trenches were completed south of the Oko West main mineralized body in Blocks 7 and 8, specifically at the West Takatu target. The 2024 trenching program covered a total of 2,524 metres. Condemnation drilling was also carried out in 2024 around the Tailings Storage Facility (TSF) and Waste Storage Facility (WSF). In the TSF area, 47 diamond drill holes (DDH) were completed, totalling 12,386 metres. In the WSF area, four (4) DDH and 60 reverse circulation (RC) holes were drilled, totalling 231 metres and 1,508 metres, respectively. The condemnation program verified that these areas were appropriate for infrastructure placement and would not overlap with economically viable near-surface mineralization.

1.10 Drilling

Drilling commenced at Oko West in December 2020, with three (3) reconnaissance holes targeting primary mineralization beneath the previously reported trench results. After the initial discovery, drilling from 2021 to 2024 has been mainly focused on delineating gold mineralization and satisfying the drill spacing required to calculate an in-pit and underground Mineral Resource Estimate (“MRE”).

Drilling methods at Oko West comprise of diamond drilling (“DDH”) and reverse circulation drilling (“RC”). As of the effective date of this report, 236,288 m of drilling and trenching has been conducted on the Project, of which 171,788 m is DDH (including wedged holes), 53,241 m is RC, and the remaining 11,259 m is trenching. Beginning late 2023, a delineation program using wedges and Down Hole Motor (DHM), or directional drilling, was started by Reunion Gold to convert underground resource to the Indicated category. A total of 14,654 m was drilled using this methodology.

Drill core recovery is considered excellent, averaging 98.2% in fresh rock. The lateritic profile is drilled with HQ-diameter drill rods, and NQ-diameter drill rods are used once hard ground conditions are encountered.

RC drilling is used for reconnaissance scout drilling to test regional soil anomalies and to test for strike extensions of known mineralization. RC drill samples are sourced from an onboard splitting system on the drill rig to ensure sample quality and representativity.

1.11 Sample Preparation, Analyses and Security

Diamond drill core samples are collected on average at every 1.2 m from drill core but vary between 0.10 and 3.90 metres. Sample intervals are marked by geologists. Samples are selected in potential mineralized zones based on logged geological features, such as rock type, mineralization, alteration and veining. Reverse circulation (RC) chip samples are collected at every metre. The splitter on the rig will produce 2 kg samples for the primary lab, the field duplicate sample and the bulk sample for storage and future reference.

Blanks, certified standards, and duplicates are inserted at the same time as the sampling process is performed. Certified reference material (CRMs or standards) and blanks include one (1) control sample every ten (10) samples interchanging between standard and blank, or the equivalent of one (1) blank and one (1) standard at every 20 samples (5%). The position of blank and standard samples is adjusted to control mineralized intervals and test lab contamination. The Company's procedures of quality control (QC) samples are designed to insert one (1) standard, one (1) blank, and one (1) field duplicate at every 20 samples generated by drilling. The primary laboratory (Actlabs) sends pulp duplicates directly to the secondary laboratory (MSA Labs) for umpire check assays.

GMIN uses a sample tag system containing the sample information, including date, target, hole or trench, interval from-to in metres, sampler name and analytical code. Access to samples is only possible by cutting the tag. At the beginning of the project, the samples were sent via boat and truck to the primary laboratory in Georgetown (Guyana), accompanied by one (1) company employee for the entire trip to witness that all samples reach the laboratory safely. Now, the samples are sealed on site, loaded on the sample truck, and sealed to the truck tarp. The truck driver signs a manifest on site and the lab confirms receipt of the truck with tags intact upon delivery.

Bulk density measurements are taken in-house on all representative core from the lithological intervals, including mineralized and non-mineralized units, with varying degrees of hydrothermal alteration and weathering.

Sample batches are prepared following the Actlabs Code RX1 procedure. Samples are weighed and dried; crushed (<5 kg) to a fineness of 80% passing 2 mm. A riffle split of 250 g is taken from the crushed material and pulverized (mild steel) to 95% passing 105 µm (140 mesh). At Actlabs, gold analysis code FAAA-1A2

is performed using a 50 g fire assay (FA) with atomic absorption spectrometry (AAS) finish. For gold values above the upper detection limit (> 3,000 ppb), samples are assayed by fire assay with gravimetric finish (FAGRA-13A). If visible gold is observed by the geologist during the logging and sampling, the analytical method 1A4 Au fire assay metallic screen is prioritized, and the sample before and after the visible gold is also analyzed using the metallic screen method.

The assay reports by both the primary and secondary labs are distributed by e-mail directly to recipients listed in the work order, including gDat Solutions, a third-party, independent database manager.

The QP concludes that the sample preparation, analysis, and security procedures applied by Reunion are acceptable. Documentation of sampling procedures used to support the diamond and reverse circulation drilling programs is considered by the QP as consistent with best industry practice. In addition, the QP believes that sample preparation, analysis and security procedures implemented by Reunion are comparable with the best industry standards, and robust controls are in place to ensure the integrity of the assay database.

1.12 Data Verification

Mr. Christian Beaulieu P.Geo., consulting geologist for GMS and qualified person (QP) under NI 43-101 visited the project on two occasions: on April 18 to April 19, 2023, and on March 11 to March 12, 2025. During these site visits, the QP inspected mineralized intervals, alteration assemblages and QAQC protocols and conducted field checks of trenches and validated drill collars. Some trenches were cleaned, and sampling protocols were assessed directly in the field with site geologists; drill core was reviewed with GMIN personnel. Drill core review permitted to observe clear relationships between gold grades (or presence of mineralization) and rock alteration / strain within mineralized domains LDZ, AU_2, AU_3, AU_3A, AU_3B, AU_4, AU_5 and LDZ_B6. It is in the opinion of the QP that GMIN work practices at Oko West are in line with the CIM Best Practice Guidelines (2019).

Prior to M. Beaulieu's site visits, two (2) former project QPs visited the project: James Purchase, P.Geo., geologist at GMS, visited the project between May 9, 2022, and May 13, 2022, and Pascal Delisle, P.Geo., Director of the Geology and Resource department at GMS visited the project between January 30, 2024, and February 2, 2024. All former QPs expressed their confidence in GMIN work practices.

During all site visits, drilling activities were ongoing. Core processing and storage facilities located on-site were toured, and the drill core from Oko West deposit was reviewed. Outcropping mineralization and trenches were visited and compared with the provided LiDAR topographic surface for survey accuracy.

Sampling protocols were also reviewed with field geologists working on the Oko West Project. Independent quarter core samples were collected, and pulps sent to an independent laboratory.

The validity of the drilling database including assay certificates, collar locations, downhole surveys and twin drill holes was reviewed. In addition to reviewing the sampling procedures, GMS conducted a comparative analysis of duplicate samples sent to both Actlabs and MSALABS. The results indicated a good correlation between the original assays and the duplicate assays, with slight variations attributable to factors such as sample size.

1.13 Metallurgical Testing and Mineral Processing

Continuing from previous test work, a feasibility level metallurgical test work program was completed August 2024, with additional test work conducted in January 2025 at BaseMet Laboratories (BML) in Kamloops, British Columbia. The objectives of the test work program were to further define the metallurgical response of the main ore domains within each weathering profile, generate sufficient metallurgical data to support a flowsheet and develop gold recoveries for the Feasibility Study (FS). The scope of the test work included chemical and mineralogical analyses, comminution test, gravity recovery, cyanide leaching and detoxification, blended composition assessments, carbon loading and oxygen uptake evaluations, and dewatering tests (including flocculant selection, static settling, and dynamic thickening tests). The relevant metallurgical data and findings from the 2024-2025 programs are provided in this Technical Report.

Samples were selected from three weathering zones (saprolite, transition, and fresh rock) and major geological units, resulting in 16 domain composites and 39 variability composites. Gold content in the samples varied between 0.67 and 16.1 g/tonne (as expected) and silver ranged between 0.1 and 21.25 g/tonne. Sulphur in the samples measured between 0.01 and 3.15 percent. Base metal content in the samples was low, i.e. 67 ppm Cu, 83 ppm Zn, 9 ppm Pb (average). Arsenic and mercury levels were low at 7 ppm As and < 1 ppm Hg. The Preg-robbing value (PRV) of select samples was measured and indicated near negligible values of preg-robbing index (PRI).

Bulk Mineral Analysis (BMA) using QEMSCAN, was conducted on the samples. Pyrite accounted for the main sulphide mineral in almost all the samples. Chalcopyrite, sphalerite and other sulphides were also detected in lower concentrations in the majority of the other samples. The non-sulphide suite of minerals varied, consisting mainly of quartz, feldspars, muscovite / illite and chlorite and clays

From a material hardness point of view, fresh rock is more competent, hard and abrasive in comparison to the saprolite and transition material. The fresh rock exhibits competent material (Axb 15th percentile of 29.4),

hard grindability (85th percentile BBWi of 16.4 kWh/tonne and BRWi of 17.9 kWh/tonne) and is mildly abrasive (average Ai of 0.139).

Gravity recoverable gold tests were performed on all samples using a Knelson concentrator. Gravity gold recovery for the Fresh Rock and Saprolite samples were approximately 30-50%.

A series of metallurgical tests were performed on samples evaluating parameters such as primary grind sizing, air vs. oxygen sparging, cyanide concentration, pre-oxidation and CIL. Overall, gold responded well to the various flowsheets, with average gold recoveries ranging between 85.5 and 99.6 percent, after gravity concentration and 48 hours of leaching, with average gold recoveries of 93.3 percent being obtained. To maximize gold extraction after 24 hours of leaching, optimum conditions were selected as a primary grind size of 75 µm K80 prior to gravity concentration, a sodium cyanide concentration of 1,000 ppm, sparged with oxygen, and subject to a 2-hour pre-oxidation stage. These conditions were used going forward with the low- and high-grade variability composite testing. Using these conditions, gold recovery to the gravity concentrates ranged between 2.4 and 36.7 percent, averaging at 11.8 percent, gold extraction at 24 hours, including the gravity recovery, ranged between 87.4 and 99.6 percent, averaging at 93.8 percent. The leach tests were performed for a period of 48 hours. Gold extraction, including gravity concentration, at 48 hours, ranged between 90.6 and 99.6 percent, averaging 95.3 percent.

There was no discernible correlation noted between feed gold content and overall gold extraction for the samples tested. FS results confirmed that Oko West ores are free milling, can recover gold via conventional gravity and cyanidation technologies.

Overall, gold recoveries from gravity-leach tests yielded the best results, with average Au recovery of 95.0% for saprolite, 93.3% for transition and 94.0% for Open Pit fresh, and 92.7% UG material types.

1.14 Mineral Resources Estimate

The MRE methodology is summarized below:

- Drillhole database validations.
- 3D modelling of host units (lithological model).
- 3D modelling of gold-bearing domains.
- Geostatistical analysis for data conditioning: mineralization domain validation, density assignment, capping assumptions, compositing and variography.
- Block modelling and grade estimation.

- Resource classification and grade interpolation validations.
- Grade and tonnage sensitivities to different cut-off grade scenarios.

The MRE considers 544 diamond drill holes (including 39 wedges DDH), 336 reverse circulation holes and 59 trenches, completed between December 2020 and September 2024 by GMIN.

The effective date of the mineral resource estimation is September 15th, 2024, and the MRE statement is listed in Table 1.1.

Table 1.1: Oko West In-pit and Underground Mineral Resources Estimate

Category	Updated MRE Tonnage (kt)	Updated MRE Au Grade (g/t)	Updated MRE Contained Gold (koz)
Pit Constrained Resource			
Indicated	73,035	2.00	4,689
Inferred	1,542	1.06	52
Underground Constrained Resource			
Indicated	7,223	3.09	718
Inferred	3,585	2.93	337
Total Open Pit and Underground			
Indicated	**80,259**	**2.10**	**5,407**
Inferred	**5,127**	**2.36**	**390**

**Notes on Mineral Resources:*
The Mineral Resources described above have been prepared in accordance with the CIM Standards (Canadian Institute of Mining, Metallurgy and Petroleum, 2014) and follow Best Practices outlined by the CIM (2019).

1. *The qualified person (QP) for this Mineral Resource Estimate (MRE) is Christian Beaulieu, P.Geo., Consulting Geologist for G Mining Services Inc.*
2. *The effective date of the Mineral Resource Estimate is September 15, 2024.*
3. *The lower cut-offs used to report open pit Mineral Resources is 0.30 g/t Au in saprolite and alluvium / colluvium, 0.34 g/t Au in transition, and 0.38 g/t Au in fresh rock.*
4. *Underground Mineral Resources are reported inside potentially mineable volume and include below cut-off material (stope optimization cut-off grade of 1.35 g/t Au).*
5. *The Oko West Deposit has been classified as Indicated and Inferred Mineral Resources according to drill spacing. No Measured Mineral Resource has been estimated. Classification of underground material was attributed on a stope-by-stope basis.*
6. *The density has been applied based on measurements taken on drill core and assigned in the block model by weathering type and lithology.*
7. *A minimum thickness of 3 metres and minimum grade of 0.30 g/t Au was used to guide the interpretation of the mineralized zones.*
8. *This MRE is based on a subblock model with a main block size of 5 m x 5 m x 5 m, with subblocks of 2.5 m x 1.25 m x 2.5 m, and has been reported inside an optimized pit shell and optimized stope shapes. Gold grades in fresh rock, transition and saprolite were interpolated with 1 m composites using Ordinary Kriging for all major domains and Inverse Distance Square for minor domains (AU_2A, AU_2B, Au_3B, AU_5 and LDZ_B6). Capping was applied on nine domains, ranging from 5 g/t Au to 45 g/t Au.*
9. *Open pit optimization parameters and cut-off grades assumptions are as follows:*
   a. *Gold price of US$1,950/oz.*

b. Total ore-based costs of US$16.43/t for saprolite and alluvium/colluvium, with a 94.5% processing recovery US$18.31/t for transition with a 93.3% processing recovery and US$20.56/t for fresh rock based on 93.9% processing recovery.
c. Inter-ramp angles of 30° in saprolite and alluvium / colluvium, 40° in transition and 50° in fresh rock.
d. Royalty rate of 8%.

10. *UG optimization parameters and cut-off grades assumptions are as follows:*
    a. Gold price of US$1,950/oz.
    b. Total ore-based costs of US$75.16/t for fresh rock.
    c. Stope height of 30 m, strike length of 20 m, maximum width of 30 m and minimum width of 2 m.
    d. The Deswik.SO (DSO) was used to constrain the Resources.
    e. Royalty rate of 3%.
11. *Tonnage has been expressed in the metric system, and gold metal content has been expressed in troy ounces. The tonnages have been rounded to the nearest 1,000 tons, and the metal content has been rounded to the nearest 1,000 ounces. Totals may not add up due to rounding errors.*
12. *These Mineral Resources assume no mining dilution and losses, however must-take material is accounted for in underground stopes.*
13. *These Mineral Resources are not mineral reserves as they have not demonstrated economic viability. The quantity and grade of reported Inferred Mineral Resources in this news release are uncertain in nature and there has been insufficient exploration to define these resources as indicated or measured; however, it is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.*
14. *There are no known factors or issues that could significantly impact the MRE, other than the typical risks associated with mining projects, such as environmental, permitting, taxation, socio-economic, marketing, and political factors, as well as additional risk factors related to indicated and inferred mineral resources.*

It was determined that the database used for estimation is reliable, and that the current drilling information is of sufficient quality for interpreting the boundaries of gold mineralization with confidence.

1.15 Mineral Reserve Estimate

The mine design and Mineral Reserve estimate were completed to a level appropriate for feasibility studies. The Mineral Reserves were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (November 29, 2019) and CIM Definition Standards for Mineral Resources and Reserves (May 10, 2014).

As such, the Mineral Reserves are based on Measured and Indicated Mineral Resources and do not include any Inferred Mineral Resources. The Inferred Mineral Resources contained within the mine design are treated as waste at a grade of 0 g/t Au.

The Proven and Probable Reserve for the Oko West project is estimated at 76.6 Mt at an average grade of 1.89 g/t Au for 4.64 million ounces of gold as summarized in Table 1.2.

It should be noted that there are no Proven Reserves; only Probable Reserves are declared.

Table 1.2: Oko Gold Project Mineral Reserve

Area	Category	Tonne (Mt)	Au Grade (g/t)	Contained Au (k oz)
Open Pit	Proven	0	0	0
	Probable	62.4	1.57	3,156
	Proven and Probable	62.4	1.57	3,156
Underground	Proven	0	0	0
	Probable	14.2	3.26	1,486
	Proven and Probable	14.2	3.26	1,486
Total (Open Pit + Underground)	Proven	0	0	0
	Probable	76.6	1.89	4,642
	Proven and Probable	76.6	1.89	4,642

**Notes on Mineral Reserves:*

1. *The Mineral Reserves were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (November 29 ,2019) and CIM Definition Standards for Mineral Resources and Reserves, (May 10, 2014). The mine design and Mineral Reserve estimate have been completed to a level appropriate for feasibility studies. As such, the Mineral Reserves are based on the Measured and Indicated Mineral Resources and do not include any Inferred Mineral Resources. The Inferred Mineral Resources contained within the mine design are classified as waste.*
2. *Mineral Reserves are estimated using a long-term gold price of US$1,800/oz.*
3. *The qualified person for the estimate is Alexandre Burelle, P.Eng. (OIQ#5019855), Mine planning and financial analysis consultant. The estimate has an effective date of April 2, 2025.*
4. *Mineral Reserves for Open Pit are estimated at a cut-off grade of 0.41, 0.37 and 0.33 g/t Au for Fresh Rock, Transition and Saprolite.*
5. *The Open Pit Strip Ratio is 6.83:1 and Dilution factor is 14%.*
6. *Mineral Reserves for Underground Mine are estimated at a cut-off grade of 1.70 g/t Au.*
7. *The underground mine dilution factor is 11.3% including 4.5% for backfill dilution*
8. *For the underground stopes, a minimum mining width of 5 m was used*
9. *The numbers may not sum due to rounding; rounding followed the recommendations in NI 43-101.*
10. *The mine design and Mineral Reserve estimate have been completed to a level appropriate for feasibility studies. The Mineral Reserve estimate stated herein is consistent with the CIM definitions and is suitable for public reporting.*

1.16 Mining Methods

The Oko West Project is planned as a mining operation that integrates both conventional open pit (OP) mining and mechanized long hole open stoping for the underground (UG) mine. The nominal milling rate will be initially set at 7.0 Mtpa to treat a blend of hard rock, saprolite and transition ores during the open pit operational period. When processing only hard rock, the process plant is designed to nominally treat 6.0 million tonnes per annum. The ramp-up period is five months, and the mill will operate for 12.3 years. The milling process is designed to operate for 13 years with stockpiles peaking at 2.8 Mt by Year 2 to maintain consistent mill feed.

The OP operation will utilize a fleet of diesel-powered equipment, including drills, haul trucks, and hydraulic shovels. The Project consists of a main pit that is deeper and centered on Block 4, with one (1) smaller sub-pit located south of the southern extension to the main pit. The OP operation will be executed in four (4) phases. The OP peak mining rate is 44.0 Mtpa over a Life-of-Mine (LoM) of 13 years. A total of

62.4 Mt of ore material will be mined at an average diluted gold grade of 1.57 g Au/t. A total of 426 Mt of combined waste and overburden will be extracted, resulting in a strip ratio of 6.8 tonnes of waste per tonne of ore material. The primary production equipment includes 22 m³ diesel-hydraulic shovels paired with 140-t off-highway mining trucks. The mining operation is planned to be owner-operated, with exception for explosives handling and blasting services. Pre-production mining is scheduled over approximately 24 months to secure construction material and to remove overburden to allow access to the ore material. A total of 47 Mt of waste as well as 2.6 Mt of ore material will be mined during the preproduction and commissioning period.

The underground (UG) operation comprises a single mine divided into two (2) zones: the main zone and the satellite zone. Both zones are accessible through a common portal located in the south pit and connected via the main ramp. The chosen mining method is long hole open stoping (LHOS), with both transverse and longitudinal stoping techniques being employed.

The LoM for the UG mine is expected to be 13 years including construction, development, pre-production, and the full production period. Over this LoM, the UG mine is expected to be in production for 10 years, including a 2-year ramp-up period. A two-year pre-production period is planned to allow sufficient underground development to be completed in order to sustain full production. The UG mine is expected to achieve an average production rate of 4,625 tpd of mineralized material, with 4,500 tpd derived from stope production and average 125 tpd from lateral development. Development of the UG mine includes approximately 51.8 km of lateral and 2.6 km of vertical development to be excavated. A total of 14.2 Mt of ore material is expected to be mined at an average diluted gold grade of 3.26 g/t Au. The primary production equipment includes 21-t diesel-powered load-haul-dump machines (LHD) coupled with 63-t underground mining trucks to handle all mined material.

The combined production for the OP mine and the UG mine is 76.6 Mt of ore at an average grade of 1.89 g/t Au.

1.17 Recovery Methods

The proposed process plant design for the Oko West Project is based on a standard metallurgical flowsheet to treat gold bearing ore via cyanidation to produce doré. The flowsheet is based on metallurgical test work, industry standards and conventional unit operations.

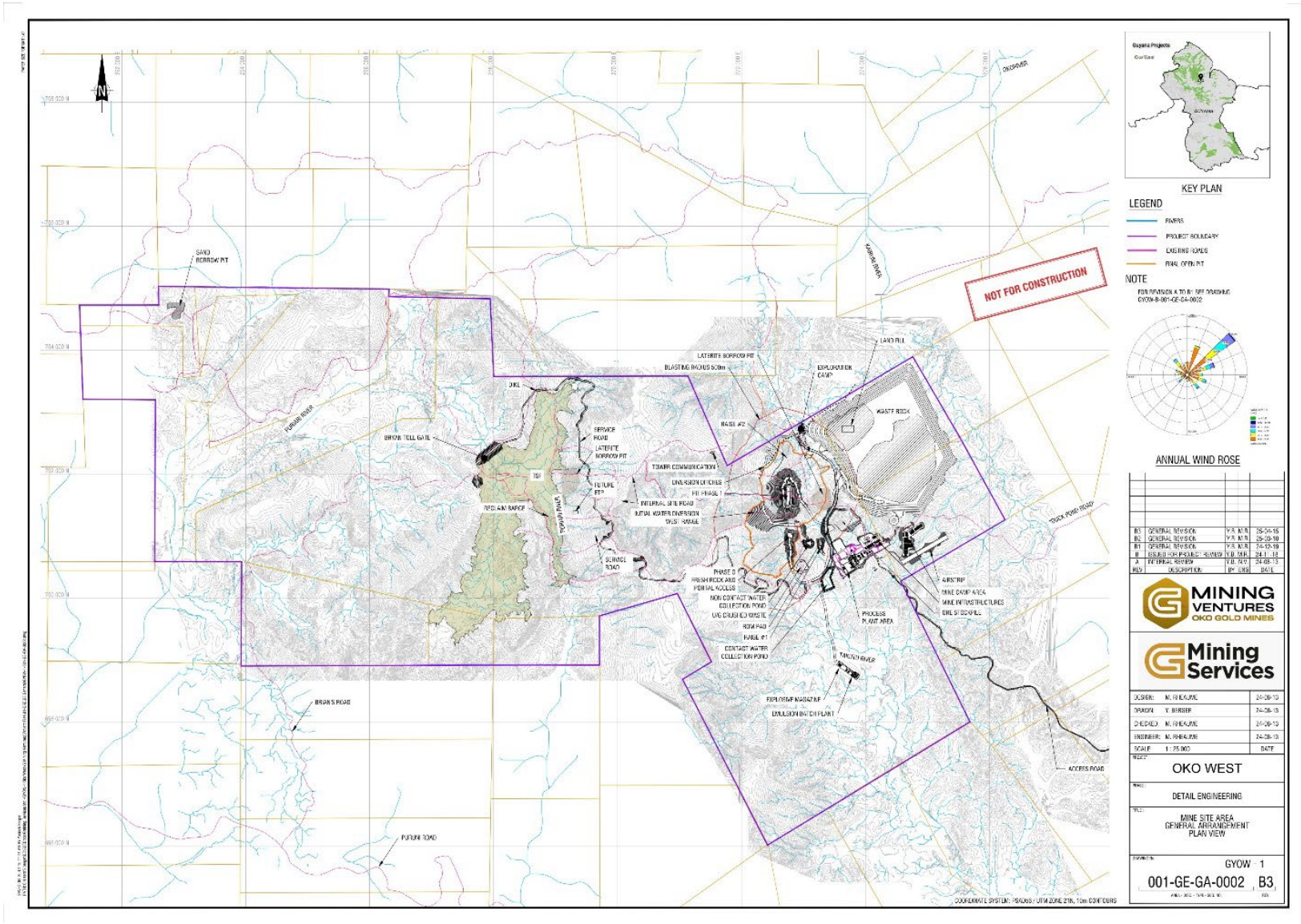
The process plant is designed to nominally treat 6 Mtpa of fresh rock (7 Mtpa with a blend of fresh rock and saprolite/transition ore) and includes the following unit operations:

- Comminution circuit (primary gyratory crusher, SAG and ball mill) to produce a primary grind size of P80 of 75 µm.
- Gravity concentration circuit to produce a gold-rich concentrate for intensive leaching and subsequent gold recovery via electrowinning.
- Pre-leach thickening.
- Cyanide leaching and carbon adsorption via a Leach-CIP carousel circuit. Leach residence time of 36 hours to achieve optimal gold extraction.
- Carbon elution via 10-t Split Pressure Zadra circuit.
- Carbon handling and regeneration.
- Electrowinning and smelting to produce doré.
- Cyanide destruction of Leach-CIP tailings using SO_2 / O_2 process to produce weak acid dissociable (WAD) cyanide levels of less than 5 ppm.
- Tailings pumping to a tailings' storage facility.
- Air and oxygen circuits.
- Water systems (domestic water, raw water, gland seal water and process water).

1.18 Project Infrastructure

The project infrastructures are identified in the general site plan in Figure 1.1.

Figure 1.1: Site Plan View



The Project requires several infrastructures to support mining and processing operations as summarized below.

- Roads and Access:
  - Road access will be via the existing Puruni road with the addition of a planned 12 km laterite surfaced road. Improvement of one (1) existing bridge on the Puruni road is also planned.
  - Service roads of 20 km will connect the various infrastructures located on the property, notably the explosives storage facility, tailings storage facilities, operations site and camp site.
  - Air access will be via a new, 850 m long airstrip that will be used during construction. This airstrip will be used for personnel, supplies, medical emergencies and gold transportation.
- Support infrastructure:
  - A gate and guardhouse will be located at the property entrance along the access road.
  - Permanent camp facility will have a 1,584-person capacity during construction with three types of camp modules. The camp will easily accommodate the operation' requirements. The permanent camp facility includes kitchen and dining area, recreation facilities, camp office and laundry facilities with associated water and sewage treatment plants.
- Mine infrastructures:
  - Facilities near the mine will allow easy access for heavy equipment while safe traffic will be ensured by segregation of light and heavy traffic.
  - Permanent mine maintenance facility will have nine (9) heavy duty bays with an additional six (6) light duty bays. The maintenance facility will include warehousing capacity, office space for the maintenance staff, tool crib and lube storage.
  - Wash bay for heavy duty vehicles will allow equipment to be washed prior to maintenance activities. The wash bay will be equipped with an oil-water separator.
  - Fuel storage will have the capacity to support site during 14 days of operation for both LFO and HFO.
  - Explosive storage facility is designed for a capacity of 1,200 t of emulsion raw materials. Storage capacity on site is sufficient for 30 days at peak consumption.
- Process Infrastructure:
  - Assay laboratory configured to process up to 500 samples per day for mine grade control, exploration and metallurgical samples.
  - Mill office, reagent building and reagent storage are within the processing plant fence.

- Waste Storage and Tailings Facility:
  - Covering an area of 3.59 Mm^2, the waste storage facility has a maximum storage capacity of approximately 216 Mm^3.
  - Enclosed within a large valley inside the prospecting licence, the tailings storage facility will store conventional tailings. Existing water courses will require building main and saddle dams, with the addition of the North dyke. The total embankment volume is estimated at 1.1 Mm^3 over the life of mine.

1.19 Market Study and Contracts

The Oko West Project will produce gold in doré bars. Oko West is authorized to export the gold doré bars from Guyana to an internationally recognized refiner. Prices are typically quoted in US dollars per troy ounce. The gold price was determined based on historical prices and the consensus of long-term estimates from mining analysts. The long-term consensus price for March 10, 2025, is $2,500 per ounce (Source: S&P Capital IQ, a product of S&P Global Market Intelligence. Accessed April 2025. © 2025 S&P Global Market Intelligence. All rights reserved). The financial analysis for the Oko West Project was based on the consensus price of $2,500 per ounce, which is significantly lower than recent spot prices. There are no refining agreements or sales contracts currently in place for the Project that are relevant to this Technical Report. The QP expects that terms contained within any sales contract that could be entered into would be typical of and consistent with standard industry practices and be similar to contracts for the supply of gold elsewhere in the world. Transportation and refining contracts for gold doré bars will be negotiated and finalized during the construction phase of the project. This Feasibility Study assumes a 99.95% gold payability factor and a refining, transportation and insurance charge of $8.00/oz.

G Mining Ventures has entered into a Master Service Agreement with G Mining Services, a Canadian engineering and mine development firm specializing in the mining sector, for the engineering and construction management of the Oko West Project.

1.20 Environmental Studies, Permitting and Social or Community Impact

A Project specific baseline study program was initiated in 2022 and completed in early 2025. The information collected has built an understanding of the spatial and temporal, local and regional environmental and social conditions that have informed the FS and allowed submission of the preliminary ESIA in December 2024 and final submission in May 2025.

The Project area does not overlap with any protected or Indigenous lands and is not located within a priority conservation zone, although it is located within a region of high biodiversity significance. Baseline data shows an environment already impacted by region-wide and degradation caused by artisanal mining for gold and timbering, with effects especially apparent in river water quality and flow and associated habitats. Artisanal mining is also the primary context for the social and community setting in the Project area and surrounding area, with active mining immediately to the west and north associated with a declining community at Oko Landing.

For many topics, such as surface water and groundwater and biodiversity, baseline studies will be advanced into ongoing regular monitoring programs, as captured in the overarching Environmental and Social Management and Monitoring Plan (ESMMP) as part of the EIA.

Key findings of the baseline include the presence of Giant Otters and an archaeological cave within the Project area, and the extent to which current artisanal mining is present in the area and has negatively affected the environment. These features represent the main receptors to the environment and social baseline from the Project. Most streams in the area were previously heavily mined by artisanal miners and have little-to-no community or cultural value, nor are they critical for biodiversity or populations of species of interest, while the quantity of water affected has little influence on downstream regional streamflow. The baseline of other environmental media, such as soil, air, noise, terrestrial habitats and groundwater are relatively undisturbed, especially away from the sites of artisanal mining in stream and river valleys.

A thorough geochemical testing program with both static and kinetic testing was undertaken by ERM between 2023 and 2025 to determine the ML/ARD characteristics of materials excavated, produced and stored by Project. The majority of materials were non-PAG, with a some volumetrically small proportion uncertain and a few marginally PAG. An ML/ARD management plan will be developed to mitigate the potential adverse effects from waste rock on the receiving environment to include periodic static testing undertaken though the life of mine to correctly identify PAG / non-PAG material, and the placement of more reactive materials in central areas of the WSF, to be encapsulated by the less reactive waste rocks through the course of operations.

The project will impact environmental and social receptors in a variety of ways, including positive impacts from the socio-economic benefits at a local, regional and national level. Construction and Operation of the Project will result in large magnitude impacts to surface water quantity, quality (in the Takutu River) and associated aquatic habitats and ecology, groundwater quantity, terrestrial habitats, soil and noise. There will be lower magnitude impacts to air, cultural heritage and other social receptors in the area of influence. All of these impacts have mitigations identified to reduce the magnitude such that residual impact significance is rated negligible or minor for most impacts, although some of these are only realized following

the end of operations as the Mine Closure Plan is implemented. Key to applying and measuring the effectiveness of mitigation is the ESMMP which collates all the monitoring and management activities and commitments for the different environmental media and manages the relationship with the local community and other stakeholders.

The Mine Closure Plan for the FS has been updated from that created for the PEA and features progressive closure of the WSF during later operations, a pit lake after operations, and transfer of ownership of the access roads, airstrip and barge landing and other useful assets. The closure of the TSF is, at this stage, uncertain and may require security provisions and water treatment in perpetuity. Consequently, a large contingency has been included in the total closure provision of $38.7M.

Key residual impacts and uncertainties at this stage are the site water management and the long-term closure of the TSF outlined above. In the current water management design and understanding of geochemical sources, discharge from the sediment pond and flocculant treatment facilities into the north tributary of the Takutu River is expected to increase concentrations for chemical constituents, with some, including but not limited to mercury and arsenic, exceeding IFC water quality guidelines. The discharge volumes are also modelled to exceed capacity in the same tributary. Further pond sizing or additional water treatment infrastructure may need to be considered in future detailed design phases.

Aside from these uncertainties, ERM is of the opinion the baseline information collected is sufficient to inform project permitting and the impact assessment undertaken has not identified any material issues of concern that would deem a project not viable in this location.

The EIA was formally submitted to Guyana's EPA in late December 2024 and remains under review. Public consultation meetings were held in January and February 2025 in local communities, providing critical input for the development of environmental and social programs aligned with regional sustainability priorities. GMIN is currently finalizing responses to all requests for clarification and supplementary information, which are expected to be submitted to the EPA by May 2025. Final environmental approval and the construction permit is expected in Q2 2025.

Also, in mid-December 2024, GMIN received an Interim Environmental Permit ("IEP") from the EPA authorizing the commencement of early works construction activities. The Company is advancing key supporting infrastructures to support full construction, including the installation of water and sewage treatment systems, camp, access roads and wharf area for logistics.

In parallel, GMIN has initiated applications for other key regulatory authorizations required for the Project's implementation, including the Mining Licence, port operation, permits for fuel use and storage, and

approvals for the installation of transmission and telecommunications towers. These complementary permits will support full-scale construction and operational readiness. All permitting efforts are guided by proactive stakeholder engagement and adherence to international environmental and social performance standards.

1.21 Capital and Operating Costs

1.21.1 Capital Costs

Life-of-mine project capital costs are estimated to total $1.7 billion consisting of the following three (3) distinct phases:

- Initial Capital Expenditure – This phase includes all costs to develop the property with a process plant designed to nominally treat 7 Mtpa blend of saprolite, transition and fresh rock. Initial capital costs total $1,041 million (including $85 million for contingency and excluding $69 million in pre-production revenue), which will be expended over 32-month of engineering, construction, pre-production and commissioning period.
- Sustaining Capital Costs – This phase includes all costs related to the acquisition, replacement, or major overhaul of assets required to sustain operations and the underground development. The plant will be configured to treat 100% fresh rock at 6 Mtpa. Sustaining capital costs are estimated to be $650 million including indirect costs but excluding contingency.
- Closure Costs – This phase includes all costs related to the closure, reclamation, and ongoing monitoring of the mine after operations. Closure costs $39 million and includes a 20% contingency.

The capital cost estimate is according to AACEI standard Class 3 and is accurate to a -5% / +10% range. The base date of the CAPEX estimate is H1-2025. The initial capital expenditure (“CAPEX”) duration is planned over a period of 34 months, assumed from March 2025 to end of December 2027 which includes three (3) months of commissioning in the last quarter of 2027. The CAPEX estimate is aligned with an owner-managed project delivery model. The initial CAPEX estimated is presented in US dollars and no allowance for escalation or exchange rate fluctuation were used.

Sustaining capital is required for various reasons. For the open pit mine, it is necessary to purchase new equipment and major components for the equipment. For the underground mine, sustaining capital includes all development work and the required equipment. Sustaining capital is required during operations for additional equipment purchases for the mine. Additional work is required for raising the main embankment of the tailings storage facility (TSF). The continued raising of the TSF will be completed by the mine

operations team with fill material from nearby barrow areas. The sustaining capital is estimated at $ 650M. Table 1.3 shows the capital expenditures.

Table 1.3: Capital Expenditures Summary

Capital Expenditures (k $)	Initial Capital Cost	Sustaining Capital Cost	Closure Cost	Total Capital Cost
100 – Infrastructure	105,889	13,404		119,293
200 – Power and Electrical	110,797	28,258		139,054
300 – Water Management	22,882	14,012		36,893
400 – Surface Operations	41,051	-		41,051
500 – Mining	104,078	575,085		679,162
600 – Process Plant	191,639	1,655		193,295
700 – Construction Indirect	134,534	-		134,534
800 – General Services / Owner's Cost	137,385	-		137,385
900 – Pre-production, Start-up, Comm.	108,173	17,589		125,762
990 – Contingency	84,962	-		84,962
Closure Cost			38,696	38,696
Total	**1,041,389**	**650,003**	**38,696**	**1,730,088**

Reclamation and closure costs include infrastructure decommissioning, site shaping and revegetation, maintenance and post closure monitoring. The total reclamation and closure cost is estimated at $38.7M.

1.21.2 Operating Costs

The OPEX includes mining, processing, power, general services and administration (G&A), transportation, refining and royalties. Power costs are separated from processing costs. The average OPEX is $ 958/oz Au or $ 54.34/t milled over the LOM. The all-in-sustaining costs (AISC) which includes closure, reclamation and sustaining capital costs averages $ 1,123/oz Au or $ 63.70/t milled (Table 1.4).

Table 1.4: Opex Costs

Operating Costs	$/t milled	$/oz
Open Pit Mining Cost	13.81	243
Underground Mining Cost	10.34	182
Processing Cost	7.24	128
Power Cost	7.95	140
General & Administration Cost	5.49	97
Refining Cost	0.45	8
Total Site Cost	**45.29**	**798**
Royalty Cost	9.05	160
Total Operating Costs	**54.34**	**958**
Sustaining Capital	8.56	151
Closure Cost	0.51	9
Land Payments	0.29	5
All-In Sustaining Costs (AISC)	**63.70**	**1,123**

1.22 Economic Analysis

The economic analysis was conducted using a 5% discount rate. Cash flows were discounted from the start of construction, all costs before this period were considered as sunk costs.

The main economic metrics used to evaluate the Project consist of net undiscounted after-tax cash flow, net discounted after-tax cash flow or NPV, IRR and payback period. A 5% discount rate is commonly used as the base case for gold projects.

A summary of the Project economic results is presented in Table 1.5. The total after-tax undiscounted cash flow over the Project life is $3,768M, and the after-tax NPV 5% is $2,163M. The after-tax Project cash flow results in a 2.9-year payback period from the commencement of commercial operations with an IRR of 31.1% before tax and 27.1% after tax.

Table 1.5: Project Economic Results Summary

Technical Report Feasibility Study Update Life-of-Mine Results		
Gold Price – Base Case	$/oz	2,500
Mine Life (operation years)	Mt	12.3
OP Mill Feed Tonnage	Mt	62.37
UG Mill Feed Tonnage	Mt	14.18
Total Ore Mined	***Mt***	***76.56***
Total Waste Mined (OP and UG)	**Mt**	***429.35***
Total Tonnage Mined (OP and UG)	**Mt**	***505.91***
Open Pit Strip Ratio	***waste: ore***	***6.83***
Average Milling Throughput	**Mtpa**	**6.17**
Average Milling Throughput	**tpd**	***16,911***
Gold Head Grade	g/t	1.89
OP Head Grade	g/t	1.57
UG Head Grade	g/t	3.26
Contained Gold	koz	4,642
Average Gold Recovery (%)	%	93.5%
Total Gold Production	koz	4,340
Average Annual Gold Production	koz	350
Operating Costs (LOM Average)		
Open Pit Mining Cost	$/t milled	13.81
Underground Mining Cost	$/t milled	10.34
Processing Cost	$/t milled	7.24
Power Cost	$/t milled	7.95
General & Administration Cost	$/t milled	5.49
Refining Cost	$/t milled	0.45
Total Site Costs	**$/t milled**	**45.29**
Total Site Costs	$/oz	798
Government Royalties	$/oz	160
Total Operating Cost	$/oz	958
AISC	**$/oz**	**1,123**
Capital Costs		
Initial Capital Costs	*$MM*	956
Contingency	*$MM*	85
Total Initial Capital Cost	***$MM***	**1,041**
UG Capital Costs (Sustaining Capital)	*$MM*	68
OP and UG Sustaining Capital	$MM	522
Life of Mine Sustaining Capital	**$MM**	**650**
Closure Costs	$MM	39
Total Capital and Closure Costs	**$MM**	**1,730**
Financial Evaluation		
After-Tax NPV 5%	$MM	2,163
After-Tax IRR	%	27.1%
Payback	years	2.9

A sensitivity analysis was conducted on the base case pre-tax and after-tax Cash Flow, NPV (5%), IRR and payback of the project using the following variables:

- Metal prices
- Operating cost
- Initial capital cost

Table 1.6 to Table 1.8 summarize pre-tax and after-tax sensitivity analysis.

Table 1.6: Gold Price Sensitivity

	Pre-Tax				After-Tax			
Gold Price ($/oz)	**NPV 0% ($M)**	**NPV 5% ($M)**	**IRR**	**Payback Period (y)**	**NPV 0% ($M)**	**NPV 5% ($M)**	**IRR**	**Payback Period (y)**
1,400	(2,005)	(1,685)	0%	16.0	(2,005)	(1,685)	0%	16.0
1,600	(1,031)	(1,044)	0%	16.0	(1,031)	(1,044)	0%	16.0
1,800	73	(317)	1%	11.7	73	(317)	1%	11.7
2,000	1,307	495	11%	7.5	1,018	337	9%	7.6
2,200	2,671	1,393	19%	4.5	2,045	1,025	17%	4.7
2,400	4,165	2,377	27%	3.1	3,169	1,768	24%	3.4
2,500	4,961	2,901	31%	2.5	3,768	2,163	27%	2.9
2,600	5,789	3,446	35%	2.1	4,391	2,573	30%	2.5
2,800	7,543	4,600	42%	1.7	5,711	3,443	37%	1.9
3,000	9,426	5,840	50%	1.3	7,129	4,376	43%	1.6
3,200	11,440	7,166	57%	1.1	8,645	5,374	49%	1.3
3,400	13,583	8,577	64%	0.9	10,258	6,436	55%	1.1
3,600	15,856	10,074	71%	0.8	11,969	7,563	61%	1.0

Table 1.7: Opex Sensitivity

	Pre-Tax				After-Tax			
OPEX	NPV 0% ($M)	NPV 5% ($M)	IRR	Payback Period (y)	NPV 0% ($M)	NPV 5% ($M)	IRR	Payback Period (y)
80%	5,645	3,351	34%	2.3	4,281	2,501	30%	2.6
90%	5,303	3,126	33%	2.4	4,025	2,332	28%	2.7
Base Case	4,961	2,901	31%	2.5	3,768	2,163	27%	2.9
110%	4,619	2,675	30%	2.7	3,512	1,994	26%	3.1
120%	4,277	2,450	28%	2.9	3,255	1,824	24%	3.2

Table 1.8: Initial Capex Sensitivity

	Pre-Tax				After-Tax			
Capex	NPV 0% ($M)	NPV 5% ($M)	IRR	Payback Period (y)	NPV 0% ($M)	NPV 5% ($M)	IRR	Payback Period (y)
80%	5,169	3,098	37%	1.9	3,924	2,318	32%	2.2
90%	5,065	2,999	34%	2.2	3,846	2,240	30%	2.6
Base Case	4,961	2,901	31%	2.5	3,768	2,163	27%	2.9
110%	4,857	2,802	29%	2.9	3,690	2,085	25%	3.2
120%	4,753	2,703	27%	3.2	3,612	2,007	23%	3.4

1.23 Adjacent Properties

According to the Guyana Geology and Mines Commission, the Oko West Prospecting Licence is surrounded by 13 medium-scale mining and prospecting permits held by various Guyanese title holders, and one group of medium-scale mining and prospecting permits controlled by G2 Goldfields, namely the Oko Gold Project.

A mineral resource estimate was produced by Micon Ltd. and published on March 10, 2025, for the Oko Gold Project. The MRE considers the "Oko Main" deposit, as well as a newly defined area south of the Oko Main called the "Ghanie" Zone. Open pit and Underground Indicated Mineral Resources are estimated at 3,147 kt, grading 7.98 g/t Au for 808 koz Au for the Oko Main zone and 10,288 kt, grading 2.01 g/t Au for 663.4 koz Au for the Ghanie zone. Open pit and Underground Inferred Mineral Resources are estimated

at 3,473 kt, grading 4.60 g/t Au for 513,5 koz Au for the Oko Main zone and 12,062 kt, grading 2.64 g/t Au for 1,024.5 koz Au for the Ghanie zone. An additional open pit Inferred resource estimated at 4,976 kt, grading 0.61 g/t Au for 97.2 koz Au was defined for the Oko Northwest zone.

The QP has been unable to verify the information on the adjacent property and the information provided herein is not necessarily indicative of the mineralization on the Oko West Project.

1.24 Other Relevant Data and Information

The schedule for the Oko Gold Project spans from late August 2024 to the end of December 2027, covering over three (3) years of multidisciplinary project development. It includes a comprehensive permitting phase, starting with early works authorizations and progressing to environmental, and construction permits through agencies such as EPA, GGMC, GFC and MARAD—running mostly through 2025 to 2026. Concurrently, procurement activities and early site infrastructure (roads, camp facilities, wharf, and utilities) are launched. Major infrastructure components such as the process plant, power generation, water and sewage systems, and mining fleet mobilization are gradually implemented through overlapping workstreams during the entire period of 2025. Construction peaks between 2026 and 2027, followed by a commissioning phase starting in Q3 2027 and commercial production starting in January 2028.

1.25 Interpretation and Conclusions

This Technical Report is prepared in accordance with the guidelines of the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101") and Form 43-101F1. The objective of this FS Report is the evaluation of the technical feasibility and economic viability of the Project, notably the development of an open pit and underground mine thereat, including processing facilities and related infrastructures. This NI 43-101 Technical Report confirms the technical feasibility and economic viability based on an open pit mining and underground operation with average gold production at 350 koz per year over a 12.3-year life-of-mine ("LOM"). It is recommended to advance the Project to the Detailed Engineering and Construction and Development phase.

1.26 Recommendations

Following the results of the financial analysis of this Feasibility Study (FS), which demonstrates a positive project economics, GMS to proceed with the Detailed engineering for the project and start the Early works as an advanced phase prior to the subsequent full construction release.

Table 1.9 summarizes the proposed budget of $41.6M to advance the project in the Detail engineering stage and perform the recommendations discussed in this section.

Table 1.9: Cost Estimate Associated with Recommendations

Description	Amount ($000)
Geology extensions and splay model testing	8,000
Mining Engineering – OP West Slope Improvement	300
OP design optimization	200
Geotechnical Testing and Break Dam	100
Environment – Biodiversity and Cave Recovery	200
Detailed Engineering	29,000
Contingency at 10%	3,780
Total	**41,580**

2. INTRODUCTION

G Mining Ventures Corp. (“GMIN” or the “Company”) mandated G Mining Services Inc. (“GMS”) as the lead consultant along with other engineering consultants to prepare a Feasibility Study (“FS”) under the supervision of the Qualified Persons “QPs” for the Oko West Project (“GYOW” or “Project”) located in the Cuyuni-Mazaruni Mining Districts, Guyana.

It was prepared in accordance with the guidelines of the Canadian Securities Administrators’ National Instrument 43-101 (“NI 43-101”) and Form 43-101F1. The objective of this report and the FS is the evaluation of the technical feasibility and economic viability of the Project, notably the development of an open pit and underground mine, including processing facilities and related infrastructures. It is based on, for the most part, a drilling database dated September 15, 2024, and the supporting geological information. The report contains all technical information pertaining to the previous exploration of the Project, drilling methods, sampling and QA/QC protocols, data verification undertaken by the Qualified Person for this Technical Report, and metallurgical test work program results. This Report provides operating and capital cost estimations, and an economic analysis of the Project.

The intention of this Technical Report is to provide sufficient, clear and unambiguous technical and scientific information relating to the Project available at the effective date of the report. The qualified persons (“QP”) understand that a copy of this Report will be filed with the Canadian Securities Commissions and be publicly available.

As of the effective date of this report, the QPs are not aware of any known litigation potentially affecting the Project. The QPs did not verify the legality or terms of any underlying agreement(s) that may exist concerning the Project ownership, permits, off-take agreements, licence agreements, royalties or other agreement(s) between GMIN and any third parties

GMIN is a gold mining company focusing on acquisition, exploration, and development of precious metal projects. Its flagship property is the Tocantinzinho Gold Project located in the State of Pará, Brazil. The Company’s common shares trade on the Toronto Stock Exchange (TSX: GMIN) and the OTC markets (OTCQX: GMINF).

On April 22, 2024, G Mining TZ Corp. (formerly G Mining Ventures Corp.) (“GMIN”), Reunion Gold Corporation (“Reunion Gold”) and Greenheart Gold Inc. (formerly 15963982 Canada Inc.) (“Greenheart Gold”, and collectively with GMIN and Reunion Gold, the “Parties”), entered into an arrangement agreement, which was subsequently amended effective June 7, 2024, setting forth the terms and conditions on which the Parties agreed to complete a plan of arrangement under Section 192 of the Canada Business

Corporations Act (the "Arrangement"). Pursuant to the Arrangement, the successor issuer, New GMIN, has acquired (i) all of the issued and outstanding common shares in the capital of GMIN, and (ii) all of the issued and outstanding common shares in the capital of Reunion Gold, thereby combining the business of these two entities.

2.1 Scope of Work

All sections of this Technical Report, excluding Section 20 – Environmental Studies, have been prepared by G Mining Services Inc., a mining consulting firm based out of Brossard, Québec, Canada. The QPs are entirely independent of the issuer (G Mining Ventures Corp.) as described in Section 1.5 of the NI 43-101 standard of disclosure for mineral projects. The QPs involved in the mandate do not hold an interest in the issuer or its related entities. The relationship between GMIN and GMS is solely professional, and GMS is being compensated based on a commercial-fee basis that is not contingent on the results presented in this Technical Report.

The QPs responsible for each section of the Technical Report are mentioned in Table 2.1.

Table 2.1: Summary of Qualified Persons

Qualified Person	Company	Title	Report Sections
Christian Beaulieu, P.Geo. (OGQ 01072)	Minéralis Services-Conseils Inc.	Mineral Resource Consultant	1.4, 1.5, 1.6, 1.7, 1.8, 1.9, 1.10, 1.11, 1.12, 1.14, 1.23, 1.26, 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 23, 25.2, 26.1 (also contributed to 1.25, 1.26, 2, 3, 25.11 and 27)
Neil Lincoln, P.Eng. (PEO 100039153)	Lincoln Metallurgical Inc.	Consulting Metallurgist	1.13, 1.17, part 1.21, 13, 17, 21.4.2, 25.5, 25.6
Alexandre Burelle, P.Eng. (OIQ 5019855)	Evomine Consulting Inc.	Mine Planning and Financial Analysis Consultant	1.15, 1.16, 1.19, 1.21.2, 1.22, 15, 16 (except 16.2 and 16.3), 19, 21.1.8, 21.4 (except 21.4.2), 22, 25.3, 25.4, 25.9, 25.10
Kevin Leahy, PhD, C.Geol.	Environmental Resources Management (ERM)	Associate Partner	1.20 1.25, 1.26, 16.3.2, 20, 21.3, 25.8, 26.4
Paul Murphy, P.Eng. (OIQ 43320)	Independent Consultant	Project Manager	1.1, 1.2, 1.3, 1.18, 1.21.1, 1.24, 1.25, 3, 16.3.1, 18 (except 18.6 and 18.7), 21.1, 21.2, 24, 25.1, 25.10, 26.5, 27
Sebastien Guido, P.Eng., M. Sc. (OIQ 5067847)	Alius Mine Consulting Inc.	Senior Engineer, Rock Mechanics	16.2, 26.2

Qualified Person	Company	Title	Report Sections
Paulo Ricardo Behrens da Franca, P.Eng.	F&Z Consultoria e Projetos.	Director	18.6, 18.7, 26.3

2.2 Sources of Information and Data

Unless otherwise stated, all the information and data pertaining to the Mineral Resource Estimate contained in the Report or used in its preparation has been provided by GMIN up to September 15, 2024. The abovenamed QPs have no reason to doubt the reliability of the information provided.

Sources of information include:

- Discussions with GMS and GMIN personnel.
- Inspection of the Oko West Project area, including drill collars, trenches, drill core, and ground conditions.
- Inspection of the principal and secondary laboratories.
- Drilling database received from independent, third-party database manager (gDat Solutions).
- Geological Interpretations, provided by GMIN.
- Exploration data, compiled and provided by GMIN.
- QPs' personal inspection of the Oko West Project
- Metallurgical test works results.
- Report on mineralogical, metallurgical and grindability characteristics of the Oko West project
- Technical and scientific reports by external consultants.
- All figures and tables cited using references in Section 27.
- A conceptual process flowsheet developed by GMS based on the specific Project testwork and similar operations

2.2.1 Previous Technical Reports

- Preliminary Economic Assessment NI 43-101 Technical Report Oko West Project, G Mining Services, September 4, 2024, October 11, 2024.
- NI 43-101 Technical Report Oko West Project Cuyuni-Mazaruni Mining Districts, Guyana, G Mining Services, Effective date February 26, 2024, Issue Date: April 11, 2024.

- NI 43-101 Technical Report Oko West Project Cuyuni-Mazaruni Mining Districts, Guyana Property, G Mining Services, Effective Date: June 1, 2023, Issue Date: July 14, 2023.
- NI 43-101 Technical Report Oko West Project Cuyuni Mining District, Guyana, G Mining Services, Effective Date: December 31, 2022, Issue Date: March 20, 2023.

2.2.2 Agreements, Mineral Tenure, Surface Rights and Royalties

The issuer provided details regarding mining titles, royalty agreements, environmental liabilities, mineral agreement and permits. The QPs are not qualified to offer any legal opinion on property titles, ownership, or potential litigation.

2.3 Site Visit

In accordance with NI 43-101 regulations, a current personal inspection was completed by the below mentioned QP to the Oko West Project as part of the data validation process.

Table 2.2: Site Visit Dates of Qualified Person

Qualified Person	Site Visit Scope	Dates
Christian Beaulieu, P.Geo.	Geology and Resources	April 18 to 19, 2023, and March 11 to March 12, 2025
Neil Lincoln, P.Eng.	Did not visit the site	N/A
Alexandre Burelle, P.Eng.	Mining	March 11 to March 12, 2025
Kevin Leahy, PhD, C.Geol.	Did not visit the site	N/A
Paul Murphy, P.Eng.	Infrastructure	March 31 to April 7, 2025
Paulo Ricardo Behrens da Franca, P.Eng.	Tailings dam and waste pile sites	February 18 to 20, 2025
Sebastien Guido P.Eng., M. Sc.	Mine Geotechnics	April 12 to April 18, 2025

2.4 Effective Date

The FS is derived using the Company's mineral resources estimate effective as September 15, 2024 (the "MRE"). The effective date of the FS report is April 28, 2025.

The issue date of the Technical Report is June 06, 2025.

2.5 Units of Measure, Abbreviations and Nomenclature

The units of measure presented in this Technical Report, unless noted otherwise, are in the metric system. All dollar figures quoted in this report refer to United States dollars (USD, US$ or $), unless otherwise noted. A list of the main abbreviations and terms used throughout this Technical Report is presented in Table 2.3. Unless otherwise specified, the source for tables and figures is GMS, 2024. All ounce units are reported in troy ounces.

Table 2.3: Table of Abbreviations

Abbreviations	Full Description
AA	Atomic-Absorption
Ag	Silver
As	Arsenic
Au	Gold
C	Carbon
CAD	Canadian Dollar
CIL	Carbon-in-leach
CoG	Cut-off Grade
CRM	Certified Reference Material
Cu	Copper
DD	Diamond Drilling
DGPS	Differential Global Positioning System
EIA	Environmental Impact Assessment
ESIA	Environmental and Social Impact Assessment
ESMMP	Environmental and Social Management and Monitoring Plan
FA	Fire Assay
Fe	Iron
FS	Feasibility Study
g	Gram
gpt or g/t	Grams per tonne
g/L	Gram per litre
GMIN	G Mining Ventures Corp.
GMIN Guyana	GMIN Ventures Guyana Inc. (previously known as Reunion Gold Inc.)

Abbreviations	Full Description
GMS	G Mining Services Inc.
gpm	Gallons per minute (US)
GPS	Global Positioning System
GFC	Guyana Forestry Commission
GGMC	Guyana Geology & Mines Commission
GCAA	Guyana Civil Aviation Authority
GLSC	Guyana Lands and Surveys Commission
GPL	Guyana Power and Light Inc
GGDMA	Guyana Gold & Diamond Miners Association
GGB	Guyana Gold Board
ha	Hectares
h (hr)	Hour
h/d	Hours per day
h/y	Hours per year
h/wk	Hours per week
ISO	International Organization for Standardization
kg	Kilograms
kg/t	Kilograms per tonne
km	Kilometre
km/h	Kilometre per hour
L	Litre
M	Mega or Millions (000,000s)
m	Metre
m^2	Square metre
m^3	Cubic metre
mg	Milligram
mm	Millimetre
ml	Millilitre
min	Minute
MRE	Mineral Resource Estimate
Mt	Million tonnes

Abbreviations	Full Description
Mtpa	Million tonnes per annum
MONR	Ministry of Natural Resources
NI 43-101	National Instruments 43-101- Canadian Standards of Disclosure for Mineral Projects
NQ	Drill Core Diameter (47.6 mm)
oz	Troy Ounce (31.10348 grams)
PEA	Preliminary Economic Assessment
PFS	Pre-feasibility Study
Pb	Lead
ppb	Parts per Billion
ppm	Parts per Million
PTCCB	Pesticide and Toxic Chemicals Control Board
QP	Qualified Person
RC	Reverse Circulation
S	Sulphur
SD	Standard Deviation
Sec	Second (time)
t	Tonnes (1,000 kg) (metric ton)
t/y or tpy	Tonnes per year
t/d or tpd	Tonnes per day
t/h or tph	Tonnes per hour
t/m^3	Tonnes per cubic metre
USD (US$)	United States Dollar
wk	Week
XRF	X-ray Fluorescence
y	Year

3. RELIANCE ON OTHER EXPERTS

This Technical Report has been prepared by GMS, under the supervision of the QPs, for GMIN. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to GMS at the time of the preparation of this Report.
- Assumptions, conditions, and qualifications as set forth in this Report.
- Data, reports, and opinions supplied by GMIN and other third-party sources.

The QPs of this Technical Report believe that the basic assumptions contained in the information indicated above are factual and accurate and that the interpretations are reasonable. The QPs of this Technical Report have, to the extent applicable, relied on this data and have no reason to believe that any material facts have been withheld. The QPs of this Technical Report have taken all appropriate steps, in their professional judgement, to ensure that the work, information, or advice from the above information is sound and the QPs do not disclaim any responsibility for this Technical Report.

The QPs have not performed an independent verification of the land title and tenure information, as summarized in Section 4 of this Technical Report, nor have they verified the legality of any underlying agreement(s) that may exist concerning the permits or other agreement(s) between third parties, as summarized in Section 4 of this technical report. For this topic, the QPs of this Report have relied on information provided by the legal department of GMIN. The QPs have relied on GMIN for guidance on cost allocation and applicable taxes, royalties, and other government levies or interests, applicable to revenue or expenditures from the Oko West Project, as described in Sections 4, 19, 20 and 21.

The authors wish to emphasize that they are QPs only in respect of the areas in this Technical Report identified in their “Certificates of Qualified Persons” submitted with this Technical Report to the Canadian Securities Administrators. Except for the purposes legislated under Canadian provincial and territorial securities law, any other use of this Technical Report by any third party is at the party’s sole risk.

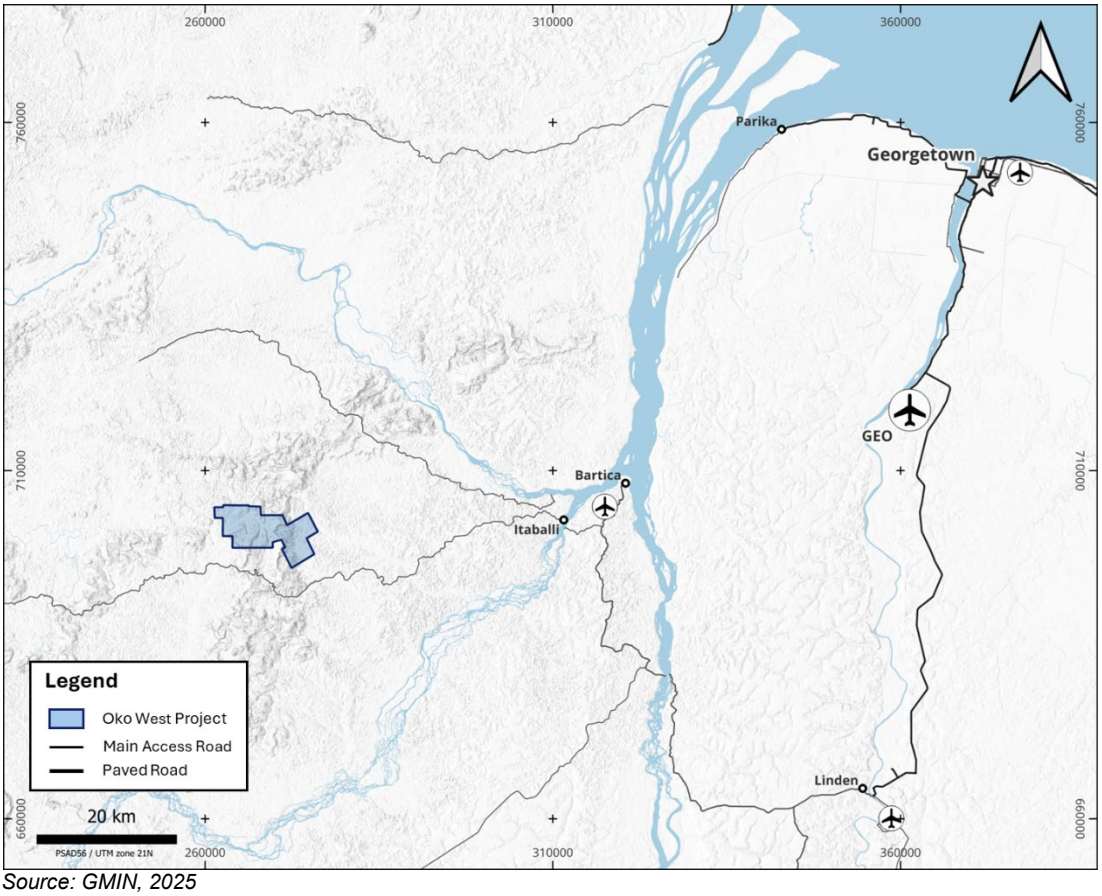
4. PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Oko West Project (the "Project") straddles the Cuyuni-Mazaruni Mining Districts (administrative Region 7) in northcentral Guyana, South America. The Project is located approximately 120 km southwest of Georgetown, the capital city of the Co-operative Republic of Guyana (hereinafter, "Guyana") and approximately 50 km from Bartica, the capital city of Region 7 (Figure 4.1). Bartica is accessible by a 20-minute direct flight from the Ogle airport in Georgetown or by road and boat from Parika on the Essequibo River. There are regular boat services between Bartica and Parika.

The Project is accessible by the Puruni and Aremu laterite roads from the town of Itaballi at the confluence of the Cuyuni and Mazaruni rivers (Figure 4.1). Several trails reach the Project area from these two (2) roads. The Project is also accessible by helicopter; the helicopter pad at the campsite is located at the coordinates 6°20´54.5" N and 59°03´13.8" W.

Figure 4.1: Oko West Project Location and Access

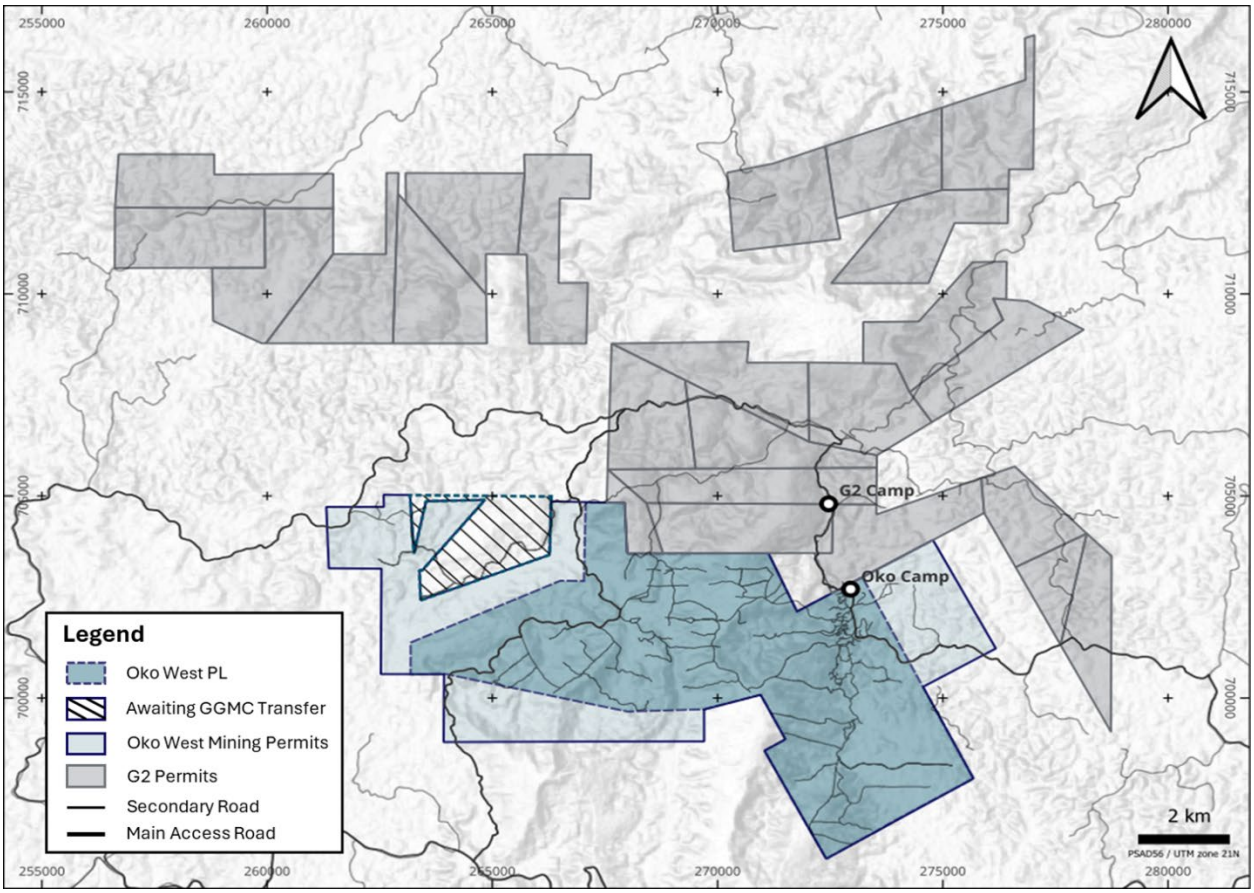


Source: GMIN, 2025

4.2 Property Description and Title

The Project comprises one Prospecting Licence ("PL 004/2022" or "GMIN's PL") issued to GMIN Ventures Guyana Inc. (previously known as Reunion Gold Inc. and hereinafter, "GMIN Guyana"), GMIN's 100%-indirectly owned Guyanese subsidiary, on September 23, 2022. GMIN's PL is valid for three (3) years and is renewable twice for a period of one (1) year each time. GMIN's PL has a surface area of approximately 10,890 acres (4,407 hectares) (Figure 4.2). In 2024, additional land has been added to the Project as medium scale permits that will be converted to a Mining Licence. The hashed zone shown in Figure 4.2 is currently being transferred to GMIN Guyana by the Guyana Geology and Mines Commission ("GGMC").

Figure 4.2: Oko West Prospecting Licence

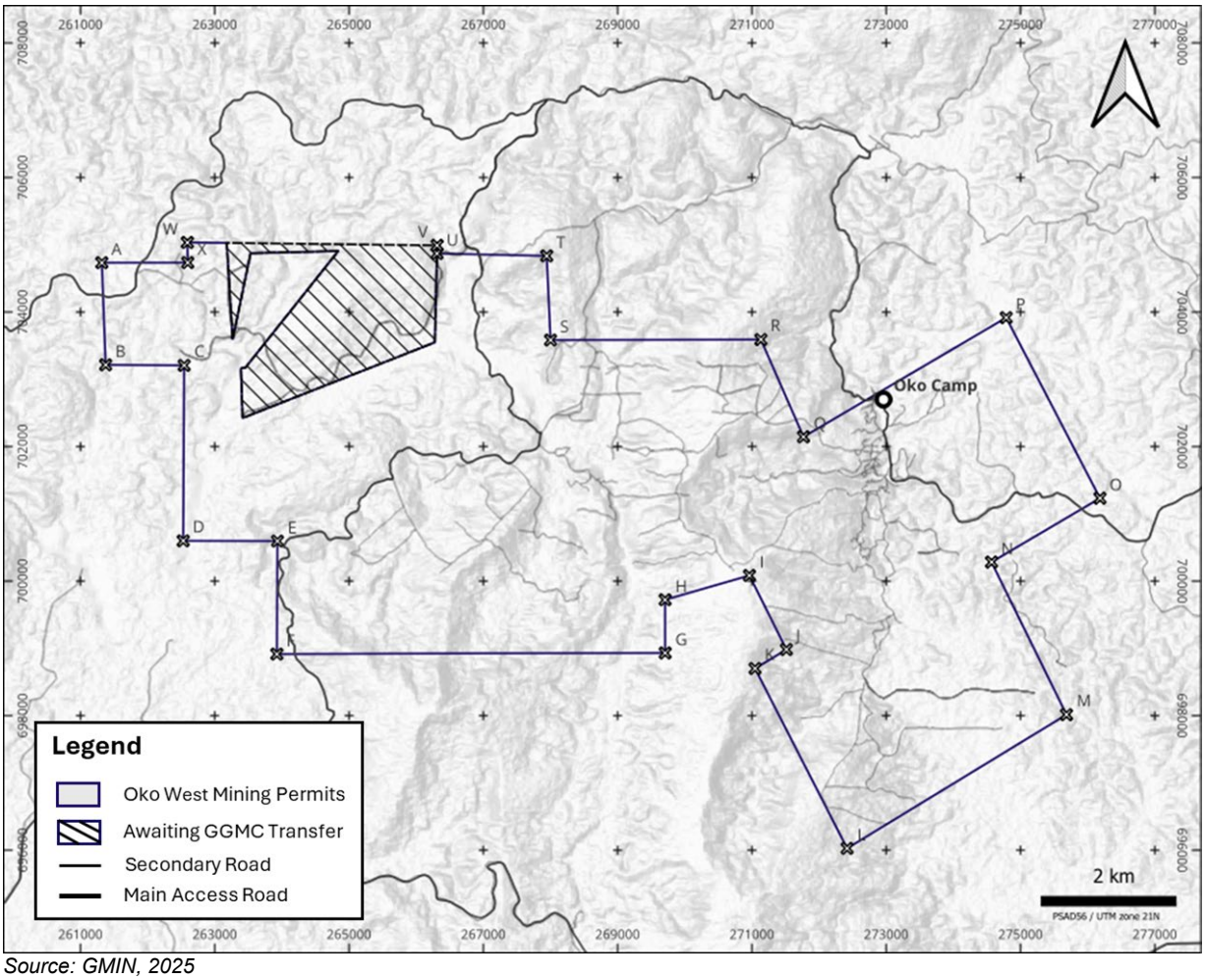


Source: GMIN, 2025

4.3 Legal Surveys

The Project's boundaries and corners are described in PL 004/2022 by geographic coordinates (latitude and longitude). These boundaries and corners have not been surveyed in the field. Table 4.1 lists the coordinates for each PL corner and Figure 4.3 shows the location of each corner.

Figure 4.3: Outline of the Oko West Property



Source: GMIN, 2025

Table 4.1: Coordinates Defining the Oko West Prospecting Licence

Coordinate ID	PSAD 56 UTM Zone 21N	
	Easting	Northing
A	261315.99	704731.99
B	261372.99	703212.99
C	262541.99	703203.99
D	262527.99	700594.99
E	263932.99	700591.99
F	263923.69	698911.12
G	269708.99	698927.99
H	269707.99	699718.99
I	270963.99	700079.99
J	271514.99	698978.99
K	271047.99	698695.99
L	272419.99	696021.99
M	275687.99	698009.99
N	274570.99	700278.99
O	276186.99	701228.99
P	274789.99	703913.99
Q	271769.99	702141.99
R	271134.99	703587.99
S	268000.99	703580.99
T	267943.99	704832.99
U	266309.99	704868.99
V	266311.99	704989.99
W	262589.99	705032.99
X	262595.99	704732.99

4.4 Oko West Mineral Tenure and Requirements

Mining in Guyana is managed by the GGMC under the Mining Act of 1989 (the "Act"). The Act establishes that the State is the owner of all subsurface mineral rights in Guyana and authorizes the GGMC to manage these resources.

The Mining Act allows for three (3) scales of operation: small, medium, and large-scale permits or licences. Small-scale claims, medium-scale prospecting permits ("PPMS") and medium-scale mining permits ("MSMP") can only be issued to Guyanese citizens and partnerships, cooperatives or companies beneficially owned by Guyanese citizens. Foreign companies can enter into joint venture arrangements with local permitholders.

Large-scale prospecting licences ("PL") and MPMS can be issued to Guyanese citizens as well as Guyanese and foreign companies. PLs cover areas between 500 and 12,800 acres. A PL grants an exclusive right of occupation and exploration within the area covered by it. PLs are valid for a period of three (3) years with two (2) rights of renewal of one (1) year each.

GMIN Guyana, as holder of PL 004/2022, has the exclusive right of occupation over the area (10,890 acres) covered by it for the purpose of exploring for gold. Rental rates are US$0.50 per acre for the first year; US$0.60 per acre for the second year, and US$1.00 for the third year. Pursuant to PL 004/2022, GMIN Guyana must submit to the GGMC semi-annual and annual progress reports as well as annual audited financial statements. The work commitment during the first year of PL 004/2022 is US$240,000, and a performance bond of 10% of that amount has been provided. Three (3) months prior to each anniversary date of GMIN's PL, a work program and budget for the following year must be presented for approval for the work to be undertaken during the following year.

In October 2022, GMIN Guyana applied with the Environmental Protection Agency of Guyana ("EPA") to obtain an Environmental Authorization ("EA") to conduct exploration activities on GMIN's PL. EPA representatives visited the Project and issued a letter confirming that they have no objection to GMIN Guyana continuing its exploration activities. This is discussed further in Section 20.

At any time during the term of a PL, its holder may apply for a mining licence for any part or all of the PL area. In connection with the application, the following will need to be submitted: a technical and economic feasibility study, a mine plan, an Environmental Impact Statement, and an Environmental Management Plan. Mining licences are usually granted for twenty years or the life of the mine for which it is granted, whichever is shorter; renewals are possible. GMS and the QP have reviewed the documentation

relating to the grant of PL 004/2022 as well as proof of payment of rental fees and the performance bond, and they are satisfied with the authenticity and compliance of the foregoing.

4.5 Oko West Project Ownership and Agreements

GMIN Guyana is the 100% registered and beneficial owner of the Project and PL 004/2022.

Pursuant to an option agreement with a Guyanese citizen (Optionor) entered into on August 14, 2018, (amended as of August 10, 2020, and January 11, 2021), GMIN Guyana became entitled to conduct exploration activities over the area comprising nine (9) MPMS held by the Optionor. This area covers approximately 9,425 acres representing 86.5% of the total Project area (the "Optioned Area"). Under this agreement, GMIN Guyana had an option to acquire a 100% interest in the mineral rights over the Optioned Area for an initial period of five (5) years. On February 23, 2023, GMIN Guyana exercised that option.

Should the Project go into commercial production, the Optionor will be entitled to receive a contingent consideration of US$5.00 per ounce of gold produced from the Optioned Area. A first payment will be due within 30 days after commencement of commercial production on all ounces included in the feasibility study mining plan. An additional US$5.00 per ounce produced above the amount in the feasibility study will be payable quarterly thereafter.

On March 19, 2024, GMIN Guyana entered into an additional option agreement with the same Optionor for the Northwest extension mining permits, consisting of three (3) MPMS, adjacent to PL 004/2022, which are valid for a period of five (5) years with a possible 2-year extension. An initial cash payment has been made to exercise this option, and this additional agreement provides for the same contingent consideration, *i.e.*, a first payment that will be due within 30 days after commencement of commercial production on all ounces included in a feasibility study mining plan, plus an additional US$5.00 per ounce produced above the amount that is determined in such mine plan will be payable quarterly thereafter. The NW extension provides for additional prospective exploration ground.

GMIN Guyana also entered into, on October 20, 2020 (as modified on November 2, 2020), an agreement to conduct exploration activities and acquire mineral rights over two (2) other MPMS held by another individual titleholder. These two (2) MPMS cover an area of approximately 1,465 acres and, subsequent to their purchase, were made part of the Prospecting Licence 004/2022. The consideration payable to the seller for such purchase, was paid in full.

GMIN Guyana purchased five (5) MPMS on December 6, 2024, from a private group of individuals for the Eastern and Southern extensions to PL 004/2022 at a cost of $4.3M. An amount equal to $40.00 per ounce of gold recovered from the area covered by these five (5) MPMS is payable to the sellers. However, these

two (2) property extensions are currently intended to allow for a more optimal project layout facilitating the positioning of the waste rock dump and tailings storage facilities.

Together, the combined surface area of PL 004/2022, of the aforementioned MPMS and of the area covered by NW option agreement result in a total surface area of 71 km^2.

4.6 Surface Rights

While the Guyana Government holds the surface rights to the PL 004/2022 area, such PL allows GMIN Guyana to occupy the area covered by it.

4.7 Royalties and Other Encumbrances

The Government of Guyana imposes a royalty of 8% on gold produced from large-scale mines within the country. The Mineral Agreement signed as of April 17, 2024, between the Government of Guyana, the GGMC and GMIN Guyana Inc. establishes an 8% of Net Smelter Returns (as defined therein) royalty applicable to open pit gold production and a 3% of Net Smelter Returns royalty from underground gold production. Also, reference must be made to the contingent consideration described in Section 4.4. There are no other known royalties, back-in rights, payments, or encumbrances on the Project.

4.8 Environmental Liabilities

There are no known environmental liabilities associated with the Project.

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

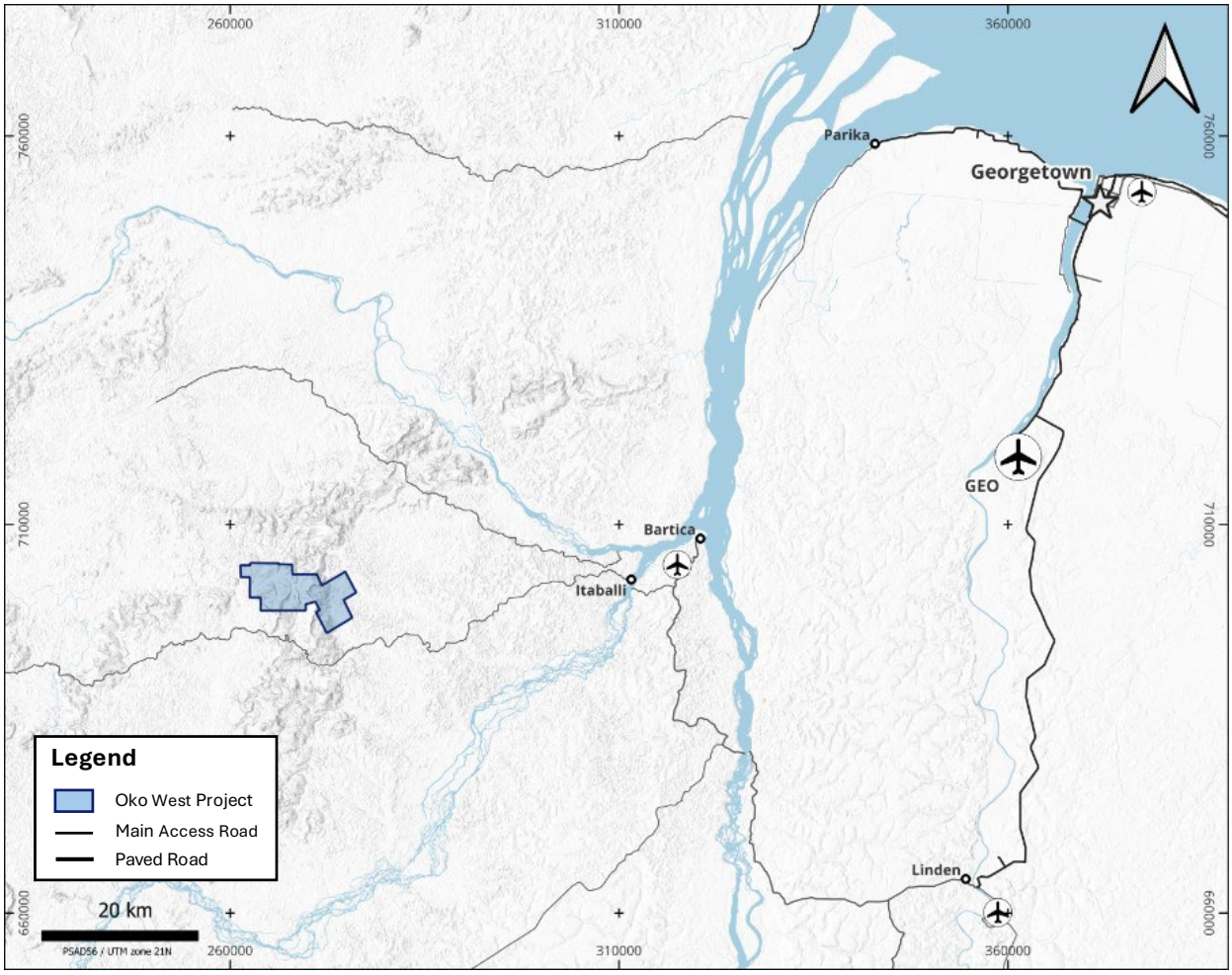
5.1 Accessibility and Roads

From the city of Georgetown, the project area can be accessed as follows (Figure 5.1 and Figure 5.2):

- **Helicopter:** A direct flight from Ogle Airport to the helicopter pad at the campsite's, lasting approximately 45 minutes.
- **Fixed-Wing Plane and Car:** A 20-minute fixed-wing flight from Ogle Airport to the Bartica airstrip, followed by a car ride to Itaballi, crossing the Mazaruni River by ferry and a 1.5-hour car ride to site. Once the airstrip at the project site is completed, it will reduce the access to a 35-minute flight between Ogle and the Project.
- **Car and Speedboat:** A 1-hour drive from Georgetown to Parika (at the mouth of the Essequibo River), followed by a 1-hour river boat ride to Itaballi, and concluding with a 1.5-hour car ride to the project site.
- **Four-Wheel Drive Vehicle:** An 8-hour journey from Georgetown to the project site, involving a paved drive to Linden, followed by laterite roads to Itaballi, including two ferry crossings. The duration may vary depending on road conditions.

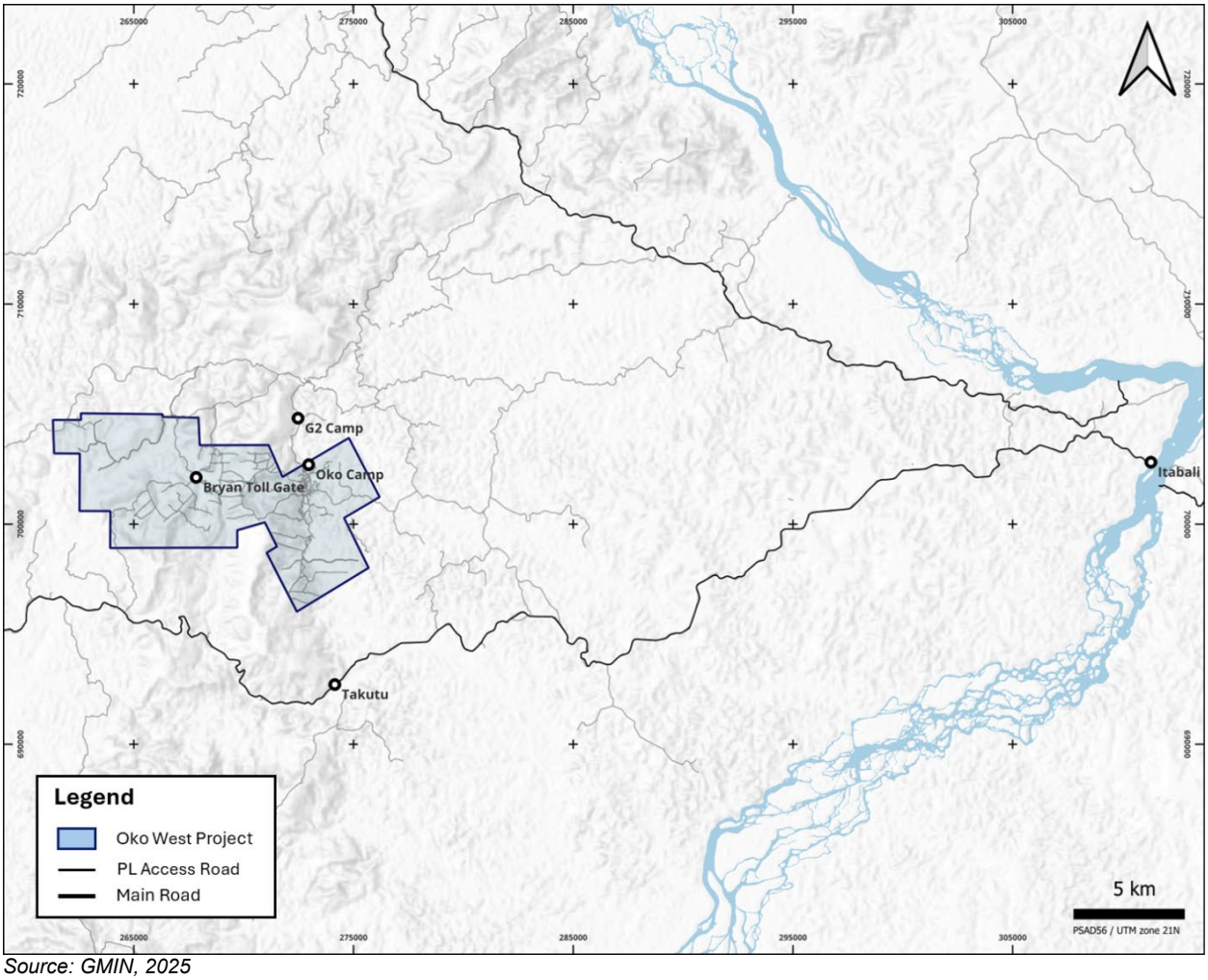
From Itaballi to the project site, one can use the Puruni or the Aremu laterite roads, requiring four-wheel drive vehicles. The Project has been using the Puruni road due to its better condition and proximity to the site. The Puruni road has been recently maintained to service the Toroparu Project, designed for mine development and copper concentrate export. A light four-wheel drive vehicle can reach the Oko West Project campsite in about three (3) hours in good weather conditions. The vehicle will follow the Puruni road until Kilometre 65 at Lion Mountain and past the mining community of Takutu (with Digicel cell tower). From this turnoff, a private road leads to Bryan's camp, with a toll controlling access to passage (6 km) to the Oko West campsite (Figure 5.2). The Oko West campsite can also be reached by another trail running north, passing through the communities of Sand Hill and Oko Landing. Alternatively, the campsite can also be reached from the Puruni road via the Duckpond trail, an old logging road, although the road condition is poor and often unpassable.

Figure 5.1: Location Map of Major Access Roads and Topographic Features



Source: GMIN, 2025

Figure 5.2: Location Map of Local Roads in the Project Area



Source: GMIN, 2025

5.2 Climate

Guyana falls under the Köppen climate classification as "Af". The climate is equatorial and humid, with two (2) dry seasons, one from approximately March to mid-April and the other from August to November. The dry seasons' onset and duration vary from year to year. The heaviest precipitation is expected in May and June. The average yearly temperature is about 26°C; in the interior regions of Guyana, one can expect typical daily highs around 34°C to 36°C and typical nighttime lows of 16°C to 18°C.

GMIN installed a weather station at the Oko West campsite, and this data is available for the environmental baseline study of climate. The Guyana Hydrometeorological Service operates several regional meteorological stations with extensive historical climate data available.

The Oko West Project has operated throughout the year without any interruption related to weather. Laterite road conditions deteriorate significantly during the rainy seasons and might cause transportation delays.

5.3 Local Resources

There are limited local resources available for the project:

- Aggregate made from the Bartica gneiss and with good specifications for concrete, currently produced at the Bartica quarry, 60 km to the east and on the other side of the Mazaruni river.
- Aggregate is also produced at a quarry at a landing along the Cuyuni River near GMIN's planned landing.
- High-quality hardwood is available for construction and foundation pylons.
- Duricrust is available for laterite road surfacing or capping.

5.4 Infrastructure

The region's infrastructure is in development, lacking power, roads, communications, and general services.

The city of Bartica (population of about 10,000), at the Essequibo, Mazaruni and Cuyuni rivers' confluence, is the primary hub for artisanal mining activity in northwest Guyana, often called the "gateway to the interior". Its main industrial activity is a quarry providing aggregate and boulders (for sea wall construction) for the entire western Guyana, shipped by barges to Parika. It houses the government administrative offices for Region 7 and has basic commercial, transportation and manufacturing facilities. It is linked to Parika by regular river speed boats transporting people and cargo. It has a hospital, an elementary-level school, and cell phone communication.

The town of Itaballi, at the left of the Mazaruni river, is the gateway for the road transportation of goods and services to all the artisanal mining operations not reachable by a river. It is also where wood logs harvested in the region get loaded into barges for transportation to the sawmills in Parika. The Cuyuni landing is the logical place to install docking facilities to service the mining operation at Oko West.

5.4.1 Services Buildings and Ancillary Facilities

The existing infrastructure at the Oko West Project includes campsite facilities designed to support mineral exploration activities and environmental studies on the property. Figure 5.3 illustrates the facilities on-site as of November 2023. The campsite has been upgraded to accommodate more people and provide better facilities. Figure 5.4 shows a recent aerial view of the camp.

Figure 5.3: Overhead View of the Oko West Camp Showing Available Facilities

Oko-West Camp Layout

Q2 / 24

1-a, 1-b	Security Check point / Planned South Gate	13-a	Ablution on platform A	25	Power generating station (3 gensets)	38	Planned admin and engineering office
2	Security house / dormitory	14-a,14-b	Recreational center and laundry	26	Swingfog and pesticide shed	39	Ablution and septic toilets (C-North)
3-a, 3-b	Drilling contractor product storage and workshop	15-b	Female staff and visitors' house (septic toilet)	27	LV Parking	40	Planned UTV garage
4	Water well: 70m deep, pump at 46m, 14 m³/hour	15-a	Male visitors' house (septic toilet)	28	Old gym	41	Site clinic annex (D&A testing)
5	Water storage and treatment: 20,000 L	16-a,16-b	Canteen and kitchen	29	Latrines (2 outhouses)		
6	Diesel farm: 2 x 22,000 L	17-b	RC sample duplicates storage	30	Surface water well		
7	Helipad	18-a, 18-b	Core stores A and B	31-a, 31-b	Core store C and core laydown area		
8-a, b, c	RGI mech. workshop, carpentry, LV service bay	19-a, b, c	Core logging shacks	33	Recycled PET storage		
9	Dormitories C1–C5, B1–B7, A1-A2	20	Core cutting facility (3 saws)	34	Drilling products storage		
10	Medical / First Aid center	21, 22	Warehouse, temporary admin office	35	Senior staff and guest house (septic)		
11	Gym expansion	23	New gasoline dispensing station	36-a, 36-b	Ablution and septic toilets		
12	Exploration office and IT / satcom	24-a, 24-b	ATV garage	37-a, 37-b	Recycling area; Garbage pits		

Source: GMIN, 2024; North to the right of the image.

Figure 5.4: Drone View of the Oko West Camp, Looking South

Source: GMIN, 2024; Looking south.

5.4.2 Power Supply and Distribution

There is no available electrical power grid in the region. The entire Guyana power system currently runs on heavy fuel or diesel thermal plants installed along the coast and at Linden and Bartica. There are no power line or substation in the project vicinity.

ExxonMobil is on track to deliver natural gas from its offshore Guyana operations to the mainland in 2025. This Gas-to-Energy (GtE) Project aims to construct an Integrated Liquid Natural Gas Liquid (LNG) plant, and a 300-megawatt (MW) combined cycle power plant at Wales, West Bank Demerara (WBD). The project will utilize natural gas from the Stabroek Block's Liza Phase 1 and 2 Projects. The pipeline, which is 12 inches in diameter, will cover approximately 200 km from offshore to onshore facilities. It is expected to transport about 50 million standard cubic feet per day (mscfpd) of dry gas, with a maximum capacity of 120 mscfpd. There are plans to produce Liquefied Natural Gas (LNG) for export which could be a future source of fuel for the Oko West power plant.

5.4.3 Water

The project area lies in the water divide between the Cuyuni river to the north and the Mazaruni river to the south. Alternatively, the project could harvest water from local creeks and store it in a reservoir. Once the tailings storage facility will be constructed, it will serve to capture water and build up an initial starter reservoir to feed the process plant and will recycle water back to the process plant during operations. The ongoing environmental baseline information will provide essential information on the local hydrology.

5.5 Physiography

5.5.1 Vegetation

The entire project region is covered by what Granville (1988) called an "upland moist forest". This equatorial evergreen forest type is the most common and floristically richest forest formation of the Guianas, found extensively on undulating terrain and well-drained ferralic and sandy soils. Their general characteristics are the presence of a high and dense canopy at 20-45 m and emergent trees up to 50-60 m. They have enormous species diversity, between 100 and 300 tree species per hectare, showing a correlation between higher rainfall and increasing diversity. The forests on higher elevations and over duricrust appear less species-rich and have a lower canopy, many lianas, and a scrubby undergrowth. The baseline environmental study provided essential flora information for the area.

5.5.2 Topography

The Oko West permit area is underlain by a belt of greenstone rocks composed mostly of mafic volcanics. The weathering of these rocks released iron, forming pisolitic concretions near the paleosurfaces. Once the water table stopped reaching the upper levels of the lateritic profile, these pisolitic concretions coalesced, forming concrete-hard duricrusts. This weathering phenomenon explains the local topography, where duricrusts hold higher elevations. The highest point in the project area, at 340 m, corresponds to a duricrust surface on the access road to the campsite (Figure 5.2). The eastern flank of this north-south ridge hosts the Oko West gold mineralization, with an average elevation of 130 m. The terrain drops to the east, featuring felsic intrusive rocks with a composition low in iron, which precludes the formation of duricrust. This results in the lowest point along Kairuni Creek, reaching 66 meters in elevation. The campsite has an average elevation of about 80 m. Further east, the topography is characterized by rolling hills over the Oko granitic pluton with an average elevation of about 100 m.

6. HISTORY

6.1 Prior and Current Ownership

GMIN Ventures Guyana Inc. is a Guyanese subsidiary wholly owned by G Mining Ventures Corp (GMIN) following the merger with Reunion Gold Corporation (RGD).

G Mining Ventures Corp. (“GMIN”) (TSX: GMIN) (OTCQX: GMINF) and Reunion Gold Corporation (“RGD”) (TSXV:RGD) (OTCQX:RGDFF) announced entering into a definitive agreement to combine the two companies on April 22, 2024. The merger via a plan of arrangement was overwhelmingly approved by GMIN shareholders as well as Reunion Gold security holders at their respective annual general and special meetings held on July 9, 2024, and was subsequently approved by the Ontario Superior Court of Justice (Commercial List) on July 11, 2024.

G Mining Ventures Guyana Inc. currently holds a 100% interest in the Oko West Project and Prospecting Licence (PL). This PL was issued on September 23, 2022, by the Guyana Geology and Mines Commission (GGMC), following the relinquishment of 11 medium-scale mining permits held by two (2) Guyanese entrepreneurs with whom Reunion had signed option agreements (see Section 4 for more details). These entrepreneurs had acquired some of these permits from other local parties and registered them to their names. The previous permit holders produced an unknown quantity of gold in an artisanal basis, primarily by alluvial mining.

6.2 Exploration History

6.2.1 Historical Exploration

The discovery of gold in the region dates back to the end of the 19th century by artisanal miners. In 1935, Dr. Grantham, from the British geological survey, described mining activity in the region. British Guiana Goldfields Ltd. recognized the area's potential and extended the road from Kartabu Puruni road to Bird Page on the Oko River, establishing a trading outpost in 1938. In 1939, it drilled some 400 Banka holes in the Oko River area and 110 holes in the Kairuni river area, a tributary of the Oko River. Despite encouraging results, they abandoned the Oko area and focused on the more profitable veins at the Aremu mine area to the northwest (Narain, 1985).

In the mid-60s, Aero Mineral Surveys Ltd., a Canadian company funded by the United Nations, flew a regional magnetic and radiometric survey over northern Guyana. From 1966 to 1979, McDonald, Schielly and Garson, working for the British Geological Survey, conducted regional field mapping and

produced compilation maps of the region. The geophysical data was a major impetus for further work, which included an investigation by Schielly in 1969 of an area with magnetic anomalies on the northern border of the project area. He produced two (2) separate maps, one showing geology and structure, and one interpretative map assessing the regional stress fields (Narain, 1985).

In 2002, the Guyana Geology and Mines Commission (GGMC) conducted the Lower Puruni Regional Geochemistry Programme, which covered the project area, identifying gold and molybdenum anomalies from stream sediment samples. Between 2010 and 2015, extensive alluvial and eluvial mining was done in the region, where local miners mined several gold-rich quartz veins at Crusher Hill, north of the Oko West Project area. There is still artisanal gold mining activity in the region.

In 2016, Sandy Lake Gold Inc. collected grab samples at Crusher Hill, a primary prospect north of Oko West, reporting high gold grades in shaft stockpiles associated with quartz and quartz-carbonate veins in narrow mineralized zones (Ilieva, 2018). This reconnaissance became the basis for Sandy Lake Gold to enter into an option agreement with Michael Vieira in 2018 for this area. Sandy Lake Gold was renamed to G2 Goldfields Inc. in 2019 and continues to explore its Oko Main zone.

There is no record of the amount of gold production from artisanal mining within the Oko West Project area.

6.2.2 Reunion Gold Exploration

In 2018, the local permit holder, Mr. Bryan Stephens, introduced Reunion Inc. to a number of claims under his control that resulted in a site visit to a mining pit to inspect outcrops and alluvial waste piles. Grab samples were collected on this initial visit and included samples BS-02, BS-03 and BS-04, which assayed 2.40, 6.00 and 45.84 g/t Au, respectively. This visit and the interpretation of a favourable local geological context prompted the option of two (2) groups of medium-scale mining permits held by Mr. Bryan Stephens, Oko West and Oko East. Reunion later relinquished the Oko East block from the agreement and proceeded to explore the Oko West claim block as described elsewhere in this report.

6.3 Historical Drilling

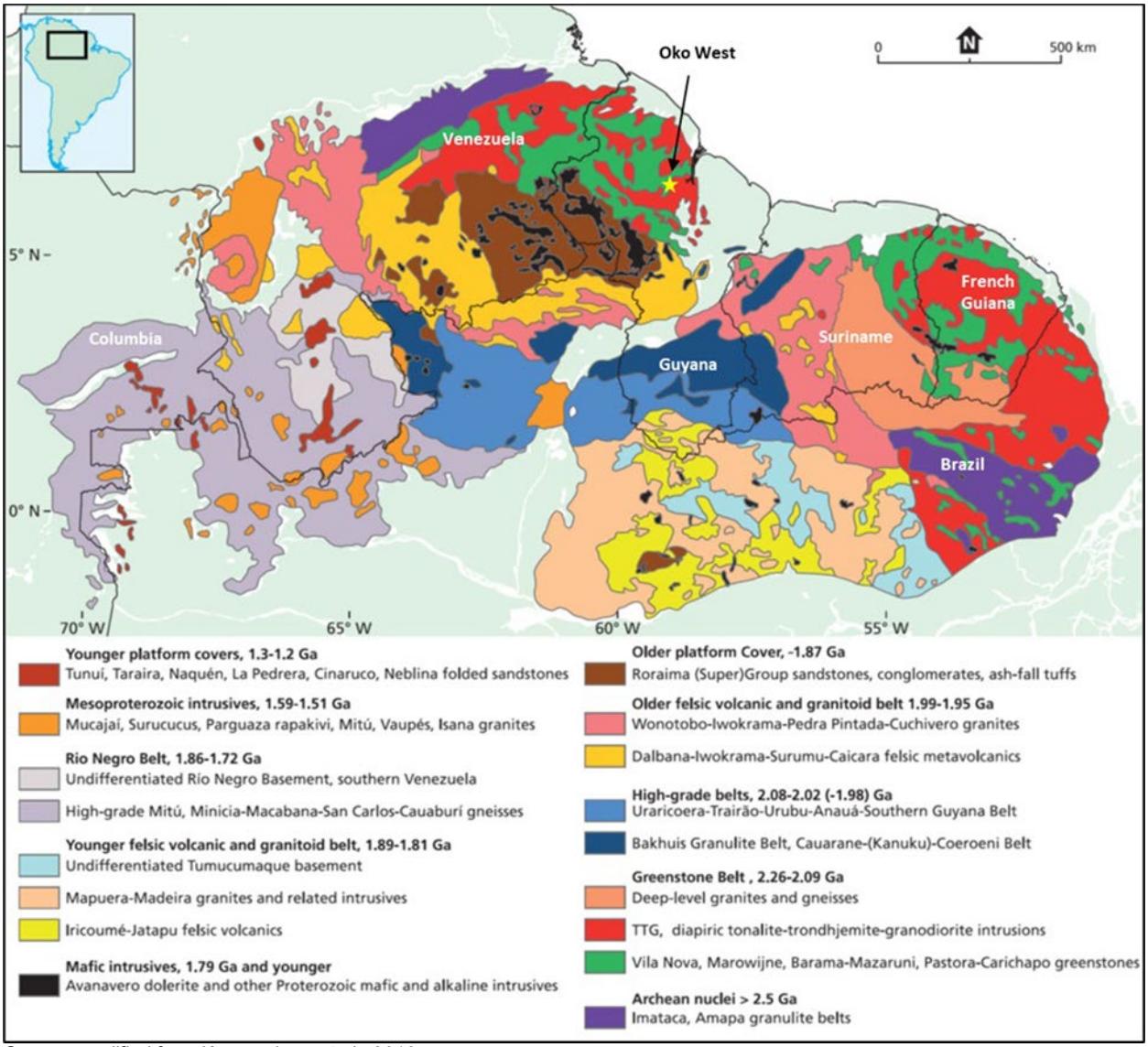
There is no known historical drilling completed within the project boundaries prior to Reunion’s exploration efforts.

7. GEOLOGICAL HISTORY AND MINERALIZATION

7.1 Regional Geology

The Oko West Project is located approximately 110 km west-southwest of Georgetown, Guyana, in the Trans-Amazonian province of the Guiana Shield (Figure 7.1).

Figure 7.1: Simplified Geological Map of the Guiana Shield



Source: modified from Kroonenberg et al., 2016

The Guiana Shield corresponds to the northeastern portion of the Amazonian Craton. With a total area of 900,000 km^2, it covers eastern Venezuela, Guyana, Suriname, French Guiana, the northern end of Brazil, and easternmost Colombia (Daoust 2016; Tedeschi et al. 2020). The Guiana Shield is mainly composed of

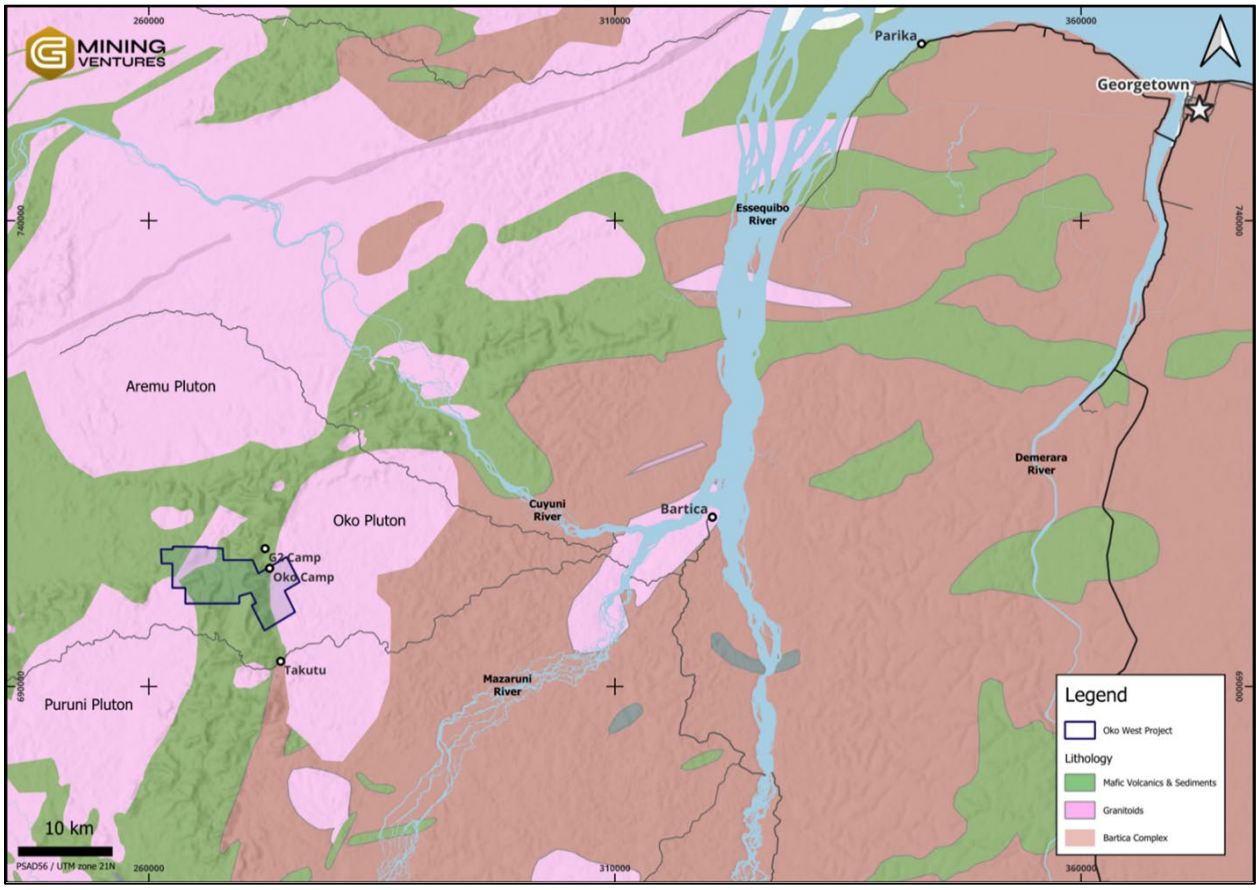
Paleoproterozoic rocks accreted during the Trans-amazonian orogeny (2.2 - 2.0 Ga) and affected by tectonic, metamorphic, and intrusive events (Vanderhaeghe et al. 1998; Milési et al. 2003). Small Archean relics are preserved in eastern Venezuela (Imataca Complex) and northern Brazil (Amapá) (Tassinari et al., 2004; Tedeschi et al., 2020).

The Trans-Amazonian Province is composed of large Rhyacian (2.20 - 2.05 Ga) granite-greenstone belts, including volcano-sedimentary rocks, metamorphosed to greenschist facies, intrusive granitoids, and TTG (tonalite-trondhjemite-granodiorite) gneisses (Vanderhaeghe et al., 1998; Santos et al., 2000; Tedeschi et al. 2018, Tedeschi et al., 2020). In Guyana, the greenstone belts are described from deepest to shallowest as basalt ± ultramafic rocks, intermediate to felsic volcanic rocks, and finally, tuffs and turbiditic sedimentary rocks (Gibbs, 1980; Tedeschi et al., 2020). They host multiple gold deposits; however, little is known about the relationship between gold mineralization, magmatism, and deformation (Tedeschi et al., 2020).

Two (2) major tectonic events, D1 and D2, which took place during the Trans-Amazonian orogeny, have been documented (Ledru et al., 1991; Gibbs and Barron, 1993; Vanderhaeghe et al., 1998; Delor et al., 2003a). The major Trans-Amazonian orogeny begins between 2.26 and 2.20 Ga, forming a juvenile oceanic crust from tholeiitic magmatism. It continues between 2.18 and 2.13 Ga with the first major tectonic event D1, associated with the N-S convergence of the Archean African and Amazonian cratons. Between 2.11 and 2.08 Ga, the convergence of the Archean African and Amazonian cratons evolves towards an oblique NE-SW convergence, with the closure of the volcanic arc basins (Delor et al., 2003b). This sinistral strike-slip regime, defined as the D2a event, is marked by granitic magmatism, minor mafic intrusions, and regional greenschist metamorphism. It also led to the folding of the volcano-sedimentary formations of the greenstone belts and the development of thrust faulting and EW to NW-SE regional sinistral strike-slip shear zones (Vanderhaeghe et al., 1998; Delor et al., 2003a; Tedeschi et al., 2020). The D2b event occurred between 2.07 and 2.06 Ga (Delor et al., 2003a). It is marked by dextral reactivation of WNW ESE strike-slip shear zones and significant crustal thinning, leading to mantle rise and regional high-temperature metamorphism (Delor et al., 2003a; Tedeschi et al., 2020) and peraluminous intrusions.

The D1 intrusive event in the project region is likely represented by the intrusion of three (3) plutons: Oko, adjacent to Oko West, Aremu to the north, and Puruni to the southwest (Figure 7.2). The Barama-Mazaruni-Supergroup volcano-sedimentary rocks (2.12 Ga) caught between these plutons were thoroughly deformed, and local structures host several gold occurrences.

Figure 7.2: Schematic Geology Map of North-Central Guyana

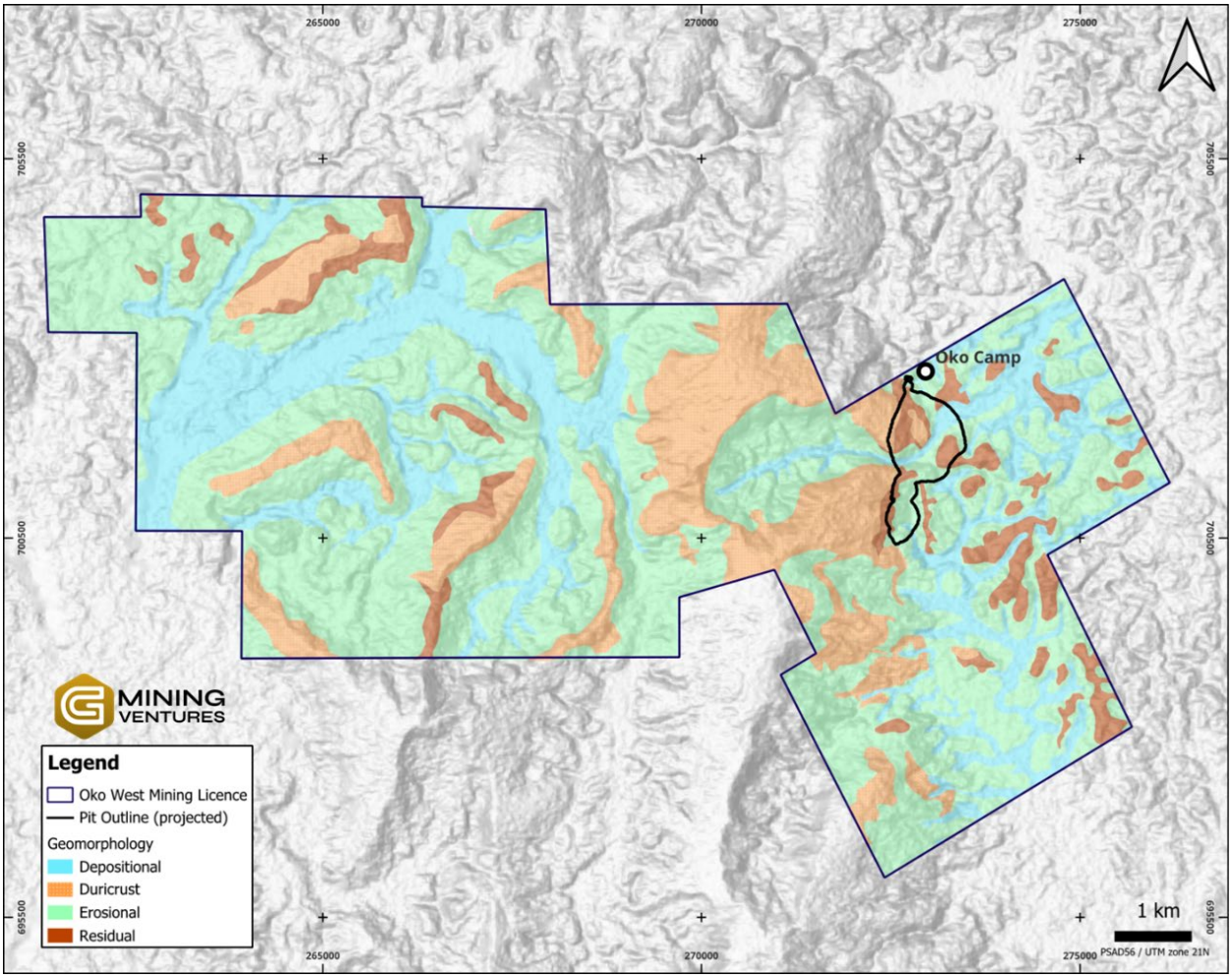


Source: GMIN 2025

7.2 Property Geology

The Oko West Project straddles the north-south striking contact between rocks of the Barama-Mazaruni Supergroup greenstone belt to the west and a granitoid pluton to the east (the Oko pluton). This sequence comprises-mafic volcanic flows, volcaniclastics, and siliciclastic and carbonaceous sediments (Figure 7.3). The following sections describe each of these geological units and their role in gold mineralization.

Figure 7.3: Oko West Permit Simplified Regolith Map



Source: GMIN, 2025

7.2.1 Regolith

Long chemical weathering typical of humid equatorial paleoclimate produced a thick lateritic profile down to a depth below 100 m from the surface. This profile is typically composed of a veneer of pisolitic colluvium and latosols overlaying a massive clay zone, which pass into a mottled zone and then saprolite / saprock before reaching unweathered rocks at depth.

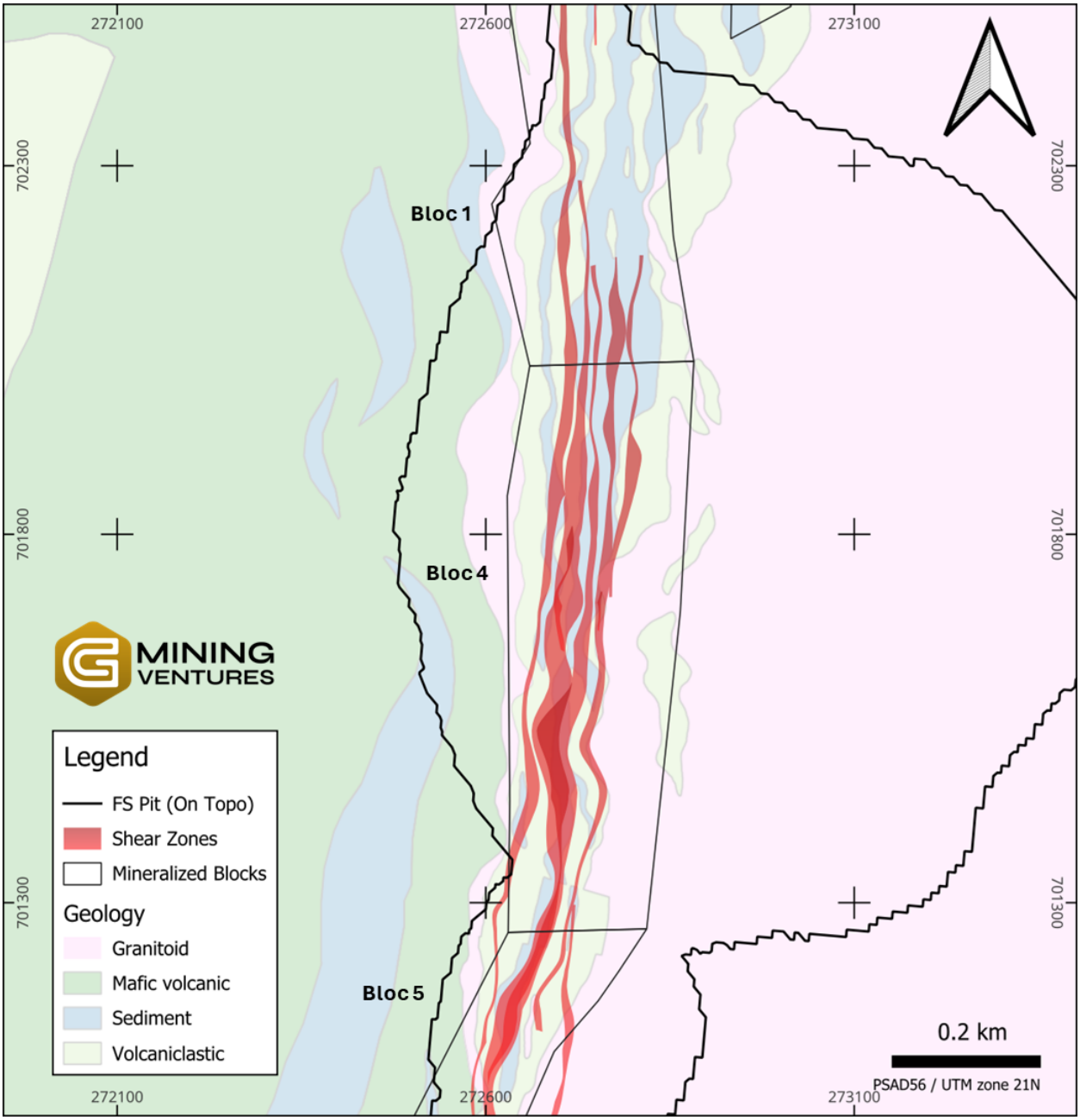
At the topographic highs in the centre of the project area (Figure 7.3), the profile is different; characterized by a superficial duricrust, either in situ or broken, passing into a massive clay zone and then the mottled zone. This duricrust, remains of a peneplain, can be several metres thick and form a protective cap, slowing the erosion of the underlying rock. To the south and at the Puruni road, this ridge is called Lion Mountain. Not coincidentally, the duricrust developed only above areas underlain by mafic volcanic rocks, rich in iron, a key ingredient to form duricrust through water table movement and bedrock leaching.

Gold mineralization at Oko West rocks outcrop on the eastern flank of a north-south trending ridge, covered by a veneer of pisolitic colluvium up to two (2) metres thick and blocks of duricrust that survived erosion. The clay layer and mottled zone below vary in thickness up to three (3) metres and do not allow the recognition of lithologies, although they are considered to be in situ. The saprolite allows the recognition of lithologies and geological structures. The transition zone, or "saprock", to unweathered rocks exhibits partially preserved mineralogy and structural features typical of the protolith. It is noticeable that weathering is deeper overlying gold mineralization, facilitated by the abundance of easily dissolved carbonate minerals and faults and shear zones that allowed weathering to reach greater depths. Overall, the lateritic or weathered profile ranges from 16 m to 118 m over the area (Exploration Blocks 1, 4 and 5), and averages 65 m in thickness. The thickness of the saprolite zone averages 43 m, and the transition zone averages 22 m. The mineralized saprolite and transition zones are entirely preserved and untouched by artisanal miners over the strike length of gold mineralization. Figure 7.4 presents a typical section through outcropping gold mineralization in the saprolite. A geological map outlining the various zones at Kairuni is presented in Figure 7.5.

Figure 7.4: Outcrop of Carbonaceous and Volcanoclastic Sediments Saprolite (Exploration Block 5)

Source: Reunion, 2022. Photo by C. Bertoni

Figure 7.5: Oko West Kairuni Zone Geology and Mineralization Map with Exploration Blocks



Source: GMIN, 2025

7.2.2 Granitoids

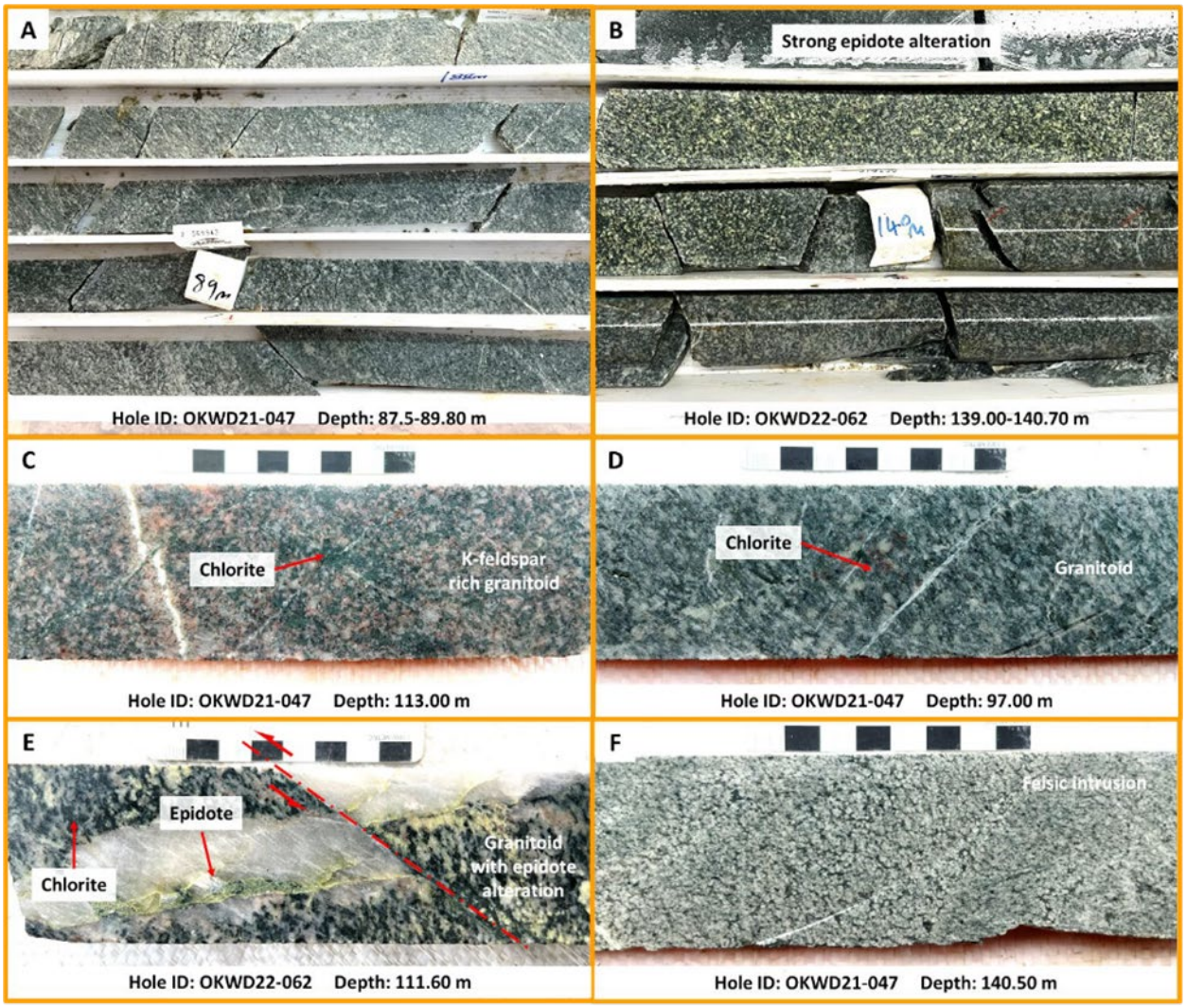
The pluton flanking gold mineralization to the east and acting as its hanging wall does not appear to belong to the Bartica Gneiss complex (Figure 7.5). It is interpreted as a later intrusion of unknown dimensions, being less foliated and lacking evidence of partial melting, hereby called the "Oko pluton" Tedeschi (2022) dated a sample from the "Bryan pit" area at 2,107 ± 6 Ma. The rock is primarily coarse-grained and slightly foliated, described petrographically by Thompson (2022) as a metamorphosed quartz---monzodiorite and

granodiorite (Figure 7.6). At the deposit scale, its contact with the sedimentary sequence is sharp and locally sheared. There is no evidence of pre- or syn-orogenic contact metamorphism. The locally fractured and sheared zones near its contact can be mineralized, but do not represent significant thicknesses.

The volcano-sedimentary sequence shows a strongly sheared contact with a footwall granitoid (see cross-section in Figure 7.7) intersected by drilling in exploration Blocks 1 to 5. The footwall granitoid could either be a faulted sliver of the hanging wall granitoid, or a sill. The footwall granitoid is truncated in Block 6 but appears locally as thin lozenges or fault slices into Block 7 and 8. The footwall granitoid likely played an important role in the development of deposit, providing a strong rheological contrast for the folding of volcano-sedimentary beds, resulting in dilation zones during D2 folding. The shear zone at the contact can be up to 20 m wide, showing even mylonitic textures hosted mainly by the sediments. The footwall granitoid has a sill-like geometry up to 120 m wide, and its composition overlaps the classification boundaries between granodiorite and quartz monzonite. Bent plagioclase, polygonised quartz grains, and multiple chlorite-white mica-rutile-magnetite shear seams are evidence of deformation during lower greenschist regional metamorphism (Thompson, 2022). The western contact of this sill is a mafic volcanic rock extending westwards to the centre of the project area (Figure 7.5 and Figure 7.7). A whole rock analysis by Tedeschi (2022, pers. comm.) reveals that the mafic volcanics have a tholeiitic basalt composition.

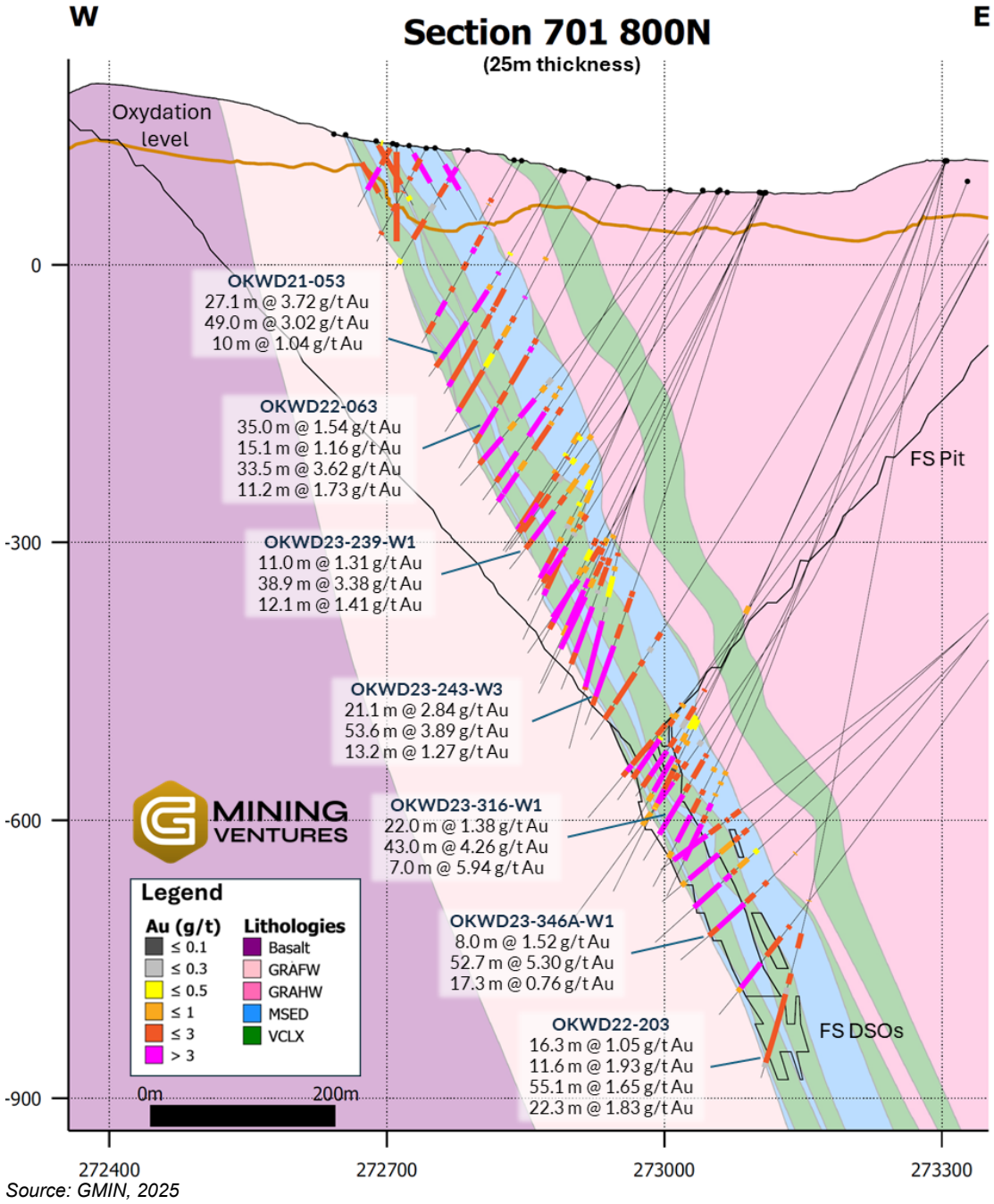
Smaller intrusive bodies (decimetric) with the same granitoid composition and texture can often be observed within the volcano-sedimentary units. Mostly parallel to the bedding, they highlight the preferential path created by slippage during folding. Larger intrusive bodies (decimetric to metric) can also be locally observed at the contact between units with different rheological characteristics, such as massive volcanic rocks and sediments. These larger intrusions are often associated with strong alteration (K-feldspath or epidote assemblages) and metasomatism affecting surrounding volcano-sedimentary rocks.

Figure 7.6: Granitoid Rock Samples



Source: Hainque et al., 2022
**Notes: A: Granitoid in hole D21-047. B: Granitoid showing strong epidote alteration in hole D22-062. C: K-felspar-rich granitoid with strong chloritization in hole D21-047. D: Granitoid with strong chloritization in hole D21-047. E: Granitoid with strong epidotization and evidence of reverse faulting in hole D22-062. F: Felsic intrusive body at the contact between units with different rheological characteristics (carbonaceous sediments and volcaniclastics) in hole D21-047.*

Figure 7.7: Geological Cross-Section 701800 N, Looking North



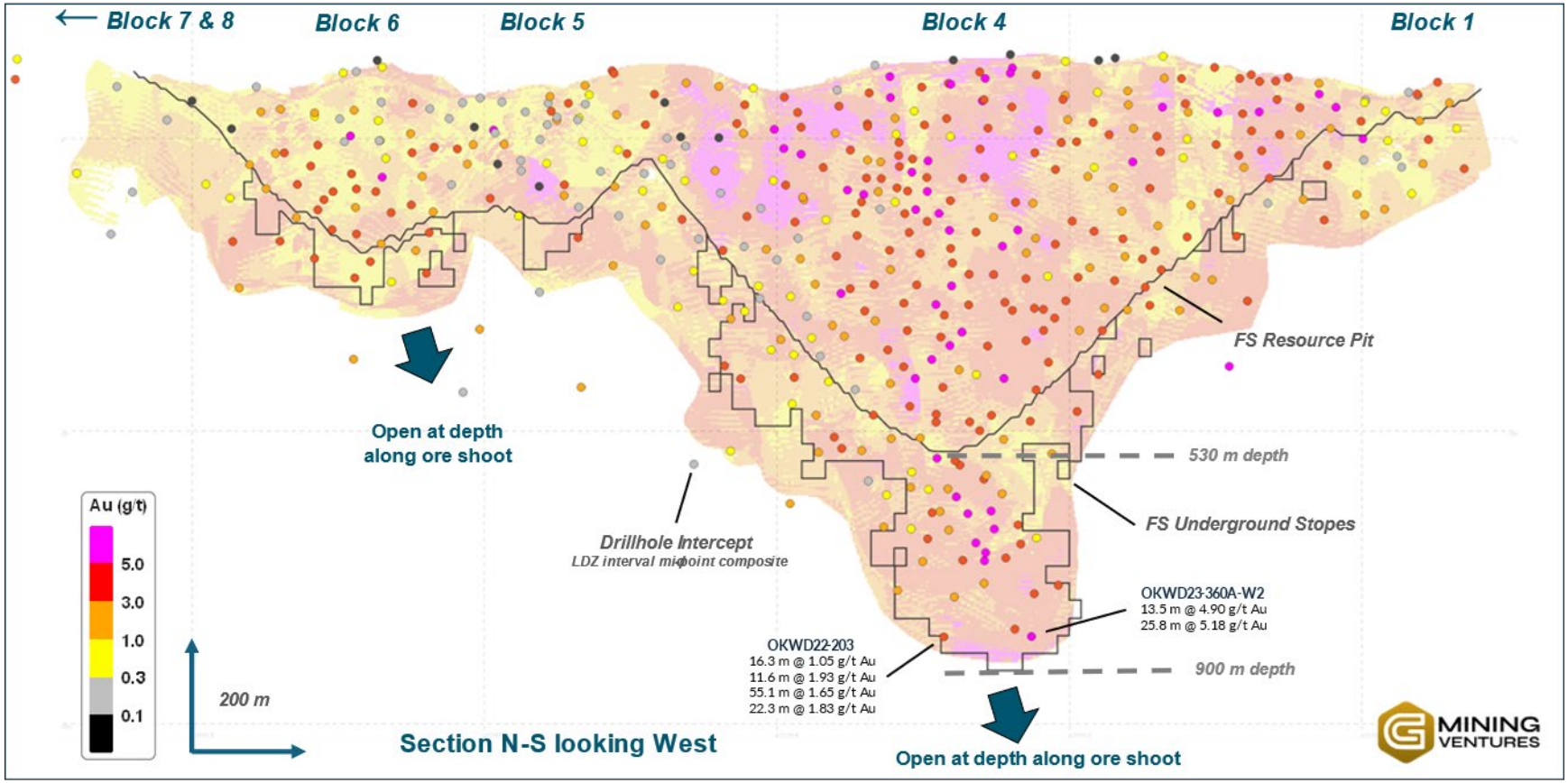
Source: GMIN, 2025

7.2.3 Volcano-Sedimentary Sequence

The sequence hosting the bulk of the Oko West gold mineralization is composed of clastic rocks: siliclastic, volcanoclastic, and carbonaceous, and is better understood in the Kairuni zone, the 2.5 km northernmost extent of known gold mineralization. Some of the volcaniclastics may be coherent volcanics, such as flows and associated hyaloclastites, but this has yet to be established. The sequence is 100 to 200 m wide and

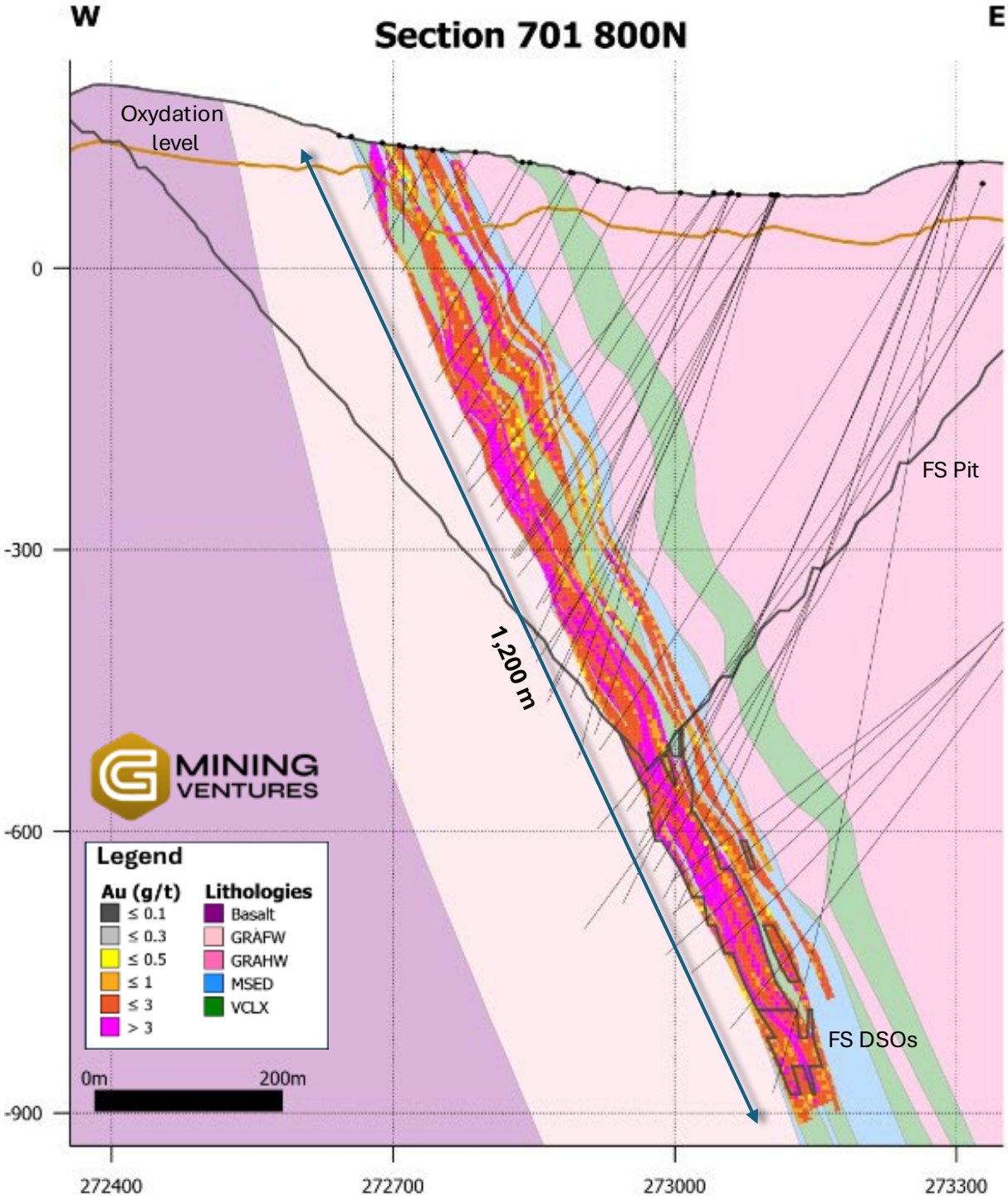
has an overall tabular geometry, dipping steeply to the east and "sandwiched" between the Oko pluton to the east and the footwall granitoid to the west (Figure 7.5). The units are intercalated and strongly deformed. The mineralized sequence has been intercepted by drilling to a depth of 980 m from the surface (drillhole OKWD22-203 and OKWD23-360A-W2) but is known to continue much further down-dip (see longitudinal section in Figure 7.8). This sequence straddles the contact with the Oko pluton for the entire length of the project area within the Kairuni and Takutu zones.

Figure 7.8: Longitudinal Inclined Section Along Mineralized Zone (Blocks 1 to 6)



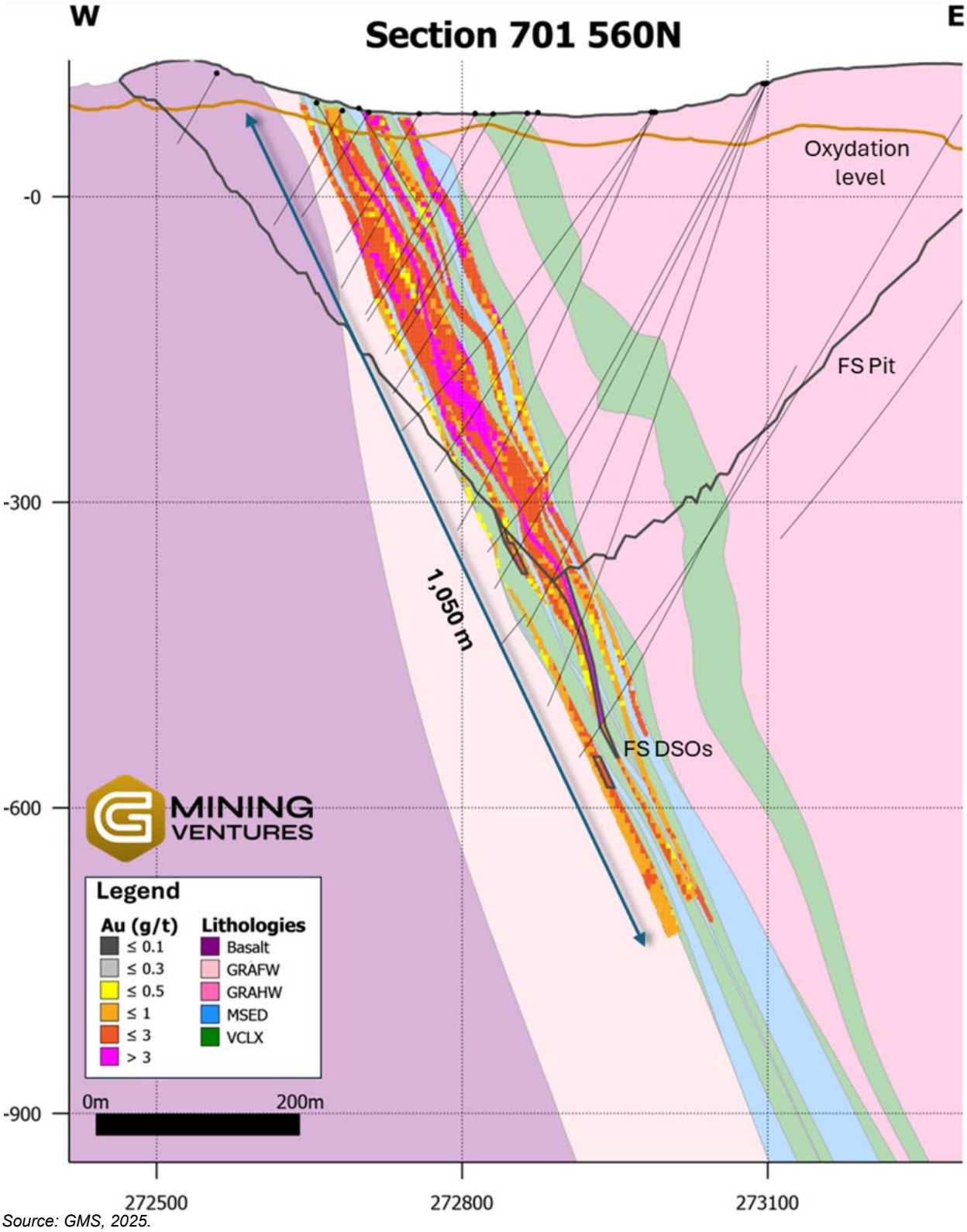
Source: GMIN, 2025.
**Note: FS Mineral Resources pit and DSO*

Figure 7.9: Geological Cross-Section 701800 N, Looking North



Source: GMS 2025.
**Note: FS Mineral Resources pit and DSO*

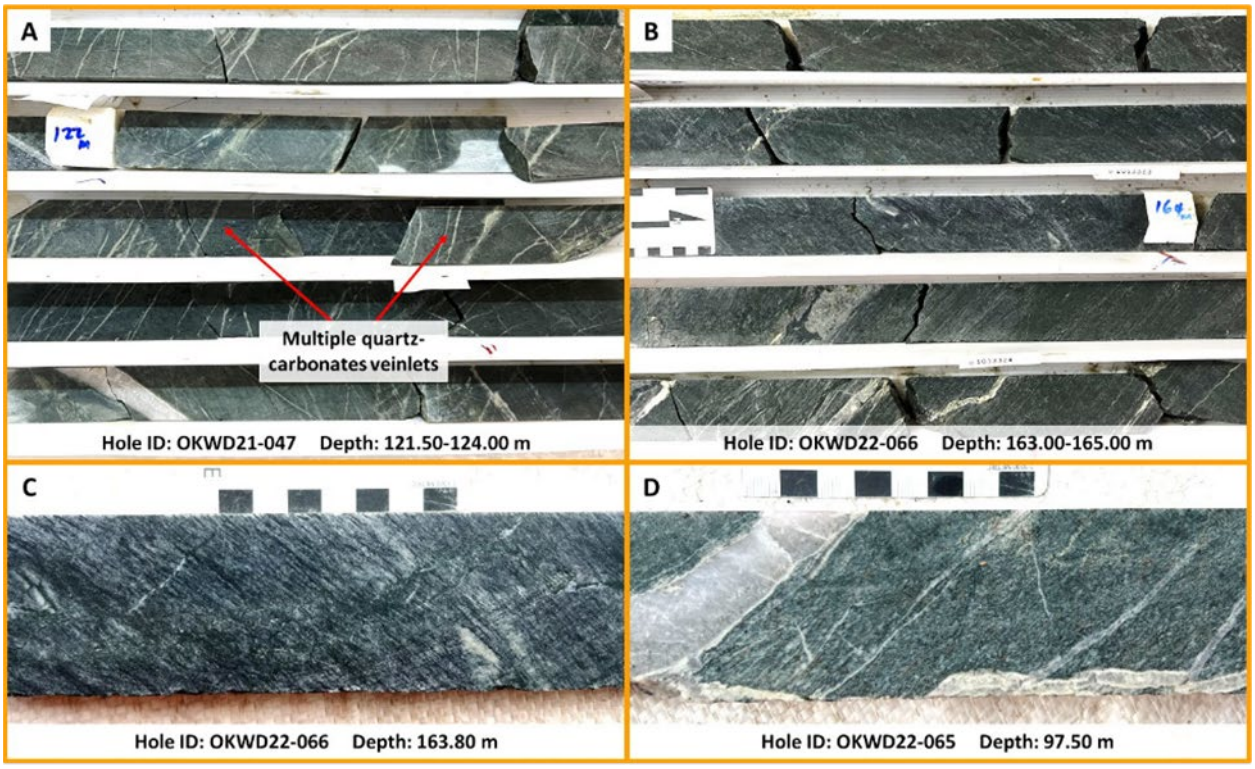
Figure 7.10: Geological Cross-Section 701560 N, Looking North



Source: GMS, 2025.
**Note: FS Mineral Resources pit and DSO Volcanoclastics*

Volcaniclastics are herein defined as clastic rock containing predominantly volcanic particles of any shape or size and does not imply any specific clast-forming, transport or depositional process (McPhie et al.,1993). They show local evidence of bedding and crossbedding as well as more massive facies with possible perlitic textures suggesting both sedimentary and effusive origins respectively. They present numerous imbricated quartz-carbonate veinlets transposed to the foliation. Volcanoclastic rocks are highly chloritized and usually have a dark green colour in unweathered rock and a purple colour in weathered rock (Figure 7.11). They are mainly composed of silt to sand sized grains but locally contain polymictic and monomictic fragments up to several cm in size. They can also show some evidence of siliciclastic elements, such as quartz and sericite, as contacts between the volcaniclastics and siliciclastics appear gradational and thus, conformable. Moreover, the frequent numerous transposed quartz-carbonate veinlets can give them a strong bedding-like characteristic. However, these volcanoclastics usually show stronger chloritization and are darker than the siltstones and sandstones.

Figure 7.11: Volcanoclastic Core Samples



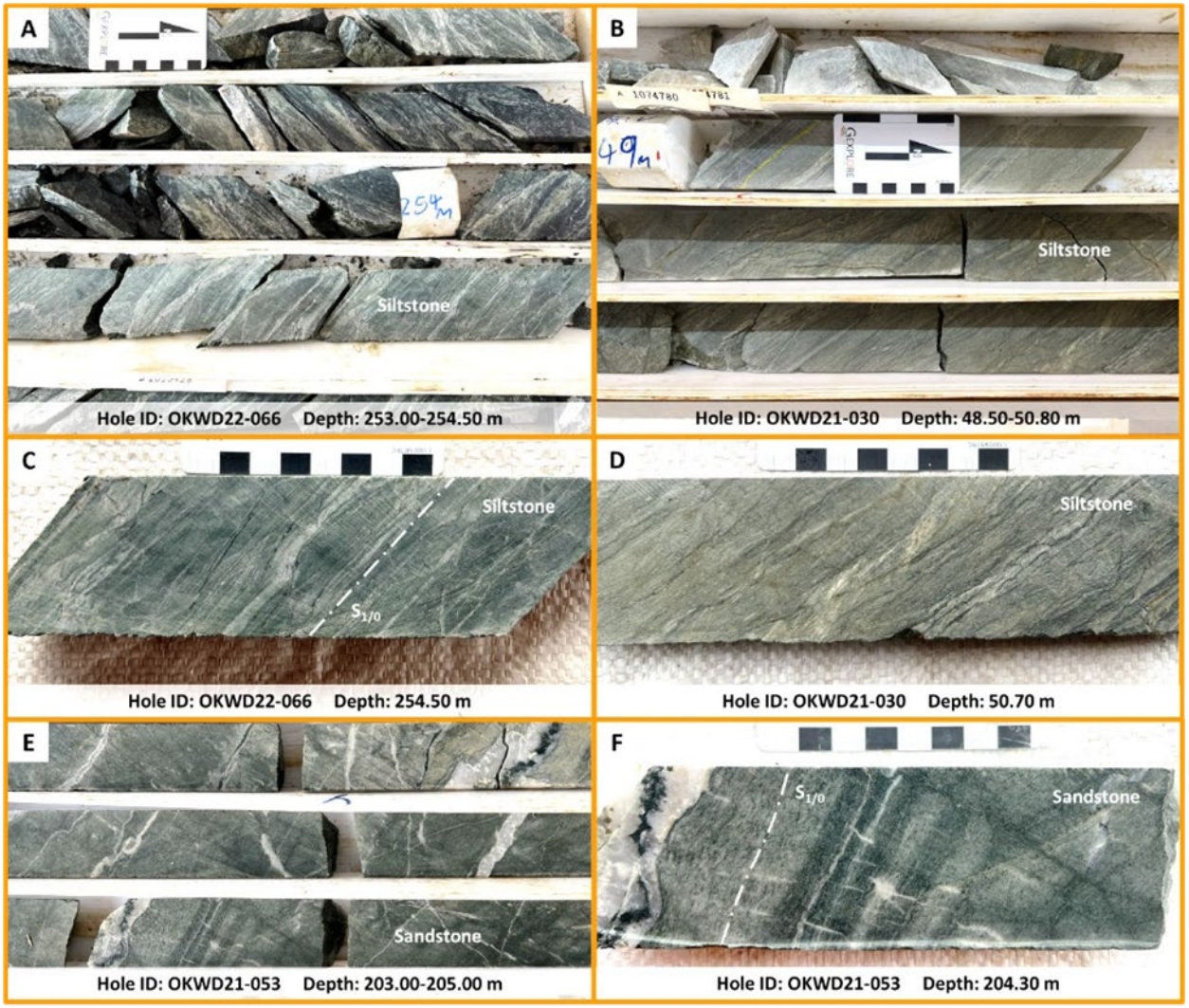
Source: Hainque et al., 2022
**Notes: A: Fine volcanoclastics with imbricated quartz-carbonate veinlets transposed to the main foliation in hole D21-047. B: Coarser and bedded volcanoclastics showing evidence of bedding in hole D22-066. C and D: Bedded volcanoclastics in holes D22 066 and D22-065.*

7.2.3.1 Siliciclastic Sediments

The siliciclastic sediments correspond to interlayered sandstones and siltstones. They present a pale grey / beige colour and are usually highly carbonatized and moderately sericitized, with quartz and probably

fully carbonitized and/or sericitized feldspar (Figure 7.12). Locally they can be pale green with weak chloritization. They show clear bedding, preserved even in zones of intense S2 fabric development, and local evidence of crossbedding. The weathered product of these sedimentary rocks is generally yellow / orange. Sandstone-dominated rocks occasionally appear near the contact with volcaniclastics. Siltstones may grade into carbonaceous sediments.

Figure 7.12: Siltstone and Sandstone Core Samples



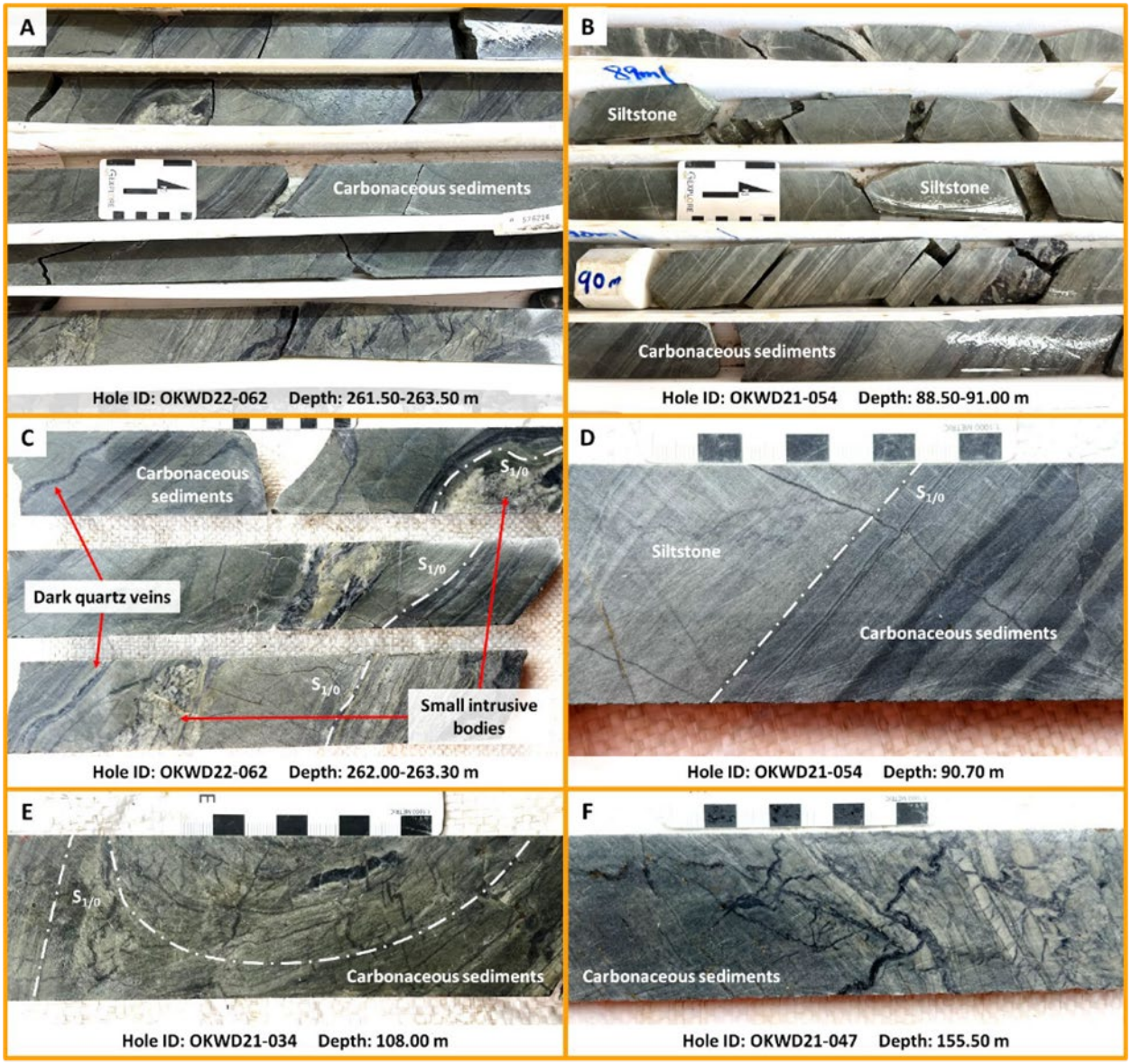
Source: Hainque et al., 2022
**Notes: A and B: Siltstone in holes D22-066 and D21-030. C and D: Bedded siltstones in holes D22-066 and D21-030. E: Sandstone dominated rock in hole D21-053, crosscut by several quartz veins. F: Well- sorted sandstones in hole D21-053.*

7.2.3.2 Carbonaceous Sediments

Carbonaceous sediments correspond to siliciclastic turbidite-like facies, alternating fine carbonaceous bearing sandstone / siltstone, and fine carbonaceous-rich shale layers (graphitic schists), as shown in Figure 7.13. While weathered, this lithology develops a grey colour and can show the remaining shale

layers. Due to the weak nature of the carbonaceous material, this lithology also composes a preferential "decollement" layer accommodating bedding-parallel slippage during deformation. This bedding-parallel slippage likely allowed responsible for small intrusive bodies (intercalated sediments and granitoid) and the emplacement of dark grey quartz extension and shear veins. The dark grey colour of these veins might be related to a chemical reaction between the carbonaceous material and the hydrothermal fluid.

Figure 7.13: Carbonaceous Sediment Core Samples



Source: Hainque et al., 2022
**Notes: A: Carbonaceous sediment in hole D22-062, with clear bedding and several dark quartz veins. B: Bedded siltstone and carbonaceous sediment in hole D21-054. C: Carbonaceous sediment in hole D22- 062, with several dark quartz veins, evidence of folding, and small intrusive bodies. D: Contact between siltstone and carbonaceous sediment in hole D21-054. E: Folded carbonaceous sediment in hole D21-034. F: Carbonaceous sediment crosscut by multiple small dark quartz veins in hole D021-047.*

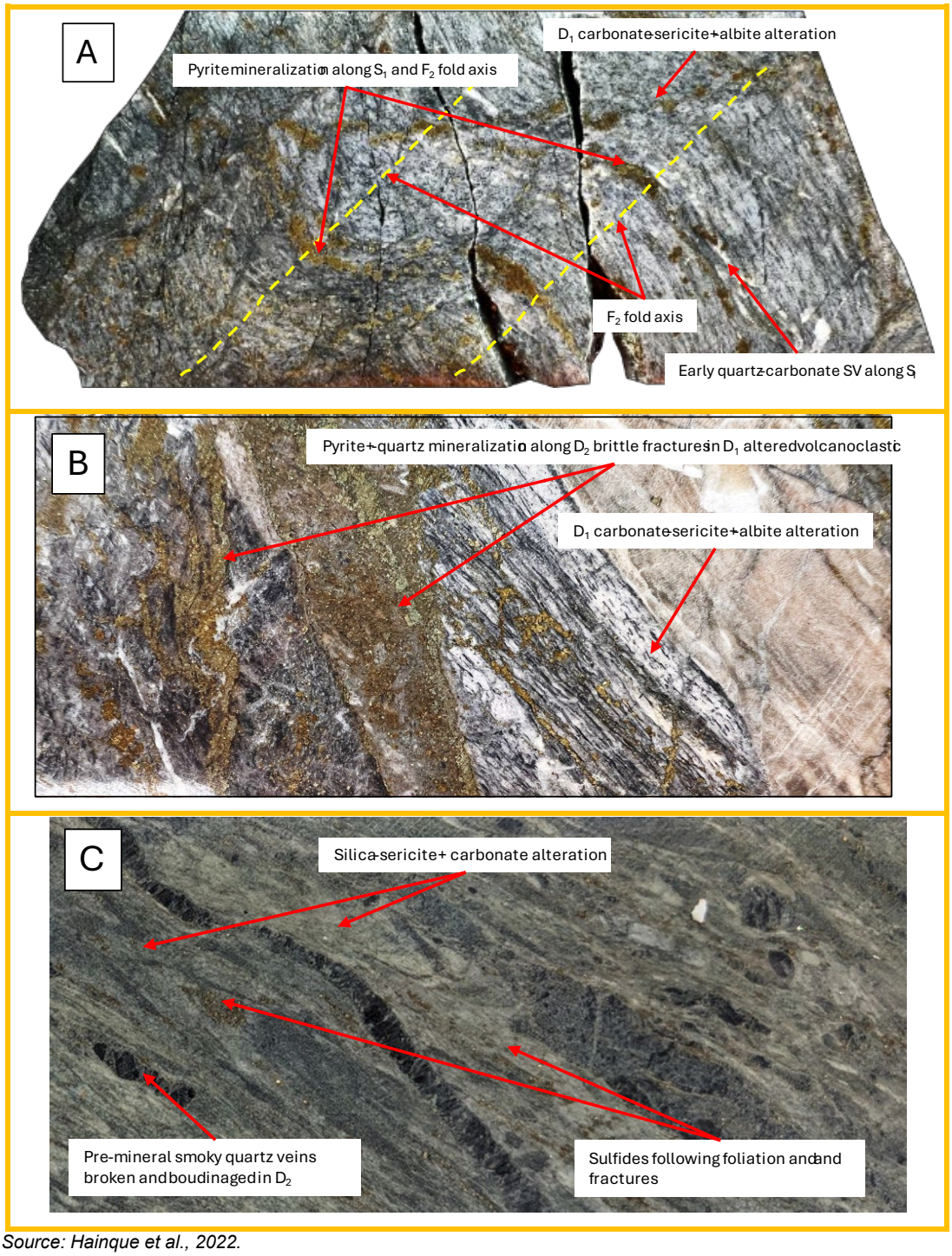
7.3 Structural Geology and Metamorphism

Rhyacian regional greenschist metamorphism and associated deformation modified clastic and plutonic rocks. The metamorphism of clay size grains in sedimentary and volcanoclastic rocks was preceded by dehydration and decarbonization. Metamorphism of granitoids is limited to the recrystallization of protolith minerals unless metamorphic fluids gain access via grain boundaries and/or litho-structural conduits (fractures, shear zones). In this case, hydration and, locally, carbonatization transformed magmatic minerals into metamorphic mineral assemblages compatible with the greenschist grade of metamorphism. The intensity or degree of change varies with fluid access, typically increasing toward pluton contacts and internal litho-structural conduits. The intrusion of granodiorite / quartz monzodiorite occurred before (syn volcanism) or in the early stage of Rhyacian orogenesis. The petrography work by Thompson (2022) shows that the metaclastites, matrix carbonate, chlorite, plagioclase, and white mica are recrystallized protolith minerals and/or products of metamorphism. Stress-related mobilization of matrix minerals in adjacent and/or nearby rocks is the preferred explanation for the formation of carbonate and carbonate-plagioclase-quartz veins. That is, overall rock compositions remain essentially unchanged.

Oko West outcrop and core observations by Lacroix (2022), Hainque et al. (2022), and Oko West Project geologists demonstrate that the area is marked by polyphase deformations, with a first N-S tight folding from an E-W shortening event, followed by a second E-W fold overprint from a N-S shortening event. Gold mineralization occurs predominantly within volcanoclastic, siliciclastic, and carbonaceous sediments, characterized by sulphide ± silica (pyrite, chalcopyrite, sphalerite) overprinting earlier silica, carbonate, and sericite alteration (Figure 7.14 and Figure 7.15). The mineralized intervals are generally associated with higher intensities of pre-mineral alteration and veining including quartz / quartz carbonate shear veins (SV) and multiple generations of extension veins (EV). The early stage EVs were transposed to the foliation during fold tightening by bedding-parallel slip. Later EVs are crosscutting S1/0 and the first generations of EVs, marking a deformation continuum.

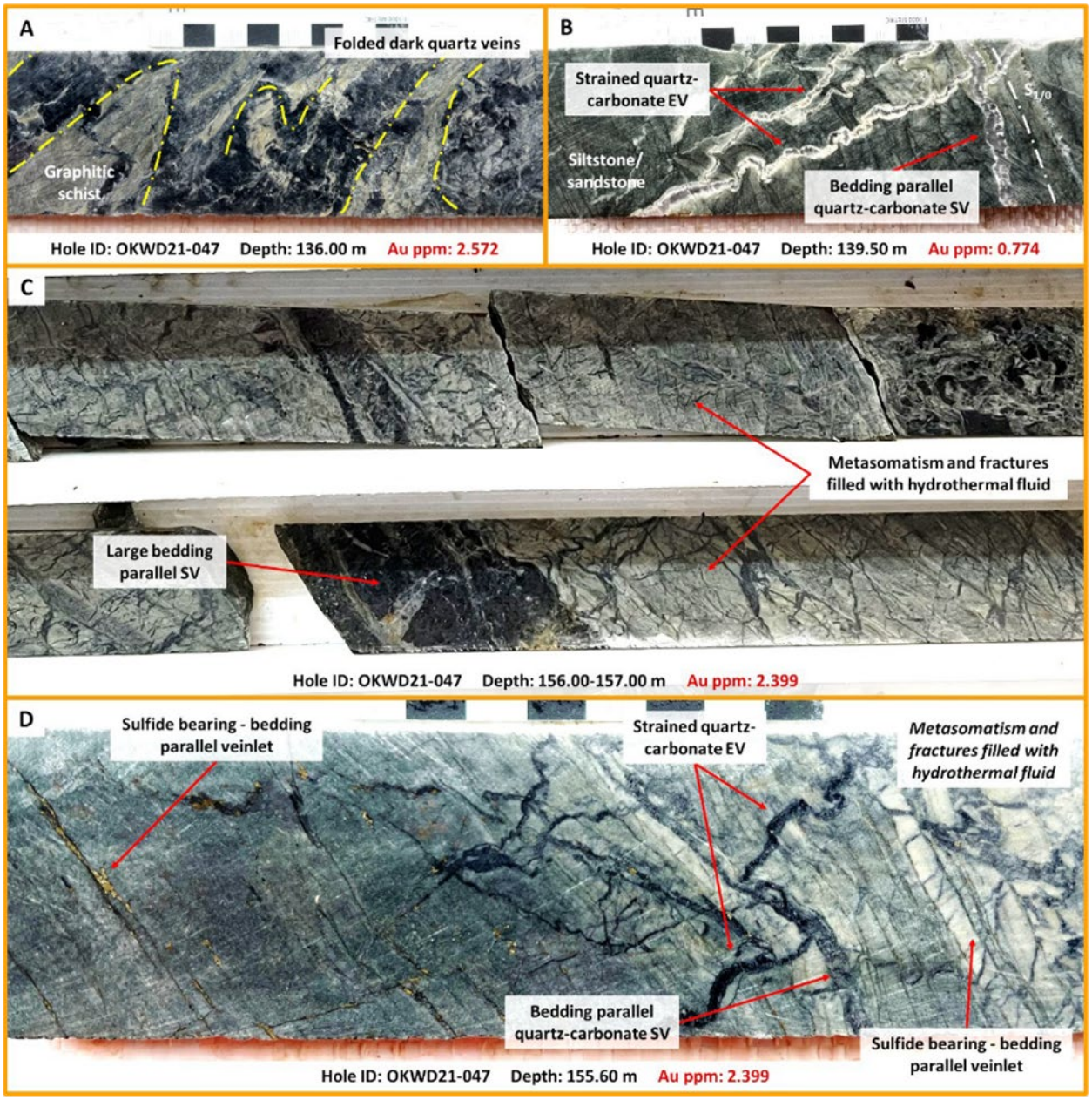
One major D1 event was probably responsible for the continuous development of the main quartz / quartz carbonate EV-SV system and related quartz, carbonate and sericite alteration (D1a and D1b). This event was characterized by tight folding, with the main volcano-sedimentary package sandwiched between the footwall and hanging wall granitoids, and responsible for the emplacement of most of the veining and alteration and possibly minor amounts of gold mineralization. The D2 event is associated with the development of S2 crenulation and mineralization. Most of the gold was emplaced during this phase in F2 fold axis and along brittle fractures brecciating the earlier D1 vein systems and associated alteration. The tight folding D1 and the broad open D2 fold events produced a subdued type-2-fold pattern, affecting the stratigraphy and mineralization. In summary, the mineralized sediments record a history of multiphase deformation that overlapped in time and space with lower greenschist regional metamorphism.

Figure 7.14: Mineralized Structures in Core Samples



Source: Hainque et al., 2022.
**Notes: Images A and B show quartz-feldspar crystal rich volcanoclastics with strong overprinting carbonate-sericite-albite pervasive alteration, subsequently folded and brecciated during D2. Image C shows the same relationships including pre-mineral smoky quartz EV-SV, in a quartz-sericite-carbonate+-albite altered siltstone. Sulphide mineralization is concentrated along F2 fold axis and D2 fractures along with pre-existing D1 fabrics, such as S1.*

Figure 7.15: Mineralized Structures in Core Samples (EV and SV)



Source: Hainque et al., 2022
**Notes: Bedding parallel SV, folded EV, and fractures due to metasomatism. A: Folded sulphide-bearing dark quartz-carbonate veins in hole D21-047. B: Quartz-carbonate SV and multiple folded EV in siliciclastic sediment in hole D21-047. C: Highly fractured interval (hardened by metasomatism) filled with sulphide-bearing quartz-carbonate veinlets (stockwork) and large SV in hole D21-047. D: Sulphide bearing fractures in a medium hardened by metasomatism in hole D21-047.*

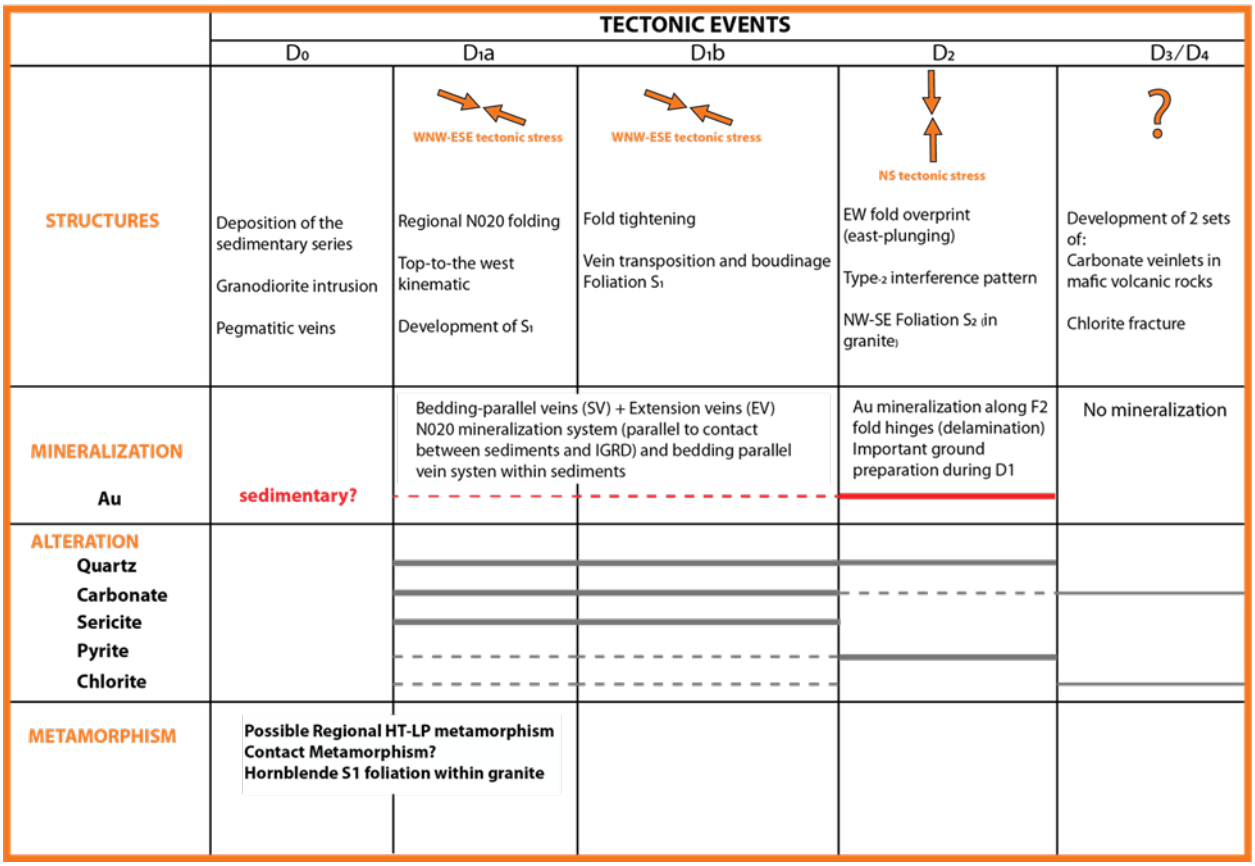
7.4 Tectonic Events and Gold Mineralization

Combining observations from the Oko West Project team and structural analyses of the drill core and field visits, Lacroix and Hainque (2024) and Hainque et al. (2024) have proposed the following geological

evolution for the mineralization at Oko West, reflecting on the regional tectonic regimes described in Section 7.3 (Figure 7.16):

- D0: Deposition of the volcano-sedimentary sediments and mafic volcanic rocks and emplacement of the granitoid intrusions responsible for early potassic veins and metasomatism.
- D1a: WNW-ESE to NW-SE compressional event related to F1 folding, associated with the development of S1. Bedding-parallel veins and "en-echelon" EV started developing during folding and may have contained minor gold.
- D1b: Late stage of tight folding, marked by the formation of penetrative foliation, S1, and the transposition of early extension veins (EV1) to S1 and further development of the EV system. The transposed EV1 veins are locally dismembered and sheared, forming sigmoidal tectonic clasts imbricated along the foliation.
- D2: N-S fold overprint, associated with the development of S2 crenulation and may be associated with the development of the late quartz-Sulphide EV2 vein system. The more discrete nature of S2 highlights less tight folding, suggesting that F2 might have a larger amplitude than F1. Most of the gold was deposited during this phase, focused along F2, S1 and D2 brittle fractures.
- D3: Development of two (2) sets of conjugated fractures / faults with possible significant offset.

Figure 7.16: Tectonic Events and Gold Mineralization at Oko West



Source: Lacroix and Hainque, 2024

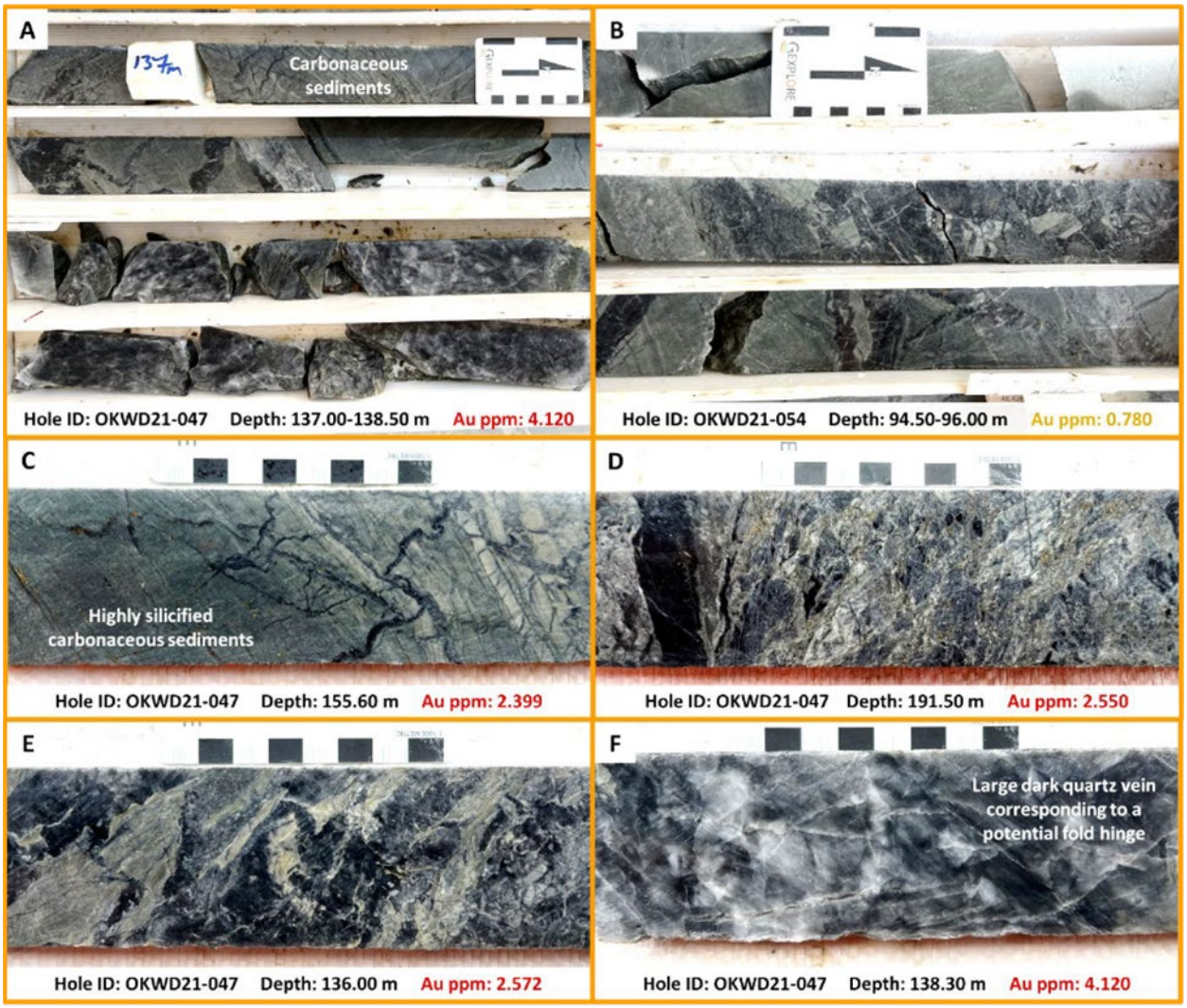
7.5 Gold Mineralization

Gold mineralization mainly occurs within volcanoclastic, siliciclastic, and carbonaceous sediments, which have an overall tabular geometry dipping to the east. Strong evidence of silica and carbonate alteration can be observed within the mineralized zone, with more intense sericitization, as well as with the presence of multiple sulphides (pyrite, chalcopyrite, sphalerite) disseminated within the altered rock, along bedding / laminations or small fractures / veinlets, or as envelopes around the quartz and quartz-carbonate veins. Despite the spatial association, most of the alteration is pre-mineral and served to harden the rock allowing for brittle deformation and dilation during the D2 mineralization stage.

Other hydrothermal alteration does not seem directly related to the mineralization, or their association with it is unclear. Chloritization moderately to highly affects all lithologies in mineralized and non-mineralized areas. Epidotization was only observed in non-mineralized areas, mainly in the upper and intermediate granitoids, in the upper volcano-sedimentary sequence between them, and locally in the lower main volcano-sedimentary package. Magnetization has been observed in mineralized and non-mineralized areas, with strong variability, even within the same unit (probably due to alteration to other oxides).

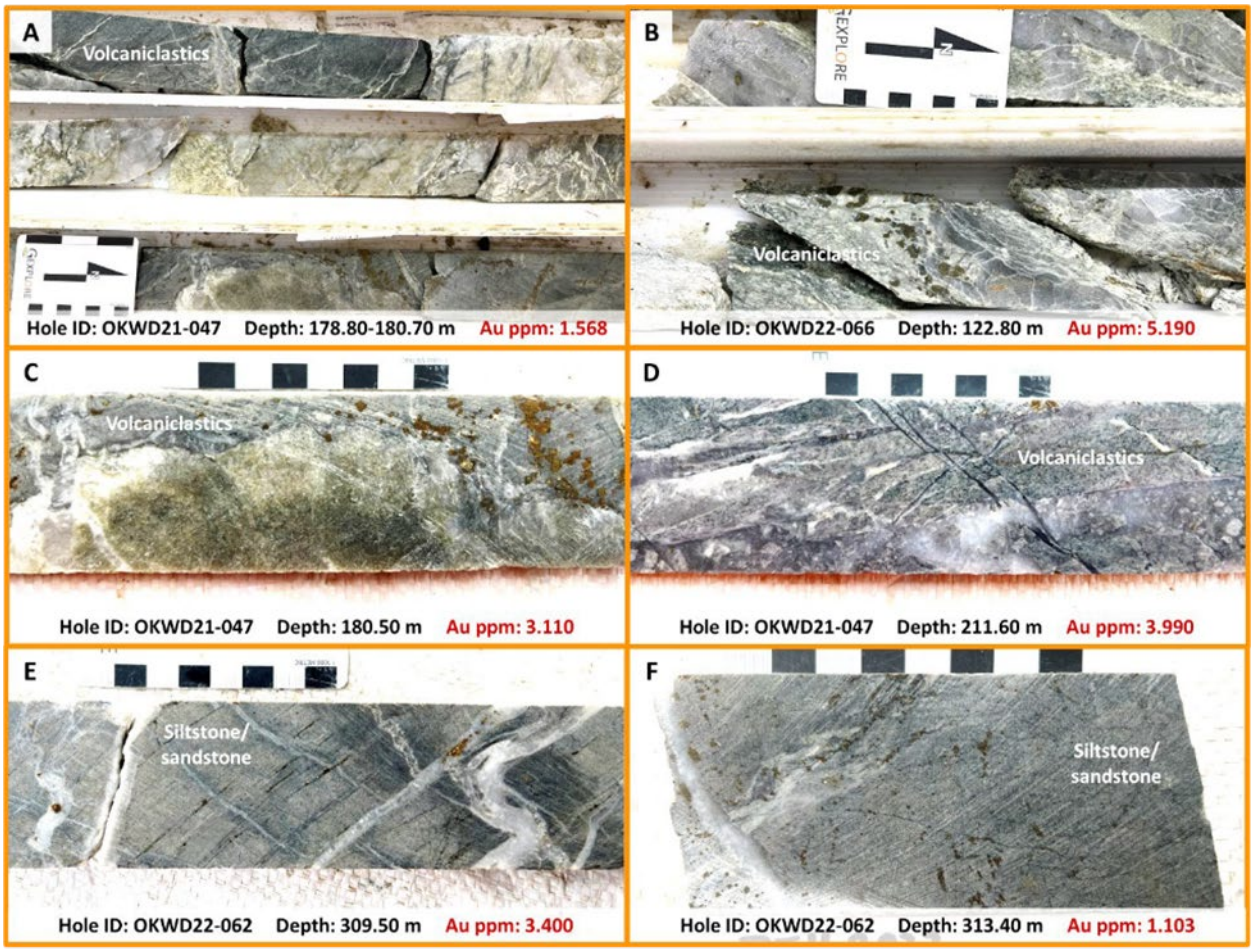
The hydrothermal fluids responsible for sulphide and gold emplacement produced a network of Sulphide bearing quartz and Sulphide-only stringers spatially associated with EV quartz-carbonate vein systems and their hydrothermal alteration that partially or totally overprint the parent rock. In the carbonaceous sediments, the quartz / quartz-carbonate vein system is characterized by a dark grey smoky colour (Figure 7.17), whereas in the volcanoclastic and siltstones / sandstone lithologies, it is characterized by a white-grey colour (Figure 7.18). The dark gray colour of the quartz / quartz-carbonate veins in the carbonaceous sediments could be related to a geochemical reaction between the carbonaceous material and the hydrothermal fluid. This carbonaceous material may have acted as a reducer for the later Sulphide and gold bearing fluids, leading to particularly high grades of gold.

Figure 7.17: Gold Mineralization in Carbonaceous Sediment Core Samples



Source: Hainque et al., 2022
**Notes: A and B: Mineralized carbonaceous sediment intervals in holes D21-047 and D21-054, with strong silicification and carbonatization. C: Mineralized stockwork in hole D21-047. D: Highly brecciated mineralized interval with strong silicification, carbonatization, and sulfidation in hole D21-047. E: Folded sulphide-bearing dark quartz-carbonate veins in hole D21-047. F: Large dark quartz-carbonate vein corresponding to a potential fold hinge in hole hinge in hole D21-047.*

Figure 7.18: Gold Mineralization in Volcanoclastic and Siltstone / Sandstone Core Samples



Source: Hainque et al., 2022
**Notes: A: Mineralized volcaniclastic interval in hole D21-047, with strong silicification and carbonatization, associated with metasomatism. B: Mineralized volcaniclastic interval in hole D22-066, with multiple large pyrites. C: Metasomatism from the hydrothermal fluid and multiple pyrites showing pressure shadows in hole D21-047. D: Highly strained and fractured mineralized volcaniclastic interval with multiple sulphide- bearing quartz-carbonate veins in hole D21-047. E and F: Mineralized siliciclastic interval with multiple sulphide-bearing quartz-carbonate veins and veinlets in hole D21-062.*

8. DEPOSIT TYPES

The relationship between gold mineralization, magmatism, and deformation in the Guiana Shield is poorly defined by the literature, despite the presence of several major deposits (Tedeschi et al., 2020).

In French Guiana, orogenic-type gold deposits are mainly related to the regional D2 tectonic-metamorphic deformation (between 2.1 and 2.0 Ga). The quartz veins-related mineralization occurs along shear zones in greenstone belts and is associated with granitic magmatism (Milési et al., 1995; Milési et al., 2003). Tedeschi et al. (2020) outline that recent data from the Karouni orogenic gold deposit in Guyana support this timing, as gold mineralization has been dated to 2.084 Ga ± 14 Ma. The Karouni orebodies are primarily related to shear-hosted quartz-carbonate-chlorite ± tourmaline-pyrite-gold veins in high MgO basalts and high TiO_2 dolerite sills and granodiorite. This mineralization type occurred within dilatational bends formed by the late dextral transcurrent movement of strike-slip shear zones. They are controlled by rheological contrast, as brittle deformation of dolerite sills and granodiorite resulted in better mineralized extensional veins than ductile deformation of basalts. Thus, gold mineralization is found at the interface between these lithologies and structures (Tedeschi et al., 2018). Similarly, at the Wenot Lake deposit at Omai, gold mineralization occurs along a shear zone straddling the contact of sedimentary and volcanic sequences (Bertoni et al., 1991). At the Rosebel gold mine in Suriname, mineralized shear zones developed along contacts between units of varying rheologies, but also, to a lesser degree, parallel to axial plane cleavages in fold noses. The bulk of mineralization at the Royal Hill deposit, for example, is hosted in bedding-parallel quartz-carbonate-tourmaline veins along lithological contacts (Wasel et al., 1997, and Daoust, 2016).

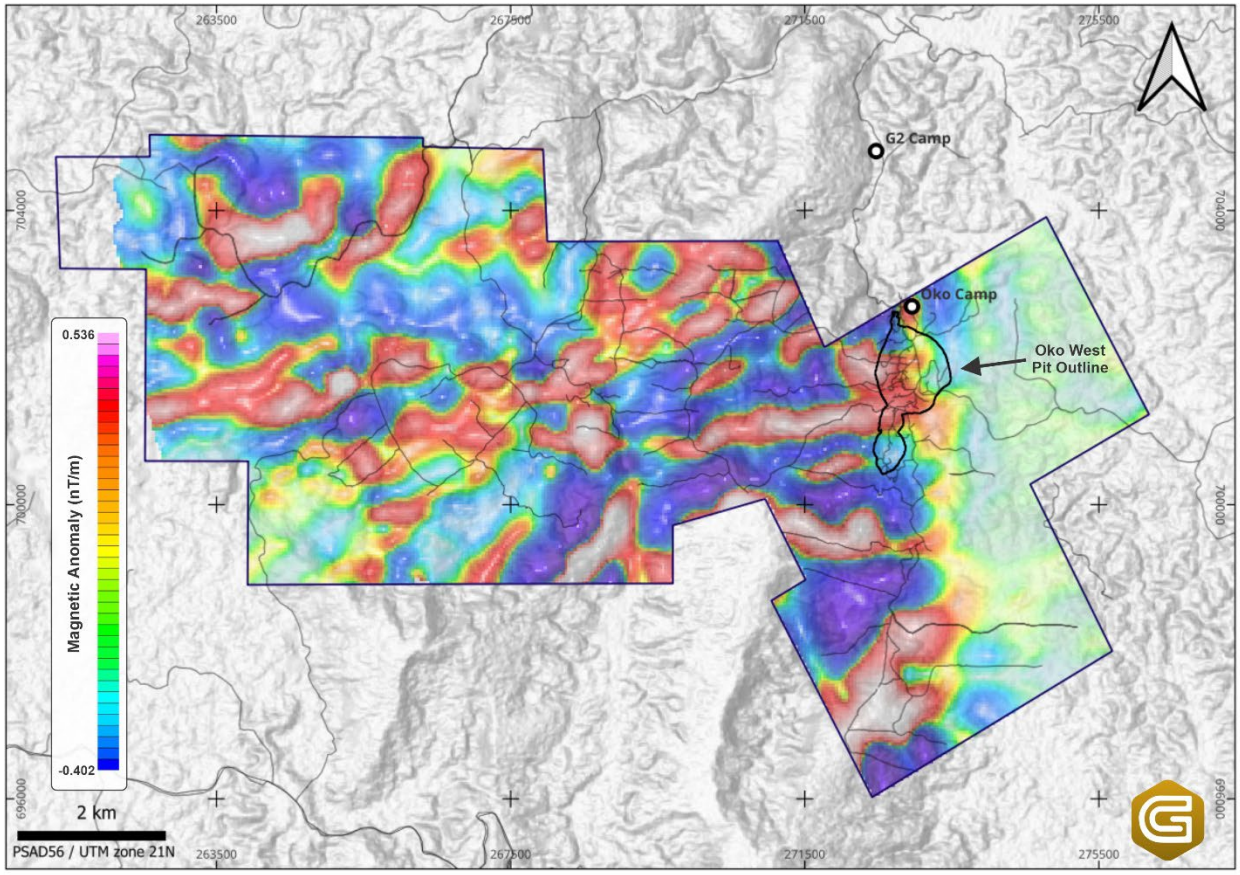
9. EXPLORATION

In August 2019, the Canadian company Terraquest covered the project area with an airborne geophysical survey of about 690-line km using the specifications below:

Parameters	Specification
Aircraft	King Air C90 Registration C-GCFZ
Primary Airborne Geophysical Sensors	High-Resolution Magnetics (Cesium Vapour), Gamma Ray Spectrometers
Base Station Sensors	High Resolution Diurnal Magnetics (Cesium Vapour); Base GPS L1/L2 12 channel, fully kinematic grade
Aircraft Magnetometer Sensitivity	+/- 0.005 nT
Magnetometer Noise Envelope (4th diff.)	+/- 0.5 nT counting at 0.1 Hz
Traverse Line Direction	344°/164°
Control Line Direction	074°/254°
Traverse Line Spacing	200 m
Traverse / Control Intersection Tolerance	± 15 m
Survey Clearance (AGL)	~60 m above the canopy
Magnetic Sample Interval	7-8 m (10 Hz sampling)
Average Local Ferry	~130 km
Aircraft Survey Velocity - Nominal	~205-240 km/h
Average Ferry Speed	~325 km/h

This geophysical survey assisted in the interpretation of the area's structural geology and lithological distribution (Figure 9.1) and clearly defined the contact between greenstone volcanics and the Oko pluton to the east. Although the line spacing of 200 m is relatively wide, the distinction between magnetic highs and lows can be readily identified.

Figure 9.1: RTP 1VD Map of Airborne Magnetic Data

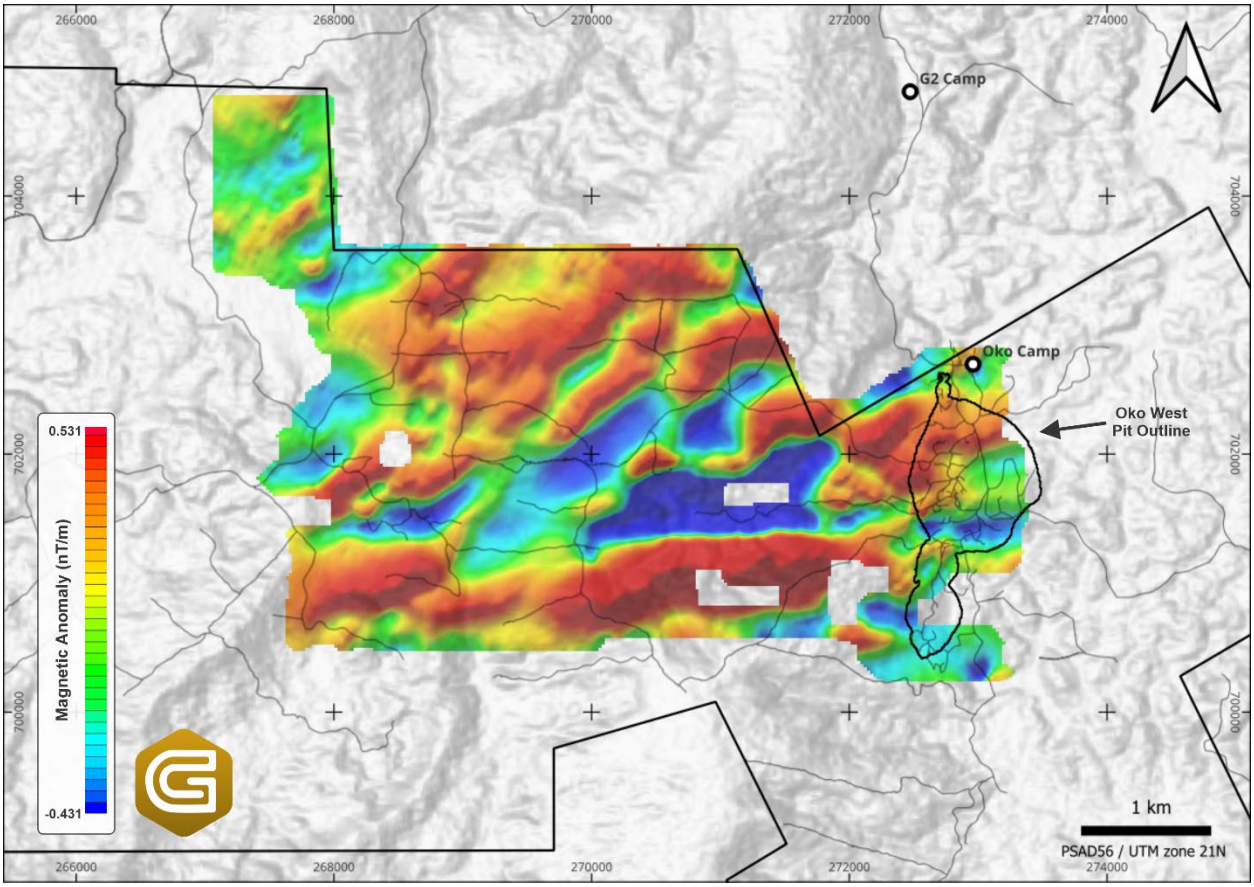


Source: GMIN, 2024

9.1.1 Ground Magnetics

Ground Magnetics survey has been ongoing on the Oko West property since early 2023 with GMIN owned equipment and personnel. Sectors of surveys are chosen quarterly based on exploration interest (Figure 9.2). Since early 2024, multiple test lines were re-surveyed using the continuous walking mode to ensure that the best functional settings were being applied and former issues resolved. Infill lines were planned and surveyed in Blocks 5, 6, 7 and 8. The data was still being processed at the time this report was written.

Figure 9.2: Map of RTE Ground Mag Coverage Over Terrain

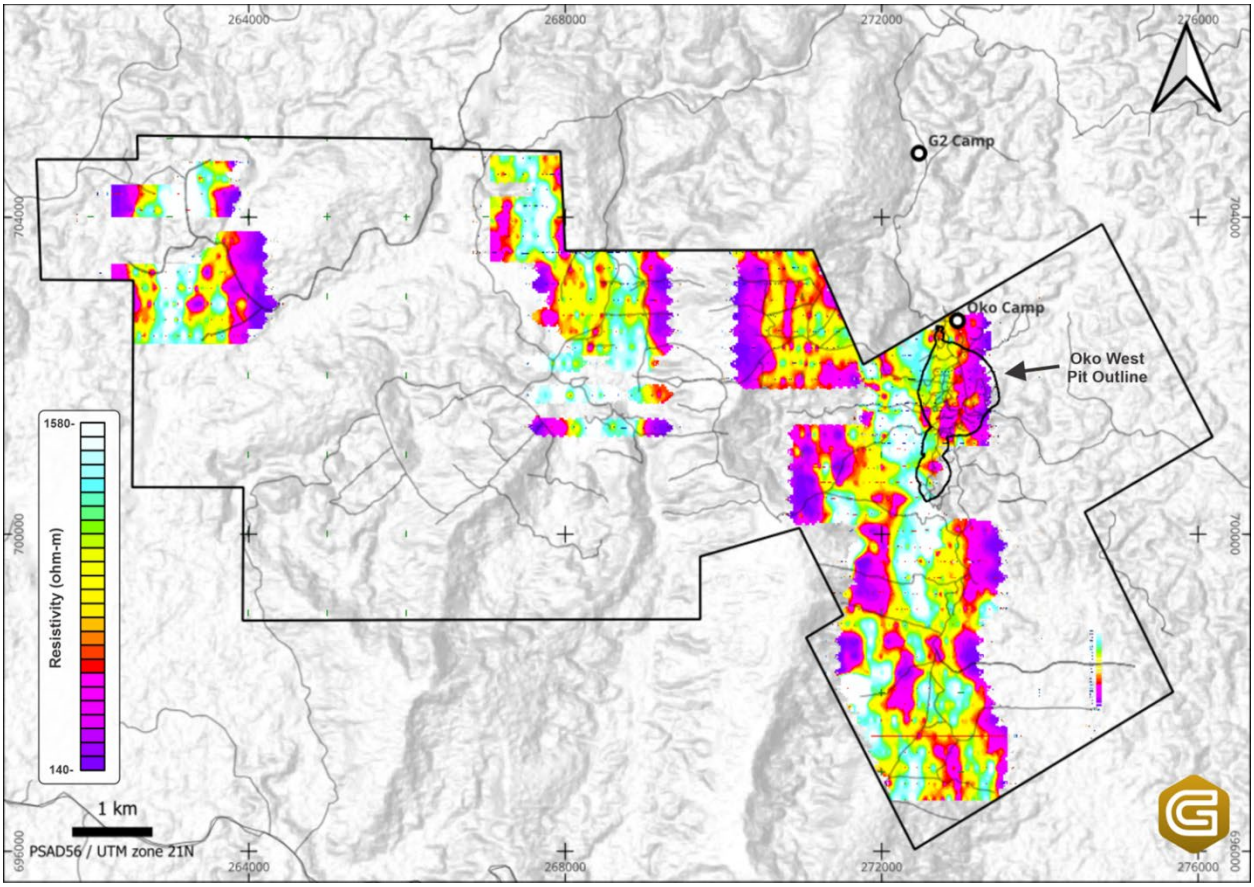


Source: GMIN, 2024

9.1.2 Induced Polarization and PDP

The induced polarization geophysics program was completed towards the end of March 2024. Five (5) areas of interest were surveyed and used for exploration targeting across the Oko West property using chargeability and resistivity data. Two (2) pole dipole (PDP) lines were done in Block 4 and three (3) in the East Takatu area. These surveys were used to plan drillholes in Blocks 7 and 8 which target potential mineralization at depth (Figure 9.3).

Figure 9.3: Map of IP Resistivity Coverage with Mineralized Domains at 0 m RL Over Terrain



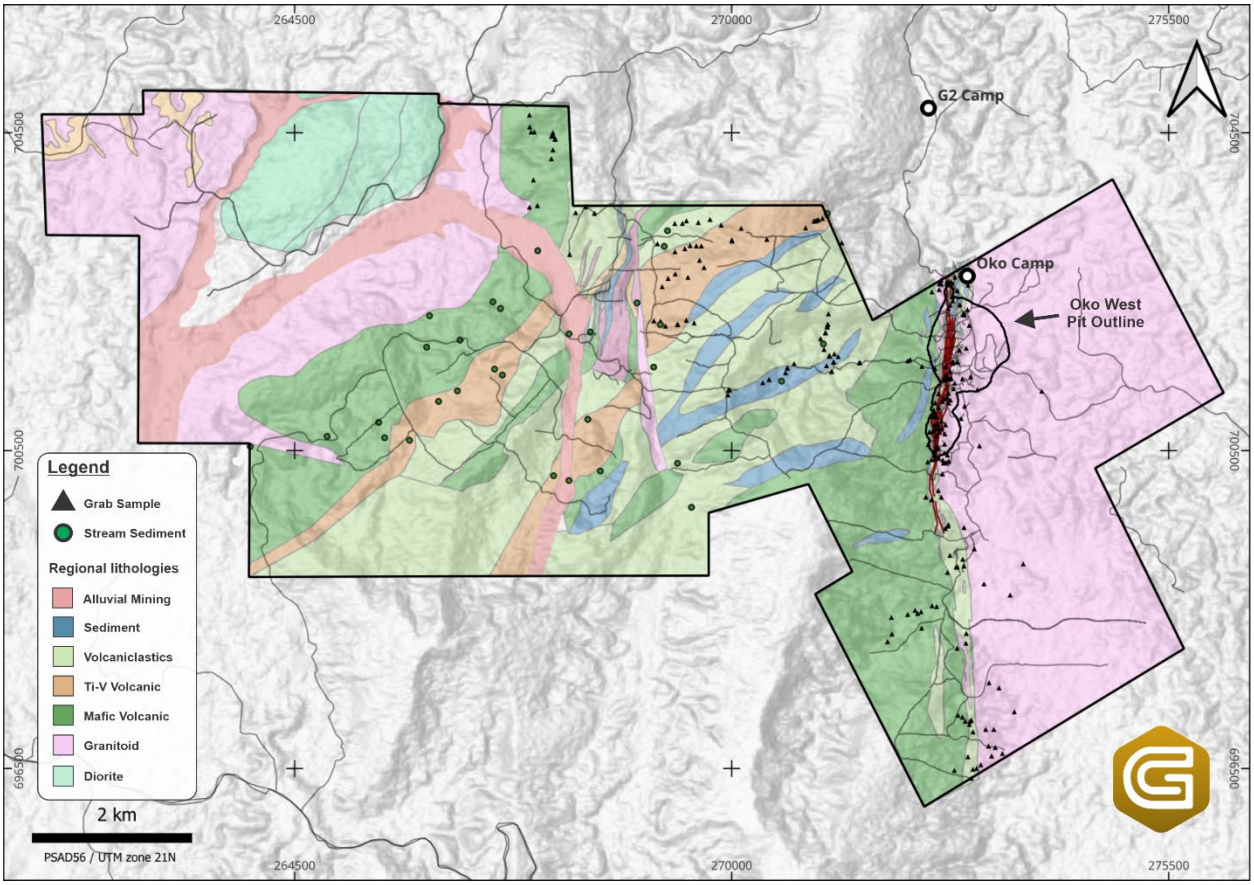
Source: GMIN, 2025

9.2 Geology Reconnaissance and Stream-Sediment Geochemistry

During September and October 2019, geologists Jorge Tachibana and Deuel Garner from RGD completed a reconnaissance of the area's geology by mapping and sampling available outcrops (Tachibana and Garner, 2019). A total of 18 rock samples, 19 regolith samples and eight (8) hand-auger samples were collected.

A stream-sediment sampling campaign collecting both BLEG samples (Bulk Leach Extractable Gold) and sediment samples for ICP-MS was carried out from 35 sites in the central-western part of the project area (Figure 9.4). This stream-sediment campaign did not cover creeks along the eastern edge of the permit area, where the gold mineralization at Oko West was later discovered.

Figure 9.4: Stream Geochemistry Survey and Reconnaissance Mapping Points Plotted on Geology



Source: GMIN, 2025

9.3 Soil Geochemistry

Early in 2020, Reunion completed a soil sampling survey covering the eastern-central portion of the Project area previously not covered by the stream-sediment survey. The eastern-central area drains many creeks with artisanal alluvial workings (

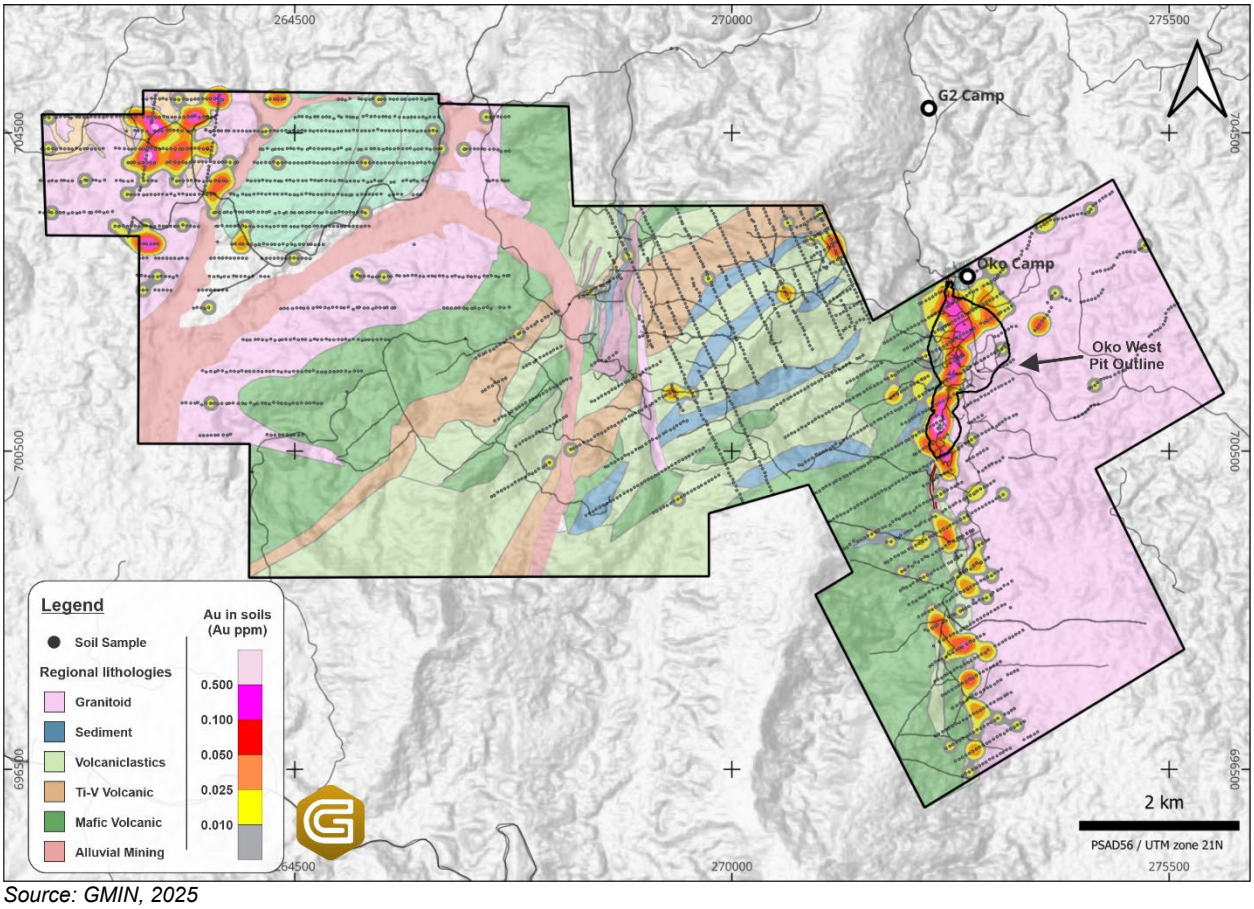
Figure 9.5). The soil grid was oriented at 050° azimuth to cover both potential north-south and east-west trending mineralized structures. The project team collected soil samples at an average depth of 30 cm, every 50 m along lines and spaced 200 m apart. This survey defined gold anomalies straddling the contact between the Oko pluton to the east and the volcano-sedimentary sequence to the west, with a strike length of approximately 6 km. The anomalies are continuous at the northern end, at the headwaters of Kairuni creek, but become discontinuous at the southern half, at the headwaters of Takutu creek. The exploration team observed that there were no primary historical workings up-terrain from the alluvial gold workings and that potentially intact bedrock mineralization could be present.

A second soil geochemistry program tested the permit's central area, exploring other potential source areas (Figure 9.5) relating to alluvial workings that straddle the western edge of the project. Results were comparatively weak and likely hindered by a thick duricrust layer blanketing the sector.

In total, the soil geochemistry program collected 1,691 samples within the project boundaries, mostly in 2019 and 2020. A total of 91 hand auger holes drilled within trenches generated an additional 351 samples within the saprolite profile.

In 2024, the soil sampling program was reinstated to better cover the western part of the property, specifically the North-West Extension. A total of 561 samples were collected and analyzed for gold, with 232 returning anomalous values above 0.01 ppm Au. These results will guide the 2025 exploration campaign and help identify new trenching and drilling targets.

Figure 9.5: Map of Soil Geochemical Program with Anomalies



Source: GMIN, 2025
**Note: Lithology Background Superimposed Over Terrain.*

9.4 Trenching

The soil geochemistry results identified significant gold anomalies at the headwaters of creeks with historical alluvial gold production needed to be explored further. The Company mobilized two (2) excavators and one (1) bulldozer from its Aremu project to launch a trenching program.

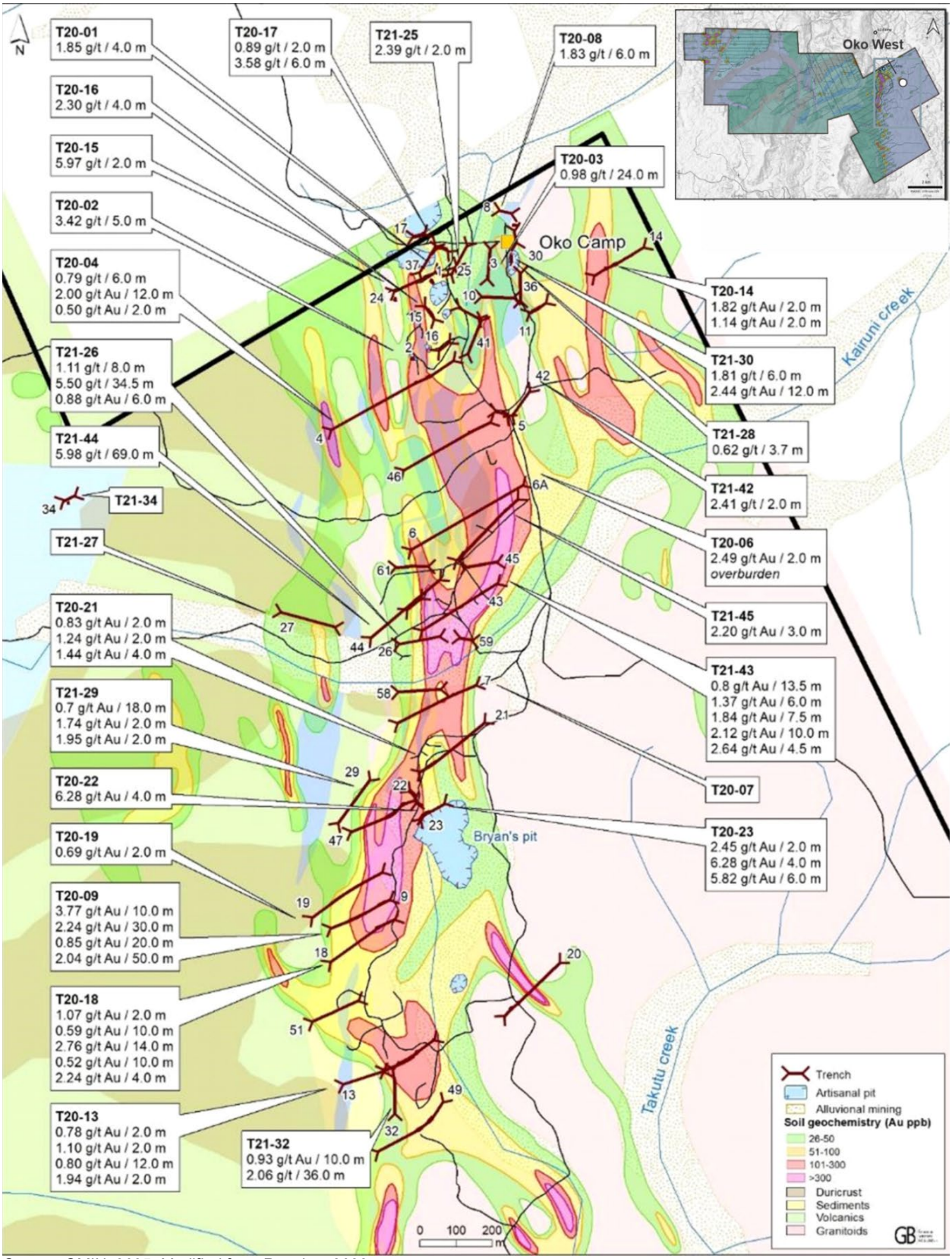
In the first quarter of 2020, the Company initiated the trenching program to test two (2) groups of anomalies, one south of the camp (exploration Blocks 1 and 3) and another next to the so-called Bryan pit (exploration Block 5 - Figure 9.6). The trenching program consisted of excavator-dug trenches along soil survey lines to a safe "shoulder" depth (approximately 1.2 m to 1.5 m) in an attempt to reach saprolitic material. Hand cut channels on the trench walls provided continuous samples. Only in-situ material was sampled, avoiding colluvium and alluvium, with an average sample length of 1.5 m (2-3 kg of material). Trenches were spaced at 200 m and attempted to traverse the entire width of the soil anomalies. Whenever saprolite was identified, the trench geology was mapped. Trenches are considered sub horizontal drillholes for database purposes and were surveyed accordingly. As the trenches were dug along soil lines and at an angle to the north-south striking shear zones, some mineralized intervals do not represent- true widths.

Trenching was interrupted during the second quarter of 2020 due to the Covid-19 pandemic and recommenced in September 2020, continuing uninterrupted until June 2021, when resource drilling commenced. During this period, trenching focused on the exploration of Block 4, intersecting several well mineralized intervals and supporting geological interpretations.

The trenching program successfully validated the soil geochemical anomalies. They confirmed the presence of significant in-situ gold mineralization in a sequence of sediments striking north-south and at the contact with the Oko pluton granitoid. Figure 9.6 summarizes the trench sampling results in the Kairuni zone. Trenching is a powerful tool for exploring areas with soil anomalies and outcropping mineralization, providing quick access to bedrock geology and systematic sampling.

In 2024, trenching on the Oko West Project targeted soil and stream sediment anomalies, focusing primarily on the western part of the property. Additional trenches were completed south of the Oko West main mineralized body in Blocks 7 and 8, specifically at the West Takatu target. The 2024 trenching program covered a total of 2,524 metres.

Figure 9.6: Map of Trenching Results in Kairuni Zone Plotted on Geology and Soil Geochemistry



Source: GMIN, 2025; Modified from Reunion, 2022

Table 9.1 summarizes trenching statistics by year.

Table 9.1: Trenching Statistics

Year	Trenches Done	Length (Metre)	Average Trench Length (Metre)	Number Samples
2020	25	3,093	124	1,385
2021	41	4,466	109	2,116
2022	13	419	32	235
2023	6	758	126	312
2024	26	2,524	97	1,210
Total	**111**	**11,259**	**101**	**5,258**

9.5 TSF and WSF Condemnation Drilling

In 2024, condemnation drilling was carried out in the Tailings Storage Facility (TSF) and Waste Storage Facility (WSF). In the TSF area, 47 diamond drill holes (DDH) were completed, totalling 12,386 metres. In the WSF area, four (4) DDH and 60 reverse circulation (RC) holes were drilled, totalling 231 metres and 1,508 metres, respectively. The condemnation program verified that these areas were appropriate for infrastructure placement and would not overlap with economically viable near-surface mineralization.

10. DRILLING

This section includes details on drilling procedures employed at the Oko West Project and a compilation of drilling statistics by drilling type. Drilling and trenching can be summarized into the following types:

- Surface trenching (TR) to gather in-situ surface samples of saprolitic and alluvium / colluvium material.
- Shallow scout Reverse Circulation (RC) drilling focusing on the definition of geochemical targets beneath the duricrust layer.
- Diamond drilling (DD), including wedges from parent DD holes, targeting depth extensions to shallow trenching and RC defined gold anomalies.

Given the relatively recent nature of the discovery of gold mineralization at Oko West, no historical drilling data exists on the Project.

10.1 Drilling Statistics by Year

A summary of the drilling and trenches campaigns performed by the Company is presented in Table 10.1. GMS is not aware of any drillholes that exist on the project before 2020.

Table 10.1: Drilling (DD-RC-Wedges) and Surface Trenches Conducted on the Project by Year, Up to September 15, 2024.

Period	Type of Hole / Trench	Number of Holes / Trench	Total Length (m)	Total Assayed Length (m)
2020	DD	3	462.0	462.0
	TR	25	3,092.6	2,739.0
2021	DD	54	6,644.0	5,443.0
	RC	109	8,686.0	8,594.0
	TR	41	4,465.7	4,029.7
2022	DD	164	42,658.2	33,867.7
	RC	199	16,651.5	16,401.0
	TR	13	419.0	411.0

Period	Type of Hole / Trench	Number of Holes / Trench	Total Length (m)	Total Assayed Length (m)
2023	DD	182	70,547.7	36,960.9
	RC	1,436	26,550.0	19,072.0
	TR	6	758.0	624.0
	Wedge (DD)	17	6,039.9	2,917.6
2024	DD	115	36,822.1	25,230.4
	RC	31	1,353.5	1,610.0
	TR	26	2,523.7	2,355.3
	Wedge (DD)	22	8,614.1	4,928.4
Total	***DD***	**518**	**157,134**	**101,964**
	RC	***1,775***	***53,241***	**45,677**
	TR	***111***	***11,259***	**10,159**
	Wedge (DD)	**39**	**14,654**	**7,846**
Total		**2,443**	**236,288**	**165,464**

10.2 General Drilling Procedures

10.2.1 Safety

Pre-start inspections require the drilling operator to undertake the following steps before commencing work:

- Inspect ‘critical’ safety components of the drill rig.
- Report any faults to ensure prompt repair, distinguishing between those faults that will halt operation at a later time and those that require immediate repair or correction.
- Ensure all crew members are wearing PPEs.

10.2.2 Hole Numbering

Drillhole numbering follows a convention by which each collar must have a unique identifier consisting of the following:

- An alpha prefix denotes the project name (e.g., Oko W.) followed by one (1) letter indicating the drill type (e.g. "D" for diamond, "R" for reverse circulation) and year.
- A dash followed by a numeric suffix (e.g. OKWR22-066).

- The drillhole is sequential from year-to-year, i.e. does not revert to "1" each year.
- Twinned or re-drilled holes have the same Hole-ID as the original with a character suffix (e.g. OKWR22-005A).
- Wedge holes have the same Hole-ID as the parent with an additional suffix consisting of a dash “W” and a sequence number (e.g., OKWD23-239-W2 is the second wedge on parent hole OKWD23-239).

10.2.3 Drill Rig Supervision

The geologist has the authority and ability to perform the following duties:

- Ensure holes are drilled by following the approved drilling program.
- Enforce all safety procedures.
- Visit the drill rig at any time during each shift.
- Issue instructions to the drilling supervisor and resolve disputes.
- Issue instructions and supervise the crew carrying out sampling activities.
- Instruct the driller to cease operations if any of the following occur:
    - Accident or safety incident.
    - Dangerous drilling conditions that could result in injury or loss of equipment.
    - Environmental incident.
    - Excessive deviation detected from down-hole surveys.
    - Excessive and continuous water influx of the hole can result in high sample contamination.
    - Examine the core / RC chips at the planned end of the hole depth.

10.2.4 Drill Site Preparation

The layout of the drill site permits safe drilling operations following these guidelines:

- Locate the approximate position of the collar with backsight and foresight according to the planned azimuth. Allow enough space on the site for all the contractor's equipment, including core storage and sumps.
- Arrange for site preparation equipment (if needed) and ensure that the site preparation is completed as specified before the excavator leaves the drill site. The site must be level to prevent trip hazards.

- Ensure the drill site is not in a hazardous location to workers or equipment, including landslides and falling trees.
- Prepare a piece of wood to temporarily mark the location of the drill collar at the end of drilling and avoid the hole from being covered.

10.2.5 Drill Rig Setup

The drilling supervisor receives instructions detailing the following for each hole: hole number, location of the collar, hole dip and azimuth, expected hole length and methods or downhole survey.

The drill rig is aligned with a Reflex TN14 gyrocompass. Prior to 2023, azimuth alignment was done using surveyed foresight (collar) and backsight (sighter) stakes (pegs) and running a string or laser sight between the foresight and back site to aid in alignment.

The following steps must be followed:

- Collar the hole within one (1) metre of the surveyed collar location.
- The hole dip (inclination) was checked using a clinometer.
- Once the rig setup is complete, obtain another set of handheld GPS coordinates for the drill collar and record this location as the current one in the database.

10.2.6 Drillhole Surveys

Collar Survey: The drill collar monument is surveyed by differential GPS to obtain accurate coordinates and elevation.

Downhole Survey: The hole is surveyed using a Reflex surveying tool (Gyro Sprint-IQ-A4 or EZ Trac A4). Survey measurements are taken either continuously or at every ten metres (10 m) while the instrument is lowered and hoisted. After the survey tool has been hoisted and disconnected, the survey information is extracted by connecting a portable tablet and downloading the data. The test is considered successful after an examination of the results at the drill site.

10.2.7 Environmental Management

Site waste control procedures include the following:

- Washing and servicing equipment, cleaning hydrocarbon or other chemical spills, and waste disposal is part of the responsibilities of the drilling contractor.
- Waste hydrocarbons, empty hydrocarbon containers and chemical containers must be placed in leak-proof receptacles and disposed of at the nearest designated hydrocarbon containment area.
- If fuel or oil spillage occurs at the drill site, the driller immediately notifies the geologist in charge, who advises collecting and disposing of the contaminated rock or soil. Clean-up of spills is at the contractor's cost if not otherwise specified in the drilling contract.
- Waste generated by drilling operations is placed in suitable containers and disposed of at a designated trash disposal site.
- The drillers will contain water generated from drilling operations in sumps at the drill site.

Site-specific rehabilitation procedures include the following:

- Removal of waste and equipment.
- Plugging of drillhole collars.
- Backfilling of sumps.
- Removal or scarifying of sample rejects from splitters.
- Clean-up of all fuel and oil spills.
- Removal of all bulk bags from the site location.

10.2.8 Recovery and Re-drilling a Hole

A drillhole is re-drilled when:

- The sample order has been lost, and sample quality and recovery have been compromised.
- Excessive deviation from the planned drillhole trace.
- The hole did not reach its target depth because drilling equipment (rods, hammers, or bits) lost down-hole could not be recovered.
- The contractor informed the geologist of dangerous drilling or poor hole conditions that hindered further progress.

The decision to re-drill a hole is also based on discussions with the contractor's on-site representative.

10.3 Diamond Drilling

Drilling by Reunion first commenced at the Oko West Project on December 9, 2020, and at the time of writing, one (1) diamond drill rig, and one (1) RC drill rig is on-site.

The following diamond drill rigs have been used:

- One (1) Longyear 90.
- One (1) Sandvik 740.
- Six (6) Sandvik 710's (Figure 10.1).

Figure 10.1: Sandvik 710 Diamond Drill Rig at the Oko West Project

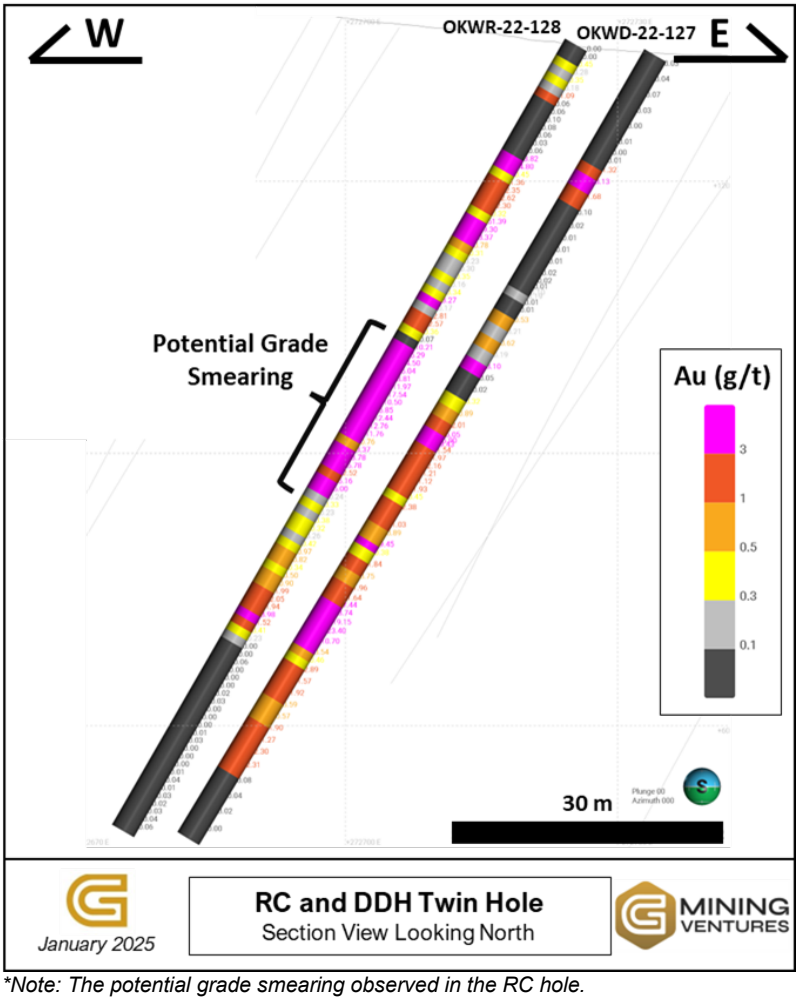
Source: GMS, 2024.

From the project outset, both diamond and reverse circulation drilling were undertaken simultaneously. Diamond drill core provides excellent geological information both in saprolite and unweathered rocks. RC drilling provides poorer-quality geological information but is a fast, cost-effective, and reliable sampling

tool for the lateritic profile. At the initial stages of the program, RC drilling was used to scout for gold mineralization in the saprolite and delineate the footprint of mineralization near surface. The resulting gold anomaly was followed up with diamond drillholes that tested not only the mineralization's lateral extent, but also depth continuity. Diamond drill core demonstrated that several initial RC holes were drilled sub-parallel into the mineralized envelope at a sub-optimal angle. As the project evolved, RC holes were restricted to the shallow western edges of the gold anomaly and subsequently entirely replaced by diamond drilling as resource delineation drilling started.

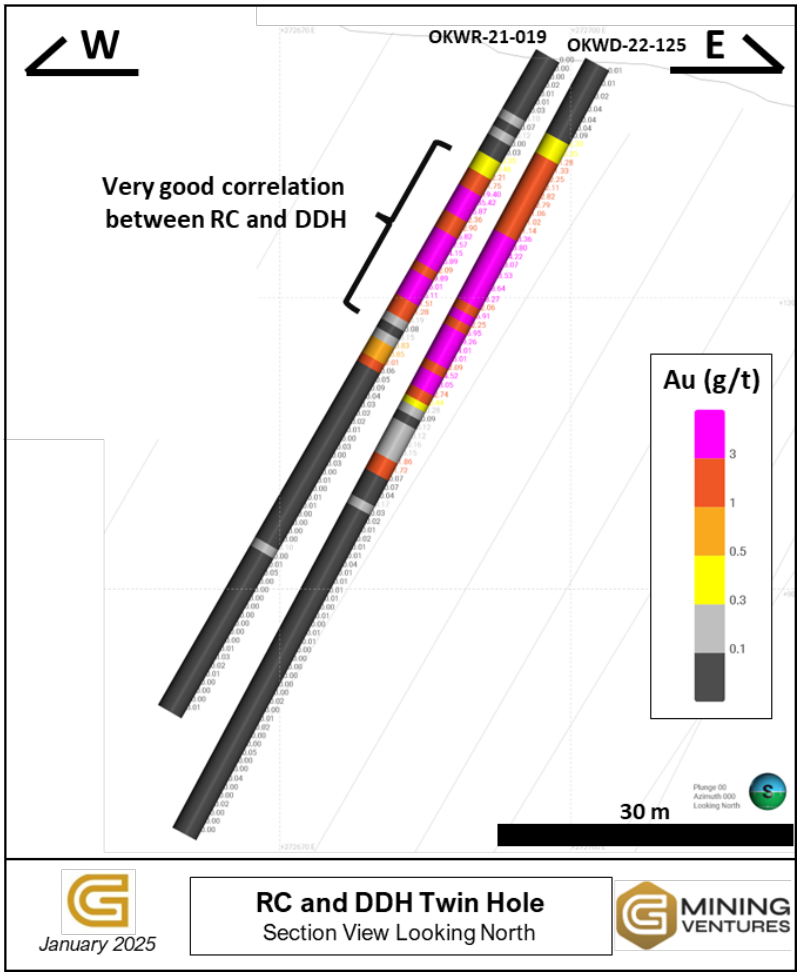
A total of 16 diamond twin holes were drilled to validate the early RC assays. Only one (1) discrepancy was acknowledged between RC OKWR22-128 and diamond twin hole OKWD22-127. The RC hole was flagged due to potential grade smearing and removed from the mineral estimation database. Figure 10.2 presents the mineralized intervals from the RC hole with the twin diamond hole. Figure 10.3 presents an example of RC and DD twin holes assayed intervals that correlates well together.

Figure 10.2: RC OKWR22-128 and Diamond Twin Hole OKWD22-127



**Note: The potential grade smearing observed in the RC hole.*

Figure 10.3: RC OKWR21-019 and Diamond Twin Hole OKWD22-125



**Note: The similar grade ranges observed in both RC and DD twin hole.*

Diamond drill core recovery is excellent in unweathered rock, often greater than 95% but poorer in RC drilling, with a sample recovery of an average of 64%. Table 10.2 presents the sample recovery statistics by type of drilling and weathering. A higher sample recovery is observed for the RC drilled within the deposit footprint and it should be noted that most of RC drilling is supported by nearby DD holes.

Table 10.2: Drilling Recovery by Weathering Domain – Inside the Deposit Footprint

Type of Hole	Weathering Domain	Num. of Measurements	Avg. Sample Recovery %
DD	Alluvium / Colluvium	1,558	84.8
	Saprolite	14,115	92.2
	Transition	6,090	92.3
	Fresh Rock	58,504	98.2

Type of Hole	Weathering Domain	Num. of Measurements	Avg. Sample Recovery %
RC*	Alluvium / Colluvium	2,844	51.1
	Saprolite	16,868	72.9
	Transition	6,746	70.7
	Fresh Rock	13,072	64.0

**Note: RC drilling recoveries are an approximation. Only Maxidrill 400 data were used for the evaluation of recoveries.*

All diamond drilling, regardless of the drill rig, followed these procedures:

- Drilling azimuths have been consistently targeted at 270 degrees, except for a few holes designed to intersect interpreted cross-cutting structures and drilled at due north or south.
- Drilling dips of most holes have been designed to start at -60 degrees. A few in-fill holes had different initial angles, and deeper drilling uses higher angles (up to -80 degrees) due to surface constraints.
- The entire lateritic profile is drilled with HQ core size (63.5 mm core diameter), changing to NQ size (47.6 mm core diameter) once in unweathered rocks.
- The drill core is placed in core boxes with pertinent Hole-ID, drill run and depth information at the drill site and transported by the drilling company to the camp core shed for logging and sampling.
- Drillholes are always initiated and stopped in the presence of a project geologist.
- The drilling crew will perform drill core orientation in unweathered rocks as often as practically possible.
- The drill crew will perform down-hole surveys using the prescribed equipment at the end of each hole.

10.3.1 Wedging

In diamond drilling, wedging is a procedure used to change the direction of the borehole using permanent or retrievable wedge. At Oko West, wedging is used to ensure specific targets are reached when deviation is observed in drill holes. Wedging is also used to create secondary holes branching off from a single “Parent Hole”, reducing the meterage needed for resource conversion during infill programs.

A summary of the wedging procedure performed by Major Drilling is described below:

- Pre-installation checks and preparation for wedge installation.
- Off-Bottom wedges and wedges placement.
- Reaming and core drilling.

- Downhole survey to verify wedges position.
- Proceed with normal drilling operations.

10.3.2 Down Hole Motor (DHM)

For hard-to-reach targets, Down Hole Motor (DHM), has been implemented in late 2023 at Oko West.

The DHM simplified procedure is as follows:

- Preparation and maintenance.
- Bypass wedge installation and hole cutting.
- DHM handling and testing.
- Drilling and testing, using Reflex tools for direction.
- Drilling verifications for direction.
- Reaming and resuming drilling.

10.4 Reverse Circulation Drilling

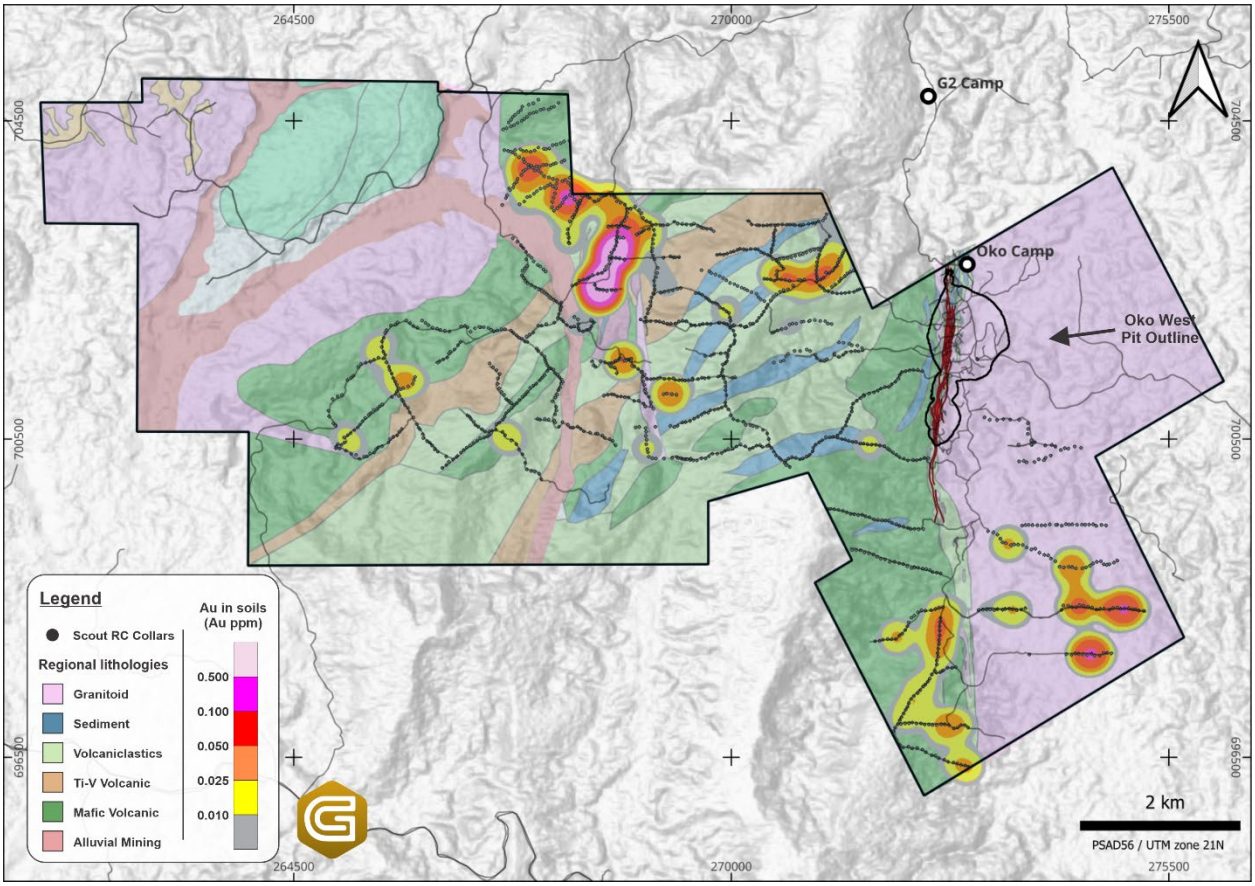
Reverse circulation drilling started on July 26, 2021, and continues using a Maxidrill 400 mounted on a Cat 315DL base (Figure 10.4) using four-inch drill pipes and a four-inch DTH hammer. It uses a Metzke cyclone with two (2) cone chutes and a rotary splitter attachable to the cyclone. A Sullair 1150 / 1350 CFM at 500/350 psi compressor mounted on a steel tracked carrier is occasionally supported by an Atlas Copco B4-41 booster. Major Drilling owns and operates this rig.

On November 7, 2022, Major Drilling mobilized a scout rig (ED-167) mounted on a Morooka MTS 850, capable of rotary air blast, air core and reverse circulation drilling to depths up to 100 m. This rig is being used for reconnaissance scout drilling of geochemical anomalies (Figure 10.5).

Figure 10.4: Reverse Circulation Rig (compressor and booster not shown)

Source: Reunion, 2022. Photo by C. Bertoni

Figure 10.5: Map of Scout RC Program with Gold Anomalies on Lithology



Source: GMIN, 2024

The drill rig has an onboard sampling system consisting of a rotary splitter attached to a Metzke cyclone (Figure 10.6). Where drilling has not intersected water and the sample is dry, each primary sample is:

- Material for an entire interval is weighed to estimate recovery. The recovery is calculated based on the theoretical mass drilled every metre by weathering status.
- Three (3) bags of samples are collected from the splitter. Samples from these bags are then remixed and shaken six (6) times in two (2) directions for homogeneity.
- The splitter on the rig separates the actual samples for lab processing, the field duplicate sample and the bulk sample for storage and future reference.
- All primary sample bags and bulk sample bags are marked with a unique sample number identifier. A pre-printed sample tag with the corresponding number is placed in the primary sample bag.
- The sample is weighed to ensure a mass of approximately 2 kg.
- Duplicate samples are placed in bags with corresponding hole numbers written on them, returned to camp and placed in an appropriate storage location.

Figure 10.6: Rotary Splitter Attached to Metzke Cyclone

Source: Reunion, 2022

For each primary sample, the corresponding reject sample is:

- Sub-sampled, sieved, and two (2) representative samples are taken for the chipboard and chip tray.
- Logged noting geologic features and photographed.

The following drilling and sampling practices are followed to minimize contamination between samples:

- Drilling stops (5 - 15 seconds) while air circulation is maintained to flush out all remaining material at the end of each sample interval. This time is occasionally reduced when drilling in a zone with a high influx of water, reducing contamination.
- The splitter is cleaned thoroughly after each rod change (6 m). In case of water influx, this cleaning frequency might be changed.
- The inside of the cyclone and the collection box are thoroughly cleaned after each drill rod change when drilling in fresh rock and, more often as required in weathered rock or when drilling under wet conditions.
- Sample bags are tied off immediately after filling to prevent contamination using a cable-tie.

The intersection of groundwater is expected but may be avoided by using an auxiliary compressor and booster. Where wet samples are unavoidable due to groundwater inflow, the following process is employed.

- Connect the booster at the first sign of moisture, ensuring that subsequent samples do not suffer contamination and recoveries improve.
- Collect a preliminary sample split (sub-sample) by spearing in six (6) different directions through the bag with a scoop or available spatula.
- The geologist should abandon the hole if water is encountered for three (3) consecutive metres before the planned hole depth is reached.

11. SAMPLE PREPARATION, ANALYSIS AND SECURITY

This section describes the sample preparation, analysis, and security procedures for the diamond drilling (DD), reverse circulation drilling (RC) and trenching programs performed on the Oko West Project. It also includes a quality assurance and quality control (QA/QC) program as part of the sample assaying process.

11.1 Core Handling and Sampling

A total of 518 DD, 1,775 RC, 39 wedges of DD holes and 111 trenches were completed between January 2020 and September 2024. The following section describes the procedures for core handling, logging, and sampling implemented by the Company on the Oko West Project.

Drill rigs are aligned with the planned drillhole information and inspected by the project geologists. The hole number, azimuth, and dip are marked at the collar of the drillhole.

Core boxes for each drillhole are labelled with a unique number consisting of an alpha prefix, usually denoting project name (OKW) followed by one (1) letter indicating the drill type (“D” for diamond, “R” for reverse circulation) and year (e.g. OKWD22-067). Twinned or redrilled holes have the same Hole-ID as the original with a letter suffix (e.g. OKWR22-005A).

Numbered boxes containing drill cores are transported to the Company logging core shack. Geotechnical measurements are then conducted; recovery, RQD, orientation marks, core diameter and hardness are recorded.

Geological logging and structural measurements are taken by the geologist in charge of the drillholes. Data is entered directly into the drillhole master database. For the core logging, the following data is retained and described: lithology, alteration, structure, mineralization, and sample intervals.

Diamond drill core samples are collected on average at every 1.20 m from drill core but vary between 0.10 m and 3.90 m. Sample intervals are marked by geologists. Samples are selected in potential mineralized zones based on logged geological features, such as rock type, mineralization, alteration, veining, etc.

Blanks, certified reference materials (CRMs or standards), and duplicates are inserted at the same time as the sampling process is performed.

The core boxes are then photographed, with details of each core box clearly visible. Digital photographs are taken dry and wet and archived for review purposes.

The core cutting is supervised by the geologists and geotechnicians to ensure that the samples are cut in order, and the sample number vs the depth are not mixed up. Samples are marked along the cutline with a different colour from the core-orientation line. The core is cut from the bottom to the top to avoid combining with the sample below. The same side of the core is always put in the sample bag. The side with the orientation line must be kept in the core box, while the other side goes into the sample bag (Figure 11.1).

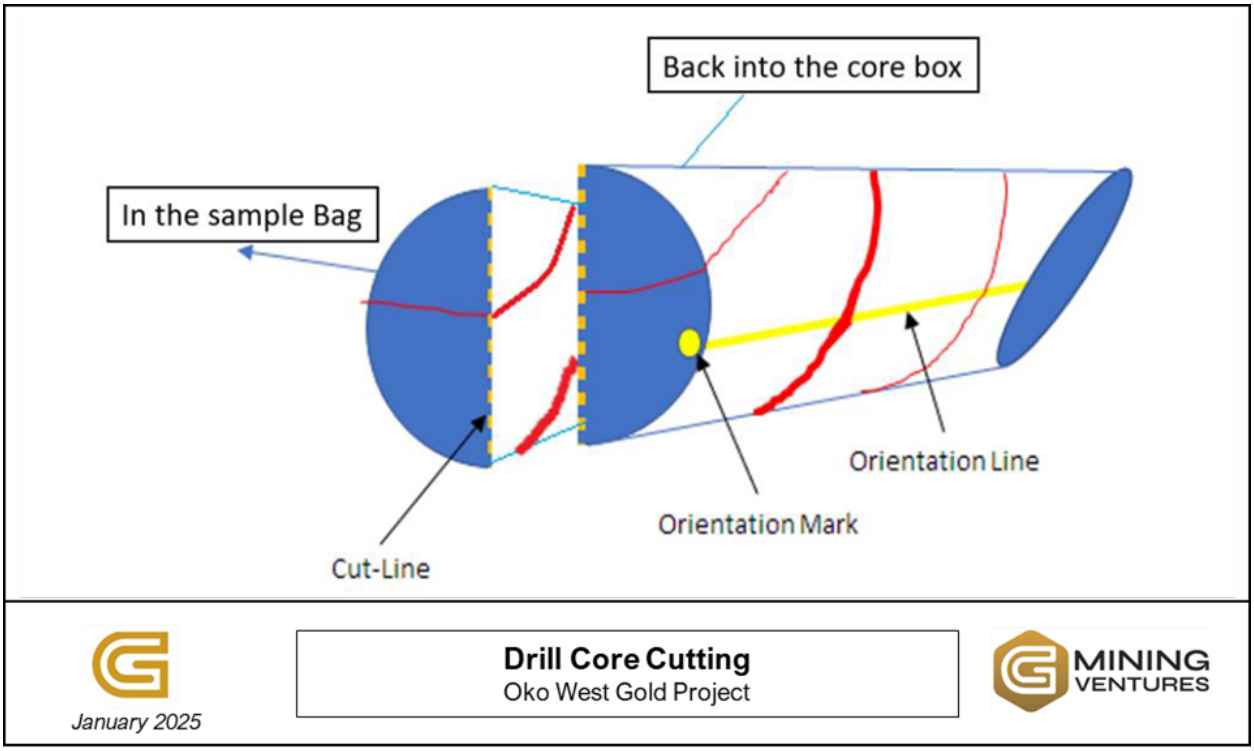
The second stub of the sample tag remains stapled in the box at the end of the sample interval to identify the sample length and location The third stub and a metal tag bearing the same sample ID is inserted into the corresponding sample bag with ½ the core.

In the case of field duplicates, the two (2) sample stubs will be stapled side by side in the box. The primary sample is always ½ core. For a ½ core duplicate, the second half of the core goes into the second sample bag, leaving no core in the box. For ¼ core duplicates, the remaining half will be cut in half again. The ¼ goes into the secondary sample bag and ¼ is kept in the core box.

Bags are closed with zip-ties immediately after the sample is placed in the bag. Samples are placed in order in the core shed outside the cutting station. Samples from different holes are kept separate. Drill core samples, blanks, and certified standards are then bagged and ready to be submitted to the primary laboratory.

The laboratory is instructed to return the rejects with the sample stubs in the bags even if there is no sample in the bag (e.g. bags containing a standard).

Figure 11.1: Drill Core Cutting



Source: Reunion, 2022.

11.2 RC Sample Handling and Sampling

Reverse circulation (RC) chip samples are collected at every metre. RC drilling protocols and contamination mitigation measures are described in Section 10.

The splitter on the rig will produce 2 kg samples for the primary lab, the field duplicate sample and the bulk sample for storage and future reference. All primary sample bags and bulk sample bags are marked with a unique sample number identifier. A pre-printed sample tag and corresponding number are placed in the primary sample bag.

Duplicate samples are placed in bags with corresponding hole numbers written on them, taken back to the exploration camp, and placed in an appropriate storage location. Rejects are sub-sampled, sieved and two (2) representative samples are taken for chipboard and chip tray to be logged, noting geologic features, and photographed (Figure 11.2).

Figure 11.2: Chip-Logging Geologist

Source: Reunion, 2022. Photo by C. Bertoni.

For the RC samples, the second and third stubs are inserted in the sample bag, still attached to each other. The laboratory will detach these, placing one in the coarse reject bag and another in the pulp reject bag.

11.3 Sample Transit, Security and Chain of Custody

The sample tags are kept in a locked cabinet and issued to the core shed personnel on an as-needed basis. The sample tag stubs are completed, and the books used are kept in a sample book library for future reference. Sample tags contain the sample information, including date, target, hole or trench, interval from--to in metres, sampler name and analytical code.

Samples submitted display the sample number and are individually tied with plastic tags and packed in rice bags (six samples per rice bag), which lists the project initials (Oko West) and the batch number (photo

below). The rice bags are also securely tied (Figure 11.3). Access to samples is only possible by cutting the tag. The laboratory is instructed to recycle the sample bags, return the rejects on the same bags, and show the original batch numbers. In the case of damaged bags, the lab is to replace them, re-writing the same labels on a new bag.

Figure 11.3: Bagging of Samples

Source: Reunion, 2022. Photo by C. Bertoni.

The proper storage of sample rejects received from the lab is essential for a proper QA/QC system; thus, insuring the integrity of samples and their easy retrieval. The laboratory must return rejects quoting their original batch or work order numbers, as these rejects will be stored by batch groups. Laboratories create their own job numbers, which are not used for storage records.

Pulps are stored in Kiva plasticized cardboard NQ core boxes by a batch of 40 samples (photo below) and stored safely in locked sea containers (Figure 11.4).

Coarse rejects are stored in raffia bags at the reject library on shelves by batch groups.

Figure 11.4: Pulp Reject Storage

Source: GMS, 2024.

At the beginning of the project, the samples were sent via boat and truck to the primary laboratory in Georgetown (Guyana), accompanied by one (1) company employee for the entire trip to witness that all samples reach the laboratory safely. Now, the samples are sealed on site, loaded on the sample truck, and sealed to the truck tarp. The truck driver signs a manifest on site and the lab confirms receipt of the truck with tags intact upon delivery. Along with the requisition form, a receipt form is submitted to the lab, listing the samples included in the batch and containing space for the signature of a lab representative at the time of delivery. The lab acknowledges the receipt of a work order by e-mail to the company. The primary laboratory, Activation Laboratories Ltd. (Actlabs) sends pulp duplicates directly to the secondary laboratory (MSA Labs) for umpire check assays.

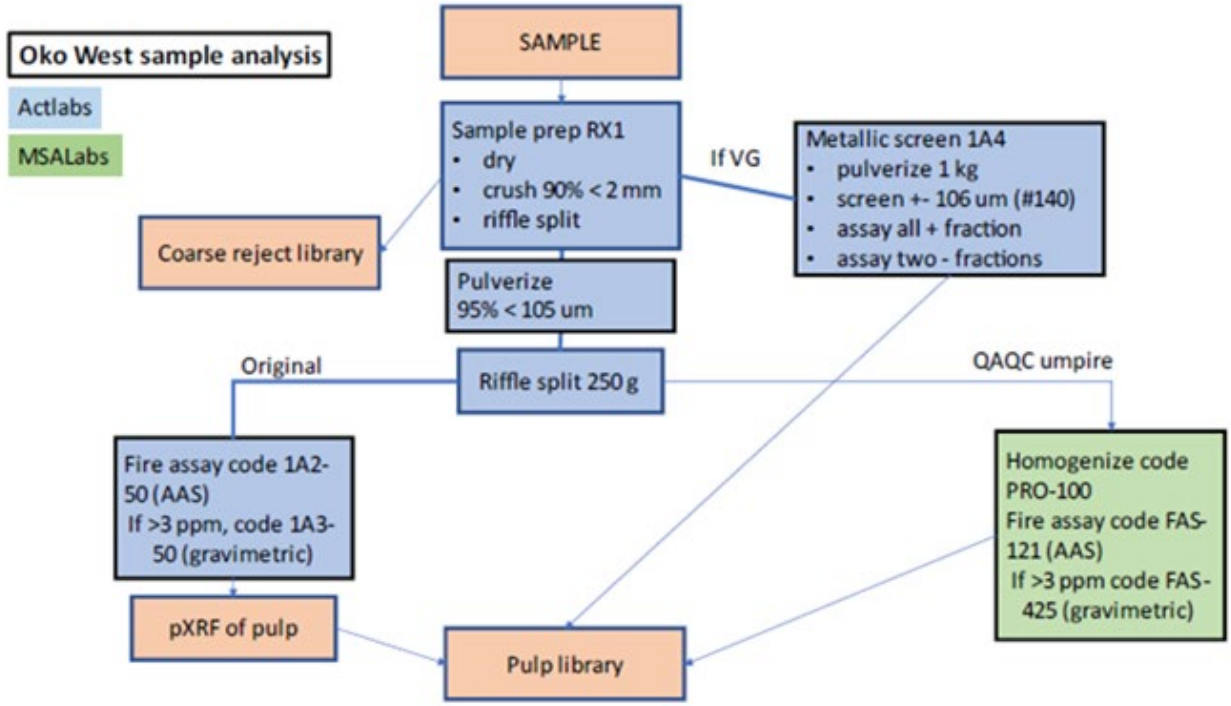
11.4 Sample Analysis Methods

Sample batches are prepared following the Actlabs Code RX1 procedure. Samples are weighed, dried, crushed (< 5 kg) to a fineness of 80% passing 2 mm. A riffle split of 250 g is taken from the crushed material and pulverized (mild steel) to 95% passing 105 µm (140 mesh). The laboratory technician uses sand to clean in between each sample.

At Actlabs, gold analysis code FAAA-1A2 is performed using a 50 g fire assay (FA) with atomic absorption spectrometry (AAS) finish. For gold values above the upper detection limit (> 3,000 ppb), samples are assayed by fire assay with a gravimetric finish (FAGRA-13A). If visible gold is observed by the geologist during the logging and sampling, the analytical method 1A4 Au fire assay metallic screen is prioritized, and the sample before and after the visible gold is also analysed using the metallic screen method.

At MSA Labs, pulp rejects samples from the primary laboratory are homogenized by mat rolling 100 times (code PRO-100) and gold assaying is performed by fire assay with AAS finish of 50 g pulverized material (FAS-121). If the gold value is above the upper detection limit (> 3 ppm), then a gravimetric finish is used (FAS-425). Figure 11.5 illustrates the analytical process selected to assay the Project gold samples.

Figure 11.5: Sample Analysis Procedures Schema as Implemented at Oko West Project



Source: Reunion, 2022.

11.5 Data Management

The assay reports by both the primary and secondary labs are distributed by e-mail directly to recipients listed in the work order, including gDat Solutions (www.gdatsolutions.com), a third-party, independent database manager. The report includes a CSV file and a PDF file, both containing the same information. The report includes assay results for both GMIN submitted samples and CRM, as well as the results for the laboratories' CRM. CRM provided to GMIN by the laboratories are part of their own internal QA/QC program. gDat imports the QA/QC results into the acQuire software, which then generates graphs and statistics for each QA/QC sample, showing whether they meet the QA/QC criteria established by GMIN. A report is sent by gDat via e-mail to the Company senior geologists, who review the results. If results are approved, gDat is informed by e-mail and incorporates the results into the database. If not, GMIN will ask the lab to re-run the samples until the results are approved. Only when approved by the project geologists will the assay results be incorporated into the database by gDat. All the e-mail correspondence is kept on record.

gDat Solutions use the acQuire Geoscientific Information Management (GIM) with complete independence. They are the only personnel authorized to introduce database changes and make copies available to third-parties. Project geologists can consult the database and download its contents via open database connectivity (ODBC). No personnel have direct access to the database.

11.6 Density Measurements

Bulk density measurements are taken in-house on all representative cores from the lithological intervals, including both mineralized and non-mineralized units, with varying degrees of hydrothermal alteration and weathering. The frequency of density measurements is determined by the project geology team. To determine the dry bulk density of a sample, the water displacement method is used. The sample is assumed to displace a volume of water equal to its own bulk volume. The sample, a piece of dry and clean drill core, is weighed to determine the dry mass using an electronic scale which is connected to a suspended basket immersed in water. The entire piece of the core is submerged in the water and weighed again. Saprolitic core samples are tightly wrapped with a thin film of plastic and weighted. The sample is immersed in water within a suspended basket and weighed again (Figure 11.6).

Figure 11.6: Density Measurement Balance Setup

Source: Reunion, 2022

11.7 Quality Assurance and Quality Control (QA/QC) Procedures

The Company's exploration team has implemented a rigorous quality assurance and quality control program (QA/QC) to ensure the validity and integrity of its sampling procedures and data management. The QA/QC protocol applies to all drill holes (DH and RC) and trench samples.

The QA/QC protocols were designed to certify drill holes sample collection, analysis, and data management and were designed in accordance with CIM Mineral Reserve and Mineral Resource Best Practice Guidelines (CIM, 2019).

The QA/QC program adopted for all drill and trench sampling is sent to the primary lab in batches corresponding to an entire trench or drillhole. Samples from different trenches or drill holes are not mixed up in the same submission order. The insertion of control samples is distributed among the original drillhole or trench samples in each batch. Certified reference materials (CRMs or standards) and blanks include one (1) control sample every ten (10) samples interchanging between standard and blank, or the equivalent of one (1) blank and one (1) standard at every 20 samples (5%). The position of blank and standard samples is adjusted to control mineralized intervals and test lab contamination.

The Project geologists request the primary lab to generate a duplicate pulp sample at a frequency of one (1) in 15 to 20 samples (about 5%) from expected mineralized and unmineralized intervals. The primary lab produces these samples and forwards them to the secondary lab for check or umpire assays, including additional standards provided by the Company.

The Company procedures of quality control (QC) samples are designed to insert one (1) standard, one (1) blank, and one (1) field duplicate at every 20 samples generated by drilling.

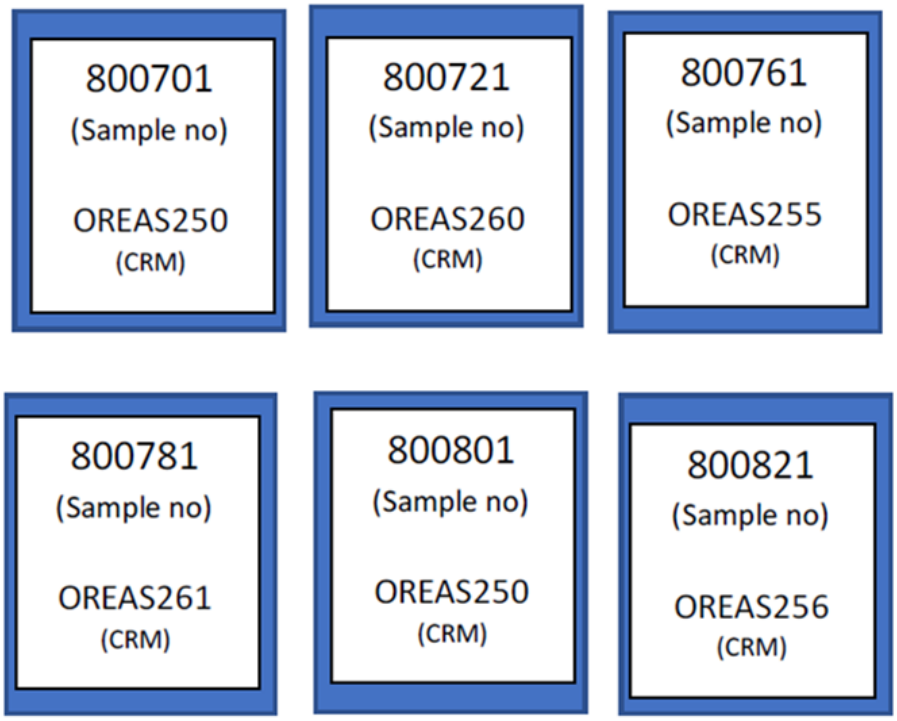
Field duplicates are taken from RC drilling. One (1) field duplicate is introduced at every 20 samples generated by drilling. This duplicate is collected directly at the rig's Metzke splitter.

For the trench channel sampling, one (1) standard, one (1) blank and one (1) field duplicate are inserted at every 20 samples generated by sub-horizontal channels near the base of the trench. The field duplicate is collected directly at the channel by enlarging it.

For diamond drill holes (DH), two (2) half core field duplicates are inserted into each mineralized zone, providing the zone is long enough. At the end of 2023, this procedure was updated to require ½ core duplicates in infill drilling (40 m spacing) and ¼ core duplicates in exploration drilling (80 m spacing or greater).

All CRMs being inserted are photographed with their respective sample numbers before the CRMs' (Figure 11.7) identification is erased, thus documenting the insertion for eventual inspection or audit.

Figure 11.7: Identification of CRMs Before Submission to Laboratory



Source: Reunion, 2022.

GMS reviewed the analytical quality control data produced by the Company for the 2020, 2021, 2022, 2023 and 2024 drilling programs to confirm that the analytical results were reliable.

Data relating to blanks, certified reference materials (CRMs), field, coarse and pulp duplicates and umpire check assays data were received from GMIN in comma-delimited format spreadsheets. From September 10, 2020, to September 15, 2024, a total of 7,104 blanks, 7,275 CRMs and 6,960 duplicates were submitted to Actlabs. The control samples represent approximately 14% of the total number of samples submitted for assaying. These totals include only final and passing batches. Analyses of data from CRMs and blank samples are normally illustrated in time-series plots to identify extreme values (outliers) or trends that may indicate issues with the overall data accuracy and precision.

11.7.1 Blanks

To monitor any contamination during the sample preparation and assaying at the laboratory, blank materials are inserted into the sampling stream every 20th sample.

The blank used in the drilling and trenching programs was sourced from a barren sample of crushed granite. The Company uses a coarse blank composed of a granitoid rock aggregate from the Toolse and Teperu quarry in Guyana (Figure 11.8). The bulk coarse blank material is kept in a covered plastic bucket at the site core shed, and the blank sample is prepared in advance by placing 1 kg of the material in a zip-lock bag. This material was recently re-tested for its gold content (work order 22-OKWG-198).

Figure 11.8: Blank Used by Company as Quality Control Sample

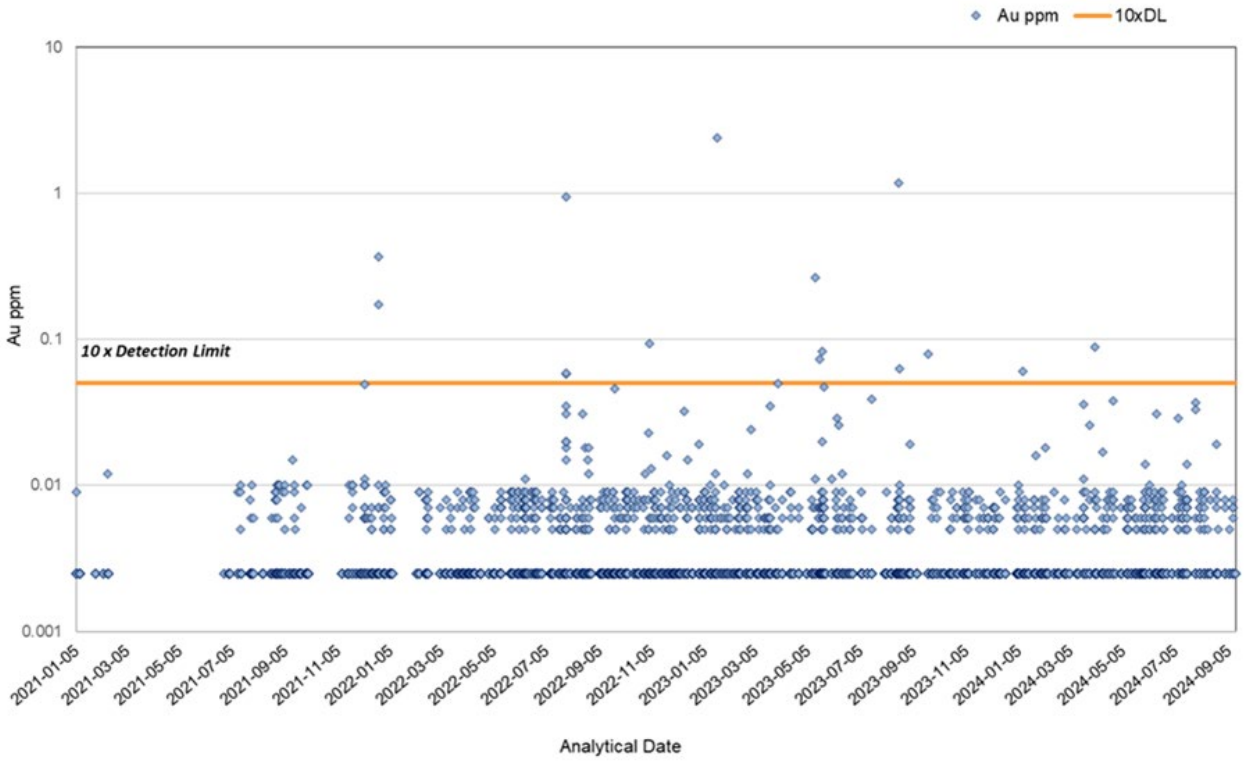
Source: Reunion, 2022.

Blank materials are considered failed when the returned gold value exceeds 0.05 ppm, which is equivalent to ten (10) times the lower detection limit (DL) of 0.005 ppm for both FAAA-1A2 and FAGRA1A3 analytical methods. A total of 7,104 blank samples were submitted to Actlabs. The results are considered good, with 99.8% of blanks falling within the accepted control limit, which demonstrates that the samples show no

systematic contamination with contiguous mineralized intervals. Some of the failures seem to be related to mislabels and the insertion of CRMs instead of the barren material.

Figure 11.9 shows the performance of the blank samples and the control limits (10 times DL).

Figure 11.9: Blanks Control Chart for Gold – 2021-2024 Drilling Program



Source: GMS, 2025.

11.7.2 Certified Reference Materials

Certified reference materials (CRMs or standards) were chosen within low and high gold grade ranges and from two (2) types of material, oxide and sulphide, used to monitor the laboratory performance and to assess bias on the analytical results. The Company purchases commercial CRMs prepared by Ore Research and Exploration (Australia) (OREAS) and Canadian Resource Laboratories. The CRM packets are kept in clean plastic containers at the field geology office and brought to the core shed only when needed to be introduced to the sample batches.

Quality control (QC) samples are classified as failures for a reference material if the results are outside ± three (3) standard deviations (SD) of the certified gold value. Failures observed in the CRM results are investigated and replaced by the re-analyzed value when it is considered necessary. Overall, the results

tabulated below indicate that the CRMs are good and minimal bias regarding the precision of the expected values vs. the assayed value is present through the assaying period.

OREAS 250b, 260 and CDN-GS-16 were removed from the quality control sampling program by the Company due to poor analytical performance at the primary laboratory, the values obtained often being between the control limits, systematically different from the certified value over several periods.

The results of the analytical quality control data produced by the CRM samples used between 2020 and 2024 drilling programs and submitted to Actlabs are summarized in Table 11.1.

Table 11.1: Summary of CRM Performance Results as Implemented on Oko West Drilling Resource Area (2020-2024) – CRM Samples Assayed by Actlabs

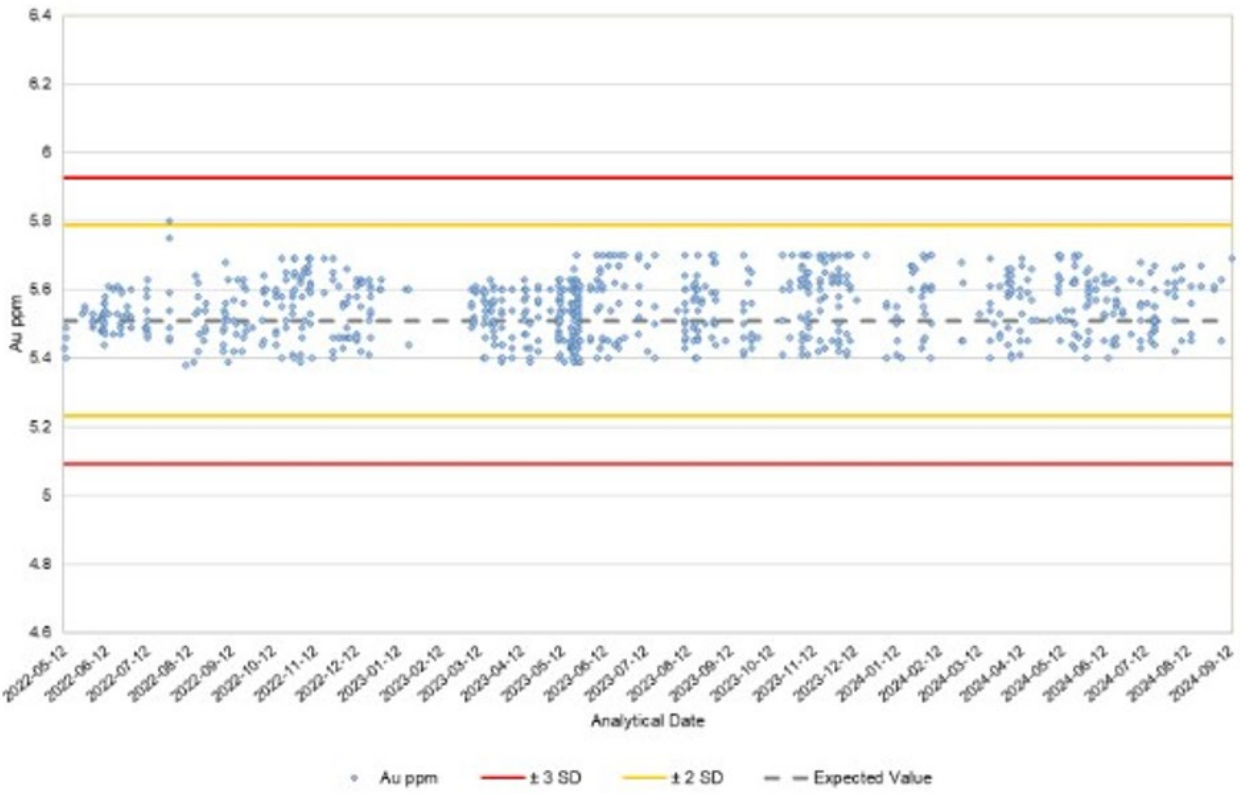
Material Type	Certified Standard (CRM)	Num. of Submitted CRM	Certified Au Value (ppm)	Standard Deviation (SD)	Num. of Failures	% Passing
Sulphide	CDN-GS-16*	11	16.48	0.315	0	100.0%
Sulphide	CDN-GS-1U*	102	0.968	0.043	0	100.0%
Sulphide	CDN-GS-P1A*	140	0.143	0.004	1	99.3%
Sulphide	OREAS211	1,428	0.768	0.027	0	100.0%
Sulphide	OREAS221*	152	1.06	0.036	1	99.3%
Sulphide	OREAS234	176	1.2	0.03	0	100.0%
Sulphide	OREAS236	1,408	1.85	0.059	0	100%
Sulphide	OREAS238*	83	3.03	0.08	0	100.0%
Sulphide	OREAS239*	84	3.55	0.086	2	97.6%
Sulphide	OREAS240	781	5.51	0.139	0	100.0%
Oxide	OREAS250*	126	0.309	0.013	0	100.0%
Oxide	OREAS250b	478	0.332	0.011	2	99.6%
Oxide	OREAS251b	514	0.505	0.017	1	99.7%
Oxide	OREAS252*	121	0.674	0.022	0	100.0%
Oxide	OREAS252b *	238	0.837	0.028	0	100.0%
Oxide	OREAS254*	217	2.55	0.076	0	100.0%
Oxide	OREAS254b	415	2.53	0.061	0	100.0%
Oxide	OREAS255*	57	4.08	0.087	0	100.0%
Oxide	OREAS256*	64	7.66	0.238	0	100.0%

Material Type	Certified Standard (CRM)	Num. of Submitted CRM	Certified Au Value (ppm)	Standard Deviation (SD)	Num. of Failures	% Passing
Oxide	OREAS256b	211	7.84	0.207	0	100.0%
Oxide	OREAS258*	43	11.15	0.259	0	100.0%
Oxide	OREAS260	157	0.016	0.0018	2	98.7%
Oxide	OREAS261	403	0.0486	0.0023	0	100.0%
Oxide	OREASH1*	6	0.012	0.001	2	66.7%

**Note: Discontinued CRMs.*

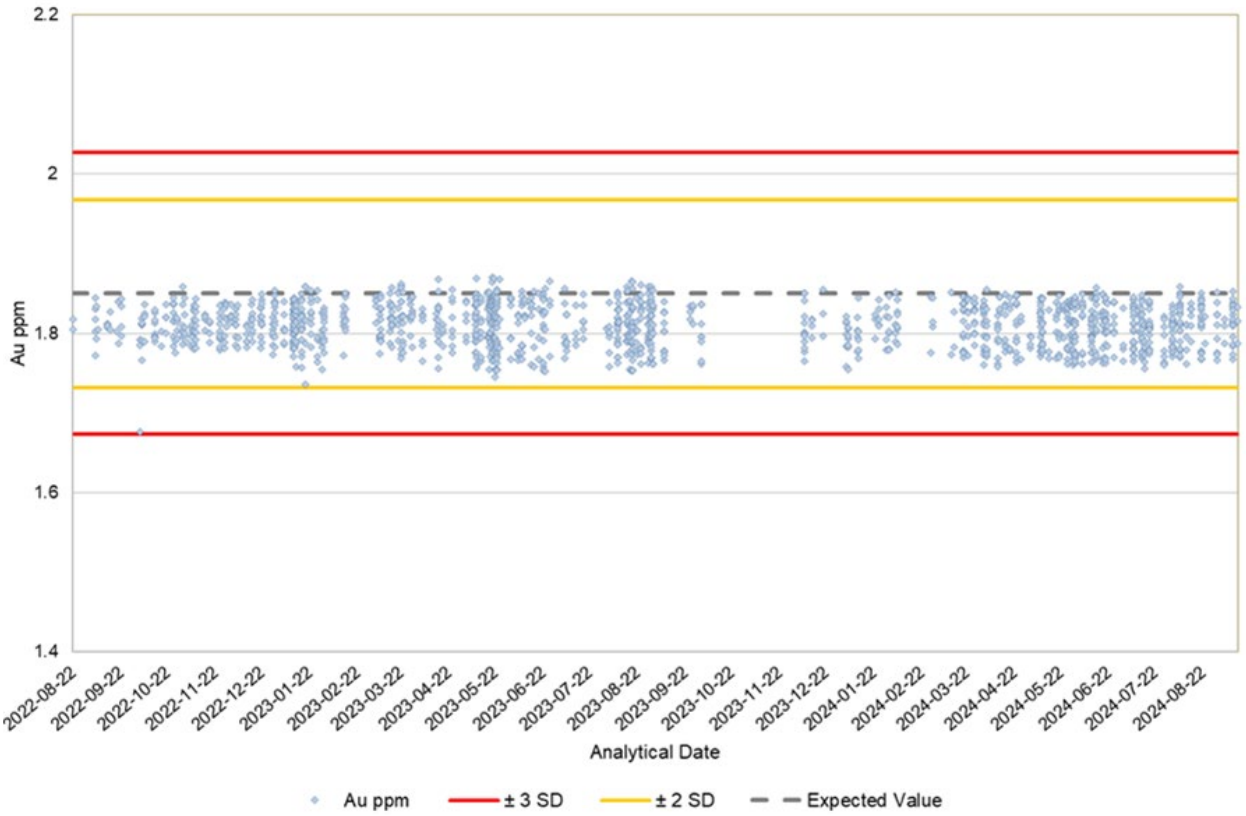
The performance of the CRMs was also validated on time-series control charts to monitor for analytical drift and abnormal assay batches. After verification, the CRMs generally performed well over time with the majority of the results within the control limits of ± 3 times standard deviation (± 3SD) of the certified recommended value. Some of the control limits of CRMs are illustrated in Figure 11.10 to Figure 11.14. The sample exceeding the expected value for OREAS 236 was identified as a mislabel.

Figure 11.10: Sample Control Chart of CRM OREAS 240



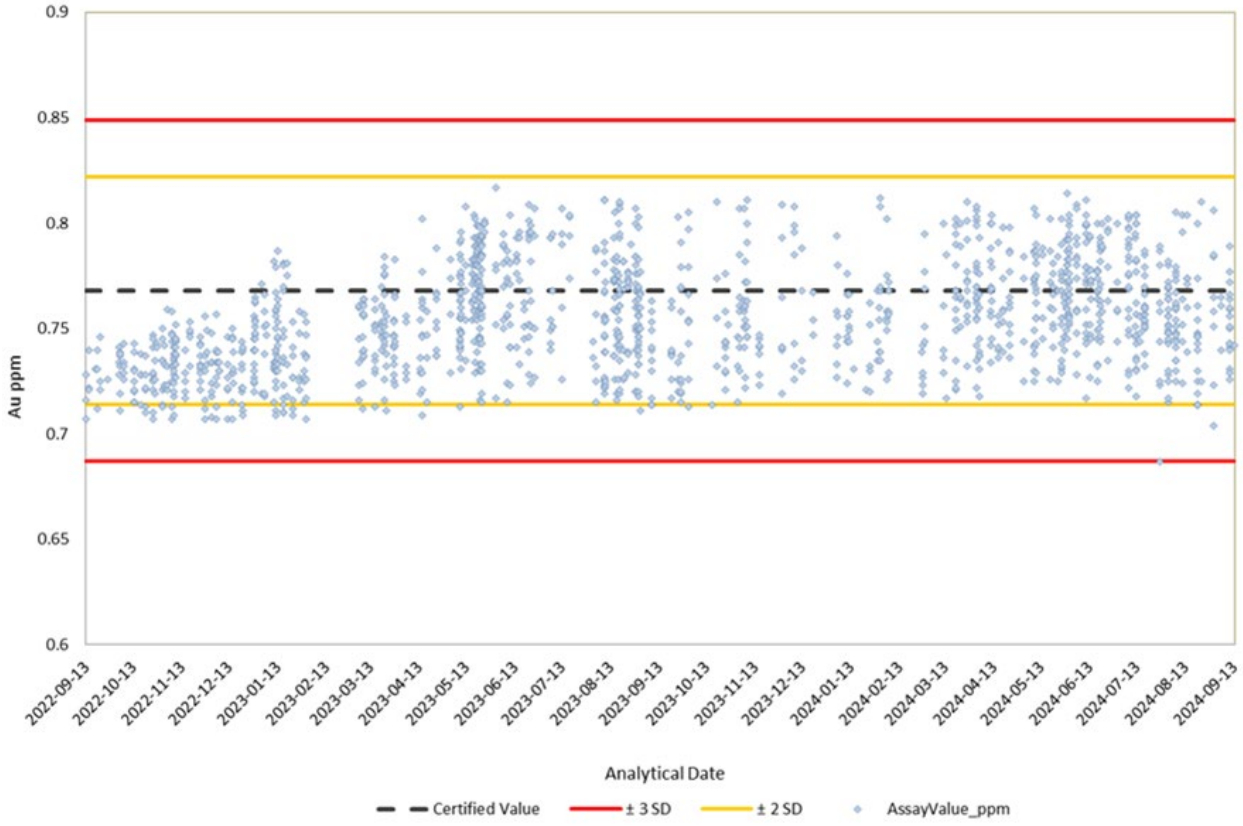
Source: GMS, 2025.

Figure 11.11: Sample Control Chart of CRM OREAS 236



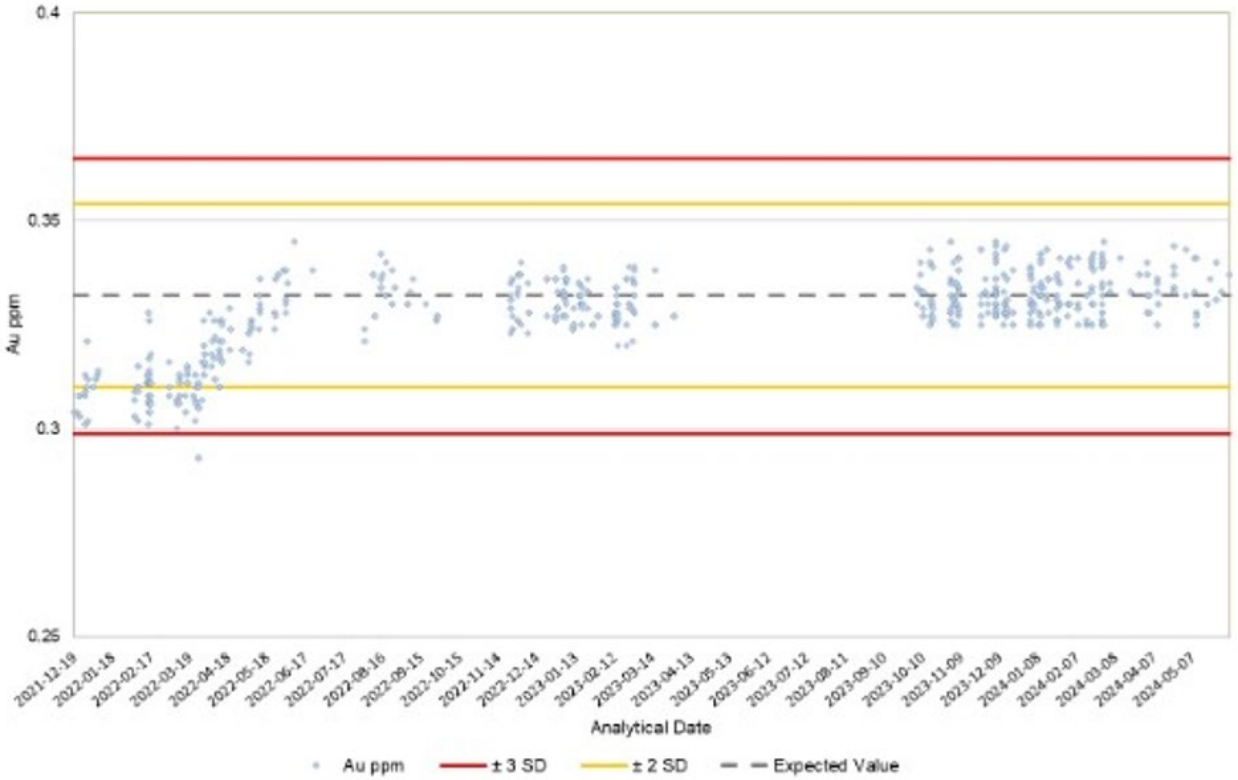
Source: GMS, 2025.

Figure 11.12: Sample Control Chart of CRM OREAS 211



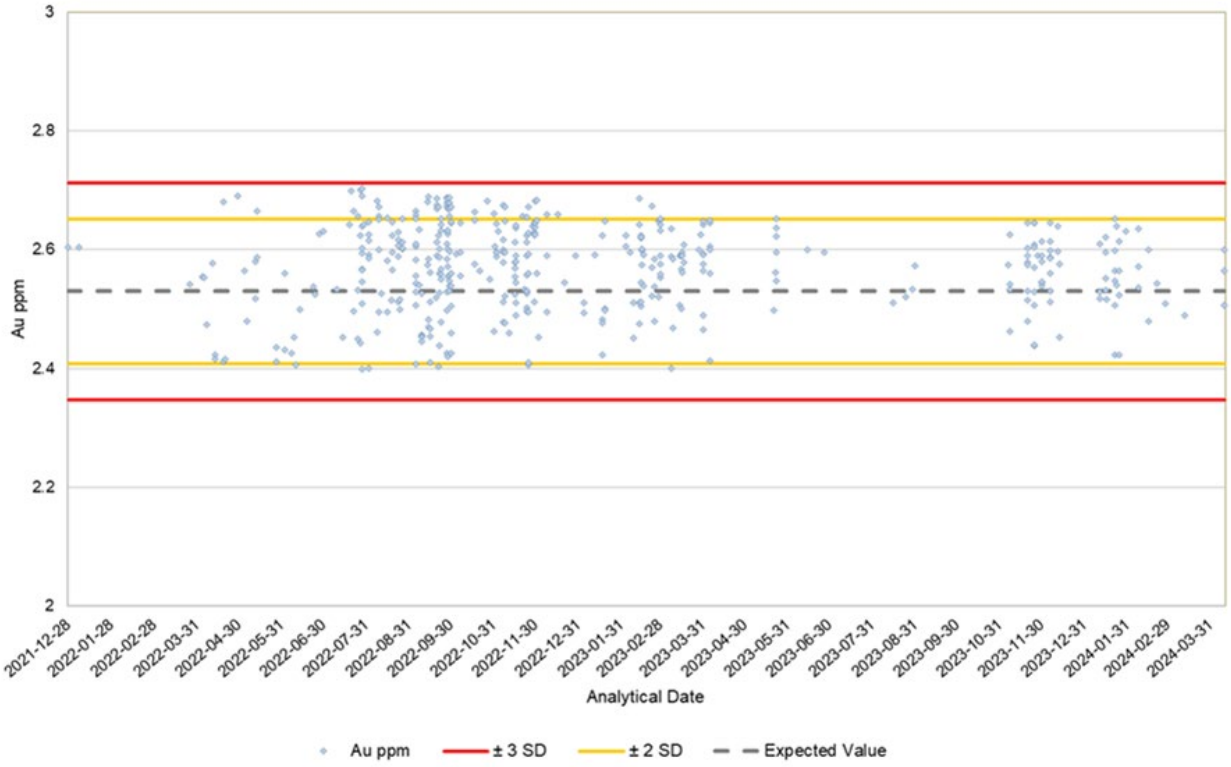
Source: GMS, 2025.

Figure 11.13: Sample Control Chart of CRM OREAS 250b



Source: GMS, 2025.

Figure 11.14: Sample Control Chart of CRM OREAS 254b



Source: GMS, 2025.

11.7.3 Duplicates

As part of the QA/QC program, duplicate samples were included as quality control (QC) measures to evaluate grade variability and monitor laboratory consistency throughout the 2020-2024 drilling campaigns.

A total of 2,428 field duplicates from RC holes, trenches, and diamond drill holes (DH) were collected and submitted for analysis using methods 1A2-50 and 1A3-50. For the RC duplicates, 979 out of 1,757 reported above 0.005 ppm and below 3.00 ppm gold for method 1A2, and 8 duplicate pairs reported above 3.00 ppm gold for method 1A3. For the trench duplicates, 214 duplicate pairs out of 271 reported above 0.005 ppm and below 3.00 ppm gold for method 1A2, and 2 duplicate pairs reported above 3.00 ppm gold for method 1A3. For the DH field duplicates, 264 out of 537 pairs reported above 0.005 ppm and below 3.00 ppm gold for method 1A2, and 111 pairs reported above 3.00 ppm gold for method 1A3.

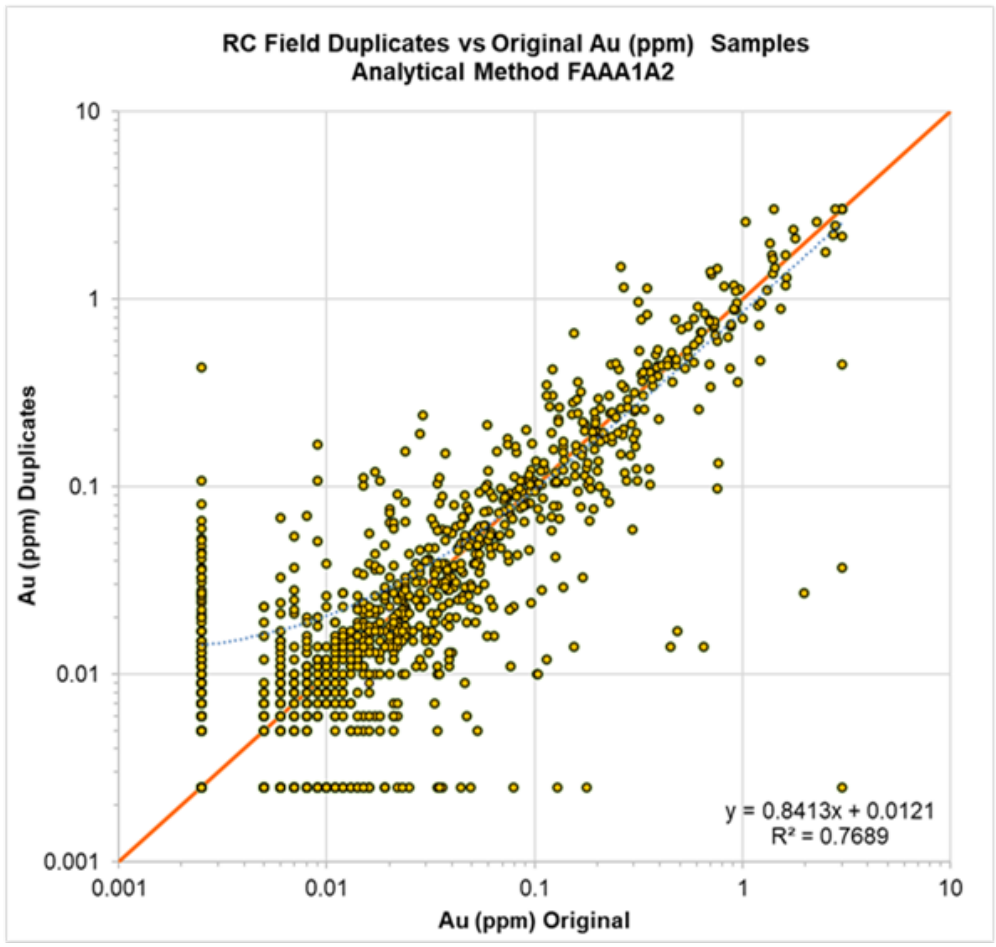
Table 11.2 summarizes the number of samples by duplicate type submitted by the Company to assess the reproducibility of assays and identify any sampling bias.

Table 11.2: Quality Control (QC) Duplicates Submitted by the Company

QC Sample Type	Total of QC Samples (FAAA-1A2)
Field Duplicate (RC)	1,744
Field Duplicate (Trench)	269
Field Duplicate (DDH – ½ & ¼ core)	414
	Total of QC Samples (FAGRA-1A3)
Field Duplicate (RC)	8
Field Duplicate (Trench)	2
Field Duplicate (DDH – ½ & ¼ core)	111
Coarse Reject Duplicate	2,214
Pulp Duplicate	2,543
Total	**6,960**
Analytical Method	**Umpire Pulp Duplicates**
FAS-121	3,008
FAS-425	108
Total	**3,116**

RC field duplicates are sent to the primary laboratory to calculate field, preparation, and analytical precision. The results of RC field duplicates vs. original gold values assayed by FAAA-1A2 are presented in Figure 11.15. The RC field duplicates returned results with a good correlation of determination (R^2) of 77% and a linear regression slope of 0.84. The RC field duplicates results are similar in average when compared to the original assays showing a relative difference of 2.2% (Figure 11.15).

Figure 11.15: RC Field Duplicates – 2020-2024 Drilling Programs

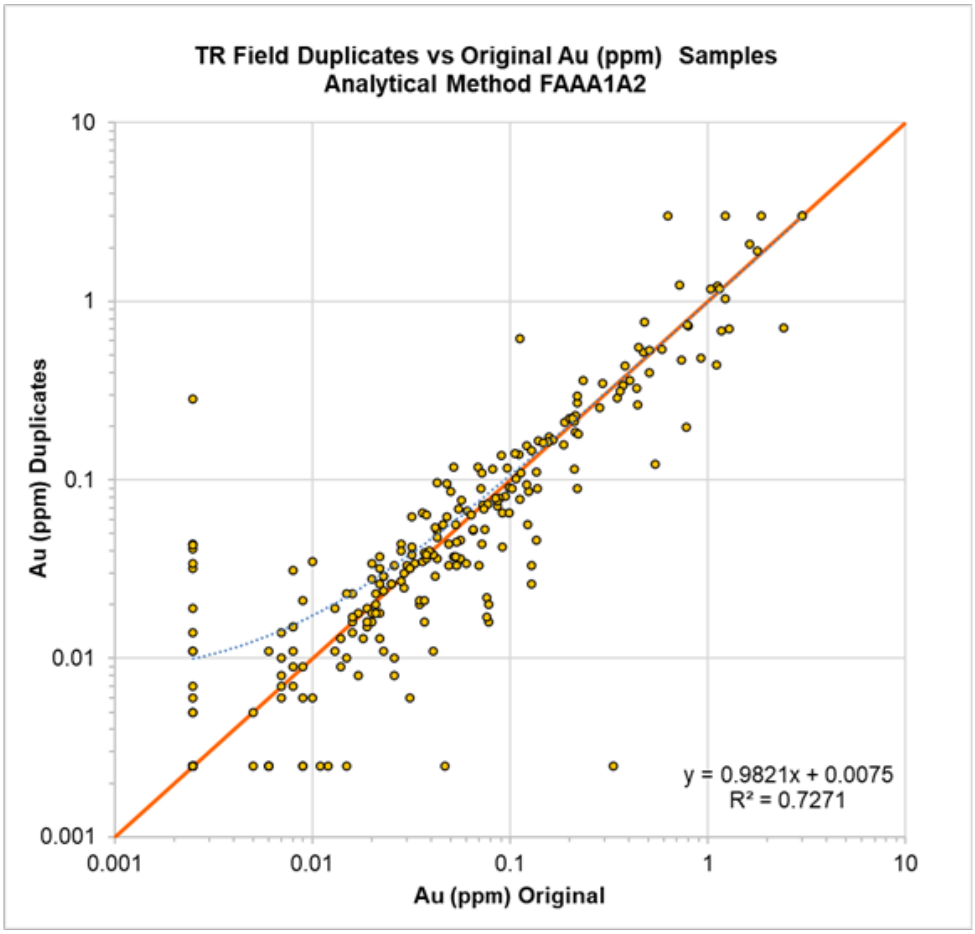


Global Statistics	Au_ppm_Original	Au_ppm_Duplicate
Mean	0.092	0.094
Median	0.007	0.007
Standard Deviation	0.33	0.34
Sample Variance	0.11	0.12
Pearson Correlation	0.88	
Minimum	0.0025	0.0025
Maximum	3	3
Count	1,744	1,744
Confidence Level(95.0%)	0.01	0.01

Source: GMS, 2025.

The results of trench field duplicates vs. original gold values assayed by FAAA-1A2 are presented in Figure 11.16. The trench field duplicates returned results with a good correlation of determination (R^2) of 72% and a linear regression slope of 0.98. The trench field duplicates are on average 2.8% lower when compared to the original assays (Figure 11.16).

Figure 11.16: Trench Field Duplicates – 2020-2024 Drilling Programs

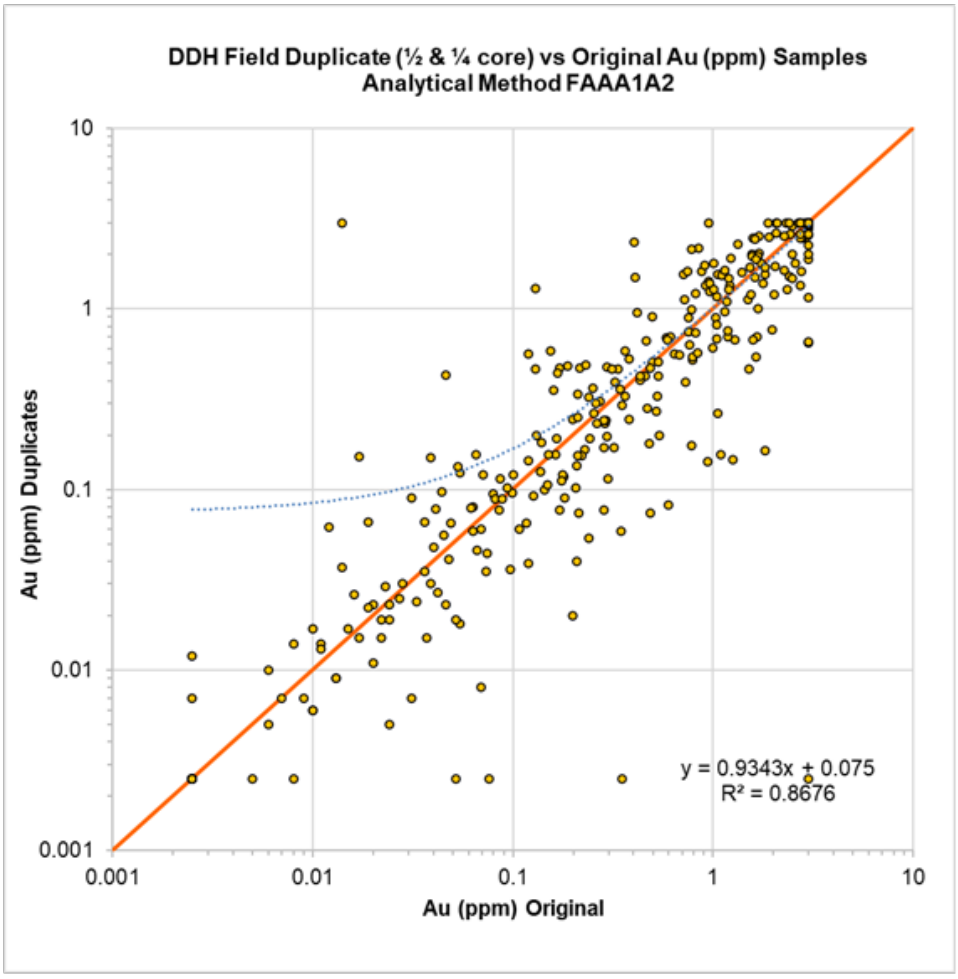


Global Statistics	Au_ppm_Original	Au_ppm_Duplicate
Mean	0.178	0.173
Median	0.034	0.037
Standard Deviation	0.47	0.41
Sample Variance	0.22	0.17
Pearson Correlation	0.85	
Minimum	0.0025	0.0025
Maximum	3	3
Count	269	269
Confidence Level(95.0%)	0.01	0.01

Source: GMS, 2025.

The results of half and quarter core drill hole field duplicates vs. original gold values assayed by FAAA-1A2 are presented in Figure 11.17. The DDH field duplicates returned results with a good correlation of determination (R^2) of 87% and a linear regression slope of 0.94. The core field duplicates are on average 1% lower when compared to the original assays (Figure 11.17).

Figure 11.17: Diamond Drill Hole (½ & ¼ Core) Field Duplicates – 2020-2024 Drilling Programs



Global Statistics	Au_ppm_Original	Au_ppm_Duplicate
Mean	1.352	1.338
Median	0.9105	0.7495
Standard Deviation	1.26	1.27
Sample Variance	1.59	1.60
Pearson Correlation	0.93	
Minimum	0.0025	0.0025
Maximum	3	3
Count	414	414
Confidence Level(95.0%)	0.01	0.01

Source: GMS, 2025.

Actlabs also conducts an internal quality control program, which includes a routine selection of coarse rejects and pulp duplicates. The preparation duplicates are taken from the coarse crushed material before the pulverizing process. The pulp duplicates are taken after the sample has been pulverized, and two (2) same-weighted pulps are analyzed by the same analytical method.

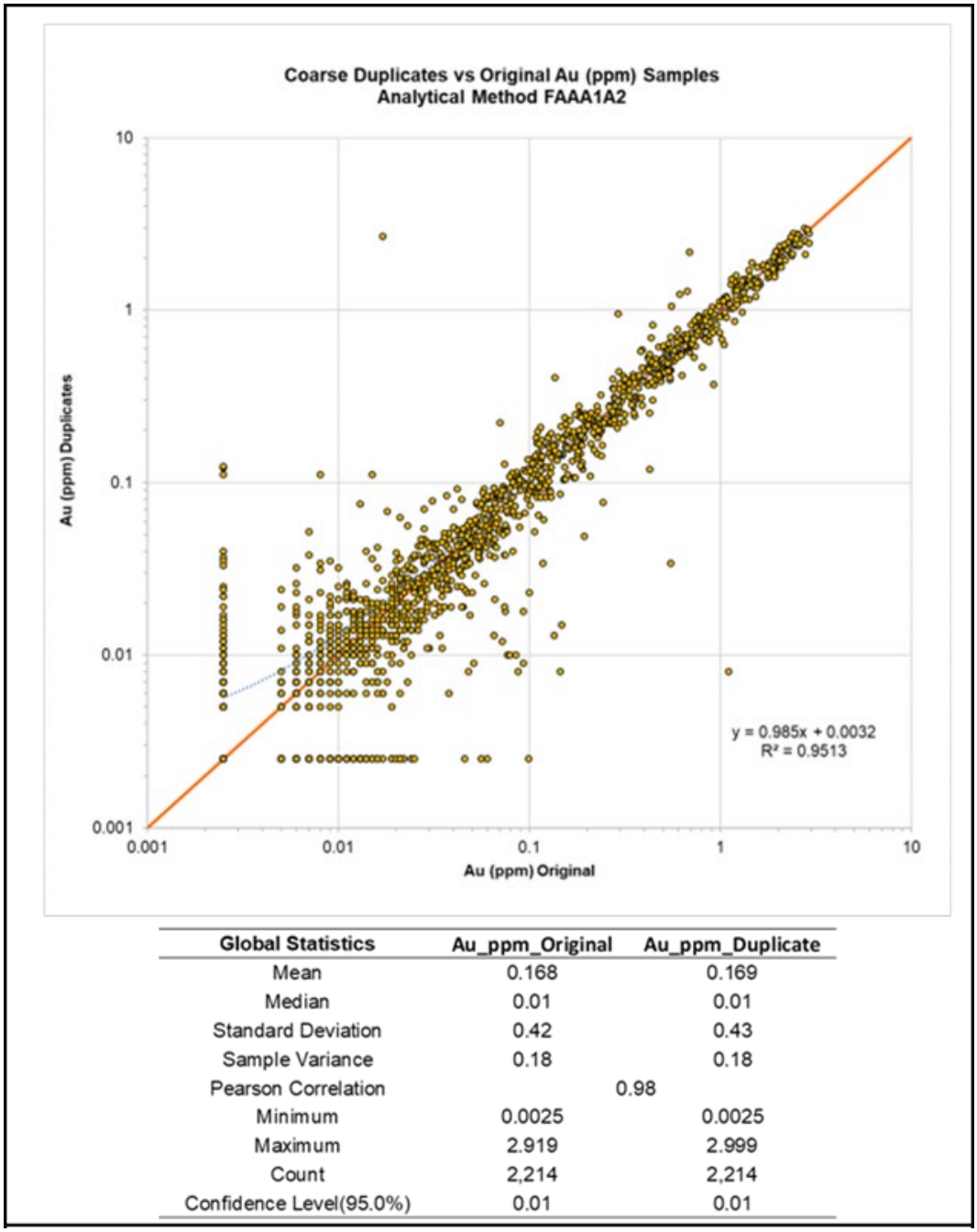
Coarse reject duplicates are created by splitting a second cut of the crushed sample following the same protocol and for the same weight as the original sample. The main objective is to determine if splitting procedures are applied consistently or if modifications to sample preparation procedures, such as the crush size of the samples, are required. Due to their particle sizes, preparation duplicates are expected to be less similar than the pulp duplicate samples.

A total of 2,214 coarse reject duplicates were analyzed for gold by fire assay with AA finish (FAAA-1A2) at Actlabs. A total of 1,398 duplicates reported gold results above 0.05 ppm and below 3.00 ppm gold for method FAAA-1A2 (Figure 11.18). The preparation duplicates for gold have 62% of the duplicate pairs reporting within ±20% tolerance limit. GMS judges this acceptable since most pairs ±20% difference is below 0.1 g/t Au. The coarse reject duplicates produced an excellent correlation of 95%. These internal QC results indicate that the duplicate grades are close to the original grade values, and good reproducibility is obtained at the primary laboratory regarding coarse reject material at the preparation stage of the analysis process.

Commercial laboratories routinely assay a second aliquot of the sample pulp, usually for one (1) in ten (10) samples. The data are used by the laboratory for their internal quality control monitoring. These data are provided to the clients at no additional cost.

A total of 2,543 pulp duplicates were analyzed for gold by fire assay with AA finish (FAAA-1A2). 1,700 duplicate pairs sent to Actlabs reported gold results above 0.005 ppm and below 3 ppm gold for method FAAA-1A2. Figure 11.19 shows a good correlation of 96% between the pulp duplicates and the original gold values.

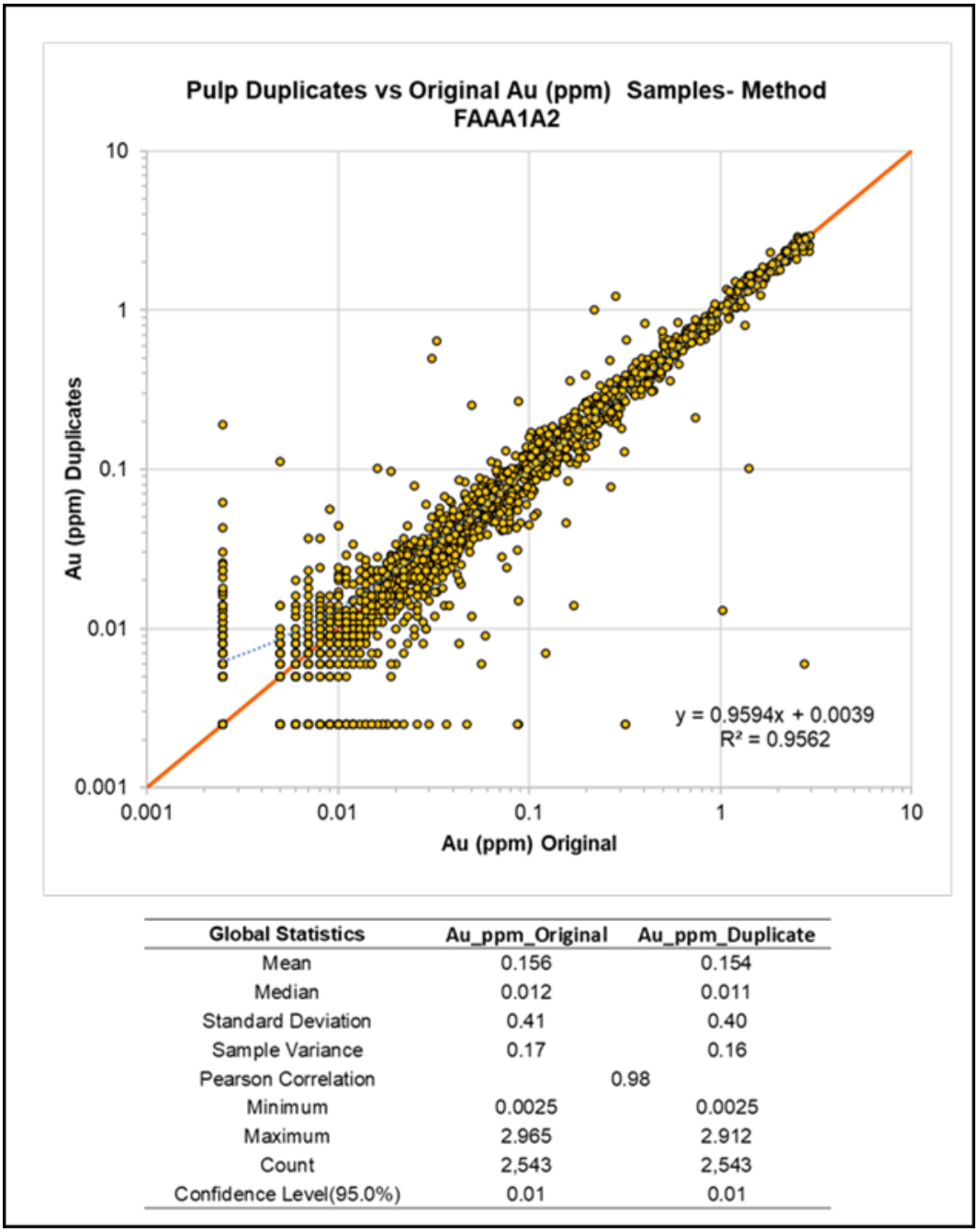
Figure 11.18: Coarse Reject Duplicates – 2020-2024 Drilling Program



Global Statistics	Au_ppm_Original	Au_ppm_Duplicate
Mean	0.168	0.169
Median	0.01	0.01
Standard Deviation	0.42	0.43
Sample Variance	0.18	0.18
Pearson Correlation	0.98	
Minimum	0.0025	0.0025
Maximum	2.919	2.999
Count	2,214	2,214
Confidence Level(95.0%)	0.01	0.01

Source: GMS, 2025.

Figure 11.19: Pulp Duplicates Check – 2020-2024 Drilling Programs



Global Statistics	Au_ppm_Original	Au_ppm_Duplicate
Mean	0.156	0.154
Median	0.012	0.011
Standard Deviation	0.41	0.40
Sample Variance	0.17	0.16
Pearson Correlation	0.98	
Minimum	0.0025	0.0025
Maximum	2.965	2.912
Count	2,543	2,543
Confidence Level(95.0%)	0.01	0.01

Source: GMS, 2025.

11.7.4 Umpire Check Assays

Umpire check assays are often used to assess the possibility of intra-laboratory bias.

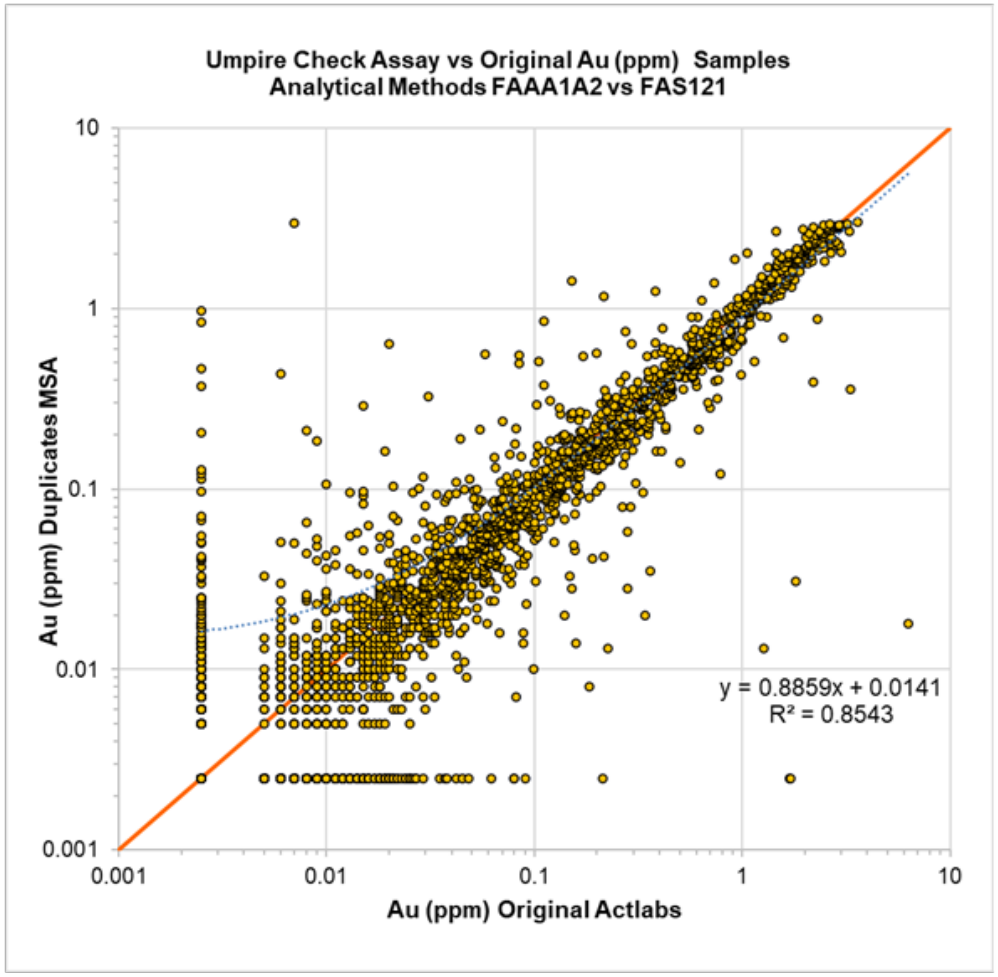
The geology team selects the samples to be submitted as check assays at a frequency of one (1) in 15 to one (1) in 20 samples (5%). The primary laboratory, Actlabs, prepares a second pulp and sends it to the

check laboratory, MSA Labs, in Georgetown, Guyana, with additional reference materials. The analytical methods at the check laboratory are comparable to the original laboratory. There is a preference for samples from unweathered rocks to avoid potential discrepancies between both laboratories. The secondary lab thoroughly homogenizes the samples and analyses them using a method as close to the primary lab as possible. The results between the primary and secondary labs are compared for discrepancies.

Reunion selected 3,008 samples for check assaying at the MSA Labs. The umpire check assays with AA finish (FAS-121) at MSA Labs have 54% of the duplicate samples reporting within ± 25%. As shown in the global statistical analysis (Figure 11.20), the mean gold grades obtained by MSA Labs pulp duplicates are slightly higher than the primary laboratory assay results. The average relative percent difference between both laboratories is 4.4%. GMS considers the check assay results as acceptable. The difference in the mean values of the umpire check assays may in part be due to the higher upper detection limit and differences in sample preparation. Most of the samples outside the 25% limit range are below 0.1 ppm.

Figure 11.20 presents the results of umpire pulp duplicates compared to the original Au values from Actlabs.

Figure 11.20: Actlabs vs MSA Labs Check Assays



Global Statistics	Au_ppm_Original	Au_ppm_Duplicate
Mean	0.188	0.196
Median	0.012	0.014
Standard Deviation	0.47	0.49
Sample Variance	0.22	0.24
Pearson Correlation	0.92	
Minimum	0.0025	0.0025
Maximum	2.999	6.283
Count	3,008	3,008
Confidence Level(95.0%)	0.01	0.01

Source: GMS, 2025.

11.8 External Audit – Qualitica Consulting Inc, September 2022

In September 2022, Qualitica Consulting Inc. was engaged to undertake an external audit of QA/QC results. The report contained a summary of QA/QC results up to September 2022, and made the following conclusions:

- There is no evidence of systematic contamination; however, it is likely that a portion of the blank failures is due to mislabelling or CRM / blank mix-ups.
- The CRM failure rate is less than 1%, and the results from CRMs are acceptable.
- A mix-up of OREAS250b and OREAS250 CRMs was investigated and resolved.
- The RC field duplicate results are acceptable.
- The coarse reject preparation duplicate results are acceptable.
- The pulp duplicate results are acceptable.
- The umpire check assay results are acceptable; however, it is recommended that MSA does not homogenize the pulps by mat rolling before running the analysis. As of 2024, mat rolling is still in practice.

11.9 QP Conclusions and Recommendations

The QP concludes that the sample preparation, analysis, and security procedures applied by the company are acceptable. Documentation of sampling procedures used to support the diamond and reverse circulation drilling programs is considered by GMS to be in line with industry best practices.

At this time, there is no re-assay procedure for duplicates that significantly exceeds the original laboratory value. GMS recommends implementing a procedure to the QA/QC protocols for outliers greatly exceeding the expected value.

In the opinion of the QP, sample preparation, analysis, and security procedures implemented by the company are comparable with the best industry practices, and robust controls are in place to ensure the integrity of the assay database. A statistical analysis of the quality control data from the 2020 to 2024 sampling programs did not expose any significant analytical issues. The QP believes that the drilling database is robust and reliable.

12. DATA VERIFICATION

12.1 Site Visits

Mr. Christian Beaulieu, P.Geo., consulting geologist for GMS and qualified person (QP) under NI 43-101 visited the project on two occasions: in April 18 to April 19, 2023, and in March 11 to March 12, 2025. During these site visits, the QP inspected mineralized intervals, alteration assemblages and QAQC protocols and conducted field checks of trenches and validated drill collars (Figure 12.1). Some trenches were cleaned, and sampling protocols were assessed directly in the field with the site geologists and technicians; drill core was reviewed with GMIN personnel. Drill core review permitted to observe clear relationships between gold grades (or presence of mineralization) and rock alteration / strain within mineralized domains LDZ, AU_2, AU_3, AU_3A, AU_3B, AU_4, AU_5 and LDZ_B6. It is the opinion of the QP that work practices at Oko West are in line with the CIM Best Practice Guidelines (2019).

Pascal Delisle, P.Geo., Director of the Geology and Resource department at G Mining Services (GMS visited the project between January 30, 2024, and February 2, 2024.

During all site visits, drilling activities were ongoing, core processing and storage facilities located on site were toured (Figure 12.2 and Figure 12.3), and the drill core from Oko West deposit was reviewed. Outcropping mineralization and trenches were visited and compared with the provided LiDAR topographic surface for survey accuracy (Figure 12.4). Sampling protocols were also reviewed with field geologists working on the Oko West Project. Independent quarter core samples were collected, and pulps sent to an independent laboratory (Section 12.2).

GMS also reviewed sampling and QAQC procedures on site and visited the preferred independent laboratory (Actlabs, Georgetown, Guyana) and the umpire laboratory (MSALABS, Georgetown, Guyana) to inspect the sample preparation facilities. During all site visits, chain-of-custody and sample security protocols were also reviewed and were found to be robust and transparent. The protocols in place are judged to be very robust: several instances of redundancy are integrated, and the staff is well trained to perform several tasks, which both contribute to reducing the risks of errors and lost samples.

Prior to the above site visits, one (1) other former QP visit was performed by James Purchase, P.Geo., geologist at GMS, between May 9, 2022, and May 13, 2022. During this visit, Mr. Purchase was able to visit drill rigs, inspect drill core, collect QP samples, validate sampling protocols, review drilling procedures with drilling contractors and field geologists working for the Project and visited the preferred independent laboratory (Actlabs, Georgetown, Guyana) and the umpire laboratory (MSALABS, Georgetown, Guyana).

Mr. Purchase also believed that work practices at Oko West were in line with the CIM Best Practice Guidelines (2019) (Oko West Technical Report; 2022).

A selection of drill collars was visited by the former and current QPs and independently verified using a handheld GPS. The comparison between the results and the database is shown in Table 12.1 for all site visits (including those from the QP in 2023 and 2025). Some examples of drill collars are shown in Figure 12.1.

Table 12.1: Validation of Drill Collar Coordinates (PSAD 1956 UTM Zone 21N)

Site Visit	Hole ID	Database		G Mining		Difference	
		X	Y	X	Y	X	Y
2022	OKWD21-41	272777.8	701902.2	272777.0	701904.0	-0.8	1.8
	OKWD21-42	272786.5	701994.8	272784.0	701997.0	-2.5	2.3
	OKWD22-73	272833.3	701651.8	272835.0	701652.0	1.7	0.2
	OKWR22-123	272705.2	702248.2	272704.0	702254.0	-1.2	5.8
	OKWR22-132	272821.3	702648.3	272819.0	702654.0	-2.3	5.7
	OKWR22-77	272900.3	701735.8	272901.0	701739.0	0.7	3.2
2023	OKWD21-015	272669.5	701604.0	272667.0	701603.0	2.1	1.2
	OKWT21-044A	272676.5	701688.0	272677.0	701687.0	-1.0	0.7
	OKWD21-057	272691.5	701656.0	272687.0	701655.0	4.2	0.9
	OKWD22-153	272703.5	701692.0	272698.0	701694.0	5.1	-1.8
	OKWD22-087	272779.5	701860.0	272777.0	701857.0	2.6	2.6
	OKWD21-051	272792.5	701795.0	272788.0	701793.0	5.0	1.8
	OKWD22-064	272792.5	701656.0	272787.0	701657.0	5.1	-1.3
	OKWD23-221	273003.5	702151.0	273002.0	702153.0	1.5	-2.0
	OKWD22-175	273045.5	701876.0	273045.0	701875.0	0.5	0.6
	OKWD22-182	273092.5	701721.0	273089.0	701722.0	3.4	-0.9
	OKWD22-178	273094.5	701719.0	273091.0	701720.0	3.9	-1.2

Site Visit	Hole ID	Database		G Mining		Difference	
		X	Y	X	Y	X	Y
2024	OKWD22-173	272813.0	701982.0	272812.5	701985.0	0.5	-3.3
	OKWD22-088	272886.9	702135.5	272886.5	702143.0	0.4	-7.4
	OKWD22-163	272758.4	701622.6	272760.5	701622.0	-2.1	0.6
	OKWD22-166B	272795.9	702072.5	272795.5	702077.0	0.4	-4.5
	OKWD23-220	272815.3	702020.6	272815.5	702025.0	-0.2	-4.4
	OKWD23-348	273089.7	701716.8	273085.5	701719.0	4.2	-2.2
	OKWD23-343	273108.1	701792.3	273105.5	701794.0	2.6	-1.7
2025	OKWD24-320-W1	273,394.1	701,734.4	273,394	701,742	0.1	-7.6
	OKWD23-321	273,316.3	701,652.7	273,316	701,646	0.3	6.7
	OKWD23-331	273,318.5	701,655.7	273,317	701,653	1.5	2.7
	OKWD23-337A	273,321.1	701,569.9	273,317	701,571	4.1	-1.1
	OKWD24-346A-W2	273,445.4	701,883.7	273,447	701,887	-1.6	-3.3
	OKWD24-364-W1	273,564.6	701,877.4	273,559	701,885	5.6	-7.6
	OKWD24-390	273,392.9	701,733.0	273,390	701,736	2.9	-3.0
	OKWD24-401A	273,318.0	701,648.7	273,315	701,654	3.0	-5.3
	OKWD24-410A	273,394.4	701,740.2	273,393	701,739	1.4	1.2
	OKWD24-421	273,318.4	701,650.2	273,315	701,650	3.4	0.2

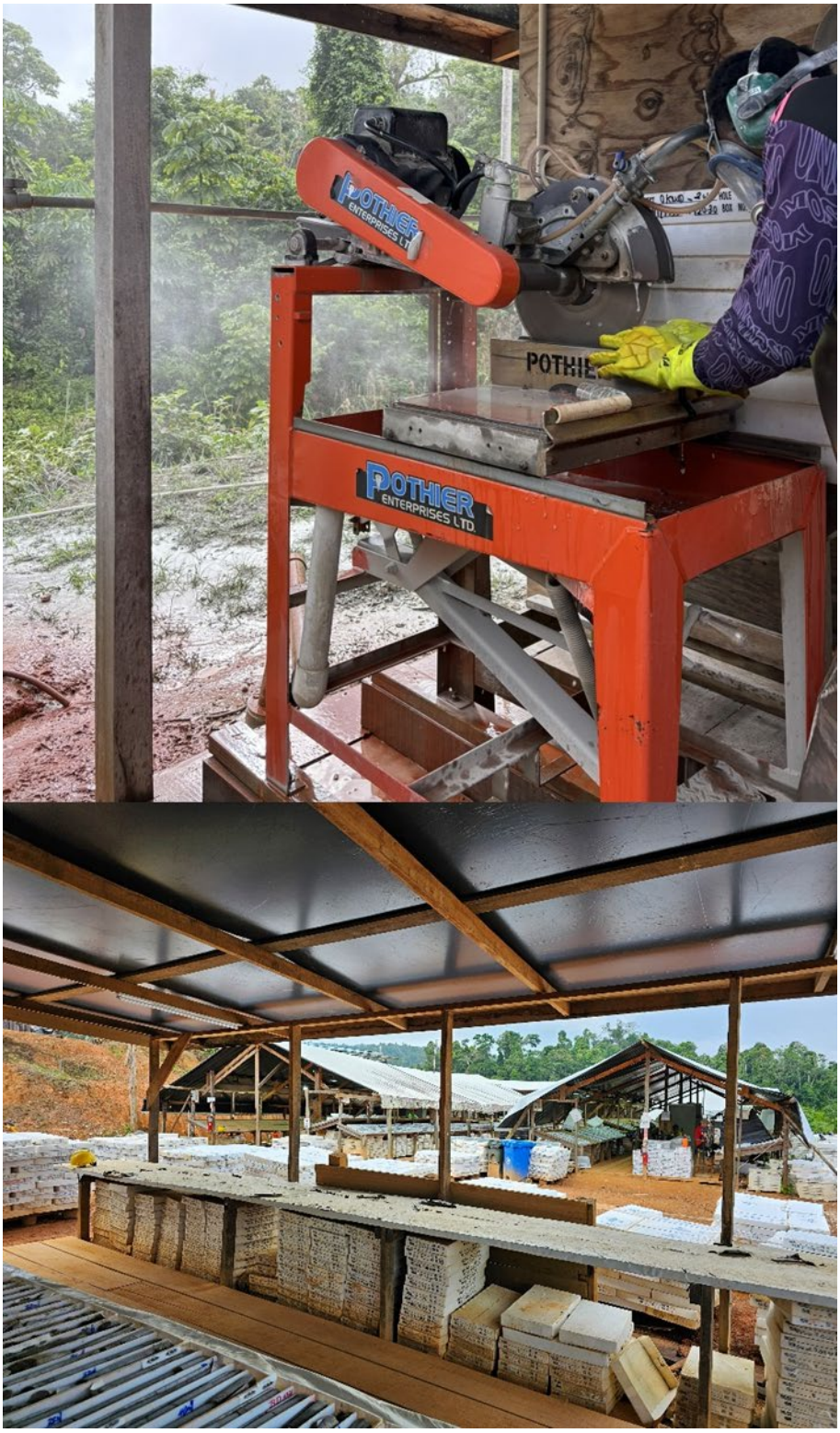
Figure 12.1: Verification of Drill Collar – OKWD23-220

Source: GMS, 2024

Figure 12.2: Sample Reject Storage in Georgetown (top) & Core Storage Facilities Onsite (bottom)

Source: GMS, 2024

Figure 12.3: Core Cutting Facility (top) and Core Sampling Facility (bottom)



Source: GMS, 2024

Figure 12.4: Outcrop Inspection (top) and Closeup of Outcrop OKWT20-001

Source: GMS, 2024
**Note: Primary textures are still visible from surface altered material (saprolite).*

12.2 QP Duplicate Samples

During Mr. Purchase's site visit, 20 quarter-core duplicates were taken from mineralized intervals derived from four (4) drillholes distributed through the strike length of gold mineralization. Both oxidized and unoxidized mineralization was sampled. The 20 duplicates were hand-delivered by the QP to Actlabs in Georgetown and were subsequently analyzed using the standard analysis protocol described in Section 11.

QAQC samples were also inserted into the sample stream, and all standards and blanks returned gold values within the expected ranges of error. The entire sampling process was supervised.

Another nine (9) quarter-core duplicates were collected by Mr. Delisle during the 2024 site visit. Samples were collected from mostly fresh material ranging from 0.01 g/t Au to 15.56 g/t Au, across the deposit strike and depth. Selected core samples were sampled manually and cut under the supervision of GMS' team in quarter core. All samples were bagged and sealed with a security tag by GMS' personnel (Figure 12.5). The samples were sent to MSALABS in Georgetown. Laboratory technicians made sure nobody tempered with the security tags when samples were received. Nine (9) pulp samples from the reject storage facility previously analysed at Actlabs in Georgetown were hand-delivered to MSALABS in Georgetown by Mr. Delisle for external umpire check assays.

The results and a comparison with the original assays are presented in Table 12.2 and Figure 12.6 for 2022 and 2024 core duplicates and in Table 12.3 for 2024 pulp re-assays.

Table 12.2: Independent Core Duplicate Results

Site Visit	Hole ID	From	To	Original Sample ID	QP	Au ppm	Au ppm QP
					Sample ID	Original	
2022	OKWD21-053	193.0	194.5	577154	791858	6.52	5.89
	OKWD21-053	194.5	196.0	577156	791860	0.79	0.52
	OKWD21-053	196.0	197.5	577157	791861	0.75	0.75
	OKWD21-053	197.5	199.0	577158	791862	4.55	3.11
	OKWD21-053	199.0	200.5	577159	791863	2.66	4.53
	OKWD21-031	66.0	67.5	1074952	791864	1.13	1.58
	OKWD21-031	67.5	69.0	1074953	791865	2.05	2.61
	OKWD21-031	69.0	70.5	1074954	791866	1.70	2.04
	OKWD21-031	70.5	72.0	1074955	791868	0.85	2.17
	OKWD21-031	72.0	73.5	1074956	791869	2.05	4.66
	OKWD22-062	280.0	281.0	576235	791870	0.71	1.54
	OKWD22-062	281.0	282.0	576236	791871	3.05	2.56
	OKWD22-062	282.0	283.0	576237	791873	1.89	2.50
	OKWD22-062	283.0	284.0	576238	791874	3.30	2.76
	OKWD22-062	284.0	285.0	576239	791875	15.62	6.65
	OKWD22-066	180.8	182.0	1013343	791877	1.14	0.97

Site Visit	Hole ID	From	To	Original Sample ID	QP Sample ID	Au ppm Original	Au ppm QP
	OKWD22-066	182.0	183.0	1013344	791878	6.16	6.65
	OKWD22-066	183.0	183.8	1013345	791879	13.83	10.17
	OKWD22-066	183.8	185.0	1013346	791881	1.28	0.68
	OKWD22-066	185.0	186.0	1013347	791882	0.41	1.49
2024	OKWD23-237A	317.0	318.0	549446	B00333475	0.01	0.01
	OKWD23-237A	305.0	306.0	549433	B00333476	15.56	8.65
	OKWD23-237A	306.0	307.4	549434	B00333477	0.13	0.39
	OKWD23-237A	293.0	294.0	549419	B00333478	0.85	0.95
	OKWD21-046	112.5	114.0	569800	B00333479	0.57	0.51
	OKWD23-243	443.3	444.5	553142	B00333480	1.54	2.17
	OKWD23-243	524.0	524.5	553240	B00333481	8.80	25.90
	OKWD21-046	175.5	177.0	569847	B00333483	4.98	8.62
	OKWD21-046	63.0	64.5	569764	B00333484	2.75	0.63

Table 12.3: Pulp Re-assay Results

Site Visit	Hole ID	From	To	Original Sample ID	Au ppm Original	Au ppm Umpire
2024	OKWD22-129	223.0	224.0	A02208	0.61	0.69
	OKWD22-129	256.8	258.6	A02242	1.06	0.98
	OKWD22-187	471.1	472.4	A08865	1.62	1.39
	OKWD22-187	479.3	480.7	A08872	6.11	5.74
	OKWD23-223	236.0	237.0	545705	0.85	0.81
	OKWD23-223	266.7	267.6	545737	4.19	3.12
	OKWD23-223	250.0	251.8	545719	1.44	1.27
	OKWD23-243	534.0	535.0	553259	1.48	1.35
	OKWD23-243	525.5	526.0	553244	40.84	44.60
	OKWD23-243	516.5	517.0	553225	20.30	19.90

The assay results from the QP duplicates sent to Actlabs (2022 site visit) show a good correlation with the original assay values found in the database. For duplicates sent to MSALABS (2024 site visit), a good correlation is also observed between assays from the database and re-assays. Some variability is expected

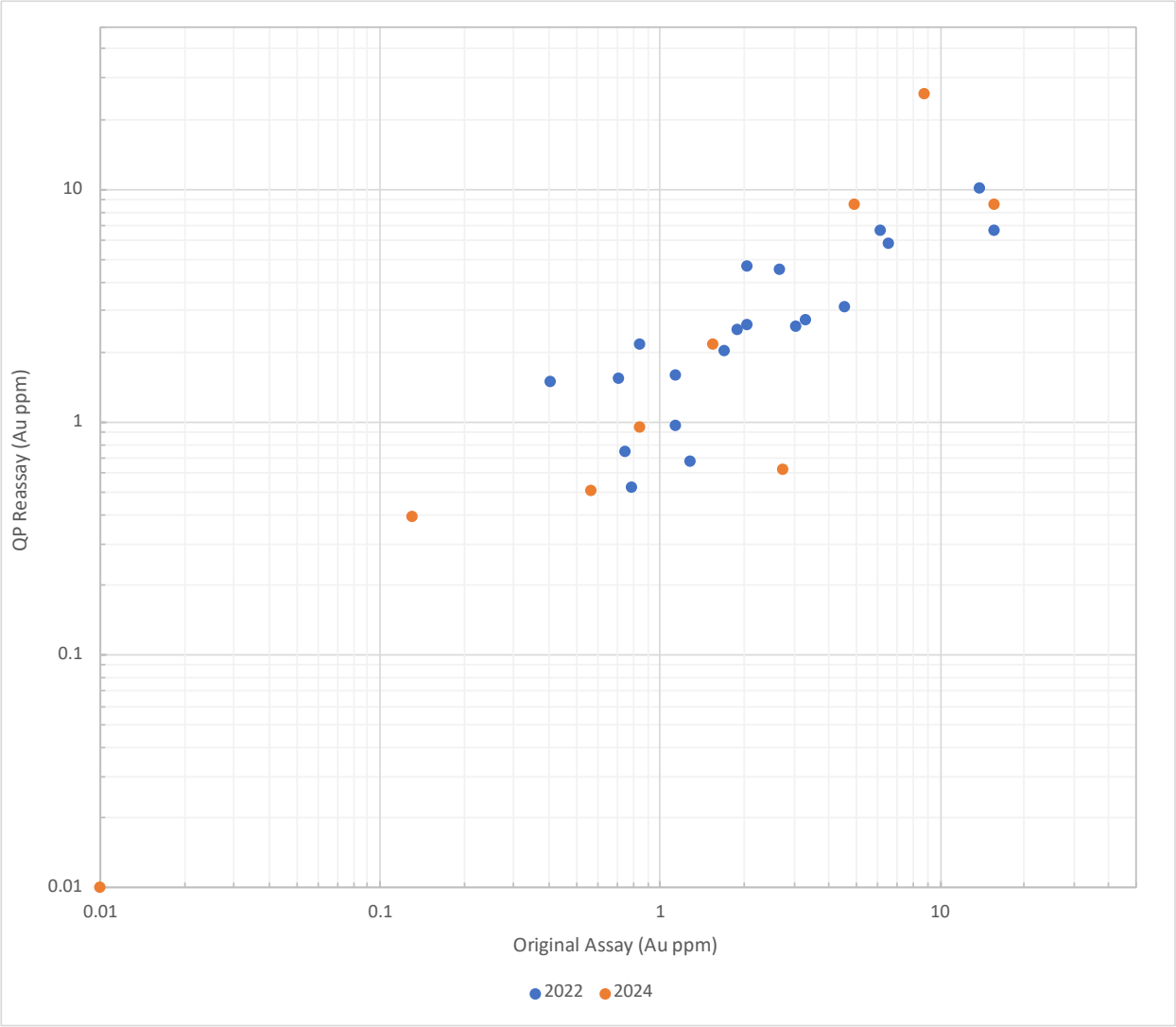
due to the nugget effect of the gold mineralization, and the difference in sample size (½ core for the original assays, ¼ core for the QP duplicate assays). No bias was identified.

For the pulp re-assay, a 2 - 26% difference is observed between original assays completed at Actlabs and pulp re-assays submitted to MSALABS. MSALABS pulp duplicate tend to be slightly lower than the primary laboratory (i.e. Actlabs). As explained in Section 11, this could be due to the homogenization done by MSALABS on pulps.

Figure 12.5: QP Sampling (Top) and QP Sample with Security Tag (Bottom)

Source: GMS, 2024

Figure 12.6: Scatter Plot Showing Original Assays (X-axis) vs. QP Duplicate Assays (Y-axis)



Source: GMS, 2024

12.3 Drill Core Inspection

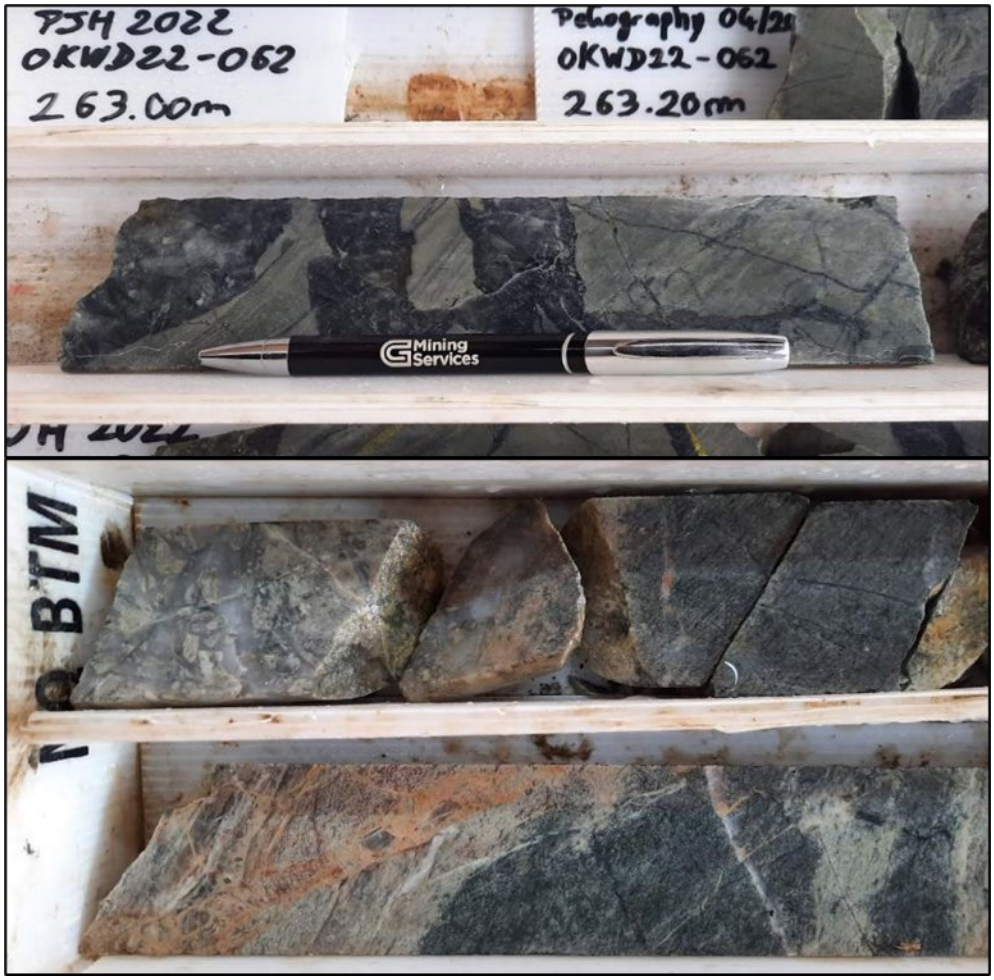
The QP personally inspected a total of 19 holes distributed evenly throughout the strike length of the deposit during his two (2) site visits. Holes inspected covered a range of dates from 2020 to 2024 and included wedged holes. GMS also reviewed numerous mineralized intersections from 16 holes distributed evenly throughout the strike length of gold mineralization during the 2024 site visit. A clear relationship was visible between gold grades and alteration / veining intensity. The frequency of smoky quartz brecciation and silicification, and sericitic “ghosting” of host rock appear to be directly relatable to the gold grades observed in assays. Visible gold was observed on numerous occasions. The QP and GMS notes that sulphide content is generally low compared to other orogenic deposits; however, pyrite accumulations were observed in mineralized intervals hosted within mafic rocks, although these occurrences are uncommon. Figure 12.7

shows some typical examples of alteration assemblages associated with mineralization. In trenches, a clear distinction between alluvium or colluvium sediments with saprolite material could be observed (Figure 12.8). Primary textures in saprolite are also very well preserved.

In oxidized intervals, gold grades were found to be directly related to the abundance of quartz fragments hosted with clays.

Unmineralized rocks in the hanging wall were found to be relatively unaltered sediments or volcaniclastics with intact primary textures exhibiting weaker foliation / shearing. The footwall rocks (also unmineralized) were also found to be weakly altered by metasomatic assemblages unrelated to mineralization, with primary igneous textures evident. These observations correlate with low-gold grades observed in the assays.

Figure 12.7: Typical Examples of Gold Mineralized Intervals Displaying Various Alteration Assemblages and Veining



Source: GMS, 2022.
**Note: (Top) Typical smoky-quartz vein with fragments of the sericite-altered host rock sediments. (Bottom) Brecciation and silicification within volcaniclastic host rock with sericitic alteration emanating from fracture planes. Fe-carbonate veining (orange) is mostly barren.*

Figure 12.8: Contact Between Overburden Material and Saprolite (top, water bottle as scale), and Primary Textures in Saprolite (bottom)

Source: GMS, 2023

12.4 Drillhole Database Verification

GMS reviewed the original assay certificates for Oko West drilling programs performed between 2020 and the closure of the database (September 15, 2024). A script was used to validate all assays recorded in the database with the gold assays reported by the laboratory in the original certificates. Four (4) discrepancies were noted and addressed to the Company database management team. Only one (1) sample

(OKWD23-264: 558705) was ignored from the database due to an upper limit detection value (≥ 3g/t Au) recorded in the database. This sample should have been re-assayed using gravimetric analysis method.

GMS also reviewed collar location, downhole surveys and diamond drill holes (DDH) twin holes. The drill holes collar locations are recorded in UTM PSAD56 Zone 21 North coordinate system. The collar elevations were validated using a 50-cm digital elevation model (DEM) provided to GMS by Reunion Gold. One (1) collar elevation was corrected (OKWD21-005) after differences of more than seven (7) metres were observed between the database and the DEM. Other collar elevations in the database were consistent with the DEM.

The consistency of drillhole surveys was assessed using LeapFrog Geo™ 3D viewer and GMS' validation script. Any discrepancies with variations exceeding 5 degrees per 100 min dip and direction were identified and subsequently examined for incoherence. Following the investigation, only two (2) surveys (DDH OKWD22-134 at 42.63 m downhole and DDH OKWD22-174 at 119.00 m) were discarded from the resource database.

Finally, DDH twin holes were conducted to examine the possibility of gold smearing in Reverse Circulation (RC) drillholes, as recommended by GMS. Upon careful evaluation, the Qualified Person (QP) expresses confidence in the accuracy of gold grades obtained through RC drilling and approves the use of RC drill holes for the resource estimation of the Oko West deposit. One (1) RC drillhole (OKWR22-128) was however excluded from the resource database due to potential gold grade smearing identified from the neighbouring DDH twin hole (OKWD22-127) assay result. Furthermore, a single trench (OKWT20-006) was removed from the database as it was entirely sampled in unconsolidated alluvial material.

12.5 Database Validation and Verification by the Qualified Person

An independent validation and verification of the database was undertaken by the Qualified Person for the Mineral Resource Estimation (Mr. Christian Beaulieu). The inspection of the database used in the estimation consisted of inspecting all relevant drilling information in Leapfrog Geo® 2024.1, such as collars, deviation, geology, weathering, assays, composite tables, interpolation profiles and integration into the block model, and checked for inconsistencies. A special attention was paid to more recent data, given that earlier data had already been reviewed by the current Qualified Person (QP), who also served as the QP during the initial Mineral Resource in 2023 (Technical Report, 2023) and the more recent PEA (PEA, 2024).

A validation of the assay database was conducted by comparing original assay certificates (PDF) provided by Actlabs with the database used for the estimation. No errors were found.

12.6 QP Commentary and Conclusions

In the opinion of the QP, the drilling, sampling and QAQC procedures meet best industry practices. Sampling equipment and logging facilities were found to be of sufficient quality, and the sample storage facilities were adequate. Although the site is remote, the company has established a robust chain of custody in relation to the storage and transportation of samples. Inspection of drill core demonstrated a good correlation between alteration and gold grades, and the independent duplicate assays correlated with the original assay database with acceptable levels of error. The QP has no concerns relating to the validity of the drilling database.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

A metallurgical test work program was completed August 2024, with additional test work conducted in January 2025 at BaseMet Laboratories (BML) in Kamloops, British Columbia. The objectives of the test work program were to further define the metallurgical response of the main ore domains within each weathering profile, generate sufficient metallurgical data to support a flowsheet and develop gold recoveries for the Feasibility Study (FS). The scope of the test work included chemical and mineralogical analyses, comminution test, gravity recovery, cyanide leaching and detoxification, blended composition assessments, carbon loading and oxygen uptake evaluations, and dewatering tests (including flocculant selection, static settling, and dynamic thickening tests). The relevant metallurgical data and findings from the 2024-2025 programs are summarized in this section.

13.2 Previous Metallurgical Test Work

Intensive cyanidation test work was completed in 2024 on select samples to determine preliminary gold recoveries to support an updated open-pit and initial underground Mineral Resource Estimate.

Subsequent comminution tests, preg-robbing tests, mineralogy analysis, gravity tests, leach and cyanide destruction tests were conducted on 18 master composites from core samples from three (3) weathering zones, i.e., saprolite, transition and fresh rock (open pit) and main geological units, i.e. volcanics, metasediments and carbonaceous sediments. Samples were also selected for two (2) gold grades, at approximately 1 g/t Au and 2 g/t Au.

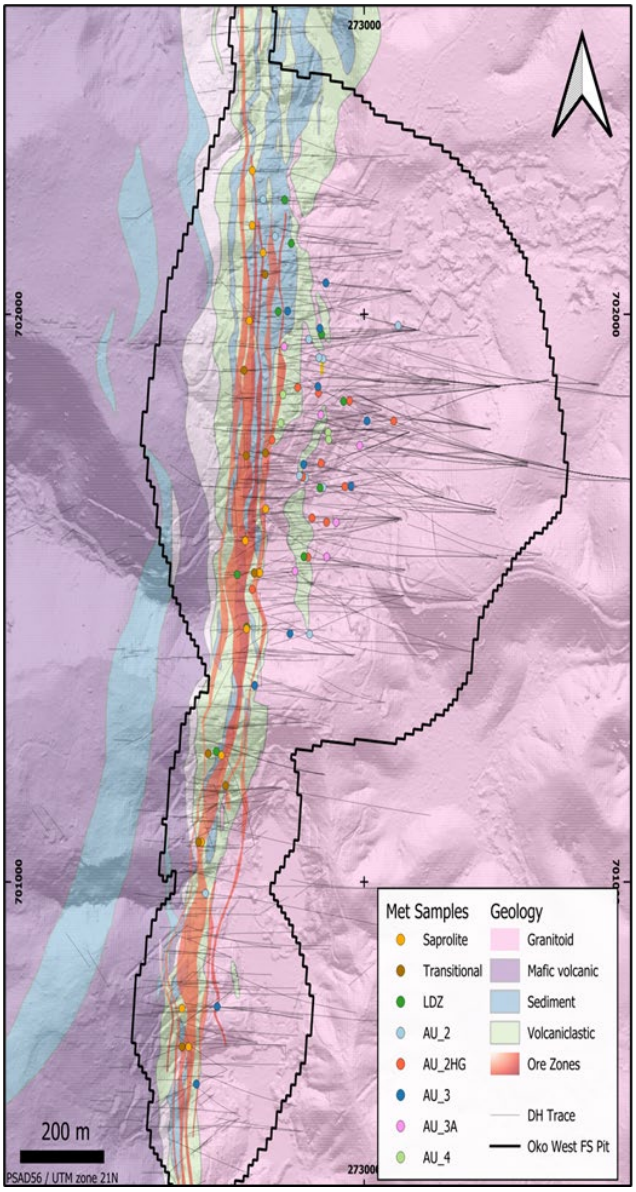
The results of the two previous metallurgical test work programs are documented in the previous NI 43-101 Technical Report, Preliminary Economic Assessment, Oko West Project, October 2024.

13.3 Sample Selection

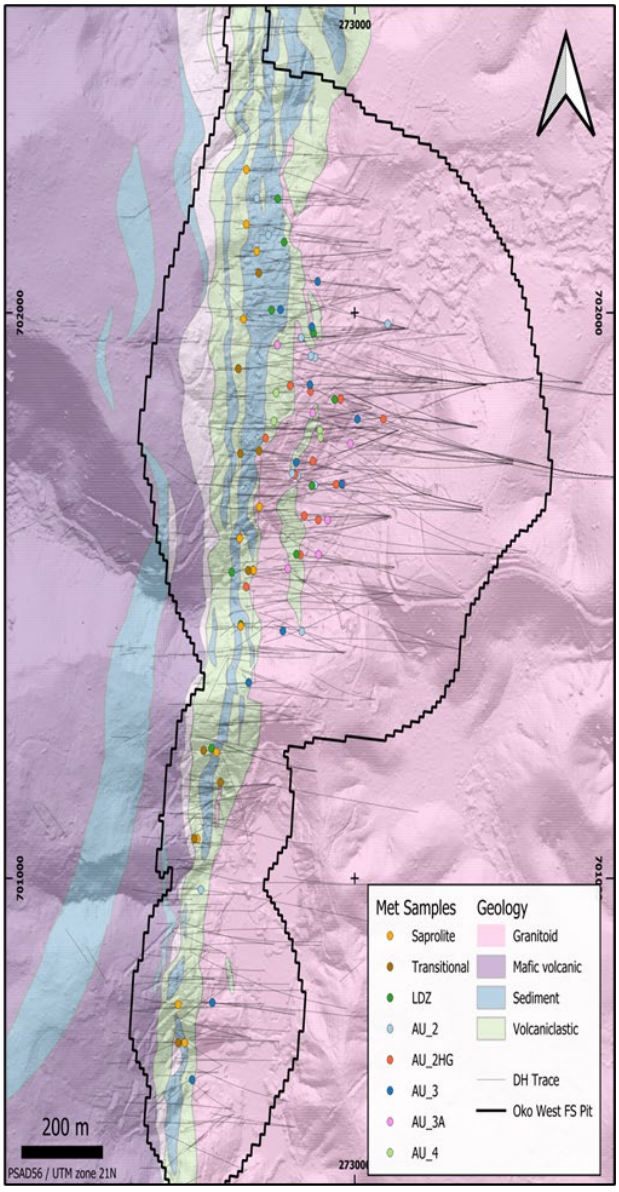
As part of the FS metallurgical test work program, 16 domain composites and three (3) variability composites per domain were initially planned, for a total of 48 variability composites. However, due to insufficient sample mass from certain domains, only 39 variability composites were ultimately constructed. The variability composites represented low, medium and high-grade ore from each domain, capturing a range of metallurgical responses across different weathering profiles and lithological units. The weathering types included Saprolite, Transitional, and Fresh Rock from both open pit and underground geological units. Sample locations are shown in Figure 13.1.

Figure 13.1: Metallurgical Sample Location with Ore Zone (a); Metallurgical Sample Location (b)

(a) With ore zone



(b) Without ore zone



13.4 Comminution Test Work

13.4.1 Test Results

The domain composites were subject to a series of comminution testing, including SAG Mill Comminution (SMC), SAG Power Index (SPI), Bond abrasion, Bond rod and Bond ball mill work index determinations. The Saprolite samples were not physically competent for select comminution test work. SPI testing was performed on the Transitional samples in lieu of SAG testing. One (1) Fresh sample was

not subject to comminution testing due to low sample mass. The Bond ball mill work index was performed at a closing screen size of 106 microns. The comminution test results are summarized in Table 13.1.

Table 13.1: Summary of Bench-Scale Comminution Testwork

Study	Sample ID	Oxidation	SMC Parameters		Bond Parameters			SPI
			Relative Density	Axb	RWi kWh/t	BWi kWh/t	Ai g	min
PEA	MET23-HSS-05	Saprolite	2.31	390	5.5	10.5*	0.054	
PEA	MET23-LSS-14	Saprolite						
PEA	MET23-HSV-06	Saprolite					0.049	
PEA	MET23-LSV-15	Saprolite						
PEA	MET23-HST-08	Transition	2.47	116	7.0	7.0	0.055	
PEA	MET23-LTS-17	Transition	2.29	228	6.4	6.8	0.028	
PEA	MET23-HTV-09	Transition	2.53	198	6.9	8.1	0.016	
PEA	MET23-LTV-18	Transition	2.41	193	5.6	5.6	0.002	
PEA	MET23-HFS-02	Fresh Rock	2.72	31.6	16.4	13.3	0.164	
PEA	MET23-LFS-11	Fresh Rock	2.94	33.3	16.8	13.5	0.114	
PEA	MET23-HFV-03	Fresh Rock	2.73	54.4	17.9	15.5	0.123	
PEA	MET23-LSV-12	Fresh Rock	2.93	35.0	15.7	14.0	0.129	
PEA	FWGR	Wast Rock	2.74	35.6				
PEA	HWGR	Wast Rock	2.72	33.3				
FS	SAP_AU2	Saprolite				7.3		
FS	SAP_AU3	Saprolite				6.3		
FS	SAP_AU3A	Saprolite				12.5*		
FS	TRANS_AU2	Transition				8.8	0.014	5.9
FS	TRANS_AU3	Transition				6.2	0.024	6.9
FS	FRESH_PIT_AU2	Fresh Rock	2.38	36.6	15.5	16.1	0.135	
FS	FRESH_PIT_AU2HG	Fresh Rock	2.89	31.8	18.2	16.6	0.210	
FS	FRESH_PIT_AU3	Fresh Rock	2.75	28.3	17.0	14.4	0.111	
FS	FRESH_PIT_AU3A	Fresh Rock	2.71	24.0	17.8	16.0	0.116	
FS	FRESH_PIT_AU4	Fresh Rock	2.70	34.7		13.6	0.145	
FS	FRESH_PIT_LDZ	Fresh Rock	2.76	35.3	15.1	19.8	0.139	
FS	FRESH_UG_AU2	Fresh Rock - UG	3.59	32.0	17.9	17.1	0.149	
FS	FRESH_UG_AU2HG	Fresh Rock - UG	2.81	30.0	18.8	17.6	0.299	
FS	FRESH_UG_AU3	Fresh Rock - UG	2.86	27.4	18.5	16.9	0.166	
FS	FRESH_UG_LDZ	Fresh Rock - UG	2.83	27.4	18.2	19.1	0.171	

**Hard BWi based on OMC Database of saprolite ores in West Africa.*

Table 13.1 presents a summary of comminution testwork results from both the PEA and the FS programs. The upper portion of the table corresponds to the PEA results, previously disclosed in the NI 43-101 technical report issued in October 2024. The discussion below focuses on the FS comminution test results presented in both sections of the table. The FS testwork was conducted on representative samples of saprolite, transition, open pit fresh rock and underground material.

The Axb parameter provides a measure of resistance to impact breakage in a SAG mill. The Axb values, as derived from the SMC tests for these samples ranged between 24.0 and 36.6.

The Bond abrasion index of these samples was variable and measured between 0.014 and 0.299, indicating the samples range between being not very abrasive to mildly abrasive.

The Bond rod mill work index of the samples ranged between 15.1 and 18.8 kWhr/tonne classifying the samples as hard with respect to rod milling.

The Bond ball mill work index of the samples ranged between 6.2 and 19.8 kWhr/tonne. There is a wide range in these results given the varying competency of the ore types. Generally, the Saprolite samples would be considered soft, and the Transitional samples would be considered medium soft, while the Fresh Rock samples had a much higher Bond ball work index and would be classified as medium hard to hard with respect to ball milling.

SPI results of 5.9 and 6.9 mins on the two (2) Transitional samples indicate the samples to be very soft.

13.4.2 Comminution Analysis

The test results were further analyzed by Orway Mineral Consultants (OMC) as follows. Table 13.2 presents the ore parameter statistics for the bench-scale comminution test work.

Statistical analysis and calculation of the coefficient of variation (Standard Deviation / Average, %) as well as precision (90% confidence level) are indications of test work ore hardness variability. For a FS, a typical target accuracy of 15% or less is recommended. The RWi, Bwi, Axb and ore relative density meet FS level accuracy, as does the Ai for fresh rock. Heterogeneity exists for the BWi for saprolite and Ai for transition and fresh rock UG.

In general, the fresh rock is more competent, hard and abrasive in comparison to the saprolite and transition material which is expected. The fresh rock exhibits competent ore (Axb 15th percentile of 34.5), hard grindability (85th percentile BWi of 15.3 kWh/t) and is mildly abrasive (Ai of 0.139). The saprolite and transition materials are very soft (Axb 15th percentile of 194-300), with soft grindability (85th percentile BWi of 7.2-8.4 kWh/t) and is mildly abrasive (Ai of 0.030-0.052).

Due to the high proportion of fresh rock in the LOM schedule, OMC carried out the major equipment sizing with fresh rock design ore parameters, as summarized in Table 13.2. The FS testwork for fresh rock open pit yielded higher competency and the resulting total grinding mill specific energy is +10% compared to the PEA data. The fresh rock underground material is marginally more competent based on Axb and harder based on BWi, with total grinding mill specific energy +8% compared to open pit material.

Two (2) saprolite samples MET23-HSS-05 and SAP_AU3A exhibited high BWi results (>10.5 kWh/t) relative to the remainder the samples tested; however, these samples fall within the OMC database of saprolite oxide ores. It is unclear if these samples were properly categorized as saprolite or if perhaps should be regrouped as transition material. Ore interpretation statistics were calculated with and without these samples to assess impact on mill modelling.

To date there has been no Bond low-energy impact (CWi) testing or unconfined compressive strength (UCS) testing. Calculation to infer a CWi value based on Axb and SG data can be done for a conservative design.

Table 13.2: Summary of Comminution Test Results Statistics by Ore Type

Statistic	BWi, kWh/t						
	Sap+ Trans*	Sap*	Trans	Fresh	Fresh UG	Waste	Overall
Count	10	2	5	10	4	-	24
Min	5.6	6.3	5.6	13.3	16.9	-	5.6
Max	12.5	7.3	8.8	19.8	19.1	-	19.8
85th / 15th Percentile	**9.9**	**7.2**	**8.4**	**16.4**	**18.4**	**-**	**17.0**
Average	**7.6**	**6.8**	**7.1**	**15.3**	**17.7**	**-**	**12.6**
Std Dev	2.1	0.7	1.3	2.0	1.0	-	4.5
Coefficient of Var	27%	10%	19%	13%	6%	-	36%
Accuracy (90% CL)	14%	12%	14%	7%	5%	-	12%

Statistic	Axb						
	Sap+ Trans*	Sap	Trans	Fresh	Fresh UG	Waste	Overall
Count	5	1	3	10	4	2	21
Min	390.3	-	228.0	54.4	34.9	35.6	24.0
Max	116.0	-	193.0	24.0	27.4	33.3	390.0
85th / 15th Percentile	**162.2**	**390.0**	**194.5**	**29.4**	**28.6**	**33.7**	**30.0**
Average	**225.0**	**390.0**	**206.3**	**34.5**	**31.1**	**34.5**	**79.2**
Std Dev	101.1	-	18.9	7.9	3.2	1.6	95.1
Coefficient of Var	45%	-	9%	23%	10%	5%	120%
Accuracy (90% CL)	33%	-	9%	12%	8%	5%	43%

Statistic	Ai, g						
	Sap+ Trans*	Sap	Trans	Fresh	Fresh UG	Other	Overall
Count	8	2	5	10	4	-	22
Min	0.002	0.049	0.002	0.111	0.149	-	0.002
Max	0.055	0.054	0.028	0.210	0.299	-	0.299
85th Percentile	0.054	0.053	0.025	0.157	0.242	-	0.166
Average	**0.030**	**0.052**	**0.017**	**0.139**	**0.196**	**-**	**0.110**
Std Dev	0.020	0.004	0.010	0.030	0.069	-	0.074
Coefficient of Var	66%	7%	59%	21%	35%	-	67%
Accuracy (90% CL)	39%	8%	44%	11%	29%	-	24%

Statistic	Ore SG						
	Sap+ Trans*	Sap	Trans	Fresh	Fresh UG	Other	Overall
Count	5	1	3	10	3	2	21
Min	2.29	-	2.29	2.38	2.81	2.72	2.29
Max	2.53	-	2.53	2.94	2.86	2.74	3.59
85th Percentile	2.49	2.31	2.49	2.92	2.85	2.74	2.89
Average	**2.40**	**2.31**	**2.41**	**2.75**	**2.83**	**2.73**	**2.72**
Std Dev	0.10	-	0.12	0.16	0.03	0.01	0.28
Coefficient of Var	4%	-	5%	6%	1%	1%	10%
Accuracy (90% CL)	3%	-	5%	3%	1%	1%	4%

**Two hard BWi samples (MET23-HSS-05 and SAP_AU3A) included in statistics calculation for "Sap+Trans" but excluded from "Sap"*

13.5 Chemical Analysis

Duplicate head cuts were removed and assayed for elements of interest in the project using standard assaying techniques. A multi-element ICP analysis and whole rock analysis (WRA) was also performed on a single head cut from each medium grade (B) sample as well as acid base accounting (ABA). The Preg-robbing index of the medium grade (B) samples was also measured (Table 13.3).

Samples were selected to represent low, medium and high grades of gold as determined by the suffix A, B and C, respectively.

Gold content in the samples was variable and measured between 0.51 and 16.1 g/tonne, averaging at 2.94 g/tonne, while silver ranged between 0.1 and 21.5 g/tonne, averaging at 1.1 g/tonne.

Sulphur in the samples measured between 0.01 and 3.15 percent, indicating a relatively small sulphide mineral component.

The Preg-robbing value of the medium grade (B) samples was also measured and indicated near negligible values of preg-robbing index for most samples.

Table 13.3: Chemical Analysis

Products	Element						PRV	PRI
	Au g/t	Ag g/t	Fe %	S %	C %	TOC %		
SAP_AU2_A	0.67	1.90	4.9	0.02	0.01	0.01	-	-
SAP_AU2_B	1.40	0.45	5.6	0.06	0.10	0.05	0.19	0.04
SAP_AU2_C	1.34	0.60	7.1	0.06	0.06	0.04	-	-
SAP_AU3_A	0.51	1.90	10.9	0.14	0.04	< 0.01	-	-
SAP_AU3_B	1.28	0.15	3.1	0.01	0.02	0.01	0.04	0.01
SAP_AU3_C	2.74	0.60	6.4	0.08	0.02	0.02	-	-
SAP_AU3A_A	0.57	0.40	7.1	0.16	0.40	0.08	-	-
SAP_AU3A_B	3.34	3.20	5.1	0.04	0.22	0.05	0.20	0.04
SAP_AU3A_C	1.90	21.25	8.5	0.16	0.25	0.04	-	-
TRANS_AU2_A	0.75	0.30	12.5	0.33	2.16	0.02	-	-
TRANS_AU2_B	1.58	0.40	12.1	0.66	1.80	0.03	<0.01	*0*
TRANS_AU2_C	1.41	0.40	9.6	0.74	1.20	0.01	-	-
TRANS_AU3_A	0.54	0.10	10.8	0.85	0.76	0.03	-	-
TRANS_AU3_B	1.87	0.20	8.3	1.43	0.86	0.04	<0.01	*0*
TRANS_AU3_C	2.82	0.15	7.8	0.51	0.39	0.04	-	-
FRESH_PIT_AU2_B	1.36	0.20	8.5	0.87	2.73	0.01	<0.01	*0*
FRESH_PIT_AU2HG_A	3.19	0.30	6.6	1.04	1.88	0.07	-	-
FRESH_PIT_AU2HG_B	7.83	0.55	8.4	1.58	2.57	0.03	<0.01	*0*
FRESH_PIT_AU2HG_C	15.7	1.15	7.2	2.21	2.38	0.06	-	-
FRESH_PIT_AU3_B	1.95	0.10	7.2	0.60	3.09	0.02	0.19	0.04
FRESH_PIT_AU3_C	1.95	0.10	7.9	0.49	2.52	0.02	-	-

Products	Element						PRV	PRI
	Au g/t	Ag g/t	Fe %	S %	C %	TOC %		
FRESH_PIT_AU3A_B	0.99	<0.1	3.9	0.53	2.02	0.08	0.15	0.03
FRESH_PIT_AU3A_C	1.44	<0.1	4.3	0.23	2.20	0.08	-	-
FRESH_PIT_AU4_A	0.92	<0.1	4.2	1.01	1.40	0.03	-	-
FRESH_PIT_AU4_B	1.18	0.10	3.6	0.86	1.85	0.02	0.04	0.01
FRESH_PIT_AU4_C	3.00	0.15	4.3	1.10	2.04	0.05	-	-
FRESH_PIT_LDZ_A	1.03	0.20	5.2	0.20	2.57	0.02	-	-
FRESH_PIT_LDZ_B	3.01	0.30	4.6	0.24	1.82	0.01	0.12	0.03
FRESH_PIT_LDZ_C	2.97	0.35	4.7	0.26	2.18	0.01	-	-
FRESH_UG_AU2_A	2.11	0.25	7.1	0.48	3.13	0.02	-	-
FRESH_UG_AU2_B	3.01	0.25	8.9	0.67	2.76	0.02	0.15	0.03
FRESH_UG_AU2_C	2.22	0.25	5.5	0.91	2.28	0.03	-	-
FRESH_UG_AU2HG_A	4.46	0.35	5.8	1.41	2.18	0.05	-	-
FRESH_UG_AU2HG_B	10.5	0.80	8.2	2.55	2.42	0.03	<0.01	*0*
FRESH_UG_AU2HG_C	16.1	1.05	9.4	3.15	2.36	0.01	-	-
FRESH_UG_AU3_A	1.50	0.30	7.2	0.50	2.85	< 0.01	-	-
FRESH_UG_AU3_B	1.66	0.35	6.8	0.62	2.65	0.06	0.09	0.02
FRESH_UG_AU3_C	2.77	0.40	8.7	1.05	3.51	0.01	-	-
FRESH_UG_LDZ_B	1.06	0.25	7.7	0.30	3.14	0.05	0.10	0.02

13.6 Mineral Analysis

Bulk Mineral Analysis (BMA) using QEMSCAN, was conducted on the medium grade (B) variability samples. This assessment provides an unsized measure of mineral composition. This analysis also provides information regarding sulphur distribution in the samples. Table 13.4 and Table 13.5 as well as Figure 13.2 and Figure 13.3 show a summary of the mineral content data for the composites and sulphur distribution data.

Pyrite accounted for the main sulphide mineral in almost all the samples. Chalcopyrite, sphalerite and other sulphides were also detected in lower concentrations in the majority of the other samples.

The non-sulphide suite of minerals varied, consisting mainly of quartz, feldspars, muscovite / illite and chlorite and clays. A number of samples contained a considerable concentration of iron-oxy hydroxides as well.

Table 13.4: Mineral Content

Mineral	Mineral Abundance - Percentage														
	FRESH PIT AU HG_B	TRANS AU3_B	FRESH PIT AU3A_B	FRESH PIT AU3_B	FRESH PIT AU4_B	FRESH PIT LDZ_B	FRESH UG_AU2_B	TRANS_ AU2_B	SAP AU2_B	SAP AU3_B	SA AU3A_B	FRESH UG_AU3_B	FRESH PIT_AU2_B	FRESH UG_AU2HG_B	FRESH UG_LDZ_B
Pyrite	2.47	2.94	0.73	0.92	2.17	0.34	1.00	0.57	0.05	0.01	0.94	0.17	1.23	4.42	0.48
Other Sulphides	0.02	0.04	0.05	0.02	0.04	0.01	0.02	0.04	0.01	0.01	0.02	0.01	0.03	0.12	0.03
Quartz	21.3	29.1	21.2	22.8	22.8	27.5	25.4	38.1	40.6	50.5	25.2	54.7	25.8	21.1	17.6
Feldspar	24.0	7.39	29.1	18.4	26.9	33.4	21.3	5.77	2.16	0.40	23.5	0.37	23.0	37.3	28.7
Muscovite / Illite	13.7	24.9	19.5	17.9	18.5	8.24	12.2	12.5	11.6	26.5	15.5	18.0	8.21	7.46	9.18
Chlorite / Clays	11.5	22.8	11.9	12.6	10.1	13.6	14.8	24.2	34.1	11.4	12.7	17.7	15.4	9.53	16.1
Other Silicates	2.17	0.27	2.19	2.11	1.99	3.37	2.62	0.48	0.24	0.07	2.15	0.55	4.16	2.54	6.40
Fe(Al)-Oxy Hydroxides	4.37	9.17	0.04	3.48	0.33	0.03	2.14	14.6	7.25	3.82	1.50	6.96	2.22	1.95	1.85
Other Oxides	3.09	1.63	0.83	1.80	0.74	0.96	3.13	2.35	3.78	6.96	1.77	1.03	3.45	3.36	2.25
Calcite	1.85	0.03	1.46	1.99	1.42	2.37	1.74	0.03	0.00	0.02	2.68	0.00	2.86	1.51	2.69
Dolomite / Ankerite	14.2	0.82	12.6	17.3	14.5	9.69	14.7	0.51	0.11	0.16	13.6	0.46	12.7	9.79	14.0
Apatite	1.17	0.80	0.32	0.62	0.39	0.40	0.77	0.74	0.06	0.05	0.51	0.01	0.87	0.87	0.65
Other	0.14	0.08	0.07	0.09	0.04	0.06	0.10	0.11	0.06	0.06	0.05	0.04	0.08	0.09	0.05
Total	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100

Table 13.5: Sulphur Distribution

Mineral	% S Distribution														
	FRESH PIT AU HG_B	TRANS AU3_ B	FRESH PIT AU3A_B	FRESH PIT AU3_B	FRESH PIT AU4_B	FRESH PIT LDZ_B	FRESH UG AU2_B	TRANS AU2_ B	SAP AU2_ B	SAP AU3_ B	SAP AU3A_B	FRESH UG_A U3_B	FRESH PIT_A U2_B	FRESH UG_AU2 HG_B	FRESH UG_LDZ_B
Pyrite	99.6	99.0	96.9	98.6	98.9	97.2	98.4	96.3	89.9	73.0	98.4	98.6	98.2	98.9	95.8
Chalcopyrite	0.22	0.65	1.77	0.89	0.80	1.95	0.94	2.12	4.95	0.48	1.42	0.49	1.43	0.17	1.72
Sphalerite	0.08	0.05	0.37	0.12	0.08	0.11	0.21	0.33	0.00	3.17	0.00	0.23	0.00	0.15	0.75
Other Sulphides	0.06	0.19	0.70	0.11	0.12	0.19	0.24	0.39	1.14	6.17	0.22	0.43	0.19	0.79	1.61
Other	0.02	0.12	0.26	0.26	0.09	0.59	0.18	0.83	4.01	17.2	0.00	0.21	0.20	0.04	0.12
Total	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100

Figure 13.2: Mineral Content

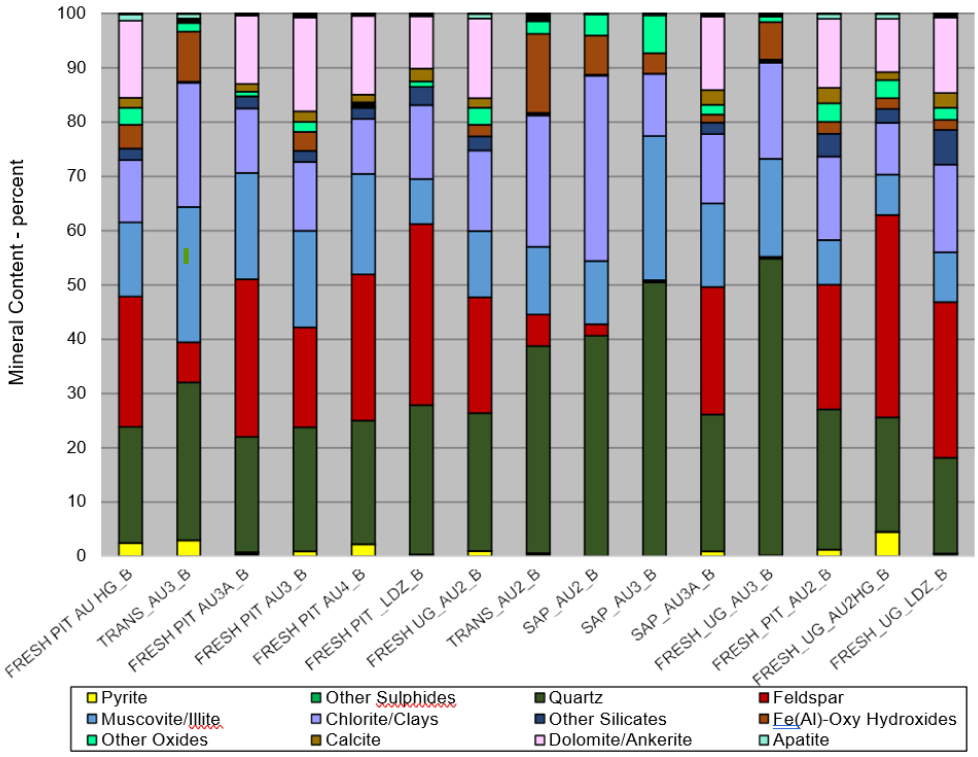
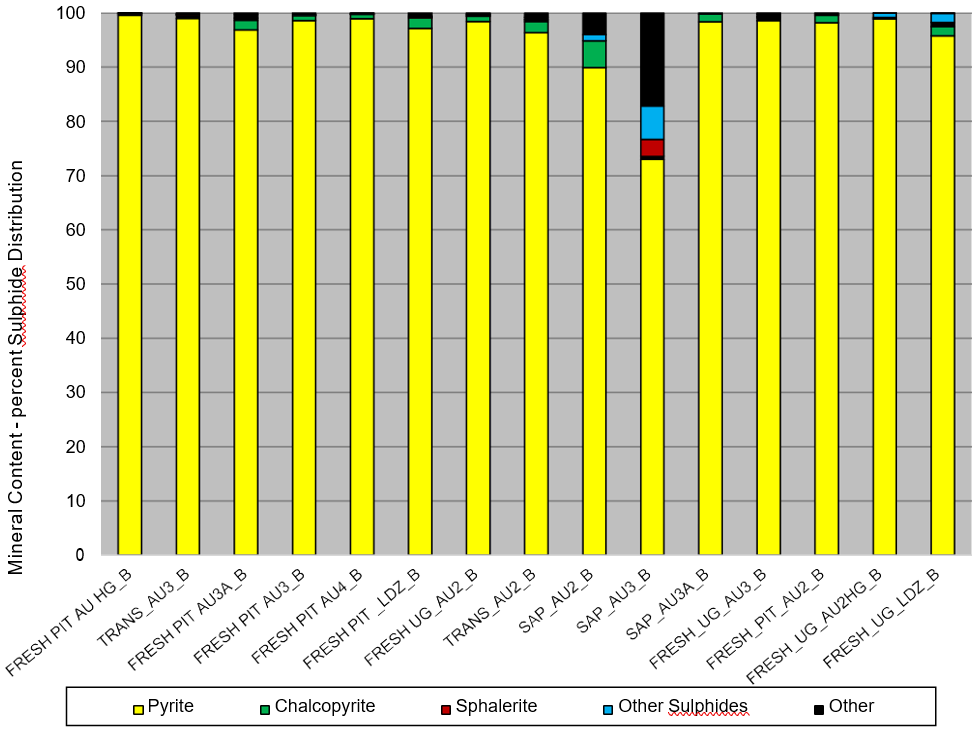


Figure 13.3: Mineral Analysis – Sulphur Distribution



13.7 Gravity Test Work

An E-GRG test was performed on three select domain samples; SAP_AU3A, FRESH_PIT_LDZ and FRESH_UG_LDZ to determine the potential for gravity recoverable gold.

These tests were performed by first preparing 20 kilograms of the feed to 100% passing 1.7 mm and passing the entire crushed material through a Knelson MD-3 concentrator at a force of 60-Gs. The concentrate was retained and sized for assay; the tailings were sub-sampled for sizing. The tailings were ground in a laboratory rod mill to a grind target of 212 µm K_{80} and processed through the Knelson Concentrator a second time (Pass 2), the concentrate and tailings were sampled for assay and sizing as per the initial pass before regrinding the tailings to a final target of 75 µm K_{80} and repassing through the Knelson concentrator (Pass 3). The final tailings were sampled, sized, and assayed by size. All assaying included gold by fire assay with concentrate fractions assayed to extinction. A flowsheet schematic of the tests along with a summary of the results is displayed in Figure 13.4 and Table 13.6.

Gravity gold recovery for FRESH_UG_LDZ measured 29.3 percent with gold concentrate assaying 34 g/tonne. For SAP_AU3A, the feed sizing was much finer than 1.7 mm K_{100} and measured around 527 µm K_{80}. Gold recovery of 50.1 percent, assaying 64 g/tonne gold was measured. Gravity gold recovery for FRESH_PIT_LDZ measured 27.9 percent, with the gold concentrate assaying 41 g/tonne.

Figure 13.4: Gravity Test Work Flowsheet

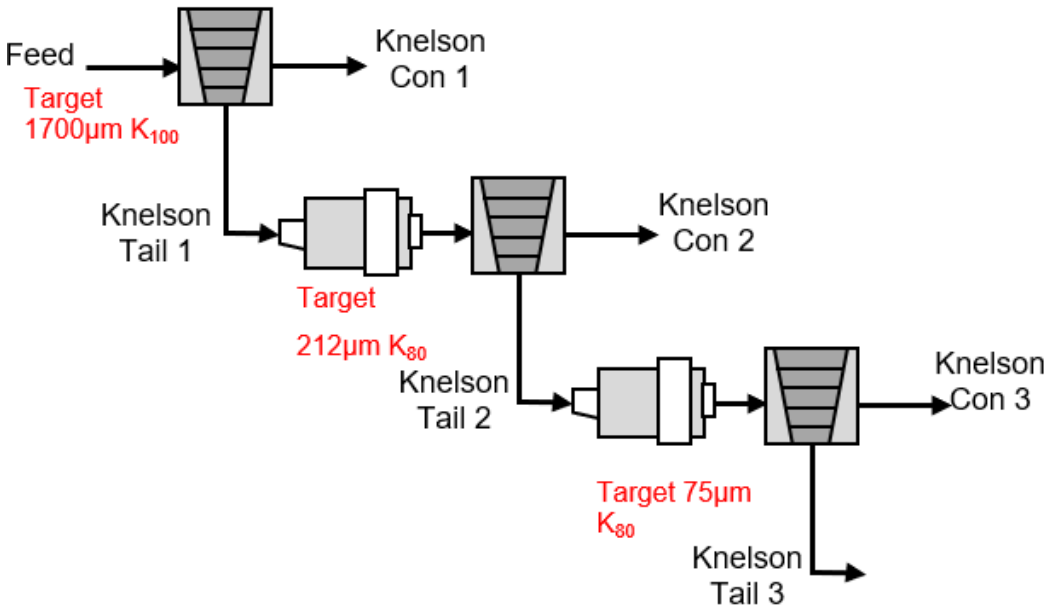


Table 13.6: E-GRG Test Results

Sample ID	Product	Weight %	Au Assay g/tonne	Au Distribution Percent
Fresh UG_LDZ	Knelson Con 1	0.5	16.4	5.4
	Knelson Con 2	0.5	47.6	13.7
	Knelson Con 3	0.4	42.9	10.3
	Knelson Tail 3	96.6	1.16	70.7
Gravity Recoverable Gold			**34**	**29.3**
SAP_AU3A	Knelson Con 1	0.4	75.6	20.1
	Knelson Con 2	0.4	69.5	18.6
	Knelson Con 3	0.4	45.3	11.4
	Knelson Tail 3	97.0	0.78	49.9
Gravity Recoverable Gold			**64**	**50.1**
FRESH_PIT_LDZ	Knelson Con 1	0.5	14.8	5.6
	Knelson Con 2	0.4	42.3	15.3
	Knelson Con 3	0.4	39.6	12.6
	Knelson Tail 3	97.4	0.82	66.4
Gravity Recoverable Gold			**41**	**27.9**

Based on the favourable GRG results, it is recommended to include a gravity circuit in the flowsheet.

13.8 Gravity Leach Testing

A series of gravity leach tests were performed on the 10 variability samples, initially evaluating primary grind size at 75 and 106 µm K_{80}. These tests also evaluated sparging with air, versus sparging with oxygen. Tests performed using air consisted of 1,000 ppm sodium cyanide (NaCN) and tests performed using oxygen consisted of 500 ppm NaCN. Tests were performed at a feed density of 40 percent solids and pH 10.5, modified with lime. Following these results, a series of tests were conducted evaluating a finer primary grind size of 53 µm K_{80}, as well as evaluating higher concentration of cyanide with oxygen sparging and no pre-oxidation step. Table 13.7 and Table 13.8 summarizes gravity leach test results.

Gold extraction of the gravity concentrates ranged between 3.6 and 30.5 percent. Overall, gold extraction was high after 48 hours of leaching and ranged between 85.5 and 99.6 percent, averaging 93.3 percent. Gold extraction after 24 hours of leaching was of particular interest, with gold recoveries ranging from 80 to over 99 percent. The optimization focused on improving gold recovery at the 24-hour mark.

Results indicated that for the majority of the samples, an increase in gold extraction was obtained, as primary grind size decreased to 75 µm K_{80}, using a higher concentration of NaCN, and using oxygen sparging. Grinding finer to 53 µm K_{80}, was not beneficial to gold extraction.

For most samples, gold recovery at 24 hours improved at higher cyanide concentrations of 1,000 ppm. A higher cyanide consumption of 1,000 ppm was used going forward.

The Transition and Saprolite samples also displayed higher lime consumption without the pre-oxidation step. A 2-hour pre-oxidation step was implemented going forward.

The remaining variability composites were tested at 75 µm K_{80}, a sodium cyanide concentration of 1,000 ppm, sparged with oxygen, and subject to a 2-hour pre-oxidation stage. Results are discussed in the following sub-section.

Table 13.7: Gravity Leach Testing Summary

Test	Sample ID	PG µm K_{80}	NaCN ppm	Air vs O_2	2 hr Pre Ox	Au Rec - Percent		
						Moz Con	Leach at Time	
							24 hr	48 hr
CN-04	FRESH_PIT_AU4_B	106	1,000	Air	-	14.4	80.4	**92.7**
CN-05		106	500	O_2	-	17.1	89.0	**93.5**
CN-06		75	1,000	Air	-	17.4	94.3	**95.1**
CN-07		75	500	O_2	-	10.8	84.2	**91.3**
CN-52		75	1,000	O_2	-	12.2	90.4	**96.6**
CN-58		53	1,000	O_2	✓	15.4	93.7	**95.8**
CIL-67		75	1,000	O_2	✓	11.3	95.3	**95.6**
CN-08	SAP_AU2_B	106	1,000	Air	-	6.5	86.6	**97.6**
CN-09		106	500	O_2	-	5.1	90.3	**97.3**
CN-10		75	1,000	Air	-	5.6	86.3	**96.2**
CN-11		75	500	O_2	-	4.6	96.6	**99.0**
CN-44		75	500	O_2	✓	5.0	96.1	**99.6**
CN-45		75	1,000	O_2	✓	4.1	99.6	**99.6**
CIL-61		75	1,000	O_2	✓	5.5	93.6	**98.2**

Test	Sample ID	PG µm K_{80}	NaCN ppm	Air vs O_2	2 hr Pre Ox	Au Rec - Percent		
						Moz Con	Leach at Time	
							24 hr	48 hr
CN-12	TRANS_AU2_B	106	1,000	Air	-	7.4	85.0	**89.0**
CN-13		106	500	O_2	-	7.7	86.2	**88.2**
CN-14		75	1,000	Air	-	6.8	86.0	**90.2**
CN-15		75	500	O_2	-	12.7	91.2	**92.2**
CN-46		75	1,000	O_2	-	5.2	91.8	**94.1**
CN-47		75	1,000	O_2	✓	10.1	91.8	**96.6**
CN-57		53	1,000	O_2	✓	20.3	97.1	**94.6**
CIL-62		75	1,000	O_2	✓	11.8	88.3	**91.4**
CN-16	FRESH_PIT_AU2HG_B	106	1,000	Air	-	16.2	84.3	**90.0**
CN-17		106	500	O_2	-	22.0	86.0	**91.5**
CN-18		75	1,000	Air	-	19.0	89.1	**92.3**
CN-19		75	500	O_2	-	19.8	86.0	**93.5**
CN-49		75	1,000	O_2	✓	10.0	92.8	**94.8**
CN-50		75	1,000	O_2	✓	6.7	94.8	**94.5**
CIL-64		75	1,000	O_2	✓	18.4	92.2	**93.2**
CN-20	FRESH_PIT_AU3A_B	106	1,000	Air	-	15.6	91.2	**97.3**
CN-21		106	500	O_2	-	18.5	90.8	**96.8**
CN-22		75	1,000	Air	-	16.3	91.8	**96.5**
CN-23		75	500	O_2	-	19.0	98.0	**97.2**
CIL-66		75	1,000	O_2	✓	8.7	95.4	**95.7**

Table 13.8: Gravity Leach Testing Summary

Test	Sample ID	PG µm K_{80}	NaCN ppm	Air vs O_2	2 hr Pre Ox	Au Rec - Percent		
						Moz Con	Leach at Time	
							24 hr	48 hr
CN-24	FRESH_UG_AU2_B	106	1,000	Air	-	18.3	90.7	**93.2**
CN-25		106	500	O_2	-	24.7	92.8	**95.8**
CN-26		75	1,000	Air	-	30.5	93.1	**94.2**
CN-27		75	500	O_2	-	27.1	92.0	**94.9**
CN-53		75	1,000	O_2	✓	8.5	94.1	**96.3**
CIL-68		75	1,000	O_2	✓	10.2	94.0	**95.9**
CN-28	FRESH_PIT_AU2_B	106	1,000	Air	-	22.3	86.2	**90.0**
CN-29		106	500	O_2	-	9.6	85.7	**88.9**
CN-30		75	1,000	Air	-	18.0	85.3	**90.4**
CN-31		75	500	O_2	-	21.5	89.5	**91.3**
CN-48		75	1,000	O_2	-	6.3	95.9	**93.1**
CIL-63		75	1,000	O_2	✓	8.9	86.4	**86.2**
CN-32	FRESH_PIT_AU3_B	106	1,000	Air	-	27.4	92.7	**93.6**
CN-33		106	500	O_2	-	29.4	89.4	**92.6**
CN-34		75	1,000	Air	-	17.5	91.1	**93.0**
CN-35		75	500	O_2	-	28.0	90.4	**94.5**
CN-51		75	1,000	O_2	-	6.4	95.4	**97.1**
CIL-65		75	1,000	O_2	✓	11.2	91.3	**94.2**
CN-36	FRESH_UG_ AU2HG_B	106	1,000	Air	-	4.7	82.9	**85.5**
CN-37		106	500	O_2	-	6.5	84.3	**86.7**
CN-38		75	1,000	Air	-	7.8	86.4	**89.4**
CN-39		75	500	O_2	-	8.0	88.5	**89.9**
CN-54		75	1,000	O_2	-	3.6	89.5	**91.4**
CN-55		75	1,000	O_2	✓	5.3	89.7	**91.9**
CN-59		53	1,000	O_2	✓	9.2	92.5	**91.9**
CIL-69		75	1,000	O_2	✓	6.1	83.1	**89.7**

Test	Sample ID	PG µm K_{80}	NaCN ppm	Air vs O_2	2 hr Pre Ox	Au Rec - Percent		
						Moz Con	Leach at Time	
							24 hr	48 hr
CN-40	FRESH_UG_AU3_B	106	1,000	Air	-	7.8	87.8	**88.3**
CN-41		106	500	O_2	-	17.6	89.9	**88.0**
CN-42		75	1,000	Air	-	16.8	89.9	**91.9**
CN-43		75	500	O_2	-	14.5	91.2	**91.1**
CN-56		75	1,000	O_2	✓	9.3	91.8	**94.8**
CN-60		53	1,000	O_2	✓	5.8	90.5	**93.7**
CIL-70		75	1,000	O_2	✓	11.8	83.2	**92.5**

13.9 Gravity-CIL Tests

The last set of tests evaluated gravity carbon in leach (CIL) testing, with leach test conditions using a primary grind size of 75 µm K_{80}, 1,000 ppm NaCN, sparged with oxygen and a 2-hour pre-oxidation step being implemented. For these tests, 40 g/L of carbon was used. Table 13.9 summarize the test results.

For the majority of these tests, the addition of carbon did not improve gold extraction or was detrimental to gold extraction. Carbon in leach was not used going forward for the variability composites.

Table 13.9: Gravity CIL Test Summary

Test	Sample ID	Au Rec - %		Consumption - kg/t		Au in CnTl
		Moz Con	48 hr CIL	NaCN	Lime	g/t
CIL-67	FRESH_PIT_AU4_B	11.3	**95.6**	0.89	0.57	0.06
CIL-61	SAP_AU2_B	5.5	**98.2**	2.29	2.89	0.03
CIL-62	TRANS_AU2_B	11.8	**91.4**	2.18	3.05	0.13
CIL-64	FRESH_PIT_AU2HG_B	18.4	**93.2**	1.62	1.21	0.49
CIL-66	FRESH_PIT_AU3A_B	8.7	**95.7**	2.25	0.76	0.05
CIL-68	FRESH_UG_AU2_B	10.2	**95.9**	1.63	1.11	0.10
CIL-63	FRESH_PIT_AU2_B	8.9	**86.2**	1.18	1.07	0.18
CIL-65	FRESH_PIT_AU3_B	11.2	**94.2**	1.66	0.83	0.10
CIL-69	FRESH_UG_AU2HG_B	6.1	**89.7**	1.72	0.91	1.00
CIL-70	FRESH_UG_AU3_B	11.8	**92.5**	1.49	0.58	0.17

13.10 Metallurgical Testing- Variability Samples

Most favorable test conditions and results from optimization testing on the medium grade (B) samples were used to evaluate gold extraction from the remaining variability samples which included the low (A) and high (C) grade samples in this test program. The flowsheet consisted of a primary grind sizing of 75 µm K_{80}, with no pre-oxidation step, 1,000 ppm of NaCN and sparged with oxygen. Figure 13.5 and

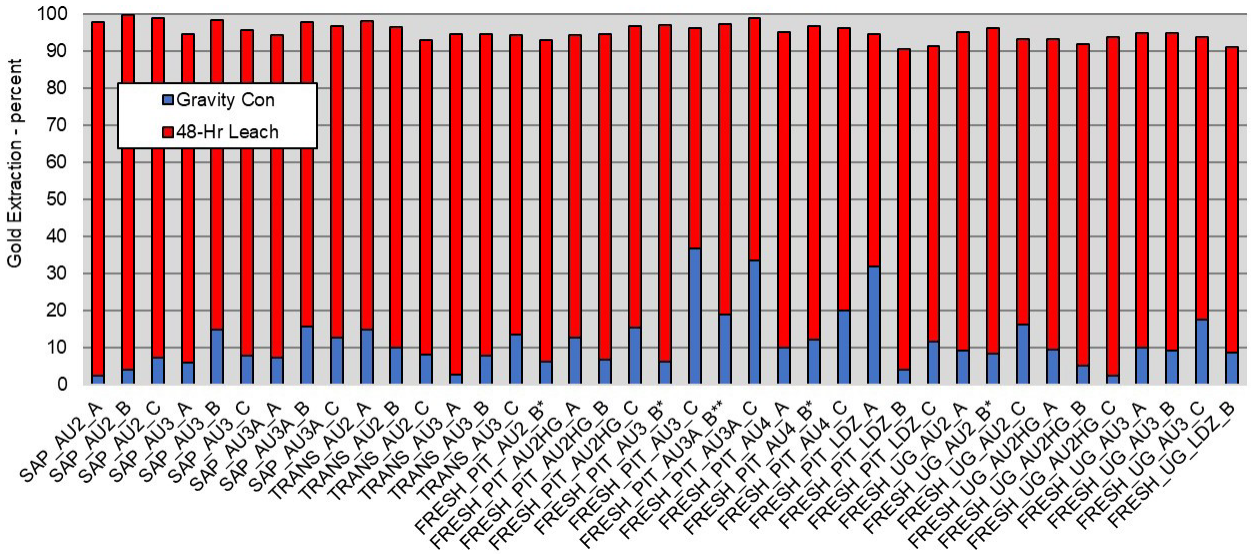
Table 13.10 display gold extraction for these samples, using the same set of test conditions. It should be noted that a number of samples only had a medium grade composite.

The samples all responded very well to the conditions tested. Gold recovery to the gravity concentrates ranged between 2.4 and 36.7 percent, averaging at 11.8 percent. Gold extraction at 24 hours, including the gravity recovery, ranged between 87.4 and 99.6 percent, averaging at 93.8 percent. The leach tests were performed for a period of 48 hours. Gold extraction, including gravity concentration, at 48 hours, ranged between 90.6 and 99.6 percent, averaging 95.3 percent.

Cyanide consumption for these samples ranged between 0.2 and 1.0 kg/tonne, averaging 0.6 kg/tonne, while lime consumption ranged between 0.8 and 10.5 kg/tonne, averaging 2.7 kg/tonne.

There was no discernible correlation noted between feed gold content and overall gold extraction for the samples tested

Figure 13.5: Optimum Overall Gold Extraction Test Results



*Note: *Pre-ox was not applied to these tests.*
***Only 500 ppm NaCN was used for this test.*

Table 13.10: Overall Gold Extraction Results at Optimum Conditions

Test	Sample ID	Moz Con Rec		Au Ext (%) at T hours		Consumption kg/t		CNTI Au	Recalc. Au	Au Head
		Mass (g)	Au (%)	24	48	NaCN	Lime	g/t	g/t	g/t
CN-71	SAP_AU2_A	0.66	2.6	98	**97.8**	0.67	6.56	0.02	0.68	0.67
CN-45	SAP_AU2_B	0.97	4.1	100	**99.6**	0.39	4.04	0.01	1.33	1.40
CN-72	SAP_AU2_C	0.94	7.4	99	**99.0**	0.61	10.45	0.02	1.46	1.34
CN-73	SAP_AU3_A	1.92	6.1	95	**94.6**	0.42	5.62	0.04	0.65	0.51
CN-95	SAP_AU3_B	0.86	14.8	96	**98.5**	0.38	1.28	0.02	1.33	1.28
CN-74	SAP_AU3_C	2.32	8.0	94	**95.7**	0.33	2.37	0.11	2.56	2.74
CN-75	SAP_AU3A_A	2.81	7.2	94	**94.3**	0.49	4.35	0.05	0.79	0.57
CN-96	SAP_AU3A_B	0.84	15.8	95	**97.7**	0.46	2.84	0.07	3.04	3.34
CN-76	SAP_AU3A_C	0.56	12.6	93	**96.7**	0.59	3.81	0.05	1.50	1.90
CN-77	TRANS_AU2_A	1.70	15.0	96	**98.2**	0.5	3.23	0.02	0.82	0.75
CN-47	TRANS_AU2_B	1.70	10.1	92	**96.6**	0.46	4.05	0.06	1.77	1.58
CN-78	TRANS_AU2_C	0.96	8.2	93	**93.0**	0.54	6.36	0.12	1.71	1.41
CN-79	TRANS_AU3_A	0.40	2.7	92	**94.5**	0.53	2.32	0.05	0.81	0.54
CN-97	TRANS_AU3_B	2.16	7.8	94	**94.7**	0.47	3.14	0.13	2.43	1.87
CN-80	TRANS_AU3_C	0.62	13.6	93	**94.2**	0.34	3.12	0.11	1.90	2.82
CN-48*	FRESH_PIT_AU2_B*	1.49	6.3	96	**93.1**	0.92	1.81	0.10	1.44	1.36
CN-81	FRESH_PIT_AU2HG_A	0.58	12.7	93	**94.2**	0.65	2.69	0.16	2.67	3.19
CN-50	FRESH_PIT_AU2HG_B	1.15	6.7	95	**94.5**	0.87	1.25	0.47	8.55	7.83
CN-82	FRESH_PIT_AU2HG_C	0.46	15.6	96	**96.9**	0.75	2.61	0.55	17.4	15.68

Test	Sample ID	Moz Con Rec		Au Ext (%) at T hours		Consumption kg/t		CNTI Au	Recalc. Au	Au Head
		Mass (g)	Au (%)	24	48	NaCN	Lime	g/t	g/t	g/t
CN-51*	FRESH_PIT_AU3_B*	1.14	6.4	95	**97.1**	0.58	1.13	0.07	2.24	1.95
CN-83	FRESH_PIT_AU3_C	0.60	36.7	95	**96.1**	0.60	2.29	0.04	1.03	1.95
CN-23**	FRESH_PIT_AU3A_B**	1.45	19.0	98	**97.2**	0.27	0.90	0.03	1.07	0.99
CN-84	FRESH_PIT_AU3A_C	0.36	33.6	96	**98.9**	0.45	1.95	0.02	1.38	1.44
CN-85	FRESH_PIT_AU4_A	0.35	10.0	92	**95.1**	0.54	1.32	0.06	1.22	0.92
CN-52*	FRESH_PIT_AU4_B*	1.99	12.2	90	**96.6**	0.46	0.84	0.05	1.47	1.18
CN-86	FRESH_PIT_AU4_C	0.72	19.9	95	**96.2**	0.57	1.54	0.12	3.06	3.00
CN-87	FRESH_PIT_LDZ_A	0.76	32.1	93	**94.6**	0.55	1.94	0.07	1.30	1.03
CN-98	FRESH_PIT_LDZ_B	0.52	4.1	87	**90.6**	0.32	1.50	0.27	2.83	3.01
CN-88	FRESH_PIT_LDZ_C	0.63	11.6	88	**91.3**	0.17	1.79	0.23	2.58	2.58
CN-89	FRESH_UG_AU2_A	0.64	9.3	97	**95.2**	0.73	1.91	0.10	1.99	2.11
CN-53*	FRESH_UG_AU2_B*	0.94	8.5	94	**96.3**	0.83	1.66	0.10	2.56	3.01
CN-90	FRESH_UG_AU2_C	0.57	16.4	93	**93.3**	0.57	1.74	0.15	2.15	2.22
CN-91	FRESH_UG_AU2HG_A	1.32	9.6	92	**93.4**	0.58	1.69	0.28	4.22	4.46
CN-55	FRESH_UG_AU2HG_B	0.95	5.3	90	**91.9**	0.95	1.09	0.80	9.79	10.5
CN-92	FRESH_UG_AU2HG_C	0.56	2.4	93	**93.8**	0.54	1.84	0.97	15.5	16.12
CN-93	FRESH_UG_AU3_A	0.76	10.1	96	**94.9**	0.64	1.95	0.11	2.04	1.50
CN-56	FRESH_UG_AU3_B	1.01	9.3	92	**94.8**	0.34	1.40	0.10	1.90	1.66
CN-94	FRESH_UG_AU3_C	0.85	17.7	94	**93.7**	0.64	2.00	0.16	2.53	2.77
CN-99	FRESH_UG_LDZ_B	1.37	8.6	89	**91.0**	0.78	3.51	0.13	1.39	1.06

13.11 Cyanide Destruction Tests

Following the domain and variability tests, bulk leach tests were performed on three (3) blends representing the initial mining years, followed by cyanide detoxification on the cyanide tailings via the SO_2 / Air method. These tests initially targeted a CNWAD value of < 5 ppm; once that was achieved, limited optimization was performed with remaining tests focusing on targeting a CNWAD value of < 1 ppm. The detoxification results are summarized in Table 13.11.

The target CNWAD value of < 5 ppm was achieved for all three (3) composites at an SO2:CNWAD ratio of 4 and a copper addition of 25 mg/L and a retention time of 60 minutes. For Blend 3, this target was achieved with a lower copper addition of 10 mg/L.

To achieve a CNWAD value of < 1 ppm, Blend 1 required higher copper addition of 35 mg/L, Blend 2 required higher copper addition of 50 mg/L, but there may be room to optimize these conditions further. Blend 3 required a copper addition of 15 mg/L. A retention time of 60 minutes was sufficient for all tests. The final cyanide detoxified slurries from each test, meeting the target CNWAD, were packaged and sent for additional cyanide speciation, as well as ABA and NAG testing.

Table 13.11: Cyanide Detoxification Summary

Sample	Detox Test	Test Parameters				Test Length		Feed / Detox Solution Assays - ppm					Target WADCN ppm
		pH	Return Time	Reagent Used		Mins	Number of Displacements	CN_{MP}	Cu	Fe	Ni	Zn	
			Mins	SO_2 g/g CN_{MP}	Cu mg/L								
Test 115 CNTL	Feed	10.4	-	-	-	-	-	390	12.6	3.00	0.88	1.95	-
	C1	8.0	120	4.0	25	360	3	1.29	0.02	4.00	0.36	<0.01	<5
	C2	8.0	60	4.0	25	180	3	1.22	<0.01	5.00	0.43	<0.01	
	C3	8.0	60	5.0	25	180	3	1.44	0.07	1.00	0.31	<0.01	<1
	C4	8.0	60	5.0	35	180	3	0.46	0.03	4.00	0.39	<0.01	
	C5	8.0	60	4.0	35	180	3	0.44	0.05	4.00	0.40	<0.01	
Test 116 CNTL	Feed	10.7	-	-	-	-	-	319	14.4	12.0	0.93	1.77	-
	C1	8.2	120	4.0	25	360	3	0.94	0.08	2.00	0.49	<0.01	<5
	C2	8.1	60	4.0	25	180	3	1.39	0.11	2.00	0.50	<0.01	
	C3	8.2	60	5.0	25	180	3	2.08	0.10	1.00	0.44	<0.01	<1
	C4	8.1	60	5.0	50	180	3	0.90	0.05	0.50	0.40	<0.01	
	C5	8.1	120	5.0	50	360	3	0.50	<0.01	0.50	0.08	<0.01	
Test 117 CNTL	Feed	10.0	-	-	-	-	-	324	8.81	22.0	0.80	1.08	-
	C1	8.0	120	4.0	25	360	3	1.22	0.15	4.00	0.41	<0.01	<5
	C2	8.2	60	4.0	25	180	3	1.49	0.11	4.00	0.34	<0.01	
	C3	8.1	60	4.0	10	180	3	1.59	0.10	8.00	0.35	<0.01	
	C4	8.2	60	5.0	25	180	3	0.63	0.02	5.00	0.16	<0.01	<1
	C5	8.2	60	5.0	15	180	3	0.56	<0.01	6.00	0.14	<0.01	

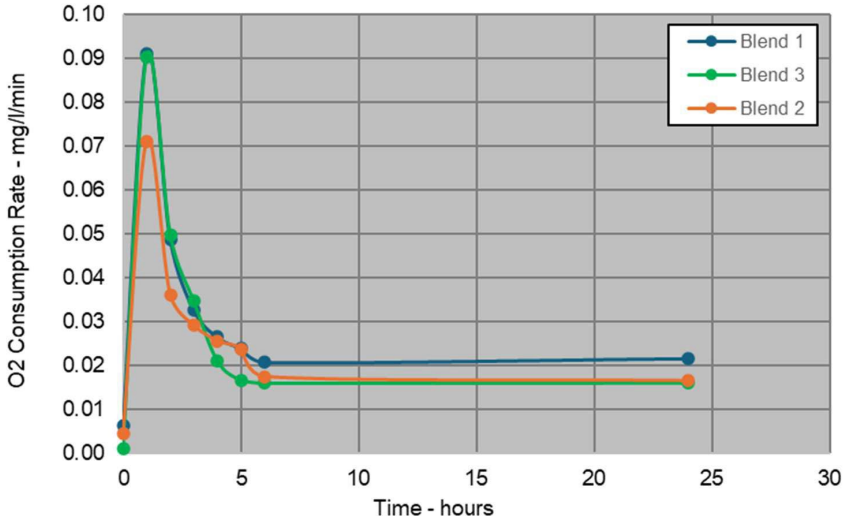
13.12 Oxygen Uptake Rate Testing

The three (3) Blend Composites were subject to oxygen uptake rate determinations. The test procedure described by G.M. Fraser was utilized. A 1-kilogram charge from each composite was ground to 75 µm K_{80} and transferred to an agitated vat, where the pH of the sample was adjusted to 10.5 and 500 ppm of NaCN was added. Each test lasted 24 hours, with dissolved oxygen measurements being taken every minute for 15 minutes at time intervals of 0, 1, 2, 3, 4, 5, 6 and 24 hours. Oxygen consumption results are summarized in Table 13.12 and Figure 13.6. For all three (3) composites, oxygen demand seems to be met within the first two (2) to three (3) hours.

Table 13.12: Oxygen Uptake Rate Summary

Sample	Oxygen Consumption @ Time - Hours							
	0	1	2	3	4	5	6	24
Blend 1	0.006	0.09	0.05	0.03	0.03	0.02	0.02	0.02
Blend 2	0.005	0.07	0.04	0.03	0.03	0.02	0.02	0.02
Blend 3	0.001	0.09	0.05	0.03	0.02	0.02	0.02	0.02

Figure 13.6: Oxygen Uptake Rate



13.13 Carbon Modeling Testing

Carbon loading test work generates carbon modeling data which supports defining a carbon management strategy. This can optimize plant design and performance around gold inventory. Test data generated are specific to the carbon tested. The carbon used for all tests was pre-attritioned carbon, manufactured / size

by ‘Calgon / 6x12 mesh’. Blend Composites 1, 2 and 3 were tested. For each sample, 8 kg of sample was subjected to a cyanide leach test. At leach termination, the pulp was split; one half retained as slurry for sequential triple contact carbon loading and the second half filtered to obtain pregnant leach solution (PLS) for carbon contacting to produce isotherm curves. Leach conditions replicated those optimized in the earlier composite testing, sparged with oxygen and subject to a 2-hour pre-oxidation period.

13.13.1 Carbon Equilibrium Testing

Carbon equilibrium loading data were generated by contacting known volumes of pregnant leach solution with varying levels of activated carbon. Data are provided in Table 13.13 and Table 13.14.

Table 13.13: Carbon Equilibrium Test Summary

Comp	Test	Carbon		Solution mL	Solution	Au mg/L	Loading
		Conc. g/L	Added grams		Initial mg/L	Final mg/L	Au g/t (calc)
Blend 1	N1	4	3.6	900	0.80	0.001	200
	N2	2	1.8	900	0.80	0.01	395
	N3	1	0.9	900	0.80	0.01	790
	N4	0.5	0.5	900	0.80	0.06	1480
	N5	0.2	0.2	900	0.80	0.29	2550
Blend 2	N1	4	3.6	900	1.18	0.002	295
	N2	2	1.8	900	1.18	0.005	588
	N3	1	0.9	900	1.18	0.04	1140
	N4	0.5	0.5	900	1.18	0.18	2000
	N5	0.2	0.2	900	1.18	0.64	2700
Blend 3	N1	4	4.0	1,000	0.73	0.002	182
	N2	2	2.0	1,000	0.73	0.005	363
	N3	1	1.0	1,000	0.73	0.01	720
	N4	0.5	0.5	1,000	0.73	0.13	1200
	N5	0.2	0.2	1,000	0.73	0.35	1900

Table 13.14: Carbon Loading Capacity

Comp	Loading Capacity g/L		
	1	0.5	0.2
Blend 1	3,293	2,505	1,744
Blend 2	2,101	1,737	1,352
Blend 3	1,835	1,477	1,109

13.13.2 Sequential Carbon in Pulp (Triple Contact)

The Fleming Kinetic Constants are determined from sequential carbon triple contact tests, in which a known mass of fresh activated carbon is contacted with a known pulp volume, kinetic solution samples are removed over a 2-hour period at which point the carbon is transferred to freshly leached pulp and contacted for four (4) hours. A final carbon transfer is made to fresh pulp and contacted for an additional 16 hours (24 total) before the carbon is assayed. The Fleming constant by ‘k’ and constant ‘n’ are summarized Table 13.15; both considered to be within the typical range.

Table 13.15: Fleming constants

Comp	Fleming Constants	
	k (hr^{-1})	'n'
Blend 1	85.8	0.84
Blend 2	93.9	0.72
Blend 3	86.1	0.87

13.14 Dewatering Tests

A series of dewatering tests were performed on the Blend 1, 2 and 3 feeds. Thickening characteristics of the sample were studied, beginning with a small-scale floc scoping to identify a candidate flocculant, followed by static settling in cylinders and dynamic thickening at bench- scale. Thickened products from dynamic thickening included rheology. Test details are discussed below.

13.14.1 Flocculant Scoping Tests

Initially, scoping tests were conducted on the Blend 1, 2 and 3 feed samples. A total of five (5) flocculants were screened using small scale settling tests in 250 mL cylinders; flocculants screened included Magnafloc (MF) MF10, MF380, MF351, MF156 and AN913. Results are summarized Table 13.16.

Flocculant was prepared to a concentration of 0.1 g/L and dosed at 10 g/t; performance was measured semi-quantitatively. Results were compared for slurry samples at a constant volume of 250 mL and feed density of 15% solids; scoping tests suggested AN913 was the most promising flocculant and formed the basis for static settling test work.

Table 13.16: Flocculant Scoping Test Results

Composite	Test	pH	Initial Density (% solids)	Floc	Floc Type	Dosage g/t	Final Density (% solids)	Settling Rate mm/s
Blend-01	F-1	7.5	15.0	MF10	anionic	10.0	37.2	1.6
		7.5	15.0	MF380	cationic	10.0	25.5	0.3
		7.5	15.0	MF351	non-ionic	10.0	33.2	0.5
		7.5	15.0	MF156	anionic	10.0	35.8	1.1
		7.5	15.0	AN913	anionic	10.0	37.9	2.2
Blend-02	F-2	7.5	15.0	MF10	anionic	10.0	46.5	3.4
		7.5	15.0	MF380	cationic	10.0	37.2	0.7
		7.5	15.0	MF351	non-ionic	10.0	42.3	2.1
		7.5	15.0	MF156	anionic	10.0	43.2	2.7
		7.5	15.0	AN913	anionic	10.0	50.2	3.3
Blend-03	F-3	7.4	15.0	MF10	anionic	10.0	50.9	4.2
		7.4	15.0	MF380	cationic	10.0	48.9	1.2
		7.4	15.0	MF351	non-ionic	10.0	51.6	4.5
		7.4	15.0	MF156	anionic	10.0	48.9	3.7
		7.4	15.0	AN913	anionic	10.0	50.3	3.8

13.14.2 Static Settling Tests

Static settling tests (Standard Kynch) were completed to evaluate flocculant dosages using the selected AN913 flocculant. The feed solids were diluted to approximately 14% solids. Testing was completed in 2 L glass cylinders; thickened solids were raked after passing the critical settling point. A summary of results is shown in Table 13.17. Static testing of the sample suggested thickening operating parameters of 40 g/t of AN913 provided sufficient static settling characteristics to commence optimization of the dynamic thickening level.

Table 13.17: Static Settling Tests

Test	Sample	Flocculant g/t	pH	Density (%)		Free Settling Velo. (m/h)
				Initial	Final	
S-1	Blend 1	40	7.5	13.4	51.3	2.7
S-2		60	7.5	13.4	49.5	15.8
S-3		50	10.5	13.3	45.3	14.6
S-4	Blend 2	40	7.5	13.8	56.0	3.8
S-5		60	7.5	13.6	44.1	12.3
S-6		50	10.5	13.6	51.6	20.0
S-7	Blend 3	40	7.4	13.6	60.2	21.7
S-8		50	7.4	13.6	60.3	21.0
S-9		40	10.5	13.6	61.4	22.1

13.14.3 Dynamic Settling Tests

The results of the preliminary static settling-thickening data formed the basis to determine the pre-optimized conditions for dynamic testing. Semi-continuous dynamic testing was conducted using a custom test unit which consisted of a 100 mm diameter laboratory thickener. General conditions established during static testing formed the basis for dynamic optimization testing. Criteria tested included feed flux loading rate and flocculant dosage is summarized by test in Table 13.18.

The highest underflow density of 52.1 to 61.4% for the samples was achieved with 40 g/t AN913SH and 0.5 t/m^2/h loading rate and pH 10.5. Turbidity measured very low at between 9 and 17 mg/L at this condition.

Table 13.18: Dynamic Thickening Tests

Test	Sample	Density (%)		Flocculant		pH	Rise Rate m/h	Loading Rate t/m^2/h	Turbidity mg/L	Unsheared Yield Stress (Pa)
		Feed	U/F	Type	g/t					
D1-A	Blend 1	15.0	52.1	AN913SH	40.0	10.5	3.1	0.5	9.0	87.1
D1-B		15.0	51.0	AN913SH	40.0	10.5	4.3	0.7	12.0	92.5
D1-C		15.0	49.6	AN913SH	40.0	10.5	6.2	1.0	13.0	63.0
D1-D		15.0	41.7	AN913SH	20.0	10.5	4.3	0.7	63.0	7.0
D1-E		15.0	53.0	AN913SH	60.0	10.5	4.4	0.7	1.0	104.8
D2-A	Blend 2	15.0	56.9	AN913SH	40.0	10.5	3.1	0.5	11.0	108.2
D2-B		15.0	54.3	AN913SH	40.0	10.5	4.3	0.7	19.0	58.6
D2-C		15.0	53.6	AN913SH	40.0	10.5	6.2	1.0	24.0	49.3
D2-D		15.0	50.2	AN913SH	30.0	10.5	4.3	0.7	29.0	22.5
D2-E		15.0	53.5	AN913SH	50.0	10.5	4.4	0.7	7.0	48.2
D3-A	Blend 3	15.0	61.4	AN913SH	40.0	10.5	3.1	0.5	17.0	80.9
D3-B		15.0	59.7	AN913SH	40.0	10.5	4.3	0.7	15.0	56.4
D3-C		15.0	58.1	AN913SH	40.0	10.5	6.2	1.0	11.0	44.8
D3-D		15.0	58.7	AN913SH	30.0	10.5	4.3	0.7	27.0	43.6
D3-E		15.0	59.6	AN913SH	50.0	10.5	4.4	0.7	10.0	91.7

13.14.4 Rheology

Un-sheared yield stress measurements were conducted on settled solids with a Brookfield DV2T Viscometer test apparatus, using a vane spindle. A summary of results is shown in Table 13.19 with viscosity at varying underflow densities.

Table 13.19: Viscosity Yield Stress Tests

Test ID	Test	Sample	% Solids	Yield Stress (Pa)
V1-A	D1-A	Blend 1	52.1	87
V1-B	D1-B		51.0	93
V1-C	D1-C		49.6	63
V1-D	D1-D		41.7	7
V1-E	D1-E		53.0	105
V2-A	D2-A	Blend 2	56.9	108
V2-B	D2-B		54.3	59
V2-C	D2-C		53.6	49
V2-D	D2-D		50.2	23
V2-E	D2-E		53.5	48
V3-A	D3-A	Blend 3	61.4	81
V3-B	D3-B		59.7	56
V3-C	D3-C		58.1	45
V3-D	D3-D		58.7	44
V3-E	D3-E		59.6	92

13.15 Process Design Criteria

Comminution process design criteria are based on processing 100% fresh rock and material characteristics are summarized in Table 13.20.

Table 13.20: Comminution Design Parameters (Fresh Rock)

Parameter	Value	Notes
Ai	0.139	Average
CWi	22.2 kWh/tonne	Inferred calculation
BRWi	17.9 kWh/tonne	85th percentile
BBWi	16.4 kWh/tonne	85th percentile
Axb	29.4	15th percentile
Specific Gravity (sg)	2.75	Average

The selected gold leach circuit is leach-CIP carousel and process design criteria summarized in Table 13.21.

Table 13.21: Leach Design Parameters

Parameter	Value	Notes
Primary P_{80} Grind Size	75	µm
Gravity Concentration Included	Y	Y/N
Operating pH	10.5	
Slurry Density	42-45	% solids w/w
Leach Residence Time	36	hours
Oxygen Sparging	20 ppm	Cone and slam jet sparging
Pre-Leach Thickener Loading Rate	0.5 (0.6)	T/m^2/h; (0.5) Lab result and (0.6) design
Carbon Constant Average	88.6	k(hr^{-1})
	0.81	'n'
Cyanide Detoxification Parameters		
Reactor Tank Number	2	
Reactor Feed Volume	1,515	m^3/h
WAD Cyanide Level in Reactor Feed	250	ppm
Reactor pH	8.0 -8.5	
Sulphur Dioxide Ratio	4.0	g SO_2: g CN_{WAD}
Copper Sulphate Ratio	0.17	g Cu: g CN_{WAD}
Oxygenation Source	Oxygen	
Residence Time	4	2 h per tank
WAD Cyanide Level in Reactor Discharge	5	ppm

13.16 Gold Recoveries

Gold extractions from the gravity-leach tests produced the best gold extraction. Test conditions were primary grind size of P_{80} of 75 µm, pH 10.5, 1,000 ppm NaCN, oxygen sparged and 36-hour leach time. The average gold extraction results per weathering type are summarized in Table 13.22. A discount factor for solution losses was applied to establish gold recoveries.

Table 13.22: Gold Recoveries

Weathering Type	Average % Au Extraction (Lab Results)	Plant Scale Average % Au Recovery Assumptions
Saprolite	95.7%	95.0%
Transition	94.0%	93.3%
Fresh Rock OP	94.7%	94.0%
Fresh Rock UG	93.3%	92.7%

14. MINERAL RESOURCE ESTIMATES

14.1 Introduction

The following chapter presents Mineral Resource Estimates (MRE) for the Oko West deposit. The MRE was prepared by Pascal Delisle, P.Geo., Director of the Geology and Resources at GMS, and Émile Boily-Auclair, P.Eng., Project Engineer in Geology at GMIN. The MRE presented in this section, along with all steps leading to its completion, has been revised and approved by Mr. Christian Beaulieu, P.Geo., Consulting Geologist for GMS and independent Qualified Person (QP) as defined in the National Instrument 43-101.

GMS personnel and the QP visited the Oko West Project on multiple occasions, between May 2022 and March 2025, to review the geological data, drilling program and sampling protocols. Independent verification samples from drill cores were collected during the May 2022 and the January 2024 visits by GMS personnel (see Section 12).

The MRE was prepared following the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (2014), and in accordance with CIM Guidelines (2019) for Estimation of Mineral Resources and Reserves. The effective date of the mineral resource estimation is September 15, 2024, and the MRE statement is listed in Table 14.1.

Table 14.1: In-pit and Underground Updated Mineral Resource Estimate at Oko West

Category	Updated MRE Tonnage (kt)	Updated MRE Au grade (g/t)	Updated MRE Contained Gold (koz)
Pit Constrained Resource			
Indicated	73,035	2.00	4,689
Inferred	1,542	1.06	52
Underground Constrained Resource			
Indicated	**7,223**	**3.09**	**718**
Inferred	**3,585**	**2.93**	**337**
Total Open Pit and Underground			
Indicated	**80,259**	**2.10**	**5,407**
Inferred	**5,127**	**2.36**	**390**

**Notes on Mineral Resources:*
The Mineral Resources described above have been prepared in accordance with the CIM Standards (Canadian Institute of Mining, Metallurgy and Petroleum, 2014) and follow Best Practices outlined by the CIM (2019).

1. *The qualified person (QP) for this Mineral Resource Estimate (MRE) is Christian Beaulieu, P.Geo., Consulting Geologist for G Mining Services Inc.*
2. *The effective date of the Mineral Resource Estimate is September 15, 2024.*
3. *The lower cut-offs used to report open pit Mineral Resources is 0.30 g/t Au in saprolite and alluvium / colluvium, 0.34 g/t Au in transition, and 0.38 g/t Au in fresh rock.*
4. *Underground Mineral Resources are reported inside potentially mineable volume and include below cut-off material (stope optimization cut-off grade of 1.35 g/t Au).*
5. *The Oko West Deposit has been classified as Indicated and Inferred Mineral Resources according to drill spacing. No Measured Mineral Resource has been estimated. Classification of underground material was attributed on a stope-by-stope basis.*
6. *The density has been applied based on measurements taken on drill core and assigned in the block model by weathering type and lithology.*
7. *A minimum thickness of 3 metres and minimum grade of 0.30 g/t Au was used to guide the interpretation of the mineralized zones.*
8. *This MRE is based on a subblock model with a main block size of 5 m x 5 m x 5 m, with subblocks of 2.5 m x 1.25 m x 2.5 m, and has been reported inside an optimized pit shell and optimized stope shapes. Gold grades in fresh rock, transition and saprolite were interpolated with 1 m composites using Ordinary Kriging for all major domains and Inverse Distance Square for minor domains (AU_2A, AU_2B, Au_3B, AU_5 and LDZ_B6). Capping was applied on nine domains, ranging from 5 g/t Au to 45 g/t Au.*
9. *Open pit optimization parameters and cut-off grades assumptions are as follows:*
   a. *Gold price of US$1,950/oz.*
   b. *Total ore-based costs of US$16.43/t for saprolite and alluvium / colluvium, with a 94.5% processing recovery US$18.31/t for transition with a 93.3% processing recovery and US$20.56/t for fresh rock based on 93.9% processing recovery.*
   c. *Inter-ramp angles of 30° in saprolite and alluvium / colluvium, 40° in transition and 50° in fresh rock.*
   d. *Royalty rate of 8%.*
10. *UG optimization parameters and cut-off grades assumptions are as follows:*
    a. *Gold price of US$1,950/oz.*
    b. *Total ore-based costs of US$75.16/t for fresh rock.*
    c. *Stope height of 30 m, strike length of 20 m, maximum width of 30 m and minimum width of 2 m.*
    d. *The Deswik.SO (DSO) was used to constrain the Resources.*
    e. *Royalty rate of 3%.*
11. *Tonnage has been expressed in the metric system, and gold metal content has been expressed in troy ounces. The tonnages have been rounded to the nearest 1,000 tons, and the metal content has been rounded to the nearest 1,000 ounces. Totals may not add up due to rounding errors.*
12. *These Mineral Resources assume no mining dilution and losses, however must-take material is accounted for in underground stopes.*
13. *These Mineral Resources are not mineral reserves as they have not demonstrated economic viability. The quantity and grade of reported Inferred Mineral Resources in this news release are uncertain in nature and there has been insufficient*

exploration to define these resources as indicated or measured; however, it is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

The total pit constrained Indicated Mineral Resource is reported at 73,035 kt grading 2.00 g/t Au, for a total of 4,689 koz of gold. The total pit constrained Inferred Mineral Resource is reported at 1,542 kt grading 1.06 g/t Au, for a total of 52 koz of gold. The underground resources are estimated from zones outside the constrained resources of the open pit. The total constrained underground Indicated Mineral Resource is reported at 7,223 kt grading 3.09 g/t Au, for a total of 718 koz of gold. The total constrained underground Inferred Mineral Resource is reported at 3,585 kt grading 2.93 g/t Au, for a total of 337 koz of gold. Mineral resources are not mineral reserves and have not demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserves. The QP has determined that there are no known factors or issues that could significantly impact the Mineral Resource Estimate (MRE), other than the typical risks associated with mining projects, such as environmental, permitting, taxation, socio-economic, marketing, and political factors, as well as additional risk factors related to indicated and inferred mineral resources Additionally, insufficient drilling at depth, below the southern part of the pit and below the underground stopes, do not allow to define more mineral resources at depth.

It was determined that the database used for estimation is reliable, and that the current drilling information is of sufficient quality for interpreting the boundaries of gold mineralization with confidence. Additionally, the assay data used for the mineral resource estimation and block modelling is considered reliable by the QP. The mineral resource estimation methodology and key assumptions considered for the MRE are described in the following sections.

14.2 Estimation Methodology

The mineral resources presented in this report have been estimated through interpolation into a sub-block model using the modelled mineralized zones of the deposit.

The estimation methodology is summarized below:

- Drillhole database validations and selection of the drillholes to be included in the mineral resource estimation.
- 3D modelling of host units (lithological model) based on available geological data (drill logs, surface geophysics, surface plan maps, drill core photography, etc.) using Leapfrog Geo™ 2024.1.
- 3D modelling of gold-bearing domains based on geology model, strain, alteration (type and intensity), Sulphide content, assay results and drill core photography using Leapfrog Geo™ 2024.1.

- Geostatistical analysis for data conditioning: mineralization domain validation, density assignment, capping assumptions, compositing and variography using Leapfrog Geo™ 2024.1 and Supervisor™ v. 8.15.1.
- Block modelling and grade estimation using Leapfrog Edge™ 2024.1.
- Resource classification and grade interpolation validations.
- Grade and tonnage sensitivities to different cut-off grade scenarios.

14.3 Resource Database

On September 15, 2024, GMS received from the Company an extract of the database in the form of a series of comma-separated spreadsheets containing information on the Oko West deposit. The database includes information such as collar locations, drillhole types, downhole surveys, assay results, drill logs, density measurements and various geological interpretations.

Drillhole collar elevations were verified using a 50-cm resolution digital elevation model (DEM) provided to GMS. One (1) drillhole collar (OKWR22-289) was pressed to the DEM after differences of more than 2 m were observed between the collar elevation and the DEM. All other drillhole collar elevations were consistent with the DEM and were left unchanged. Drillhole surveys were verified for inconsistencies using Leapfrog Geo™ 3D viewer and a GMS validation script. Variations in dip and direction of more than 5 degrees per 100 m were flagged and investigated for consistency. Following the investigation, only two (2) surveys (DDH OKWD22-134 at 42.63 m downhole and DDH OKWD22-174 at 119.00 m) were discarded from the resource database.

Diamond drillhole (DDH) twin holes were completed to validate the inclusion of Reverse Circulation (RC) drillholes in the MRE. After review, the QP is confident that the RC drilling is a good representation of gold grades. Only one (1) RC drillhole (OKWR22-128) was discarded from the resource database after potential smearing of gold grades was observed from the adjacent DDH twin hole (OKWD22-127). Finally, one (1) trench was taken out of the database because it was completely sampled in unconsolidated alluvial material (OKWT20-006).

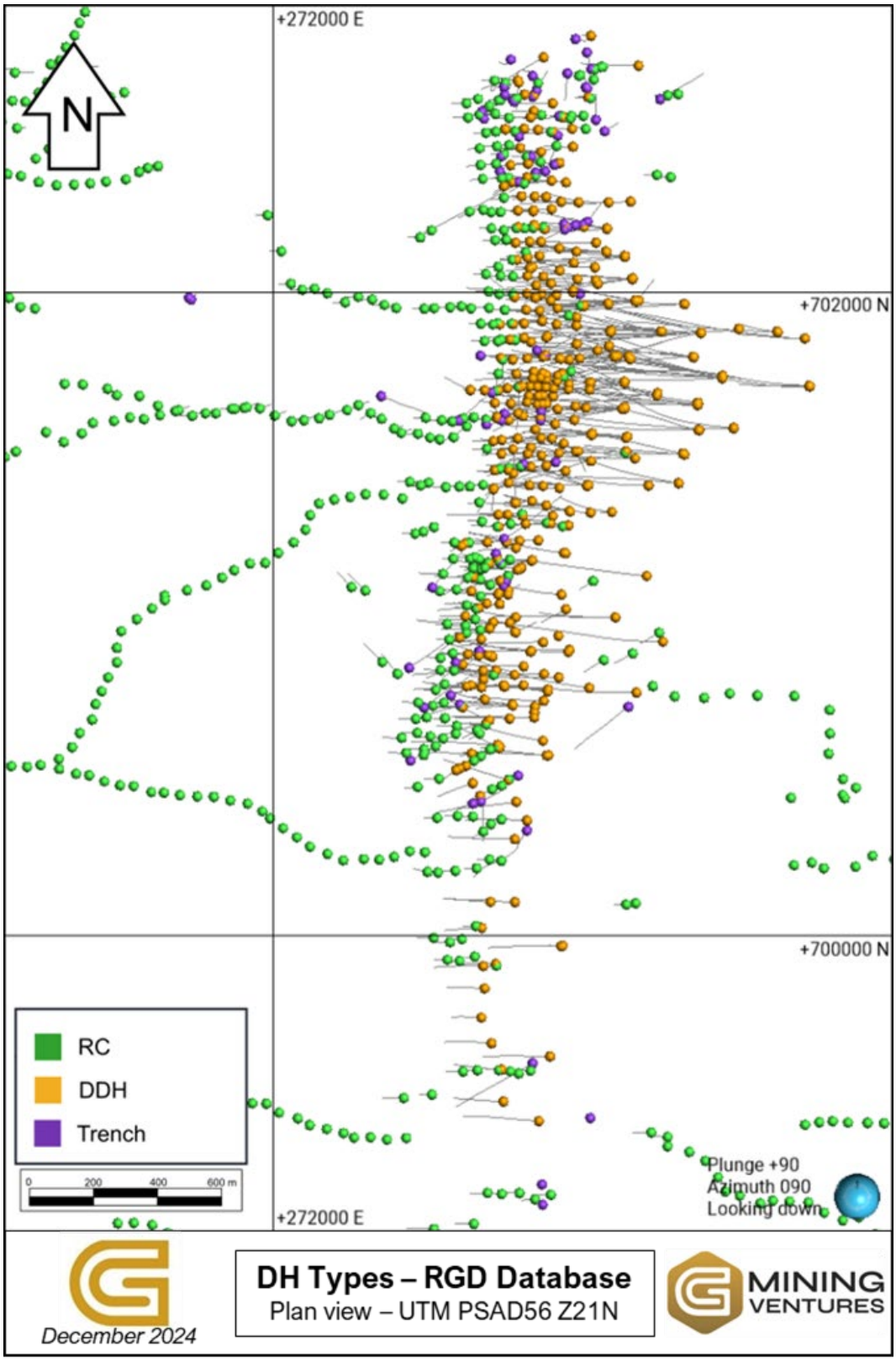
Some drillholes from previous drill campaigns on the Oko West Project have been drilled downdip of the mineralized shear zones resulting in local clusters of assays not reflecting the current drill spacing of the Project. This inconsistency in distribution is affecting the accuracy of the resource estimation. To address this issue, drillholes from the original database were excluded from the resource estimate if the assayed intervals of mineralization were more than three (3) times the average length of the mineralized domains. This step ensures that the resource estimation is based on data that better reflects the current drill spacing

and orientation of mineralized lenses of the Project. Table 14.2 summarizes the original drillhole database received from the company and the filtered one used for the resource estimation of the Oko West deposit. Figure 14.2 presents a plan view of the Oko West drillhole collars used in the estimate.

Table 14.2: Summary of Drillholes and Assays Used in the Oko West Resource Estimate

Bore Hole Types	Original Database				Resource Estimate Database			
	Total Number of Drillholes	Total Drilled Length (m)	Total Assayed Length (m)	Total Number of Assays (#)	Total Number of Drillholes	Total Drilled Length (m)	Total Assayed Length (m)	Total Number of Assays (#)
Trenches	111	11,259	10,159	5,258	59	8,166	8,080	4,212
RC	1,775	53,241	45,677	42,671	366	24,730.5	24,718.5	24,277
DD	518	157,134	101,964	83,364	505	155,689	98,895	81,174
Wedges (DD)	39	14,654	7,846	7,153	39	13,534	7,782	7,098

Figure 14.1: Drillhole Database, Coloured by Drillhole Type



Source: GMS, 2024.

14.4 Geological Models

14.4.1 Lithology and Weathering Models

A 3D geological model of the Oko West deposit was developed by GMS, incorporating drill hole logs, core photos, geochemical datasets, and prior interpretations provided by GMIN. This updated model aims to enhance the characterization of the various lithologies present in the Oko West deposit. The modelled units are presented below, from the footwall (West) to the hanging wall (East) of the mineralized domains:

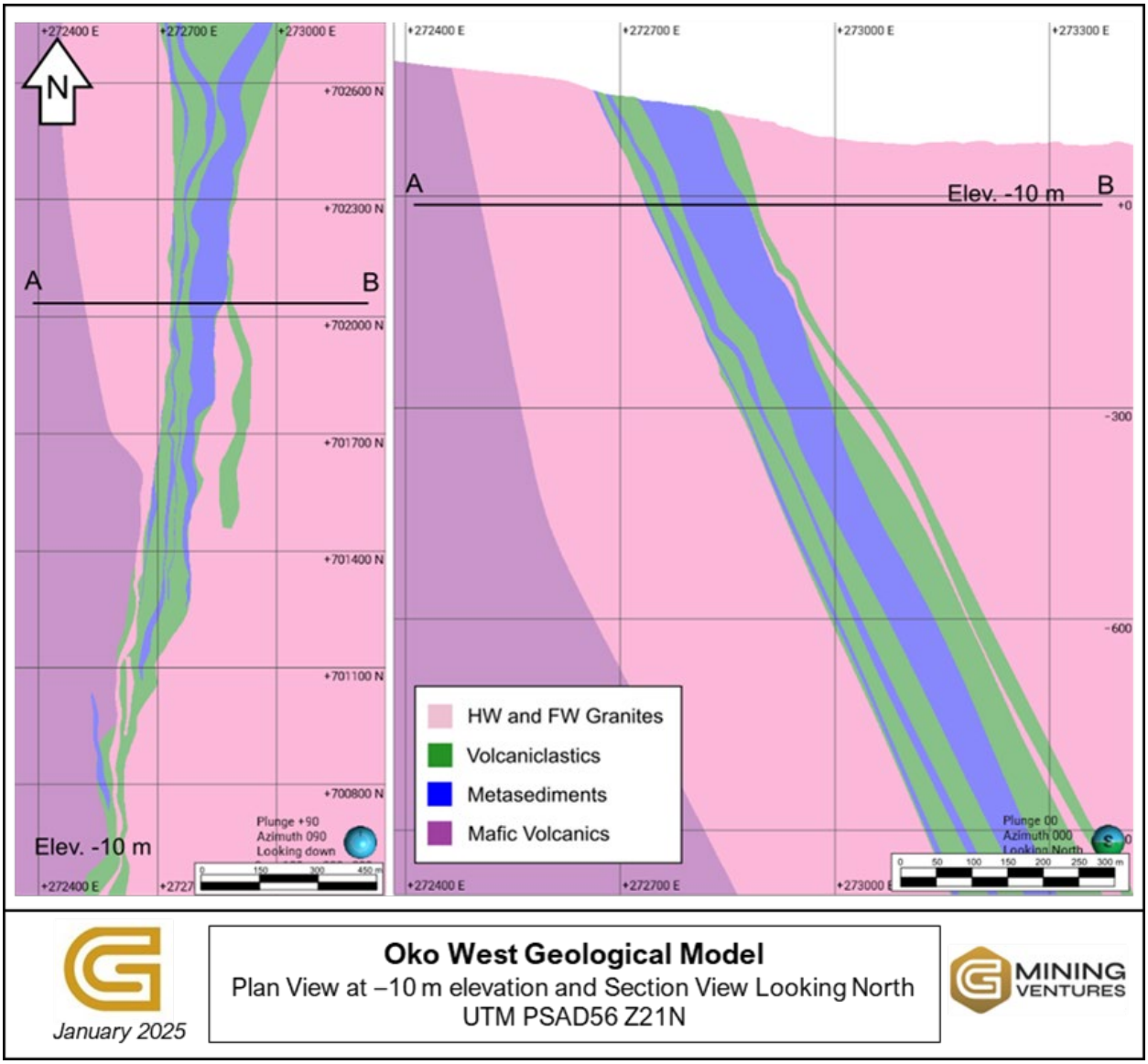
1. Mafic volcanics (MBAS).
2. Footwall granite (GRAFW).
3. Volcanoclastic units (VCLX).
4. Sedimentary package, including carbonaceous sediments (MSED).
5. Hanging wall granite (GRAHW).

A weathering model was created by GMS using simplified drillhole logs. An alluvium / colluvium model was integrated to the weathering model and was created using lithology and alteration logs. Manual selections and editing were completed to smooth surfaces and avoid over-estimation of the alluvium model. The weathering model units are presented below:

1. Alluvium / colluvium (OVBN).
2. Saprolite (SAP).
3. Transition material (TRANS).
4. Fresh rock (FRSH).

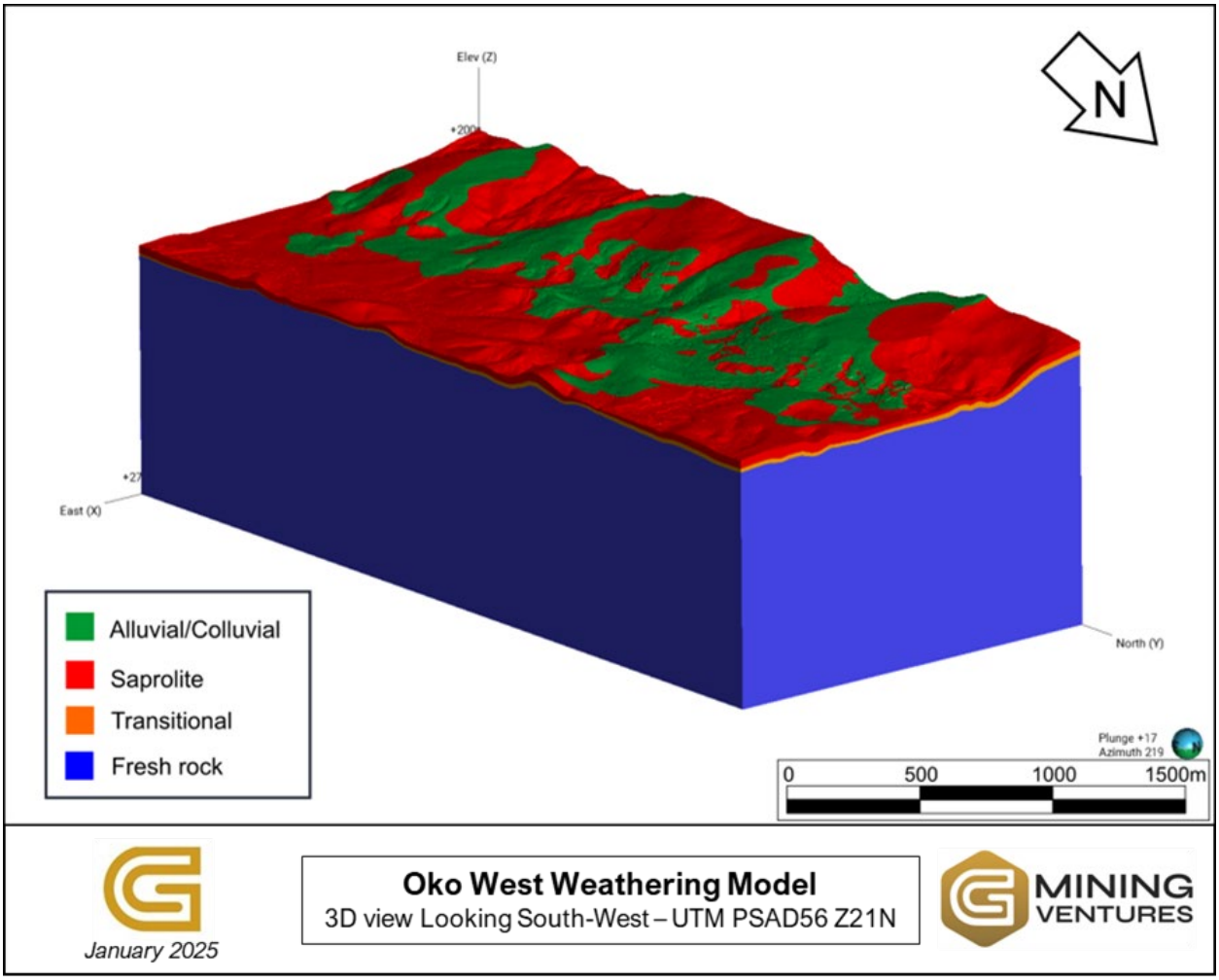
Both models are created in Leapfrog Geo™ and are used to assign density in the block model. The geological model was also used to guide the mineralization domain interpretation. Figure 14.2 and Figure 14.3 presents the geological and weathering models used for this resource estimate.

Figure 14.2: Geological Model Plan View and Vertical Section



Source: GMS, 2025.

Figure 14.3: Weathering Model Isometric View



Source: GMS, 2025.

14.4.2 Mineralization Model

The drillhole assay intervals were used to model ten (10) distinct gold-bearing mineralized domains using Leapfrog Geo™ interval selection method. North- and South-verging holes were drilled to confirm the existence or the exact location of the interpreted east-southeast subvertical fault. Since no faulting evidence has been found, the fault was removed from the mineralization and geological models. The change in mineralization continuity could be explained by the proposed polyphase deformation of the Oko West deposit and its influence on high grade ore shoots along reactivated S1/S0-parallel structures during D2 (Hainque and al., 2025). Figure 14.4 and Figure 14.5 show examples of the mineralization domains in plan view and vertical sections. The mineralization model was built based on several parameters or inputs, such as lithologies (geological model), shear and brecciation intensity, alteration types and intensity, strain, sulphide content, vein types, vein density and gold content to properly assign mineralized intervals to their respective domains to preserve domain stationarity. Most of the main zones are running parallel to the

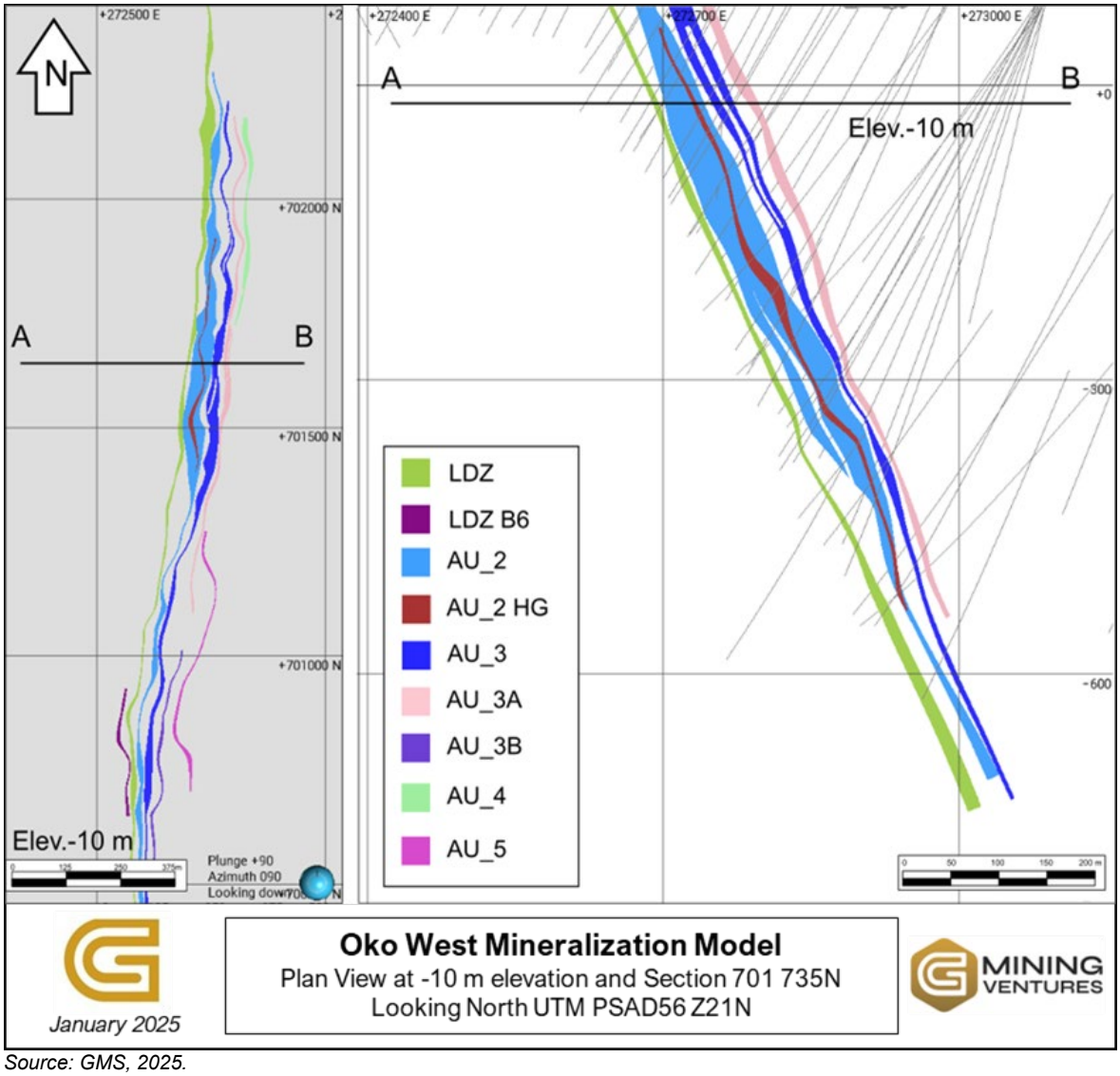
footwall granite. A cut-off of 0.30 g/t Au and a minimum true thickness of three (3) metres were used to constrain the mineralized domains. Within the main zone (AU_2), a high-grade sub-domain was modelled using a cut-off of 5.00 g/t Au, resulting in 146 continuous drill intercepts averaging 8.83 g/t Au over an estimated true thickness of 5.85 m. This high-grade ore domain was modelled using grade, strain intensity, sulphide content and lithological domain (Figure 14.5). Additionally, a total of seven (7) internal waste sub-domains were modelled where continuity of low-grade / unmineralized intervals could be demonstrated on more than one (1) section. Sub-domains were created to reduce smearing between grade populations where clear boundaries could be interpreted.

The models were improved by using manual editing such as insertions of points, polylines and structural data. Figure 14.4 presents a plan and cross-section view of the major mineralized domains. The modelled domains are listed below, from footwall to hanging wall:

- Lower Deformation Zone (LDZ).
- Lower Deformation Zone Block 6 (LDZ_B6).
- AU_2:
  - AU_2_HG.
- AU_2A.
- AU_2B.
- AU_3.
- AU_3A.
- AU_3B.
- AU_4.
- AU_5.

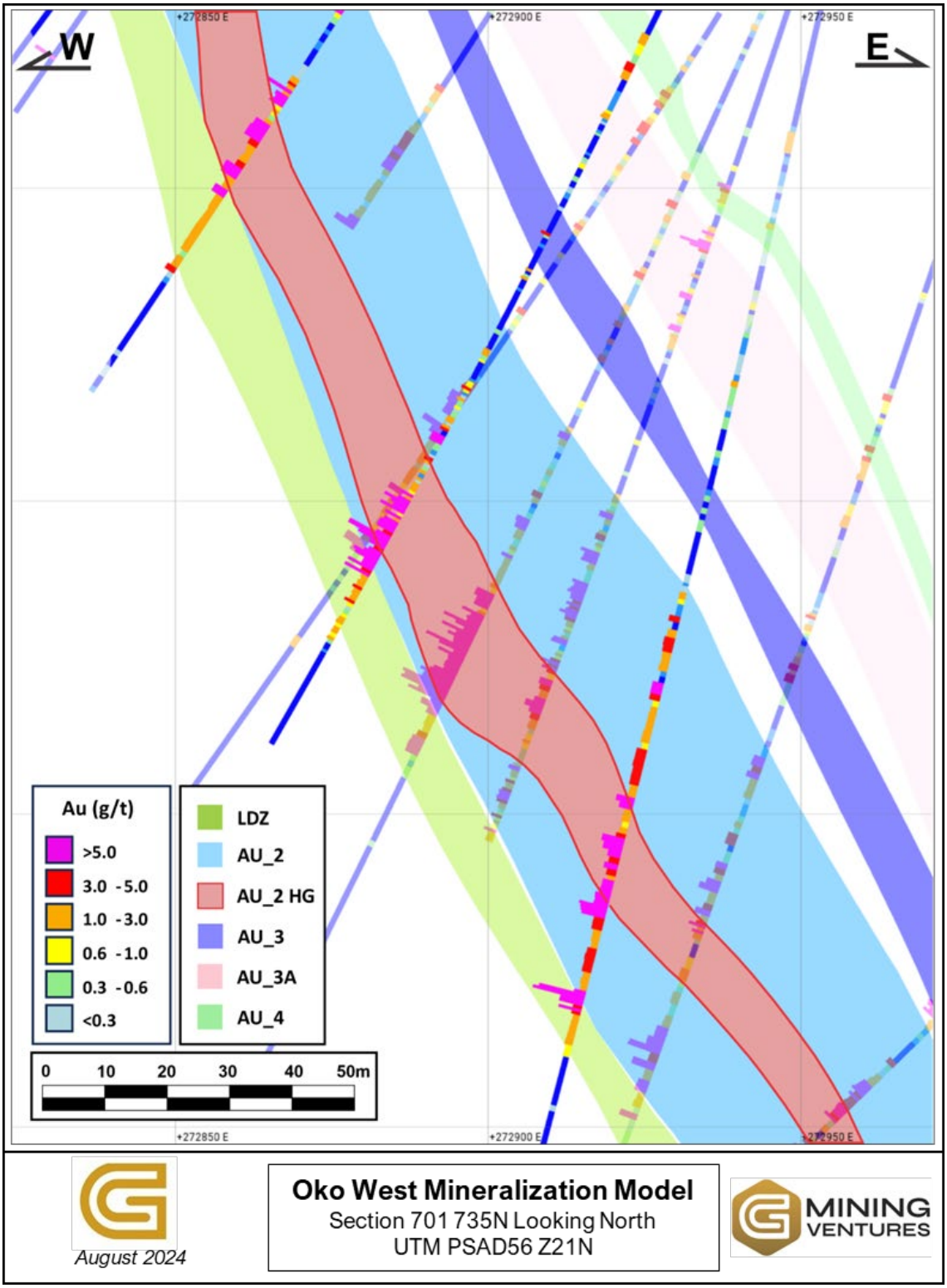
The main mineralized domains (*i.e.*, LDZ, AU_2, AU_3, AU_3A) span over 3,000 m in strike-length at surface and extend to depths of approximately 1,000 m. All mineralized zones are characterized by a consistent N5° orientation and a general 60° dip to the east.

Figure 14.4: Oko West Mineralization Model



Source: GMS, 2025.

Figure 14.5: Oko West Mineralization Model - Zoom In



Source: GMS, 2024.

14.5 Assays, Capping and Compositing

14.5.1 Raw Assays

Statistics of the Oko West deposit assays are presented below. The assay values reported below detection limits were assigned half the detection limit for statistical analysis and grade estimation purposes. Missing analysis within the database was assigned a value of 0.00 g/t Au. Table 14.3 presents the descriptive statistics of gold assays used for the resource estimations of the Oko West deposit.

Table 14.3: Oko West Gold Assays Statistics (Length Weighted)

Oko West Assays	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	Standard Deviation	CV	Median (g/t Au)	Length (m)
LDZ	4,478	0.00	50.76	1.63	3.07	1.89	0.68	4,708
LDZ_B6	221	0.00	11.58	0.87	1.79	2.05	0.24	233
AU_2	8,904	0.00	157.29	1.85	4.18	2.27	0.83	9,046
AU_2 HG	1,032	0.05	84.67	8.84	8.97	1.01	6.45	886
AU_2A	48	0.00	42.57	1.27	4.98	3.92	0.37	52
AU_2B	106	0.00	4.91	0.74	0.91	1.23	0.42	121
AU_3	3,605	0.00	1,106.05	2.09	15.59	7.48	0.61	3,819
AU_3A	2,461	0.00	65.84	1.46	3.17	2.18	0.54	2,473
AU_3B	337	0.00	12.72	0.69	1.13	1.63	0.36	423
AU_4	1,120	0.00	129.09	1.52	5.70	3.74	0.54	1,091
AU_5	278	0.00	12.01	0.62	0.96	1.55	0.37	354
Total	**22,590**	**0.00**	**1,106.05**	**2.00**	**7.51**	**3.77**	**0.75**	**23,209**

14.5.2 Capping

Capping is a technique used to mitigate the impact of outliers, specifically extremely high-grade values, on the estimation of mineral resources. It involves establishing a threshold or limit on the maximum value that can be utilized in the estimation process.

Capping analysis was conducted independently for each mineralized domain, examining assay statistics (including, but not limited to, coefficient of variation), histograms, cumulative probability plots, and conducting decile analyses to identify domains for potential grade capping. Additionally, the spatial

distribution of outliers was assessed in three dimensions (3D) to identify any high-grade clusters or localized high-grade areas within the mineralized domains that would warrant higher capping.

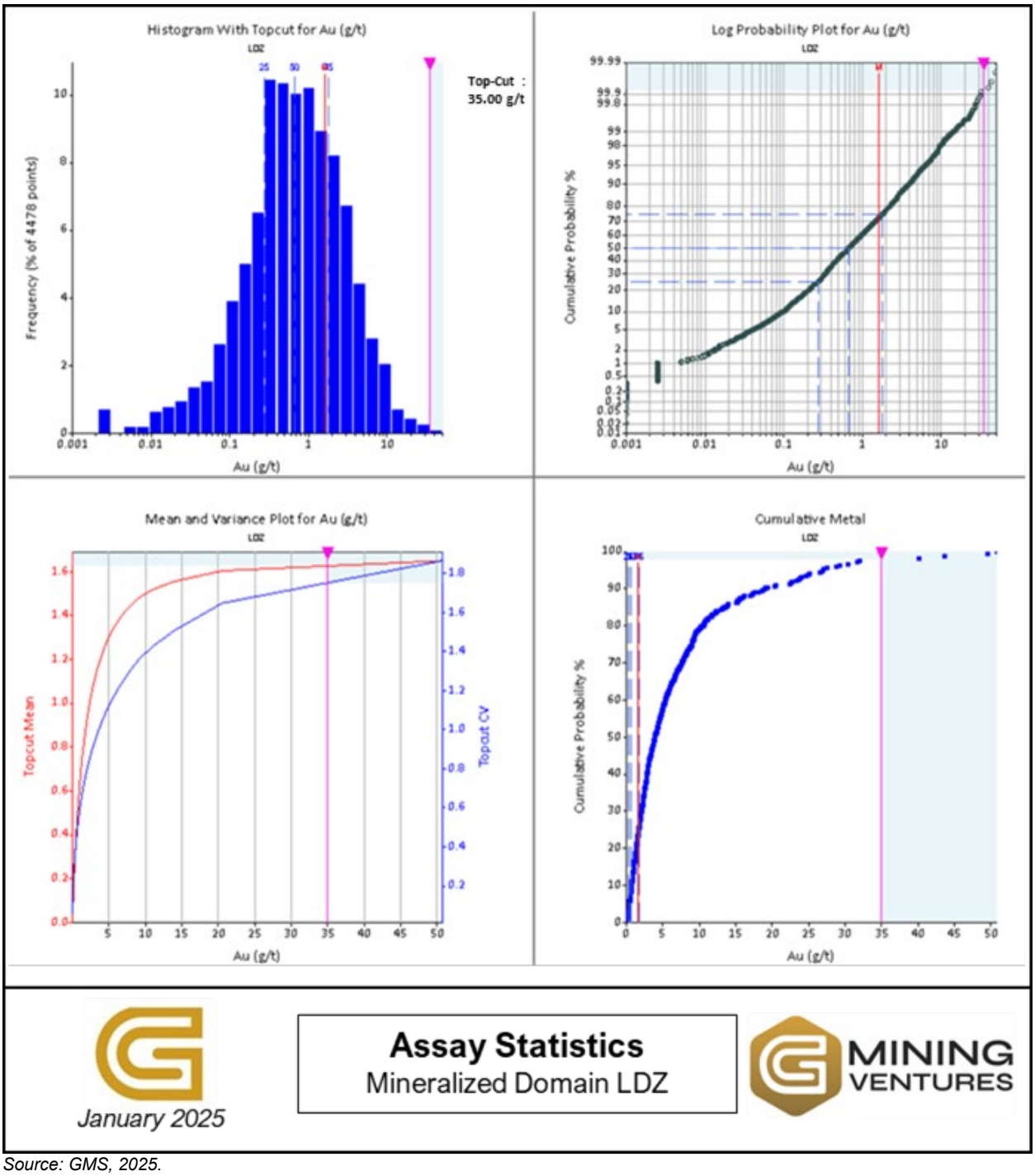
Capping assumptions for Oko West mineralized domains are presented in Table 14.4.

Table 14.4: Capping Applied to Oko West Mineralized Domains

Oko West Domains	Capping (Au g/t)
LDZ	35.0
LDZ Block 6	8.00
AU_2	45.0
AU_2 HG	No capping
AU_2A	5.00
AU_2B	No capping
AU_3	40.0
AU_3A	35.0
AU_3B	6.00
AU_4	20.0
AU_5	7.50

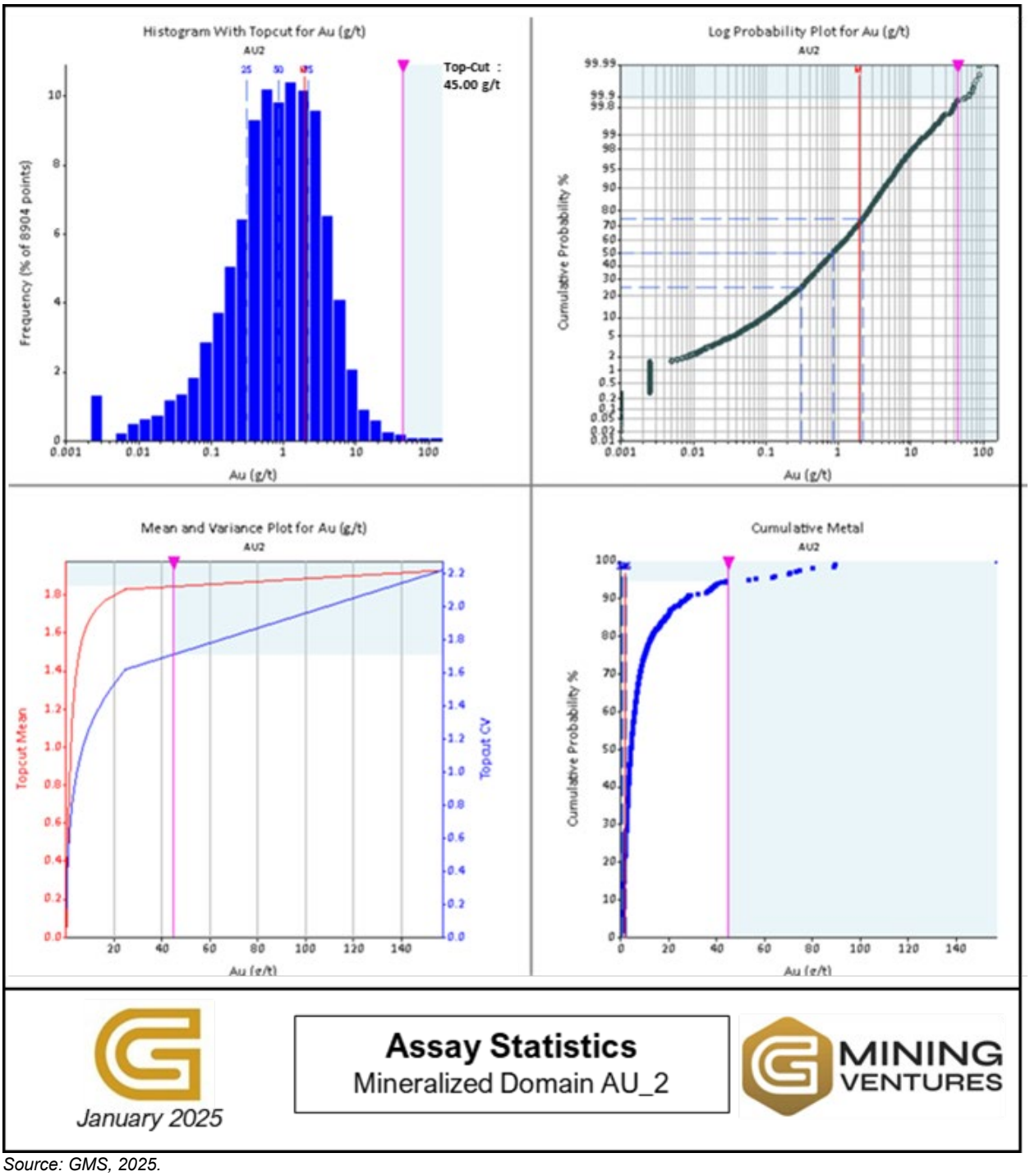
Figure 14.6 to Figure 14.8 present histograms, log probability plots, mean and variance plots, as well as cumulative metal plots for gold within the Oko West LDZ, AU_2 and AU_2HG mineralized domains, respectively. Table 14.5 compares statistics for uncapped and capped assays of the Oko West deposit, per domain. Domain AU_3 is highly influenced by only a few anomalous very high-grade samples.

Figure 14.6: Histograms, Log Probability Plots, Mean and Variance Plots, and Cumulative Metal Plots for Gold Within the LDZ Mineralized Domain



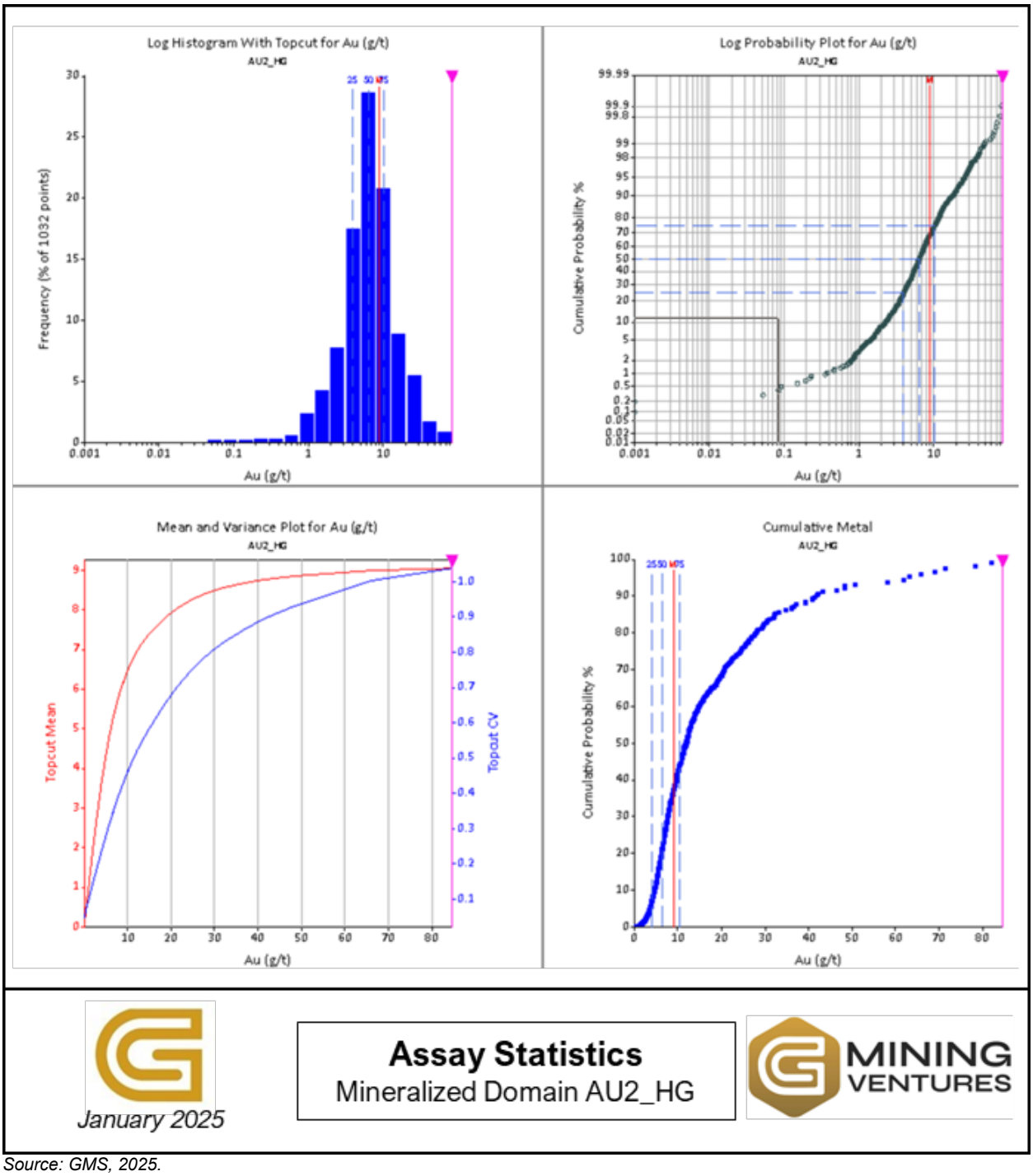
Source: GMS, 2025.

Figure 14.7: Histograms, Log Probability Plots, Mean and Variance Plots, and Cumulative Metal Plots for Gold Within the AU_2 Mineralized Domain



Source: GMS, 2025.

Figure 14.8: Histograms, Log Probability Plots, Mean and Variance Plots, and Cumulative Metal Plots for Gold Within the AU_2HG Mineralized Domain



Source: GMS, 2025.

Table 14.5: Oko West Uncapped & Capped Assays Statistics, per Domain (length weighted)

Domain	Num. of Assays	Au Uncapped (g/t Au)			Num. of Assays Capped	Au Capped (g/t Au)			Metal Cut (%)
		Max	Mean	CV		Max	Mean	CV	
LDZ	4,478	50.76	1.63	1.89	4	35	1.62	1.82	0.6
LDZ_B6	221	11.58	0.87	2.05	5	8.00	0.84	1.93	3.9
AU_2	8,904	157.29	1.85	2.27	12	45.0	1.80	1.83	2.4
AU_2 HG	1,032	84.67	8.84	1.01	0	84.67	N/A	N/A	N/A
AU_2A	48	42.57	1.27	3.92	2	5.00	0.74	1.48	41.6
AU_2B	106	4.91	0.74	1.23	0	4.91	0.74	N/A	N/A
AU_3	3,605	1,106.05	2.09	7.48	13	40	1.73	2.18	16.8
AU_3A	2,461	65.84	1.46	2.18	4	35.0	1.44	2.05	1.0
AU_3B	337	12.72	0.69	1.63	3	6.00	0.68	1.47	2.5
AU_4	1,120	129.09	1.52	3.74	6	20.0	1.29	1.81	15.6
AU_5	278	12.01	0.62	1.55	1	7.5	0.61	1.37	2.1

14.5.3 Compositing

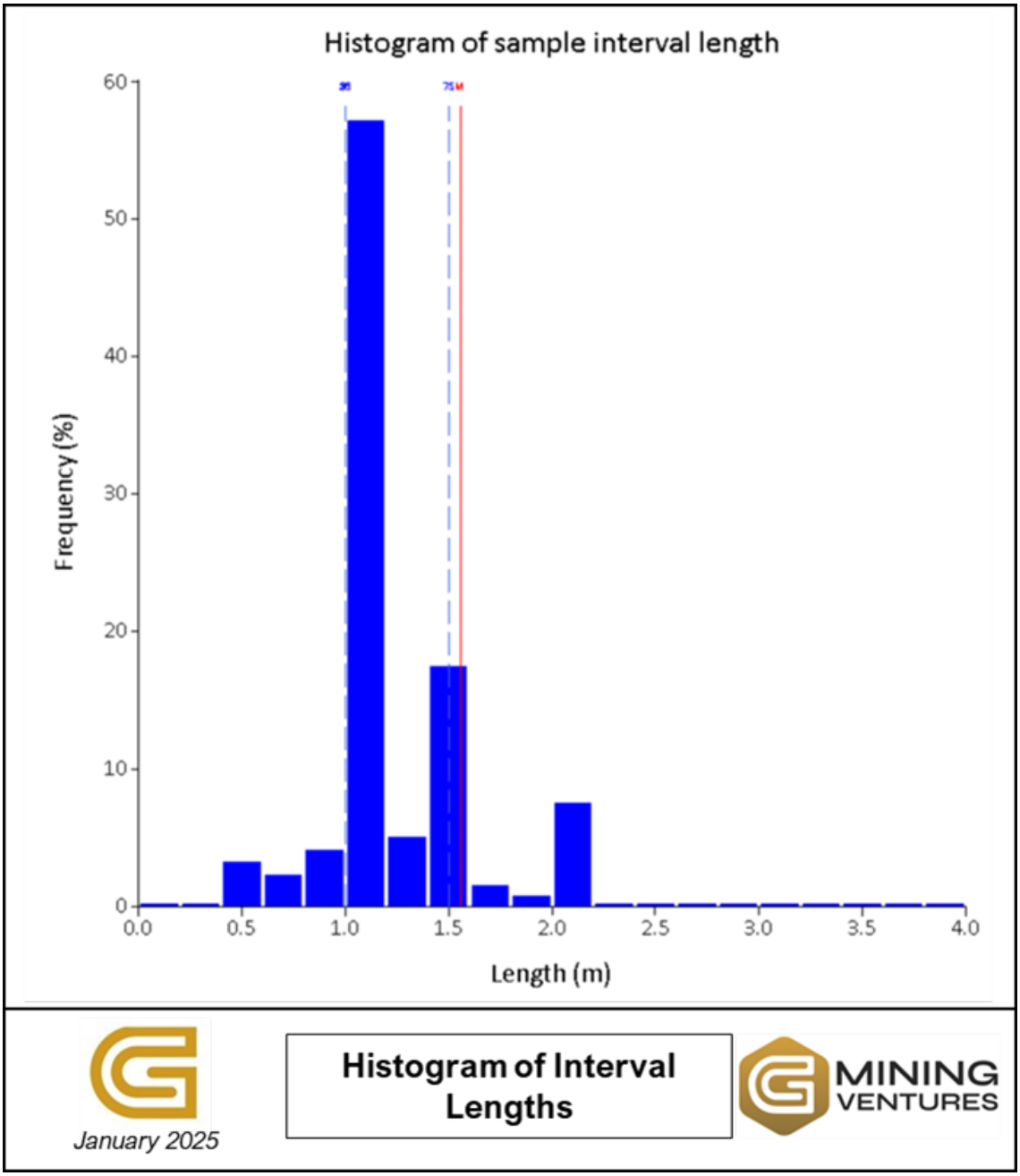
Following the application of assay capping, the samples were composited downhole within the boundaries of each mineralized domain. The length of the composites was determined through statistical analysis of the sample lengths, taking into consideration factors such as the most sampled interval length (i.e. mode), block sizes, and modelled mineralized domain sizes.

Composites of 1 m were retained, with residuals of less than 0.3 m distributed equally through the interval length. A sample coverage of each composite interval of at least 50% was needed for composites to be created. Table 14.6 presents the statistics for uncomposited and composited samples, per mineralized domain. Figure 14.9 presents the histogram of interval length on the Oko West Project, while Figure 14.10 compares values before and after compositing.

Table 14.6: Uncomposited and Composited Statistic by Mineralized Domain

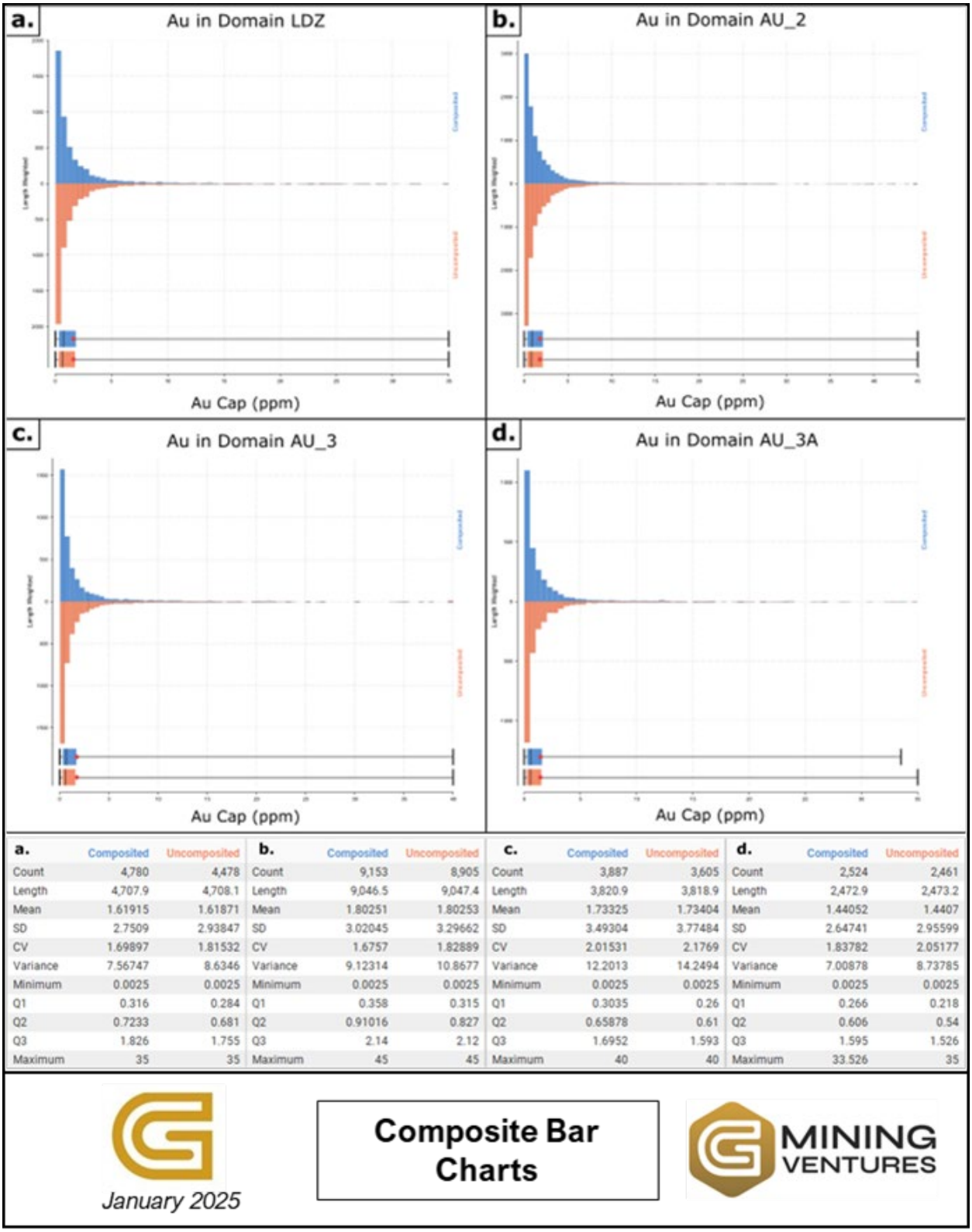
Zone	Uncomposited (Length-Weighted)				1 m Composites			
	Number Samples	Length	Average	CV	Number Samples	Length	Average	CV
LDZ	4,478	4,708	1.62	1.82	4,780	4,708	1.62	1.70
LDZ_B6	221	233	0.84	1.93	238	233	0.86	1.78
AU_2	8,904	9,046	1.80	1.83	9,153	9,046	1.80	1.68
AU_2 HG	1,032	886	8.84	1.01	909	886	8.84	0.90
AU_2A	48	52	0.74	1.48	55	53	0.72	1.33
AU_2B	106	121	0.74	1.23	123	121	0.73	1.19
AU_3	3,605	3,819	1.73	2.18	3,887	3821	1.73	2.02
AU_3A	2,461	2,473	1.44	2.05	2,524	2,473	1.43	1.83
AU_3B	337	423	0.68	1.47	435	425	0.68	1.42
AU_4	1,120	1,091	1.29	1.81	1122	1,091	1.27	1.59
AU_5	278	354	0.61	1.37	367	355	0.61	1.32

Figure 14.9: Oko West Database Histogram of Sampled Interval Lengths



Source: GMS, 2025.

Figure 14.10: Composited and Uncomposited Assays Comparative Bar Charts



a.	Composited	Uncomposited
Count	4,780	4,478
Length	4,707.9	4,708.1
Mean	1.61915	1.61871
SD	2.7509	2.93847
CV	1.69897	1.81532
Variance	7.56747	8.6346
Minimum	0.0025	0.0025
Q1	0.316	0.284
Q2	0.7233	0.681
Q3	1.826	1.755
Maximum	35	35

b.	Composited	Uncomposited
Count	9,153	8,905
Length	9,046.5	9,047.4
Mean	1.80251	1.80253
SD	3.02045	3.29662
CV	1.6757	1.82889
Variance	9.12314	10.8677
Minimum	0.0025	0.0025
Q1	0.358	0.315
Q2	0.91016	0.827
Q3	2.14	2.12
Maximum	45	45

c.	Composited	Uncomposited
Count	3,887	3,605
Length	3,820.9	3,818.9
Mean	1.73325	1.73404
SD	3.49304	3.77484
CV	2.01531	2.1769
Variance	12.2013	14.2494
Minimum	0.0025	0.0025
Q1	0.3035	0.26
Q2	0.65878	0.61
Q3	1.6952	1.593
Maximum	40	40

d.	Composited	Uncomposited
Count	2,524	2,461
Length	2,472.9	2,473.2
Mean	1.44052	1.4407
SD	2.64741	2.95599
CV	1.83782	2.05177
Variance	7.00878	8.73785
Minimum	0.0025	0.0025
Q1	0.266	0.218
Q2	0.606	0.54
Q3	1.595	1.526
Maximum	33.526	35

Source: GMS, 2025.
**Note: a. LDZ, b. AU_2, c. AU_3, d. AU_3A.*

14.6 Density Measurements

Density measurements were collected systematically in DDH every five (5) to ten (10) metres or at every geological domain transition (i.e., weathering and/or lithology shift) (Procedures, 2022). The weight in water, weight in air method was used by the personnel to measure the density of chosen core samples. These data were used to assign median specific gravity to each modelled geological and weathering domain. Table 14.8 presents the number of measurements and median density used in the resulting 16 density domains. Figure 14.11 presents a cross-section of the Oko West density model.

Table 14.7: Host Rocks Density Statistics by Weathering Profile

Weathering Domain	Geological Domain	Count	Mean	Minimum	Median	Maximum
Alluvium / Colluvium		126	1.90	1.49	1.85	2.94
Saprolite	Granite FW	64	1.95	1.09	1.93	2.60
	Granite HW	2,461	1.88	0.66	1.88	2.87
	Metasediment	432	1.93	0.64	1.91	3.06
	Mafic Volcanic	0	NA	NA	NA	NA
	Volcaniclastic	509	1.91	0.68	1.88	5.25
Trans	Granite FW	35	2.27	1.78	2.33	2.69
	Granite HW	903	2.25	1.26	2.23	3.91
	Metasediment	108	2.21	1.66	2.15	3.10
	Mafic Volcanic	13	2.44	2.13	2.42	2.87
	Volcaniclastic	307	2.23	1.05	2.17	3.00
Fresh Rock	Granite FW	1,102	2.71	1.33	2.75	3.88
	Granite HW	4,991	2.71	0.73	2.73	7.73
	Metasediment	1,528	2.74	1.14	2.76	5.24
	Mafic Volcanic	587	2.85	2.00	2.88	3.82
	Volcaniclastic	2,771	2.79	1.01	2.83	6.05

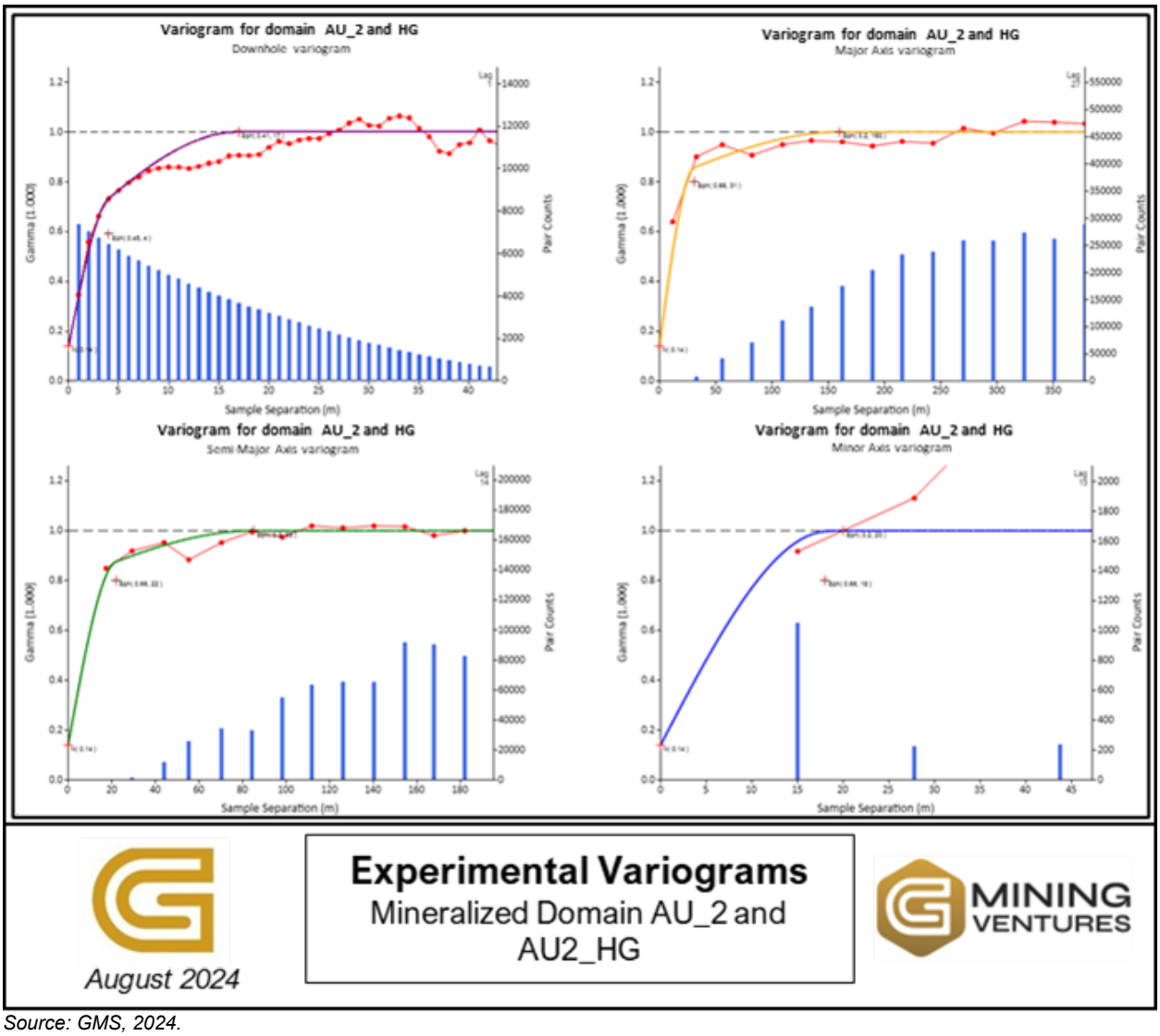
Figure 14.11: Density Model Coloured by Density Value

Source: GMS, 2024.
**Note: Weathering materials can be distinguished between saprolitic material (green), transitional material (yellow) and fresh rocks (orange and pink).*

14.7 Variography

Variography is a statistical tool used in resource estimation to evaluate the spatial distribution of grades within a mineralized domain. Experimental variograms were produced for each mineralized domain, based on the 1-m composites presented above. Some similar domains were merged to obtain more coverage and facilitate variogram interpretation (e.g. domain AU_2 and AU_2HG). Variograms for LDZ_B6, AU_2A, AU_2B, AU_3B and AU_5 could not be adequately interpreted due to the insufficient number of composites. Figure 14.12 presents variogram parameters, while Table 14.8 presents an example of variography for the Oko West AU_2 mineralized domain (combined with the high-grade domain).

Figure 14.12: Oko West Experimental Variograms for Mineralized Domain AU_2 and AU_2_HG Combined



Source: GMS, 2024.

Table 14.8: Variogram Parameters Used for the Mineral Resource Estimation per Domain

Domain	Direction			Nugget	Structure 1				Structure 2			
	Dip	Dip Azimuth	Pitch		Sill 1	Major	Semi-Major	Minor	Sill 2	Major	Semi-Major	Minor
LDZ	66	94	70	0.18	0.64	41	34	8	0.18	150	129	12
AU_2	65	96	73	0.22	0.67	31	22	18	0.11	160	85	20
AU_3	66	95	70	0.27	0.55	30	26	8	0.18	95	40	10
AU_3A	67	92	70	0.25	0.64	40	42	10	0.11	100	50	12
AU_4	66	95	70	0.22	0.54	44	27	8	0.24	70	45	10

**Note: All parameters are normalized with a total Sill of 1.0.*

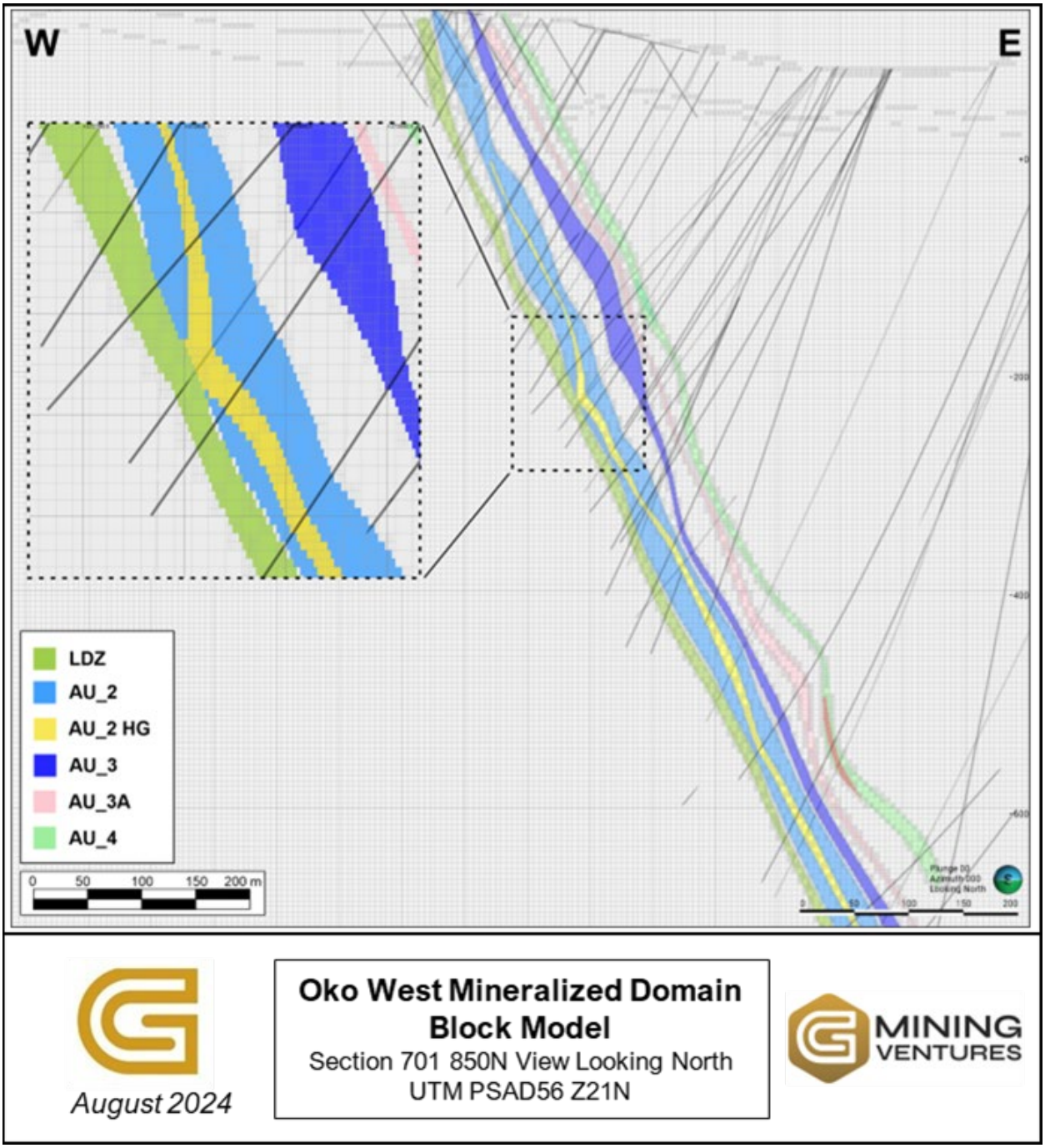
14.8 Block Modelling

The modelling of each block was carried out by adopting a parent block size of 5.0 m x 5.0 m x 5.0 m and a sub-block count of 2 x 4 x 2 for a minimum block size of 2.5 m x 1.25 m x 2.5 m. The block size was chosen based on the width of the mineralized zones, the nominal drill spacing and the anticipated open pit and underground mining methods. To validate the accuracy of the block size against the mineralization wireframe volumes, GMS compared the volumes. The blocks for each domain are good representations of their respective 3D model volumes. Table 14.9 presents block model parameters, while Figure 14.13 presents a cross-section of the block model mineralization domain trigger.

Table 14.9: Oko West Block Model Parameters

Description	Easting (m)	Northing (m)	Elevation (m)
Origin Coordinates	273,726.0	698,844.0	350.0
Parent / Sub-block Count	5/2	5/4	5/2
Minimum Block Size	2.5	1.25	2.5
Number of Blocks	809	425	269
Rotation	275°		

Figure 14.13: Mineralized Domain Block Model



Source: GMS, 2024.

14.9 Block Model Interpolation

The Ordinary Kriging (OK) interpolation method was used to interpolate block grades based on the variogram models presented in Section 14.7. Inverse Squared Distance (ID^2) interpolation method was used in areas with limited data and where robust variogram models were not achievable (see Section 14.7).

For domains interpolated using OK, a discretization of 5 x 5 x 5 in X, Y and Z was applied to the blocks. For OK and ID^2, a four-pass approach applied by domains was used, with an increasing ellipsoid size after each pass estimation. A fifth pass was added for a minority of blocks that were not evaluated within the first four (4) passes (search distance up to 160 m and minimum of 4 composites). Blocks falling outside mineralized domains but within the mineralized volcanoclastic units were interpolated using ID^2 and a four-pass approach, similar to the estimation used for this MRE. This dilution model was used by the engineering team for Whittle and stope optimizations. However, for mineral resource statement, all blocks falling outside the mineralized domains were assigned a value of 0.00 g/t Au.

For ellipsoid orientation, Leapfrog Edge's "dynamic anisotropy" (DA) was used based on the geometry of each domain. The variable ellipsoid orientation (or DA) was validated for each domain and no inconsistencies were observed. Table 14.8 to Table 14.11 present the parameters, restrictions and search criteria used for the Oko West resource estimation.

For domains LDZ_B6 and AU_3B, outlier restrictions were applied to the fourth pass with a 60 m threshold above which composite grades higher than 5 g/t Au were clamped to prevent high-grade smearing.

Table 14.10: Search Ellipsoids by Mineralized Domains and Search Passes

Domain	Interpolation Method	Ellipsoid Ranges (m)											
		Pass 1			Pass 2			Pass 3			Pass 4		
		Max	Int	Min	Max	Int	Min	Max	Int	Min	Max	Int	Min
LDZ	OK	60	50	15	80	65	20	120	100	30	160	120	30
LDZ_B6	ID2	50	40	10	70	60	20	120	100	30	160	120	30
AU_2	OK	60	50	15	80	65	20	120	100	30	160	120	30
AU_2 HG	OK	60	50	15	80	65	20	120	100	30	160	120	30
AU_2A	ID2	60	50	15	80	65	20	120	100	30	160	120	30
AU_2B	ID2	60	50	15	80	65	20	120	100	30	160	120	30
AU_3	OK	60	50	15	80	65	20	120	100	30	160	120	30
AU_3A	OK	60	50	15	80	65	20	120	100	30	160	120	30
AU_3B	ID2	50	40	10	70	60	20	120	100	30	160	120	30
AU_4	OK	60	50	15	80	65	20	120	100	30	160	120	30
AU_5	ID2	60	50	15	80	65	20	120	100	30	160	120	30

Table 14.11: Sample Search Criteria by Passes

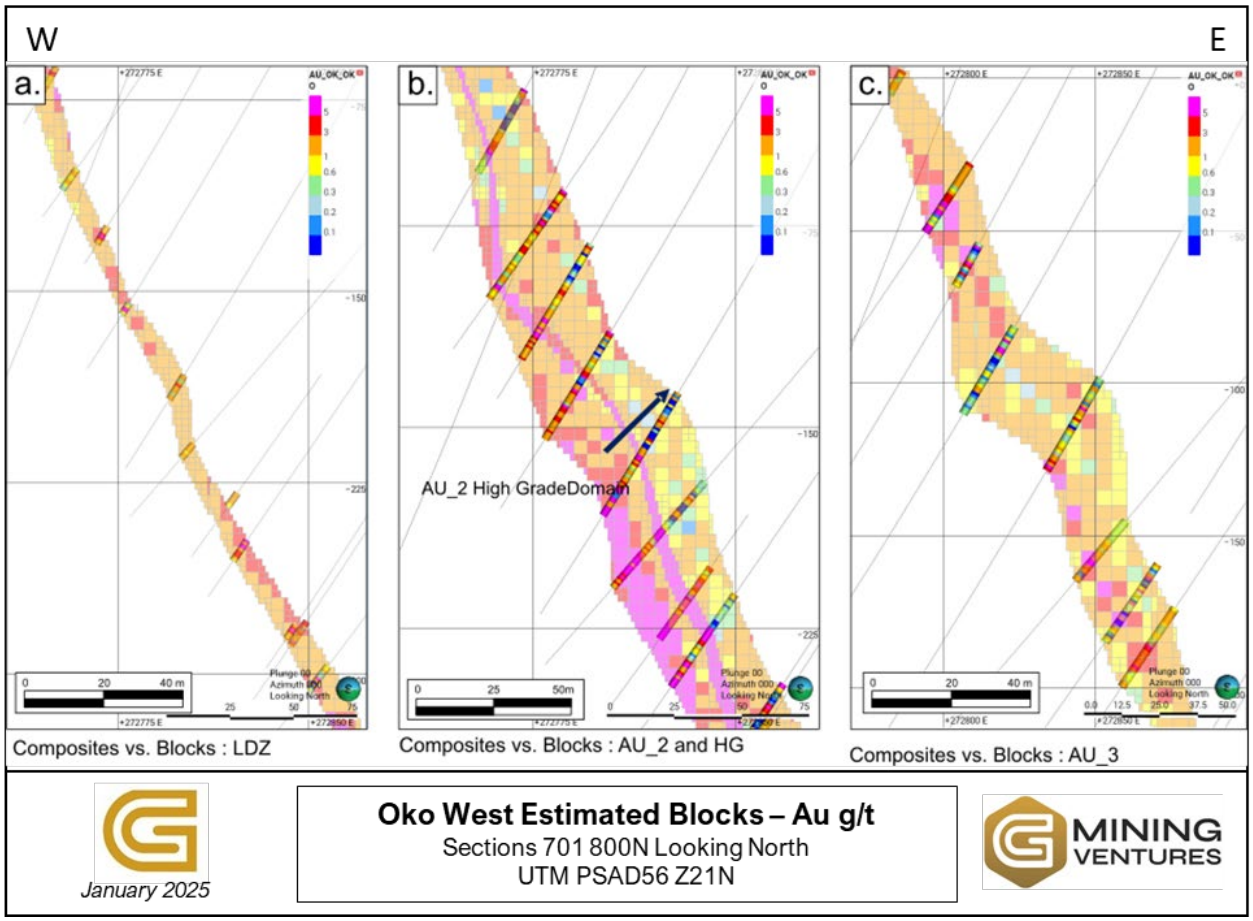
Domain	Pass	Composites			Minimum DDH
		Min.	Max	Max / DDH	
LDZ, AU_2, AU_2 HG, AU_2A, AU_2B, AU_3, AU_3A, AU_4, AU_5	Pass 1	7	12	3	3
	Pass 2	7	12	3	3
	Pass 3	7	12	3	3
	Pass 4	4	12	3	2
LDZ_B6, AU_3B	Pass 1	7	12	3	3
	Pass 2	6	12	3	2
	Pass 3	4	9	3	2
	Pass 4	1	9	3	1

14.10 Grade Estimation Validation

14.10.1 Visual Validation

A visual validation was conducted to confirm that the ellipsoid orientation matches the orientation of the modelled veins and the distribution of grades. To ensure that the estimated blocks are a robust interpretation of the composites, various validation methods were used. Visual checks of the block model, per vertical section and plan view, were used as validation of the interpolation outputs. Figure 14.14 presents global cross-sections of the interpolated block models against composites. In general, the estimated block gold grades are good representations of composites gold grades.

Figure 14.14: Oko West Block Model and Composites Visual Validation



Source: GMS, 2025.
**Note: a. LDZ, b. AU_2, c. AU_3. The High-Grade domain is constrained within AU_2.*

14.10.2 Global Statistical Validation

To ensure proper composite representation in each domain, a statistical comparison was made between the global composite mean and global interpolated block means for various interpolation methods. Table 14.12 shows a summary of global means, per domain. Based on the results, the interpolation using OK is judged to be valid and a good representation of composite grades, and no important bias is observed between the interpolation methods.

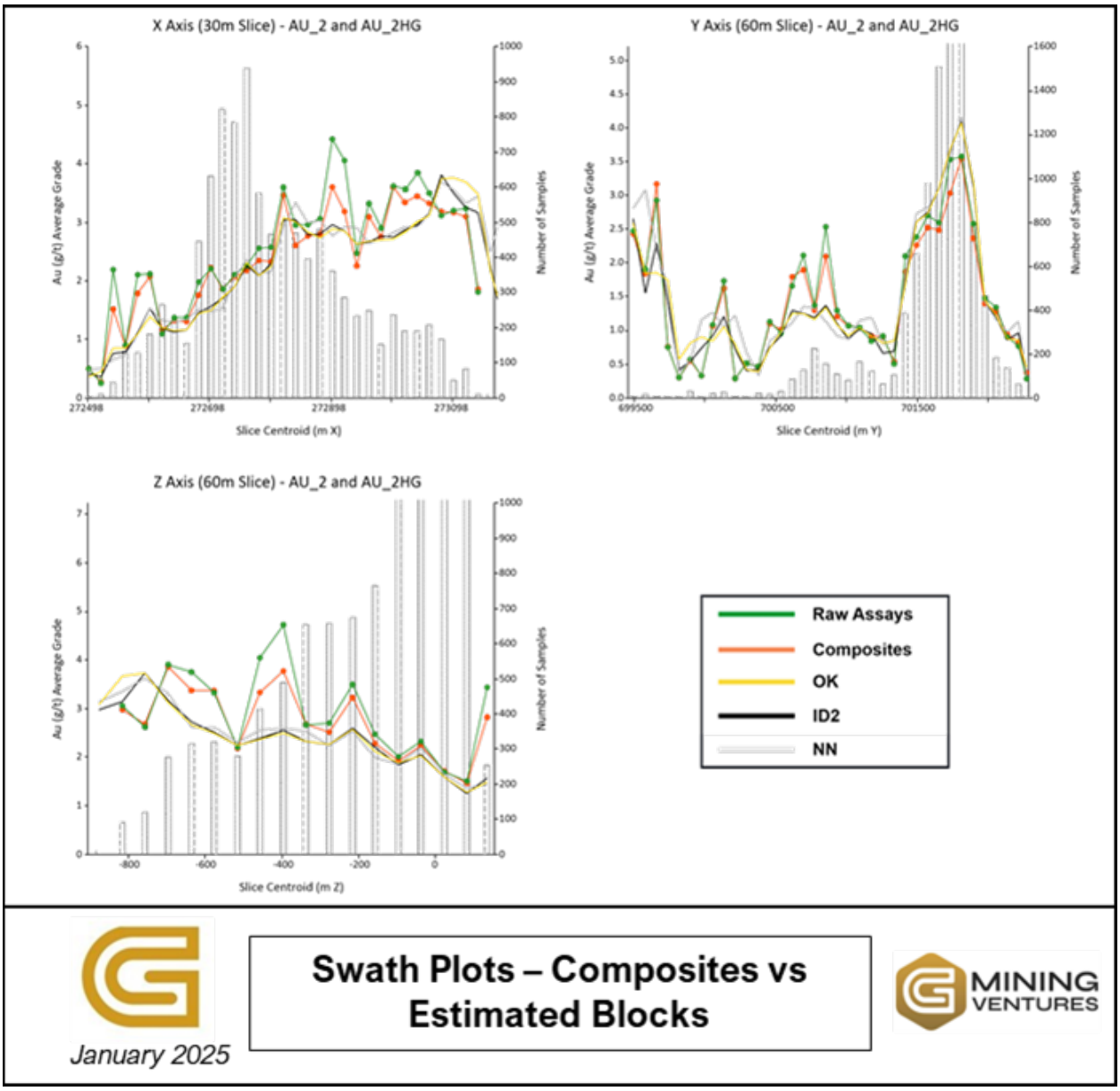
Table 14.12: Mean Grade Comparison Between Composites and Blocks, per Domain (Volume Weighted)

Oko Domains	Comp (g/t Au)	Block OK (g/t Au)	Block ID^2 (g/t Au)	Block NN (g/t Au)
LDZ	1.62	**1.49**	1.48	1.49
LDZ_B6	0.86	**0.80**	0.82	0.80
AU_2	1.80	**1.66**	1.68	1.66
AU_2 HG	8.84	**8.67**	8.61	8.81
AU_2A	0.72	0.69	**0.68**	0.69
AU_2B	0.73	0.57	**0.56**	0.61
AU_3	1.73	**1.51**	1.50	1.47
AU_3A	1.43	**1.44**	1.43	1.50
AU_3B	0.68	**0.63**	0.62	0.66
AU_4	1.27	**1.30**	1.28	1.27
AU_5	0.61	0.53	**0.52**	0.59

14.10.3 Local Statistical Validation – Swath Plot

Finally, swath plots were created to validate local estimation. The method involves comparing the predicted values of a block from the interpolation model to the actual values obtained from drillhole samples (i.e., composites). When enough samples are available for swath plot analysis, peaks and troughs in composite grades generally follow peaks and troughs in block grades. Figure 14.15 presents swath plots along the X, Y and Z-axis. In general, composite gold grades are well represented within estimated block gold grades and the smoothing inherent to OK appears reasonable.

Figure 14.15: Swath Plots for X (Along Strike), Y (Along Cross-strike) and Z for Domain AU_2 and AU_2HG Combined



Source: GMS, 2025.

14.11 Mineral Resources

14.11.1 Mineral Resources Classification

The estimated blocks were classified according to CIM's "Definition Standards for Mineral Resources and Mineral Reserves" (2014) and adhere to the CIM "Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines" (2019). The mineral resources at Oko were classified as Indicated and Inferred mineral resources.

As stated in CIM's "Definition Standards for Mineral Resources and Mineral Reserves":

"An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, density, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit."

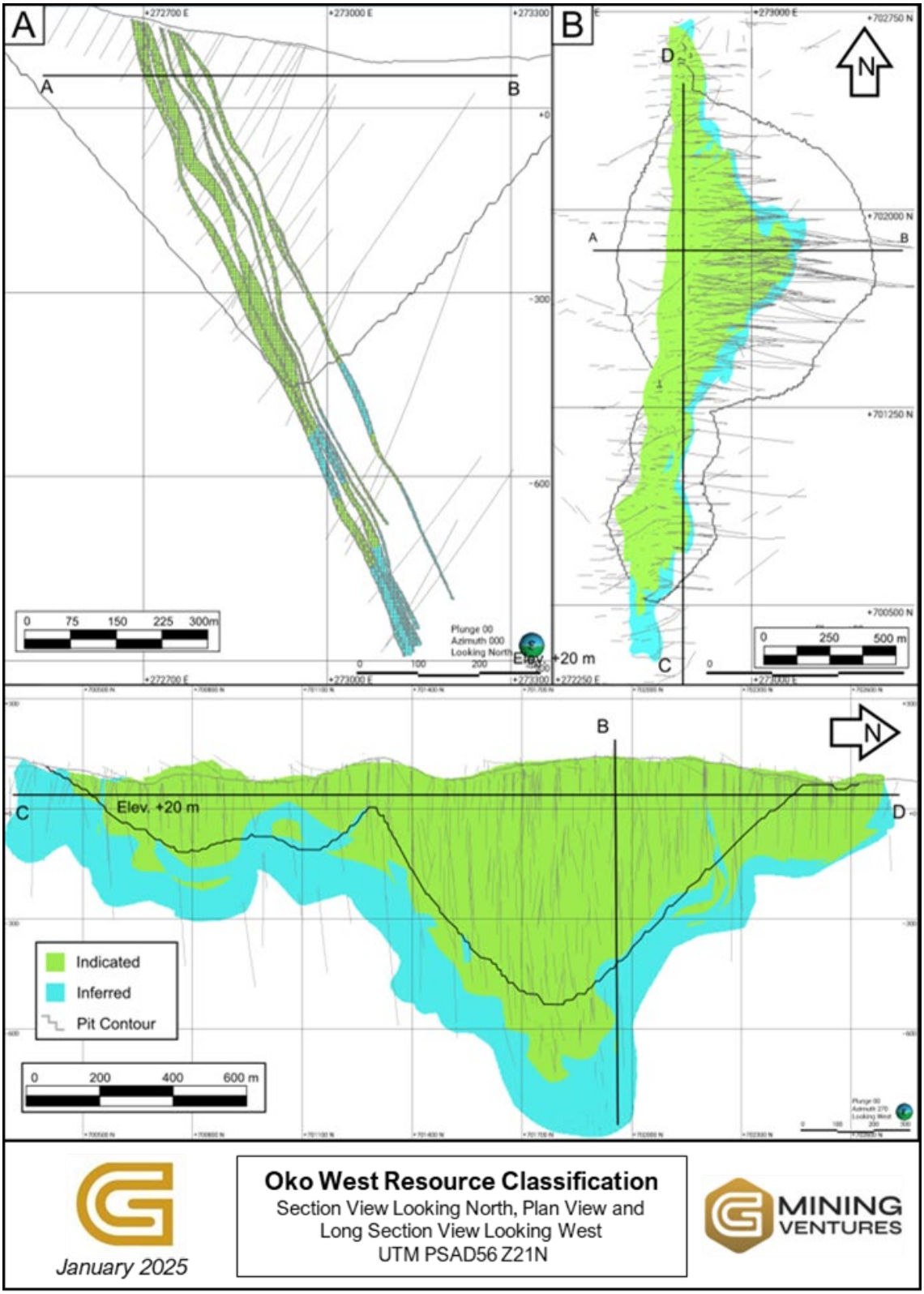
"An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity."

GMS considered variogram ranges, drillhole spacing, confidence in the geological interpretation and recovery methods to determine parameters that will define the resource categories. The final mineral resource classification is mostly based on average drillhole spacing and manual editing to avoid isolated blocks. The principal assumptions to classify the Mineral Resources as Indicated and Inferred are summarized below:

- No Measured Mineral Resources are defined at Oko West at this stage of the Project.
- Indicated Mineral Resources are defined where blocks have an average distance to the nearest three (3) drillholes of less than 45 m.
- Inferred Mineral Resources are defined where blocks have an average distance to the nearest three (3) drillholes of less than 80 m. This limit corresponds to sectors with sparse drilling and less lateral and horizontal continuity.
- Final categories of all domains were manually edited to avoid isolated clusters of blocks.
- Blocks inside each individual stope were reclassified according to the dominant class. A few stopes were downgraded from Indicated to Inferred or upgraded from Inferred to Indicated, based on the surrounding information (drilling, block data, confidence in the geological model).

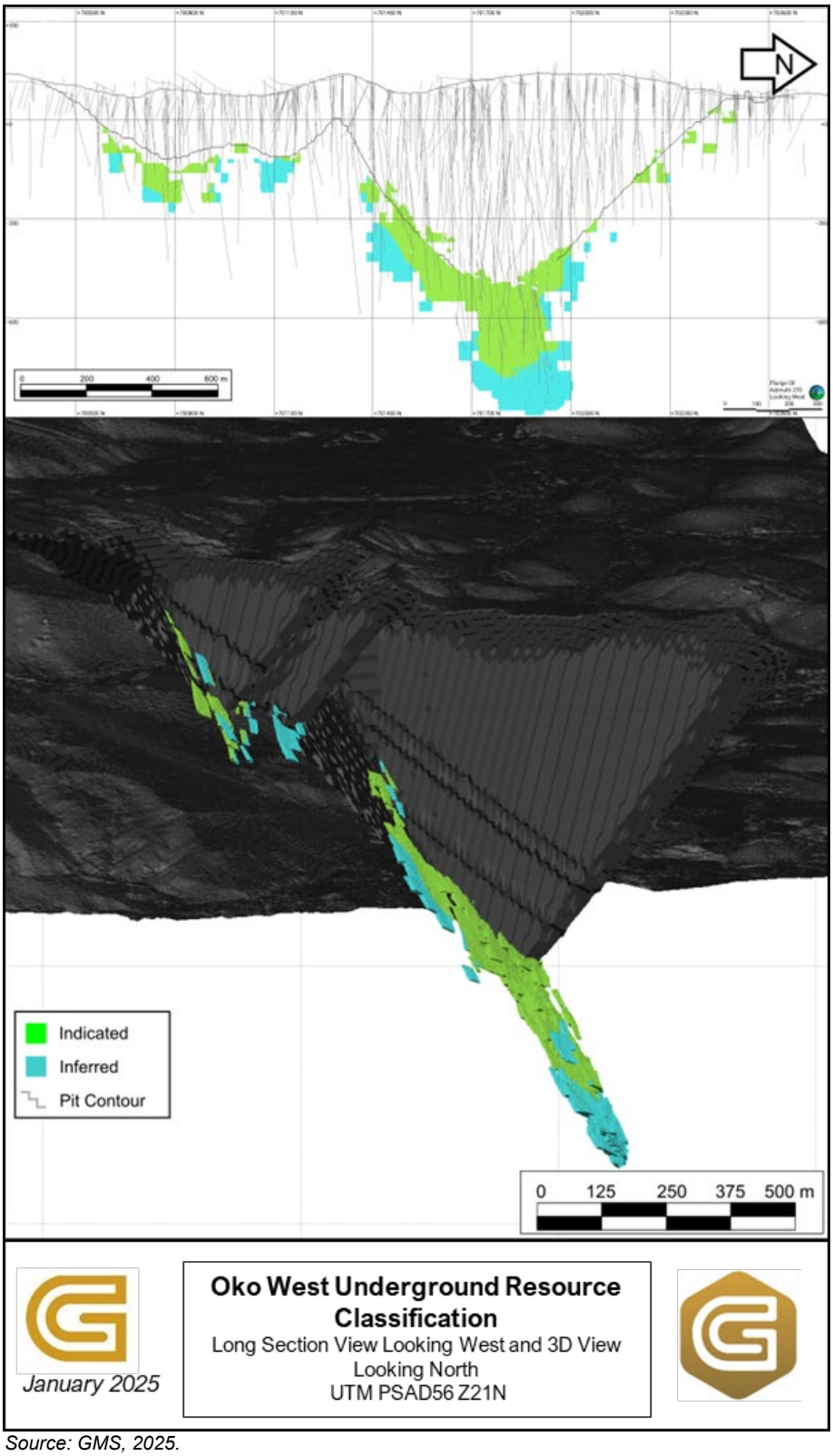
The final classification of mineral resources is displayed in Figure 14.16 for the in-pit resource and Figure 14.17 for the underground resource constrained within stopes optimized using Deswik Stope Optimizer (DSO).

Figure 14.16: Open-pit Mineral Resource Classification with Pit Outline Optimized Using Whittle



Source: GMS, 2025.

Figure 14.17: Underground Mineral Resource Classification Constrained Within Stopes Optimized from DSO



Source: GMS, 2025.

14.11.2 Reasonable Prospects of Eventual Economic Extraction (RPEEE)

The Oko West deposit is constrained by a Whittle pit shell for the open-pit part, and stopes modelled using Deswik Stope Optimizer (DSO) for the underground part of the deposit. Whittle pit shell and underground stopes were modelled by GMS mine engineering personnel. To define the resource pit and underground stopes, different parameters were selected according to the surface alteration intensity (i.e., weathering) of the host rocks. The parameters for pit and stopes optimization and cut-off grade assumptions are presented in Table 14.13 and Table 14.14. The optimization of the mineral resource pit is presented in Figure 14.16, while the optimization of the underground mineral resource stope is presented in Figure 14.17.

Table 14.13: Parameters Used for Open Pit Whittle Optimization and Open-pit Cut-off Grade Assumptions

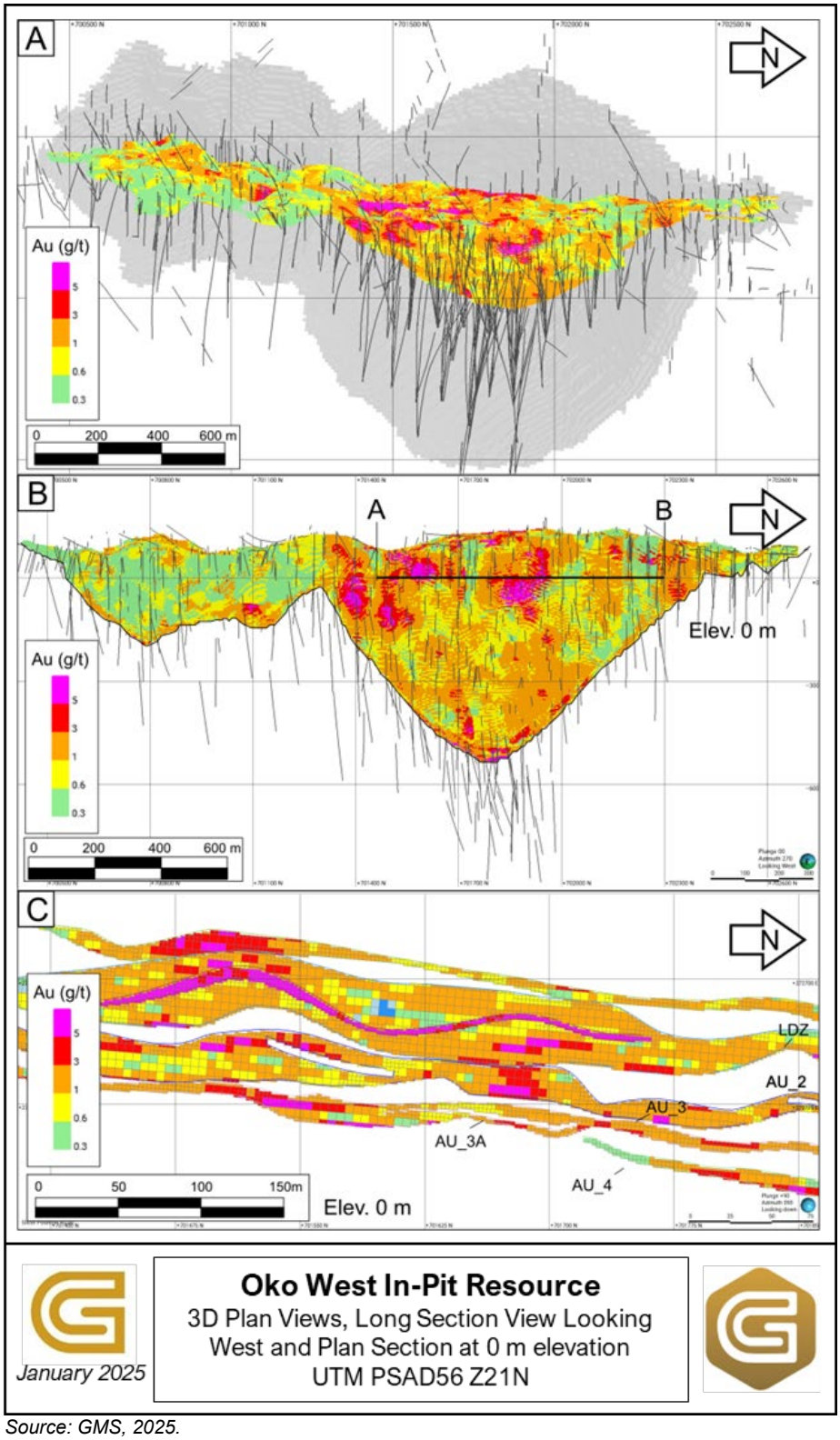
Optimization Parameters		Resources Parameters		
		Fresh Rock	Trans	Saprolite
Electricity Cost	*USD/kWh*	0.160	0.160	0.160
Discount Rate	*%*	5%	5%	5%
Gold Price	*USD/oz*	1,950.00	1,950.00	1,950.00
Payable Metal	*%*	99.95%	99.95%	99.95%
Transport & Refining Cost	*USD/oz*	8.00	8.00	8.00
Royalty Rate	*%*	8.00%	8.00%	8.00%
Royalty Cost	*USD/oz*	156	156	156
Net Ore Value	*USD/oz*	1,785.03	1,785.03	1,785.03
Nominal Milling Rate	*t/d*	16,438	16,438	16,438
Plant Throughput	*kt/y*	6,000	6,000	6,000
Recovery	*%*	93.9	93.3	94.5
Total Ore Based Cost	*USD/t milled*	15.77	13.52	11.64
Optimization Parameters		**Resources Parameters (Open-Pit)**		
		Fresh Rock	**Trans**	**Saprolite**
Mining Dilution	%	-	-	-
Mining Loss	%	-	-	-

Optimization Parameters		Resources Parameters		
		Fresh Rock	Trans	Saprolite
Total Mining Reference Cost	*USD/t mined*	2.37	2.22	1.73
Incremental Bench Cost	*USD/10 m bench*	0.04	0.04	0.04
IRA SLOPE	*degree*	50	40	30
Gram Value	*USD/g*	57.39	57.39	57.39
Total Ore Based Cost	*USD/t*	20.56	18.31	16.43
Used COG (Inc. Proc. Rec.)	*g/t*	0.38	0.34	0.30

Table 14.14: Parameters Used for Stope Optimization and Underground Cut-off Grade Assumptions

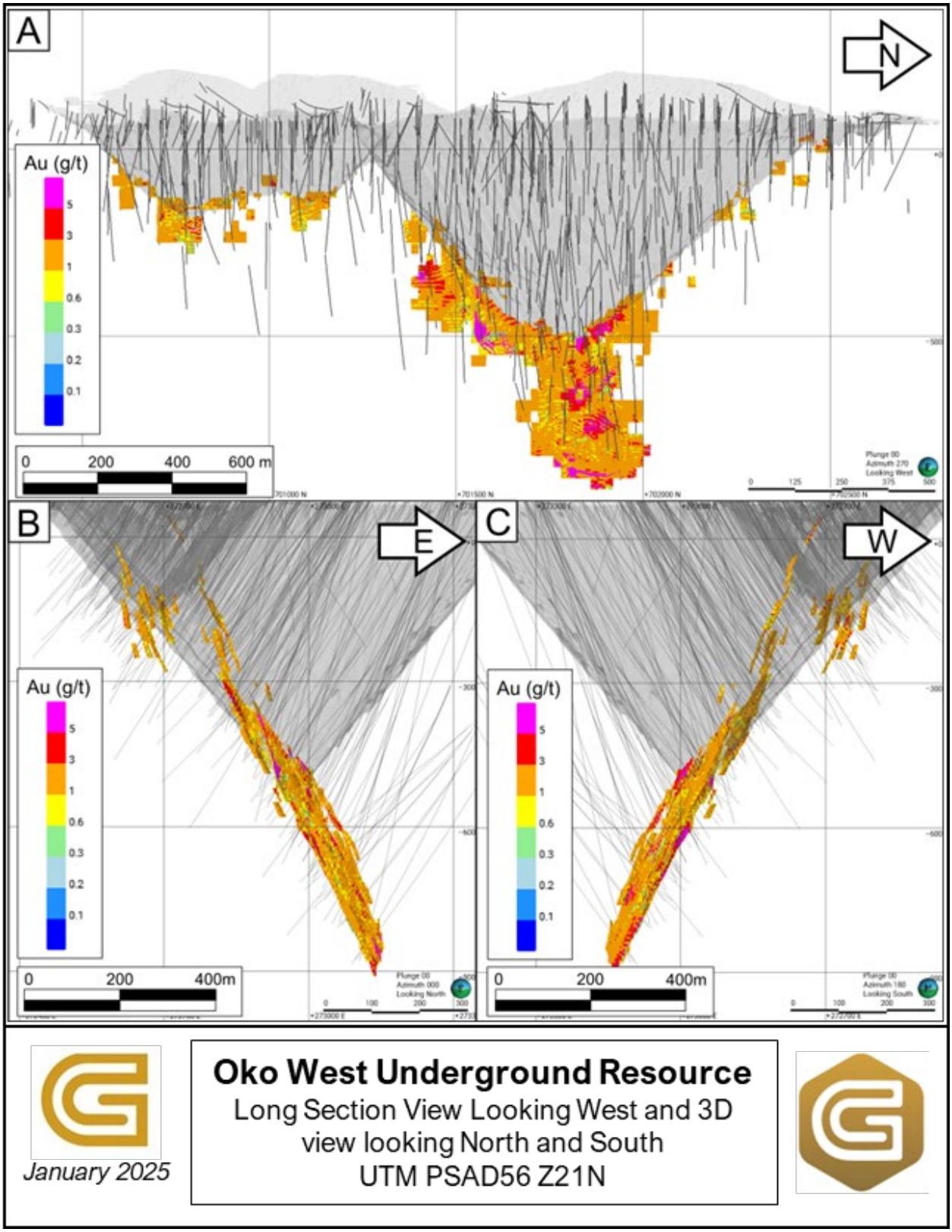
Optimization Parameters		Resources Parameters (Underground)		
		Fresh Rock	Trans	Saprolite
Royalty Rate	*%*	3.00%	3.00%	3.00%
Mining Dilution	*%*	-	-	-
Mining Recovery	*%*	100%	-	-
Total Mining Reference Cost	*USD/t mined*	44.38	-	-
Stope Height	*m*	30	-	-
Strike Length	*m*	20	-	-
Maximum Width	*m*	30	-	-
Minimum Mining Width	*m*	2	-	-
HW Dilution	*m*	-	-	-
FW Dilution	*m*	-	-	-
Minimum Pillar Width	*m*	5	-	-
Minimum Dip	*degree*	50	-	-
Stope Optimizer Sub-Shapes	*Elev. & Dir.*	Yes	-	-
Gram Value	*USD/g*	60.52	-	-
Total Ore Based Cost	*USD/t*	75.16	-	-
Used COG (Inc. Proc. Rec.)	*g/t*	1.35	-	-

Figure 14.18: Open-Pit Optimization with Block Model Coloured by Gold Grades (g/t)



Source: GMS, 2025.

Figure 14.19: Underground Stope Optimization with Block Model Coloured by Gold Grades (g/t)



Source: GMS, 2025.

14.12 Mineral Resource Statement

The Oko West deposit open-pit Mineral Resource is stated using a lower cut-off of 0.30 g/t Au in alluvium / colluvium and saprolite, 0.34 g/t Au in transitional material, and 0.38 g/t Au in fresh rocks. The resources are constrained within the resource pit. Results for the open-pit part of the deposit are presented by weathering profile in Table 14.15. Open pit Indicated Mineral Resources are estimated at 73,035 kt, grading 2.00 g/t Au for 4,689 koz Au. Open pit Inferred Mineral Resources are estimated at 1,542 kt, grading 1.06 g/t Au for 52 koz Au.

Table 14.15: Oko West Deposit In-pit Mineral Resource Estimate
Effective Date Sept. 15, 2024

Resource Classification	Weathering Profile	Tonnage	Grade	Gold Content
		(kt)	(g/t Au)	(koz Au)
Indicated	Alluvium / Colluvium	0	-	0
	Saprolite	6,020	1.68	325
	Transition	2,788	1.82	163
	Fresh Rock	64,228	2.03	4,201
	Total	**73,035**	**2.00**	**4,689**
Inferred	Alluvium / Colluvium	953	0.82	25
	Saprolite	51	0.54	1
	Transition	1	0.91	0.04
	Fresh Rock	537	1.52	26
	Total	**1,542**	**1.06**	**52**

**Notes:*

1. *The mineral resources described above have been prepared in accordance with the CIM Standards (Canadian Institute of Mining, Metallurgy and Petroleum, 2014) and follow Best Practices outlined by the CIM (2019).*
2. *The qualified person (QP) for this Mineral Resource Estimate (MRE) is Christian Beaulieu, P.Geo., Consulting Geologist for G Mining Services Inc.*
3. *The effective date of the Mineral Resource Estimate is September 15, 2024.*
4. *The lower cut-offs used to report open pit Mineral Resources is 0.30 g/t Au in saprolite and alluvium / colluvium, 0.34 g/t Au in transition, and 0.38 g/t Au in fresh rock.*
5. *The Oko West Deposit has been classified as Indicated and Inferred Mineral Resources according to drill spacing. No Measured Mineral Resource has been estimated.*
6. *The density has been applied based on measurements taken on drill core and assigned in the block model by weathering type and lithology.*
7. *A minimum thickness of 3 metres and minimum grade of 0.30 g/t Au was used to guide the interpretation of the mineralized zones.*
8. *This MRE is based on a subblock model with a main block size of 5 m x 5 m x 5 m, with subblocks of 2.5 m x 1.25 m x 2.5 m, and has been reported inside an optimized pit shell. Gold grades in fresh rock, transition and saprolite were interpolated with 1 m composites using Ordinary Kriging for all major domains and Inverse Distance for minor domains (AU_2A, AU_2B, AU_3B, AU_5 and LDZ_B6). Capping was applied on nine domains, ranging from 5 g/t Au to 45 g/t Au.*
9. *Open pit optimization parameters and cut-off grades assumptions are as follows:*
   a. *Gold price of US$1,950/oz.*
   b. *Total ore-based costs of US$16.43/t for saprolite and alluvium/colluvium, with a 94.5% processing recovery US$18.31/t for transition with a 93.3% processing recovery and US$20.56/t for fresh rock based on 93.9% processing recovery.*

c. Inter-ramp angles of 30° in saprolite and alluvium/colluvium, 40° in transition and 50° in fresh rock.
d. Royalty rate of 8%.
10. Tonnage has been expressed in the metric system, and gold metal content has been expressed in troy ounces.
11. The tonnages have been rounded to the nearest 1,000 tons, and the metal content has been rounded to the nearest 1,000 ounces. Totals may not add up due to rounding errors.
12. These mineral resources are not mineral reserves as they have not demonstrated economic viability. The quantity and grade of reported inferred mineral resources in this news release are uncertain in nature and there has been insufficient exploration to define these resources as indicated or measured; however, it is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated mineral resources with continued exploration.
13. These mineral resources assume no mining dilution and losses.

Stope-constrained underground mineral resources are presented in Table 14.16. The underground Resources are estimated from zones outside the constrained Resources of the open pit. The underground stope optimization was designed using a cut-off grade of 1.35 g/t Au – all material within the stopes is reported. Underground Indicated Mineral Resources are estimated at 7,223 kt, grading 1.85 g/t for 718 koz Au. Underground Inferred Mineral Resources are estimated at 3,585 kt, grading 2.93 g/t for 337 koz Au. The bulk of the ounces are located below the deepest limits of the open pit MRE, which represents the extension of the high-grade zone in Block 4 at depth.

Table 14.16: Oko West Deposit Underground Mineral Resource Estimate
Effective Date Sept. 15, 2024

Category	Tonnage (kt)	Au grade (g/t)	Contained Gold (koz)
Indicated	7,223	3.09	718
Inferred	3,585	2.93	337

**Notes:*
1. The mineral resources described above have been prepared in accordance with the CIM Standards (Canadian Institute of Mining, Metallurgy and Petroleum, 2014) and follow Best Practices outlined by the CIM (2019).
2. The qualified person (QP) for this Mineral Resource Estimate (MRE) is Christian Beaulieu, P.Geo., Consulting Geologist for G Mining Services Inc.
3. The effective date of the Mineral Resource Estimate is September 15, 2024.
4. Underground Mineral Resources are reported inside potentially mineable volume (i.e., must take material) and include below cut-off material (stope optimization cut-off grade: 1.35 g/t Au).
5. The Oko West Deposit has been classified as Indicated and Inferred Mineral Resources according to drill spacing. No Measured Mineral Resource has been estimated.
6. The density has been applied based on measurements taken on drill core and assigned in the block model by weathering type and lithology.
7. A minimum thickness of 3 metres and minimum grade of 0.30 g/t Au was used to guide the interpretation of the mineralized zones.
8. This MRE is based on a subblock model with a main block size of 5 m x 5 m x 5 m, with subblocks of 2.5 m x 1.25 m x 2.5 m, and has been reported inside optimized stope shapes. Gold grades in fresh rock, transition and saprolite were interpolated with 1 m composites using Ordinary Kriging for all major domains and Inverse Distance for minor domains (AU_2A, AU_2B, AU_3B, AU_5 and LDZ_B6). Capping was applied on eight domains, ranging from 5 g/t to 45 g/t Au.
9. UG optimization parameters and cut-off grades assumptions are as follows:
a. Gold price of US$1,950/oz.
b. Total ore-based costs of US$75.16/t for fresh rock.
c. Stope height of 30 m, strike length of 20 m, maximum width of 30 m and minimum width of 2 m.
d. The Deswik.SO (DSO) was used to constrain the resources.
e. Royalty rate of 3% payable to the Government of Guyana.
10. Tonnage has been expressed in the metric system, and gold metal content has been expressed in troy ounces.
11. The tonnages have been rounded to the nearest 1,000 tons, and the metal content has been rounded to the nearest 1,000 ounces. Totals may not add up due to rounding errors.
12. These mineral resources are not mineral reserves as they have not demonstrated economic viability. The quantity and grade of reported inferred mineral resources in this news release are uncertain in nature and there has been insufficient exploration to define these resources as indicated or measured; however, it is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated mineral resources with continued exploration.

13. *These mineral resources assume no mining dilution and losses, however must-take material is accounted for in underground stopes.*

Mr. Christian Beaulieu, P.Geo., is not aware of any factors or issues that materially affect the mineral resource estimate other than normal risks faced by mining projects in the province in terms of environmental, permitting, taxation, socio-economic, marketing, and political factors, and additional risk factors regarding indicated and inferred resources.

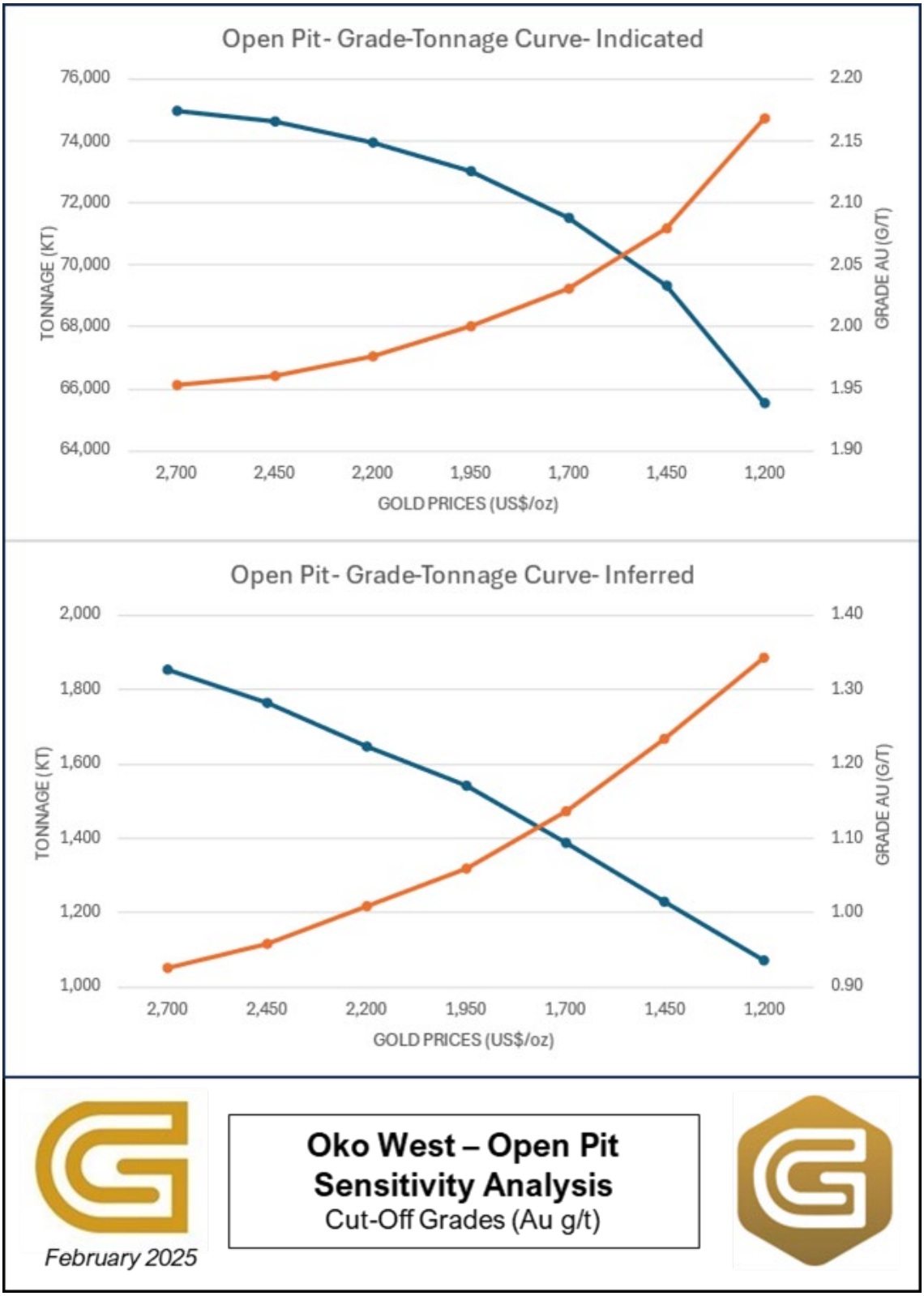
These mineral resources are not mineral reserves as they have not demonstrated economic viability. The quantity and grade of reported inferred mineral resources in this Report are uncertain in nature and there has been insufficient exploration to define these resources as indicated or measured; however, it is reasonably expected that the majority of Inferred mineral resources could be upgraded to Indicated mineral resources with continued exploration.

14.12.1 Cut-Off Grade Sensitivities

The sensitivity of the open pit and underground resources to different cut-off grades scenarios is summarized in Table 14.17 and Table 14.18. Figure 14.20 presents the grade-tonnage curves for varying gold cut-offs of the indicated and inferred open-pit mineral resource, Figure 14.21 presents the grade-tonnage curves for varying gold cut-offs of the indicated and inferred stopes-constrained underground mineral resource. The tonnages and grade at differing cut-offs shown below are for comparison purposes only and do not constitute an official Mineral Resource. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

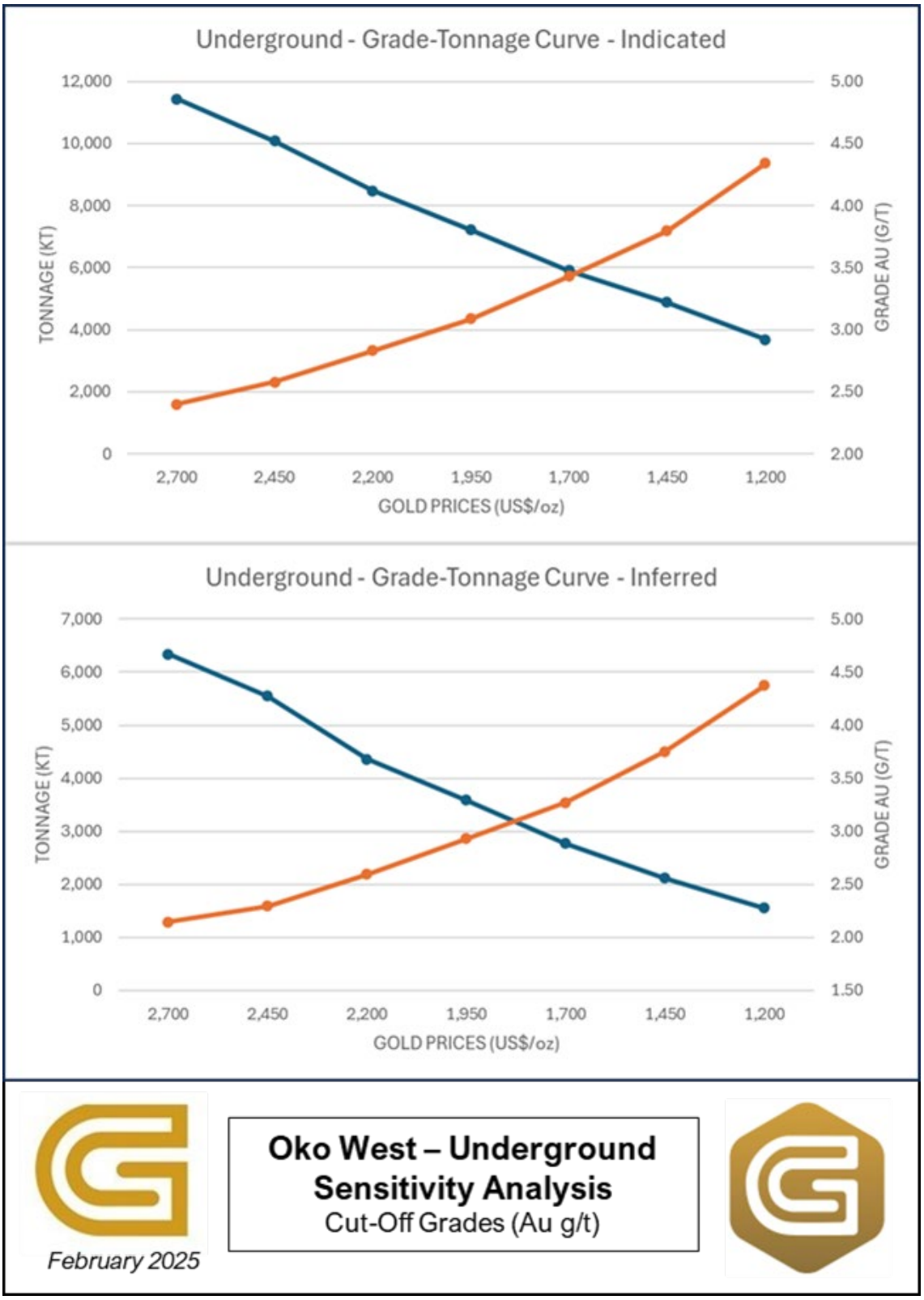
As seen from the following table and graphs, the Oko West open-pit deposit shows a low sensitivity to cut-off grades. The gold content decrease in the Indicated category remains stable with increasing cut-offs below the US$1,450/oz cut-off. The Inferred category constrained within the pit does not reflect the sensitivity of the deposit and only accounts for about 1% of the total gold content. As expected, the underground part of the deposit is more sensitive to increasing cut-offs. However, the gold content decrease remains relatively linear with increasing cut-offs for gold prices above US$1,200/oz (or 2.20 g/t Au).

Figure 14.20: Indicated and Inferred Grade-Tonnage Curves for In-pit Resource



Source: GMS, 2025.

Figure 14.21: Indicated and Inferred Grade-Tonnage Curves Within Underground Stopes Modelled at Different Cut-off Grades Using DSO



Source: GMS, 2025.

Table 14.17: Oko West In-Pit Cut-off Grade Sensitivity

Cut-off Grade Sap / Trans / Fresh (g/t)	Gold Price (US$/oz)	Indicated			Inferred		
		Tonnage	Grade	Gold Content	Tonnage	Grade	Gold Content
		(kt)	(g/t)	(koz)	(kt)	(g/t)	(koz)
0.22 / 0.25 / 0.28	2,700	74,968	1.95	4,709	1,852	0.92	55
0.24 / 0.27 / 0.30	2,450	74,656	1.96	4,706	1,766	0.96	54
0.27 / 0.30 / 0.34	2,200	73,936	1.98	4,699	1,648	1.01	53
COG*	**1,950**	**73,035**	**2.00**	**4,686**	**1,542**	**1.06**	**52**
0.35 / 0.39 / 0.44	1,700	71,507	2.03	4,669	1,389	1.14	51
0.41 / 0.46 / 0.51	1,450	69,345	2.08	4,637	1,228	1.23	49
0.49 / 0.56 / 0.62	1,200	65,531	2.17	4,569	1,073	1.34	46

Note(s):

1. *COG tonnage and grades are calculated at the MRE cut-off grade (0.30 g/t Au in colluvium / alluvium and saprolite, 0.34 g/t Au in transition and 0.38 g/t Au in fresh rock).*
2. *The tonnages and grade at differing cut-offs shown above are for comparison purposes only and do not constitute an official Mineral Resource Estimate.*

Table 14.18: Oko West Underground Cut-off Grade Sensitivity

Cut-off Grade (g/t)	Gold Price (US$/oz)	Indicated			Inferred		
		Tonnage	Grade	Gold Content	Tonnage	Grade	Gold Content
		(kt)	(g/t)	(koz)	(kt)	(g/t)	(koz)
0.95	2,700	11,440	2.40	883	6,343	2.15	438
1.05	2,450	10,089	2.58	837	5,552	2.30	410
1.20	2,200	8,484	2.83	773	4,362	2.59	364
COG*	1,950	7,223	3.09	718	3,585	2.93	337
1.55	1,700	5,912	3.43	652	2,772	3.27	291
1.80	1,450	4,885	3.80	596	2,119	3.75	256
2.20	1,200	3,689	4.34	515	1,553	4.38	218

Note(s):

1. *COG tonnage and grades are calculated at the MRE cut-off grade (1.35 g/t for underground material in fresh rock).*
2. *The tonnages and grade at differing cut-offs shown above are for comparison purposes only and do not constitute an official Mineral Resource Estimate.*
3. *The tonnages and grade at differing cut-offs shown above are constrained within optimized stope.*

15. MINERAL RESERVE ESTIMATES

15.1 Introduction

The mine design and Mineral Reserve estimate were completed to a level appropriate for feasibility studies. The Mineral Reserves were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (November 29, 2019) and CIM Definition Standards for Mineral Resources and Reserves (May 10, 2014).

As such, the Mineral Reserves are based on Measured and Indicated Mineral Resources and do not include any Inferred Mineral Resources. The Inferred Mineral Resources contained within the mine design are treated as waste at a grade of 0 g/t Au.

The Proven and Probable Reserve for the Oko West project is estimated at 76.6 Mt at an average grade of 1.89 g/t Au for 4.64 million ounces of gold as summarized in the Table 15.1.

It should be noted that there are no Proven Reserves; only Probable Reserves are declared.

Table 15.1: Oko Gold Project Mineral Reserve

Area	Category	Tonne (Mt)	Au Grade (g/t)	Contained Au (k oz)
Open Pit	Proven	0	0	0
	Probable	62.4	1.57	3,156
	Proven and Probable	**62.4**	**1.57**	**3,156**
Underground	Proven	0	0	0
	Probable	14.2	3.26	1,486
	Proven and Probable	**14.2**	**3.26**	**1,486**
Total (Open Pit + Underground)	Proven	0	0	0
	Probable	76.6	1.89	4,642
	Proven and Probable	**76.6**	**1.89**	**4,642**

**Notes on Mineral Reserves:*

1. *The Mineral Reserves were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (November 29 ,2019) and CIM Definition Standards for Mineral Resources and Reserves, (May 10, 2014). The mine design and Mineral Reserve estimate have*

been completed to a level appropriate for feasibility studies. As such, the Mineral Reserves are based on the Measured and Indicated Mineral Resources and do not include any Inferred Mineral Resources. The Inferred Mineral Resources contained within the mine design are classified as waste.

2. *Mineral Reserves are estimated using a long-term gold price of USD 1,800/oz.*
3. *The qualified person for the estimate is Alexandre Burelle, P.Eng. (OIQ#5019855), Mine planning and financial analysis consultant. The estimate has an effective date of April 2, 2025.*
4. *Mineral Reserves for Open Pit are estimated at a cut-off grade of 0.41, 0.37 and 0.33 g/t for Fresh Rock, Transition, and Saprolite.*
5. *The Open Pit Strip Ratio is 6.83:1 and Dilution factor is 14%.*
6. *Mineral Reserves for Underground Mine are estimated at a cut-off grade of 1.70 g/t.*
7. *The underground mine dilution factor is 11.3% including 4.5% for backfill dilution.*
8. *For the underground stopes, a minimum mining width of 5 m was used.*
9. *The numbers may not sum due to rounding; rounding followed the recommendations in NI 43-101.*
10. *The mine design and Mineral Reserve estimate have been completed to a level appropriate for feasibility studies. The Mineral Reserve estimate stated herein is consistent with the CIM definitions and is suitable for public reporting.*

15.2 Open Pit

15.2.1 Mineral Resource Block Model

The Mineral Resource block model (Oko West 2.5_0.5_2.5) was imported into the Deswik CAD™ software as a single block model. For the open pit optimizations, the model provided was regularized and reblocked into a 5 m x 5 m x 5 m block model.

15.2.2 Open Pit Optimization

Open pit optimization was conducted in GEOVIA WhittleTM version 2022 to determine the optimal economic shape of the open pit to guide the pit design process. This software uses the pseudoflow algorithm. The pseudoflow algorithm works on a block model of the ore body, progressively constructing lists of related blocks that should, or should not, be mined. The method uses the values of the blocks to define a pit outline that has the highest possible total economic value, subject to the required pit slopes defined as structure arcs as well as physicals constraints under the form of heavy blocks in the software.

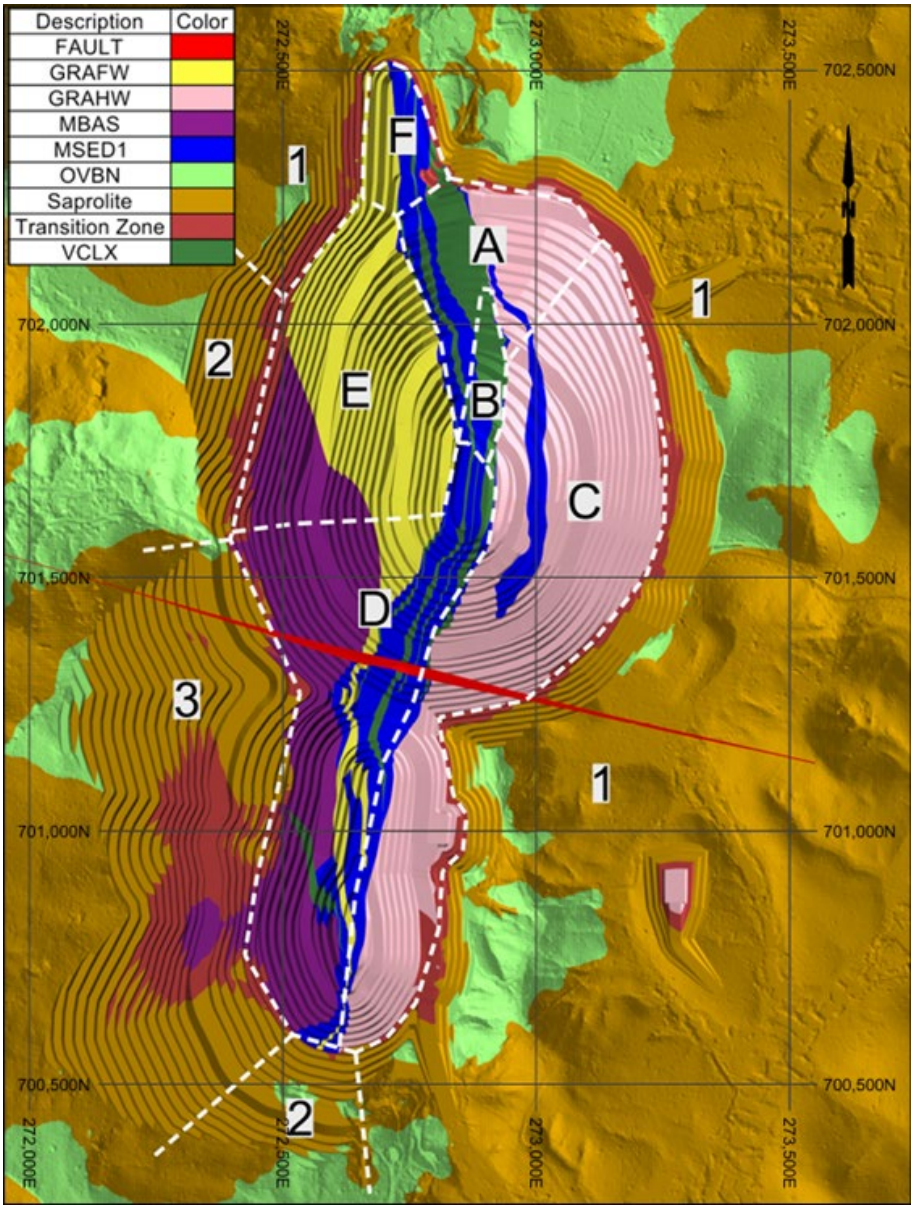
To ensure the software can process the block model, the reblocked block model of 5 m x 5 m x 5 m is reblocked into 10 m x 10 m x 10 m in Whittle™. This induces a difference in the material tonnage estimates between Whittle and the shells interrogated in Deswik with the 5 m x 5 m x 5 m block model.

The pit optimizations performed to generate optimal pit limits to guide the ultimate pit design were based only on Measured and Indicated Resource category blocks.

15.2.3 Slope Recommendations

Alius Mine Consulting Inc. delivered a geotechnical study with design parameters varying by weathering, slope direction and lithology. Further details are provided in Section 16. The analysis takes into consideration rock mechanics as well as joint discontinuity orientations for various sectors and lithology. The geotechnical domains considered are presented in Figure 15.1.

Figure 15.1: Geotechnical Domains for Saprolite and Rock



Source: Alius, 2025.
**Note: Not to scale.*

The recommendations provided for saprolite slopes consider triaxial tests that were performed on samples collected across the footprint of the project. There is a significant difference in saprolite strength between the east and west side of the pit, attributable to differences in the geology of the original bedrock before weathering and the global thickness of saprolite. Depending on the face orientations from geotechnical discontinuities, the Inter-Ramp Angle (IRA) in fresh rock in the main pit varies from 40 to 55. Due to the depth of the pit, geotechnical berms have been added to the design parameters in order to contain large rockfalls. Finally, the geotechnical design parameters include a 10 m wide bench that should be maintained at the transition between saprolite and bedrock, which will facilitate the installation of a collector berm and

mitigate saprolite exposure at the pit crest. The geotechnical berm design criteria are summarized in Table 15.2.

Table 15.2: Geotechnical Berms Design Criteria

Geotechnical Berm Requirements	
Starts when 80 m of rock is exposed; Ends at 80 m from the pit bottom; Spaced vertically at 180 m intervals.	
Catch berm	16 m wide catch berm
At the transition between saprolite and bedrock	10 m wide catch bench

**Note: A ramp itself can't be considered as a geotechnical berm if no space for a catch berm is included inside.*

The Overall Slope Angle (OSA) design criteria considers both, the IRA and geotechnical berms with resulting design parameters presented in Table 15.3, which are used for pit optimization.

Table 15.3: Detailed Slope Design Parameters

Rock TYPE and Sector	Bench Height (m)	Bench Width (m)	Bench Face Angle, Degrees	Inter-Ramp Angle "IRA", Degrees	Overall Slope Angle "OSA", Degrees
Saprolite and Overburden, Sector 1	10.0	8.0	60	36	**36**
Saprolite and Overburden West, Sector 2	10.0	13.0	60	28	**28**
Saprolite and Overburden West, Sector 3	10.0	15.5	60	25	**25**
Transition	10.0	10.0	70	40	**38**
Rock = A / 335-40	20.0	8.5	55	42	**36**
Rock = B / 10-40	20.0	8.5	52	40	**34**
Rock = C / 40-190	20.0	8.5	75	55	**46**
Rock = D / 190-260	20.0	9.5	70	50	**42**
Rock = E / 260-335	20.0	8.5	75	55	**46**
Rock Sub Pits = F	20.0	8.5	80	59	**49**

15.2.4 Mining Dilution and Ore Loss

A specific algorithm has been developed to calculate the mining dilution and ore material recovery for the Oko West deposit. This algorithm considers the type of rock weathering to determine a width of block dilution between ore material block and waste block contact. This dilution skin varies from 0.5 m in saprolite

to 1.5 m in the transition material and fresh rock. The dilution impact is therefore dependant on to the Cut-Off Grade and the number of interfaces between ore material and waste.

For each ore block in contact with a waste block, the grade of the dilution skin of the waste block grade is considered in the ore block. A new overall density and grade are then recalculated for the ore block.

The mining dilution and ore loss algorithm is applied to the regularized block model. The reserves statement within the pit design accounts for it.

15.2.5 Pit Optimization Parameters and Cut-Off Grade

A summary of the pit optimization parameters is presented in Table 15.4 for a nominal mining rate of 6.0 Mt/y. Gold price for the Mineral Reserve estimate is set at $1,800/oz. Royalties for the open pit component of this project are at 8% of NSR. The reference mining cost is estimated from the previous PEA study with updated salaries and diesel costs at $2.37/t for fresh rock, $2.22/t for transition, and $1.73/t for saprolite. The incremental cost comes from the same analysis and is set at $0.04/t per 10 m bench.

Total ore material-based cost including processing, general & administration, rehabilitation, and sustaining capital was taken to be $20.56/t milled for fresh rock, $18.31/t milled for transition material and $16.43/t milled for saprolite. Metallurgical recoveries are evaluated respectively at 93.9% in fresh rock, 93.3% in transition, and 94.5% in saprolite through various core samples analyzed in a laboratory.

Such ore-based costs give a marginal cut-of-grade including process recovery of 0.41 g/t for fresh rock, 0.37 g/t for transition material and 0.33 g/t for saprolite. Those cut-off grade values will be used for the Mineral Reserves statement calculation.

The Overall Slope Angles presented in Section 15.2.3 are used in the optimization parameters and vary from 26 to 49 degrees.

Table 15.4: Economic Optimization Parameters per Rock Type

Parameters	Units	Fresh Rock	Transition	Saprolite
Economic Parameters				
Electricity Cost	*US$/kWh*	0.1600	0.1600	0.1600
Discount Rate	*%*	5.0%	5.0%	5.0%
Gold Price	*US$/oz*	1,800	1,800	1,800
Payable Metal	*%*	99.95%	99.95%	99.95%

Parameters	Units	Fresh Rock	Transition	Saprolite
Transport & Refining Cost	*US$/oz*	8.00	8.00	8.00
Royalty Rate (Calculated)	*%*	8.00%	8.00%	8.00%
Royalty Cost	*US$/oz*	144.00	144.00	144.00
Net Ore Value	*US$/oz*	1,647.10	1,647.10	1,647.10
Processing Assumptions				
Nominal Milling Rate	*t/d*	16,438	16,438	16,438
Plant Throughput	*kt/y*	6,000,000	6,000,000	6,000,000
Recovery	*%*	93.9%	93.3%	94.5%
Processing Costs				
Plant Labour Unit Rate	*US$/t milled*	1.61	1.61	1.61
Consumables (reagents & media)	*US$/t milled*	5.57	5.25	4.99
Plant Maintenance & Lab.	*US$/t milled*	1.83	1.86	1.85
Total Processing Cost	***US$/t milled***	**9.01**	**8.72**	**8.45**
Power Costs				
Variable Power Consumption	*kWh/t*	42.3	30.0	19.9
Power Cost	***US$/t milled***	**6.76**	**4.80**	**3.18**
Other Ore Based Costs				
General & Administration Costs	*US$/t milled*	4.15	4.15	4.15
Rehabilitation & Closure	*US$/t milled*	0.20	0.20	0.20
Sustaining Capital	*US$/t milled*	0.44	0.44	0.44
Total Other Ore Based Costs	***US$/t milled***	**4.79**	**4.79**	**4.79**
Total Ore Based Costs				
Total Ore Based Cost	***US$/t milled***	**20.56**	**18.31**	**16.43**
Marginal Cut-Off Grade Gold Recovered				
Cut-Off Grade	*g/t Au*	**0.41**	**0.37**	**0.33**
Mining Assumptions				
Nominal Daily Mining Rate	*t/d*	192,000	192,000	192,000
Mining Rate	*kt/y*	70,080	70,080	70,080
Mining Dilution	*%*	13.0%	15.0%	7.0%
Mining Loss	*%*	5.0%	7.0%	3.0%

Parameters	Units	Fresh Rock	Transition	Saprolite
Mining Costs				
Total Mining Reference Cost	*US$/t mined*	2.37	2.22	1.73
Incremental Bench Cost	*US$/10 m bench*	0.040	0.040	0.040

15.2.6 Pit Optimization Results

The Whittle nested shell results are presented in Table 15.5 using only Measured and Indicated resources. The nested pit shells are generated using revenue factors to scale up and down from the base case selling price.

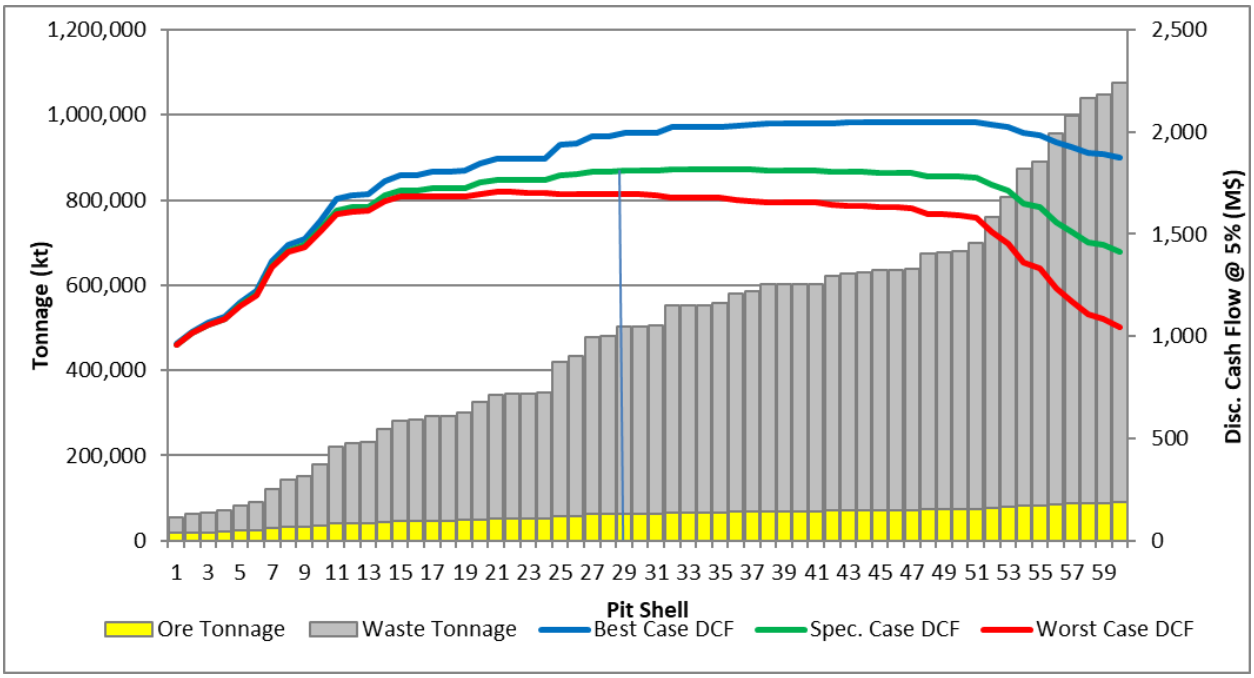
Table 15.5: Measured and Indicated Mineral Resource Whittle Shell Results

Pit Shell	Best Case Disc. @ 5% (M$)	Specified Case Disc. @ 5% (M$)	Worst Case Disc. @ 5% (M$)	Total Tonnage (kt)	Ore Tonnage (kt)	Strip Ratio (W:O)	Grade (g/t)	In-Situ Gold (k oz)
5	962	956	956	55,221	17,502	2.16	2.59	1,025
6	1,022	1,014	1,013	61,715	18,903	2.26	2.49	1,102
7	1,066	1,057	1,056	67,028	20,076	2.34	2.40	1,163
8	1,095	1,085	1,083	70,883	20,911	2.39	2.32	1,205
9	1,165	1,152	1,149	82,131	23,022	2.57	2.23	1,310
10	1,221	1,206	1,202	91,212	24,627	2.70	2.17	1,395
11	1,366	1,344	1,336	122,136	29,268	3.17	2.12	1,642
12	1,448	1,422	1,412	143,042	31,957	3.48	2.09	1,793
13	1,479	1,450	1,439	151,219	33,019	3.58	2.05	1,850
14	1,564	1,525	1,510	179,433	36,391	3.93	2.02	2,028
15	1,672	1,615	1,597	219,511	40,592	4.41	2.01	2,266
16	1,691	1,630	1,610	228,062	41,546	4.49	1.98	2,312
17	1,697	1,635	1,614	230,499	41,954	4.49	1.95	2,329
18	1,757	1,687	1,659	262,797	44,626	4.89	1.93	2,488
19	1,788	1,714	1,682	281,980	46,006	5.13	1.93	2,576
20	1,791	1,715	1,682	284,246	46,291	5.14	1.91	2,586
21	1,803	1,723	1,687	291,967	47,061	5.20	1.90	2,624
22	1,803	1,723	1,687	292,044	47,075	5.20	1.89	2,624

Pit Shell	Best Case Disc. @ 5% (M$)	Specified Case Disc. @ 5% (M$)	Worst Case Disc. @ 5% (M$)	Total Tonnage (kt)	Ore Tonnage (kt)	Strip Ratio (W:O)	Grade (g/t)	In-Situ Gold (k oz)
23	1,813	1,727	1,682	299,974	48,168	5.23	1.86	2,662
24	1,845	1,751	1,698	324,821	49,954	5.50	1.86	2,769
25	1,867	1,765	1,706	343,671	51,237	5.71	1.85	2,846
26	1,869	1,766	1,705	345,153	51,458	5.71	1.84	2,854
27	1,870	1,766	1,704	346,236	51,577	5.71	1.83	2,858
28	1,871	1,767	1,704	346,919	51,640	5.7	1.82	2,861
29	1,936	1,789	1,693	420,261	58,219	6.2	1.79	3,157
30	1,946	1,795	1,695	432,755	58,800	6.4	1.78	3,201
31	1,979	1,806	1,697	478,919	61,881	6.7	1.78	3,375
32	1,980	1,806	1,696	479,576	61,947	6.7	1.77	3,378
33	1,995	1,810	1,694	502,241	63,277	6.9	1.77	3,459
34	1,995	1,810	1,694	502,299	63,286	6.9	1.76	3,459
35	1,997	1,809	1,691	506,158	63,707	6.9	1.74	3,475
36	2,022	1,816	1,679	553,632	66,402	7.3	1.74	3,637
37	2,022	1,816	1,679	553,703	66,414	7.3	1.74	3,638
38	2,022	1,816	1,679	553,794	66,424	7.3	1.73	3,638
39	2,024	1,815	1,676	558,375	66,832	7.4	1.72	3,655
40	2,032	1,815	1,669	578,877	67,913	7.5	1.71	3,721
41	2,034	1,814	1,663	585,560	68,464	7.6	1.70	3,744
42	2,039	1,813	1,655	601,924	69,400	7.7	1.69	3,796
43	2,040	1,813	1,654	602,317	69,476	7.7	1.69	3,798
44	2,040	1,813	1,654	603,298	69,570	7.7	1.68	3,801
45	2,040	1,813	1,653	603,483	69,584	7.7	1.67	3,802
46	2,044	1,805	1,641	622,937	70,785	7.8	1.66	3,862
47	2,045	1,804	1,639	628,646	71,027	7.9	1.66	3,879
48	2,045	1,803	1,637	630,242	71,135	7.9	1.65	3,883
49	2,045	1,801	1,633	635,336	71,425	7.9	1.64	3,898
50	2,046	1,801	1,633	635,439	71,436	7.9	1.64	3,899
51	2,046	1,798	1,628	639,528	71,746	7.9	1.63	3,910

Pit Shell	Best Case Disc. @ 5% (M$)	Specified Case Disc. @ 5% (M$)	Worst Case Disc. @ 5% (M$)	Total Tonnage (kt)	Ore Tonnage (kt)	Strip Ratio (W:O)	Grade (g/t)	In-Situ Gold (k oz)
52	2,047	1,784	1,601	673,269	73,389	8.2	1.62	4,002
53	2,048	1,781	1,596	678,109	73,737	8.2	1.62	4,017

Figure 15.2: Pit by Pit Graph M&I Resource



The shell selection is presented in Figure 15.2 and in Table 15.6. Pit shell 29 was selected as the optimum final pit shell, with a corresponding revenue factor of 0.75. This shell has a total tonnage of 420 Mt, including 58 Mt of mineralized material. The pit shell was selected as being near-optimal in terms of discounted cash flow in the specified-case scenario in Whittle™ while not exposing the project to the additional risk of larger revenue factor shells that comes at little to null incremental value.

Trade-off analyses were also made to determine the best limit between the open pit and underground designs. Pit Shell 29 was found to be NPV-optimal for the project. It balances the Life of Mine between the OP and UG operations and optimizes the strip ratio for the open pit.

Table 15.6: M&I Pit Shell Selection

Shell Selection	Best	Spec.	Worst	Selection
Shell Number	53	36	25	29
Shell RF	0.99	0.82	0.71	0.75
Shell Price (US$/oz)	1,733	1,435	1,243	1,313
Total Tonnage (kt)	678,109	553,632	343,671	420,261
Waste Tonnage (kt)	604,372	487,230	292,435	362,042
Strip Ratio (W:O)	8.20	7.34	5.71	6.22
Ore Tonnage (kt)	73,737	66,402	51,237	58,219
Grade (g/t)	1.62	1.74	1.85	1.79
In-situ Gold (k oz)	4,017	3,637	2,846	3,157
DCF @ 5% (M$)	2,048	1,816	1,706	1,789
LOM (Y)	13.5	12.1	9.4	10.6

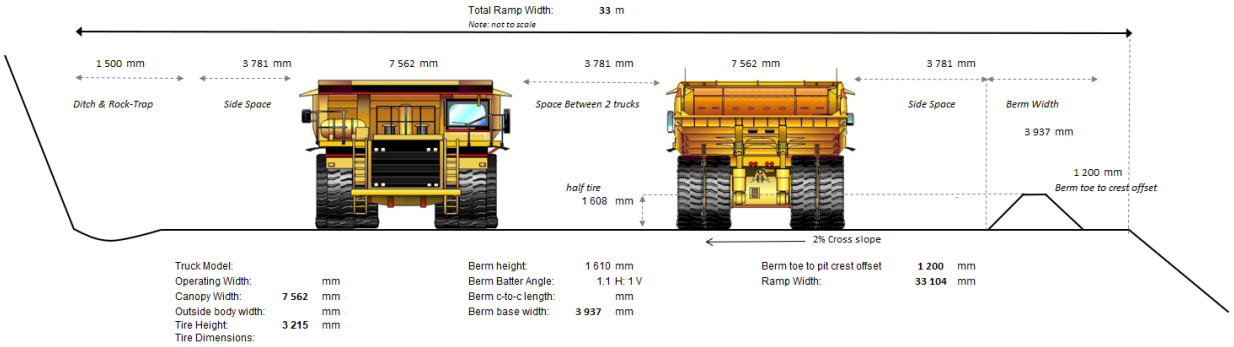
15.2.7 Mine Design

15.2.7.1 Ramp Design Criteria

The ramps and haul roads are designed for the largest equipment considered, that being a Komatsu HD1500-8 150 short ton class haul truck with a canopy width of 7.6 m. For double lane traffic, industry's best practice is to consider a travelling surface of at least three and a half times the width of the largest vehicle. The ramp's maximum gradient is set at 10%.

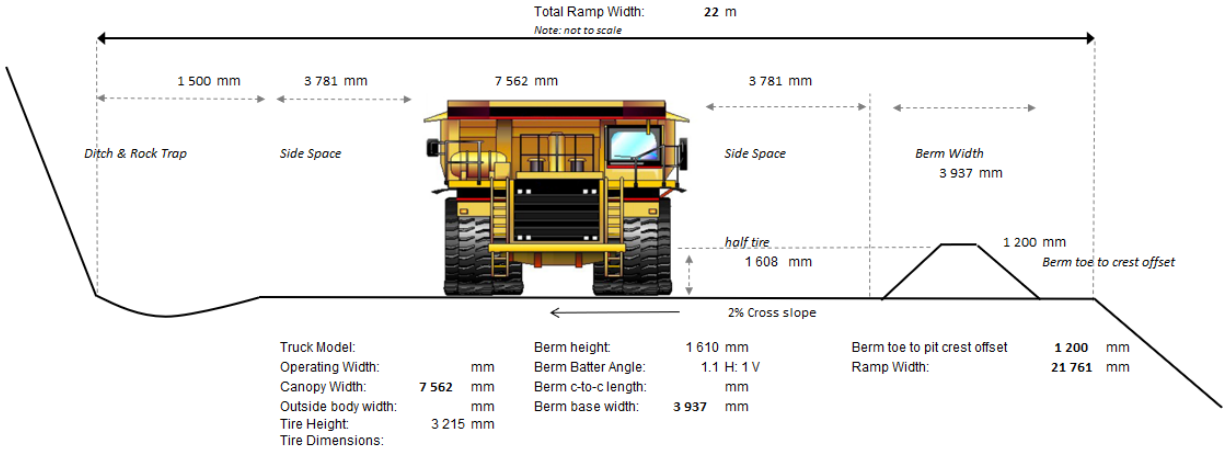
A safety berm on the outside edge will be constructed with crushed rock to a height equal to the rolling radius of the largest tire using the ramp. The rolling radius of the truck tire is 1.61 m. These shoulder barriers are required wherever a drop-off greater than 3 m exists and will be designed at 1.1H:1V. A ditch planned on the highwall will capture run-off from the pit wall surface and ensure proper drainage of the running surface. The ditch will be 1.5 m wide. To facilitate drainage of the roadway, a 2% cross slope on the ramp is planned. The double-lane ramp width is 33 m wide, and the single-lane ramp is 22 m wide. Single-lane ramps are introduced in pits bottom when the benches start narrowing and when the mining rates will be significantly reduced. Double and single-lane ramp configurations are shown in Figure 15.3 and Figure 15.4, respectively.

Figure 15.3: Double-Lane Ramp Design Criteria



Source: GMS, 2025.
**Note: Not to scale.*

Figure 15.4: Single-Lane Ramp Design Criteria

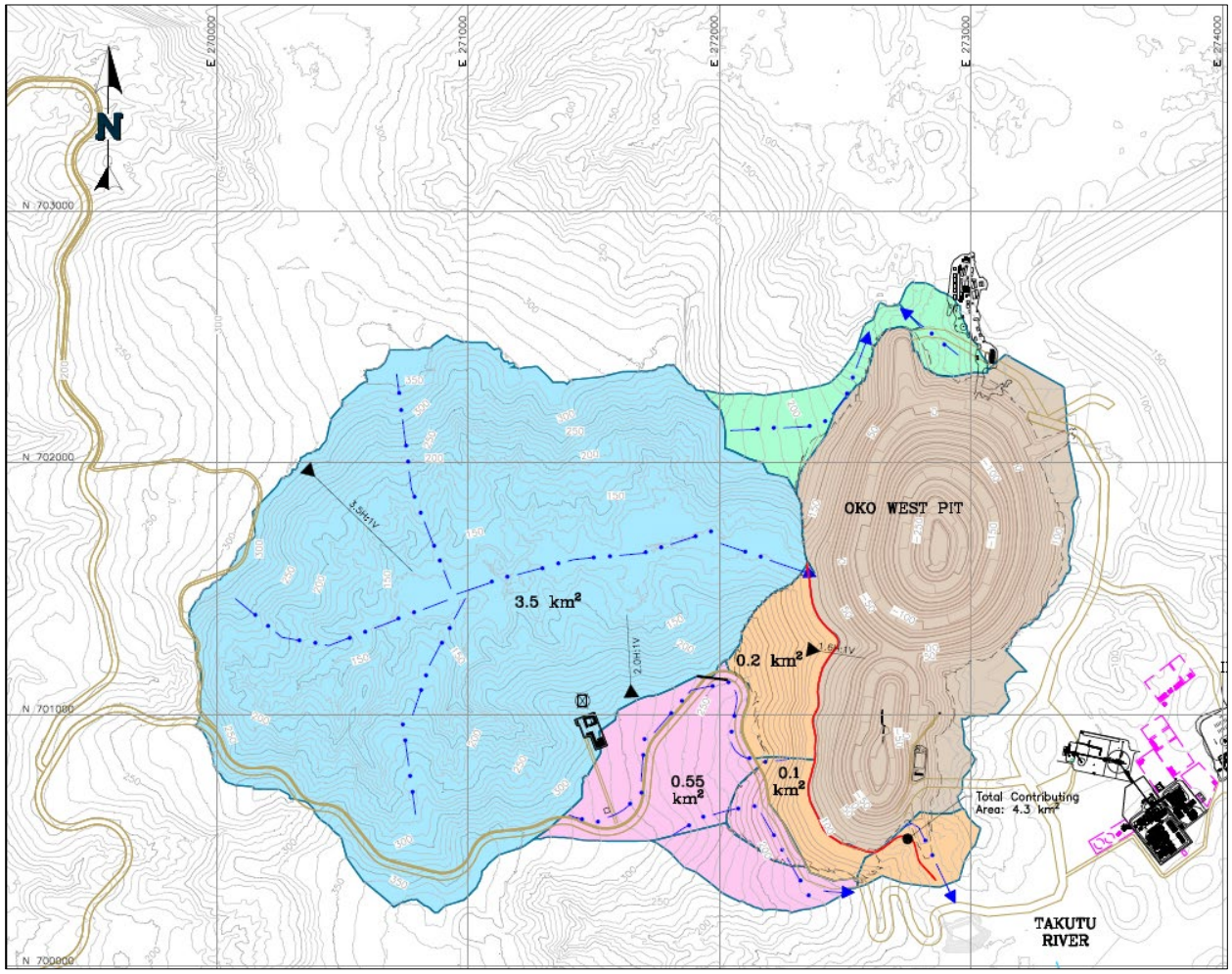


Source: GMS, 2025.
**Note: Not to scale.*

15.2.7.2 Diversion Ditch

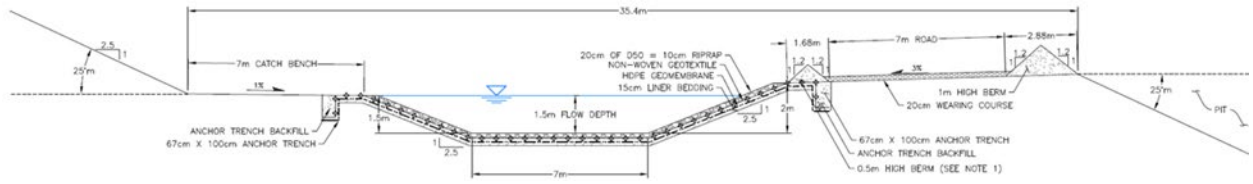
Newfields was mandated to establish a hydrologic model of the area surrounding Oko West pit to assess the stormwater runoff to be controlled within to the ultimate pit area. The evaluation involved the identification of contributing stormwater catchments and the selection of the storm criteria with the peak run-off. The study concludes that an important watershed, illustrated in blue in Figure 15.5, will direct water toward the pit. To avoid managing such a large volume of water in the pit, it was decided to include a diversion ditch in the final pit design to keep the water of this watershed outside of the pit. The diversion ditch design is detailed in Figure 15.5. It includes a catch bench width of 7 m, a 7 m wide single-lane road access for maintenance, safety berms, and 16.8 m for the channel itself for a total of 35.4 m. Further details about the Newfields study are provided in Section 16.

Figure 15.5: Watersheds Around Oko West Pit (= Blue, Green, Pink and Orange areas) and Deviation Ditch Implementation (=Red)



Source: Newfields, 2025.
**Note: Not to scale.*

Figure 15.6: Diversion Ditch Design



Source: Newfields, 2025.

15.2.7.3 Open Pit Mine Design Results

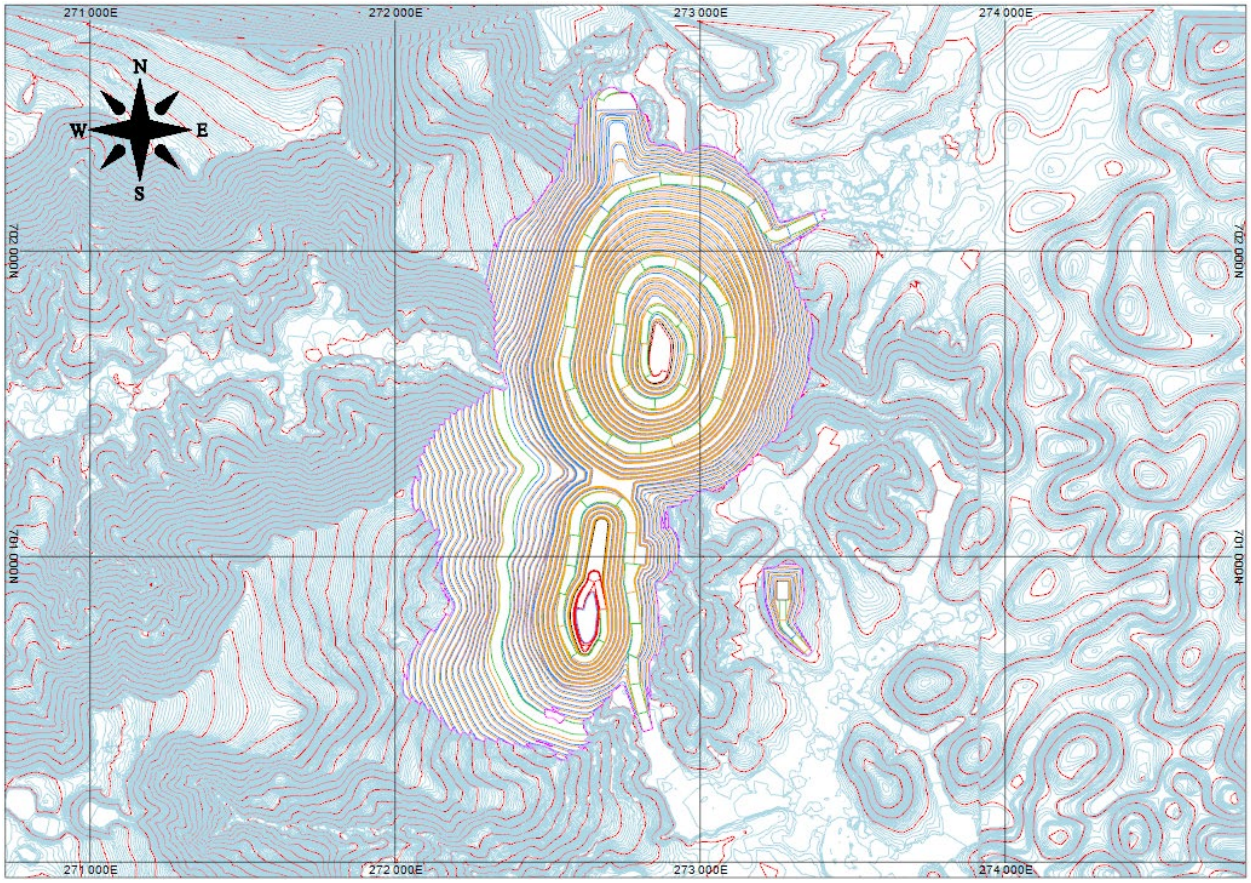
The Oko West deposit forms a single pit presented in Figure 15.6. The pit design includes two (2) pit bottoms, the north pit being larger and the south pit being smaller. It is approximately 2.2 km long by 1.1 km wide and reaches a depth of 510 m in the north pit centered on block 4 and 370 m in the south pit

centered on blocks 5 and 6. The final pit design has two (2) exits in order to facilitate access to the crusher and waste management facility and to allow mining each section of pit independently. The upper section of the ramp in the south will be shared with underground mining trucks as the underground portal will be located within it.

The pit design includes a diversion ditch along its southwest wall to redirect water coming from the watershed immediately west of the pit.

The box cut required for the fresh air raise of the underground mine is located outside of the main pit, to the southeast.

Figure 15.7: Final Pit Design Plan View



Source: GMS, 2025.
**Note: Not to scale.*

15.2.8 Open Pit Mine Reserves Statement

The Mineral Reserve and stripping estimates are based on the final pit design presented in the previous section. It includes the diversion ditch on the southwest wall and the box cut required for the fresh air raise.

Table 15.7 presents a reconciliation of the Mineral Resources and Mineral Reserves contained within the pit design. Waste tonnage (Table 15.8) includes Inferred Mineral Resources.

No Proven Mineral Reserve can be reported as no Measured Resources are declared.

The Probable Mineral Reserves are inclusive of mining dilution and ore loss. The total ore tonnage before dilution and ore loss is estimated at 58.7 Mt at an average grade of 1.67 g/t Au for 3,159 koz. The dilution envelope around ore blocks results in a dilution tonnage of 8 Mt with an average gold content of 0.18 g/t Au for 47 koz. This dilution tonnage represents 14% of the ore tonnage before dilution. Ore Loss is calculated with cut-off re-applied on those diluted blocks, representing 4.4 Mt at 0.36 g/t Au for 50 koz. This gives a total of 62.4 Mt of Probable Mineral Reserve with an average grade of 1.57 g/t for 3,156 koz.

Table 15.7: Resource to Reserve Reconciliation

Resource to Reserves Reconciliation Open Pit	Tonnage (kt)	Grade (g/t Au)	Contained Gold (koz)
Ore Before Dilution and Ore Loss	58,696	1.67	3,159
Add Dilution	8,052	0.18	47
Less: Ore Loss	-4,374	0.36	-50
Probable Mineral Reserve	**62,374**	**1.57**	**3,156**

Table 15.8: Oko Open Pit Mineral Reserves and Quantities

Final Pit Quantities	
Probable Mineral Reserve	
Probable Mineral Reserve	62,374 kt
Gold Grade	1.57 g/t
Contained Gold	3,156 koz
Waste Material (including Inferred Mineral Resources)	
Waste Tonnage	426,232 kt
Total Tonnage	**488,607 kt**

15.3 Underground Mining

15.3.1 Mineral Reserve Calculation Methodology

Underground Mineral Reserves have been estimated applying mining considerations to the Mineral Resource block model. The block model was provided by GMS and was imported into Deswik CAD™ software as a single block model. The model provided was a 5m x 5m x 5m parent block model and sub-blocked to a minimum of 2.50 m x 1.25 m x 2.50 m block size. The evaluation of Mineral Reserves for the Oko Gold FS includes the Mineral Resources categorized as Measured and Indicated. Deswik's stope optimizer was used as a first pass to determine economic zones for extraction by using different stope optimization parameters and Equivalent Linear Overbreak Slough (ELOS) dilution on hanging wall and footwall. The output of the optimizer was verified to remove areas that could not be deemed economic. Stope economics in the new sections of the orebody were then calculated with consideration for potential capital development requirements by zone to ensure profitability. The data of stope shapes that have a reasonable expectation of economic extraction were then adjusted for external backfill dilution and mining recovery. The results were compiled to form the Mineral Reserve estimate.

15.3.2 Dilution and Mining Losses

Dilution parameters were assigned to each stope to estimate the additional dilution experienced during mining operations. A 0.5 m Equivalent Linear Overbreak Slough (ELOS) was applied to the stope hanging wall. For the footwall, from 2.0 m to 0.5 m ELOS dilution was applied depends on proximity with the LBXPMZ structure, as recommended by Alius. To reflect dilution caused by backfill, an additional dilution factor was applied after the stope optimization process, along with a mining recovery of 95%. Table 15.9 shows the ELOS dilution for the footwall depending on the proximity of the structure named LBXPMZ. Table 15.10 shows the various backfill dilution factors used depending on the stope type while Figure 15.7 illustrates the different stope types for transverse stoping to reflect the multiple possible situations.

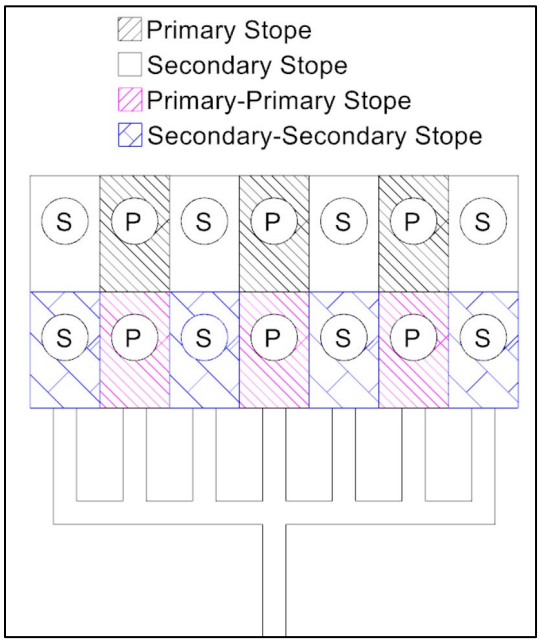
Table 15.9: Footwall Dilution

Parameters	Units	Value
±2 m from RQD < 10-20	m	2.0
±2 m from RQD < 40-50	m	1.0
Elsewise	m	0.5

Table 15.10: Underground Mine Backfill Dilution Parameters

Parameters	Units	Value
Sublevel Transverse Stoping		
Primary Stope	%	2.0
Secondary Stope	%	7.0
Primary - Primary Stope	%	3.0
Secondary - Secondary Stope	%	9.0
Sublevel Longitudinal Stoping		
Primary Stope	%	5.0

Figure 15.8: Underground Mine Transverse Stope Types



Source: GMS 06-06-2024 (not to scale).

Table 15.11 indicates the parameters used for mining recovery for all mining methods.

Table 15.11: Mining Recovery by Mining Method

Parameters	Units	Value
LHOS - Transverse	%	95
LHOS - Longitudinal	%	95
Development	%	100

15.3.3 Underground Cut-Off Grades and Optimization Parameters

The break-even cut-off grade (COG) is the minimum mineral or metal content in the mineralized material for its extraction and processing to be profitable. It represents the grade at which the costs of extraction, processing, and refining would be equal to the revenues derived from the selling price of the valued commodity. To evaluate the Potentially Economical Portion of the Mineral Resource Estimate, a cut-off grade was calculated for the selected mining method (LHOS). Table 15.12**Erreur ! Source du renvoi introuvable.** identifies the parameters for fresh rock used to estimate the mine's cut-off grade for reserves, and Table 15.13 identifies the detailed mining operating cost (OPEX) estimation for underground mining.

Table 15.12: Underground Mine Cut-offs Calculation Parameters

Parameters	Units	Value
Gold Price	*$/oz*	1,800
Discount Rate	*%*	5.0
Royalty Rate	*%*	3.0
TC & RC		
Transportation & Refining	*$/oz*	8.00
Payable Metal Au	*%*	99.95
Mill Recovery		
Average Mill Au Recovery	*%*	93.0
Mineralized Material Based Costs		
Mine Operating Cost - LHOS	*$/t milled*	51.85
Processing Cost (Fresh Rock)	*$/t milled*	9.01
Power Cost	*$/t milled*	11.77
General & Administration Cost	*$/t milled*	10.00
Cut-Offs		
Cut-Off Value (No Sustaining Capital)	***$/t milled***	**82.63**
Cut-Off Grade (No Sustaining Capital)	***g/t***	**1.70**

Table 15.13: Underground Mine Detailed LHOS Mining Cost

Parameters	Units	Value
Definition Drilling & Geology	$/t	3.15
Stope Preparation	$/t	5.30
Drilling & Blasting	$/t	3.35
Mucking & Hauling	$/t	5.65
Backfilling	$/t	12.70
Mine Supervision	$/t	3.40
Technical Services	$/t	3.90
Mine Services	$/t	7.60
Maintenance Services	$/t	4.10
Electrical Services	$/t	2.70
Total Mining Cost – LHOS	**$/t**	**51.85**

A zone-by-zone approach was used to validate the economic viability of the various zones by including their specific sustaining capital cost in the economic analysis.

A series of iterations with the stope optimizer tool of Deswik™ software were performed to obtain the best possible stope shapes. The stope geometry and cut-off grade parameters used in the stope optimizer tool are summarized in Table 15.14.

Table 15.14: Underground Mine Stope Optimizer Parameters

Parameters	Units	Value
Stope Height	m	30
Strike Length	m	20
Maximum Mining Width (HW to FW)	m	30
Minimum Mining Width (HW to FW)	m	5
HW Dilution	m	0.5
FW Dilution	m	2.0 to 0.5
Minimum Dip	°	50
Crown Pillar Thickness	m	30
Side Ratio (Top-Bottom)	-	2.25

Parameters	Units	Value
Side Ratio (Front-Back)	-	2.25
Cut-Off Grade	g/t	1.70

15.3.4 Underground Mineral Reserve

The underground Mineral Reserve estimates are based on the final underground mine design presented in Chapter 16. Table 15.15 presents a summary of underground Mineral Reserves.

Table 15.15: Mineral Reserves Statement

Parameters	Units	Value
Undiluted Stope		
Tonnage	kt	13,084,401
Grade	g/t Au	3.58
Au content	oz	1,507,569
ELOS Dilution (Footwall & Hanging Wall)		
Tonnage	t	882,009
Grade	g/t Au	1.31
Au content	oz	37,130
Additional Ore Dev		
Tonnage	t	316,784
Grade	g/t Au	1.51
Au content	oz	15,405
Backfill Dilution		
Tonnage	t	592,959
Mining Loss		
Tonnage	t	695,122
Grade	g/t Au	3.30
Au content	oz	73,699
Mining Reserve (Diluted & Recovered)		
Tonnage	**t**	**14,181,032**
Grade	**g/t Au**	**3.26**
Au content	**oz**	**1,486,405**

15.3.5 Factors that May Affect the Mineral Reserves

Factors that may affect the Mineral Reserves include:

- Geological complexity, geological interpretation, and Mineral Resource block modelling.
- Commodity prices, market conditions and foreign exchange rate assumptions.
- Long-term consumables price assumptions.
- Operating cost assumptions.
- Sustaining capital costs.
- Rock quality and geotechnical constraints, dilution and mining recovery factors.
- Hydrogeological assumptions.
- Metallurgical process recoveries.
- Mining recovery and dilution assumptions.
- There are no other environmental, legal, title, taxation, socioeconomic, refining, political or other relevant factors known to the QP that would materially affect the estimation of Mineral Reserves that are not discussed in this Report. It is reasonably expected that all necessary government approvals will be issued for the project to proceed.

16. MINING METHODS

16.1 Summary

The Oko West Project is planned as a mining operation that integrates both conventional open pit (OP) mining and mechanized long hole open stoping for the underground (UG) mine. The nominal milling rate will be initially set at 7.0 Mtpa to treat a blend of hard rock, saprolite and transition ores during the open pit operational period. When processing only hard rock, the process plant is designed to nominally treat 6.0 million tonnes per annum. The ramp-up period is five (5) months, and the mill will operate for 12.3 years. The milling process is designed to operate for 13 years with stockpiles peaking at 2.8 Mt by Year 2 to maintain consistent mill feed.

The OP operation will utilize a fleet of diesel-powered equipment, including drills, haul trucks and hydraulic shovels. The Project consists of a main pit that is deeper and centred on Block 4, with one (1) smaller sub-pit located south of the southern extension to the main pit. The OP operation will be executed in four (4) phases. The OP peak mining rate is 44.0 Mtpa over a Life-of-Mine (LoM) of 13 years. A total of 62.4 Mt of ore material will be mined at an average diluted gold grade of 1.57 g/t Au. A total of 426 Mt of combined waste and overburden will be extracted, resulting in a strip ratio of 6.8 tonnes of waste per tonne of ore material. The primary production equipment includes 22 m^3 diesel-hydraulic shovels paired with 140-t off-highway mining trucks. The mining operation is planned to be owner-operated, with an exception for explosives handling and blasting services. Pre-production mining is scheduled over approximately 24 months to secure construction material and to remove overburden to allow access to the ore material. A total of 47 Mt of waste as well as 2.6 Mt of ore material will be mined during the pre-production and commissioning period.

The underground (UG) operation comprises a single mine divided into two (2) zones: the main zone and the satellite zone. Both zones are accessible through a common portal located in the south pit and connected via the main ramp. The chosen mining method is long hole open stoping (LHOS), with both transverse and longitudinal stoping techniques being employed.

The LoM for the UG mine is expected to be 13 years, including construction, development, pre-production, and the full production period. Over this LoM, the UG mine is expected to be in production for 10 years, including a 2-year ramp-up period. A two-year pre-production period is planned to allow sufficient underground development to be completed in order to sustain full production. The UG mine is expected to achieve an average production rate of 4,625 tpd of mineralized material, with 4,500 tpd derived from stope production and an average of 125 tpd from lateral development. Development of the UG mine includes approximately 51.8 km of lateral and 2.6 km of vertical development to be excavated. A total of 14.2 Mt of

ore material is expected to be mined at an average diluted gold grade of 3.26 g/t Au. The primary production equipment includes 21-t diesel-powered load-haul-dump machines (LHD) coupled with 63-t underground mining trucks to handle all mined material.

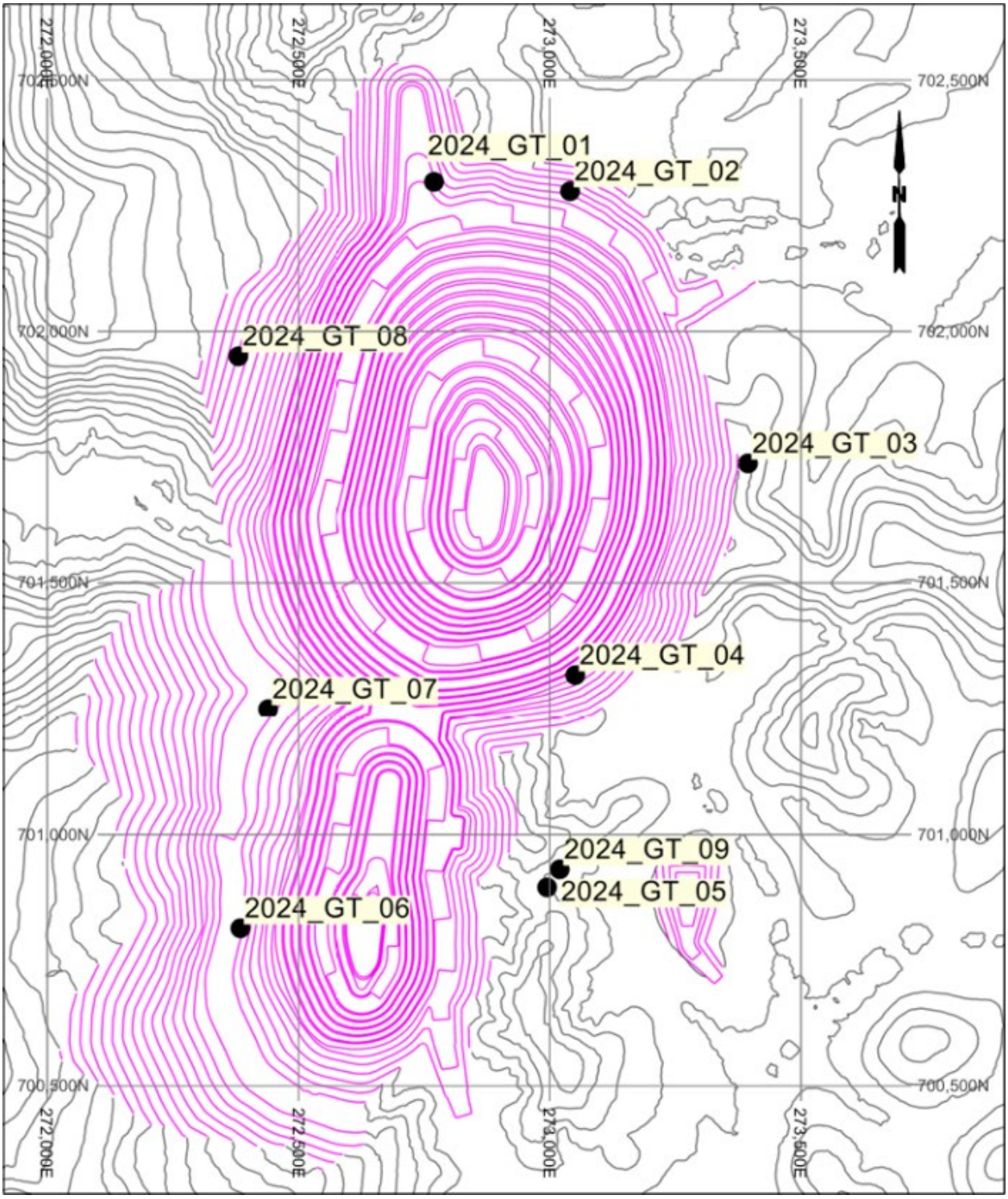
16.2 Geotechnical Considerations

Alius Mine Consulting Inc. (“Alius”) conducted a geotechnical evaluation of the Project that included data from two (2) geotechnical field investigation programs designed to characterize the geotechnical conditions of the residual soils and rock mass in the vicinity of the open pit (OP) slopes and the underground (UG) mine. The various elements of the geotechnical evaluation and findings are presented in the following sections.

16.2.1 Geotechnical Field Campaigns

A geotechnical field campaign was completed during the spring and the summer of 2024 for soil characterization purposes (saprolite). The campaign included soil descriptions by TEC3 and classification of nine (9) drill holes, including seven (7) drill holes in the vicinity of the open pit. The drill hole locations of the field campaign are presented in Figure 16.1.

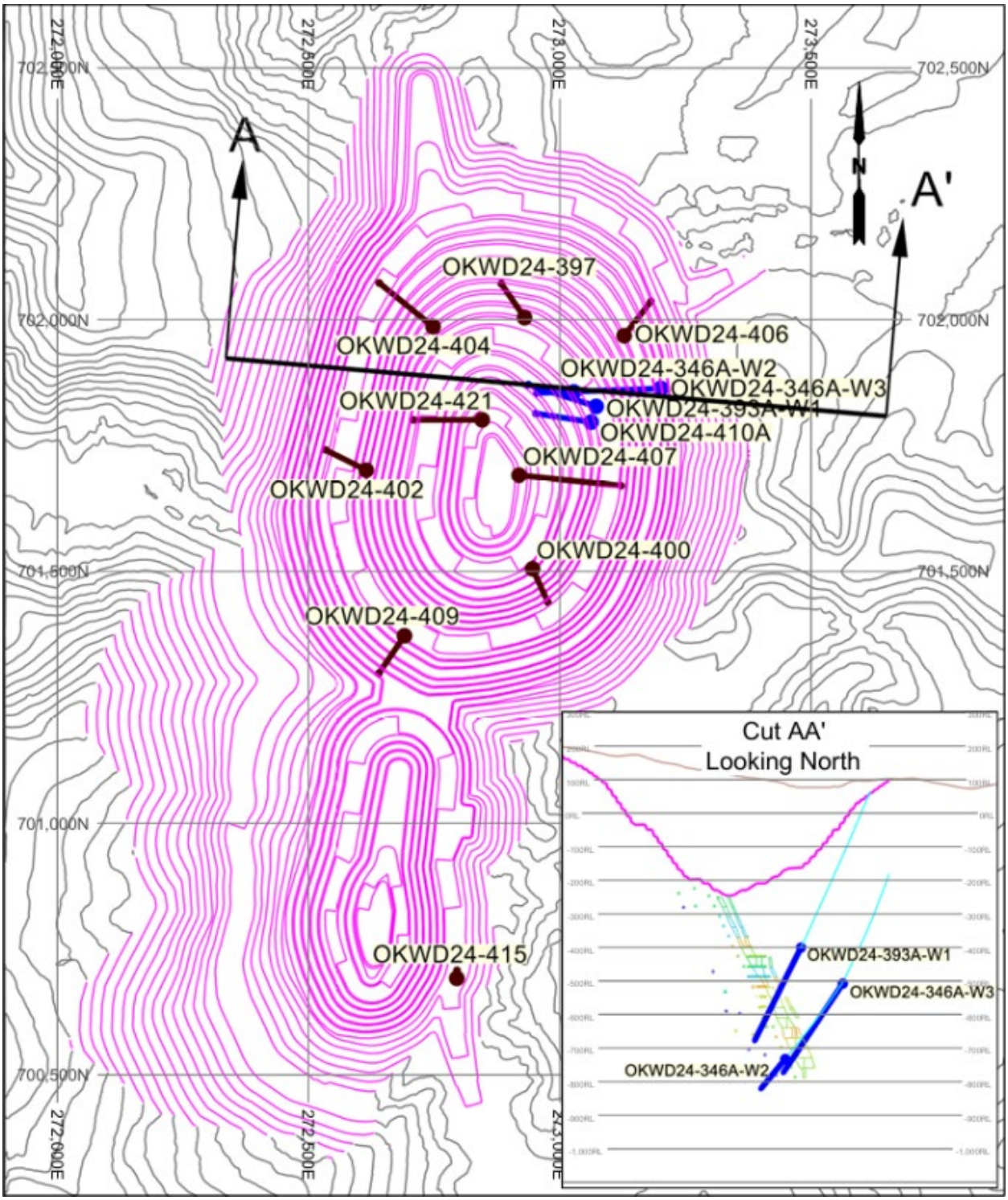
Figure 16.1: Plan View of Open Pit and Soil Geotechnical Drill Holes



Another geotechnical field campaign was completed during Spring and Summer 2024 to characterize the rock mass. The campaign included drilling of 3,270 metres of triple tube NQ3 oriented core and logging of

2,440 metres of rock core, with the involvement of Alius for ensuring quality assurance and quality control. The field campaign is presented in Figure 16.2.

Figure 16.2: Plan View of OP & UG Rock Geotechnical Drill Holes



16.2.2 Residual Soil and Rock Mass Characterizations

Residual soil strength was assessed using Standard Penetration Tests (SPT), and soil samples were collected from split tubes and Shelby tubes and sent to Baitali for laboratory testing in Guyana. A total of 88 tests were conducted to determine particle size distribution, unit weight, moisture content, consolidation properties, Atterberg limits and consolidated undrained triaxial compressive strength (CIU). Soils (saprolite) were analyzed based on their protoliths as well as their degree of weathering and consistency.

The saprolite generally consists of inorganic clays of high plasticity, classified as CH according to the Unified Soil Classification System (USCS). The material composition is approximately 40% clay, 30% silt and 30% gravel and sand. The final interpreted strength properties of the saprolite, based on CIU laboratory test results, are summarized in Table 16.1.

Table 16.1: Interpreted Strength Properties of Saprolite

Material	Protolith	Interpreted Strength Properties	
		Friction Angle (°)	Cohesion (kPa)
Saprolite	MBAS	18	94
	GRAHW & GRAFW	23	102

Field data collection to characterize the rock mass included intact rock properties and discontinuity (bedding, foliation, joints, fault and shear zones) descriptions to derive GSI, RMR_{76} and Q' classifications. Relevant information included: RQD, fracture frequency, rock strength index and structure orientation (dip and dip direction) as well as point-load data. Representative samples of each rock domain were collected from the geotechnical drill holes for laboratory testing on intact rock and discontinuities at the Alius rock mechanics laboratory in Quebec City, Canada. A total of 135 laboratory tests were performed, including point-load testing (PLT), Brazilian tensile strength (BTS) tests, uniaxial compressive strength (UCS) tests, triaxial compressive strength (TCS) tests, and direct shear tests (DST) on discontinuities. Table 16.2 presents the intact rock strength properties of all relevant lithologies.

Table 16.2: Summary of Intact Rock Strength Properties of Main Lithologies

Lithology	UCS		BTS		TCS	Hoek Brown intact rock properties	
	Nb.	Average (MPa)	Nb.	Average (MPa)	Nb.	σ_{ci} (MPa)	m_i
GRAFW	9	138	9	10	9	117	14.0
GRAHW	6	126	6	10	10	158	18.7

Lithology	UCS		BTS		TCS	Hoek Brown intact rock properties	
	Nb.	Average (MPa)	Nb.	Average (MPa)	Nb.	σ_{ci} (MPa)	m_i
MBAS	4	189	2	8	4	191	25.6
MSED1	11	110	6	8	6	119	17.5
VCLX	9	130	5	11	7	103	11.1

*Notes: 1. Nb Number of tests; σ_{ci} intact rock unconfined compressive strength; m_i Hoek-Brown intact rock constant.
2. UCS results for intact or combined failure types only.
3. BTS and TCS results for valid tests only.*

Table 16.3 summarizes the rock mass classifications for all litho-structural domains. These domains generally correspond to lithologies, except for HWUT and LBXPM Z, which represent deformation zones encompassing multiple units. The LBXPMZ was modelled implicitly by GMIN along the footwall contact (Block 4) as two (2) wireframes, which correlate well with zones exhibiting RQD values of 10-20% and 40-50%, respectively, also corresponding to the areas of lower RMR and Q' values. Given that the LBXPMZ fractured zone is generally expected to impact footwall and hanging wall stability of stopes, it was incorporated into stope sizing and dilution estimates, as well as ground support considerations (Sections 16.2.5.2 and 16.2.5.4). Rock mass classifications were assessed by lithology and depth and found to remain consistent with depth, leading to their categorization by litho-structural domain only.

Table 16.3: Rock Mass Classifications

Litho-Structural Domain	Rock Mass Rating			Q' Index		
	$RMR_{76,P25}$	$RMR_{76,P50}$	Quality	Q'_{P25}	Q'_{P50}	Quality
GRAFW	84	87	Excellent	49	100	VG
GRAHW	83	89	Excellent	40	96	Good–VG
HWUT	82	87	Excellent	25	34	Good
LBXPMZ	39	65	Poor–Good	0.1	1	VP–Poor
MBAS	82	92	Excellent	79	200	VG–EG
MSED1	80	87	Excellent	33	99	Good–VG
VCLX	78	87	Good–Excellent	26	100	Good–VG

*Notes: 1. P25 and P50 refer to the 25th and 50th percentiles of the cumulative distribution function.
2. $Q' = RQD/J_n \times J_r/J_a$.
3. Q' Quality: EG extremely good; VG very good; VP very poor, assuming $J_w/SR = 1$.*

16.2.3 Stress State

At this stage, in-situ stress conditions have been estimated using the World Stress Map, given the limited availability of local stress data in Guyana. The major principal stress (σ_1) is assumed to be horizontal and oriented perpendicular to the orebody, with a magnitude of 1.5 times the vertical stress component (σ_v). The intermediate principal stress (σ_2) is assumed to be horizontal and parallel to the orebody, with a magnitude of 1.3 times σ_v. The minor stress (σ_3), set as the vertical stress, is calculated from the material weight derived from the block model. Unit weight for the overburden surrounding the pit area is set to 27 kN/m^3. Table 16.4 summarizes the in-situ stress tensor considered for the project.

Table 16.4: In-Situ Stress State

Horizontal / Vertical Stress Component	Principal Stress Component	Magnitude Gradient (MPa)	Orientation	
			Trend (°)	Plunge (°)
Horizontal max (σ_H)	Major (σ_1)	0.041 × Z	090	00
Horizontal min (σ_h)	Intermediate (σ_2)	0.035 × Z	180	00
Vertical (σ_v)	Minor (σ_3)	0.027 × Z	--	90

**Note: Z = depth below surface, in metres.*

16.2.4 Open Pit Mining

The following sections present the methodology to derive pit slope design recommendations.

16.2.4.1 Open Pit Geotechnical Domains

Open pit geotechnical domains are based on the rock mass classifications, structural orientations interpreted from stereographic projections, laboratory-derived strength envelops, and expected rock mass behaviour within the main weathering domains (saprolite, transition and fresh rock).

For the Oko West Project, geotechnical domains generally align with lithological units, as geomechanical properties are relatively consistent across the site. However, two additional domains were delineated because of their fractured nature. Details of the open pit geotechnical domains are provided in Table 16.5.

Table 16.5: Open Pit Geotechnical Domains

Geotechnical Domain	Weathering Domain	Lithological Units	Considered Boreholes
GTD-GRA	Saprolite	GRAFW, GRAHW	GT-02, GT-03, GT-04
GTD-BAS		MBAS	GT-06, GT-07, GT-08
GTD-TZ	Transition zone	All	GT-01, GT-02, GT-03, GT-04, GT-06, GT-07, GT-08
GMD-GRAFW	Fresh rock	GRAFW	OKWD24-402, OKWD24-404, OKWD24-409, OKWD24-420
GMD-GRAHW		GRAHW	OKWD24-400, OKWD24-406, OKWD24-407
GMD-MBAS		MBAS	OKWD24-402, OKWD24-404, OKWD24-409
GMD-SED		MSED1, VCLX	OKWD24-397, OKWD24-404
GMD-HWUT	Fresh rock (fractured)	Multiple	OKWD24-393A-W3, OKWD24-397, OKWD24-410A
GMD-LBXPMZ		Multiple	OKWD24-346A-W3, OKWD24-393A-W3, OKWD24-409, OKWD24-410A, OKWD24-420

**Notes: N/A = Not applicable; GTD = Geotechnical Domain (saprolite & transition); GMD = Geomechanical Domain (fresh rock).*

Discontinuity sets were also determined for each geotechnical domain, based on the relevant borehole. For the open pit geotechnical analyses, only discontinuities logged above a depth of 400 m were used, while structures located below this depth were used in the underground design analyses. The major discontinuity sets identified and relevant to further analyses are presented in Table 16.6.

Table 16.6: Major Discontinuity Sets Identified per Geotechnical Domain

Geotechnical Domain	Discontinuity Set 1		Discontinuity Set 2		Discontinuity Set 3	
	Dip (°)	Dip Direction (°)	Dip (°)	Dip Direction (°)	Dip (°)	Dip Direction (°)
GMD-GRAFW	75	094	81	136	62	057
GMD-GRAHW	37	181	75	286	55	310
GMD-MBAS	74	059	-	-	42	076
GMD-SED	49	210	-	-	-	-

**Notes: Values represent the average pole.*

16.2.4.2 Slope Stability Analyses

In the saprolite weathering domain, the material exhibits high cohesive properties, making slope height a key factor influencing overall stability. Due to variations in residual soil thickness across the open pit area, domain GTD-BAS has been divided into two (2) sectors based on saprolite thickness, which exceeds 120 metres in some locations.

For the saprolite, slope stability assessments were conducted using 2D limit equilibrium analyses, assuming the implementation of proactive water management measures to mitigate instability and excessive erosion. These measures may include diversion channels designed to maintain proper drainage within the transition zone layer, as well as bench sumps to help manage water on saprolite slopes. Recommendations for the transition weathering domain (GTD-TZ) rely on operational experience and benchmarking in South American mines, as this domain presents challenges in both assessment and testing.

Open pit benches were designed using a risk-based approach to address the limitations of a deterministic approach. By integrating geotechnical logging data with the proposed pit shell, the probability of occurrence and the probability of sliding were derived for wall orientations (dip and dip direction) to determine the bench-scale probability of failure (PoF). Using estimated failure volumes, catch bench widths (BW) were optimized to an established risk level to contain material spills, where risk is defined as a function of both PoF and proportion of spills exceeding containment capacity. Given that steeper benches typically generate larger failure volumes, narrower BWs may be insufficient for containment. Establishing a risk tolerance threshold enables the optimization of inter-ramp angles (IRA) in a structurally controlled rock mass by balancing bench face angles (BFA) and bench widths, resulting in a more realistic design, as some failures may be operationally manageable. Proper mining practices and slope geometry, particularly adequate catch bench widths, are required to not compromise workspace safety by effectively containing spills. This risk-based approach for bench-scale analyses is appropriate when rock mass failure is expected to be structurally controlled (strong rocks), as is the case at Oko West.

Using the recommended slope design guidelines, overall pit slope stability was assessed using 3D numerical modelling (finite-volume approach). The model incorporated the different lithologies, as well as the multiple pit phases to assess the factor of safety, for both static and pseudo-static conditions, considering fully drained slopes.

16.2.4.3 Slope Design Guidelines

Design guidelines have been established for the various pit sectors, as shown in Figure 16.3. Table 16.7 presents the slope design guidelines for each pit sector. These guidelines include three (3) sectors for the

saprolite weathering domains, one (1) sector for the transition zone weathering domain and six (6) sectors for the fresh rock weathering domain.

Due to the overall excellent quality and strength of the fresh rock, slope stability is expected to be controlled primarily by structural features rather than rock mass strength. Consequently, domains A, B and D have lower inter-ramp angles to mitigate risks of planar failures. Additionally, 16 m wide geotechnical berms are recommended in rock slopes at vertical intervals of 180 m, to reduce overall slope angles and provide additional design contingency.

Figure 16.3: Open Pit Sectors

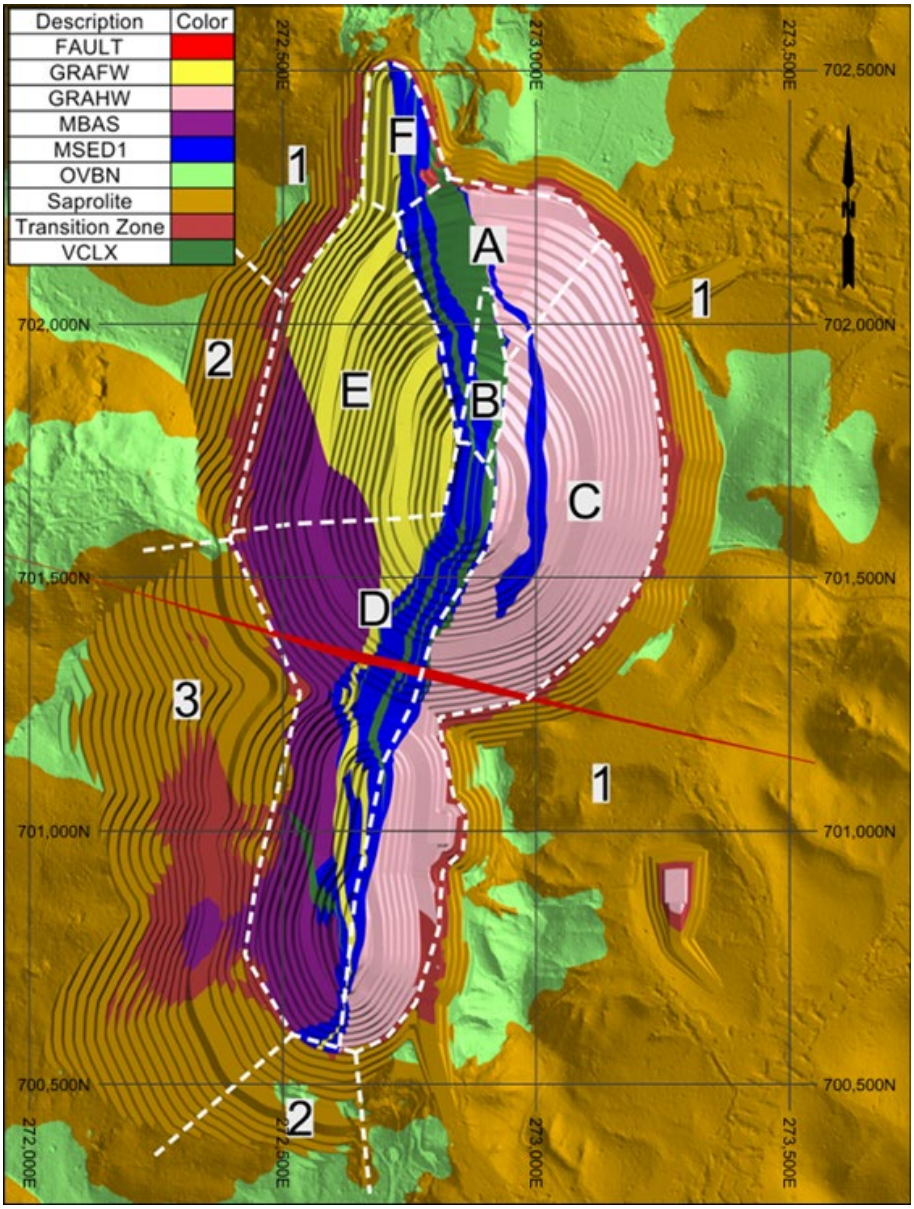


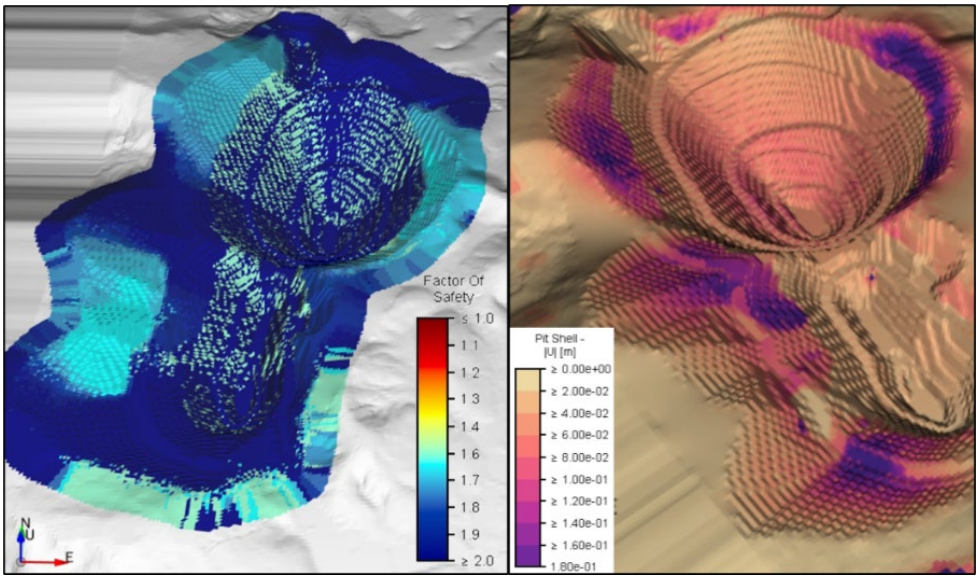
Table 16.7: Open Pit Slope Design Guidelines

Pit Sectors	Geotechnical Domains	Az Range (°)	BH (m)	BW (m)	BFA (°)	IRA (°)	Geotechnical Berm
1	GTD-BAS	N/A	10	13.0	60	28	N/A
2	GTD-BAS	N/A	10	15.5	60	25	N/A
3	GTD-GRA	N/A	10	8.0	60	36	N/A
All	GTD-TZ	0-360	10	8.5	70	40	N/A
A	GMD-GRAHW GMD-SED	90-125 20-100	20	8.5	55	42	16 m; every 180 m
B	GMD-SED	100-180	20	8.5	52	40	16 m; every 180 m
C	GMD-GRAHW	125-285	20	8.5	75	55	16 m; every 180 m
D	GMD-GRAFW GMD-MBAS GMD-SED	305-350 305-350 215-355	20	9.5	70	50	16 m; every 180 m
E	GMD-GRAFW GMD-MBAS	350-060	20	8.5	75	55	16 m; every 180 m
F	GMD-GRAFW GMD-SED	0-360	20	8.5	80	59	16 m; every 180 m

**Notes: BH: bench height; BW: bench width; BFA: bench face angle; IRA: inter-ramp angle.*

The slope design guidelines were assessed through numerical modelling to confirm that the safety factors were adequate. Figure 16.4 presents the static safety factors (FoS) distribution projected onto the final pit walls. All FoS values exceed 1.5, with a minimum of 1.55 observed in the saprolite zone.

Figure 16.4: Static Factor of Safety (left) and Displacements (right) of Final Pit from 3D Model



16.2.5 Underground Mining

The following sections present the methodology to derive underground design recommendations.

16.2.5.1 Underground Geotechnical Domains

Lower and upper bounds of the rock mass classifications, corresponding to the 25th and 50th percentiles of the cumulative distribution functions (CDF), were used for underground design. As the MSED1 and VCLX lithological units intersect multiple times within the mineralized envelope, and both units exhibit similar geomechanical characteristics, the VCLX properties (slightly lower) were assumed for both.

16.2.5.2 Stope Sizing and Dilution Estimates

Stope surface stability was assessed using the empirical stability charts developed by Potvin (1988) and Mawdesley *et al*. (2001), along with the ELOS method (Clark, 1998) for dilution estimation. These approaches consider rock mass classification, induced stress conditions, intact rock strength, joint orientations, as well as stope geometry (hydraulic radius) and orientation.

Analyses were performed for both transverse and longitudinal stopes with two (2) transverse scenarios:

1. Stopes mined entirely within the VCLX unit.
2. Stopes mined in the VCLX unit and located near the LBXPMZ deformation zone.

As summarized in Table 16.8, proximity to the LBXPMZ has a significant influence on dilution. When the stope footwall lies within ± 2 m of the 40–50% RQD wireframe, an ELOS of 1.0 m is estimated. If the footwall is located within ± 2 m of the 10–20% RQD wireframe, an ELOS of 2.0 m is estimated. In cases where both conditions are met, the higher ELOS value of 2.0 m governs.

Table 16.8: Stope Dimensions and Dilution Estimates

Stope Dimensions	Transverse (VCLX)	Transverse (Near LBXPMZ)	Longitudinal (VCLX)
Vertical Height (m)	30	30	30
Strike Length (m)	20	20	30
Width (m)	30	30	10
ELOS – Hanging wall (m)	0.5	0.5	0.5
ELOS – Footwall dilution (m)	0.5	1.0 (RQD 40-50%) 2.0 (RQD 10-20%)	1.0 (RQD 40-50%) 2.0 (RQD 10-20%)

Stope Dimensions	Transverse (VCLX)	Transverse (Near LBXPMZ)	Longitudinal (VCLX)
ELOS – Endwalls (m)	0.0 (ore) 0.5 (CRF)	0.0 (ore) 0.5 (CRF)	0.0 (ore) 0.5 (CRF)
ELOS – Back (m)	0.0 (ore)	0.0 (ore)	0.0 (ore)

16.2.5.3 Backfill

Stopes will be backfilled using cemented rock fill (CRF) to avoid leaving rib pillars. The following CRF strengths are required to ensure lateral stability of the exposed surfaces (either end walls, hanging walls or footwalls) based on the limit equilibrium method proposed by Li & Aubertin (2012). A strength of 470 kPa is required for transverse stopes, while 270 kPa is sufficient for longitudinal stopes.

When development in backfill is needed to extract sill-level stopes, higher binder content will be necessary to ensure stability of the stope above. For underhand exposures, a CRF strength of 2.4 MPa is recommended.

16.2.5.4 Ground Support

Permanent development is planned to be supported on a 1.2 × 1.2 m pattern, using 2.4 m resin rebars (back) and 1.8 m friction bolts (walls), with 4 × 4” 6-gauge mesh. 6-metre cable bolts are planned in intersections. Temporary development, mainly stope draw points and ore sills, is planned to be supported on a 1.2 × 1.2 m pattern, using 1.8 m inflatable bolts (back) and 1.8 m friction bolts (walls), with 4 × 4” 6-gauge mesh. 6-metre cable bolts are planned in intersections. Detailed ground support guidelines are provided in Table 16.9.

Back support for transverse stopes was evaluated for various widths, considering rock mass classification variability. Stopes less than 8 m wide do not require cable bolting or long support. However, for wider stopes (up to 30 m), approximately 70% will require support with longer cable bolts (9 m). On average, all stope backs should be supported using 8 m cable bolts, installed in a 2 × 2 m pattern. Additionally, long-support provisions are advised for stope brows, with flexibility for use in other areas as needed. Finally, stopes with hanging wall dips of less than 60° should be cable bolted. Again, detailed ground support guidelines are provided in Table 16.9.

Ground support for the portal development has not yet been analyzed in detail; however, this will be comprehensively addressed during the detailed engineering phase of the Project.

Table 16.9: Ground Support Guidelines

Excavation Type	Details	Typical Dimensions (m)		Max Span (m)	Back Support		Wall Support		Recommendations for Low RQD (LBXPMZ)
		Height	Width		Type	Details	Type	Details	
Lateral	Decline / ramp	5.0	5.5	6.5	Primary	RB 2.4; 1.2 × 1.2 m	Primary	SS 1.8 m; 1.2 × 1.2 m	Support to floor + SC 75 mm
	Permanent (waste)	5.0	5.0	6.5	Primary	RB 2.4; 1.2 × 1.2 m	Primary	SS 1.8 m; 1.2 × 1.2 m	Support to floor + SC 75 mm
	Temporary (ore)	5.0	5.0	5.0	Primary	RB 1.8; 1.2 × 1.2 m	Primary	SS 1.8 m; 1.2 × 1.2 m	Support to floor + SC 75 mm
	Large Infrastructure	8.0	8.0	8.0	Primary Secondary	RB 2.4; 1.2 × 1.2 m RB 3.0; 2.0 × 2.0 m	Primary Secondary	RB 2.4 m; 1.2 × 1.2 m SS 1.8 m; 1.2 × 1.2 m	Avoid
	In / Under Backfill	5.0	5.0	5.5	Primary	SX 2.4; 1.2 x 1.2 m + SC 75 mm	Primary	SX 2.4; 1.2 x 1.2 m + SC 75 mm	N/A
Intersections	Typical	6.0	8.5	8.5	Secondary	RB 3.0; 2.0 × 2.0 m	-		SC 75 mm + MS 3.0 m
	Atypical	6.0	12.0	12.0	Secondary	9 x CB or SSX 6 m	-		SC 75 mm + MS 3.0 m
Vertical	Escapeway	-	2.0	2.0	-		No support, if lined		Avoid
	Raisebore	-	6.0	6.0	-		No support required. (Use RB 2.1 m; 1.2 × 1.2 m, if needed)		Avoid
Stopes	Brow	-		Refer to stope guidelines	Secondary	9 × CB 6 m	-		SC 75 mm + MS
	Back	-			Secondary	Refer to stope guidelines	-		Support to floor + SC 75 mm

**Notes:*

1. ***CB** 15 mm Cable bolt; **RB** Rebar #6; **MS** #0-gauge mesh-straps ; **SC** Shotcrete ; **SS** Split Set 35 mm; **SX** 12 t Swellex; **SSX** 24 t Super Swellex.*
2. *If secondary support is recommended, primary support must first be added (using the appropriate excavation type recommendation).*
   For example, a 7-m intersection in the ramp requires back support of RB 2.4 m on a 1.2 x 1.2 m pattern (primary), followed by RB 3.0 m on a 2.0 x 2.0 m pattern (as a second pass).
3. *Lateral development: assumes wall support to 1.5 m from floor.*
4. *Lateral development: assumes 1.5 x 2.7 m galvanized welded wire mesh (WWM), #6 gauge.*
5. *Vertical development: assumes chain-link or WWM.*

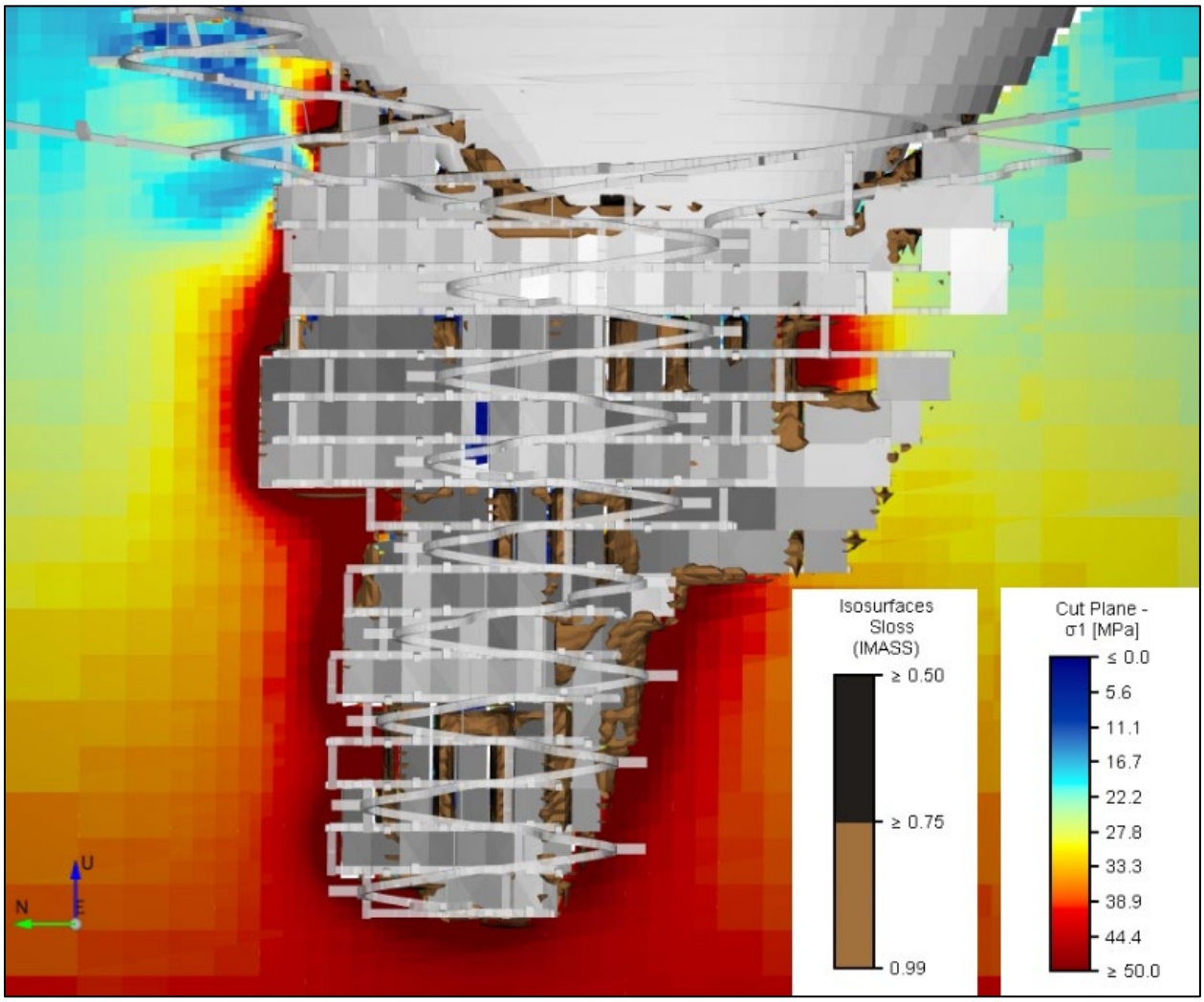
16.2.5.5 Mine Design Considerations

A 3D numerical model (using a finite-volume approach) was developed to assess interactions between the preliminary open pit and underground designs and to establish stand-off distance guidelines for key infrastructure. Following completion of the final mine design, a detailed 3D model was constructed to review the mining sequence, identify areas of elevated stress, and assess potential seismic hazards. Key findings are summarized below:

1. OP and UG Interaction: The proposed mining sequence is designed to minimize interaction between the open pit and underground mine. Stopes located near the final pit outline will be mined and backfilled prior to the pit reaching those areas, resulting in the pit effectively sitting on the backfilled stopes.
2. Stand-Off Distances: Based on the numerical modelling results, a minimum distance of 20 m is recommended between haulage drifts and adjacent stopes, while the ramp should maintain a stand-off distance of 80 m from the stopes.
3. Sequencing and Stress: The underground operation will involve multiple simultaneous mining sequences, mainly using a primary-secondary sequence, to maintain productivity and operational flexibility. Stress concentrations are expected on sill levels, in retreat zones between converging fronts, between lenses, and within waste pillars. These areas may experience spalling, local instabilities, and movement along geological discontinuities. Higher stress levels anticipated at depth may also result in fracturing and potential seismicity, although quantifying the magnitudes is currently limited by the absence of in situ stress measurements.
4. Seismic Hazard: Seismic hazard was assessed by analyzing seismic potency outputs from the 3D model. Due to the strong and stiff nature of the rock mass, the potential for seismic activity is anticipated primarily at depth, in areas of converging mining fronts and around waste pillars. At this stage, the current mine plan does not warrant the implementation of seismic monitoring or dynamic ground support. However, this may need to be revisited depending on the rock mass response as mining progresses.
5. Mine Portal: The portal design has not been assessed in detail; however, this will be comprehensively addressed during the detailed engineering phase of the Project.

Figure 16.5 presents the major principal stress (σ_1) and the damage indicator (Sloss) from the 3D model at the end of 2038. The damaged (Sloss, for strength loss) isosurfaces visible on the figure represent the areas experiencing plastic deformations due to high stress concentrations.

Figure 16.5: Major Principal Stress (σ_1) & Damage Indicator (Sloss) Surrounding Mined Stopes at Year 2038 from 3D Model



16.3 Hydrology and Hydrogeology

16.3.1 Hydrology

Newfields was mandated to establish a hydrologic model of the area surrounding the Oko West pit to assess the stormwater runoff contributing to the ultimate pit area. The evaluation involved the identification of contributing stormwater catchments and the selection of design storm criteria for the peak runoff. Newfields considered a one-in-a-thousand-year stormwater event, which implies 242 mm of precipitation over a 24 h period.

Based on such an event, the hydrological model estimates that a flow of 28.4 m^3/s will need to be managed on the western flank of the pit with a diversion channel. The diversion channel was sized to manage such a volume of water.

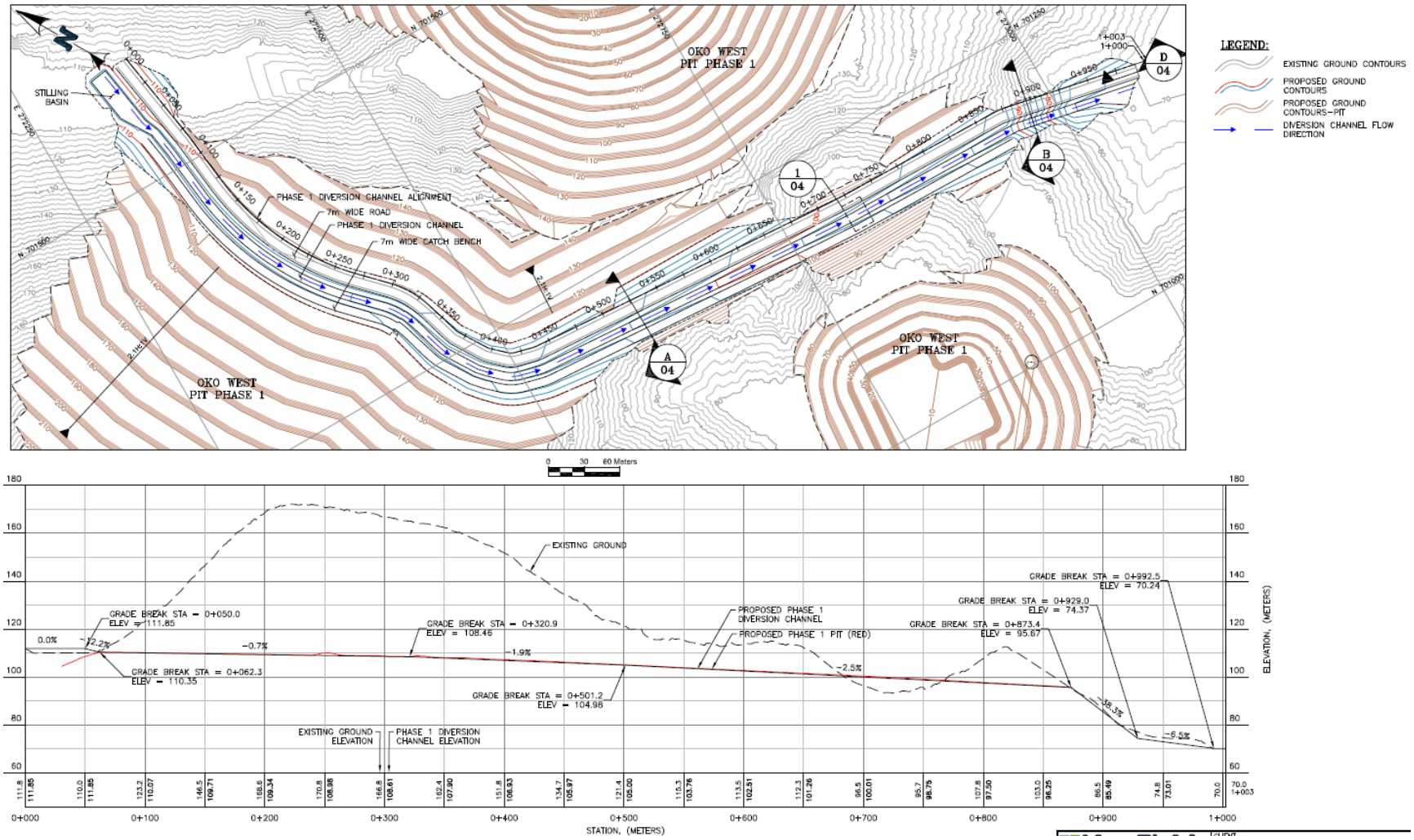
The diversion channel will be constructed in two phases. The first phase will be 1,000 m in length, and the second phase will be 1,700 m in length.

The first phase will start at the intersection between the final pit design and the water catchment area. The first 300 m of length will be common to both phases of diversion channel construction. It will then follow the limit of the phase 2 pit and discharge on the southeast. This allows to pushback excavation of waste material to be delayed until the second year of operations. The slopes of the channel will typically vary from 0.7% to 2.5%, with a 38.0% discharge point, as illustrated in Figure 16.6.

The second and final phase of the diversion channel will follow the western and southern walls of the ultimate pit and will discharge to the southern lobe of the pit. It will be required for the mining of the phase 3 pit. The slope of the channel will typically be 0.7%, with a 28.0% drop at the discharge point as shown in Figure 16.7.

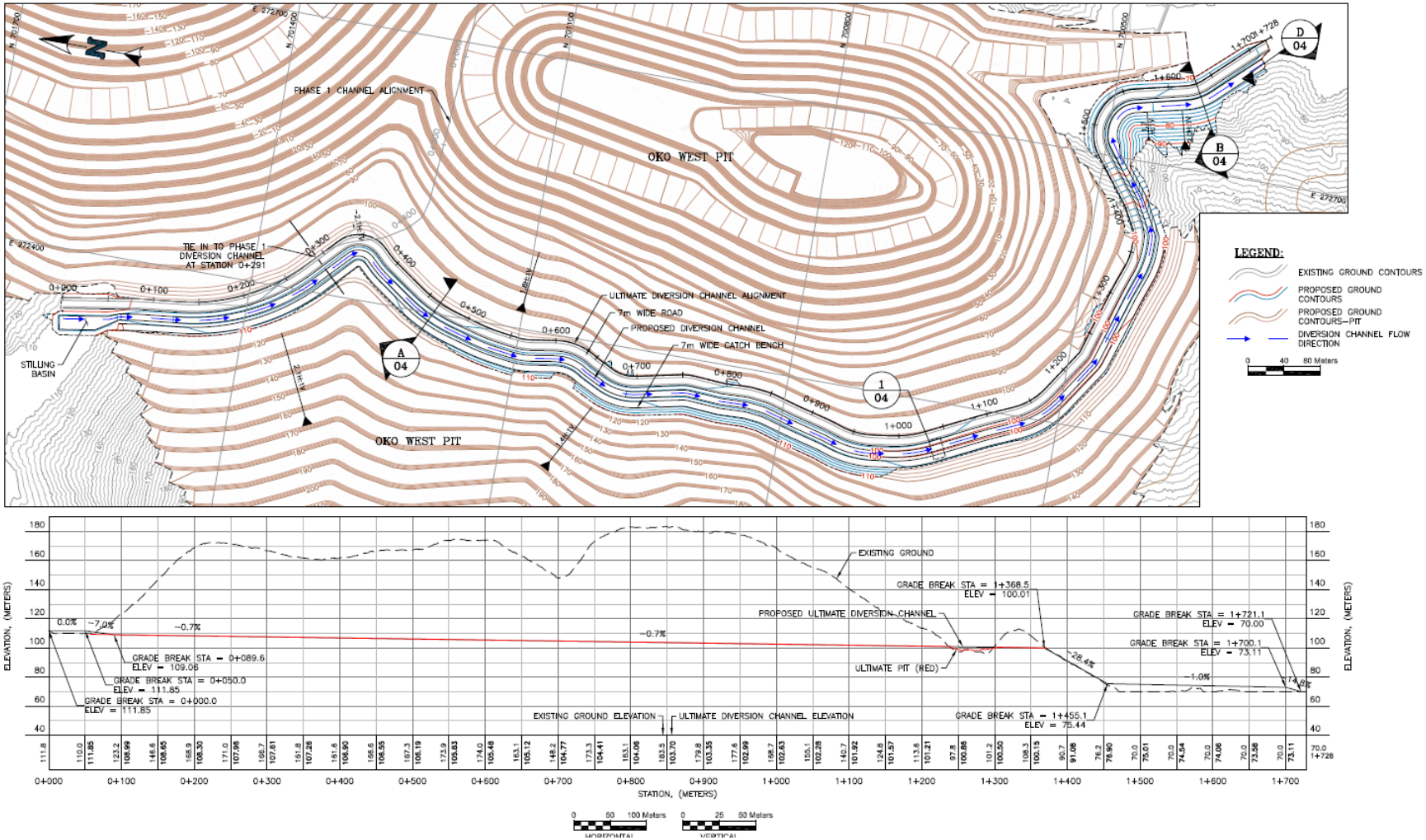
In both phases, the typical design section will include a 7 m wide catch bench, the channel itself, and a 7 m wide road to provide maintenance access along the entire length of the diversion channel, as discussed in Chapter 15. The channel is sized so that the depth of runoff from the design storm is level with the top of the diversion ditch, which is 0.5 m lower than the road. The entire channel will be geomembrane-lined to minimize erosion and seepage into the pit.

Figure 16.6: Plan View of the Phase 1 Diversion Channel Design



Source: Newfields, 2025.
**Note: Not to scale*

Figure 16.7: Plan View of the Phase 2 Diversion Channel Design



Source: Newfields, 2025
**Note: Not to scale*

16.3.2 Hydrogeology

ERM completed a hydrogeological study in December 2024 for the entire site. This study calculates the groundwater inflow volumes for both the open pit and the underground mines. Three (3) scenarios have been completed through a numerical model made with MODFLOW, which have been calibrated with measurements in the field. The three (3) scenarios modelled are for the minimum flow, the likely flow, and the high flow model.

The “likely flow” model was adopted for planning purposes. For this scenario, the water inflow from groundwater was at most 1 000 m³/day.

The assumptions considered for groundwater infiltration into the underground mine are presented in Chapter 16.5.7.2

16.4 Open Pit Mining

16.4.1 Mine Phases

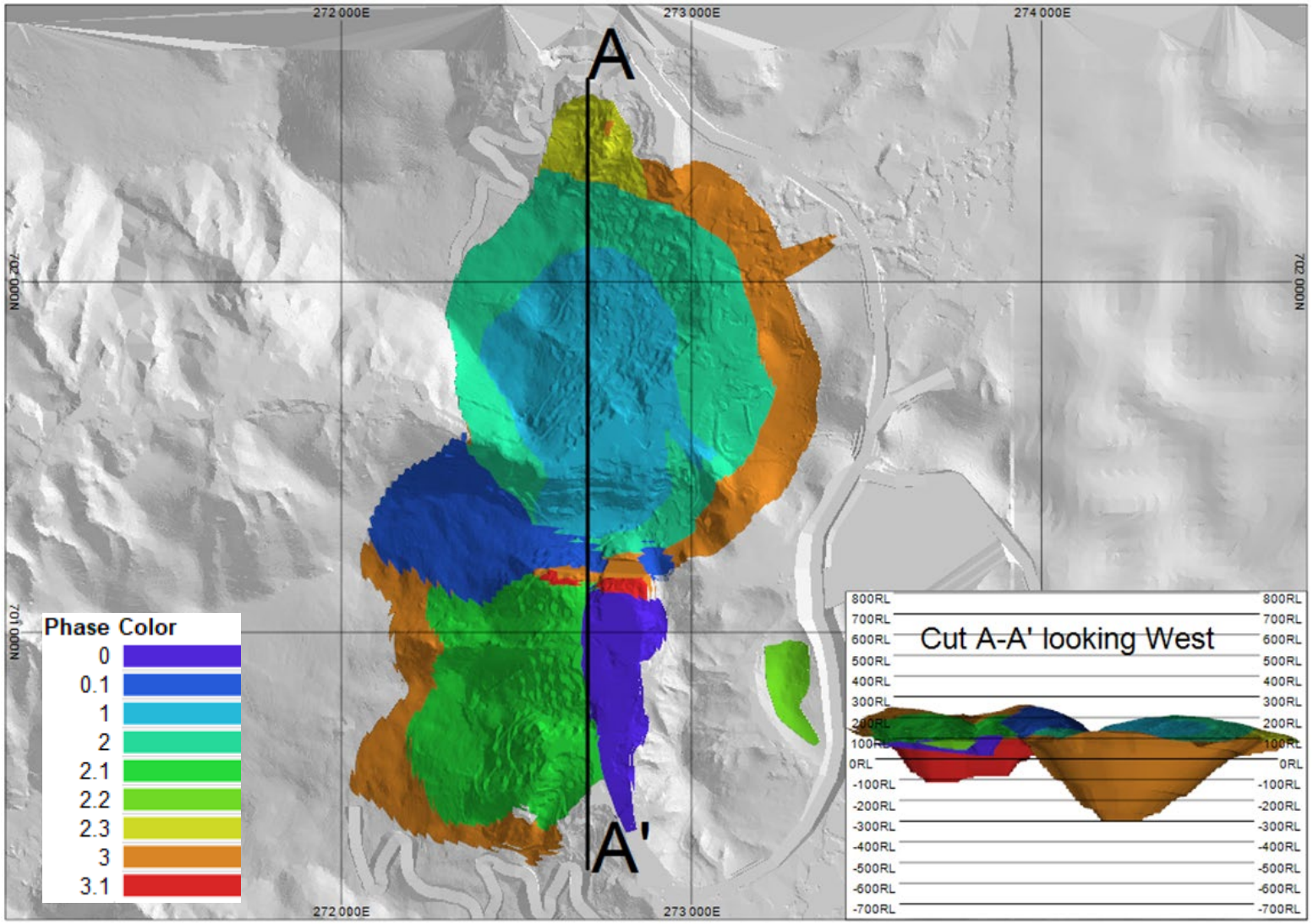
The open pit mine designs were developed using optimal Whittle™ shells. An optimized mining sequence was computed to identify the optimal shells for defining pushbacks and aiming to maximize the project’s Net Present Value (NPV). Shells 5, 15, and 29 were selected as discussed in Chapter 15. The mining of the Oko West open pit is planned to include four (4) phases. Phase 1 was guided by pit shell 5, Phase 2 was guided by shell 15, and Phase 3 was guided by 29. Phase 0 is scheduled during early works: the first phase of the diversion channel and access to the underground mine portal, which provides waste rock for haul road construction. The details of each mining phase are summarized in Table 16.10. The final configuration of the pit is presented in Figure 16.8, which also represents the configuration of the different pit phases and sub-phases.

Table 16.10: Physical Quantities per Open Pit Phase

Mining Reserves by Phase		Phase 0	Phase 1	Phase 2	Phase 3	TOTAL
Total tonnage	kt	13,471	48,727	170,034	256,374	488,607
Waste Tonnage	kt	13,455	32,604	148,518	231,656	426,232
Saprolite Waste	kt	11,429	17,453	49,572	35,176	113,631
Trans waste	kt	904	3,566	15,905	16,458	36,833
Fresh Rock Waste	kt	1,122	11,585	83,041	180,022	275,769
Ore Tonnage	kt	17	16,123	21,516	24,718	62,374

Mining Reserves by Phase		Phase 0	Phase 1	Phase 2	Phase 3	TOTAL
Saprolite Ore Tonnage	kt	16	4,523	2,695	423	7,657
Saprolite Ore Grade	g/t Au	0.88	1.56	0.90	0.66	1.28
Trans Ore Tonnage	kt	0	2,099	740	702	3,541
Trans Ore Grade	g/t Au	-	1.82	0.92	0.81	1.43
Fresh Rock Ore Tonnage	kt	0	9,501	18,082	23,593	51,176
Fresh Rock Ore Grade	g/t Au	0.46	1.82	1.62	1.56	1.63
Strip Ratio (W:O)		813.1	2.0	6.9	9.4	6.8

Figure 16.8: Open Pit Phase Limits



Source: GMS 2025
**Note: Not to scale.*

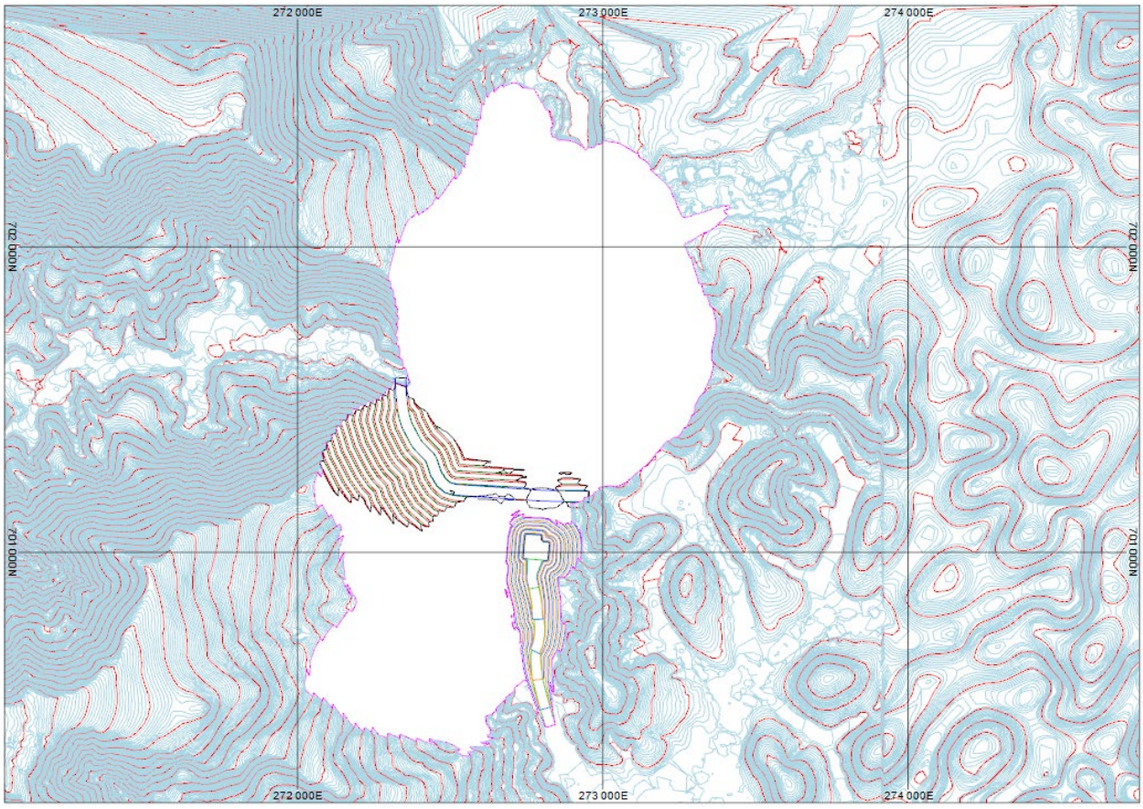
16.4.2 Ramp Designs

The pit is located on the east side of the property, northwest of the planned crusher and mill location. The pit will expand from its centre to the final wall over four (4) main phases. In each phase, the haulage ramp transitions from a double lane to a single lane on the lower benches, optimizing the capture of ore material at the bottom of the pit. Pit exits are planned to the north for the main pit to allow for the shortest haulage distance to the crusher and the waste dump, and to the south for the south pit in order to minimize waste mined. Ramp designs for the mining phases are as follows:

- Phase 0 includes two (2) small pits: the first is required for the diversion channel designed to capture water coming from the west watershed for the initial pit phases (1 and 2); the second was designed to reach the planned underground (UG) portal while minimizing excavated volumes. It also provides access to waste rock for construction requirements. The phase 0 pits feature double-lane haulage ramps.
- Phase 1 is the initial operation phase, starting at the centre of the deposit. The ramp exit is located on the east wall, allowing for the closest route to the crusher. Phase 1 measures 800 m in length, 550 m in width, and reaches a maximum depth of 250 m from the top of the hill to the bottom.
- Phase 2 consists of four different sub pits: the main pit in the centre of the deposit and reaches the final wall on the west side with a ramp exit to the south, with the shortest route to the crusher. On the north end, a small excavation of a single bench is required for an underground return air raise. In the south pit, all of the remaining saprolite ore is mined. Finally, a box cut is required to the south-east near the crusher to allow excavating the fresh air raise for the underground mine. The main pit of Phase 2 is 1,100 m in length, 950 m in width, and reaches a maximum depth of 410 m.
- Phase 3 is the final nested pi. It involves mining the last phase of the diversion channel in order to reach the bottom of the ultimate pit. The main pit is 1,400 m in length and 1,100 m in width, and reaches a maximum depth of 510 m.
- In the south pit, this phase’s ramps progress spiral starting from the underground portal. An area of 35 m in width by 60 m in length is kept at the portal. It is designed to allow both the open pit and underground truck fleets to safely operate simultaneously. The final south pit is 885 m in length and 850 m in width, and reaches a maximum depth of 360 m.

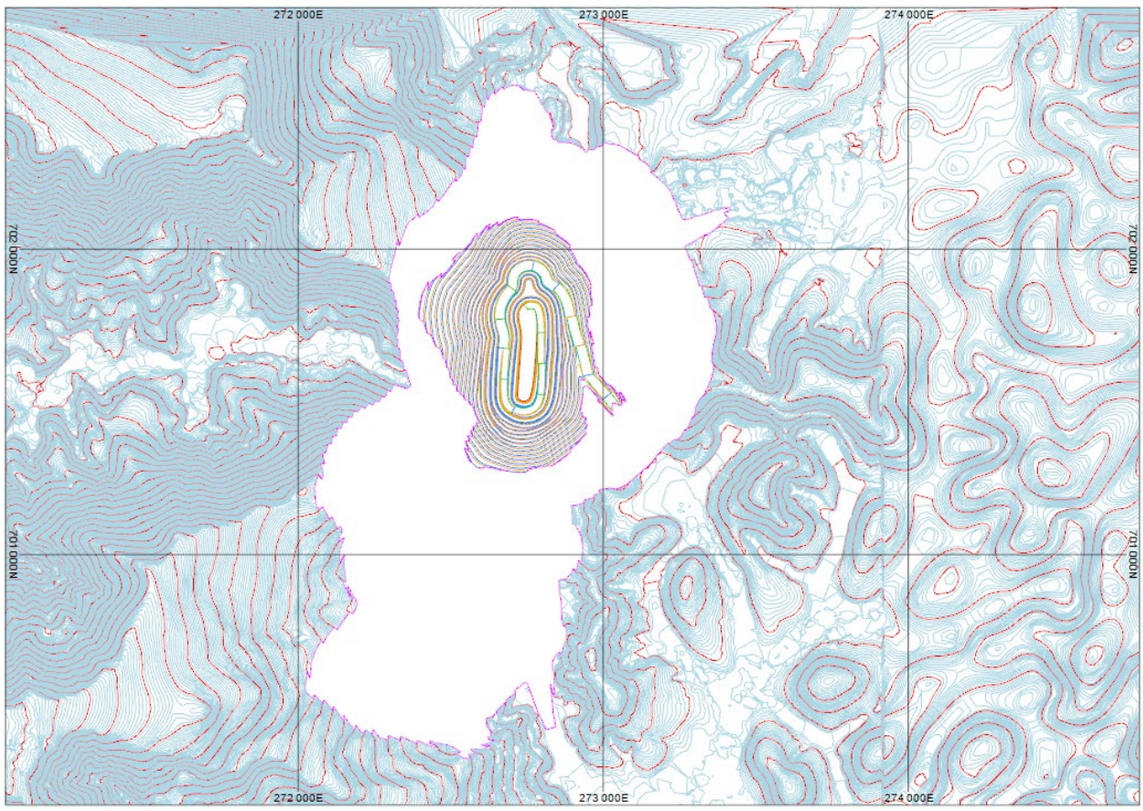
The different pit phases are presented in Figure 16.9 to Figure 16.12.

Figure 16.9: Phase 0 Open Pit Design



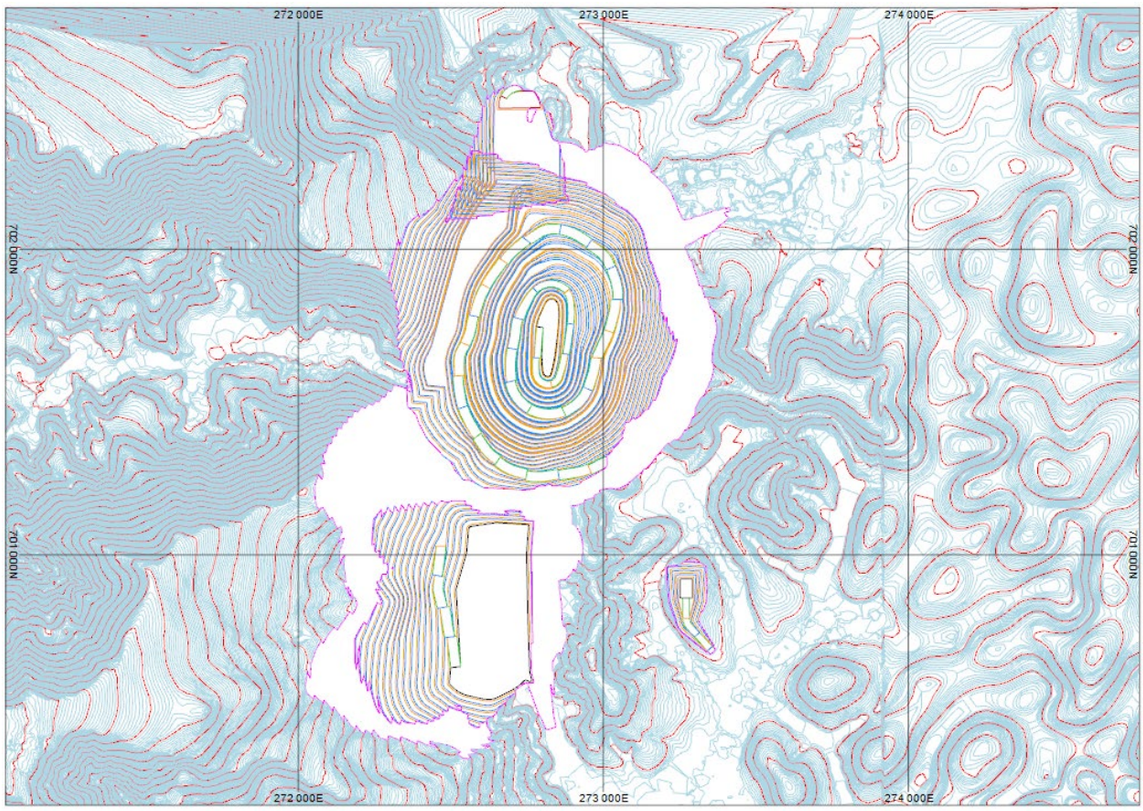
Source GMS, 2025
**Note: Not to scale.*

Figure 16.10: Phase 1 Open Pit Design



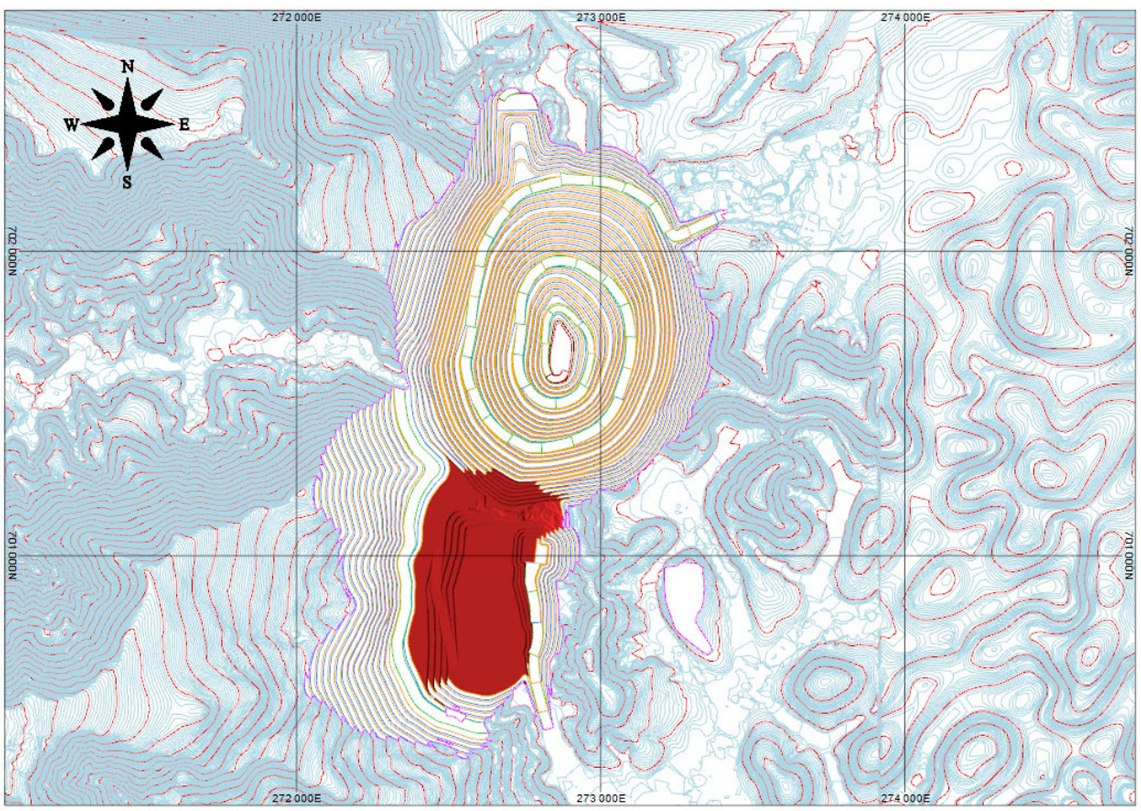
Source GMS 2025
**Note: Not to scale*

Figure 16.11: Phase 2 Open Pit Design



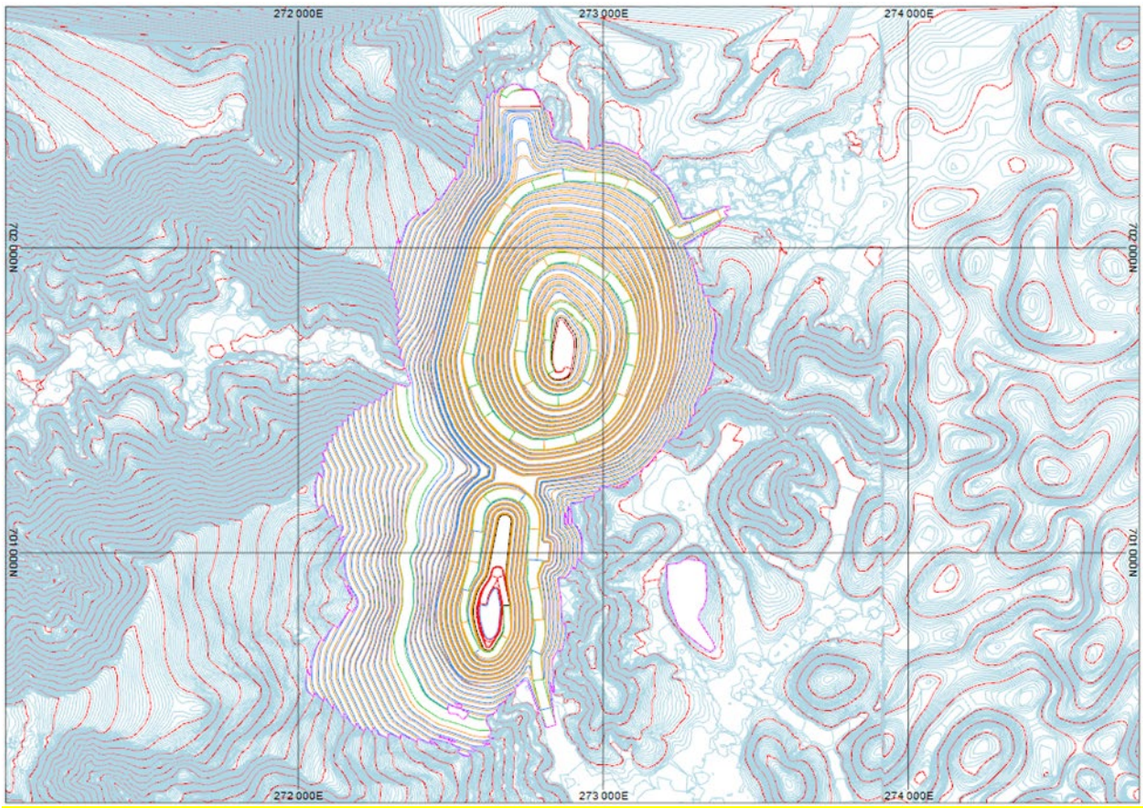
Source GMS 2025.
**Note: Not to scale*

Figure 16.12: Phase 3 Open Pit Design



Source GMS 2025
**Note: Not to scale*

Figure 16.13: Final Phase Open Pit Design

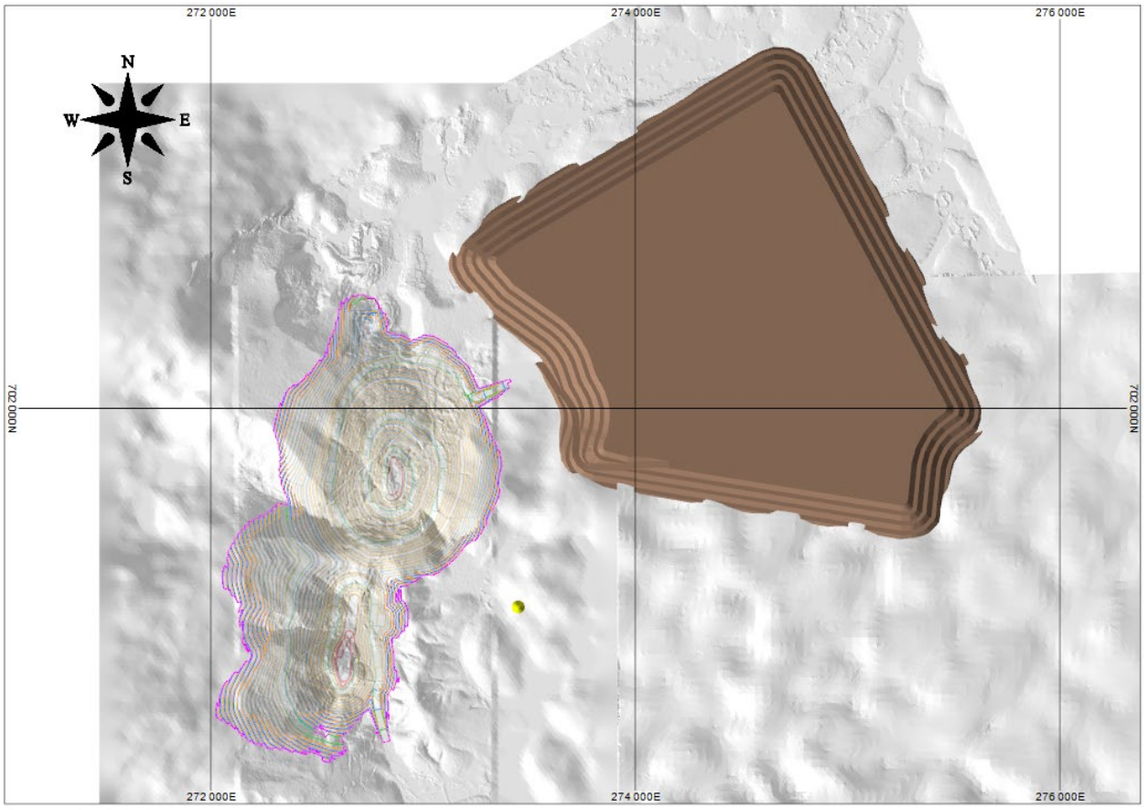


Source GMS 2025
**Note: Not to scale*

16.4.3 Waste Rock Storage Facility

A total of 426 million tonnes of waste rock, transition material, and saprolite will be mined in the open pit over the LoM. All waste material will be hauled to the waste storage facility (WSF) and stacked using tracked dozers. The material pile will be constructed in layers oriented in the north-south and west-east axes. The geotechnical firm TEC3 was mandated to design the slopes and berm widths for the WSF. Figure 16.14 illustrates the location of the WSF.

Figure 16.14: Waste Storage Facility

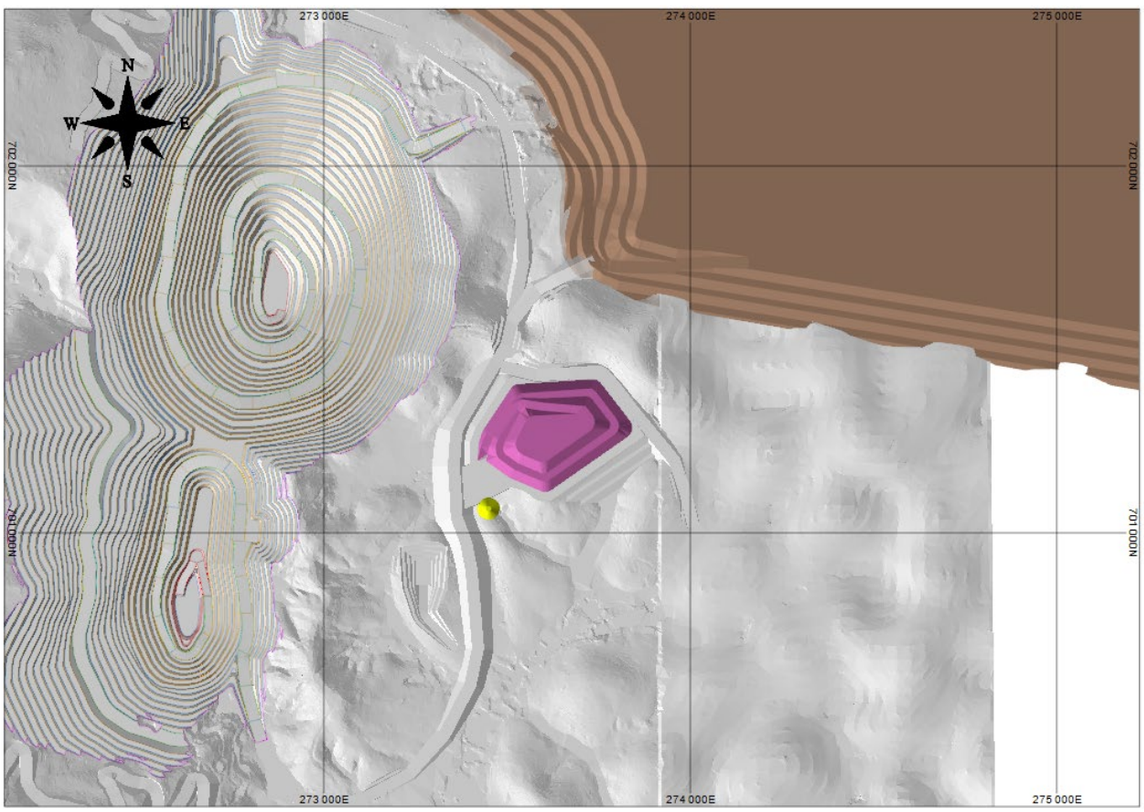


Source GMS 2025.
**Note: Not to scale*

16.4.4 Ore Stockpile

Ore material will be stockpiled south of pit, next to the crusher, to ensure a steady flow of ore material to the mill and minimal rehandling. Figure 16.15 shows the position of the ore stockpile, with a capacity of 3.2 Mm³ which is more than that required in the mining sequence. The ore stockpile design parameters are the same as those of the WSF.

Figure 16.15: Ore Stockpile Location (Stockpile = Purple, Crusher = Yellow)



16.4.5 Mine Haul Roads

The haul roads from the pit to the WSF, the ore stockpile, the crusher and other facilities will mostly be constructed during the pre-production period. Some haul roads will be constructed during operations as the pit evolves. Over the Life-of-mine (LoM), a total of 11 km of mine haul roads will be constructed outside of the pit itself. The main haul road that connects the pit, the ore stockpile, the crusher and the waste storage facility was designed to minimize the average haul distance at a maximum slope of 4% in order to maintain good speed and efficient cycle times.

16.4.6 Open Pit Production Schedule

The LoM production schedule for the open pit mine was optimized using Minemax^{tm} Scheduler, which is an industry-leading schedule optimizer using CPLEX software. Minemax^{tm} Scheduler is an automated mine scheduling tool which leverages multi-period optimization to determine the maximum net present value (“NPV”) while imposing various physical constraints and targets. The optimization includes mine sequencing and mining rate, stockpile usage and rehandling, and fleet usage. The strategic optimal plan from Minemax^{tm} was then further detailed on an annual basis using Deswik^{tm} to track material movements, stockpile inventory, mill blending, waste movements, and equipment usage.

Open pit mining activities are planned over a duration of 15 years, which includes three (3) years of pre-production. The mining rate will ramp up to reach a maximum rate of 44 Mtpa in Year -1 until Year 6, and it will decrease afterwards. Figure 16.16 presents the open pit mining schedule by material type (without stockpile reclaim movement). Variations in the quantity of open pit ore mined are attributable to three (3) factors:

- The contribution of underground mine production to mill feed is variable, ramping up in years 3 and 4, before remaining relatively stable for the balance of the LoM, decreasing the requirement for ore material from the open pit.
- Ore saprolite availability in the earlier years of the LoM allows for a higher mill throughput.
- Availability of ore material from the stockpile, notably in year 2.

Figure presents the yearly tonnage mined by phase, whereas Figure 16.17 presents the yearly tonnage mined by weathering type. In any given year, there are up to three (3) phases mined simultaneously, with one phase typically serving as the primary source of ore and the others focusing on waste removal. The mine plan kept a maximum sinking rate of 60 m per year per phase.

Details of mine production showing mined grades and material movement are presented in Table 16.11. Figure 16.18 through Figure 16.21 depict the progression of the pit and waste storage facility over the LoM until the end of open pit mining in Year 13.

Figure 16.16: Open Pit Mining Schedule by Material Type (Without Stockpile Reclaim)

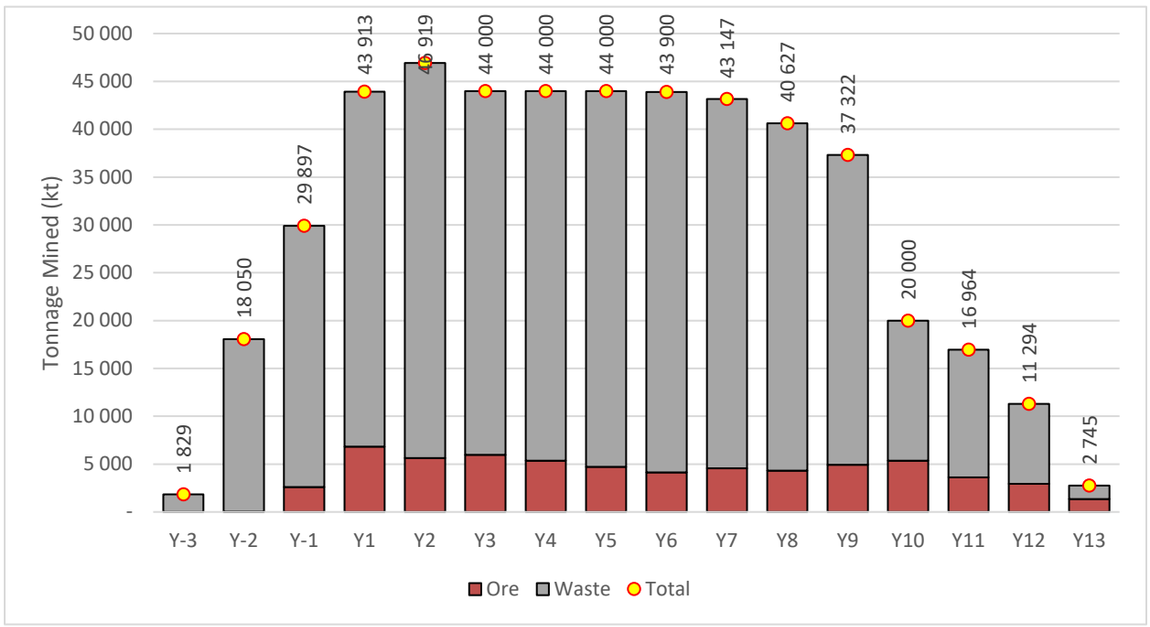


Figure 16.17: Open Pit Mining Schedule by Phase (Without Stockpile Reclaim)

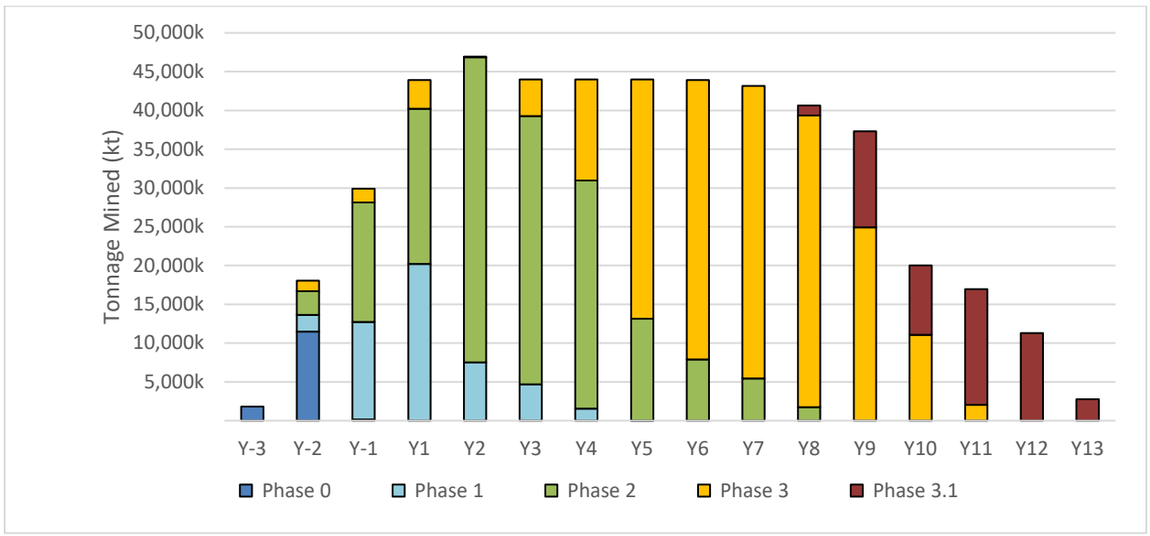


Figure 16.17: Open Pit Mining Schedule by Weathering Type (Without Stockpile Reclaim)

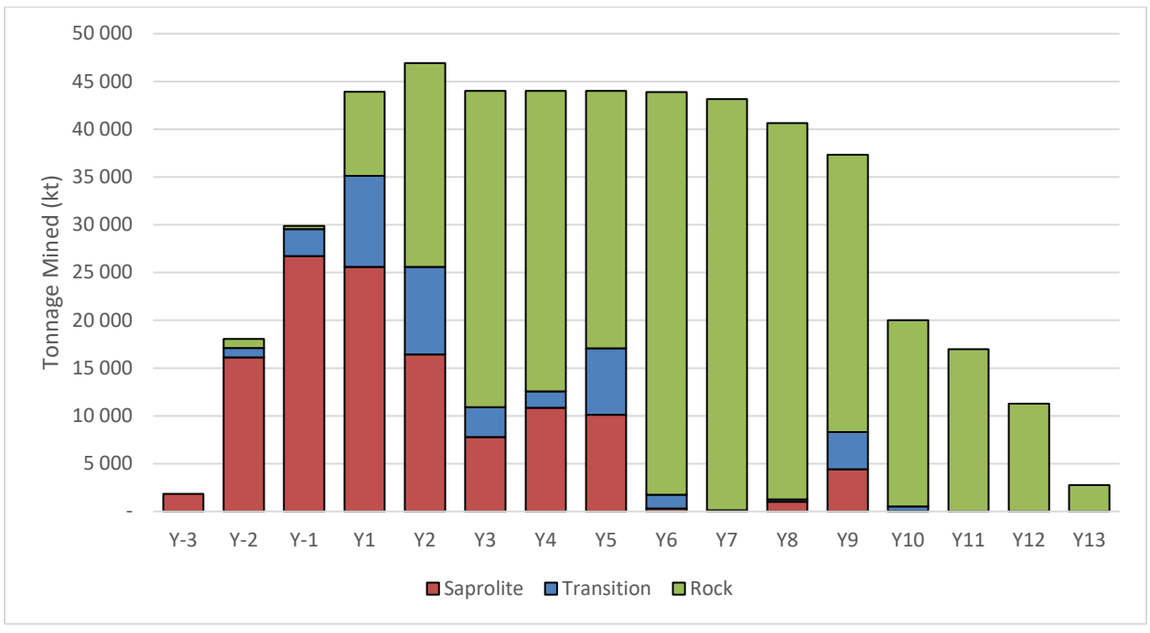
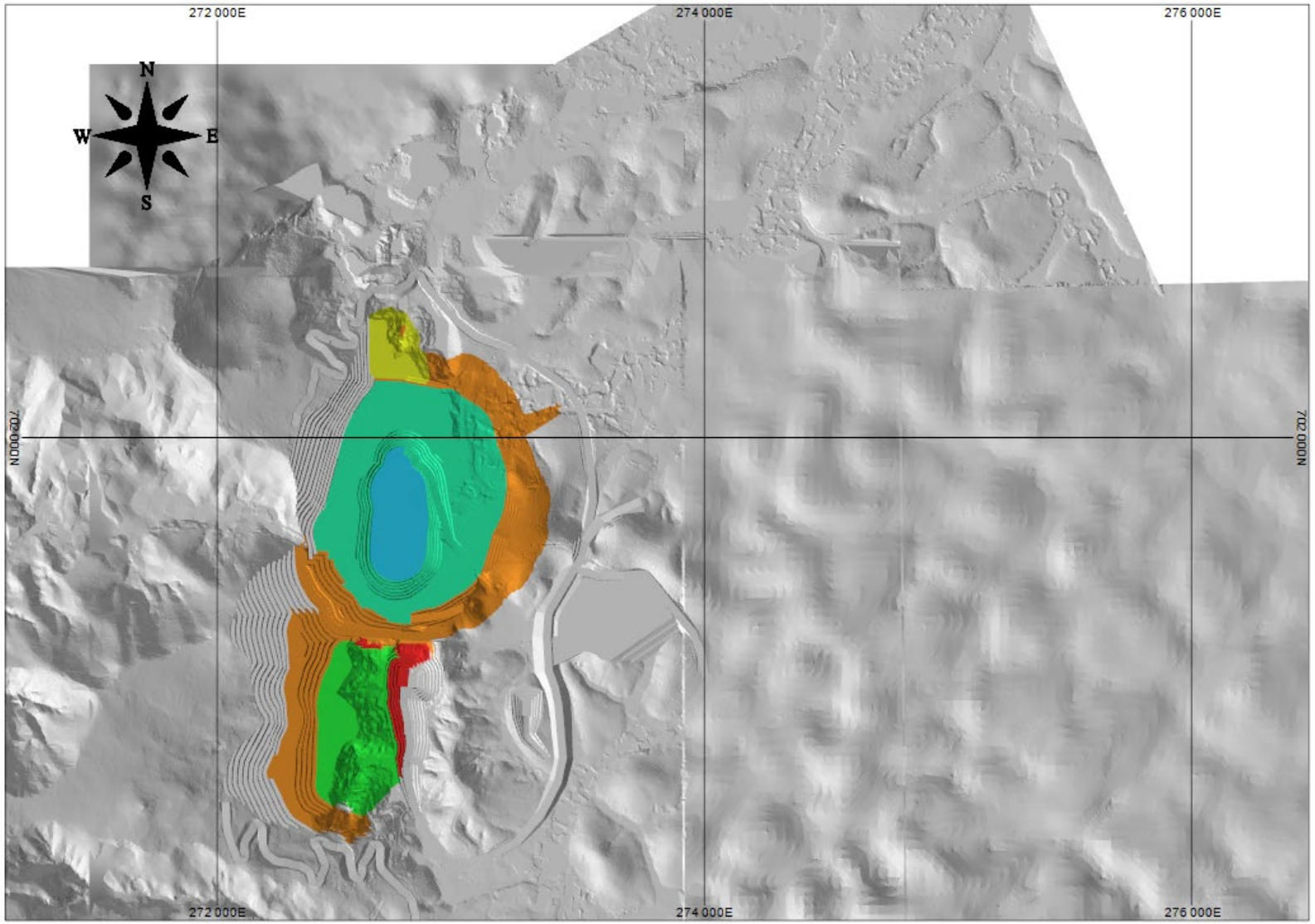


Table 16.11: Open Pit Mining Schedule Summary

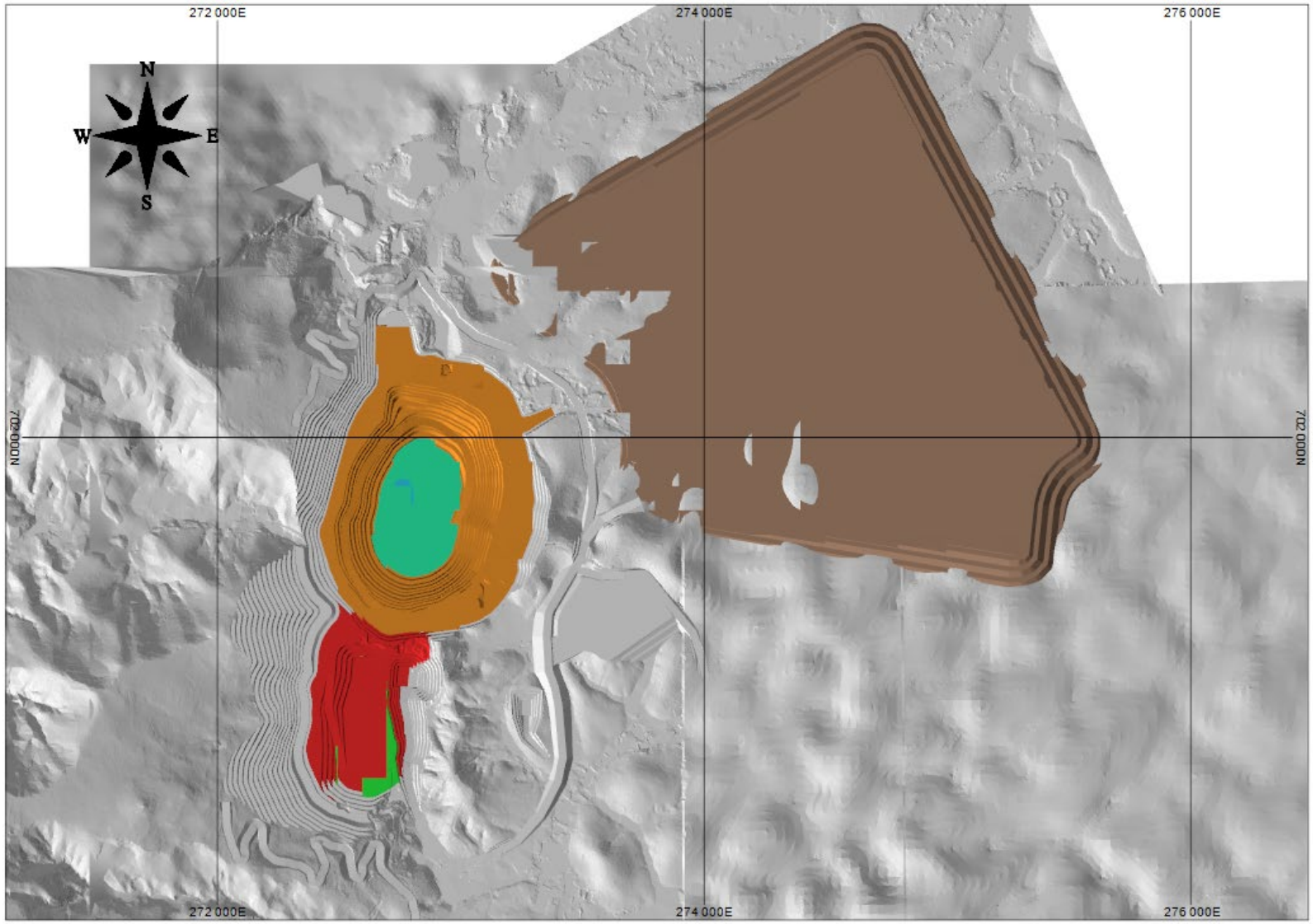
Description	Unit	Total	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
MINED - OPEN PIT																		
OP Waste	kt	426,232	1,829	18,030	27,281	37,083	41,289	38,015	38,650	39,275	39,765	38,596	36,315	32,387	14,633	13,359	8,341	1,385
Saprolite Waste	kt	113,631	1,829	16,087	24,300	22,594	15,405	6,997	10,718	10,090	313	120	964	4,188	27	-	-	-
Trans. Waste	kt	36,833	-	998	2,631	7,646	8,485	3,038	1,596	6,866	1,421	-	247	3,435	469	-	-	-
Fresh Rock Waste	kt	275,769	-	945	350	6,843	17,399	27,980	26,336	22,319	38,031	38,476	35,104	24,763	14,138	13,359	8,341	1,385
OP Ore Tonnage	kt	62,374	-	19	2,616	6,830	5,630	5,985	5,350	4,725	4,135	4,552	4,311	4,935	5,367	3,606	2,953	1,360
OP Ore Grade	g/t Au	1.6	-	0.9	1.5	1.5	1.5	1.5	1.7	1.6	1.4	1.5	1.6	1.6	2.1	1.6	1.3	1.2
OP Ore Ounces	koz	3,156	-	1	127	338	268	290	286	247	187	215	217	254	362	186	124	53
Saprolite Ore Tonnage	kt	7,657	-	19	2,419	2,984	1,012	801	144	10	-	-	39	229	-	-	-	-
Saprolite Ore Grade	g/t Au	1.3	-	0.9	1.5	1.5	0.9	0.8	0.6	0.4	-	-	0.6	0.7	-	-	-	-
Saprolite Ore Ounces	koz	314	-	1	113	141	30	20	3	0	-	-	1	5	-	-	-	-
Trans. Ore Tonnage	kt	3,541	-	-	181	1,898	683	77	114	104	1	-	5	449	29	-	-	-
Trans. Ore Grade	g/t Au	1.4	-	-	2.3	1.7	1.2	0.9	1.1	0.8	0.4	-	0.5	0.7	0.7	-	-	-
Trans. Ore Ounces	koz	163	-	-	13	104	26	2	4	3	0	-	0	11	1	-	-	-
Fresh Rock Ore Tonnage	kt	51,176	-	0	16	1,948	3,934	5,107	5,092	4,611	4,134	4,552	4,267	4,257	5,337	3,606	2,953	1,360
Fresh Rock Ore Grade	g/t Au	1.6	-	0.5	1.2	1.5	1.7	1.6	1.7	1.6	1.4	1.5	1.6	1.7	2.1	1.6	1.3	1.2
Fresh Rock Ore Ounces	koz	2,679	-	0	1	94	212	268	280	244	187	215	217	238	361	186	124	53
Strip Ratio (W:O)	-	6.8	-	926.9	10.4	5.4	7.3	6.4	7.2	8.3	9.6	8.5	8.4	6.6	2.7	3.7	2.8	1.0
OP Waste & Ore Tonnage	kt	488,607	1,829	18,050	29,897	43,913	46,919	44,000	44,000	44,000	43,900	43,147	40,627	37,322	20,000	16,964	11,294	2,745

Figure 16.18: Mine Development Y1



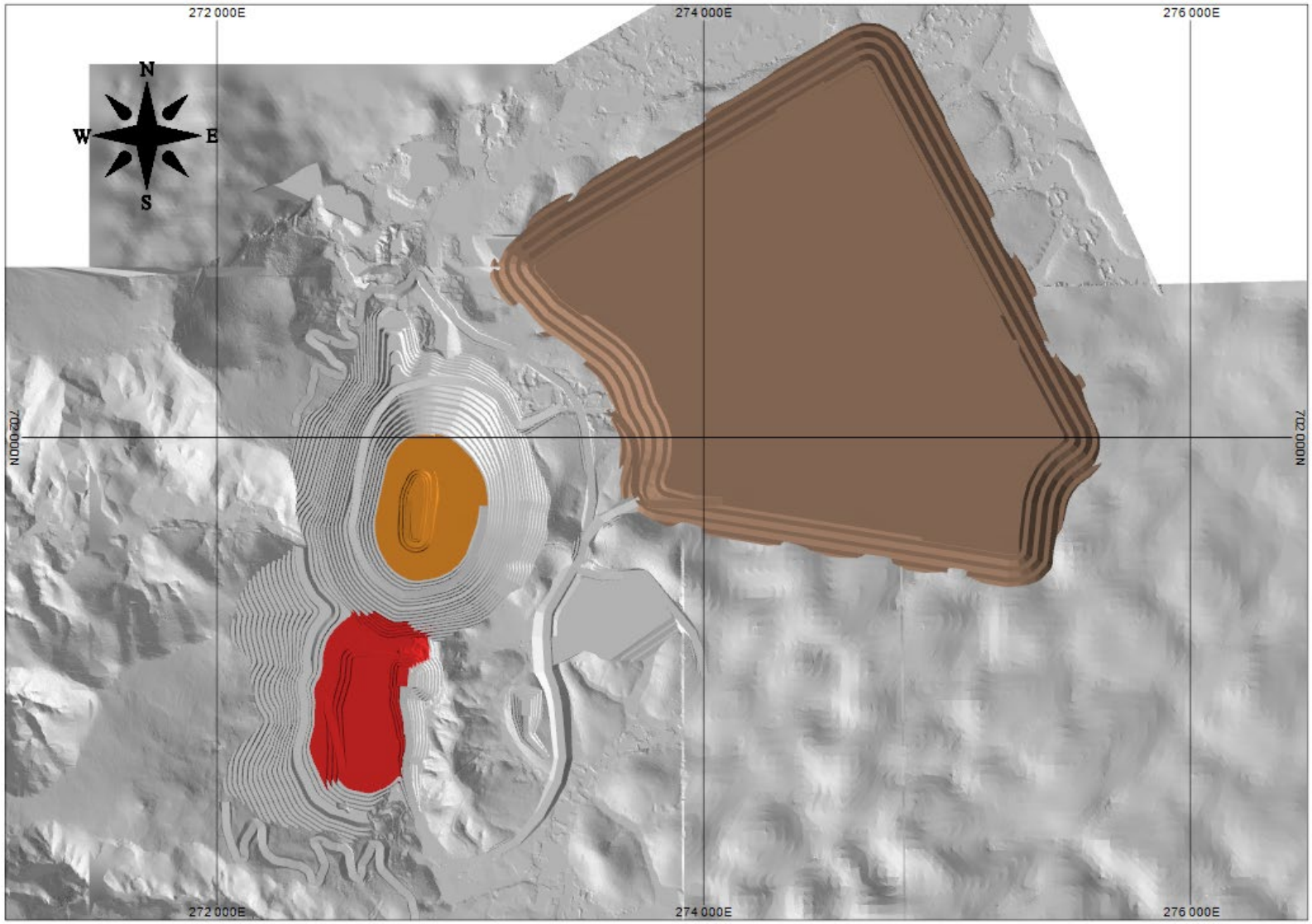
Source GMS 2025
**Note: Not to scale*

Figure 16.19: Mine Development Y4



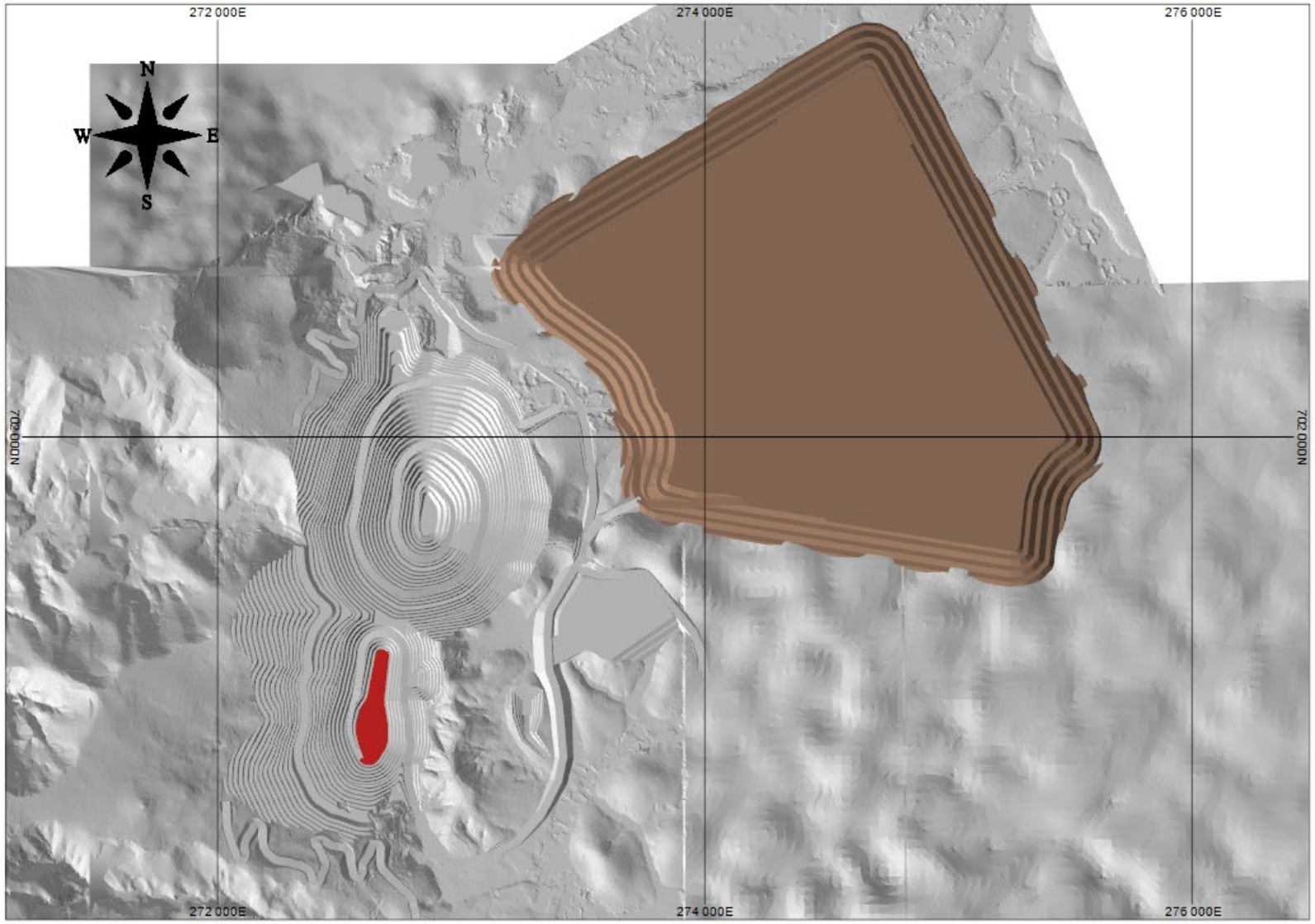
Source GMS 2025
**Note: Not to scale*

Figure 16.20: Mine Development Y9



Source GMS 2025
**Note: Not to scale*

Figure 16.21: Mine Development – Y13 (End of LoM)



Source GMS 2025
**Note: Not to scale.*

16.4.7 Mine Operations and Equipment Selection

16.4.7.1 Drilling and Blasting

Four (4) different drilling patterns will be used depending on the hardness of the rock and the destination of the material (ore material or waste). Single-pass drills will be used to ensure optimal productivity.

Table 16.12 describes all the different parameters considering materials and destination. Saprolite will be drilled for grade control purposes only. Nevertheless, 25% of mined saprolite will be assumed to have been blasted, as the limit between saprolite and transition material is not easily defined. The saprolite and transition material will be drilled using 6.75-inch holes and rotary drills. For the transition material, 2.0 m of subdrill will be added to maintain good pit floors.

For fresh rock, two (2) different patterns are planned. The DTH drilling method will be the most efficient method on this harder rock with 8.5-inch holes for waste and ore. 1.5 m of subdrilling will be considered. For waste material, a wider drilling pattern will be used, with a 5.50 m burden and 6.25 m spacing. In order to optimize feed fragmentation to the crusher, a smaller pattern will be used for ore with a 5.50 m burden and 5.50 m spacing. Powder factors will be 0.36 kg/t in ore material and 0.32 kg/t in waste. Most of the blast holes will be initiated with NONEL detonators paired with two (2) 450 g prime boosters.

It is expected to have a better production drilling rate with rotary drills for saprolite and transition material, with 90.0 metres per engine hour and 60 metres per engine hour, respectively. For fresh rock, it is expected to have better productivity with DTH drills, with 24.5 metres per engine hour.

Controlled blasting techniques will be used, including buffer blasts and pre-splitting. The pre-splitting consists of closely spaced holes along the designed excavation limit. The holes are loaded with a light charge and detonated simultaneously or in groups separated by short delays. Firing the pre-split row creates a fracture along the excavation limit and helps to prevent wall rock damage by venting explosive gases and reflecting shock waves. A pre-split drill rig was selected for this application. Explosives and Blasting services will be supplied by a third-party provider who will be responsible for supplying and delivering explosives with emulsion trucks, loading the holes, and initiating the blasts in safe conditions. The third party will provide all the equipment and workforce to operate the explosive plant, as well as loading and blasting services. At peak production, this will require 21 people. The owner mine’s engineering department will be responsible for designing blast patterns.

Table 16.12: Drill and Blast Parameters

Drill & Blast Parameters		Ore Fresh Rock	Waste Fresh Rock	Saprolite	Transition
Drill Pattern		**DTH**	**DTH**	**Rotary**	**Rotary**
K_S: Spacing / Burden		1.00	1.15	1.17	1.17
K_B: Burden / Diameter		25.46	25.47	35.00	35.00
K_J: Subdrill / Burden		0.27	0.32	0.00	0.33
K_T: Stemming / Burden		0.85	0.85	0.67	0.67
K_H: Height / Burden		1.82	1.82	1.67	1.67
Explosive Density	*g/cm³*	1.20	1.20	1.20	1.20
Hole Angle	*degrees*	0.00	0.00	0.00	0.00
Hole Diameter	*in*	8.50	8.50	6.75	6.75
Diameter (D)	*m*	0.216	0.216	0.171	0.171
Burden (B)	*m*	5.50	5.50	6.00	6.00
Spacing (S)	*m*	5.50	6.30	7.00	7.00
Subdrill (J)	*m*	1.50	1.75	-	2.00
Stemming (T)	*m*	4.70	4.70	4.00	4.00
Bench Height (H)	*m*	10.0	10.0	10.0	10.0
Blasthole Length (L)	*m*	11.5	11.75	10	12
Pattern Yield					
Rock Density	*t/bcm*	2.77	2.77	1.90	2.20
BCM / hole	*bcm/hole*	303	347	420	420
Yield per Hole	*t/hole*	838	960	798	924
Yield per metre drilled	*t/m drilled*	73	82	80	77
Explosive Column (LE)	*m*	6.80	7.05	6	8
Volume of Explosives / Hole	*m³*	0.25	0.26	0.14	0.18
Weight of Explosives / Hole	*kg*	299.01	309.72	166.23	221.63
Powder Factor	*kg/t*	0.36	0.32	0.21	0.24
Powder Factor	*kg/bcm*	0.99	0.89	0.40	0.53

Drill & Blast Parameters		Ore Fresh Rock	Waste Fresh Rock	Saprolite	Transition
Drill Productivity					
Re-drills	*%*	5.0%	5.0%	5.0%	5.0%
Pure Penetration Rate	*m/min*	1.02	1.02	3.00	2.0
Pure Penetration Rate	*m/hr*	61.2	61.2	180.0	120.6
Overall Drilling Factor (%)	*%*	40.00%	40.00%	50.00%	50.00%
Overall Penetration Rate (production hour)	*m/hr*	24.5	24.5	90.0	60.3
Drilling Efficiency	*t/hr*	1 784	2 000	7 182	4 643
Drilling Efficiency	*holes/hr*	2.13	2.08	9.00	5.03

The loading fleet consists of four (4) 22 m^3 diesel hydraulic shovels, one (1) 9 m^3 diesel backhoe, and one (1) 11.5 m^3 diesel front-end loader. The 22 m^3 shovel will be utilized for both waste and ore material loading, while the front-end loader will be primarily used to rehandle the ore stockpile and to supplement ore loading as needed. The 9 m^3 backhoe will be used to excavate saprolite on the upper benches or sectors with difficult access. Table 16.13 presents the productivity assumption for the loading fleet. The number of units is represented in whole numbers for the purpose of these productivity assumptions.

Table 16.13: Loading Fleet Productivity Assumptions

		Saprolite	Saprolite	Transition	Fresh Rock	Rehandle
Loading Unit		**Front Shovel 24 m^3**	**Backhoe 9 m^3**	**Front Shovel 22 m^3**	**Front Shovel 22 m^3**	**Loader 11.5 m^3**
Haulage Unit		**Mining Truck (150 short tons)**	**Articulated Truck**	**Mining Truck (150 short tons)**	**Mining Truck (150 short tons)**	**Mining Truck (150 short tons)**
Rated Truck Payload	*t*	140	40	140	140	140
Heaped Tray Volume	*m^3*	93	24	82	82	82
Bucket Capacity	*m^3*	24.0	9.0	22.0	22.0	11.5
Bucket Fill Factor	*%*	95%	89%	92%	89%	85%
In-situ Dry Density	*t/bcm*	**1.90**	**1.90**	**2.20**	**2.77**	**2.77**
Moisture	*%*	15%	15%	11%	6%	6%
Swell	*%*	43%	43%	40%	64%	64%
Wet Loose Density	*t/lcm*	1.53	1.53	1.74	1.79	1.79

		Saprolite	Saprolite	Transition	Fresh Rock	Rehandle
Loading Unit		**Front Shovel 24 m^3**	**Backhoe 9 m^3**	**Front Shovel 22 m^3**	**Front Shovel 22 m^3**	**Loader 11.5 m^3**
Haulage Unit		**Mining Truck (150 short tons)**	**Articulated Truck**	**Mining Truck (150 short tons)**	**Mining Truck (150 short tons)**	**Mining Truck (150 short tons)**
Actual Load per Bucket	*t*	34.84	12.24	35.15	35.06	17.50
Passes (decimal)	#	4.02	3.27	4.00	4.01	8.02
Passes (whole)	#	4	3	4	4	8
Actual Truck Wet Payload	*t*	139	37	141	140	140
Actual Truck Dry Payload	*t*	121	32	127	132	132
Actual Heaped Volume	*m^3*	91	24	81	78	78
Payload Capacity	*%*	100%	92%	100%	100%	100%
Heaped Capacity	*%*	98%	100%	99%	96%	95%
Cycle Time						
Hauler Exchange	*min*	0.60	0.60	0.60	0.60	0.70
First Bucket Dump	*min*	0.10	0.10	0.10	0.10	0.10
Average Cycle Time	*min*	0.56	0.56	0.60	0.64	0.70
Load Time	*min*	2.38	1.82	2.50	2.62	5.70
Cycle Efficiency with Wait Time	*%*	75%	75%	75%	75%	75%
Nb. of Trucks Loaded per Hr	#	18.91	24.73	18.00	17.18	7.89
Productivity						
Productivity	*t/hr*	2,291	789	2,290	2,272	1,043
Effective Hours per Year	*hrs/y*	5,214	5,214	5,214	5,214	5,540
Annual Production Capacity	*kt/y/unit*	11,946,453	4,116,266	11,940,669	11,847,067	5,776,972

16.4.7.2 Hauling

Most of the haulage will be performed by 140-t class off-highway mining trucks for waste and ore material. Articulated trucks will be used for upper benches and difficult access roads. The ore material will be hauled to the crusher located outside of the pit, while the waste will be hauled to the WSF as shown in Figure 16.14.

The mining trucks will be equipped with specific boxes allowing greater volumes for saprolite material and will be modified afterwards when reaching fresh rock. The objective is to maximize the payload capacity of the trucks regardless of material type and density.

The truck requirements have been calculated in Deswik.LHS (Landform and Haulage) software. This software links the mining schedule to the material movements and determines optimal haulage routes and simulates haulage times using rimpull data from the fleet. The following assumptions were used when running the simulations.

- 50 km/h speed limit on roads and 30 km/h on benches.
- 30 km/h speed limit when loaded or going downhill.
- Average rolling resistance of 2%.
- Time for loading and spotting is variable depending on the loader, truck and material. This value ranges from 1.8 to 5.7 minutes (Figure 16.13).

Figure 16.22 depicts the average cycle times for ore material and waste over the LoM. The cycle times increase as the pit gets deeper due to increased uphill haulage required. Plateaus or dips in the cycle time are caused by new pushbacks starting from surface, temporarily reducing the cycle time. The cycle times presented do not include loading times.

Figure 16.23 depicts the total truck requirements over the years. Twenty-four (24) mining trucks will be required to maintain production at the peak mining rate, coupled with six (6) 40-ton articulated trucks.

Figure 16.22: Open Pit Average Haulage Cycle Times by Material Type

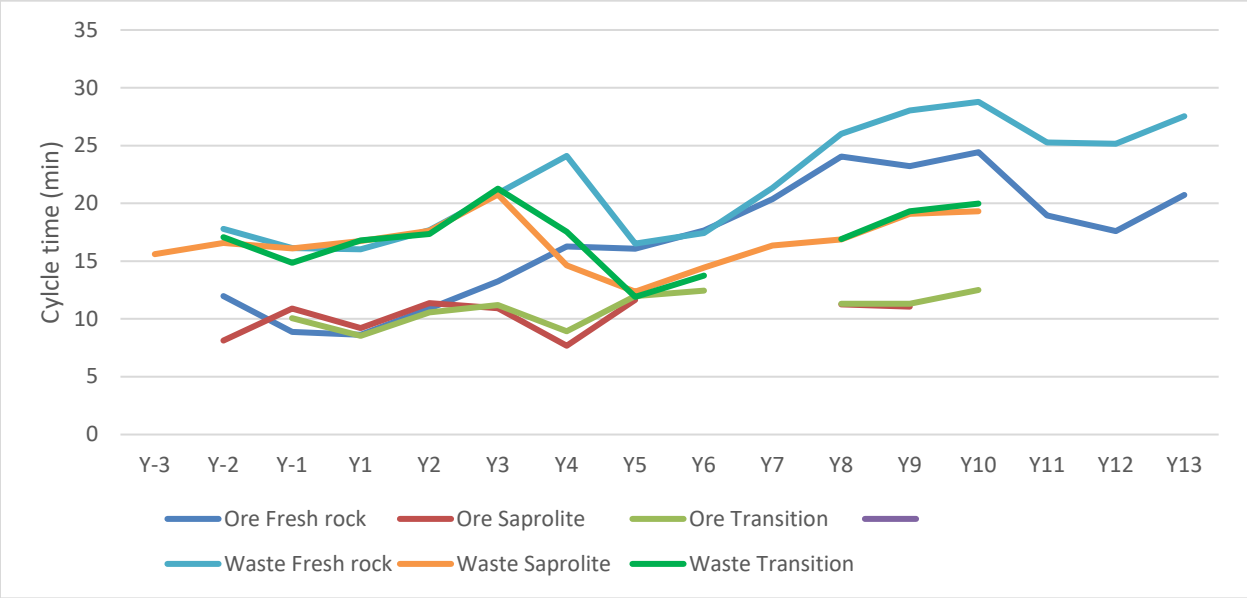
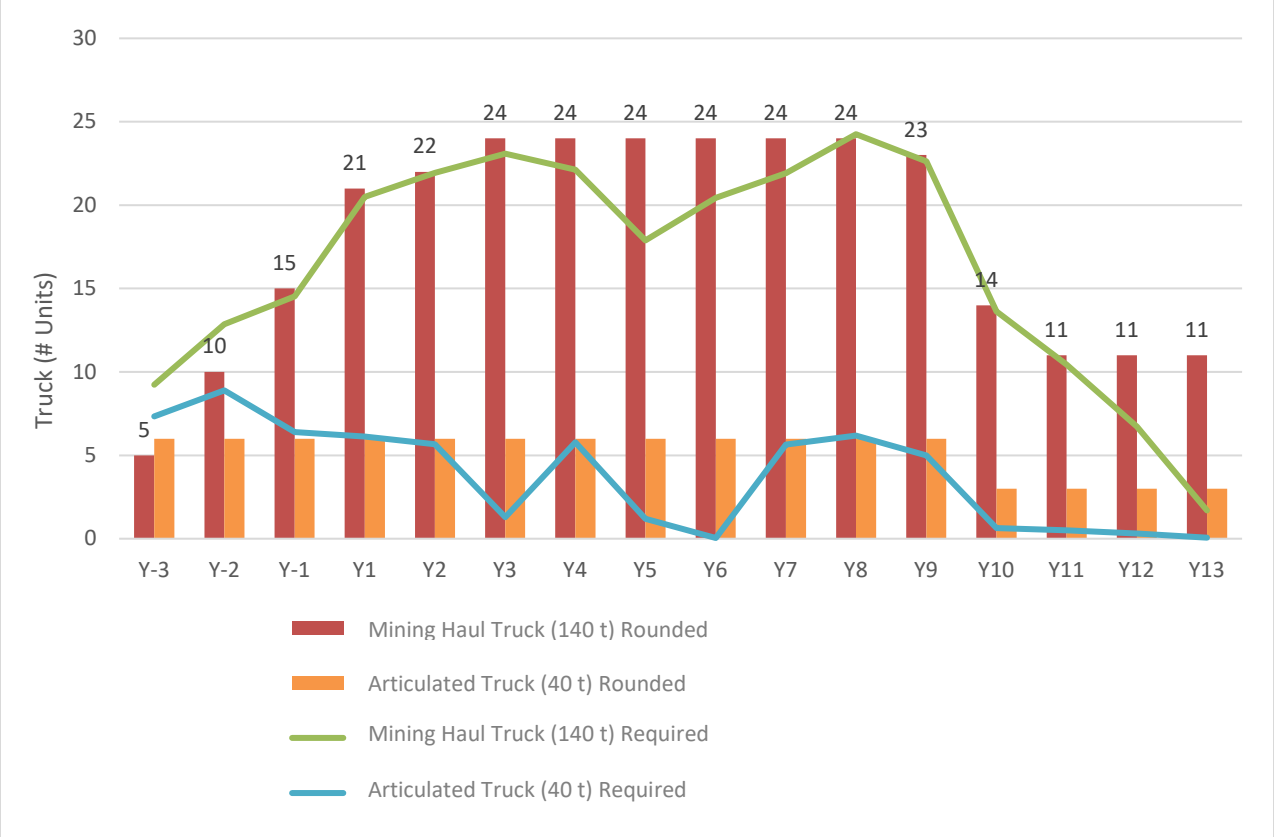


Figure 16.23: Open Pit Truck Requirements



16.4.7.3 Support Operations

Support equipment requirements are based on typical open pit mine operation and maintenance needs to safely support the loading, hauling, and drilling fleets.

Support equipment is planned for maintaining dump areas, stockpiles, pit floors, ditches and mine roads. The fleet of support equipment consist of the following:

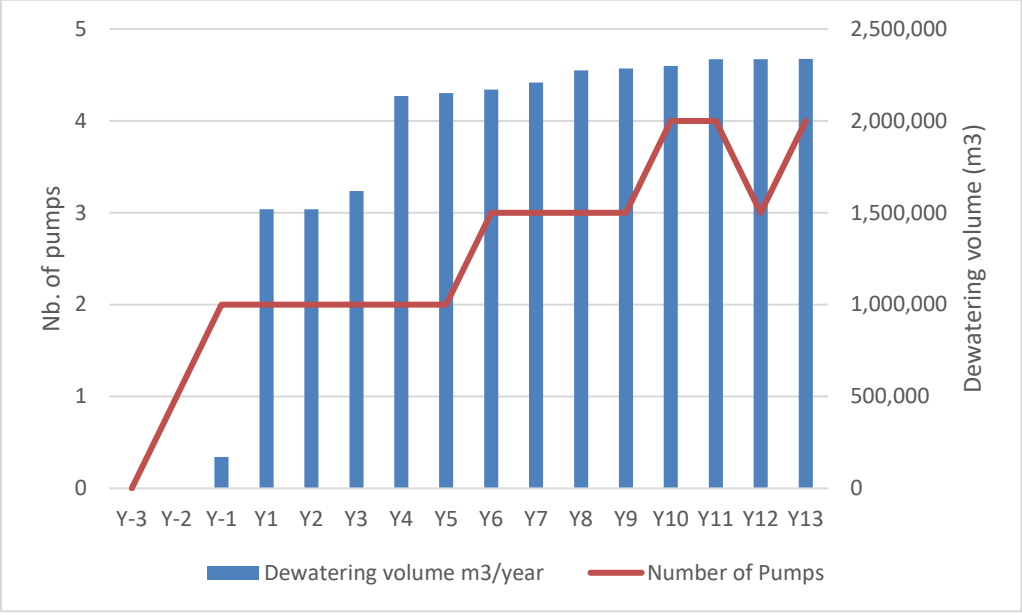
- 6 x 600 HP dozers for dump maintenance.
- 4 x 18 ft blade motor graders for road upkeep.
- 2 x water articulated trucks for dust suppression.
- 2 x 496 HP wheel dozers.

16.4.7.4 Mine Dewatering

Initial dewatering for the open pit mining is expected to be minimal and will be performed with 6.0-inch diesel pumps.

Dewatering during the operations will reach a maximum of 2.3 Mm^3 per year of water, requiring up to four (4) pumps in series. Dewatering volumes and the quantity of pumps over time are presented in Figure 16.24.

Figure 16.24: Open Pit Dewatering Volumes and Quantity of Pumps Over Time



16.4.8 Mining Fleet Requirements

Table 16.13 summarizes the gross operating hours considered for calculating equipment fleet requirements. The mine is expected to operate 22 hours per day, 355 days per year. This accounts for shift changes and 10 days of weather-related delay. Additional delays and applied factors are considered in productivity calculations for each fleet, as calculated in Table 16.14.

Additional equipment will be procured to facilitate the maintenance activities and support the operation, such as fuel and lube trucks, a forklift, a telehandler, a low-boy trailer and a tractor for moving the tracked equipment. Other small equipment, such as mechanical service trucks, generators and welding machines, is also included.

Table 16.15 and Table 16.16 present the equipment purchase schedule for the LoM.

Table 16.14: Equipment Usage Assumption

Equipment Usage Assumptions		Shovels	Loaders	Trucks	Drills	Pumps
Days in Period	days	365	365	365	365	365
Weather, Schedule Outages	days	10.0	10.0	10.0	10.0	10.0
Shifts per Day	shift/day	2.0	2.0	2.0	2.0	2.0
Hours per Shift	h/shift	12.0	12.0	12.0	12.0	12.0
Availability	%	85.0	85.0	85.0	85.0	90.0
Use of Availability	%	90.0	90.0	90.0	85.0	95.0
Utilization	%	76.5	76.5	76.5	72.25	85.5
Effectiveness	%	80.0	85.0	87.0	85.0	90.0
OEE	%	61.2	65.0	66.6	61.4	77.0
Total Hours	hours	8,760	8,760	8,760	8,760	8,760
Scheduled Hours	hours	8,520	8,520	8,520	8,520	8,520
Down Hours	hours	1,278	1,278	1,278	1,278	852
Delay Hours	hours	1,304	978	847	923	728
Standby Hours	hours	724	724	724	1,086	383
Operating Hours	hours	6,518	6,518	6,518	6,156	7,285
Ready Hours	hours	5,214	5,540	5,670	5,232	6,556

Table 16.15: Major Equipment Purchase Schedule

Major Equipment	Total	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Production Drill (6-10")	5	1	-	1	1	1	-	-	1	-	-	-	-	-	-	-	-
Auxiliary Pre-split Drill (4.5-8")	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Diesel Hydraulic Shovel (22 m^3)	4	2	-	1	1	-	-	-	-	-	-	-	-	-	-	-	-
Excavator (90-t)	1	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Wheel Loader (10.7 m^3)	1	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-
Mining Haul Truck (150-t)	24	8	2	7	4	3	-	-	-	-	-	-	-	-	-	-	-
Articulated Dump Truck (40-t)	12	6	-	-	-	-	-	6	-	-	-	-	-	-	-	-	-
Track Dozer (600 HP)	13	2	2	2	-	-	1	3	-	2	-	-	-	1	-	-	-
Motor Grader (18 ft)	8	1	-	2	1	-	-	-	1	1	2	-	-	-	-	-	-
Water Truck (30 kL tank)	6	1	-	1	-	-	1	1	-	-	-	1	1	-	-	-	-
Wheel Dozer (496 hp)	3	-	-	2	-	-	-	-	-	-	1	-	-	-	-	-	-

Table 16.16: Support Equipment Purchase Schedule

Support Equipment	Total	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Excavator (49-t)	2	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Excavator (90-t)	1	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-
Hydraulic Hammers for Excavator 49-t	6	1	-	1	-	-	-	1	1	-	-	-	1	1	-	-	-
Wheel Loader (271 hp)	2	-	-	1	-	-	-	-	-	-	1	-	-	-	-	-	-
Knuckle Boom Truck (10-t)	1	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-
Telehandler (6-t)	2	-	-	-	-	-	-	-	-	2	-	-	-	-	-	-	-
Manlift (45 ft)	1	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-
Mechanic Service Truck (6 wheel)	7	1	-	3	-	-	-	-	-	-	1	1	1	-	-	-	-
Tire Handler Truck	2	-	-	1	-	-	-	-	-	-	-	-	1	-	-	-	-
Mining Haul Truck (240-t)	1	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-
Fuel & Lube Truck (17 kL) ADT	6	1	-	1	-	-	-	2	-	-	-	-	2	-	-	-	-
Trailer Lowboy (400 t)	1	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-
Pickup Truck (1/2 t)	57	8	-	16	-	-	-	17	-	-	-	-	16	-	-	-	-
Pit Bus	5	1	-	-	1	-	-	-	2	-	1	-	-	-	-	-	-
Mobile Air Compressor (185 CFM)	3	-	-	1	-	-	-	-	-	1	-	-	1	-	-	-	-
Welding Machine Electric	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Welding Machine Diesel (400 A)	2	1	-	-	-	-	-	-	-	-	-	1	-	-	-	-	-
Light Plant	4	3	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-
Water pump 3 in - Gasoline	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Water Pump LH 6 in - Diesel	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Water Pump HH 10 in - Diesel	12	-	1	1	-	-	-	-	3	-	-	3	1	-	2	1	-

Support Equipment	Total	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Wheel Loader (527 hp)	3	-	1	-	-	-	-	-	1	-	-	1	-	-	-	-	-
Wheel Loader (271 hp)	3	-	1	-	-	-	-	1	-	-	-	-	1	-	-	-	-

16.4.9 Mobile Crushing Plant

The production of crushed material will be necessary for blasthole stemming purposes and for road maintenance. It is assumed that the required aggregate material production will occur internally with a mobile crusher on site. Waste rock to feed the aggregate crushing plant will come from pit pre-production.

Once in commercial production the UG backfill will require waste material from the OP mine. This material will be crushed to minus 6 inches by the gyratory crusher and sent underground with UG trucks. A loader will be added to the fleet to meet the 1.5 Mtpa fill requirement starting in Year 5.

16.4.10 Mine Maintenance

Maintenance will be performed by the owner's personnel. The maintenance department and personnel requirement has been structured to fully manage this function, performing maintenance planning and training of employees. However, reliance on dealer and manufacturer support will be key for the initial years of the project and major component rebuilds will be supported by the equipment dealer throughout the LoM. Tire monitoring, rotation and/or replacement will also be conducted internally.

16.4.11 Mine Management

The operation team is responsible for achieving production targets in a safe and efficient manner. The engineering and geology team will support the operations team by providing short-term and long-term planning, grade control, surveying, mining resources and reserves estimation and other technical functions. The mine dispatch system is provisioned and will be managed by the mine operation team.

16.4.12 Pit Slope Monitoring

A pit slope monitoring system will be used to gather any information on micro and macro movements of the pit walls. It will consist of strategically placed prisms that are surveyed under a controlled environment (windless, rainless, and stationary).

16.5 Underground Mining

16.5.1 Underground Mining Method

The selected underground mining method is the long hole open stoping (LHOS) mining method with either transverse or longitudinal stoping. The stoping sequence will be ascending from an initial undercut. Stopes

will be mined using an upper access for drilling and a lower access for mucking. Stopes within sill pillars will require the upper access to be redeveloped through backfill to allow access for drilling.

LHOS is a commonly used underground mining method for competent hard rock orebodies. The first phase of LHOS is the mine development phase. During this phase, the following excavations are developed: decline, level access, haulage drifts, draw points and other infrastructures. These excavations provide access to the stoping area and support the production activities. The development phase also includes the development of both an overcut drift and an undercut drift.

The overcut drift is developed to allow production drilling of the stoping area, while the undercut allows for ore extraction from the stope. Once the development of the drilling drift and extraction drift is completed, a slot raise, typically raise bored or conventionally drilled and blasted, is created to provide an initial void for production drilling and blasting activities.

The production drilling phase involves drilling long vertical or inclined holes at regular intervals along the length and width of the stoping area. The blasting phase begins by loading the production drill holes with bulk explosives and stemming once production drilling is completed.

During the blasting phase, the controlled use of explosives allows to fracture the rock surrounding the slot raise. Once the rock is blasted and the blasting gases cleared from the mine, the fragmented material is removed from the stope through the undercut drift with a load-haul-dump (LHD) unit. Depending on the size and geometry of the stopes, it can take up to 3 or 4 blasting and mucking cycles to extract all the material from the stope. The broken material from the stopes is loaded into haul trucks and then transported to surface.

LHOS is a non-entry mining method, since the stoping area is not accessible to personnel once production begins. The use of some remote-controlled load-haul-dump (LHD) units is required to completely remove the blasted material from the stope. The final phase of LHOS is the backfilling phase. Depending on multiple factors like the variant of LHOS used and the mining sequence, stopes can be filled with cemented material, uncemented material or a combination of both. Cemented backfill can include paste fill, hydraulic fill and cemented rockfill (CRF) while uncemented backfill is typically rockfill. The selected underground cemented backfill method for the Oko Gold project is CRF.

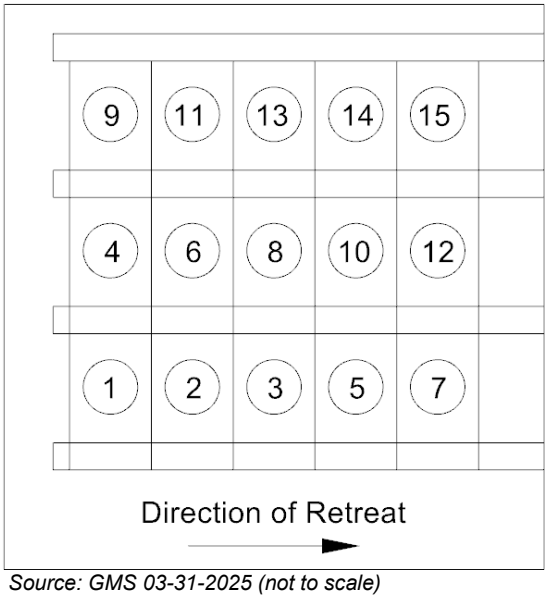
This non-entry mining method offers several advantages including, but not limited to high productivity, high operational flexibility, low operating costs, efficient mineralized material extraction and improved worker safety.

The longitudinal variant of LHOS is used for the following situations:

- Stoping thickness is narrower than 8 m.
- Adjacent to longitudinal stopes where transverse LHOS is not practical.

The longitudinal mining areas are accessed by developing the overcut and undercut ore drifts inside the stoping area along the strike of the orebody. Once the development is completed to the extremity of the longitudinal mining area, the production cycle of the initial stope can start. Subsequent stopes will be mined in the same cycle while retreating towards the main access located either at an extremity or near the middle of the orebody. Figure 16.25 illustrates the typical longitudinal mining sequence.

Figure 16.25: Typical Longitudinal Stoping Sequence



Source: GMS 03-31-2025 (not to scale)

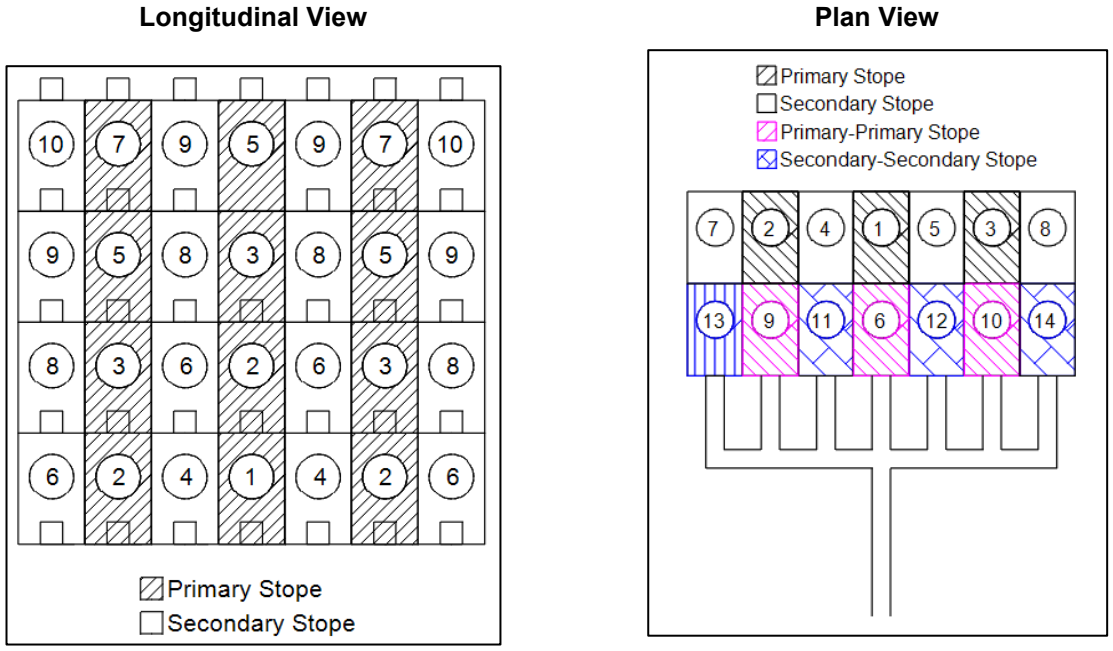
The transverse variant of LHOS is used for the following situations:

- Stoping area is wider than 8 m.
- Adjacent to transverse LHOS where longitudinal LHOS is not practical.

The transverse mining area will be accessed by developing a haulage drift parallel to the strike of the orebody. This haulage drift shall maintain a reasonable distance from the stoping area to preserve its integrity during the production phase. Perpendicular to the haulage drift, a series of parallel draw points evenly spaced along the haulage drift will be used to access the undercut and overcut ore drifts developed over the full width of the orebody. Once the haulage drift is established, the draw points and the ore drifts are developed, the production cycle of the initial primary stope can begin. Subsequent primary stopes are mined in the exact same cycle while retreating towards the extremities of the orebody. Stopes will be

sequenced in an overhand approach so that when two (2) lifts of primary stopes are mined, the secondary stope between these two (2) primaries can be mined on the first lift. This primary-secondary transverse mining approach is advantageous in terms of production and operational flexibility, as multiple stopes can be in operation simultaneously. One disadvantage of the method is that a haulage drift must be excavated along the entire length of the mineralized zone. However, the production cycle can still begin even if the haulage drift is not completely excavated to the extremities of the orebody. Figure 16.26 illustrates the typical sequence of transverse stoping.

Figure 16.26: Typical Transverse Stoping Sequence



Source: GMS 06-06-2024 (not to scale)

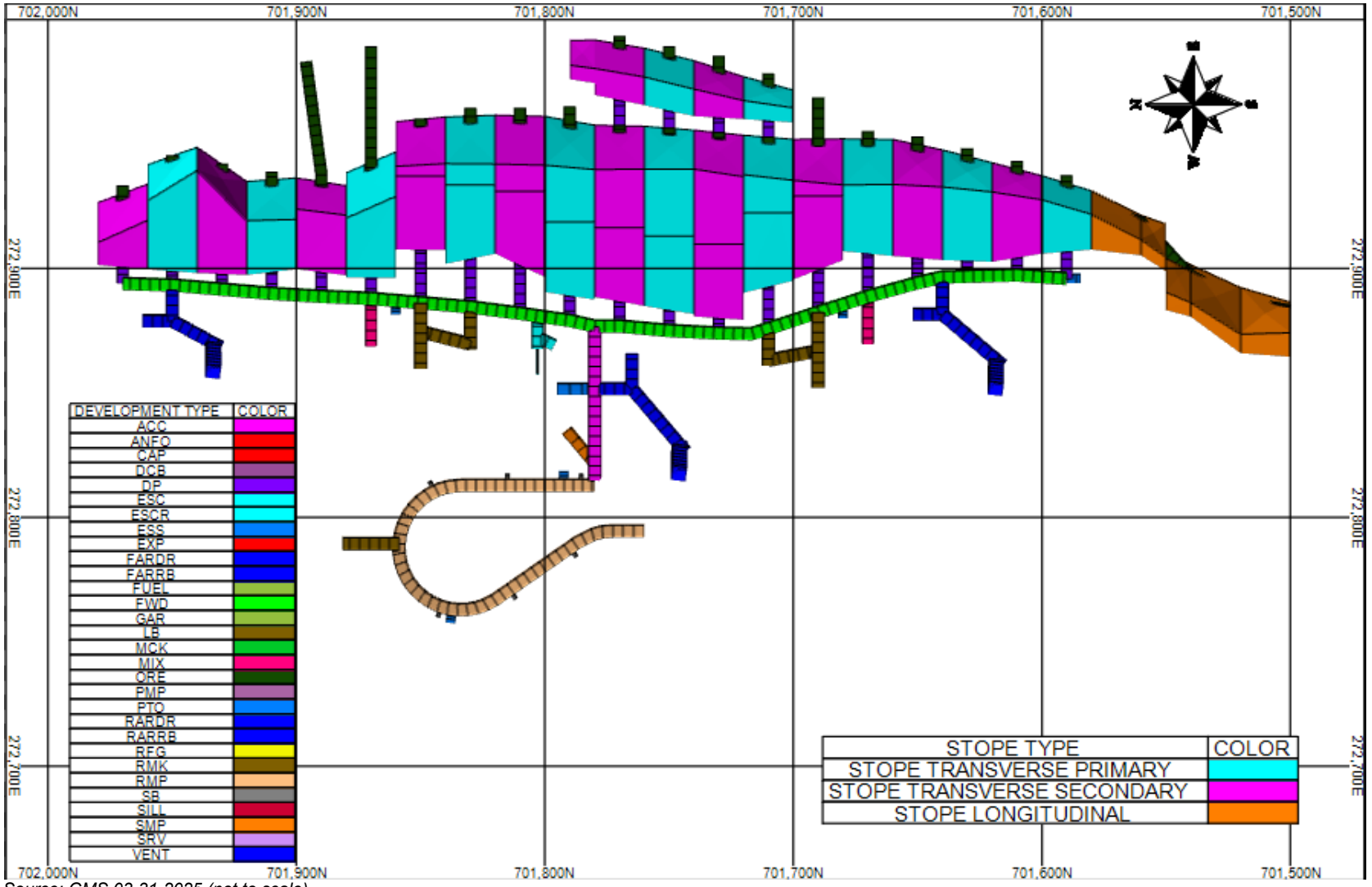
16.5.2 Underground Mine Design

16.5.2.1 Development Design

The Oko West UG mine will be accessed by a single decline with a portal located inside an initial phase of the open pit. Each production level will be accessed by a level access and a haulage drift that leads to the crosscuts that are either driven perpendicular to the ore body for transverse stoping or within the direction of the ore body for longitudinal stoping. For a typical level access, the following infrastructure must be included: sump, electrical bay and fresh air access. For a typical haulage drift, the following infrastructure must be included: loading bay (both sides), mixing pit (both sides), access to return air raise (north side), access to secondary fresh air raise (south side) and safety egress. Figure 16.27 and Figure 16.28 show

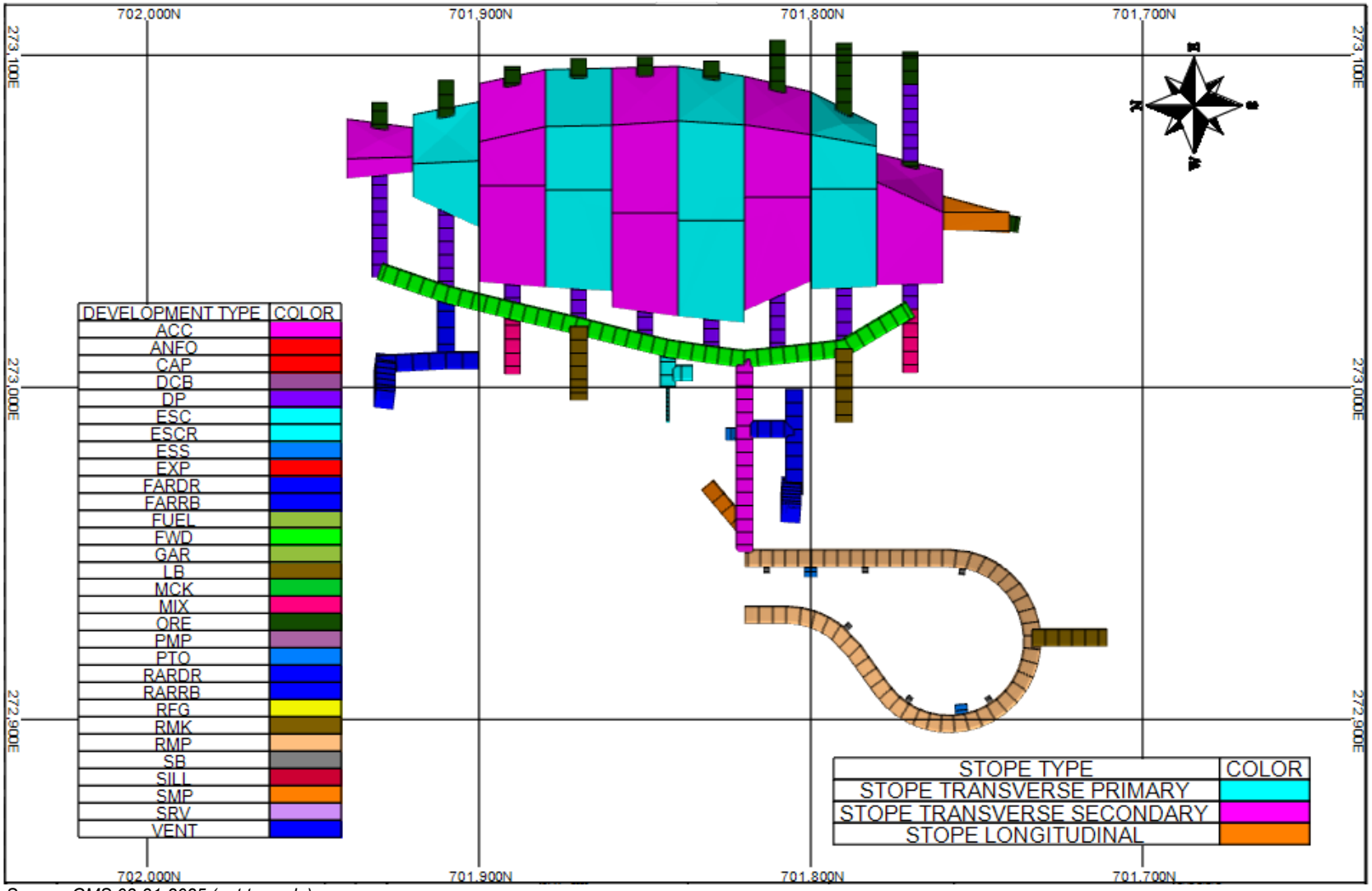
plan views of a typical underground mine production levels, while Figure 16.29 and Figure 16.30 show a longitudinal and a section view of the lateral and vertical development of the mine.

Figure 16.27: Typical Larger Level Plan View



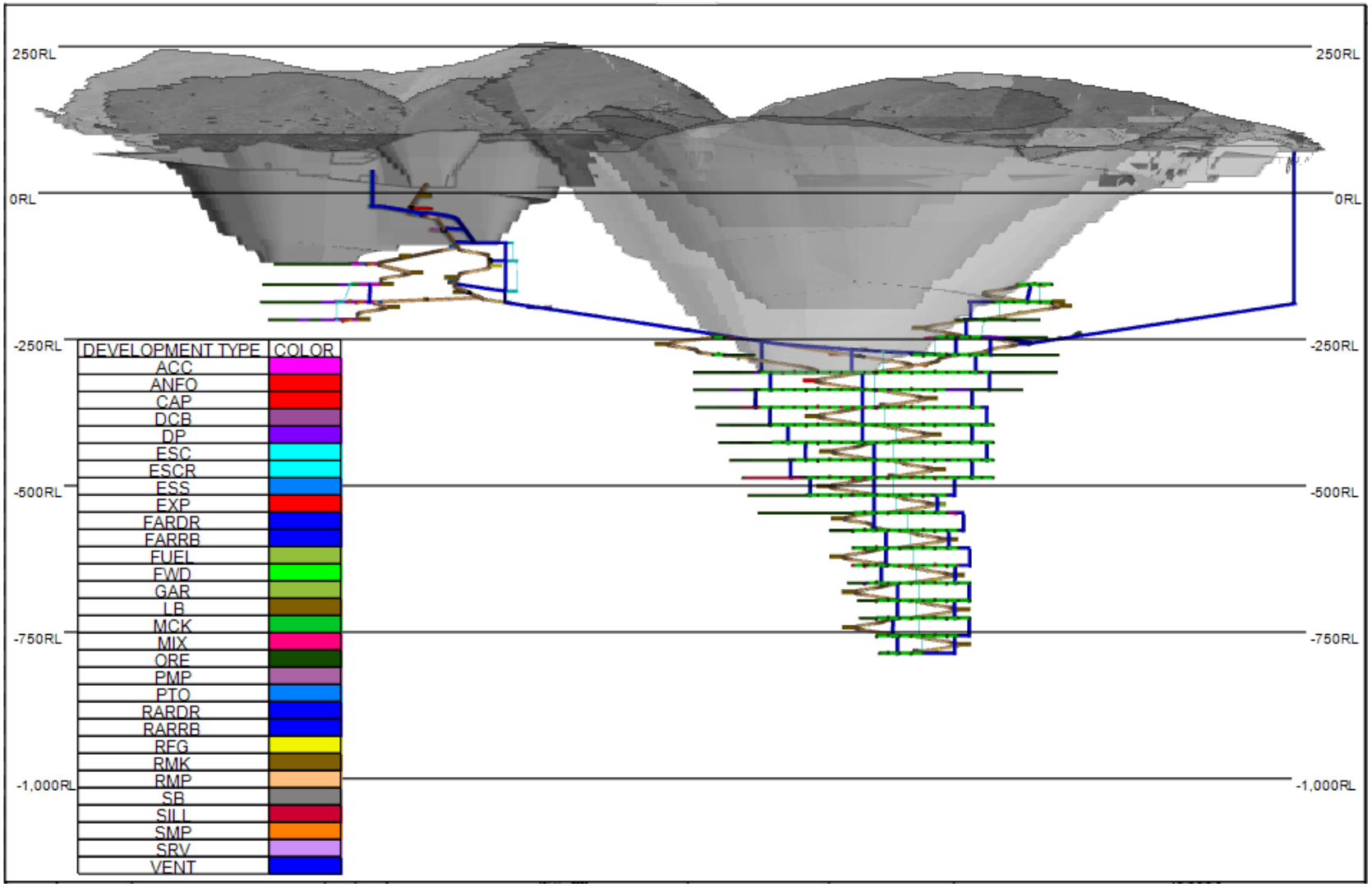
Source: GMS 03-31-2025 (not to scale)

Figure 16.28: Typical Smaller Level Plan View



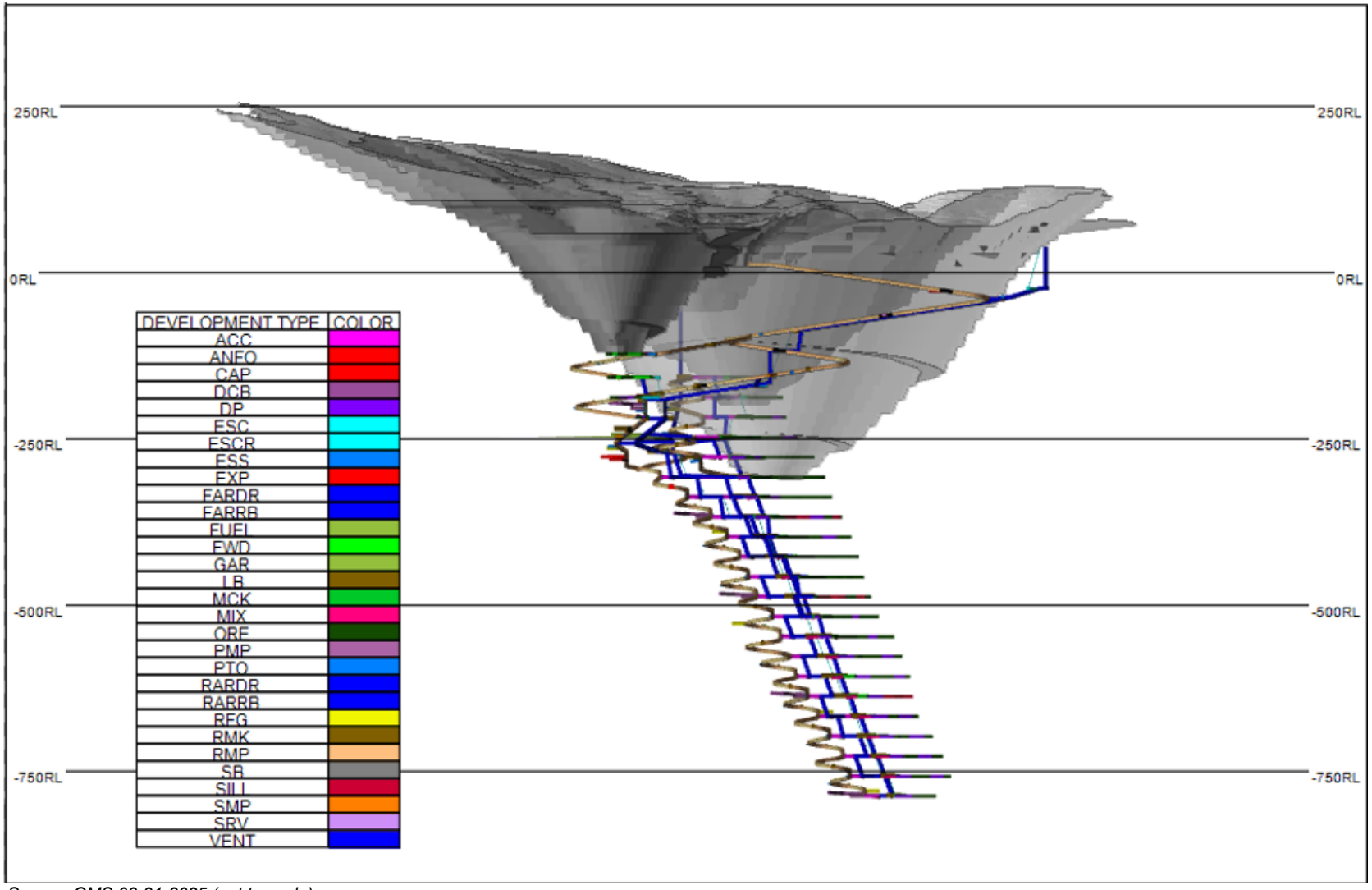
Source: GMS 03-31-2025 (not to scale)

Figure 16.29: Mine Development Longitudinal View - Looking West



Source: GMS 03-31-2025 (not to scale).

Figure 16.30: Underground Mine Development Longitudinal View - Looking North



Source: GMS 03-31-2025 (not to scale).

Table 16.17 lists some of the most relevant parameters used for mine design, while Table 16.18 lists the various development factors applied to the mine design. Table 16.19 lists the different development design criteria present in the mine and their respective dimensions.

Table 16.17: Underground Mine Lateral Development Parameters

Parameters	Units	Measurements / Specifications
Ramp Gradient	%	Nominal +/-15
Ramp Gradient at Level Intersections	%	Reduced to 10, then 5 & Flat (0) at Intersections
Turn Radius (Ramp)	m	25
Ramp - Orebody Offset (minimum)	m	80
Footwall Drift - Orebody Offset (minimum)	m	20

Table 16.18: Underground Mine Development Factors

Parameters	Units	Value
Development Allowances		
Ramp (TDB for fans, intersection slashes, cut-outs for pump boxes)	%	5
Lateral Development (CAPEX) (e.g., safety bays, TDB for fans, intersection slashes)	%	5
Lateral Development (OPEX) (e.g., TDB for fans, cutouts for remote mucking)	%	2.5
Infrastructure (e.g., slashes, TDB for fans and services)	%	2.5
Sills (Ore Drift)	%	2.5
Raises	%	0
Overbreak		
Development in Waste	%	10
Development in Mineralized Material	%	0
Raises	%	0

Table 16.19: Underground Mine Development Type and Dimensions

Development Type	Width (m)	Height (m)	Length (m)
Lateral CAPEX Development			
Decline	5.5	6.0	Variable
Level Access	5.5	6.0	Variable
Haulage Drift	5.5	6.0	Variable
Remuck	5.5	5.0	20
Safety Bay	2.0	2.0	2.0
Sump	5.0	5.0 (Flat Back)	12.5

Development Type	Width (m)	Height (m)	Length (m)
Ventilation Drift	5.5	6.0	Variable
Ventilation Raise Access	5.5	6.0	Variable
Refuge	5.0	5.0	20.0
Gear Bay	5.5	7.5	20.0
Parking Bay (Garage)	11.0	6.5	15
Wash Bay (Garage)	8.0	6.5	20.0
Maintenance Bay (Garage)	24.0	10.0	22.0
Tire Bay (Garage)	10.0	6.5	6.0
Tool Crib (Garage)	7.0	6.5	20
Electric Cut-out (Garage)	5.0	5.0	3.5
Office Bay (Garage)	5.0	5.0	6.5
Welding Bay (Garage)	5.0	5.0	3.5
Sump (Garage)	5.0	5.0	5.0
Explosive Mag	9.0	5.5	25.0
Cap Mag	6.5	5.5	10.5
Electrical Sub-Station	5.5	5.0	12.5
Safety Egress Access	5.0	5.0	Variable
Loading Bay (LHD)	5.5	5.5	11.5
Loading Bay (TRUCK)	5.5	7.5	16.0
Pumping Station	6.5	6.0	20.0
ANFO Parking	6.5	5.5	20.0
Mix Pit	5.0	5.0	16.0
Decantation Bay	5.0	5.0	Variable
Fuel Bay	10.0	6.0	25.0
Passing Bay	10.0	6.0	20
PTO	4.0	2.0	3.0
Vertical CAPEX Development			
Fresh Air Raise (Raise Bore)	7.0 Dia.		Variable
Fresh Air Raise (Drop Raise)	6.0	6.0	Variable
Return Air Raise (Raise Bore)	6.0 Dia.		Variable
Return Air Raise (Drop Raise)	6.0	6.0	Variable

Development Type	Width (m)	Height (m)	Length (m)
Emergency Egress	1.2 Dia.		Variable
OPEX Development			
Drawpoint	5.0	5.0	20
Ore Drift	5.0	5.0	Variable

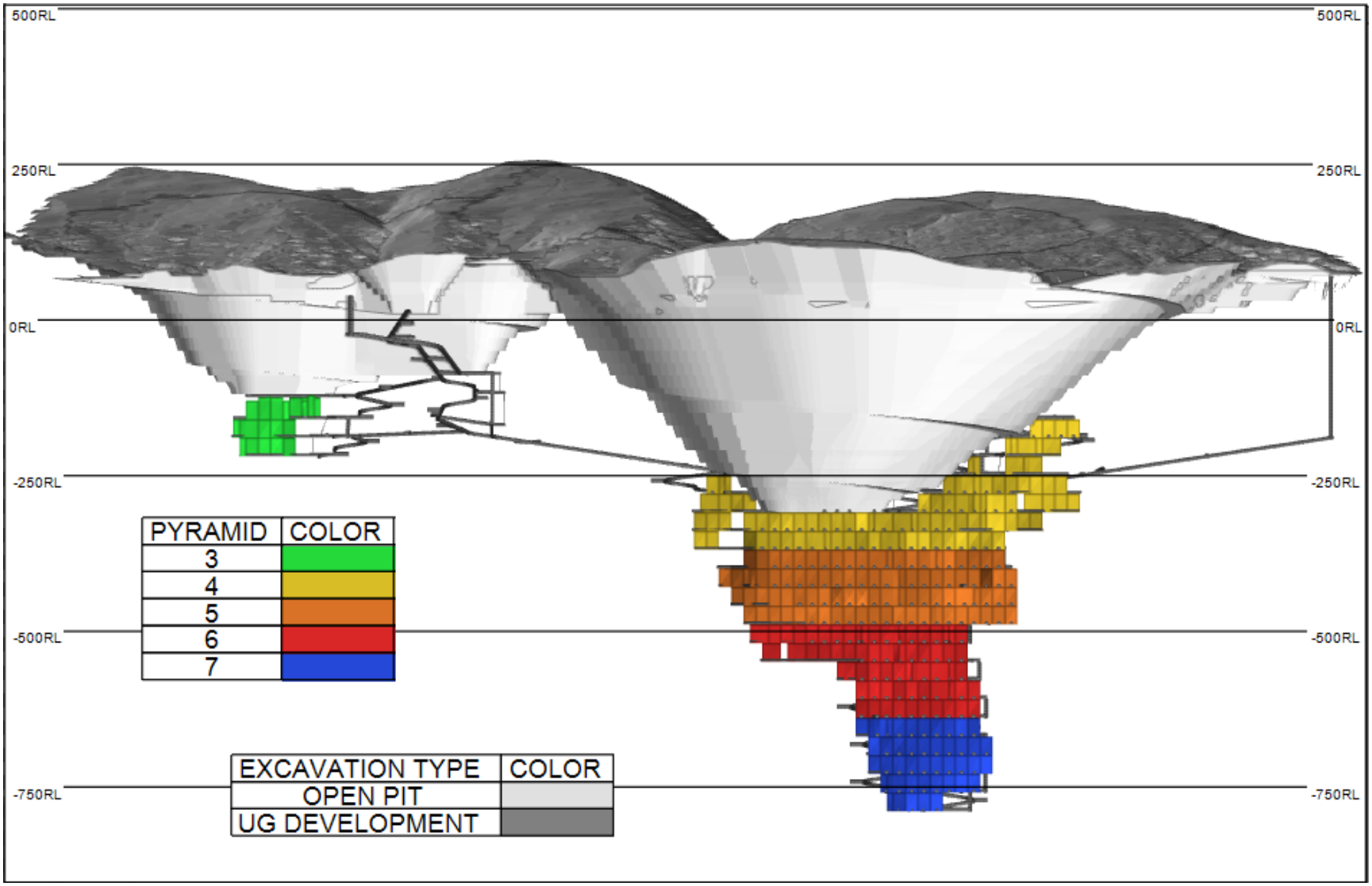
For each final level of the mining horizons in stopes using the transverse mining method, a sill-out will be developed to ensure proper breakthrough of these stopes. Additionally, on the final levels where there are underlying stopes, once these stopes have been blasted and backfilled, a sill development will be undertaken to provide access from above for drilling. These developments must be executed just-in-time to avoid ground support complications.

16.5.2.2 Stope Design

The Oko West orebody is divided into two (2) distinct mining zones, which include the main zone, located directly under the main open pit, and one (1) smaller satellite zone located south of the main zone under the south open pit. The underground mine has a total of 26 production levels. To increase operational flexibility and achieve the production target, these zones were divided into several mining blocks. Figure 16.31 and Figure 16.32 show the five (5) different mining blocks / pyramids of the underground mine.

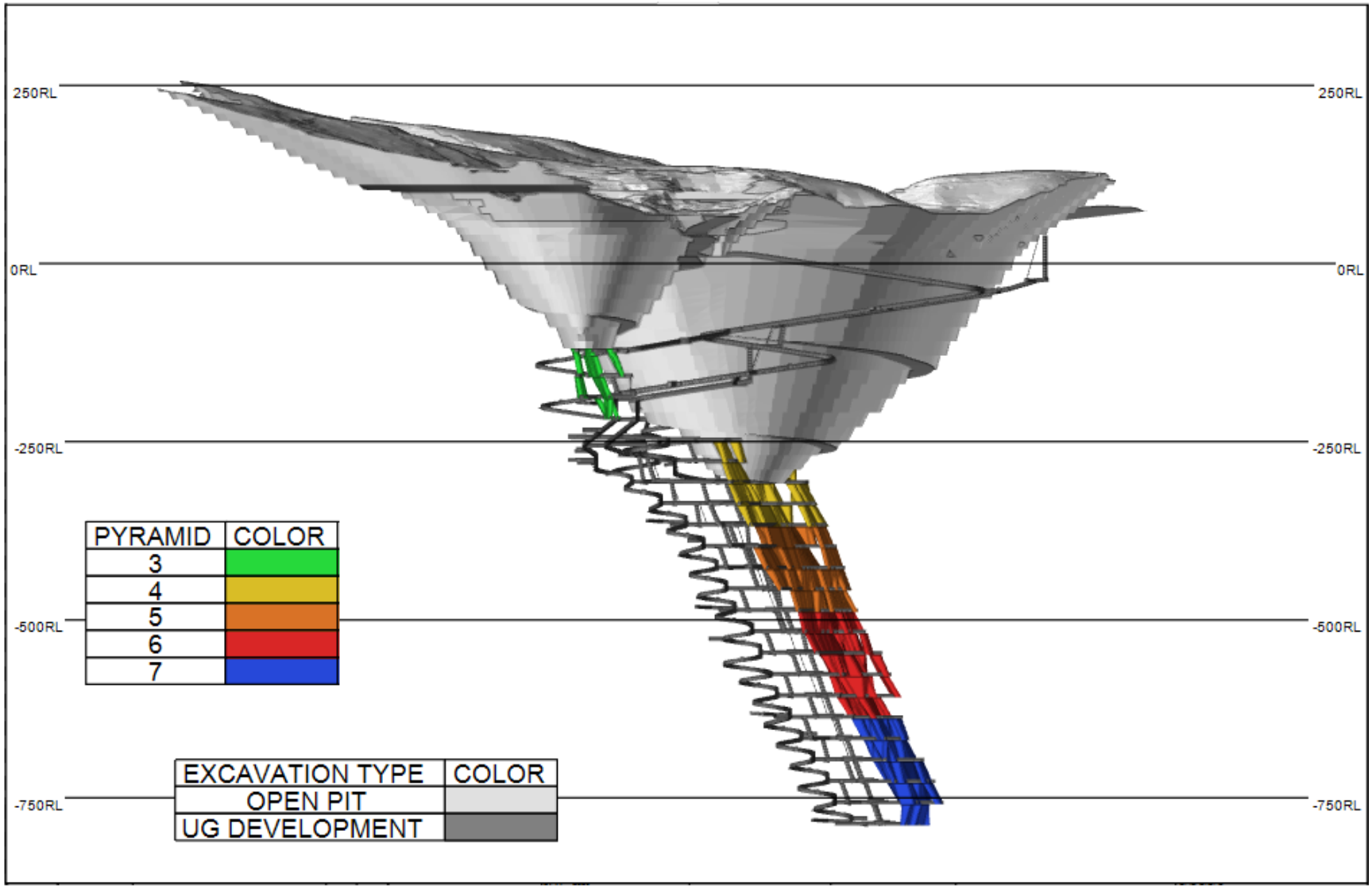
Development and production activities will occur simultaneously in all mining blocks to ensure that multiple development headings and stopes are available. The orebody is sub-vertical, with the stope thickness averaging 15.7 m and stope tonnage averaging 26,700 t. This is strongly reflected in the fact that 93% of the tonnage of the orebody is mined with the transverse stoping variant and only 7% is mined with the longitudinal stoping variant of the LHOS mining method. Figure 16.33 and Figure 16.34 both show the distribution of the mining methods in the mine. Finally, Figure 16.35Figure shows a plan view of the underground mine with the open pit.

Figure 16.31: Underground Mine Longitudinal View by Mining Blocks - Looking West



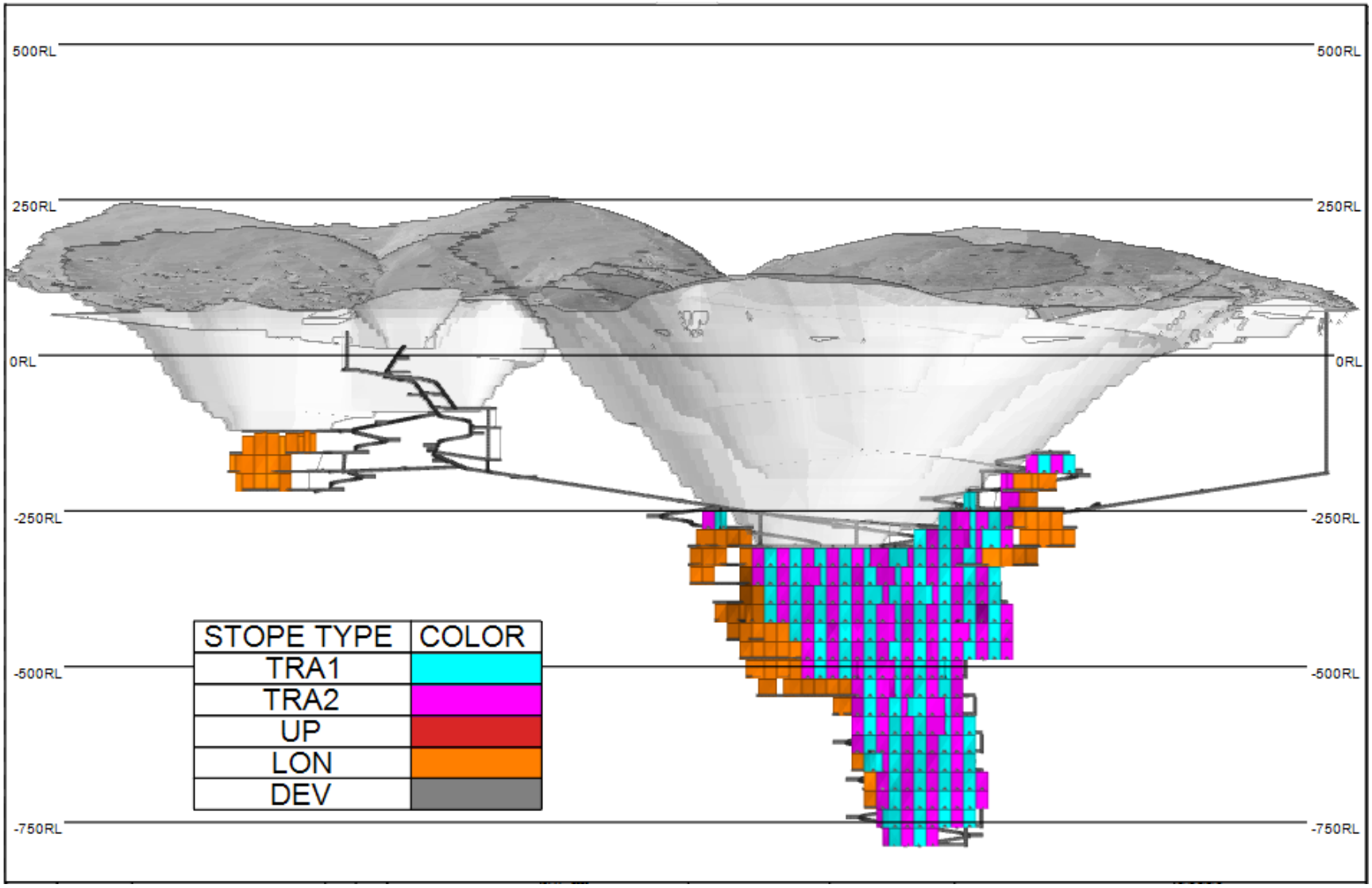
Source: GMS, 03-31-2025
**Note: Not to scale.*

Figure 16.32: Underground Mine Longitudinal View by Mining Blocks - Looking North



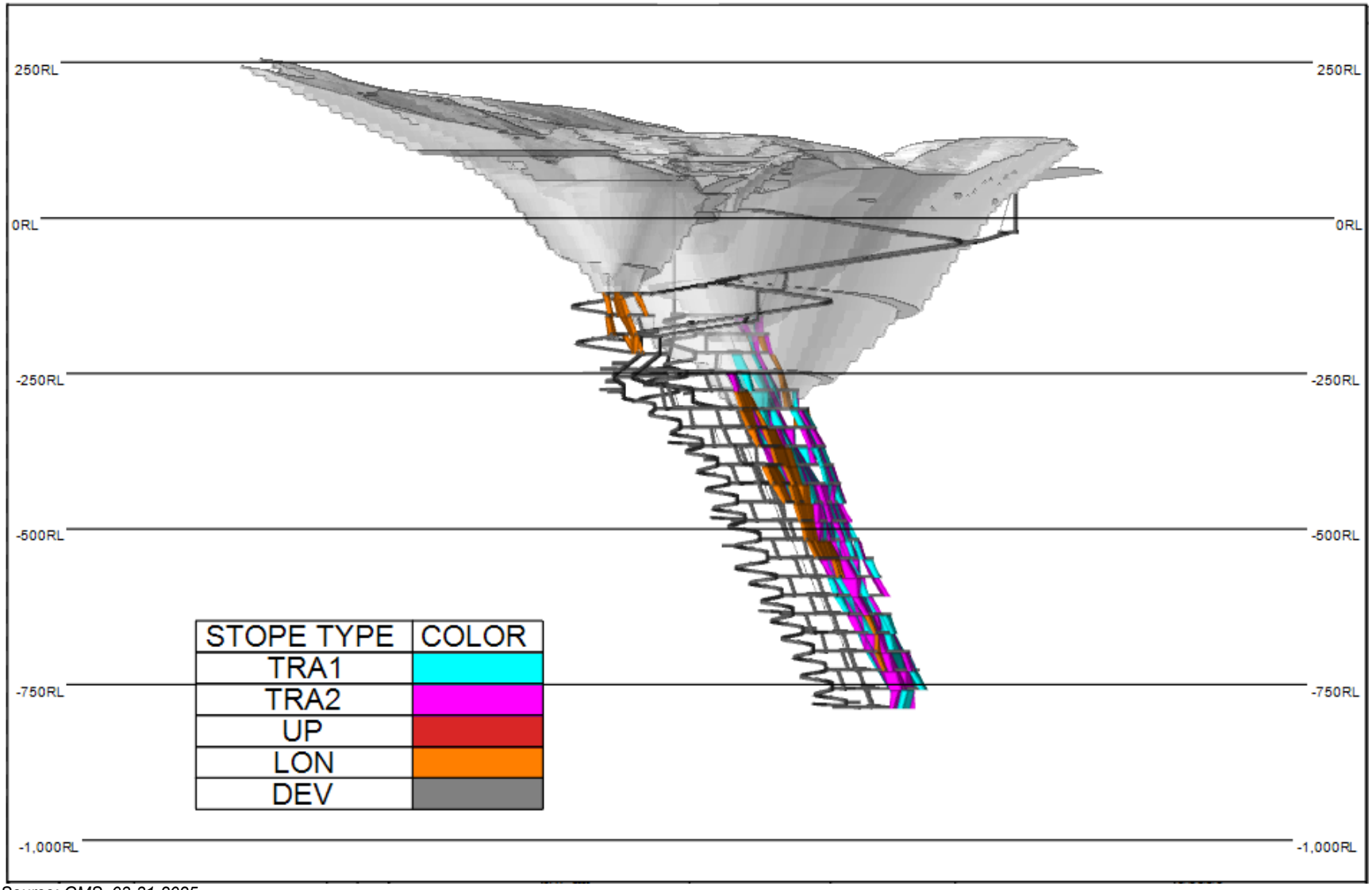
Source: GMS, 03-31-2025
**Note: Not to scale.*

Figure 16.33: Underground Mine Longitudinal View by Mining Method - Looking West



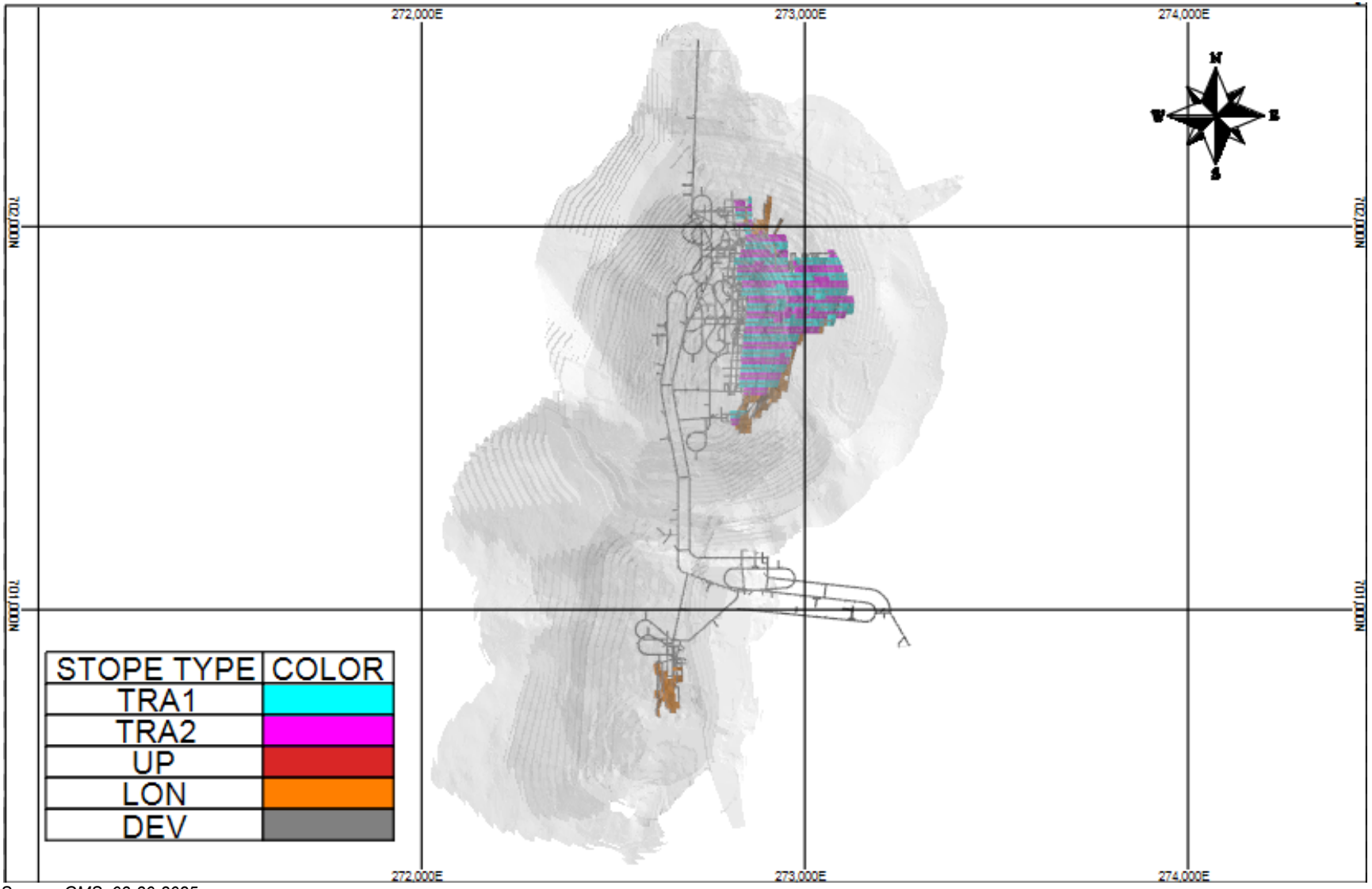
Source: GMS, 03-31-2025
**Note: Not to scale.*

Figure 16.34: Underground Mine Longitudinal View by Mining Method - Looking North



Source: GMS, 03-31-2025
**Note: Not to scale.*

Figure 16.35: Underground Mine Plan View with the Open Pit



Source: GMS, 03-30-2025
**Note: Not to scale.*

16.5.2.3 Physicals Summary

Table 16.20 summarizes the total development statistics of the underground mine. The stoping and development tonnages and grades are presented in Table 16.21, including both ore and waste development quantities.

Table 16.20: Underground Mine Development Meters Summary

Development Type	Unit	Value
Lateral Development		
Main Decline	m	8,261
Level Access	m	1,749
Haulage Drift	m	5,092
Infrastructures	m	11,014
Sub-total CAPEX Development	m	26,116
OPEX Development	m	25,727
Total Lateral Development	m	51,843
Vertical Development		
Fresh Air Raise - Raise Bore	m	62
Fresh Air Raise - Drop Raise	m	1,001
Return Air Raise - Raise Bore	m	260
Return Air Raise - Drop Raise	m	520
Emergency Egress	m	798
Total Vertical Development	m	2,641

Table 16.21: Underground Mine Physical Quantities Summary

Parameters	Unit	Value
Development		
Development Waste	t	3,041,382
Development Ore	t	973,148
	g/t Au	2.75
Production		
Stoping Ore	t	13,207,319
	g/t Au	3.30

Parameters	Unit	Value
Total		
Total Underground Ore	t	14,180,467
	g/t Au	3.26

16.5.2.4 Development and Production Rates

The targeted underground mine production rate is set at 4,625 tpd or 1.7 Mt of ore per year, including stope production (4,500 tpd) and lateral development within mineralized material (125 tpd).

To ensure efficient production, loading bays were included in the mine design on each side of the larger production levels. They are designed with two (2) drifts, one (1) of them used to position the truck for loading and the second one, excavated perpendicularly and higher, to allow the LHD to load the trucks easily. The production rate varies slightly since the quantity of development ore produced is not constant over the LoM. Multiple mining blocks are mined simultaneously to maintain the targeted underground mine production rate. The production rate for the Oko West UG mine is calculated using the Deswik™ mining sequence, considering the different rates shown in Table 16.22.

Table 16.22: Underground Mine Scheduler Rates

Parameters	Unit	Value
Single Face Development Rate	m/d/unit	5.0
Multi Face Development Rate	m/d/unit	10.0
Stope Preparation	d	5
Stope Cables	m/d	120
Slot Raise Drilling Rate	m/d	5
Production Drilling Rate - Transverse	m/d	200
Production Drilling Rate – Longitudinal & Drop Raise	m/d	150
Production Drilling Factor - Transverse	t/m drilled	10.0
Production Drilling Factor - Longitudinal	t/m drilled	8.0
Blasting Delay	d	3
Mucking Rate	t/d	1,400
Rockfill Rate	t/d	1,200
Cemented Rockfill Rate	t/d	1,000
Short Cure Time	d	7

Parameters	Unit	Value
Long Cure Time	d	28
Maximum Stoping	t/d	4,500

16.5.3 Development and Production Sequencing

Following the completion of the portal excavation and construction, the project will transition into the development phase, beginning with the advancement of the main decline toward the satellite zone.

During the first year of development, a progressive ramp-up in performance is planned. The development rate will increase from 25% to 100% of capacity by increments of 25% every quarter.

The development of the primary ventilation and safety egress networks is a priority as they are essential to allow stoping to begin. The main priority remains the development of the ramp to access the main zone, and particularly the bottom of the first mining block in the main zone. Access to the main zone is critical for transitioning from initial production to full production, as it is the largest zone and essential for sustaining the full production rate. Also, it is important to complete the stope near the pit as quickly as possible in order not to interfere with pit operations.

Stoping in the satellite zone begins approximately two and a half (2.5) years after the start of development. The satellite zone will be in production for 13 months, and the main zone will be in production through the entire LoM. The mineralized material production profile from the underground mine is summarized by zone in Figure 16.36Figure . The production plan for mineralized material from the underground mine is presented in Table 16.23. The development plan is presented in Table 16.24.

Figure 16.36: Underground Mine Mineralized Material Production by Zone

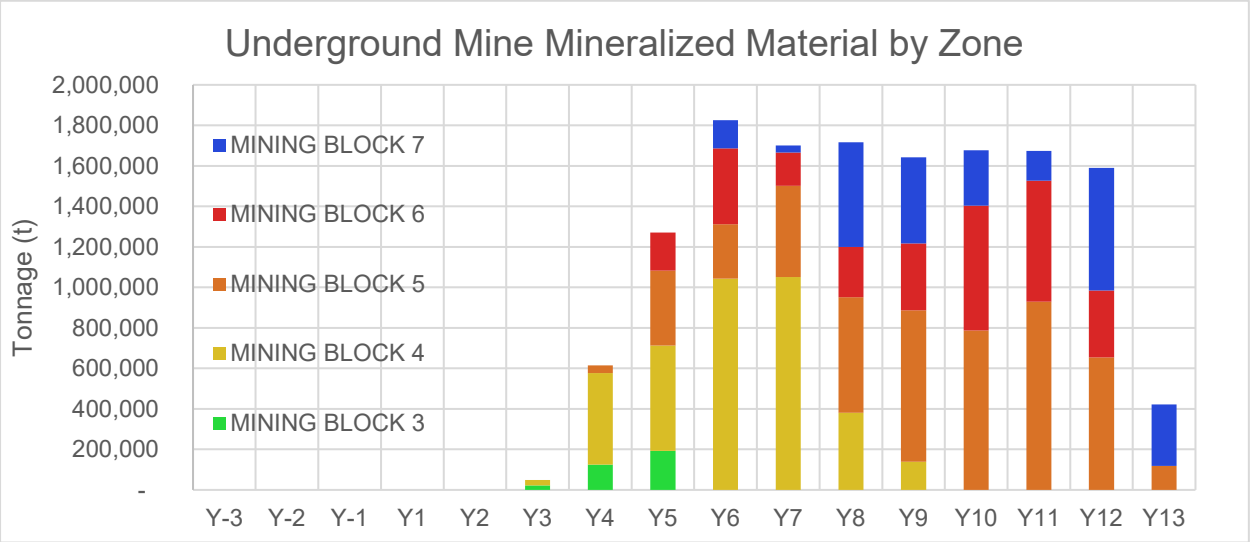


Table 16.23: Underground Mine Production Plan

Parameters	Units	Total	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Development Mineralized Material	kt	973	-	-	-	-	-	49	237	197	183	59	88	68	34	31	26	-
	g/t Au	2.75	-	-	-	-	-	2.87	2.33	2.95	2.84	2.83	2.26	3.33	2.89	2.77	4.00	-
	koz	86	-	-	-	-	-	5	18	19	17	5	6	7	3	3	3	-
Stoping Mineralized Material	kt	13,207	-	-	-	-	-	-	378	1,073	1,642	1,642	1,627	1,573	1,643	1,643	1,564	422
	g/t Au	3.30	-	-	-	-	-	-	3.15	3.23	3.38	3.13	3.26	3.19	3.19	3.19	3.57	4.30
	koz	1,400	-	-	-	-	-	-	38	111	178	165	171	162	169	168	179	58
Total Underground Mineralized Material	kt	14,180	-	-	-	-	-	49	615	1,270	1,826	1,701	1,716	1,641	1,677	1,674	1,590	422
	g/t Au	3.26	-	-	-	-	-	2.87	2.83	3.18	3.32	3.12	3.21	3.20	3.19	3.18	3.58	4.30
	koz	1,486	-	-	-	-	-	5	56	130	195	171	177	169	172	171	183	58

Table 16.24: Underground Mine Development Plan

Parameters	Units	Total	Y -3	Y -2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Lateral Development																		
Main Decline	m	8,261	-	-	-	340	1,377	2,737	926	1,231	973	228	448	-	-	-	-	-
Level Access	m	1,749	-	-	-	-	-	691	337	293	278	49	101	-	-	-	-	-
Haulage Drift	m	5,092	-	-	-	-	-	438	1,506	1,655	1,020	78	302	93	-	-	-	-
Infrastructures	m	11,014	-	-	-	102	1,786	2,840	1,906	1,899	1,752	141	507	81	-	-	-	-
Sub-total CAPEX Development	m	26,116	-	-	-	442	3,163	6,706	4,675	5,078	4,022	496	1,359	174	-	-	-	-
OPEX Development	m	23,486	-	-	-	-	-	1,618	5,749	5,306	4,607	1,601	1,734	1,287	582	537	466	-
Total Lateral Development	m	49,602	-	-	-	442	3,163	8,324	10,424	10,385	8,629	2,097	3,093	1,461	582	537	466	-

Parameters	Units	Total	Y -3	Y -2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Vertical Development																		
Raise Bore	m	260	-	-	-	-	-	141	119	-	-	-	-	-	-	-	-	-
Drop Raise	m	1,583	-	-	-	-	139	156	352	374	391	29	86	55	-	-	-	-
Emergency Egress	m	798	-	-	-	-	144	111	56	174	233	26	54	-	-	-	-	-
Total Vertical Development	m	2,641	-	-	-	-	283	408	527	548	624	55	141	55	-	-	-	-

16.5.4 Underground Mine Equipment

The requirements in terms of underground equipment were determined based on the number of operating hours needed to achieve the projected production and development rates outlined in the mine plan.

During the production years, haulage cycles consider the distances from the loading point in the footwall drifts to the level access, followed by the ramp ascent to the surface stockpile. Mucking and hauling cycles are determined based on the average distance between stopes and the loading point of trucks.

The quantities of auxiliary equipment were estimated based on the size of the operation or derived from other equipment requirements. Table 16.25 shows the results of the equipment requirements for the mine pre-production and the full-production stages of the LoM.

Table 16.25: Underground Mine Mobile Equipment Fleet

Equipment	Qty Pre-Production	Qty Production
Jumbo – 2 Boom	1	3
Bolter (plate form bolter)	2	7
Production Drill – Hydraulic Top Hammer	-	3
Production Drill – ITH	1	1
Cable Bolter	1	1
LHD – 21T	1	7
LHD – 10T	1	2
Truck – 63T	1	12
Explosive Truck - Development	1	1
Explosive Truck - Production	-	2
Scissor Lift - Development	1	2
Scissor Lift - Construction	1	2
Attachment - Fan Handler	1	1
Boom Truck	1	2
Fuel & Lube Truck	-	1
Water Truck	-	1
Block Holer	-	1
Cassette Truck	1	2
Cassette - Crane	-	1

Equipment	Qty Pre-Production	Qty Production
Cassette - Emulsion Loading Deck	-	1
Cassette - Flat Deck	-	1
Cassette - Fuel & Lube	-	1
Shotcrete Sprayer	1	1
Shotcrete Mixer	1	1
Grader	-	1
Light Vehicle	5	25
Light Vehicle – Mine Rescue	1	1
Tractor - Mechanics	1	2
Tractor - Electricians	0	2
Mobile Air Compressor	1	1
Backhoe Loader	1	2
Shotcrete and Concrete Transport	-	4
V-30	1	1

16.5.5 Underground Mine Ventilation

Ventilation requirements for the underground mine are primarily based on diesel emissions from the equipment fleet. The Canmet dilution factor was used to determine the required airflow per equipment. When Canmet dilution rates were not available, the minimum dilution rate of 0.06 m^3/kW was used. A utilization factor was also applied to assess the required CFM. This factor was determined based on the duration and intensity of diesel-powered equipment operation underground, Tabledetailing the typical fresh air ventilation requirements for equipment. Ventsim™ designs have been created and simulations performed for the maximum production scenario.

Table 16.26: Underground Mine Fresh Air Requirements per Equipment

Equipment	Engine	HP	CFM/EQUIP	Utilization Factor	CFM Requirement
Jumbo – 2 Boom	Deutz TCD3.6, 100 kW, Tier 3, Stage IIIA	134	5,100	40%	2,040
Bolter	Cummins QSB4.5, 127 kW, Tier 3, Stage IIIA	170	13,600	40%	5,440
Production Drill - Top Hammer	Cummins QSB4.5, 119 kW, Tier 3, Stage IIIA	160	13,600	40%	5,440

Equipment	Engine	HP	CFM/EQUIP	Utilization Factor	CFM Requirement
Production Drill – Raise Bore	Deutz TCD 2012, 155 kW, Tier 3, Stage IIIA	208	19,706	40%	7,882
Cable Bolter	Cummins QSB4.5, 119 kW, Tier 3, Stage IIIA	160	13,600	40%	5,440
LHD - Type I	Volvo TAD1344VE, Tier 2, Stage II	472	28,700	75%	21,525
LHD - Type II	Volvo TAD1340VE, Tier 2, Stage II	343	20,500	75%	15,375
Truck - Type I	Volvo TAD1643VE-B, Tier 2, Stage II	758	54,000	75%	40,500
Explosive Truck – Development	Cummins QSB4.5, 127 kW, Tier 3, Stage IIIA	170	13,600	60%	8,160
Explosive Truck – Production	Cummins QSB4.5, 127 kW, Tier 3, Stage IIIA	170	13,600	60%	8,160
Scissor Lift – Development	Cummins QSB4.5, 127 kW, Tier 3, Stage IIIA	170	13,600	60%	8,160
Scissor Lift - Construction	Cummins QSB4.5, 127 kW, Tier 3, Stage IIIA	170	13,600	60%	8,160
Boom Truck	Cummins QSB6.7, 144 kW, Tier 3, Stage IIIA	193	18,000	60%	10,800
Personnel Carrier	Cummins QSB4.5, 127 kW, Tier 3, Stage IIIA	170	13,600	60%	8,160
Deck Truck	Cummins QSB6.7, 144 kW, Tier 3, Stage IIIA	193	18,000	60%	10,800
Fuel & Lube Truck	Cummins QSB6.7, 144 kW, Tier 3, Stage IIIA	193	18,000	60%	10,800
Water Truck	Cummins QSB4.5, 127 kW, Tier 3, Stage IIIA	170	13,600	60%	8,160
Block Holer	Cummins QSB4.5, 127 kW, Tier 3, Stage IIIA	170	13,600	60%	8,160
Scaler	Cummins QSB4.5, 127 kW, Tier 3, Stage IIIA	170	13,600	60%	8,160
Cassette Truck	Cummins QSB4.5, 127 kW, Tier 3, Stage IIIA	170	13,600	60%	8,160
Shotcrete Sprayer	Cummins QSB4.5, 127 kW, Tier 3, Stage IIIA	170	13,600	60%	8,160
Shotcrete Mixer	Cummins QSB4.5, 127 kW, Tier 3, Stage IIIA	170	13,600	60%	8,160

Equipment	Engine	HP	CFM/EQUIP	Utilization Factor	CFM Requirement
Grader	Cummins QSB4.5, 119 kW, Tier 3, Stage IIIA	170	13,600	60%	8,160
Light Vehicle	Toyota 1HZ PCNA	127	7,300	60%	4,380
Light Vehicle - Mine Rescue	Toyota 1HZ PCNA	127	7,300	60%	4,380
Tractor – Mechanics	V3800-TIEF4ZA6 105 HP (78 kW)	105	4,100	60%	2,460
Tractor – Electricians	V3800-TIEF4ZA6 105 HP (78 kW)	105	4,100	60%	2,460
Tractor	V3800-TIEF4ZA6 105 HP (78 kW)	105	4,100	60%	2,460
Backhoe Loader	SAA4D104E-1	99	9,408	60%	5,645
Shotcrete and Concrete Transport	Cummins QSB6.7, 144 kW, Tier 3, Stage IIIA	193	18,000	60%	10,800

Table 16.27 provides estimates of fresh air requirements to ventilate the Oko West UG mine.

Table 16.27: Underground Mine Fresh Air Requirements

Equipment	Qty Total	CFM Requirement
Jumbo – 2 Boom	3	6,120
Bolter	7	43,520
Production Drill – Type I	3	16,320
Production Drill – Type II	1	7,882
Cable Bolter	1	5,440
LHD – Type I	7	172,200
LHD – Type II	2	30,750
Truck – Type I	12	567,000
Explosive Truck - Development	1	8,160
Explosive Truck - Production	2	16,320
Scissor Lift - Development	2	16,320
Scissor Lift - Construction	2	16,320
Boom Truck	2	21,600

Equipment	Qty Total	CFM Requirement
Fuel & Lube Truck	1	10,800
Water Truck	1	8,160
Block Holer	1	8,160
Cassette Truck	2	16,320
Shotcrete Sprayer	1	8,160
Shotcrete Mixer	1	8,160
Grader	1	8,160
Light Vehicle	25	109,500
Light Vehicle - Mine Rescue	1	4,380
Tractor - Mechanics	2	4,920
Tractor - Electricians	2	4,920
Backhoe Loader	2	11,289
Shotcrete and Concrete Transport	4	43,200
Fresh Air Requirements (Sub-Total)	**-**	**1,174,082**
Contingency (10%)	**-**	**117,408**
Fresh Air Requirements (Total)	**-**	**1,291,490**

A phased approach will be used to implement the ventilation system. Initially, a temporary system will be put in place to supply fresh air and allow the development of the main decline until the permanent ventilation network is developed and commissioned.

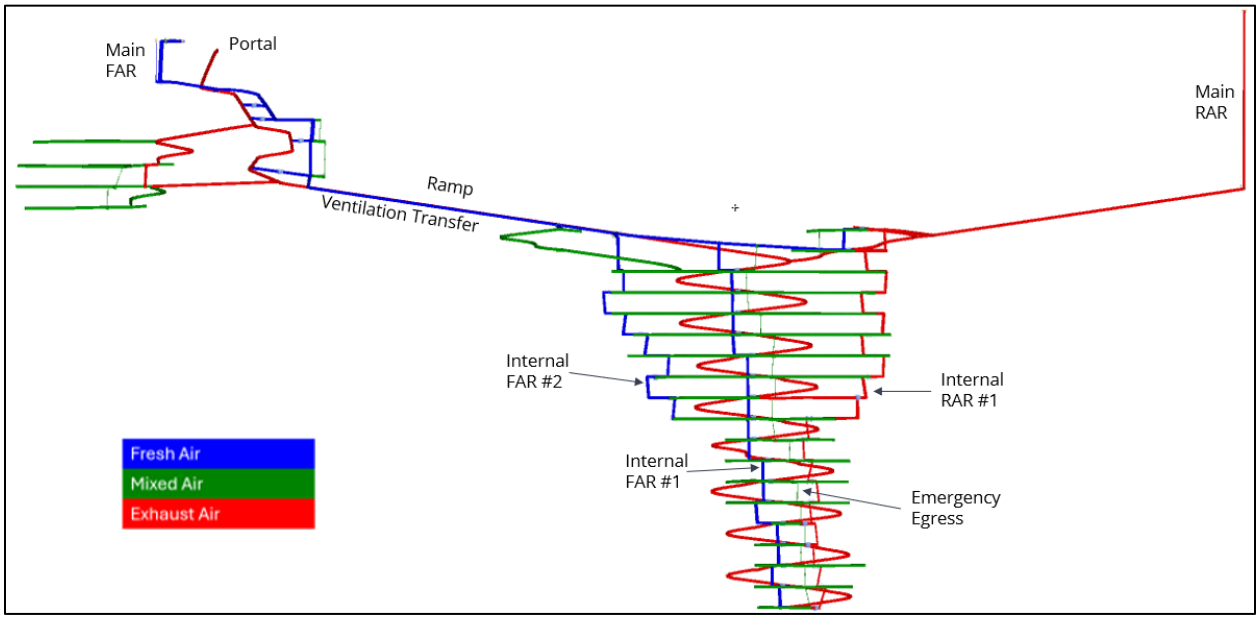
The temporary ventilation system is designed to provide fresh air requirements for one (1) LHD and two (2) haul trucks. This fresh air volume will be provided by two (2) 54” rigid ventilation ducts. Fresh air is supplied to each ventilation line by a pair of axial fans installed in series.

The permanent system is designed to accommodate the fresh air requirements of the initial pre-production phase and the ramp-up to full production. The proposed permanent ventilation system is a push ventilation system that consists of one (1) fresh air raise (FAR) intake located east of the south pit near the primary crusher. A box cut will be dug to allow the excavation of the FAR directly into fresh rock. Two (2) primary fans will be installed in parallel on the surface, both directing airflow into the mine through the FAR. A transfer ventilation drift, running parallel to the decline, will be excavated to facilitate the movement of fresh air from the satellite zone to the main zone. For exhaust ventilation, one (1) return air raise (RAR) located north of the main zone, along with the main decline, will be utilized.

A total of 1.29M cfm is required to provide sufficient air for the workforce and equipment underground.

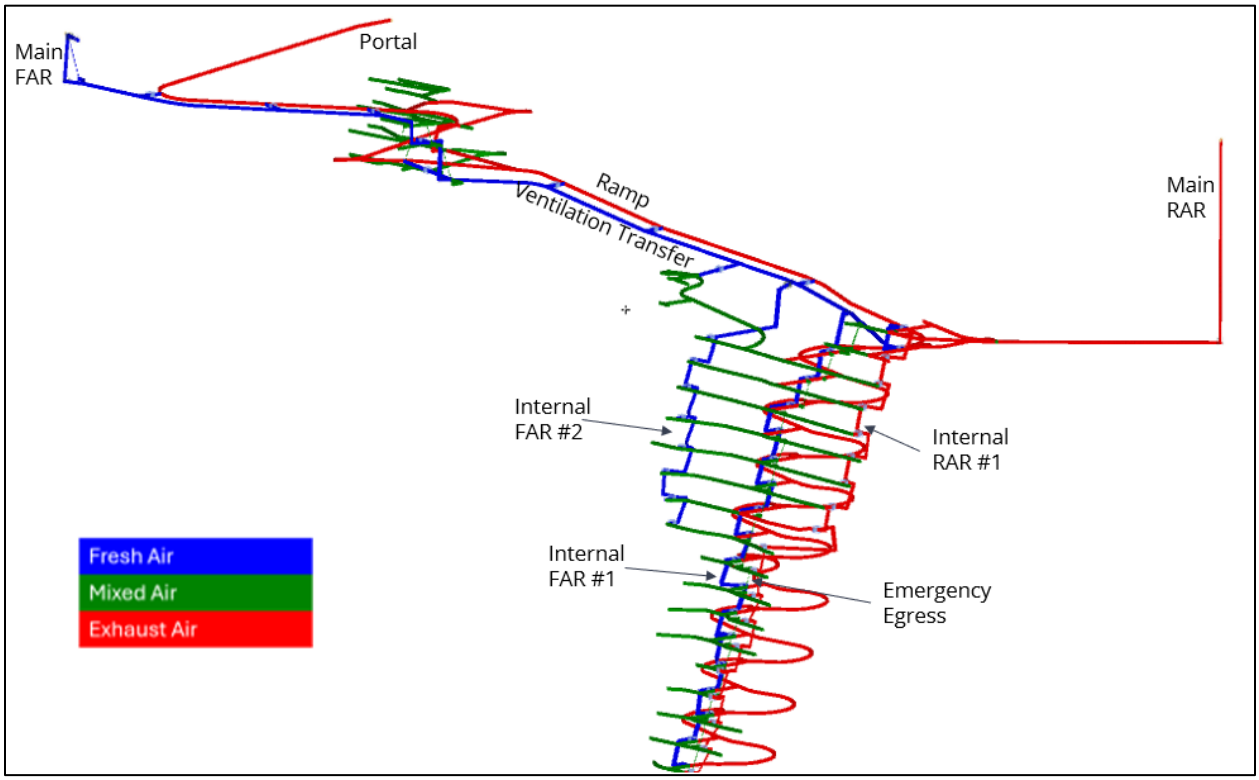
Figure 16.37 and Figure 16.38 both illustrate the permanent underground mine ventilation network.

Figure 16.37: Underground Mine Ventilation Network – Looking West



Source: GMS, 03-30-2025
**Note: Not to scale.*

Figure 16.38: Underground Mine Ventilation Network – Isometric View



Source: GMS, 03-30-2025
**Note: Not to scale.*

Both the main FAR and main RAR ventilation raises will be excavated with a raise-boring machine from surface and are varying in length and diameter. The FAR and RAR are then converted into a series of drop raises to deliver fresh air at depth.

This permanent ventilation system will be operating at a variety of pressures and flows to suit the various operating conditions of the mine. The installation of ventilation louvers at the fresh air raise access on every level will ensure that an adequate amount of fresh air is distributed to the correct workplace. The larger production levels in the main zone will receive fresh air through two (2) FARs—one positioned centrally and the other in the southern section. The southern FAR will terminate at level 520. This setup allows to reduce the pressure in the mine and ensure flow-through ventilation. Access to the RAR network is located north of the level access.

Table 16.28 summarizes the different fan design and cooling parameters.

Table 16.28: Underground Mine Ventilation & Cooling System Details

Parameters	Units	Value
Main Fan System		
Fans	kW	2 x 1,035
Pressure	In. wg.	16.0
Airflow	cfm	1,300,000
Temporary Fan System		
Fans	kW	4 x 300
Pressure	In. wg.	12.0
Airflow	cfm	150,000

16.5.6 Underground Mine Cooling

Due to the presence of multiple heat sources in the underground mine, such as auto-compression, heat from groundwater inflow, heat exchange with surrounding strata, and equipment-related heat from diesel combustion, a cooling system is required to maintain temperatures below the considered threshold of 28°C wet-bulb in deep-level operations.

The mine cooling system will include a surface bulk air cooling plant consisting of a bulk air cooler (BAC) to lower the temperature of the incoming ventilation air, a refrigeration plant that supplies chilled water to the BAC, and a condenser cooling tower that efficiently dissipates the extracted heat into the atmosphere.

The BAC follows a conventional surface cooling design, drawing ambient air directly into the cooler. Once the air is cooled, it is directed through the main surface fans and delivered underground to the working areas. The BAC is designed to deliver more cooling capacity than the minimum required, compensating for additional thermal loads from fan heat compression and solar radiation.

Table 16.29 summarizes the different fan design and cooling parameters.

Table 16.29: Underground Mine Cooling System Details

Parameters	Units	Value
System	Name	Bulk Air Cooler
Cooling	kW	4,420
Process	MW cooling	15.5

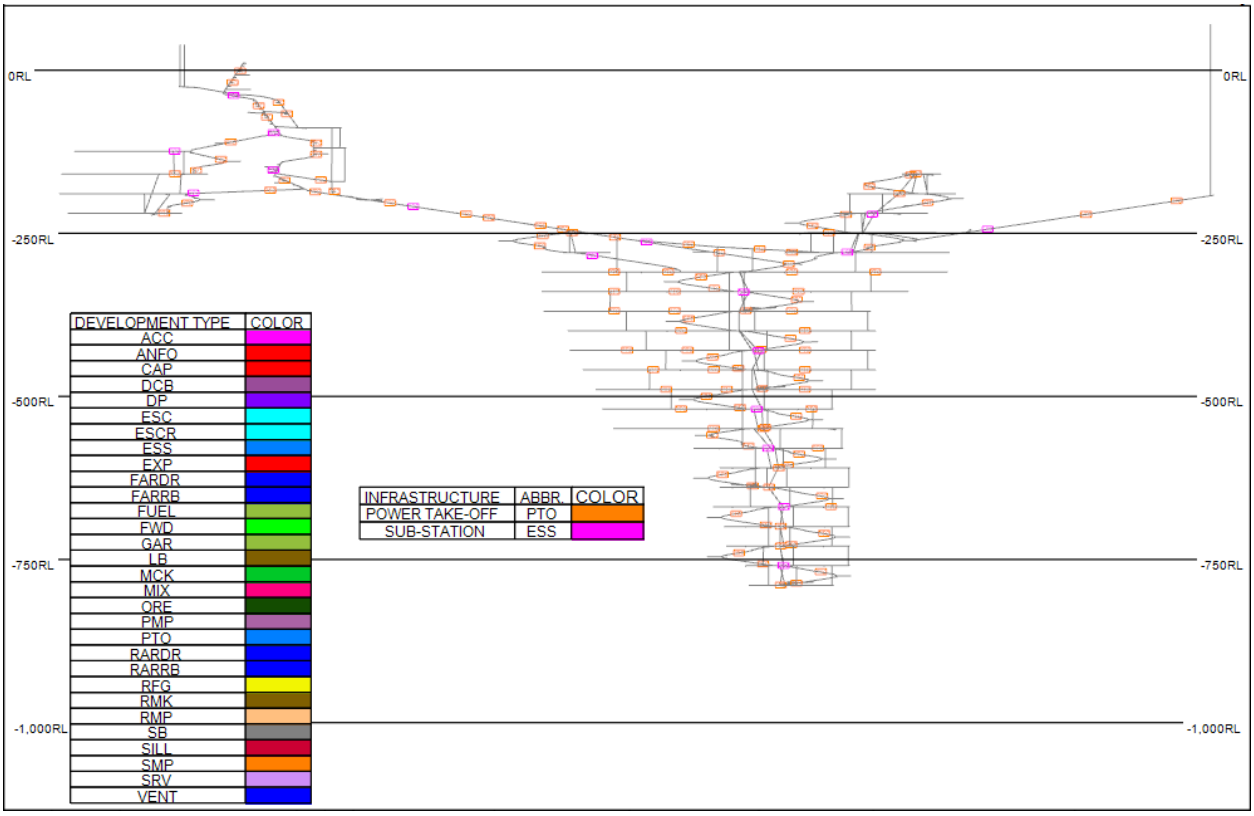
Parameters	Units	Value
Process	kW installed	1 x 5,000

16.5.7 Underground Mine Services

16.5.7.1 Underground Electrical Distribution

The power generation infrastructures will be located close to the processing plant and will power the underground mine with a 13.8 kV, 60 Hertz (HZ) distribution network stepping down to a voltage of 600 V. The permanent system will bring power by cable passing through service holes from surface to underground near the main Fresh Air Raise, then the cable will be installed via the decline to bring power to the main zone. During decline development, one (1) mobile sub-station will be installed every 380 m. As development progresses, the substation will be relocated to the subsequent electrical bay. Near infrastructures with high power requirements, such as pumping stations or the garage, mobile substations will initially be used to power the development and then left in place to supply the infrastructures. For the main area, an electrical bay will be developed at each of the three (3) production levels. Each substation will supply its level as well as the immediate upper and lower levels. To supply the upper and lower levels, the electrical cables will pass through service holes. Figure 16.39 illustrates the power distribution network.

Figure 16.39: Underground Electrical Distribution Network



16.5.7.2 Dewatering

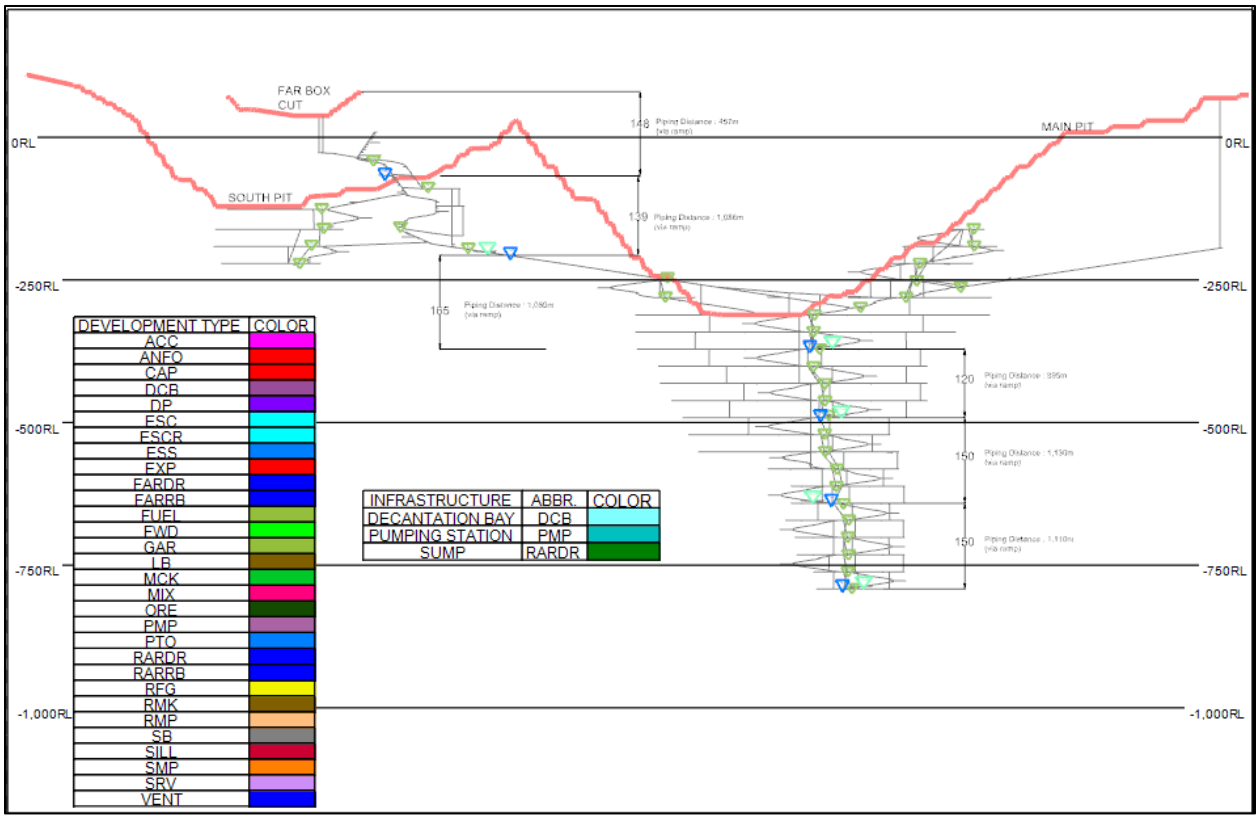
A mine water balance was completed for the entire mine, and the overall flows were attributed to each zone of the mine, based on the production rate and natural groundwater inflow assumptions based on ERM reports. Mine operations' water consumption was calculated based on the equipment lists and their respective water consumption. In each mine block, water from all production levels will flow by gravity to the lower point of the level, the sump at the entry and then descend to the deepest production level sump through a series of drain holes. From there, dirty water will be pumped to a decantation bay located on the other side of level access within the ramp. A permeable membrane will then separate clean water from slimes by trapping fine particles. The filtered water will flow into a clean water reservoir in the pumping station. Then the water will be pumped to surface by a series of five (5) pumping stations to a surface pond. To reduce power requirements for the pumping station of mining block #3, an intermediate pumping station will be installed between mining block #3 and the surface. Water stored in the surface pond will then be reused for the mine operations' water supply.

Table 16.30 shows the underground mine dewatering assumptions, while Figure 16.40 illustrates the dewatering network, and Table 16.31 details the pumping requirements.

Table 16.30: Underground Mine Dewatering Assumptions

Parameters	Unit	Value
Mine Operations Water	USGPM	595
	(l/min)	2,252
Natural Ground Water	USGPM	459
	(l/min)	1,739
Total Dewatering	USGPM	991
	(l/min)	3,752

Figure 16.40: Underground Mine Dewatering Network



Source: GMS 03-30-2025 (not to scale).

Table 16.31: Underground Mine Pumping Requirements Details

Parameters	Units	Value
Intermediate Pumping Station (090L)		
Pipe Length	m	457
Dewatering Flow Capacity	USGPM	2,478
Total Head	m	152

Parameters	Units	Value
Pump Efficiency	%	75%
Pump Power	hp	416
Mining Block #3		
Pipe Length	m	1,086
Dewatering Flow Capacity	USGPM	2,121
Total Head	m	179
Pump Efficiency	%	75%
Pump Power	hp	419
Mining Block #4		
Pipe Length	m	1,050
Dewatering Flow Capacity	USGPM	2,121
Total Head	m	175
Pump Efficiency	%	75%
Pump Power	hp	410
Mining Block #5		
Pipe Length	m	895
Dewatering Flow Capacity	USGPM	1,963
Total Head	m	138
Pump Efficiency	%	75%
1Pump Power	hp	299
Mining Block #6		
Pipe Length	m	1,130
Dewatering Flow Capacity	USGPM	1,804
Total Head	m	174
Pump Efficiency	%	75%
Pump Power	hp	346
Mining Block #7		
Pipe Length	m	1,110
Dewatering Flow Capacity	USGPM	1,274
Total Head	m	161
Pump Efficiency	%	75%

Parameters	Units	Value
Pump Power	hp	225

16.5.7.3 Cemented Rockfill

Cemented rockfill (CRF) has been selected as the fill material for the underground mine. CRF is a mix of waste rock and cement slurry. In this case, the cement slurry will be produced on surface at the same concrete batch plant used for construction purposes. Cement bulk bags will be transported to the plant by surface trucks to feed the batch plant and produce cement slurry. Cement slurry will be transported underground to the desired location by transmixers and/or agitator trucks. The waste rock for rockfill and cemented rockfill will primarily be sourced from development waste, while any additional waste required will be obtained from surface operations. Waste material sourced from surface operations will be crushed to minus 6 inches and backhauled underground in the same trucks that transported the mineralized material to the surface. On production levels, the truck will dump waste into a draw point that is mucked out. Transmixer will dump cement slurry into a cement slurry reservoir located at another identified drawpoint. Cement will be pumped to the mixing pit, and an LDH will add the waste into the mixing pit to reach the required cement content for the stope. The planned cement content is 5% for regular stopes and 7% for sill level stopes, where stopes will be mined under a backfilled stope. Figure 16.41 and Figure 16.42 illustrate the CRF process.

Figure 16.41: General CRF Process Map

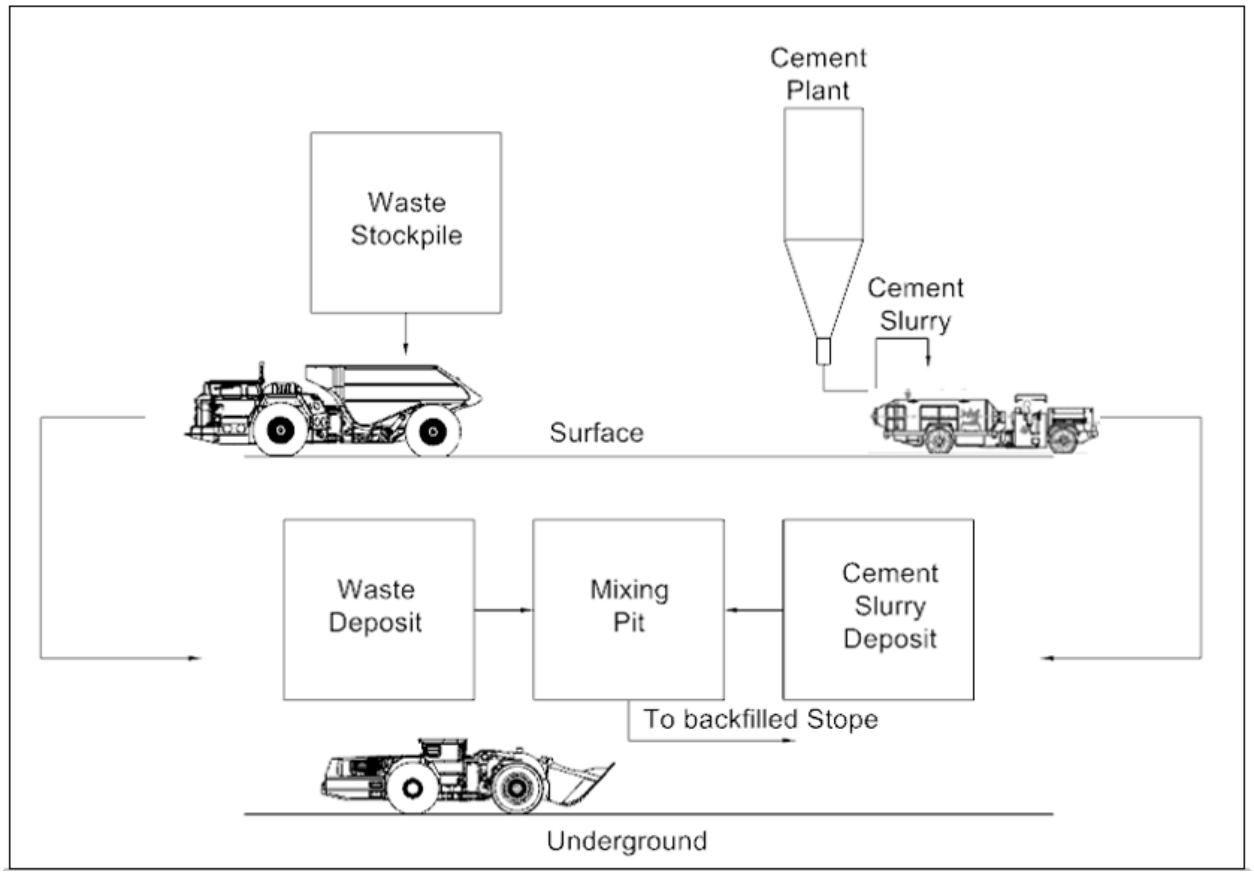
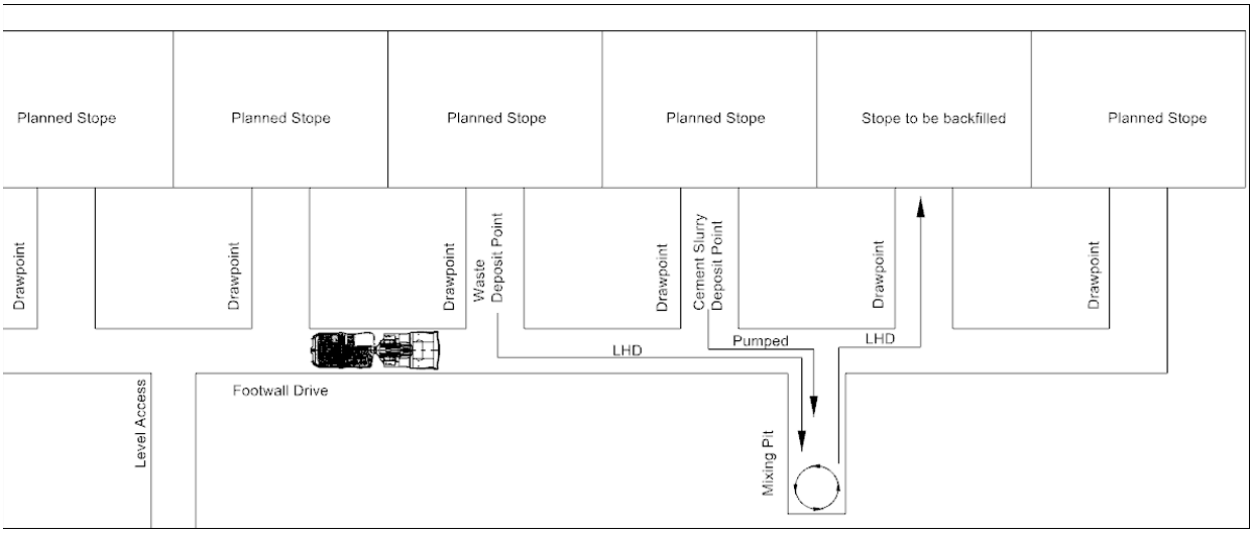


Figure 16.42: Production Level CRF Process Map



16.5.7.4 Compressed Air

The compressed air supply will be provided by a series of electrical compressors installed on surface near the main fans. The compressed air piping network will be installed via the Main Fresh Air Raise, then along the main ramp. Compressed air will provide power to the dewatering pumps of the development headings, handheld drills, equipment door, to air-powered actuators as well as any other air-powered equipment. The compressors will also provide an emergency air supply to the refuge stations.

During the development phase, prior to the installation of the permanent compressed air network, a temporary compressor will be utilized to supply compressed air for mining operations. This temporary unit will be positioned near the portal to ensure efficient air delivery to the work areas.

16.5.7.5 Communications

The underground communication network consists of an LTE system that will be installed on site and will be expanded over the LoM. Mobile equipment operators, light vehicles, and supervisors will be equipped with LTE phones to communicate with personnel on the surface. LTE will also be used to control the underground ventilation network and will allow the use of the automation features for some equipment.

16.5.7.6 Fuel Storage and Distribution

Fuel will mostly be stored on surface. However, there will be two (2) underground fuel and lube bay infrastructures to provide fuel and lube for equipment that rarely returns to the surface. A fuel and lube truck are also planned as part of the mobile equipment fleet to distribute the fuel and lube to underground equipment that cannot travel quickly to the fuel and lube station for refuelling. In addition to servicing mobile equipment, the fuel and lube truck will also be used to fill the fuel bays.

16.5.7.7 Explosives Storage and Handling

Two (2) underground explosive and detonator magazines will be installed in designated locations: one (1) near surface in the satellite zone and one (1) in the main zone. Explosives will be delivered to the portal by the selected explosive supplier, then will be transported to the underground magazines by flatbed service truck for later use.

16.5.7.8 Personnel and Underground Material Transportation

Supplies and personnel will access the underground mine via the main ramp. Several personnel carriers, such as land cruisers, will be used to transport workers from the surface to the underground mine. Supervisors and technical services will also use some light vehicles for transportation underground. The maintenance team, as well as mechanical and electrical personnel, will utilize maintenance tractors equipped with attachments such as forks, buckets, booms, or other tools to assist them in performing their duties. Two (2) flatbed trucks equipped with a service boom will be used to move supplies from the surface to the underground active headings, stopes and material storages. Also, two (2) cassette trucks with flat deck and crane cassette will be available to transport materials.

16.5.7.9 Equipment Maintenance

The maintenance for all equipment is done following the supplier's recommendations. To maximize equipment lifespan, rebuilds are planned following supplier's rebuild cycle recommendations. Most major mechanical maintenance activities will be carried out at the surface workshop. However, some major repairs, as well as all minor maintenance and small emergency interventions, will be performed underground in a dedicated garage excavated and equipped for this purpose.

The underground garage will include several key facilities to support operational and maintenance activities, such as a parking area, a washing bay, three maintenance bays, an office for supervisors, a tool crib, a tire storage bay, a welding bay, an electrical workshop area, and a sump for water management.

16.5.8 Underground Mine Safety Measures

16.5.8.1 Emergency Exits

The main decline will provide primary egress from the underground workings. In regard to the secondary exit, egress raises will be excavated between production levels. Also, some of the ventilation transfer drifts between the different zones will serve as secondary egress. The independent safety egress raises will be equipped with prefabricated modular Laddertube^{tm} systems.

16.5.8.2 Refuge Stations

Refuge stations will be positioned so that all employees can access a refuge in less than 10 minutes from the moment they leave their workplace or at a maximum of 1,000 metres from any workplace. Refuge stations used in the underground mine will consist of either portable or fixed installations. Portable, self-

contained refuge stations will be procured and deployed to follow the lateral development and prevent delays associated with constructing permanent refuge stations. A total of seven (7) fixed refuge stations are planned as part of the permanent infrastructure to support production activities.

Each refuge station will be equipped with the following:

- Telephone or radio for communication with surface, independent of mine power supply.
- Compressed air, water lines, and water supply.
- Emergency lighting.
- Hand tools and sealing material.
- Plan of the underground work showing all exits and the ventilation plans.
- All other necessary items according to the applicable regulations.
- Fire protection.

16.5.8.3 Mobile Equipment Safety

Underground mobile vehicles will be equipped with automatic fire suppression systems in accordance with best practice. Fire extinguishers will be provided and maintained in accordance with regulations and best practices at electrical installations, pump stations, fuel station, service garages and wherever a fire hazard exists. Every vehicle will carry at least one (1) fire extinguisher of adequate size and proper type.

16.5.8.4 Mine Rescue

Fully trained and equipped mine rescue teams will be established in accordance with applicable regulations. Mine rescue equipment, including a dedicated underground emergency vehicle and a foam generator, will be available on site.

Rescue teams will be trained for surface and underground emergencies. An emergency response plan will be developed and continuously updated as the mine and regulations evolve.

16.5.8.5 Emergency Stench System

A mine stench gas warning system will be installed in the main surface ventilation system, initially in the temporary system and later in the permanent system. This system is designed to inject a specific dosage of stench gas, calculated based on the airflow quantity. This gas with a particular smell would alert the

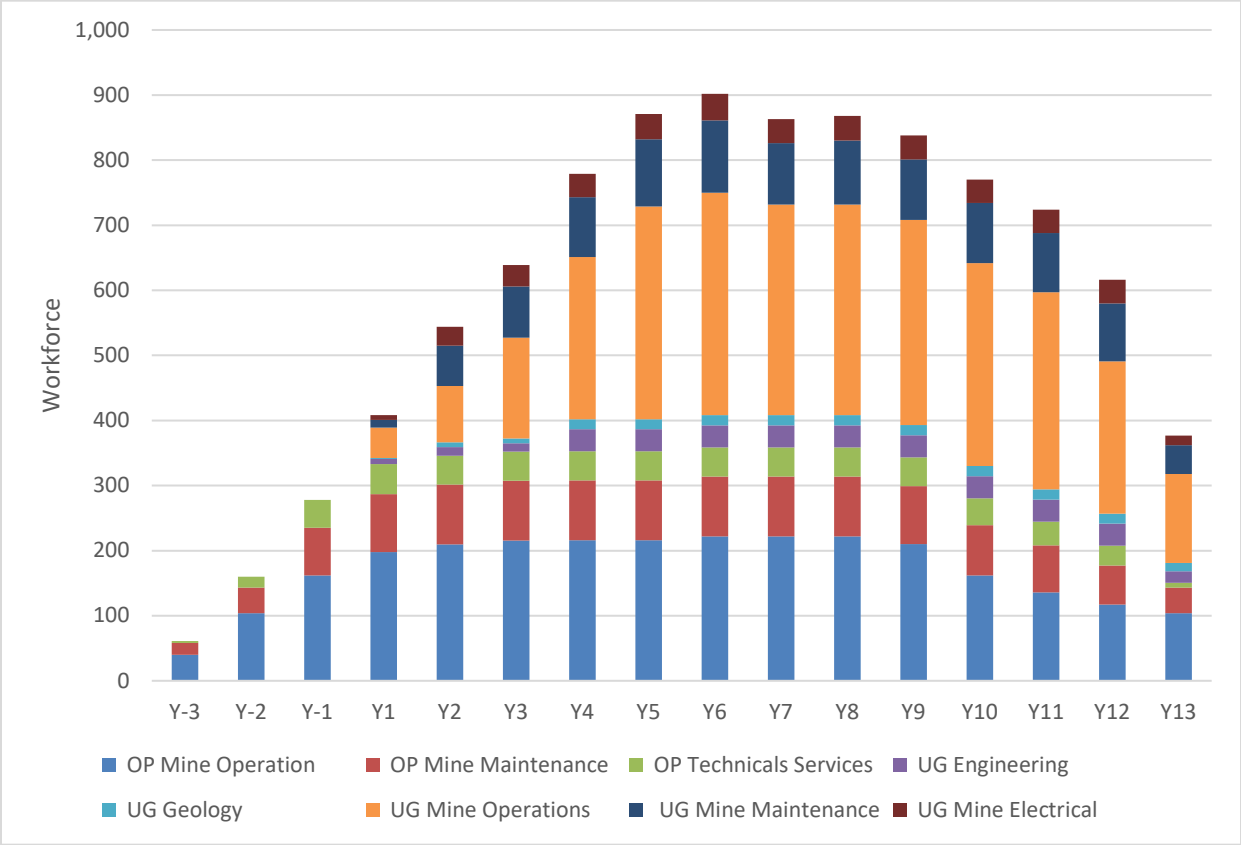
workers to an emergency as soon as they smell the gas. Another mine stench gas warning system will be installed at the mine's compressed air system as a second means to alert underground workers in the event of an emergency.

16.6 Mine Workforce

Mine personnel were divided into hourly and staff positions and divided between mine operations, mine maintenance, mine electrical, engineering and geology. Hourly positions were mostly associated with a shift roster of 14 days on and 7 days off, and as such, each unit of equipment requires three (3) operators hired in hourly positions.

Staff positions in management, supervision or technical services roles that require a continuous seven (7) days a week presence will also be on the roster schedule. Most of the staff positions are considered expats on a 23/19 schedule. The combined UG and OP mine workforce peaks at 879 individuals in Year 6. Figure 16.43 shows the workforce requirements over the LOM.

Figure 16.43: OP and UG Mine Workforce



16.6.1 Open Pit Mine Workforce Requirements

Table 16.32 to Table 16.35 shows the estimated OP workforce requirements over the LOM. The OP workforce peaks at 359 individuals in Year 6. Some positions will be shared with the UG mine workforce.

Table 16.32: Open Pit Mine Operations Workforce

Mine Operations	Type	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Mine Superintendent – Expat – CAN	Staff	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Mine Ops. General Foreman – Expat – CAN	Staff	0	0	3	3	3	3	3	3	3	3	3	3	3	3	3	2
Supervisor – Expat – CAN	Hourly	2	4	8	8	8	8	8	8	8	8	8	8	8	4	2	0
Mine D&B Supervisor – Expat – CAN	Hourly	0	0	2	2	4	4	4	4	4	4	4	4	4	2	2	2
Dispatcher – Local – GUY	Hourly	0	0	4	4	4	4	4	4	4	4	4	4	4	2	2	2
Training Supervisor – Expat – SUR	Hourly	0	4	4	4	2	2	2	2	2	2	2	2	2	2	2	0
Clerk – Local – GUY	Hourly	0	1	1	1	0.5	0.5	1	1	1	1	1	1	1	1	1	0
Driller – Local – GUY	Hourly	2	2	6	6	12	12	12	12	15	15	15	12	9	6	6	4
Auxiliary Drill Operator – Auxiliary Pre-split Drill (4.5-8") – Local – GUY	Hourly	0	3	0	3	3	3	3	3	3	3	3	3	3	3	3	3
Shovel / Excavator Operator LU 1 – Local – GUY	Hourly	4	6	9	12	12	12	12	12	12	12	12	9	9	6	3	3
Shovel / Excavator Operator LU 2 - Local – GUY	Hourly	2	4	4	3	3	3	3	3	3	3	3	3	3	3	3	4
Loader Operator LU 4 - Local – GUY	Hourly	0	0	3	3	3	3	3	3	3	3	3	3	3	3	3	4
Haul Truck Operator 1 – Local – GUY	Hourly	10	30	45	63	66	72	72	72	72	72	72	69	42	33	33	33
Haul Truck Operator 2 – Local – GUY	Hourly	12	18	18	18	18	18	18	18	18	18	18	18	9	9	9	9
Dewatering Labour – Local – GUY	Hourly	0	0	2	6	6	6	6	6	6	6	6	6	6	6	6	6
Dozer Operator – Track Dozer 1 – Local – GUY	Hourly	4	12	15	18	18	18	18	18	18	18	18	15	9	9	6	3
Grader Operator – Local – GUY	Hourly	0	3	6	9	12	12	12	12	12	12	12	12	12	9	6	6
Water Truck Operator – Water / Sand Truck – Local – GUY	Hourly	0	0	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Dozer Operator – Wheel Dozer – Local – GUY	Hourly	0	0	3	6	6	6	6	6	6	6	6	6	3	3	3	3
Shovel / Excavator Operator – Local – GUY	Hourly	2	4	4	4	4	4	4	4	4	4	4	4	4	4	0	0
Shovel / Excavator Operator – Local – GUY	Hourly	2	4	4	4	4	4	4	4	4	4	4	4	4	4	0	0

Mine Operations	Type	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Utility Equip. Operator – Wheel Loader – Local – GUY	Hourly	0	0	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Crane Operator – Local – GUY	Hourly	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Loader Operator LU 4 – Local – GUY	Hourly	0	3	3	3	3	3	3	3	6	6	6	6	6	6	6	3
Loader Operator LU 5 – Local – GUY	Hourly	0	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Ore Crusher Operator – Local – GUY	Hourly	0	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Sub. Total Mine Operations		**40**	**104**	**162**	**198**	**209.5**	**215.5**	**216**	**216**	**222**	**222**	**222**	**210**	**162**	**136**	**117**	**104**

Table 16.33: Open Pit Maintenance Workforce

Mine Maintenance Workforce	Type	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Superintendent - Expat - CAN	Staff	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
General Foreman - Expat - CAN	Staff	0	0	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Supervisor - Expat - SUR	Staff	0	2	8	8	8	8	8	8	8	8	8	8	8	8	8	4
Senior Planner - Expat - CAN	Staff	0	1	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Planner - Expat - SUR	Staff	0	1	2	2	2	2	2	2	2	2	2	2	2	0	0	0
Mechanical Engineer - Expat - SUR	Staff	0	0	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Trainer - Expat - SUR	Staff	0	2	2	2	2	2	2	2	2	2	2	2	2	2	2	0
Clerk - Local - GUY	Hourly	0	0	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mobile Mechanic - Local - GUY	Hourly	6	18	30	42	45	45	45	45	45	45	45	42	30	27	21	12
Electrician - Local - GUY	Hourly	3	3	6	6	6	6	6	6	6	6	6	6	6	6	3	3
Welder / Machinist - Local - GUY	Hourly	3	3	6	6	6	6	6	6	6	6	6	6	6	6	3	3
Fuel & Lube Technician - Local - GUY	Hourly	0	2	4	8	8	8	8	8	8	8	8	8	8	8	8	4
Tool Crib Attendant - Local - GUY	Hourly	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2

Mine Maintenance Workforce	Type	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Helper - Local - GUY	Hourly	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Sub. Total Mine Maintenance		**18**	**39**	**73**	**89**	**92**	**92**	**92**	**92**	**92**	**92**	**92**	**89**	**77**	**72**	**60**	**39**

Table 16.34: Open Pit Mine Engineering and Geology Workforce

Mine Geology and Engineering Workforce	Type	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Chief Geologist - Expat - CAN	Staff	0	1	1	1	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5
Senior Geologist - Expat - CAN	Staff	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Resource Geologist - Expat - CAN	Staff	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Production Geologist - Expat - SUR	Staff	0	1	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Geology Technician - Local - GUY	Hourly	0	1	2	2	2	2	2	2	2	2	2	2	2	2	0	0
Grade Control Technician - Local - GUY	Hourly	0	1	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Grade Control Labourers / Samplers – Local – GUY	Hourly	0	2	8	8	8	8	8	8	8	8	8	8	8	8	8	1
Chief Mine Engineer	Staff	0	1	1	1	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5
Assistant Chief Mining Engineer	Staff	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Long-Term Planning Engineer	Staff	0	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0
Short-Term Planning Engineer	Staff	1	2	3	3	3	3	3	3	3	3	3	3	3	2	1	0.5
Drill & Blast Engineer	Staff	0	0	2	2	2	2	2	2	2	2	2	2	2	2	2	0
Senior Geotechnical Engineer	Staff	0	1	1	2	2	2	2	2	2	2	2	2	1	1	1	0
Geotechnical Engineer	Staff	0	1	2	4	4	4	4	4	4	4	4	4	2	2	2	1
Junior Mine Engineer	Staff	0	0	2	2	2	2	2	2	2	2	2	2	2	2	0	0
Dispatch system coordinator	Staff	0	0	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Mining Technician	Hourly	0	0	4	4	4	4	4	4	4	4	4	4	4	2	2	0

Mine Geology and Engineering Workforce	Type	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Senior Surveyor	Hourly	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	0
Surveyor	Hourly	1	2	4	4	4	4	4	4	4	4	4	4	4	2	2	0
Clerk	Hourly	0	1	1	1	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5
Sub Total		**3**	**17**	**43**	**46**	**44.5**	**44.5**	**44.5**	**44.5**	**44.5**	**44.5**	**44.5**	**44.5**	**41.5**	**36.5**	**30.5**	**7.5**

Table 16.35: Open Pit Mine Total Workforce

Total Workforce Open Pit Mine	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Mine Operations	40	104	162	198	209.5	215.5	216	216	222	222	222	210	162	136	117	104
Mine Maintenance	18	39	73	89	92	92	92	92	92	92	92	89	77	72	60	39
Mine Engineering and Geology	3	17	43	46	44.5	44.5	44.5	44.5	44.5	44.5	44.5	44.5	41.5	36.5	30.5	7.5
Total Workforce	**61**	**160**	**278**	**333**	**346**	**352**	**352.5**	**352.5**	**358.5**	**358.5**	**358.5**	**343.5**	**280.5**	**244.5**	**207.5**	**150.5**

16.6.2 Underground Mine Workforce Requirements

All site-based positions will be on a rotational schedule. A workforce of 543 is expected to be employed for the Oko Gold underground mine. Again, some positions are shared with the open pit mine workforce and are thus not accounted for in this section. The underground mine labour is described in Table 16.36 through Table 16.41. Finally, Table 16.42 provides a summary of both the OP and UG mines workforce.

Table 16.36: UG Engineering Workforce

UG Engineering	Type	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Chief Mine Engineer	Staff	-	-	-	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5
Assistant Chief Mining Engineer	Staff	-	-	-	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0
Long-Term Planning Engineer	Staff	-	-	-	1.0	1.0	1.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.0
Short-Term Planning Engineer	Staff	-	-	-	1.0	1.0	1.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.0
Production Engineer	Staff	-	-	-	-	-	-	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.0
Drill & Blast Engineer	Staff	-	-	-	-	-	-	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.0
Junior Mine Engineer	Staff	-	-	-	-	-	-	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	-
Ventilation Engineer	Staff	-	-	-	-	1.0	1.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.0
Senior Geotechnical Engineer	Staff	-	-	-	-	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Geotechnical Engineer	Staff	-	-	-	-	-	-	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0
Construction Engineer	Staff	-	-	-	-	1.0	1.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	-
Drill & Blast Technician	Hourly	-	-	-	-	-	-	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.0
Geotech. Technician	Hourly	-	-	-	-	1.0	1.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.0
Ventilation Technician	Hourly	-	-	-	-	-	-	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.0
Senior Surveyor	Hourly	-	-	-	1.0	1.0	1.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.0
Surveyor	Hourly	-	-	-	2.0	2.0	2.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	2.0
Clerk	Hourly	-	-	-	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5
Draftsman	Hourly	-	-	-	-	1.0	1.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.0

Table 16.37: UG Geology Workforce

UG Geology	Type	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Chief Geologist	Staff	-	-	-	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5
Senior Geologist	Staff	-	-	-	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Resource Geologist	Staff	-	-	-	-	-	-	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	-
Exploration Geologist	Staff	-	-	-	-	-	-	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	-
Production Geologist	Staff	-	-	-	-	1.0	1.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0
Junior Geologist	Staff	-	-	-	-	1.0	1.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.0
Grade Control Technician	Hourly	-	-	-	-	2.0	2.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0
Grade Control Labourers	Hourly	-	-	-	-	2.0	2.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0

Table 16.38: UG Mine Operations Workforce

UG Mine Operations	Type	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Mine Superintendent	Staff	-	-	-	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Mine Ops. General Foreman	Staff	-	-	-	1.0	1.0	1.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.0
Clerk	Hourly	-	-	-	0.5	0.5	0.5	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Supervisor	Staff	-	-	-	3.0	6.0	9.0	15.0	21.0	21.0	21.0	21.0	21.0	21.0	18.0	15.0	9.0
D&B Supervisor	Staff	-	-	-	-	-	-	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0
Trainer	Staff	-	-	-	8.0	8.0	8.0	12.0	12.0	12.0	12.0	12.0	12.0	12.0	12.0	6.0	-
Dispatcher	Staff	-	-	-	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0
Miner 1	Hourly	-	-	-	0.0	0.0	5.0	15.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	15.0	15.0
Miner 2	Hourly	-	-	-	0.0	0.0	5.0	15.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	15.0	15.0
Miner 3	Hourly	-	-	-	0.0	0.0	5.0	15.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	15.0	15.0

UG Mine Operations	Type	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Long-Hole Driller	Hourly	-	-	-	-	6.0	6.0	6.0	9.0	12.0	12.0	12.0	9.0	9.0	9.0	9.0	3.0
Blasters	Hourly	-	-	-	6.0	6.0	12.0	18.0	18.0	18.0	18.0	18.0	18.0	18.0	18.0	18.0	12.0
Scoop Operators	Hourly	-	-	-	6.0	6.0	9.0	18.0	27.0	30.0	30.0	30.0	30.0	27.0	27.0	27.0	9.0
Truck Operators	Hourly	-	-	-	3.0	6.0	12.0	18.0	33.0	42.0	42.0	42.0	42.0	42.0	39.0	39.0	12.0
Jumbo Operator	Hourly	-	-	-	3.0	3.0	9.0	9.0	9.0	9.0	3.0	3.0	3.0	3.0	3.0	3.0	-
Rockbolter	Hourly	-	-	-	3.0	9.0	18.0	24.0	24.0	24.0	12.0	12.0	6.0	6.0	6.0	6.0	-
Cable Bolter	Hourly	-	-	-	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	-
Scissor Lift Operator	Hourly	-	-	-	3.0	9.0	12.0	12.0	12.0	12.0	12.0	12.0	12.0	12.0	9.0	6.0	3.0
Level services	Hourly	-	-	-	-	-	3.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0
Grader Operator	Hourly	-	-	-	-	-	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0
U/G Constructions	Hourly	-	-	-	-	6.0	8.0	12.0	12.0	12.0	12.0	12.0	12.0	12.0	12.0	6.0	6.0
Sumps and Services Labour	Hourly	-	-	-	-	-	-	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	3.0
Boom Truck Operator	Hourly	-	-	-	-	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	3.0
Fuel Truck Operator	Hourly	-	-	-	-	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0
Lamps-Dry	Hourly	-	-	-	-	1.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0
Labour	Hourly	-	-	-	-	-	3.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	3.0
Labour spare	Hourly	-	-	-	-	-	4.0	12.0	12.0	12.0	12.0	12.0	12.0	12.0	12.0	6.0	3.0

Table 16.39: UG Mine Maintenance Workforce

UG Mine Maintenance	Type	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Superintendent	Staff	-	-	-	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Superintendent Assistant	Staff	-	-	-	-	-	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0

UG Mine Maintenance	Type	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
General Foreman	Staff	-	-	-	-	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0
Reliability Engineer	Staff	-	-	-	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.0	-
Mechanical Engineer	Staff	-	-	-	-	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.0
Trainer	Staff	-	-	-	-	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.0
Senior Planner	Staff	-	-	-	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Maintenance Planner	Staff	-	-	-	-	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	1.0
Administrative Assistant	Staff	-	-	-	-	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	-
Leader Mech.- Mobile Equip.	Hourly	-	-	-	-	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0
Mechanics - Mobile Equipment	Hourly	-	-	-	7.0	19.0	35.0	48.0	59.0	67.0	50.0	54.0	49.0	48.0	47.0	45.0	20.0
Welders - Mobile Equipment	Hourly	-	-	-	-	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0
Mechanics - Fixed Equipment	Hourly	-	-	-	-	9.0	9.0	9.0	9.0	9.0	9.0	9.0	9.0	9.0	9.0	9.0	3.0
Welder - Fixed Equipment	Hourly	-	-	-	-	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0
Labour - Fixed Equipment	Hourly	-	-	-	-	9.0	9.0	9.0	9.0	9.0	9.0	9.0	9.0	9.0	9.0	9.0	3.0

Table 16.40: UG Mine Electrical Workforce

UG Mine Electrical	Type	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Superintendent	Staff	-	-	-	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
General Foreman	Staff	-	-	-	-	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	-
Supervisor	Staff	-	-	-	-	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0
Senior Planner	Staff	-	-	-	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	-
Planner	Staff	-	-	-	-	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	-
Electrical Engineer	Staff	-	-	-	-	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	-

UG Mine Electrical	Type	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Trainer	Staff	-	-	-	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	-
Electricians - Mobile Equipment	Hourly	-	-	-	2.0	5.0	9.0	12.0	15.0	17.0	13.0	14.0	13.0	12.0	12.0	12.0	5.0
Electricians - Instr. Technicians	Hourly	-	-	-	-	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0
Electricians - Fixed Equipment	Hourly	-	-	-	-	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	3.0

Table 16.41: UG Summary Workforce

Total Workforce UG Mine	Y-3	Y -2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
UG Engineering	-	-	-	8.0	13.0	13.0	34.0	34.0	34.0	34.0	34.0	34.0	34.0	34.0	34.0	18.0
UG Geology	-	-	-	1.5	7.5	7.5	15.5	15.5	15.5	15.5	15.5	15.5	15.5	15.5	15.5	12.5
UG Mine Operations	-	-	-	46.5	86.5	154.5	249.0	327.0	342.0	324.0	324.0	315.0	312.0	303.0	234.0	137.0
UG Mine Maintenance	-	-	-	12.0	62.0	79.0	92.0	103.0	111.0	94.0	98.0	93.0	92.0	91.0	89.0	44.0
UG Mine Electrical	-	-	-	7.0	29.0	33.0	36.0	39.0	41.0	37.0	38.0	37.0	36.0	36.0	36.0	15.0
Total UG Workforce	-	-	-	75.0	198.0	287.0	426.5	518.5	543.5	504.5	509.5	494.5	489.5	479.5	408.5	226.5

Table 16.42: Total Mining Workforce

Total Workforce Mines	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Total Underground	-	-	-	75.0	198.0	287.0	426.5	518.5	543.5	504.5	509.5	494.5	489.5	479.5	408.5	226.5
Total Open Pit	61	160	278	333	346	352	352.5	352.5	358.5	358.5	358.5	343.5	280.5	244.5	207.5	150.5
Total Workforce	**61**	**160**	**278**	**408**	**544**	**639**	**779**	**871**	**902**	**863**	**868**	**838**	**770**	**724**	**616**	**377**

16.7 Combined Production

The open pit mine is scheduled for a pre-production period lasting 25 months. During this time, the mine will produce a total of 50 Mt of waste and overburden, along with 2.6 Mt of ore material. Additionally, 1 Mt of material will be produced during the pre-production phase, which will be used for construction and to prepare the portal for the underground mine. The focus during this phase will be on efficiently removing non-valuable material to access the mineralized zones and preparing the site for full-scale operations. In the last three (3) months of the pre-production period, the processing plant will initiate hot commissioning and gradually ramp up its production capacity, aiming to achieve a full output of 7 million tonnes per annum (Mtpa). This capacity is made possible by the significant amount of saprolite being sent to the processing plant. Saprolite is much easier to process than other materials due to its fine nature, allowing for quicker and more efficient extraction and processing. This gradual increase will ensure a smooth transition to full production, allowing for the testing and optimization of processes to maintain high efficiency and reliability as the mine moves into full-scale operations. This phased approach is critical to ensure the seamless integration of all operational components and to maximize the economic potential of the mineral resource.

As the processing plant begins production, the underground mine will advance the decline development from the portal excavated by the open pit mine. Following a three-year development period, the underground mine is expected to ramp up production for the next two years before reaching its nominal throughput.

Table 16.43 shows annual production data for the Life-of-Mine (LOM). Additionally, Figure 16.44 shows mine production by type of material, Figure 16.45 shows total gold production, and Figure 16.46 shows gold production by mine.

Table 16.43: Yearly LOM Production Details

Description	Unit	Total	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
MILL FEED																		
Saprolite Ore Tonnage	kt	7,657	-	-	447	2,895	2,382	1,431	121	33	78	1	39	229	-	-	-	-
Saprolite Ore Grade	g/t Au	1.28	-	-	1.47	1.47	1.29	0.99	0.72	0.69	0.69	0.69	0.60	0.73	-	-	-	-
Saprolite Ore Ounces	koz	314	-	-	21	137	99	46	3	1	2	0	1	5	-	-	-	-
Trans. Ore Tonnage	kt	3,541	-	-	181	1,898	683	77	114	104	1	-	5	449	-	-	29	-
Trans. Ore Grade	g/t Au	1.43	-	-	2.25	1.70	1.16	0.91	1.07	0.83	0.39	-	0.49	0.75	-	-	0.65	-
Trans. Ore Ounces	koz	163	-	-	13	104	26	2	4	3	0	-	0	11	-	-	1	-
OP Fresh Rock Ore Tonnage	kt	51,176	-	-	17	1,948	3,934	5,107	5,092	4,611	4,134	4,288	4,271	3,840	4,309	4,405	3,858	1,360
OP Fresh Rock Ore Grade	g/t Au	1.63	-	-	1.17	1.50	1.67	1.63	1.71	1.64	1.41	1.47	1.57	1.73	2.06	1.75	1.41	1.22
OP Fresh Rock Ore Ounces	koz	2,679	-	-	1	94	212	268	280	244	187	202	216	213	286	248	175	53
Total OP Processed Ore	kt	62,374	-	-	644	6,741	7,000	6,616	5,327	4,748	4,213	4,289	4,316	4,519	4,309	4,405	3,887	1,360
Total OP Processed Grade	g/t Au	1.57	-	-	1.68	1.54	1.49	1.48	1.67	1.62	1.40	1.47	1.56	1.58	2.06	1.75	1.41	1.22
Total OP Processed Ounces	koz	3,156	-	-	35	335	336	316	286	247	189	202	217	230	286	248	176	53
UG Ore Tonnage	kt	14,181	-	-	-	-	-	71	673	1,269	1,787	1,711	1,723	1,711	1,691	1,595	1,570	380
UG Ore Grade	g/t Au	3.26	-	-	-	-	-	2.85	2.79	3.23	3.29	3.16	3.20	3.22	3.26	3.21	3.62	3.77
UG Ore Ounces	koz	1,486	-	-	-	-	-	7	60	132	189	174	177	177	177	165	183	46
Total UG & OP Processed Ore	kt	**76,555**	-	-	644	6,741	6,000	6,687	6,000	6,016	6,000	6,000	6,039	6,229	6,000	6,000	5,458	1,740
Total UG & OP Processed Grade	g/t Au	**1.89**	-	-	1.68	1.54	1.49	1.50	1.80	1.96	1.96	1.95	2.03	2.03	2.40	2.14	2.04	1.78
Total UG & OP Processed Ounces	koz	**4,642**	-	-	35	335	336	322	347	379	378	376	394	407	463	413	358	100

Figure 16.44: Processing by Rock Type

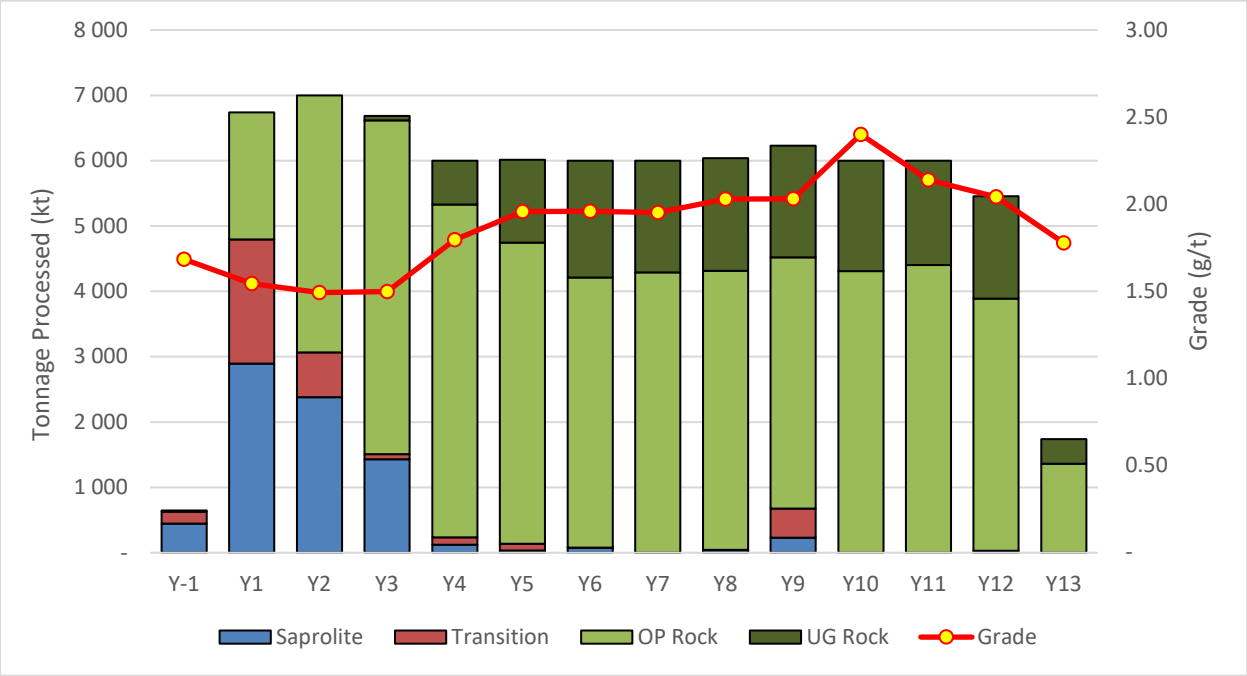


Figure 16.45: Gold Processed

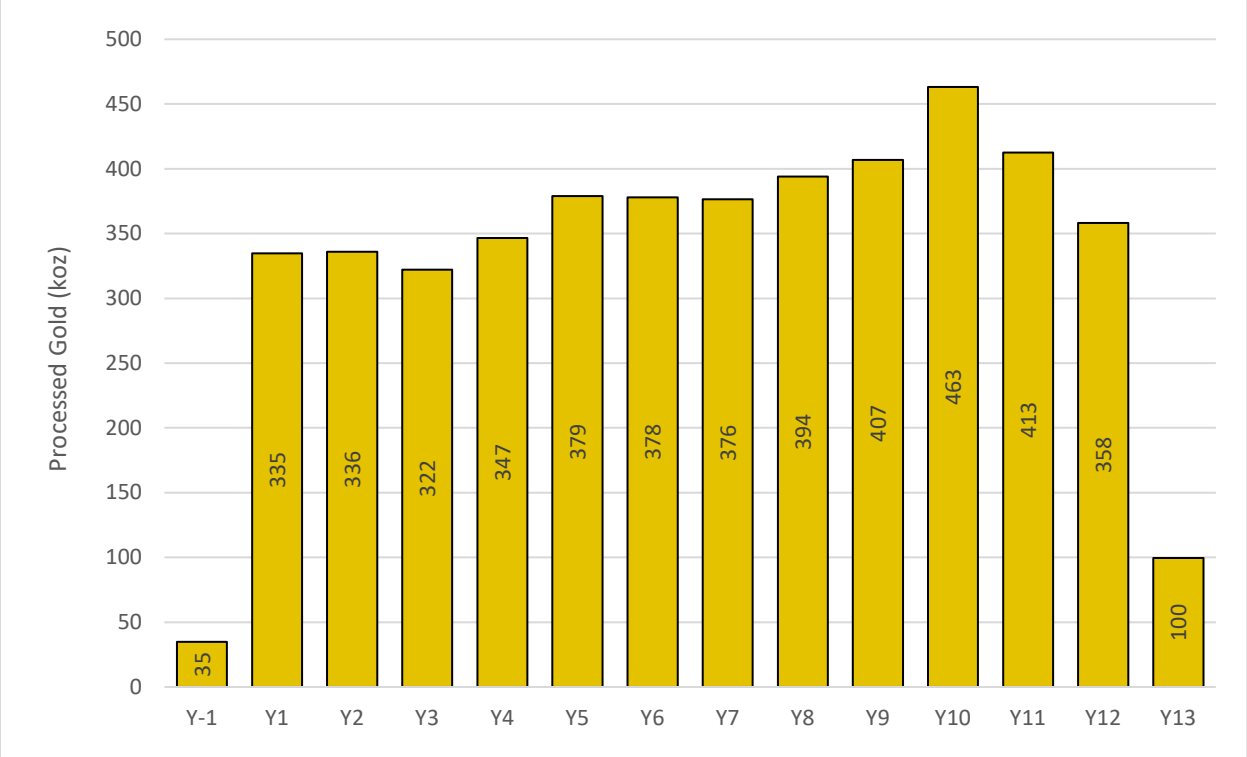
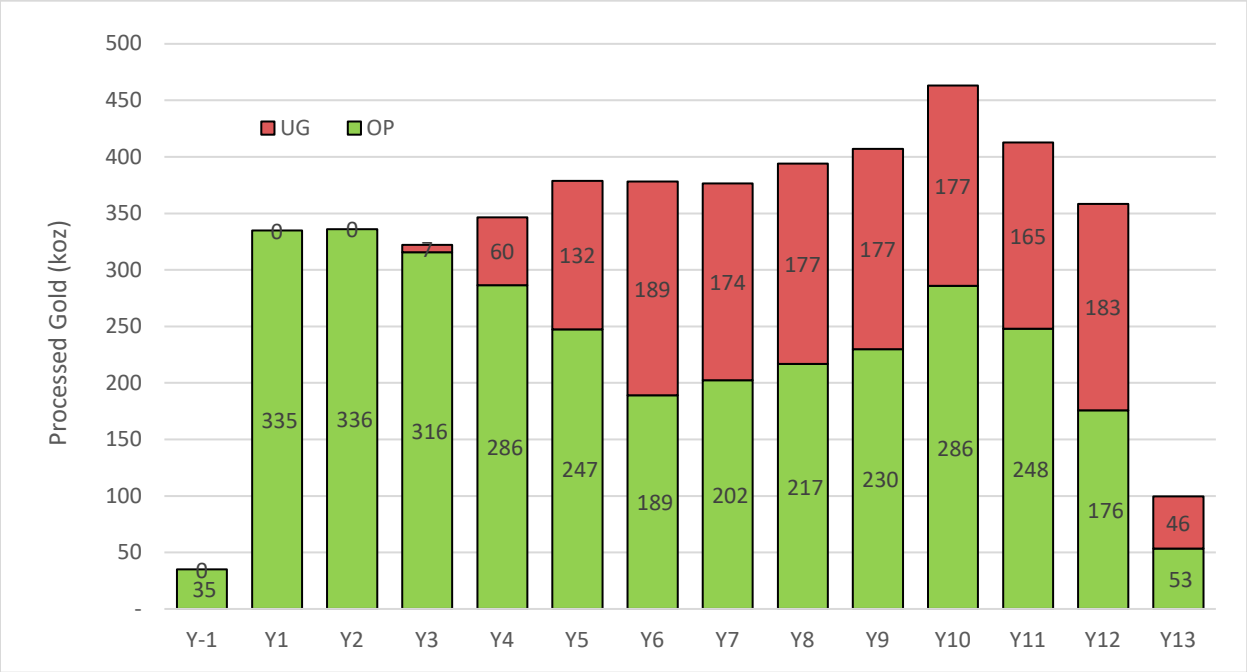


Figure 16.46: Gold Production per Mine



17. RECOVERY METHODS

17.1 Introduction

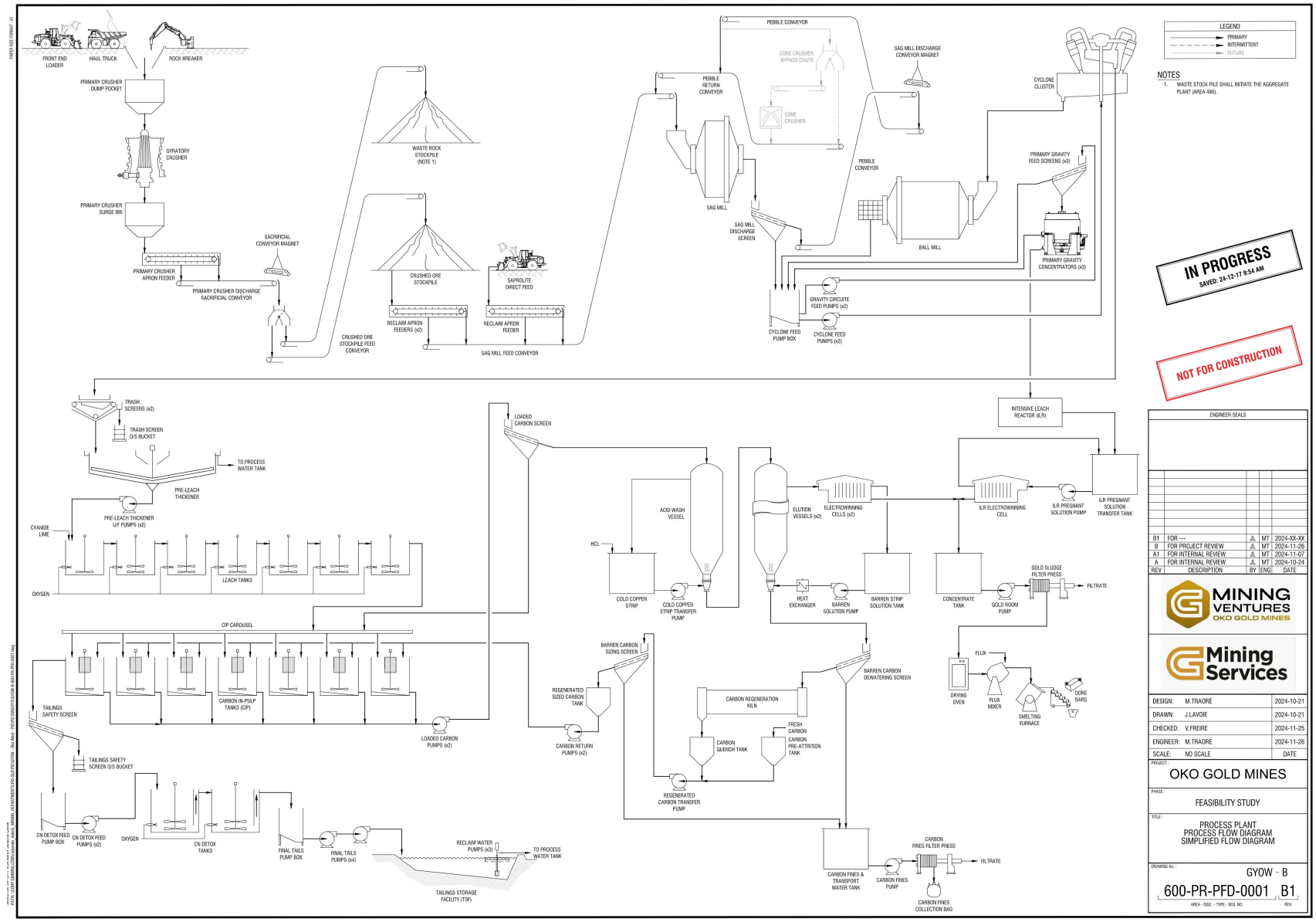
The proposed process plant design for the Oko West Project is based on a standard metallurgical flowsheet to treat gold bearing material to produce doré. The flowsheet is based on metallurgical test work described in Section 13, industry standards and conventional unit operations.

The process plant is designed to nominally treat 6 Mtpy of fresh rock and will consist of comminution, gravity concentration, cyanide leach and adsorption via leach / carbon-in-pulp (CIP), carbon elution and gold recovery circuits. CIP tailings will be treated in a cyanide destruction circuit and pumped to a tailings' storage facility. Figure 17.1 presents the overall flowsheet for the Oko West Project.

The key project design criteria for the process plant are listed below:

- Nominal throughput of 7 Mtpy when processing a blend including soft ores (i.e. saprolite and transition) with a throughput of 6 Mtpy when processing 100% fresh rock.
- Crushing plant availability of 70%.
- Grinding, gravity, CIP, gold recovery and tailings handling circuit availability of 92% through the use of standby equipment in critical areas, inline crushed material stockpile and reliable power supply.
- Comminution circuit to produce a primary grind size of (P80) 80% passing 75 µm.
- Leach residence time of 36 hours to achieve optimal gold extraction.
- Cyanide destruction circuit designed to produce weak acid dissociable (WAD) cyanide levels of less than 5 ppm.
- Sufficient process plant control to minimize the need for continuous operator interface and to allow for manual override and control if and when required.
- Equipment selection based on suitability for the required duty, reliability, and ease of maintenance.
- Plant layout that provides ease of access to all equipment for operating and maintainability, while facilitating concurrent construction activities in multiple areas of the plant.

Figure 17.1: Overall Flowsheet



17.2 Process Design Criteria

The proposed process plant will consist of the following unit operations:

- Primary crushing of material.
- Coarse material stockpile and reclaim.
- Saprolite feeder to reclaim conveyor.
- Grinding consisting of semi-autogenous (SAG) mill and ball mill with hydrocyclones producing a final product P80 of 75 µm.
- Gravity concentration to produce a gold-rich concentrate for intensive leaching and subsequent gold recovery via electrowinning.
- Pre-leach thickening.
- Cyanide leaching and carbon adsorption via a leach-CIP circuit respectively.
- Carbon elution via Split Pressure Zadra circuit.
- Carbon handling and regeneration.
- Electrowinning and smelting to produce doré.
- Cyanide destruction of CIP tailings using SO_2 / O_2 process.
- Tailings pumping to a tailings' storage facility.
- Air and oxygen circuits.
- Water systems (potable water, treated water, gland seal water and process water).

Key process design criteria are summarized in Table 17.1.

Table 17.1: Key Process Design Criteria

Area	Criteria	Unit	Nominal Value
General	Nominal Annual Throughput (blend with soft ores)	t/y	7,000,000
	Nominal Daily Throughput (blend with soft ores)	t/d	19,200
	Nominal Annual Throughput (fresh rock)	t/y	6,000,000
	Nominal Daily Throughput (fresh rock)	t/d	16,500
	Crusher Plant Availability	%	70
	Process Plant Availability	%	92
	Design Gold Head Grade	g/t	1.58
	Average Fresh Rock Gold Recovery – Open Pit	%	94.0%
	Average Fresh Rock Gold Recovery - Underground	%	92.7%
	Average Transition Material Gold Recovery	%	93.3%
	Average Saprolite Gold Recovery	%	95.0%
Crushing & Storage	Crusher Work Index	kWh/t	15.0-22.2
	Run of Mine (ROM), Maximum Size	mm	910
	Crusher Circuit Product Size (P80)	mm	115
	Fresh Rock Stockpile Capacity (live)	h	12
Grinding	SMC A x b (15^{th} percentile) – Fresh Rock	-	29.4
	Bond Ball Mill Work Index (85^{th} percentile) – Fresh Rock	kWh/t	16.4
	Bond Rod Mill Work Index (85^{th} percentile) – Fresh Rock	kWh/t	17.9
	Grinding Circuit Product Size (P80)	µm	75
Gravity Concentration	Type	-	3 x KC-QS48 or equivalent
	Intensive Leach Reactor	-	CS6000 or equivalent
Pre-Leach Thickening	Thickener Underflow Density	%w/w	42 (high sap) 45 (fresh rock)
	Solids Loading	t/m^2h	0.6
Leach-CIP	Leach Residence Time	h	36
	Leach Tanks	-	7
	CIP Carousel Tanks	-	7
DR	Elution Batch Size (Carbon)	t	10

Area	Criteria	Unit	Nominal Value
Cyanide Destruction	Cyanide Destruction Technology	-	SO_2 / O_2
	Number of Tanks	-	2
	Total Retention Time	h	2

17.3 Process Plant Description

17.3.1 Primary Crushing

Material from the open pit will be transported to the plant by rear dump trucks. The trucks will tip directly to the primary crusher dump pocket. However, if the trucks are not permitted to directly tip into the dump pocket, then the truck load will be dumped onto the ROM pad. The ROM pad will be primarily utilized for short term or emergency storage and material blending as required by the mine plan. ROM material will be reclaimed to the dump pocket by a front-end loader.

Material from the primary crusher dump pocket will feed a gyratory crusher. A rock breaker will be installed to assist in breaking down oversize material retained above the gyratory crusher. Crushed material from the gyratory crusher will discharge to the primary crusher surge bin. An apron feeder and sacrificial conveyor will withdraw the crushed material. A belt magnet at the sacrificial conveyor discharge will recover any trash metal. The sacrificial conveyor will convey crushed material to the stockpile feed conveyor, which will convey material to the crushed material stockpile. The stockpile feed conveyor will be fitted with a weightometer to monitor the primary crusher throughput and to control the apron feeder variable speed drive (VSD).

The crushing circuit will be serviced by a single dust collection system consisting of multiple extraction hoods, ducting, and a baghouse. Dust collected from this system will be discharged onto the stockpile feed conveyor.

17.3.2 Material Stockpiles

The stockpile area has provision to store crushed fresh rock from the gyratory crusher and waste material for construction purposes. The sacrificial conveyor diverter chute will send crushed rock to either the crushed ore stockpile feed conveyor or the waste rock stockpile radial feed. The fresh rock stockpile will have a live capacity of approximately 9 kt (equivalent to 12 hours of mill feed). Two (2) reclaim apron feeders located underneath the stockpile will be installed with variable speed drives (VSDs) to control the reclaim rate feeding the grinding circuit. One (1) additional apron feeder will be installed for direct feed to

the SAG mill feed conveyor During the initial production years, saprolite material will be directly fed to the mill via this apron feeder.

The apron feeders installed within the surface tunnel are equipped with a water spray system at the chute and an integrated dust collection system designed to mitigate airborne particulate matter generated during the material handling. The surface tunnel is designed with an engineering ventilation system to maintain adequate airflow, ensuring occupational health and safety compliance while minimizing dust accumulation. This ventilation system is designed to control air quality, humidity, and temperature, which are critical factors for the safe and efficient operation of mechanical equipment within the surface tunnel.

17.3.3 Grinding

Reclaimed material from the stockpile will feed a 10.4 m diameter by 5.4 m effective grinding length (EGL) SAG mill via the SAG mill feed conveyor. The SAG mill will be installed with 12,000-kW dual pinion synchronous motors and a VSD to control the speed of the SAG mill. A belt-scale on the SAG feed conveyor will monitor the feed rate. Process water will be added to the SAG mill to maintain a 75% slurry discharge density. SAG mill discharge will pass through a screen to remove grinding media scats and a small number of pebbles. The SAG screen undersize will report to the cyclone feed pump box, combining with ball mill discharge and gravity screen oversize (O/S). SAG screen oversize will be conveyed back to the SAG mill feed conveyor.

Slurry from the cyclone feed pump box will be pumped to a cyclone cluster of 18 (11-14 operating / 4-7 standby) 508 mm hydrocyclones for size classification. The cyclone overflow, at a final target product P80 of 75 µm, will flow via gravity to the trash screens prior to the leach circuit. The hydrocyclones have been designed for a 350% circulating load.

Cyclone underflow will feed a 7.3 m diameter by 11.5 m EGL ball mill with 12,000 kW dual pinion fixed speed motors. Slurry will overflow from the ball mill to a trommel screen, attached to the ball mill discharge end. Trommel undersize will discharge into the cyclone feed pump box.

A portion of the cyclone feed will feed a gravity separation circuit for coarse gold recovery.

17.3.4 Gravity Gold Recovery

The gravity recovery will consist of three (3) centrifugal gravity concentrator units equipped with a feed screen and an intensive cyanidation unit. The gravity gold recovery unit will be located in a secured area within the grinding area structure.

Gravity feed slurry will be screened using a vibrating screen to remove +2 mm material. The oversized material will overflow directly to the cyclone feed pump-box. The undersize product will feed a KC-QS48 or equivalent gravity concentrator. Gravity concentrator tailings will discharge into the cyclone feed pump box.

Periodically, the centrifugal concentrator will be bypassed and switched to flushing mode using fresh water to recover the collected concentrate. The collected concentrate will be pumped to the intensive leach reactor unit (ILR).

The gravity concentrate will be batch processed in the intensive cyanidation unit in 24-hour intervals. The gravity concentrate will be leached to dissolve gold in a leach solution that includes sodium cyanide, caustic solution, and a leach accelerant. After the leach cycle is complete, the pregnant solution will be pumped to the electrowinning circuit while the intensive cyanidation unit residue will be pumped to the cyclones feed pump box.

17.3.5 Pre-Leach Thickening

Cyclone overflow will flow to two (2) trash screens in parallel and then gravity fed to a 46 m diameter pre-leach feed thickener to increase slurry density for the downstream cyanidation process. Flocculant will be added to the thickener feed to promote the settling of solids. The thickener overflow will report to a pre-leach thickener overflow tank which is then pumped to the process water tank.

17.3.6 Leaching

Pre-leach thickener underflow at 42% w/w solids (for high saprolite feed) or 45% w/w solids (for fresh rock feed) will be pumped to the leach circuit consisting of 7 x 6,048 m^3 tanks (live volume). The leach tanks will provide a total retention time of 36 hours, and the tanks will be sparged with oxygen. Slurry from the leach circuit will then flow by gravity to the CIP circuit for carbon adsorption.

Sodium cyanide will be added to the leach circuit to dissolve the gold and lime slurry will be stage-added to maintain circuit alkalinity at a pH of 10.5-11, preventing the creation of hydrogen cyanide gas.

17.3.7 Carbon-in-Pulp

Leached slurry will flow to a CIP pump cell plant consisting of seven 7.5 m diameter x 12.2 m high CIP tanks operating in a carousel arrangement. Each tank will be installed with a pumping inter-stage screen mechanism for retaining activated carbon. In the CIP carousel circuit, the feed and discharge points of each tank rotate to simulate the counter-current movement of carbon against the slurry. Carbon management is

simplified, as each cell contains a discrete batch of carbon. As the slurry flows through the CIP circuit, gold and silver cyanide complexes will be adsorbed onto the activated carbon until precious metal values in the solution progressively decrease. The average carbon concentration in the CIP circuit is expected to be approximately 50 g/L to maximize adsorption.

Once per day, loaded carbon from a CIP tank will be pumped to the loaded carbon screen where the slurry will be separated and the loaded carbon transferred to the acid wash circuit. The slurry will then be transferred back to the CIP circuit. Fresh activated carbon from the regeneration circuit will be pumped into the CIP circuit.

Tailings from the CIP circuit will be pumped to a carbon safety screen to capture any residual fine carbon particles. Captured carbon will be collected in bags and processed to recover any residual gold and silver. The carbon safety screen undersize will flow via gravity to the cyanide destruction circuit.

17.3.8 Cyanide Detoxification

The cyanide destruction circuit will consist of two (2) 3,000 m^3 mechanically agitated tanks, providing a total retention time of two (2) hours. The conventional SO_2 / O_2 process will be used for cyanide destruction. Treated slurry will flow by gravity to the cyanide destruction tailings pump box for pumping to the tailings' storage facility.

The cyanide destruction circuit will treat CIP tailings, process spills from various contained areas and process bleed streams: cold cyanide barren solution effluent, acid wash effluent and area sump pump discharge.

Oxygen will be sparged into the cyanide destruction tanks. Lime slurry will be added to maintain the pH of 8.5 and copper sulphate will be added as a catalyst. Sodium metabisulphite (SMBS) will be dosed into the system as a source of SO_2. The process will reduce WAD cyanide in solution to 5 mg/L. The total cyanide and WAD cyanide level in solutions will eventually drop to lower levels via natural degradation and dilution from rainwater in the tailings' storage facility.

17.3.9 Acid Wash and Elution

Loaded carbon from the CIP circuit will be pumped and screened to the acid wash column where it will be treated with hydrochloric acid to remove inorganic foulants such as calcium, magnesium, sodium salts, and silica. The carbon will first be rinsed with fresh water. Acid will then be pumped from the acid wash circulation tank to the acid wash column and then pumped upward through the acid wash vessel and

overflow back to the acid wash circulation tank. The carbon will then be rinsed with fresh water to remove the acid and any mineral impurities. Fresh acid will be pumped from drums into the acid wash tank when required.

A recessed impeller pump will transfer acid washed carbon from the acid wash vessel into one of the elution vessels using recycled carbon transfer water. Carbon slurry will discharge directly into the top of one of the elution vessels.

The carbon stripping (elution) cycle will utilize barren solution to strip gold rich carbon to create a pregnant solution. The strip circuit will be equipped with two (2) strip columns that can hold 10 t of carbon each to allow more than one (1) strip per day, depending on the feed to the plant. During the strip cycle, solution containing approximately 2.0% hydroxide and 0.2% sodium cyanide, at a temperature of 150°C and 500 kPa will be circulated through the strip vessel. Solution exiting the top of the elution vessel will be cooled below its boiling point by the heat recovery heat exchanger. Heat from the outgoing solution will be transferred to the incoming cold solution. The heated barren solution will then be heated again through the primary heat exchanger using heated water to bring the solution to its final temperature.

The hot barren solution will then be pumped into the elution column through the carbon bed and recirculated multiple times creating a pregnant solution. A barren solution tank will store barren solution, and a pregnant solution tank will store pregnant solution. The elution column can also be used as a cold strip circuit to remove copper from carbon if copper levels are too high.

17.3.10 Carbon Regeneration

Once stripped of gold, transport water will transfer the carbon from the elution vessel to the carbon dewatering screen. The screen acts as both a dewatering screen and a carbon sizing screen, where fine carbon particles will be removed. Oversize carbon from the screen will discharge by gravity to the carbon regeneration kiln feed hopper. Screen undersize carbon, containing carbon fines and water, will drain by gravity into the carbon fines tank. An 800 kg/h diesel fired kiln will be utilized to treat 10 t of carbon per day, equivalent to 100% regeneration of carbon. The regeneration kiln discharge carbon will be transferred to the carbon quench tank by gravity, cooled by process water and stored in the regenerated sized carbon tank prior to being pumped back into the CIP circuit.

To compensate for carbon losses by attrition, fresh carbon is added to the carbon pre-attrition tank along with fresh water to mix and activate the carbon. The fresh carbon will then drain into the regenerated carbon tank.

17.3.11 Electrowinning and Gold Room

The pregnant solution generated from the elution column will be pumped to two (2) electrowinning cells from the pregnant solution tank. These cells will operate on a single-pass basis to produce a gold sludge. The barren solution will be collected in the barren solution tank.

The primary flow from the barren solution tank returns the solution to the elution circuit where it will be reused as barren stripping solution for the elution column.

The pregnant solution generated by the intensive cyanidation unit in the gravity circuit will be pumped to a separate pregnant solution tank and then be pumped to a dedicated electrowinning cell. Pregnant solution will be recirculated through the dedicated electrowinning cell until all gold is deposited onto the electrowinning cathodes.

The electrowinning cathodes will be manually transferred from the electrowinning cells to the cathode washing tank where a high-pressure washer will be used to dislodge gold sludge from the cathode surface. The sludge will be filtered by a filter press. The resulting filter cake will be dried in a drying oven and the resulting filtrate will be pumped back to the barren solution tank within the refinery.

The dried filter cake will then be transferred manually into the electric smelting furnace with flux materials where it will be batch smelted into gold doré bars and stored in a secure vault.

17.3.12 Tailings Storage Facility

The tailings storage facility will receive tailings from the cyanide destruction circuit. The cyanide content in the tailings will be further reduced by natural cyanide degradation and dilution from rainwater to meet IFC standard prior to release in the environment. Tailings pond supernatant (reclaim water) will be pumped back to the process water tank using vertical pumps on a barge.

17.4 Reagents

Reagents consumed within the process plant will be prepared on site and distributed via various reagent handling and makeup systems. These reagents include sodium cyanide, quicklime, hydrochloric acid, sulphamic acid, sodium hydroxide, copper sulphate, sodium metabisulphite, antiscalant, flocculant, leach accelerant (leach Aid) and activated carbon.

For the management of unexpected reagent spills, the reagent preparation and storage facilities will be located within containment areas designed to accommodate more than the content of the largest tank. Where required, each reagent system will be located within its own containment area to facilitate its return to its respective storage vessel and to avoid the mixing of incompatible reagents. Storage tanks will be equipped with level indicators, instrumentation, and alarms to ensure spills do not occur during normal operation. Appropriate ventilation, fire and safety protection, eye wash stations and showers, and Material Safety Data Sheet (MSDS) stations will be located throughout the facilities. Sumps and sump pumps will be provided for spillage control.

The reagents will be mixed, stored, and then delivered to the ILR, pre-leach thickener, leach, CIP, acid wash, elution, and cyanide destruction circuits. Dosages will be controlled by flow meters and control valves. The capacity of the storage tanks will be sized to typically handle one (1) day of production. The reagents will be delivered in dry form, except for hydrochloric acid, sulphamic acid, and antiscalant, which will be delivered as solutions.

17.4.1 Sodium Cyanide

Sodium cyanide will be used as a gold lixiviant. The cyanide will be shipped in briquette form by road to site in 18-t ISO containers and stored in the cyanide mixing facility; separate from the reagent storage and mixing facility. The sodium cyanide will be mixed with fresh water to form a cyanide solution for use in the Leach-CIP circuit.

17.4.2 Quicklime

Quicklime will be used as a pH modifier and will be supplied in dry form in bulk bags. Quicklime will be slaked, sent to a grit separator, and then added into a mix tank to prepare a milk of lime slurry, and transferred to a storage tank before addition into the process.

17.4.3 Copper Sulphate

Copper sulphate ($CuSO_4$) will be used as a catalyst for cyanide destruction. The copper sulphate will be supplied as a dry flake in one and a fifth (1.2)-tonne bulk bags and stored in the reagent storage area adjacent to the reagents mixing facility. The copper sulphate will be mixed with fresh water to form a copper sulphate solution ready for use in the processing facility.

17.4.4 Sodium Metabisulphite

Sodium metabisulphite ($Na_2S_2O_5$), also known as SMBS, will be the source of SO_2 for the cyanide destruction process and will be supplied in one and fifth (1.2)-tonne bulk bags as a dry reagent. SMBS will be stored in the reagent storage area where it will be transferred to the mixing facility to produce a SMBS solution prior to use in the cyanide destruction process.

17.4.5 Sodium Hydroxide

Sodium hydroxide (NaOH), also known as caustic soda, will be used as a pH modifier and will be supplied as solid beads in one and a fifth (1.2)-tonne bulk bags. Caustic soda will be mixed with fresh water prior to being used in the ILR, gold elution circuit, and cyanide mixing tank.

17.4.6 Hydrochloric Acid

Hydrochloric acid (HCl) will be used to remove inorganic carbonates from carbon in the acid wash process within the elution plant. They will be supplied in totes and stored in the reagent storage area adjacent to the reagent mixing facility.

17.4.7 Flocculant

Flocculant is a liquid polymer that will be used in the thickener to settle solids. It will be supplied in 700 kg bulk bags as a dry reagent. Flocculant will be shipped by road to site, offloaded by forklift, and stored in the reagent storage area adjacent to the reagents mixing facility. Flocculant will be diluted using fresh water and further diluted using an inline mixer with process water prior to being added into the processing facility.

17.5 Plant Services

17.5.1 Plant & Instrumentation Air

Three (3) air compressors will provide plant and instrument air for the process plant. Plant air receivers will act as a buffer storing air to account for variations in demand prior to being distributed throughout the process plant including the oxygen generation plant. Instrument air will be dried before being stored in the instrument air receivers and distributed throughout the plant.

17.5.2 Oxygen Generation

An oxygen generation plant will be used to provide industrial grade oxygen for the Leach-CIP circuit and cyanide destruction circuit. The plant air compressors will supply air to the oxygen generation circuit. The oxygen generation plant will include an oxygen plant air drier, a Pressure Swing Adsorption (PSA) oxygen generator, and an oxygen plant receiver.

17.5.3 Industrial Water

Raw water will be pumped to the plant industrial water tank by pumps from the catchment pond. The plant industrial tank will serve as storage for treated water supply.

Treated water in the tank will be used to supply the following services:

- Primary crushing circuit dust suppression water.
- Mining pit underground equipment.
- Reagent preparation water.
- Slurry pumps gland seal water.
- Cooling water systems; i.e., elution circuit, mill motor cooling.
- High pressure wash water in the refinery.
- Make-up water for the process water system.

Raw water from the industrial water tank will be pumped through filters to remove particulates.

17.5.4 Domestic Water

Feed to the domestic treatment plant is supplied by pumps from the catchment pond. The domestic water will feed the camp, mine maintenance facility, mine infrastructures, processing plant infrastructures, sewage treatment plant, fire water tank, domestic use such as washroom, showers and kitchen and emergency domestic water tank. The emergency domestic water will be used in the process plant for safety showers.

17.5.5 Fire Water

Feed to the fire water tank is supplied from the domestic water treatment plant. The fire water tank will have a minimum capacity of 108 m^3 and will feed the plant and permanent campfire suppression systems, fire hydrants and hose reels.

17.5.6 Gland Seal Water

Water for the gland seal water system will be supplied by raw water from the raw water tank. The gland seal water tank will store and distribute gland water to the plant with gland seal water pumps in a duty-standby configuration.

To prevent particulates from causing damaged gland seals throughout the plant, the water feeding the gland water tank will pass through 25-micron particulate filters.

17.5.7 Process Water

Process water will comprise of pre-leach thickener overflow, raw water from the catchment pond, contact water, and tailings reclaim water. Process water will be stored in the process water storage tank and distributed by the process water pumps, in a duty-standby configuration.

17.6 Metallurgical Accounting

Several samplers will be provided throughout the plant to generate composite shift samples from key process streams. Two (2) types of sampling will be performed, metallurgical and process control sampling.

Metallurgical samplers will be used to generate shift composite samples that will be assayed for plant metallurgical accounting. The following process streams will be equipped with metallurgical samplers:

- Primary cyclone overflow.
- CIP Tailings.

The metallurgical samplers will sample feed and tailings product which will allow an accurate metal balance of the plant to be completed.

Process control sampler will generate samples used to monitor unit processes in the plant. The process control samplers will be used to generate shift composite samples on process streams that will provide plant operation performance data.

The following process streams will be equipped with process control samplers:

- Leach feed.
- Leach tailings.
- Final tailings.
- Pregnant solution to electrowinning.
- Barren solution after electrowinning.

All samplers will produce 5-10 L of slurry that can be transported to the assay laboratory for further analysis.

A weightometer on the stockpile feed conveyor will measure primary crushed ore tonnage, and a weightometer on the SAG mill feed conveyor will determine mill feed tonnage.

A manual belt cut sampling point on the SAG mill feed conveyor will allow for the collection of a mill feed head grade sample for cross-checking with the calculated head grade. This sample will also be utilized to establish the moisture content of the mill feed.

Regular surveys of the gold and silver in circuit will allow a reconciliation of precious metals in the feed compared to doré production.

Water supplied and used in the various areas will be continuously monitored.

Reconciliation of the reagents used over relatively long periods will be achieved by delivery receipts and stock takes. On an instantaneous basis, reagent usage rates to unit operations will be measured and accumulated using flowmeters.

17.7 Plant Control System

The following provides a broad overview of the control strategy that will be employed for the process plant.

The general control philosophy for the process plant will be one with a moderate level of automation and remote-control facilities to allow critical process functions to be carried out with minimal operator

intervention. Instrumentation will be provided within the plant to measure and control key process parameters.

The main control room, located in the process office, will house PC-based operator interface terminals (OIT) and a single server. These workstations will act as the control system supervisory control and data acquisition (SCADA) terminals. The control room is intended to provide a central area from where the plant is operated and monitored and from which the regulatory control loops can be monitored and adjusted. All key process and maintenance parameters will be available for trending and alarming on the process control system (PCS).

Additional OITs will be provided for data logging and engineering / programming functions.

A field touch panel will be installed in the feed preparation area to allow local operator control of the crushing plant to facilitate ease of operation for rock breaking and stockpiling if required. A second field touch panel will be installed in the elution area to allow local operator control of the elution sequence. A third field touch panel will be supplied for the grinding circuit area.

The process control system that will be used for the plant will be a programmable logic controller (PLC) and SCADA-based system. The PCS will control the process interlocks and PID control loops for non-packaged equipment. Control loop set-point changes for non-packaged equipment will be made at the OIT.

In general, the plant process drives will report their ready, run, and start pushbutton status to the PCS and will be displayed on the OIT. Local control stations will be located in the field in proximity to the relevant drives. These will, as a minimum, contain start and latch-off-stop (LOS) pushbuttons that will be hard-wired to the drive starter. Plant drives will predominantly be started by the control room operator after the equipment has been inspected by an operator in the field.

The OITs will allow drives to be selected to Auto, Local, Remote, Maintenance or Out-of-Service modes via the drive control popup. Statutory interlocks, such as emergency stops and thermal protection, will be hardwired and will apply in all modes of operation. All PLC-generated process interlocks will apply in Auto, Local and Remote modes. Process interlocks will be disabled or bypassed in Maintenance mode with the exception of critical interlocks, such as lubrication systems on the mill.

Local selection will allow each drive to be operated by the operator in the field via the local start pushbutton, which is connected to a PLC input. Remote selection will allow the equipment to be started from the control room via the drive control popup. Maintenance selection will allow each drive to be operated by maintenance personnel in the field via the local start pushbutton, which is connected to a PLC input. A

PLC output will be wired to each drive starter circuit for starting and stopping drives. Status indication of process interlocks as well as the selected mode of operation will be displayed on the OIT.

Vendor-supplied packages will use vendor standard control systems as required throughout the project. Vendor packages will generally be operated locally with limited control or set-point changes from the PCS system. General equipment fault alarms from each vendor package will be monitored by the PCS system and displayed on the OIT. Fault diagnostics and troubleshooting of vendor packages will be performed locally.

The use of actuated isolation or control valves will be implemented around the plant for automatic control loops or sequencing as part of the plant control or the elution sequence. All actuated valves and control valves will be operated from the OITs with remote position indication available. Automatic control valves will be controlled by PID loops within the PCS.

The PCS will perform all digital and analogue control functions, including PID control, for all non-packaged plant. Faceplates on the PCS displays will facilitate the entry of set-points, readout of process variables (PVs) and controlled variables (CVs), and entry of the three (3) PID parameters (proportional, integral and derivative).

The majority of equipment interlocks will be software configurable. However, selected drives will be hard wired to provide the required level of personal safety protection (e.g., the emergency stop buttons associated with every motor and the pull wire switches associated with conveyors).

All alarm and trip circuits from field or local panel-mounted contacts will be based on fail-safe activation. Alarm and trip contacts will open on abnormal or fault condition. If equipment shutdown occurs due to loss of mains power supply, the equipment will return to a de-energized state and will not automatically restart upon restoration of power.

Sequential group starts and sequential group stops will not be incorporated for non-packaged plant equipment, except for the elution circuit. However, in any process, critical safety and equipment protection interlocks will cause a cascade stop in the event of interlocked downstream equipment stopping (e.g., trip of SAG mill feed conveyor will result in stop of the upstream apron feeder). Standard vendor packages may include automatic sequence start / stop controls within the vendor package only.

17.8 Plant Consumption

17.8.1 Energy

The power demand for the process plant, along with the rest of the project, will be provided by a dedicated power plant. The power demand for the future operation is discussed in Section 18.7.

17.8.2 Reagents & Consumables

Reagent storage, mixing and pumping facilities will be provided for all reagents for the process plant. Reagents and consumables usage are summarized in Table 17.2 and Table 17.3, assuming 100% fresh rock.

Table 17.2: Reagents Consumption

Description	Delivered Form	Average Usage (kg/t)
Sodium Cyanide	Briquette ISO tank	0.32
Lime (@ 90% CaO)	1.5T bag	1.44
Hydrochloric Acid (32% strength)	1,100 L IBC	0.06
Sodium Hydroxide	1.2 t bags (dry)	0.03
Copper Sulphate	1.2 t bags (dry)	0.14
SMBS	1.2 t bags (dry)	1.19
Flocculant	700 kg bags (dry)	0.05
Activated Carbon	500 kg bags (dry)	0.04

Source: GMS, 2024

Table 17.3: Consumables Consumption

Description	Delivered Form	Usage
Gyratory Crusher – Mantle	lot	2 sets / year
Gyratory Crusher – Concave Segments	lot	11 set / year / segment
SAG Mill Liners	lot	2.0 sets / year
Ball Mill Liners	lot	2.0 sets / year
Cyclone – Body	lot	1.0 sets / year
Cyclone – Vortex & Spigot	lot	4.0 sets / year
Trash Screen Panels	lot	2.0 sets / year

Description	Delivered Form	Usage
Loaded Carbon Screen Panels	lot	1.0 sets / year
Barren Carbon Screen Panels	lot	1.0 sets / year
Carbon Safety Screen Panels	lot	2.0 sets / year
Interstage Screens Panels	lot	2.0 sets / year
SAG Mill Grinding Media (125 mm)	bulk	0.39 kg/t
Ball Mill Grinding Media (50 mm)	bulk	0.63 kg/t

Source: GMS, 2024

17.9 Process Plant Personnel

The personnel for the process plant will consist of management, operations, maintenance, and laboratory. Operating staff will work 11-hour days and night shifts on a 2-week on-1-week off rotation cycle and management will work 12-hour days.

Annual process plant personnel requirements are provided in Table 17.4.

Table 17.4: Process Plant Personnel

Department	Position	Compliment
	Mill Manager	1
Process Management	Mill Superintendent	1
	Assistants	3
Metallurgy	Chief Metallurgy	1
	Senior Metallurgist	1
	Metallurgist	2
	Technicians	6
Assay Laboratory	Chief Chemist	1
	Chemist	1
	Assayers	6
	Sample Prep.	15

Department	Position	Compliment
Process Operations	General Foreman	2
	Trainers	2
	Supervisors Expat NA	6
	Supervisors NA	6
	Lead Operators	15
	Operators	39
	Laborers	12
Process Maintenance	Mechanical General Foreman	2
	Electrical General Foreman	2
	Mechanical Planner	2
	Electrical Planner	2
	System Integrator	2
	Supervisor Mechanic	4
	Electrical Supervisor	2
	Technician Electrician	6
	Lead Mechanic	21
	Mechanics	42
	Laborers	12
Power Generation	General Superintendent	1
	General Foreman	1
	Supervisors	2
	Operator Maintenance	6
	Maintenance Technician	6
Total		**233**

Source: GMS, 2025

18. PROJECT INFRASTRUCTURE

The Project infrastructure is designed to support the operation of an open pit (OP) and underground (UG) mine feeding a process plant with a nominal 7-6 Mtpa, operating on a 24-hour per day, 7-day per week basis. It is designed in consideration of local conditions and topography.

18.1 Site Layout

Figure 18.1 presents the project site plan with the onsite infrastructure. This plan is designed to minimize environmental impacts, provide secure site access, minimize construction costs, and enhance operational efficiency.

18.1.1 Roads and Drainage

Currently, there are existing roads and a trail network accessible by light vehicles or all-terrain vehicles (ATVs). A new network of laterite-surfaced roads will be constructed to accommodate both light and heavy vehicles. The heavy haul roads surrounding the mine and construction quarry will provide access for large trucks to key facilities, including the maintenance facilities, fuel bay, wash bay, crusher, waste storage area, and tailing dam areas. All other site areas will be accessible via roads designed for light vehicles.

The project site will be accessed mostly by the existing Puruni Road, with the addition of a planned 13 km laterite-surfaced road. Upgrades to specific segments of the Puruni road are designed to improve gradients, surface quality, and drainage systems. A short bridge must be upgraded to support the heaviest equipment to reach site. A short connecting road from the Oko Wharf landing to the Puruni road will complete the land access to site. To ensure reliable communication along the access route, a comprehensive radio communication system will be implemented, supported by strategically placed repeater towers.

The site will feature approximately 20 km of service roads, interconnecting key infrastructures such as the airstrip, explosives storage facility, tailings storage facility, operations sites and camp site. Few borrowed pits have been identified for laterite within the existing impacted area of the project, connected by the service roads. A potential sand pit at the western extremity has also been located as an opportunity to reduce our quantity of imported sand. Additional washing equipment is expected for the sand quality of the infrastructures, such as TSF and critical concrete.

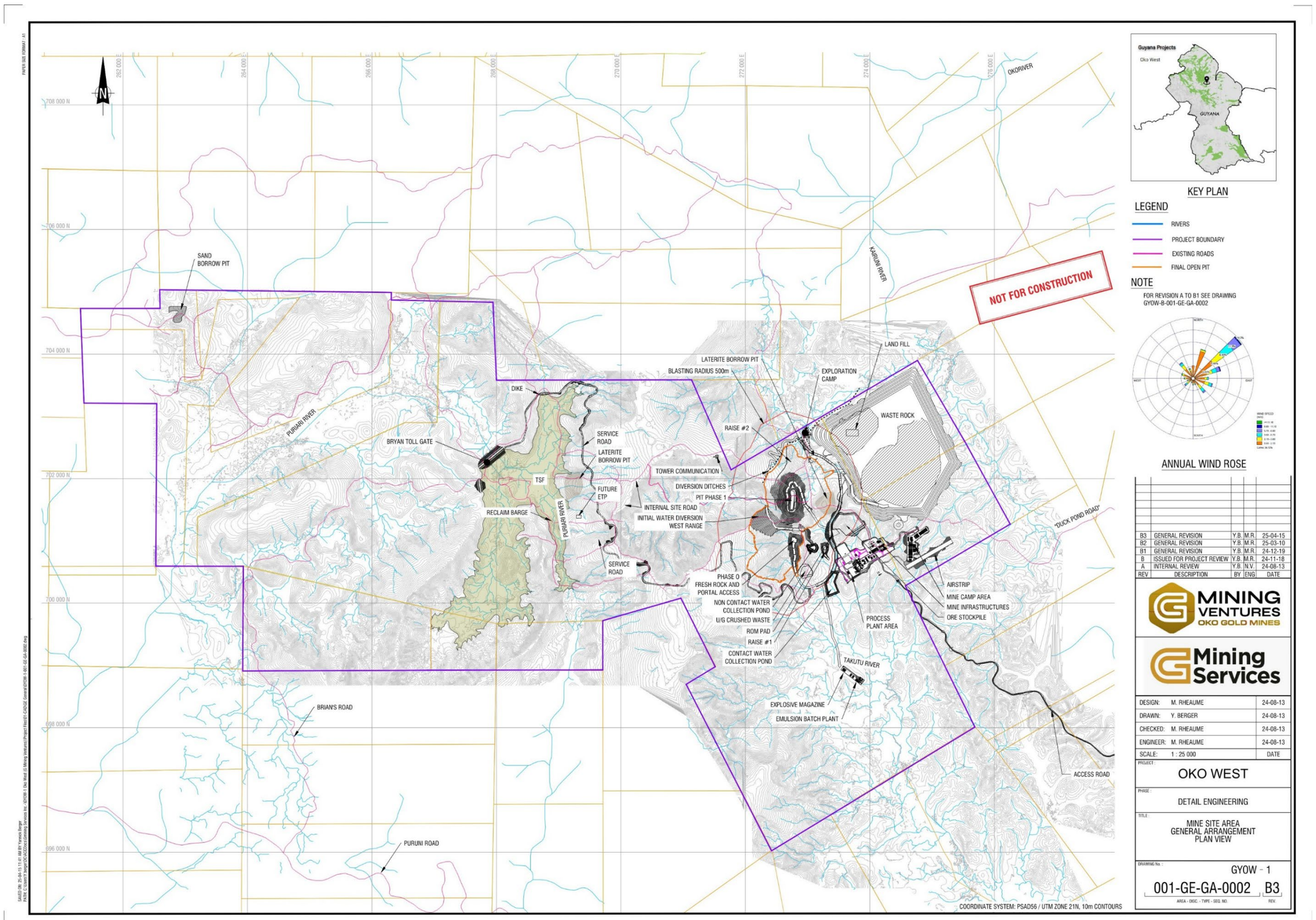
Drainage channels and culverts will be installed to divert water away from critical infrastructure, such as the plant site, process plant, open pit, and waste storage. Plant site pads are designed with a slope directing

water away from buildings towards catchment ditches. Surface water from the vicinity of the truck facilities that are suspected to carry hydrocarbons will be collected and diverted to designated oil / water separators.

A dedicated water diversion ditch will be constructed to redirect surface water originating west of the pit, preventing it from flowing through the northern pit area. Currently, this water naturally traverses the northern section of the pit and must be controlled early in the Project.

To mitigate this risk, an initial water re-routing system will be implemented during the construction phase, directing flow toward the southern water catchment. As the southern pit is developed, a permanent diversion system will be established to ensure effective long-term water management and minimize operational disruptions. This phased approach allows for adaptive water control while aligning with site development.

Figure 18.1: General Site Plan



18.2 Site Infrastructure

The primary buildings have been strategically positioned to facilitate efficient construction access and to use the existing topography, thereby minimizing the volume of bulk earthworks. This placement also adheres to geotechnical recommendations, ensuring structural stability and safety.

Infrastructure buildings have been designed as modular prefabricated structures combined with steel and cladding where required. Buildings will be equipped with smoke, carbon monoxide and heat detectors, as well as appropriate fire extinguishers with one (1) main fire alarm panel. All infrastructure will comply with the European and Caribbean standard building codes referenced to the North American standard code. As detailed in the fire water system section, the main fire protection system will be an independent fire hydrant loop.

The camp buildings, such as the Kitchen, Administration Office, Laundry, Recreational Centre, Gym and Ablution Units, will be placed within walking distance. This will optimize the electrical and piping network and create a PPE-free zone, away from the industrial area.

An access gate and guard facility, including a search and site access control building, will be installed at the site entrance. Preliminary screening of all traffic entering and leaving the property will be conducted at this location.

The control building will house the security access control office and provide control of all personnel entering and leaving site. Only site-approved and security-cleared vehicles will be allowed to proceed beyond this point.

A new 850-m-long airstrip, classified as category 2, will be constructed to accommodate most aircrafts in the country. This airstrip will facilitate the transport of senior personnel, critical supplies, medical emergencies and the export of doré.

18.3 Camp Accommodations

18.3.1 Dormitory

The permanent camp is designed to accommodate a construction peak of 1,584 individuals on site, reducing it to 900 individuals during operations. Additional to this quantity, the Oko Wharf will have 50 beds, and the upgraded exploration camp will have 350 beds.

The buildings will be divided into several units, also referred to as dormitories, each consisting of rooms on a single level. Table 18.1 details the camp capacity.

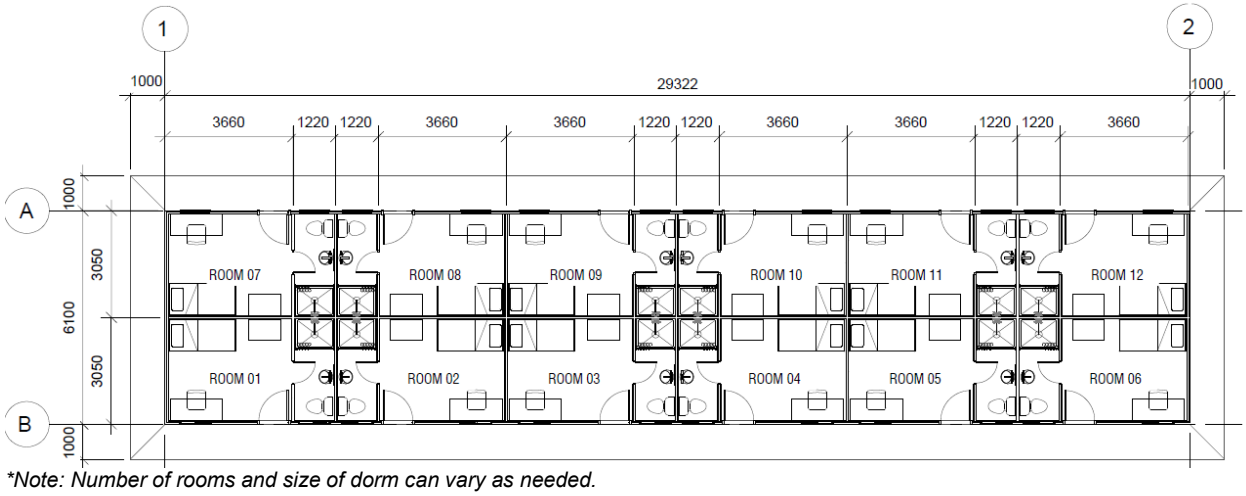
Table 18.1: Camp Capacity

141 Permanent Camp										
Building Type	Total Area (m^2)	Block Dimensions (m)				Module Description				
		L (m)	W (m)	H (m)	Area (m^2)	Room per Block	Ppl per Block	Block	Total Rooms	Bed
Type A	1,640	42	4.9	2.5	205	12	12	8	96	96
Type B	2,635	54	4.9	2.5	264	24	24	10	240	240
Type C (M)	4,412	54.8	7.3	3	401	24	96	11	264	1,056
Type C (F)	1,023	54.8	9.3	3	512	24	96	2	48	192
Total	**9,711**							**31**	**648**	**1,584**

**Note: Distribution of beds and number of rooms per building may vary. Quantities shown in the table are based on a typical type of dorm.*

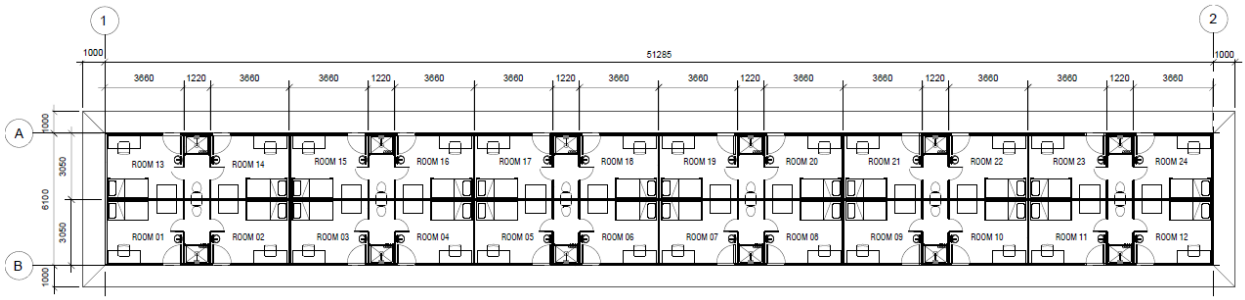
For a Type A building, the capacity is 12 people, each with an individual bathroom, along with one (1) janitor room and one (1) E-room. Type B buildings consist of 24 individual rooms, with one (1) bathroom for every two (2) rooms, plus one (1) janitor room and one (1) E-room. Type C buildings have a maximum capacity of 96 people accommodated in bunk beds with four (4) people per room, one (1) janitor room and one (1) E-room per block. After the construction peak, the bunk beds are eliminated, and the capacity of the Type C building is reduced to 48 beds. Figure 18.2, Figure 18.3 and Figure 18.4 illustrate the typical camp dorms.

Figure 18.2: Typical Camp Dorm – Type A



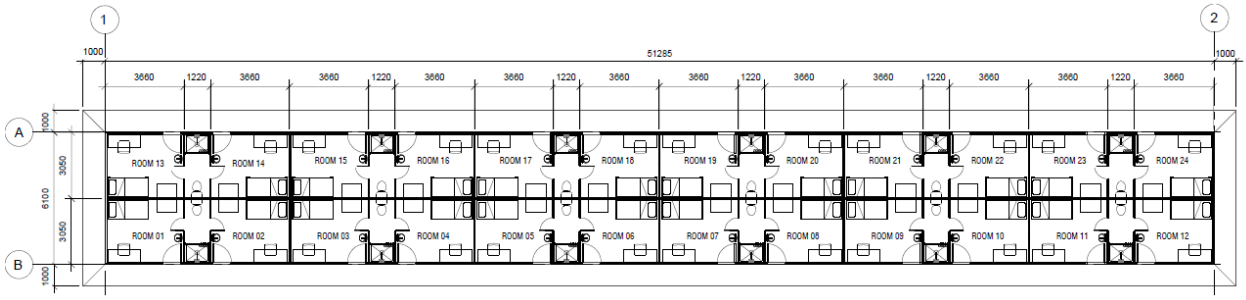
**Note: Number of rooms and size of dorm can vary as needed.*

Figure 18.3: Typical Camp Dorm – Type B



**Note: Number of rooms and size of dorm can vary as needed.*

Figure 18.4: Typical Camp Dorm – Type C



**Note: Number of rooms and size of dorm can vary as needed.*

18.3.2 Kitchen & Lunchroom

The kitchen and lunchroom design assumes a service for 1,500 people, including cooking areas, serving areas, hygienist zones, staff offices, freezer rooms and a storage area. Cooking stations will have commercial-grade appliances and proper ventilation. Designated meal preparation and serving spaces will manage high volumes, and cleaning zones will comply with health regulations. Offices for management and staff meeting rooms are included. Freezer rooms will ensure food safety, and a well-organized storage area will be provided.

HVAC equipment will be outside the main building, insulated to minimize noise and heat transmission. A maintenance road will encircle the complex, providing back access to avoid disrupting pedestrian traffic at the main entrance. This layout ensures efficient, safe, and comfortable operations with strategic placement and circulation planning supporting optimal flow. Figure 18.5 and Figure 18.6 illustrate the planned kitchen and lunchroom.

Propane will only be needed in the kitchen, therefore, an individual bottle of 60 lbs will be delivered to site and replaced by a contracted supplier.

Figure 18.5: Typical Kitchen & Lunchroom – 3D View

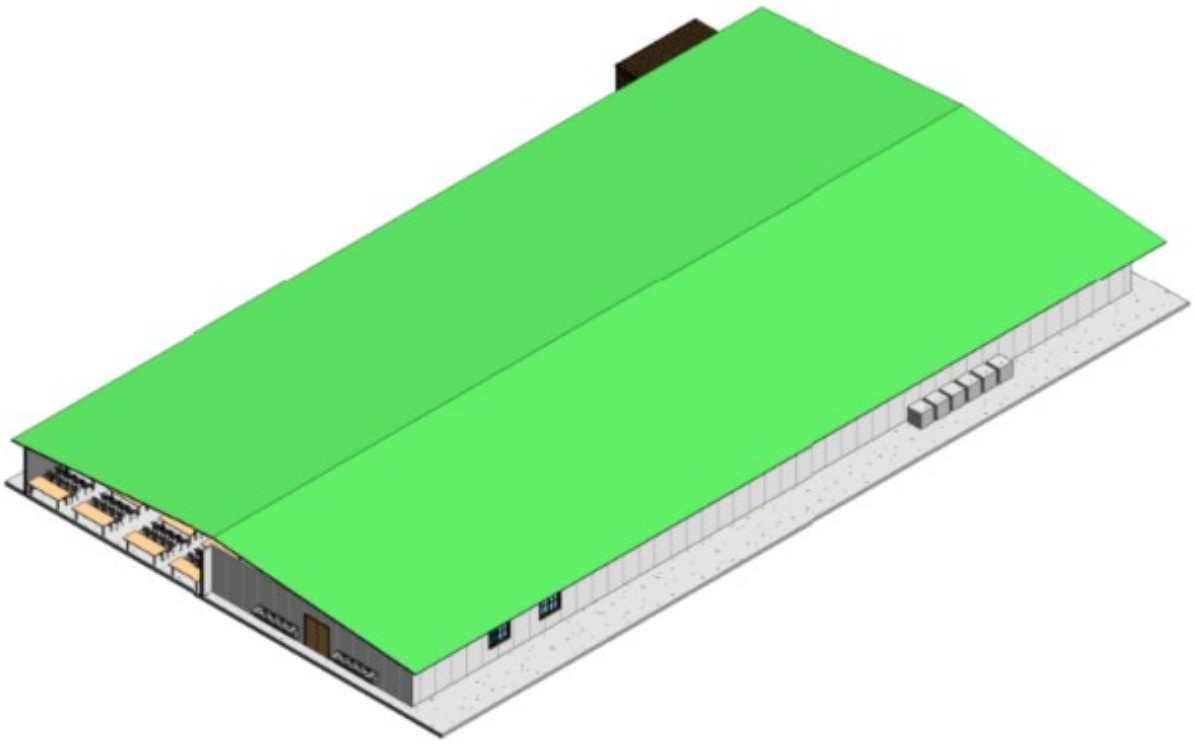
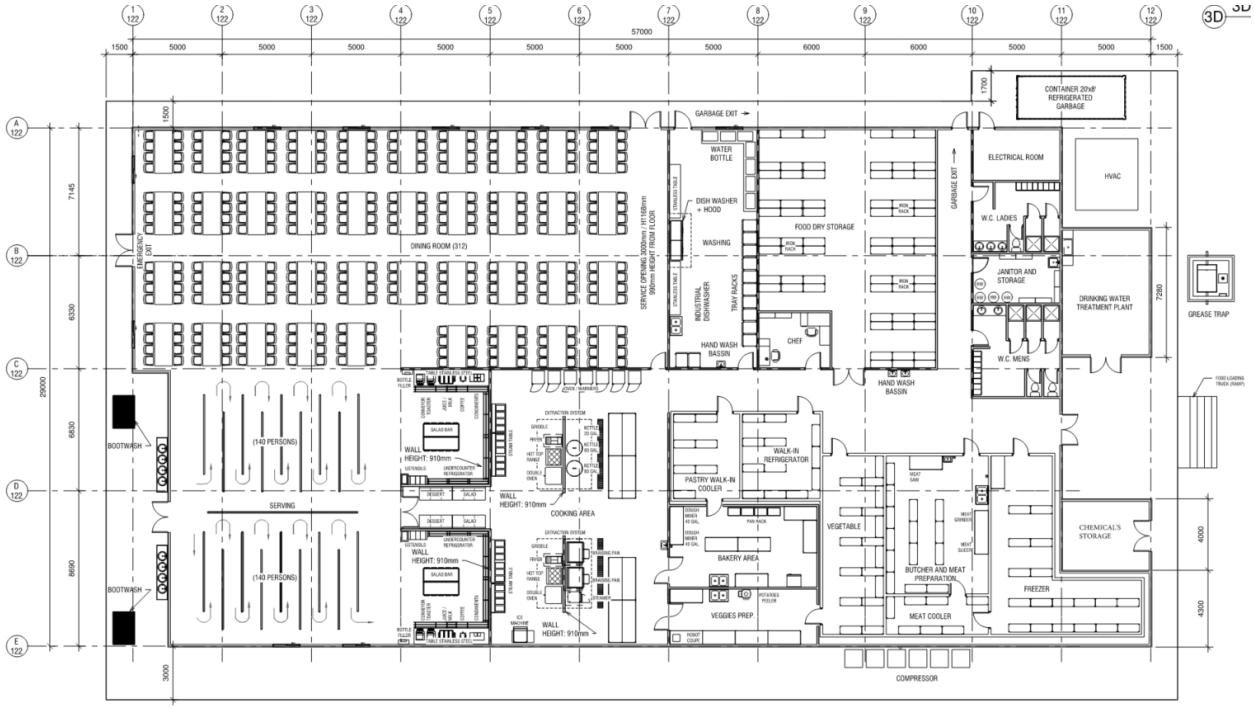


Figure 18.6: Typical Kitchen & Lunchroom – Plan View



**Note: Building capacity can vary as needed.*

18.3.3 Camp Office Welcome Centre, Laundry and Recreation Centre

The camp office is a single-story building with an approximate footprint of 750 m^2. It is located adjacent to the laundry room and the recreational room. The building will house offices and open areas for the Transportation Supervisor and Camp Supervisor, meeting rooms, a janitorial room, a storage room and a washroom. Figure 18.7 and Figure 18.8 illustrate the camp office and laundry area.

The gym will feature ablution units, an exercise equipment area, aerobic exercise spaces and other fitness activities. Furthermore, recreational areas will include sports fields to provide sufficient space for outdoor activities.

The laundry is equipped with industrial electric dryers and washing machines. Air will be compensated into the room to balance the air extracted from the dryers.

The Recreational Centre's future expansion facilities will include a game area, internet access with shared computers, a TV room, reading rooms, a convenience store and other amenities.

All these facilities will be located in proximity to the main accommodation area to maintain a prudent distance that minimizes undesirable noise and pedestrian traffic disruptions.

Figure 18.7: 3D View of Camp Office / Laundry / Rec Centre Area

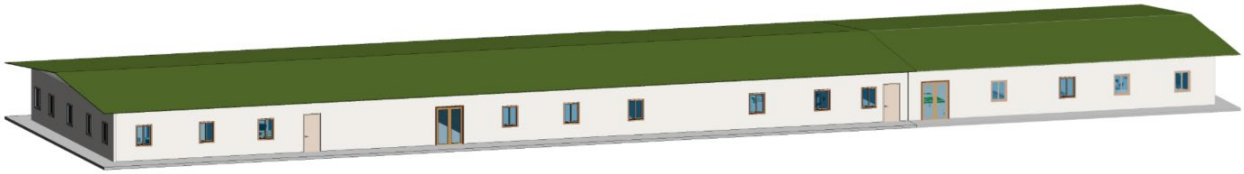
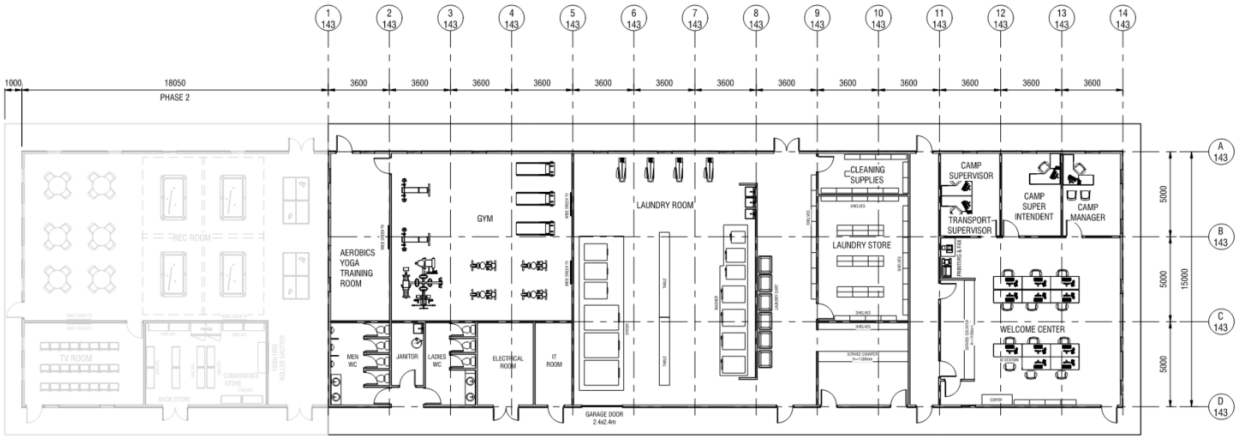


Figure 18.8: Camp Office / Welcome Centre / Rec. Centre - Plan View

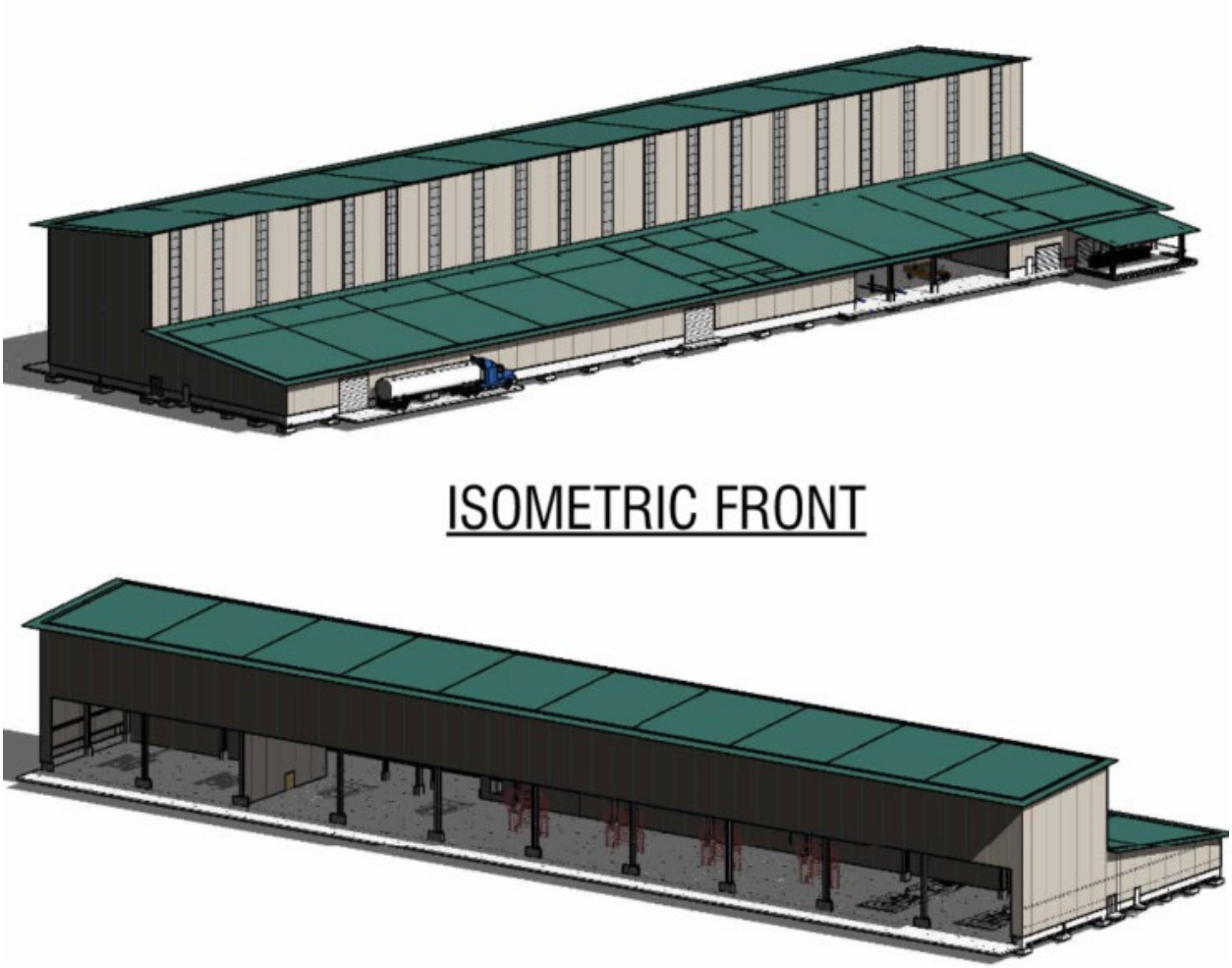


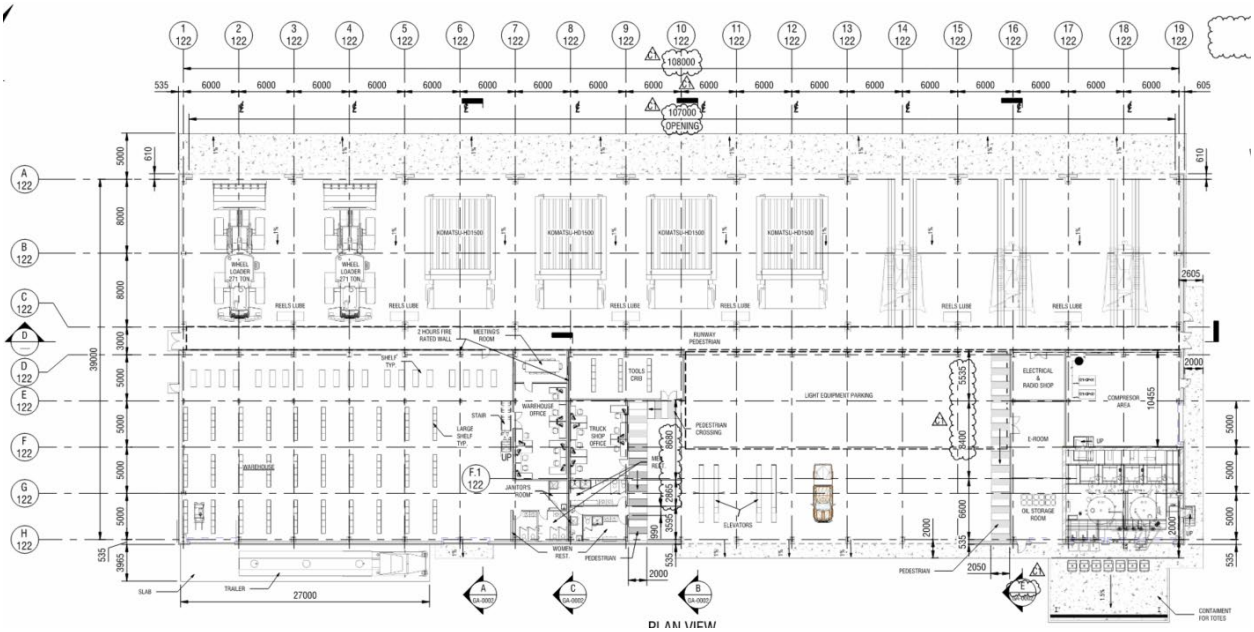
18.4 Mine Infrastructure

18.4.1 Mine Maintenance Facility & Warehouse Area

The Mine Maintenance Facility and Warehouse (Figure 18.9) is a single-story building and will be located in the Balance of Plant (BOP) area. The mining machinery will have easy access to its restricted sector. The facility includes nine (9) service bays sized for haulage trucks and six (6) light vehicle bays. A lubricant and grease storage sector will include the various products which will be distributed to the various bays with fixed piping and pumps. Three (3) dedicated areas meet these needs: a bulk delivery room with permanent tanks for high consumption, an oil storage interior room, and an outside shelter for tote delivery type with enough room for both full and empty totes. The truck shop will also include office space, a kitting room, tool storage, a meeting room, restrooms and lockers. The building construction is based on a conventional steel structure with an overhead crane; insulated sandwich panels will be used for the walls and roof of the building. The warehouse area will be contained in this building and will have all the necessary facilities for its operation. All water on the facility floor will be collected in a centralized trench leading to an oil separation system discharging at the contact water pond.

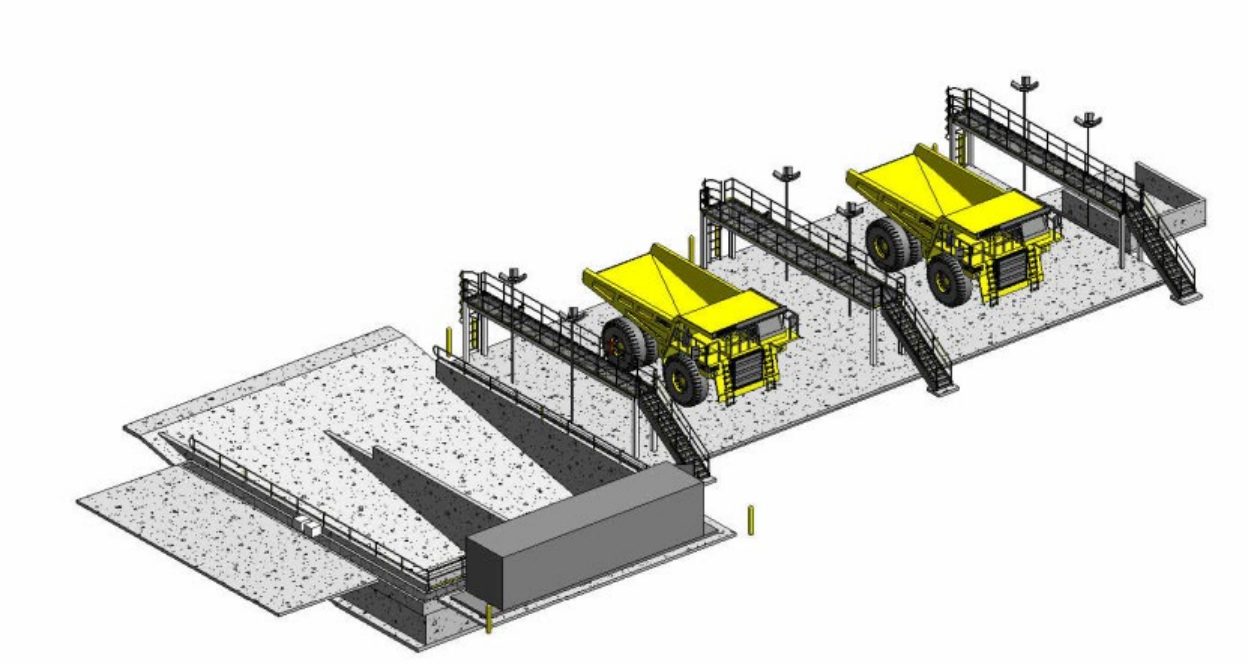
Figure 18.9: Maintenance Facility and Warehouse Area





A wash bay (Figure 18.10), also sized to service haulage trucks, will be located near the maintenance facility. Wash water will be captured, settled, and oil-separated, and the water will be recycled for the wash bay needs. When there is an excess of water in the facility, it shall be sent to the contact water pond. One (1) section will be dedicated to open-pit trucks and one (1) to light vehicles and underground trucks.

Figure 18.10: Wash Bay



18.4.2 Main Administration Building

The Main Administration Building, shown in Figure 18.11, has a footprint of 1,200 m^2. The ground slab will be structural concrete supporting a light structure. This one-story building will be used by the site management and general services, mine management and technical services, mine rescue facility, medical centre and ambulance parking, with other facilities, mine dispatch and various meeting rooms. The main camp kitchen lunchroom, being at less than 500 m, the personnel will walk or take the buses to reach the lunchroom.

Figure 18.11: Main Administration Building

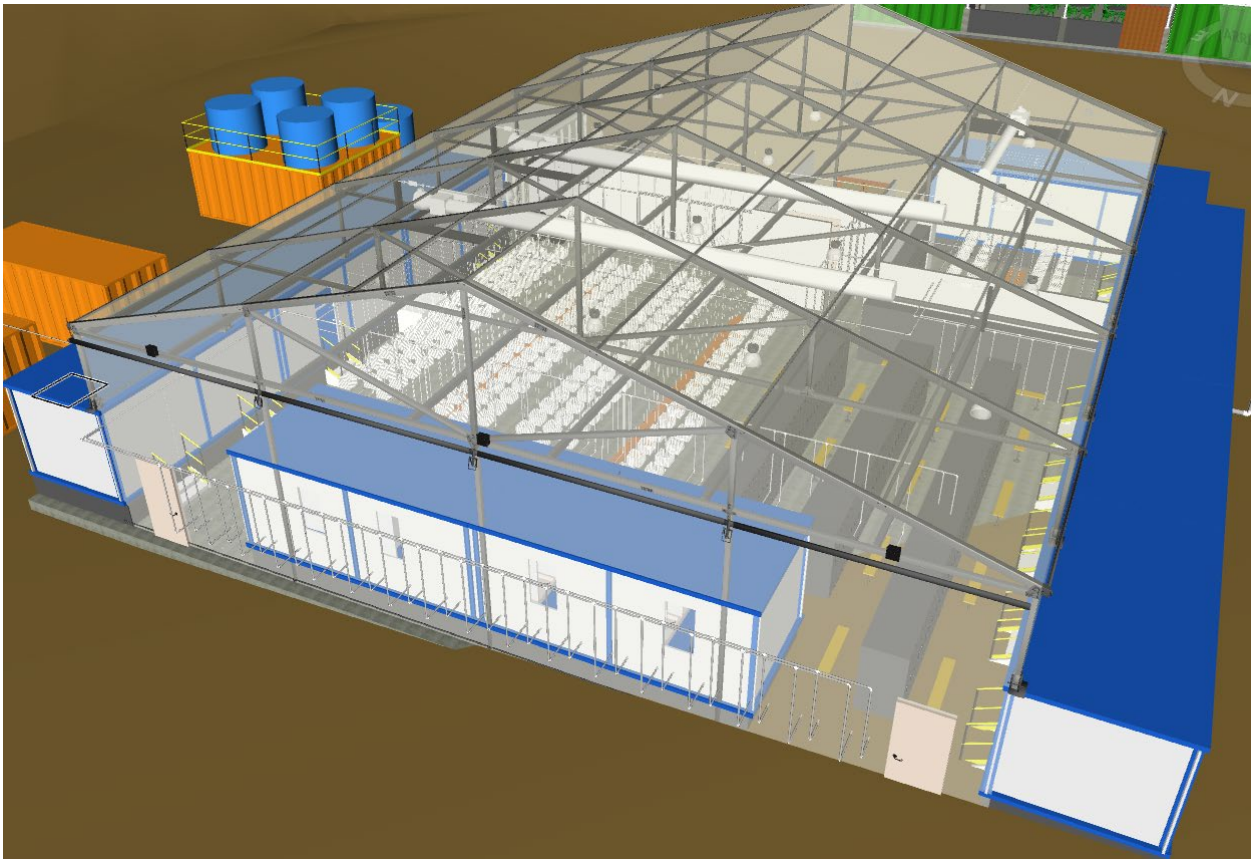
18.4.3 Mine Dry

As part of the underground development, a Mine Dry (Figure 18.12) will be constructed to provide an essential change room and drying infrastructure for underground personnel. The facility will consist of two (2) main zones: a dry area and a change room. These will serve as part of the sustaining infrastructure required for continuous underground operations.

To optimize operational efficiency and promote workforce well-being, the Mine Dry will be strategically located adjacent to the Mine Administration Building. This proximity will enhance logistical synergy, facilitate shared services, and improve overall employee experience.

Special attention will be given to the design of the ventilation and dehumidification systems. These will be engineered to accommodate the high moisture load associated with drying humid and soiled underground clothing. Airflow rates, humidity control, and energy recovery will be sized and specified to meet operational and hygienic standards, ensuring rapid drying and mould prevention while maintaining energy efficiency.

Figure 18.12: Mine Dry - Plan View



18.4.4 Explosive Storage

The explosive storage facility (Figure 18.13) comprises two (2) primary components: an emulsion production and storage plant and designated explosive magazines, co-located at a secure area on the mine site. This infrastructure is designed to support the full spectrum of blasting activities during mine operations.

The emulsion plant is engineered to produce up to 100 tons of explosive emulsion over a 12-hour production cycle. This capacity ensures alignment with operational blasting demands, including during peak activities such as bench development or stope production.

The on-site setup allows for both bulk emulsion storage and the mixing and loading of explosives into delivery trucks, enabling direct transport to blast sites. This modular system increases efficiency while maintaining segregation of components until the final loading stage, minimizing risk.

The explosive facility includes dedicated storage for all necessary blasting reagents and initiating systems, detailed as follows:

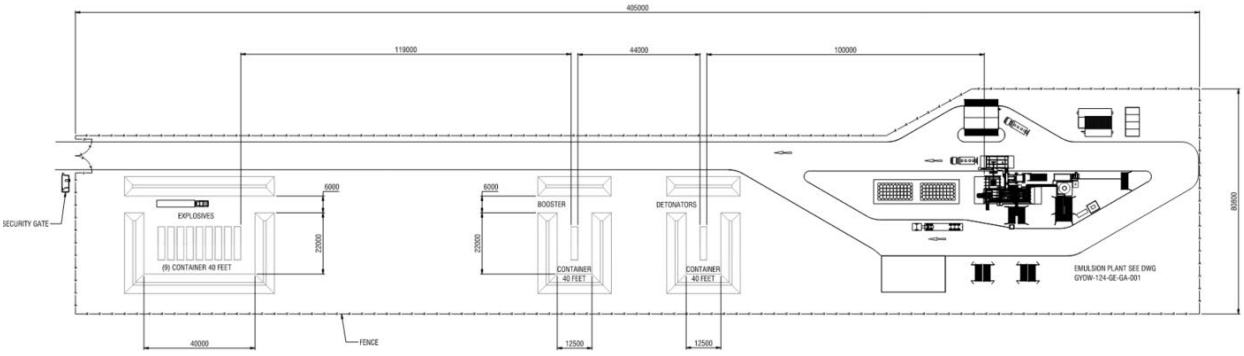
- Ammonium Nitrate (AN): A purpose-built structure provides storage for up to 1,000 tonnes of AN, equivalent to approximately 30 days of full operation. This structure is ventilated, climate-controlled, and compliant with applicable safety distances and separation standards.
- Fuel Oil / Diesel: Stored in double-walled, above-ground tanks located within a containment berm. Capacity meets operational blending requirements, with daily transfer monitored and logged.
- Other Chemical Reagents (if required for specific emulsion blends): Secured in a separate chemical storage area with controlled access and spill containment.
- Detonators, Boosters, and Blasting Caps: Stored in certified magazines, ensuring segregation by type and compatibility. Quantities are controlled through site-specific blast plans and inventory logs.

The explosive storage facility is strategically located approximately 1 kilometre south of the nearest permanent infrastructure on site. This placement ensures compliance with minimum separation distances, reducing potential risks to personnel and critical assets. The area was selected for its natural isolation and terrain compatibility, minimizing the need for extensive land modification while preserving a secure operational buffer.

The explosive storage area is enclosed within a secured and fenced perimeter, with controlled access gates and regular security patrols.

To address elevated consumption during periods of intensified blasting activity, supplemental storage capacity for AN is maintained at the barge landing area. This auxiliary site supports logistical continuity and ensures the main plant maintains an uninterrupted supply even during seasonal resupply constraints.

Figure 18.13: Explosive Storage

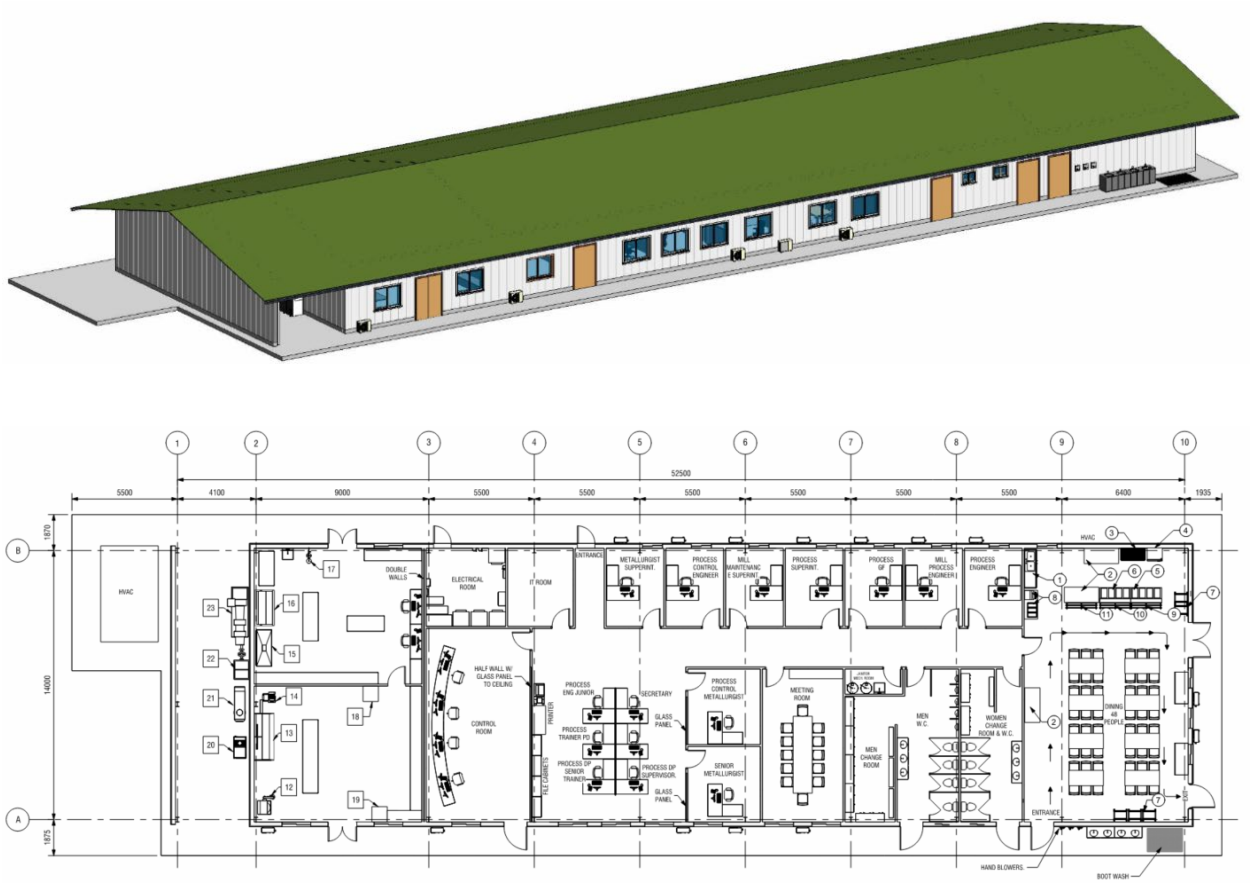


18.5 Process Infrastructure

18.5.1 Mill Offices

The Mill Office building (Figure 18.14) has an area of 735 m^2. It will have several offices for management, maintenance and operating personnel as well as a processing plant control room. In addition, the building has a portion for the metallurgical laboratory, a meeting room, a lunchroom, and men's and women's changing rooms and washrooms. The building will have a ground slab of structural concrete and steel structure with insulated panels for walls and roof.

Figure 18.14: Mill Office



To meet operational and safety constraints at the processing plant, a dedicated satellite lunchroom will be constructed within the fenced control area as part of the mill office. This facility is the only satellite lunchroom planned for the project and is specifically intended to eliminate the need for plant personnel to exit the controlled zone during meal breaks. This approach helps maintain strict access control, minimizes exposure to external contaminants, and enhances overall time efficiency.

- The lunchroom is designed to accommodate all operational personnel assigned to the process plant in two (2) meal shifts (batches).
- A fixed seating configuration allows for efficient turnover between meal groups, minimizing downtime and maintaining continuous operational coverage in the plant.
- Food will not be prepared on site, but rather delivered from the central kitchen facility, ensuring consistency with health, hygiene, and quality assurance protocols.
- The lunchroom will be equipped with warming units, refrigeration cabinets, and sanitary hand-washing stations.

18.5.2 Assay Laboratory

A complete sample preparation and assay laboratory facility (Figure 18.15) will be built adjacent to the administrative building. It will include complete lab equipment packages to provide assays for mine grade control, process control, effluent monitoring and other environmental requirements.

This facility will consist of a 17 m x 71 m structural steel building with offices for staff and equipped with an air compressor and storage space for various lab consumables. It will be capable of processing 500 samples per day. The facility includes specialized zones for sample reception, preparation, wet chemistry, furnace operations, PAL leach testing, and instrumentation. It also includes washrooms, offices, and meeting space. The structure is designed with a 4-metre clear ceiling height to allow for optimal ventilation, circulation, and extraction.

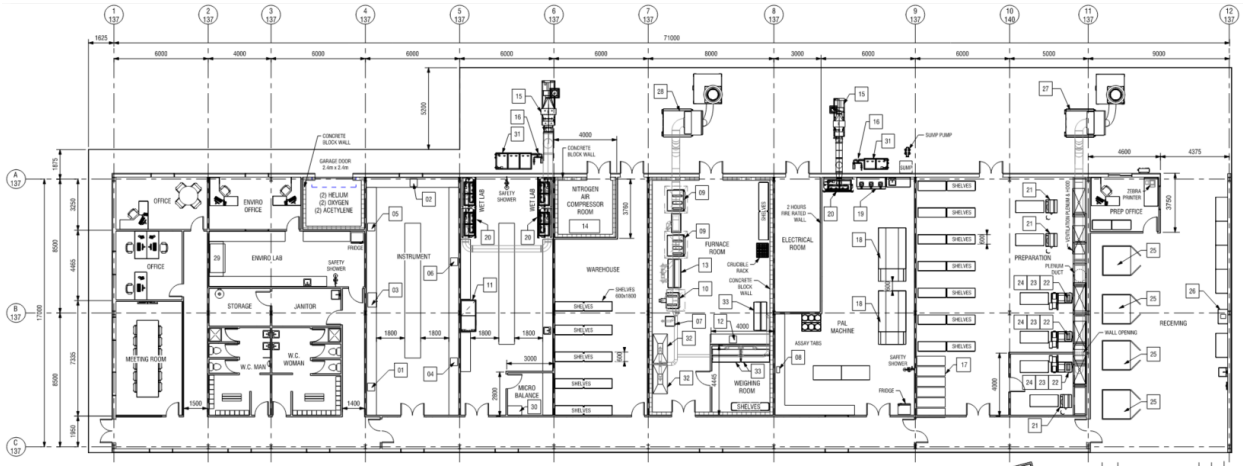
The ventilation system for the Analytical Laboratory is designed to meet industrial safety and performance standards. Each operational area that handles volatile chemicals, dust-generating equipment, or high-temperature processes is equipped with dedicated extraction hoods and plenums, ensuring contaminant capture at the source. The wet chemistry zones include four (4) 6-foot fume hoods connected to a dual fume scrubber system for acid vapour neutralization. In high-risk areas such as the PAL room and furnace room, local exhaust ventilation is supplemented with gas detectors (e.g., cyanide sensors) and sump-pump drainage systems. A Make-Up Air Unit (MUA) compensates for the extracted air volume, maintaining pressure balance and preventing cross-contamination between lab areas. All ducting systems are designed with solid ceiling panels, corrosion-resistant materials, and airflow balancing dampers. Air quality is maintained via high-efficiency filtration and thermal conditioning through HVAC units with dedicated zones.

The Metallurgical Laboratory, located within the process plant, includes ventilation plenums for sample crushing and grinding, with an outdoor baghouse dust collector ensuring dust control. Wet benches and

hotplate stations are paired with localized fume extraction. The laboratory's compressed air and process water circuits are designed to prevent aerosolized contamination and maintain laboratory hygiene. Additionally, the two (2) satellite laboratories, housed in insulated containers, are strategically deployed to allow localized sampling closer to active operations, with one located at the top of the Leach tank and another one after the cyanide destruction.

Integrated into the Analytical Laboratory, the Environmental Laboratory shares the main ventilation and fume scrubbing infrastructure. Its equipment layout considers airflow dynamics to protect delicate measurements (e.g., balances, photometers) from draft disturbances while providing effective evacuation of hazardous fumes.

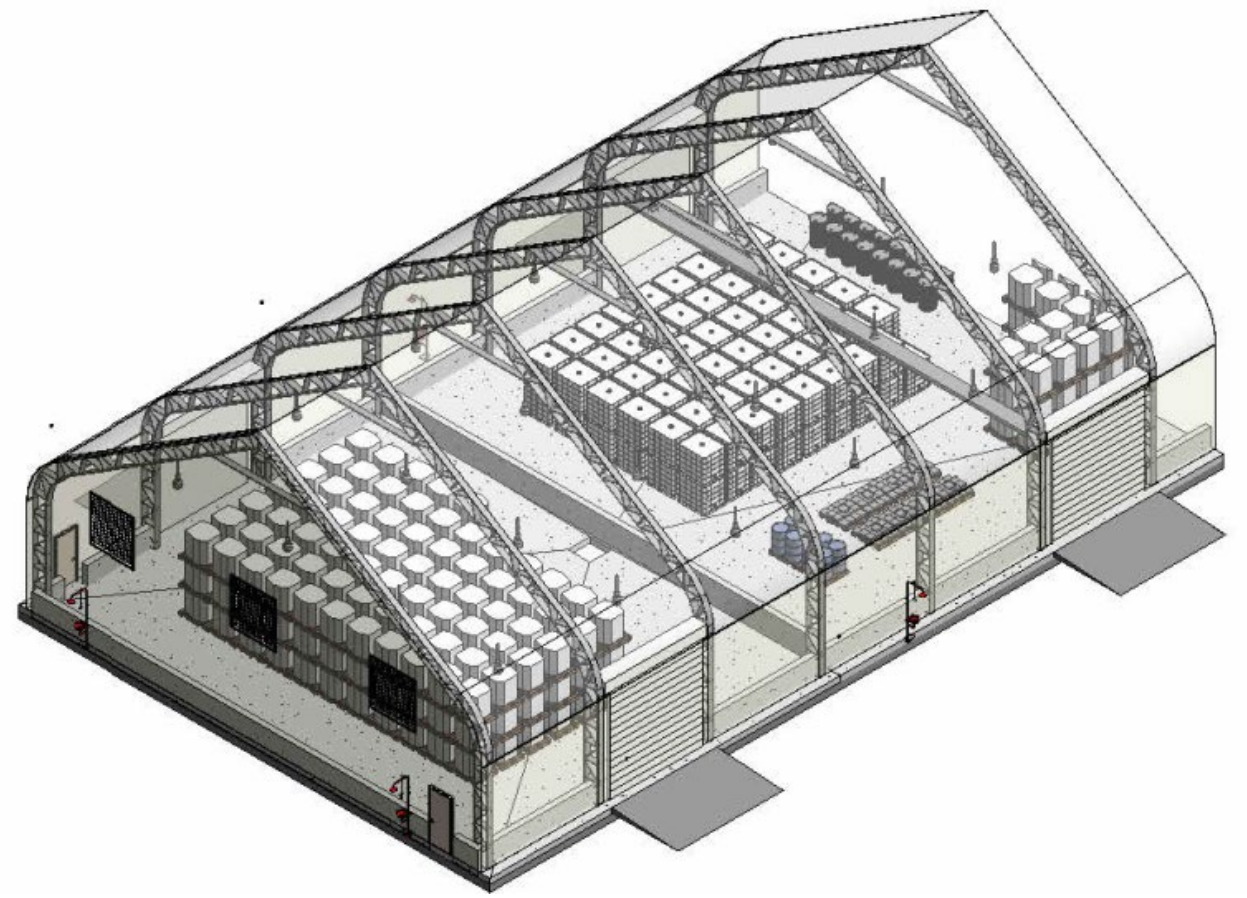
Figure 18.15: Assay Laboratory



18.5.3 Reagent Storage

All process plant reagents, except cyanide, SMBS and lime, will be stored in a warehouse made of a fabric top building on a concrete slab with approximately 640 m^2 of covered storage space. The reagents will be segregated by walls or curbs where required to avoid any potential cross-contamination. Liquid spillage in this area will be contained within the building. The floor plan is shown in Figure 18.16.

Figure 18.16: Reagent Storage Floor Plan



Separate areas will house the sodium cyanide and the combined SMBS and lime. Sodium cyanide will be shipped to site and stored in iso-containers. The delivery trucks will return with an empty iso-container for exchange. The isotainers will be stored on compacted soil.

SMBS and lime storage areas will be stored in warehouses made of a fabric top building on compacted soil.

The reagent storage areas will be within the fenced and gated area of the process plant.

18.6 Waste Storage Facility

18.6.1 Introduction

The Waste Storage Facility (WSF) was designed by TEC3 Geotecnia e Recursos Hídricos Ltda., a Brazilian consulting company specialized in Mining Geotechnics.

TEC3 evaluated locational alternatives for the waste storage facility; out of the three (3) possible locations, the WSF Option 2 (WSF 2) was chosen for its proximity to the pit and insertion in the property limits.

The total waste generated during the life of mine is approximately 234 million m³, considering the waste dry density after deposition of 2.0 t/m³. Initially, there is a predominance of saprolite waste generated during the pre-stripping stage of the open pit; however, this will diminish when bedrock is exposed, and the mine development continues. According to the mine waste schedule, the expected waste volume consists of 70% rock waste and 30% saprolite / soil waste; the soil will be disposed of within the dump, contained by rock disposal. Figure 18.17 and Figure 18.18 show the layout and a typical cross-section of the WSF 2.

Figure 18.17: WSF 2 Layout with Cross-Sections Used in the Stability Analysis

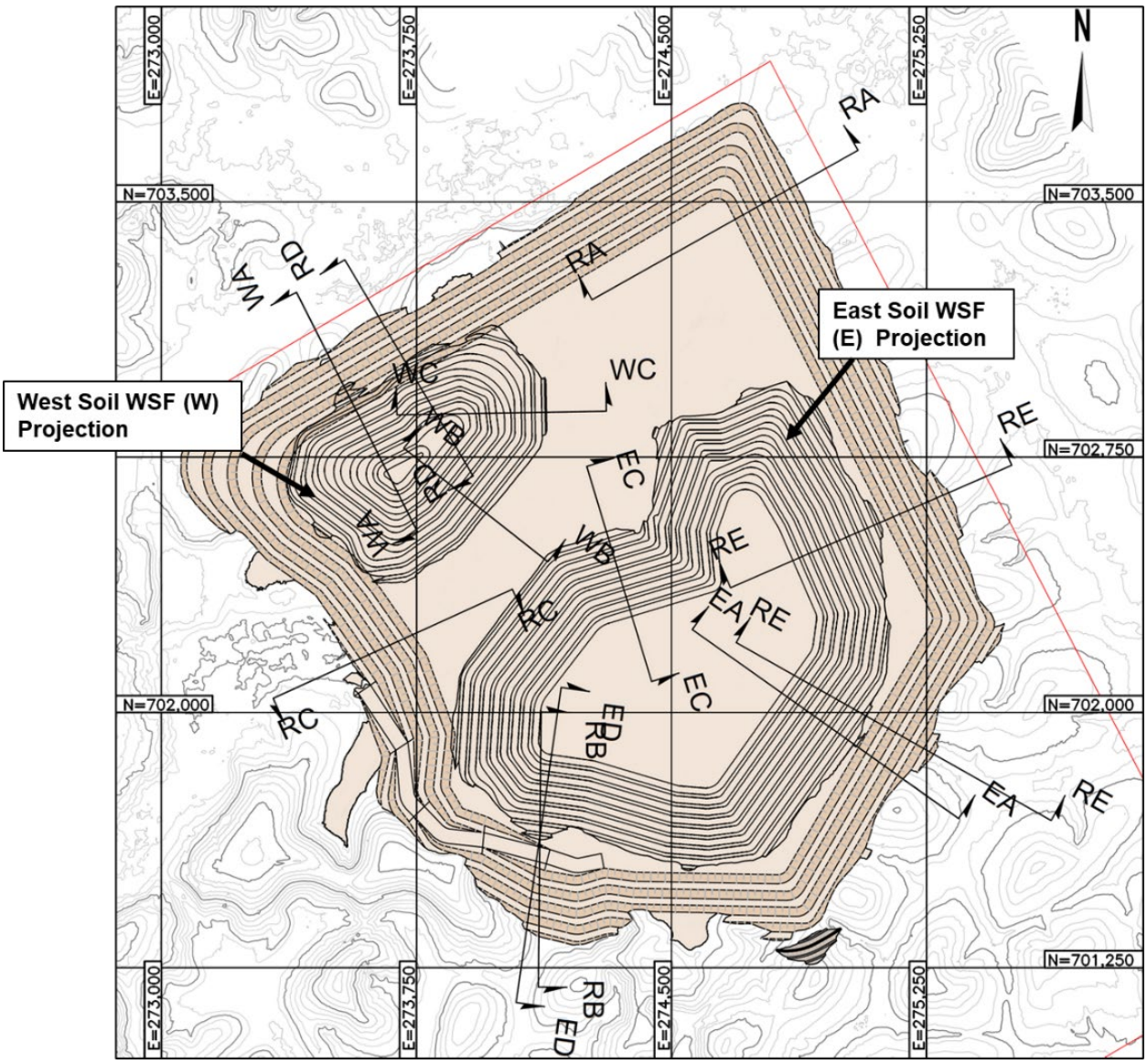
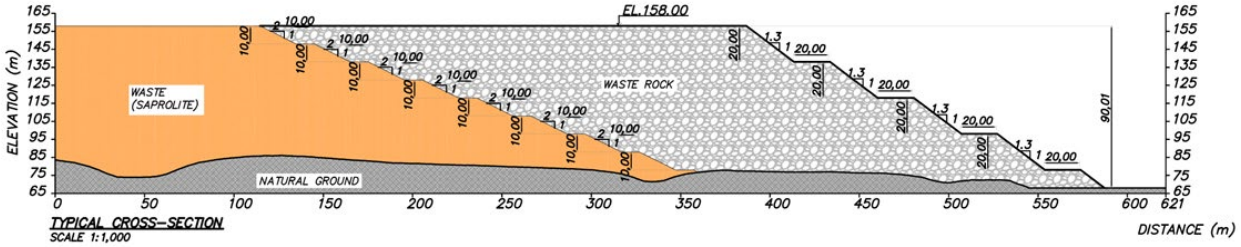


Figure 18.18: Typical Cross-Section of the WSF 2



18.6.2 Site Investigation

To characterize the WSF 2 foundation materials, a geological-geotechnical investigation campaign was carried out, including rotary drilling with SPT, *in situ* permeability tests, test pit investigations, and collecting disturbed and undisturbed soil samples (Shelby tubes and Block Samples) for laboratory testing. Laboratory tests were planned but not carried out; strength parameters were derived from correlation with the field investigation results (nSPT). It is recommended that some tests be run, primarily in the residual soils, which are the most representative foundation material.

Generally, the foundation presents at the top a 15- to 30-cm-thick superficial organic soil layer, with a silty-clay texture. Beneath it lies residual soil / laterized mature residual soil (MRS), with an average thickness of 3.40 m and an average nSPT of 12 blows (medium consistency). Beneath the mature residual soil, young residual soil (YRS) is found, with an average nSPT of 15 blows. The thickness varies from 9 m to 34.5 m. Below the young residual soil appears the saprolite (SAP), which differs from the residual soil by an average nSPT of 47 blows. The average thickness of this material is approximately 6 m. The bedrock below is typically granite / granitoid, more weathered in the contact zone, and fresh at a 37.5 m depth, on average.

Alluvial soils occur in lowland regions, with contributions from mining activities, with observed thicknesses of up to 6 m (can be thicker). The water table (WT) depth in the WSF 2 area ranges from surface outcrops in lowland areas to a depth of up to 22.05 m. The results of the permeability tests indicated hydraulic conductivity on the order of 10^{-5} cm/s or 10^{-7} m/s for both YRS and SAP.

The mine waste was not characterized; the finer fraction parameters were estimated based on the designer's experience. The geomechanical parameters for the waste rock were defined by known methodologies (Leps, 1970).

A summary of the strength parameters of the materials is shown in Table 18.2.

Table 18.2: WSF 2 Strength Parameters

Material	Γ (kN/m³)	c' (kPa)	φ' (°)	s_u/σ'_v
Waste Saprolite Material	17	5	30	-
Waste Rock Material	20	Leps (1970) – Low density		
Alluvium	15	0	20	0.20 (Minimum Shear Strength = 5 kPa)
MRSG (Mature Residual Soil – Granitoid Rock)	17	5	27	-
YRSG (Young Residual Soil – Granitoid Rock)	18	5	30	-
SAPG (Saprolite – Granitoid Rock)	19	5	35	-
Weathered Granitoid Rock	27	160	45	-

18.6.3 Design Elements

18.6.3.1 Geometry

The design was carried out according to industry best practices, such as the ones described in the Guidelines for Mine Waste Dump and Stockpile Design (HAWLEY and CUNNING, 2017) and is compliant with local regulations. The construction method will be ascending, with densification induced by equipment traffic during the waste material disposal and spreading stages.

Table 18.3 shows a summary of the geometrical characteristics of the WSF 2.

Table 18.3: Geometric Characteristics of the WSF 2

WSF Option 2 Summary		
Area (m^2)		3.59×10^6
Maximum Height (m)		96
Crest Elevation (m)		158
Volumetric Capacity (m^3)	Total	2.34×10^8
	Soil Sector	6.92×10^7
	Rock Sector	1.65×10^8
Bench Height (m)	Soil Sector	10
	Rock Sector	20

WSF Option 2 Summary		
Berm Width (m)	Soil Sector	10
	Rock Sector	20
Side Slope (H – Horizontal; V – Vertical)	Soil Sector	2.0H:1.0V
	Rock Sector	1.3H:1.0V
Overall Slope Angle (°)	Soil Sector	19
	Rock Sector	26

18.6.3.2 Internal Drainage

The design considers underdrains to collect and conduct infiltration water and groundwater. A network of primary and secondary drains is proposed (Figure 18.19). Three (3) cross-section types are considered, depending on whether the drain is to be constructed under rocky or saprolite waste. Where the drains will be built over the alluvium layer, it is planned to consolidate this layer with the disposal of waste rock over the alluvium, before the drain construction (Figure 18.20). Testing this consolidation before the operation is recommended if waste rock is available.

The design considers the proper transition layers to meet filtration and retention needs, and a minimum safety factor of 2.5 for flows. Drains include an additional height of 2.50 m to prevent longitudinal sectioning of the drain caused by differential settlement of the WSF 2. Electric piezometers will be placed to monitor the dissipation of pore pressure in the alluvial foundation areas during loading with waste material placement.

Figure 18.19: WSF 2 Internal Drainage System Layout

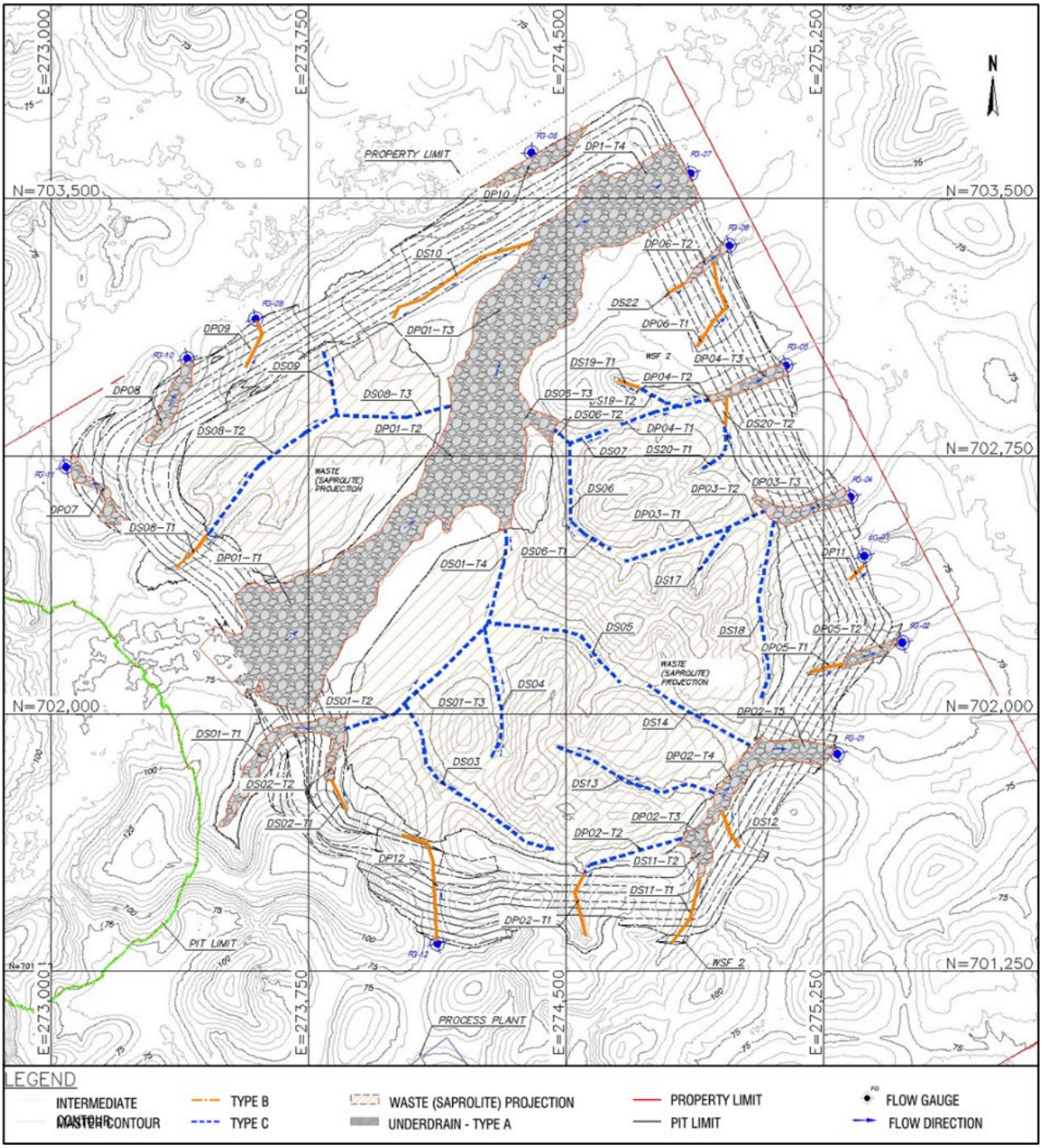
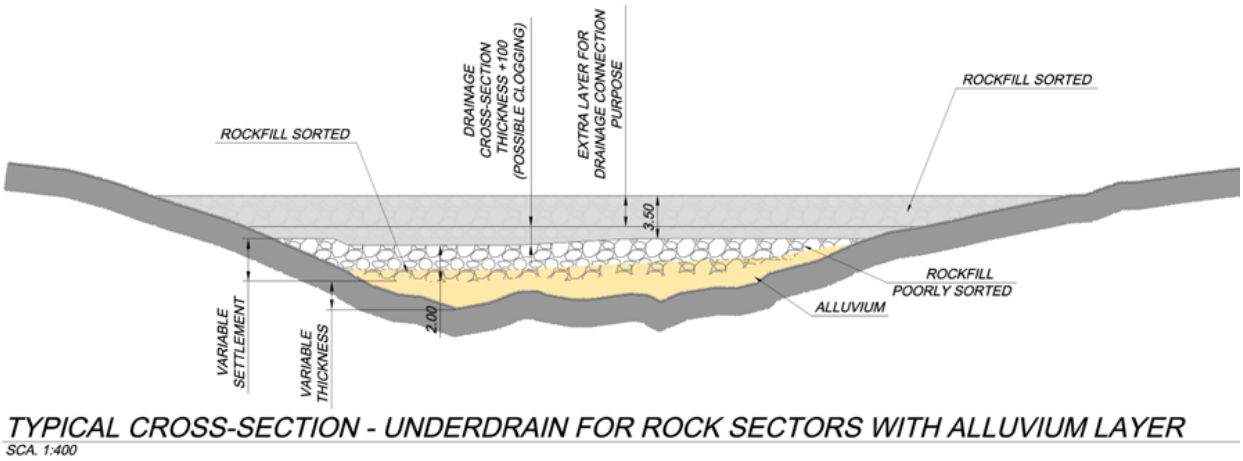
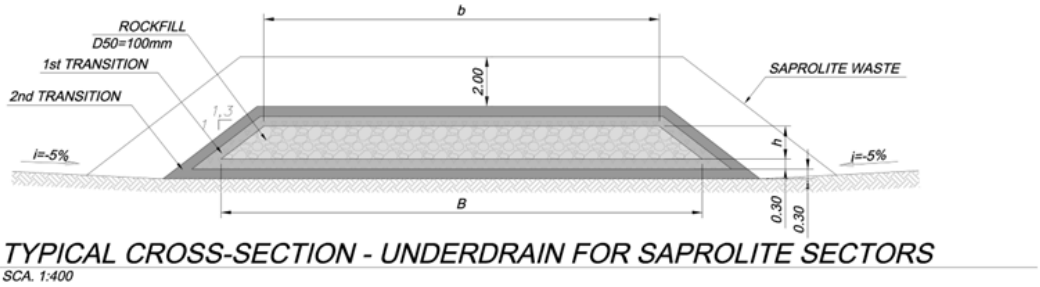
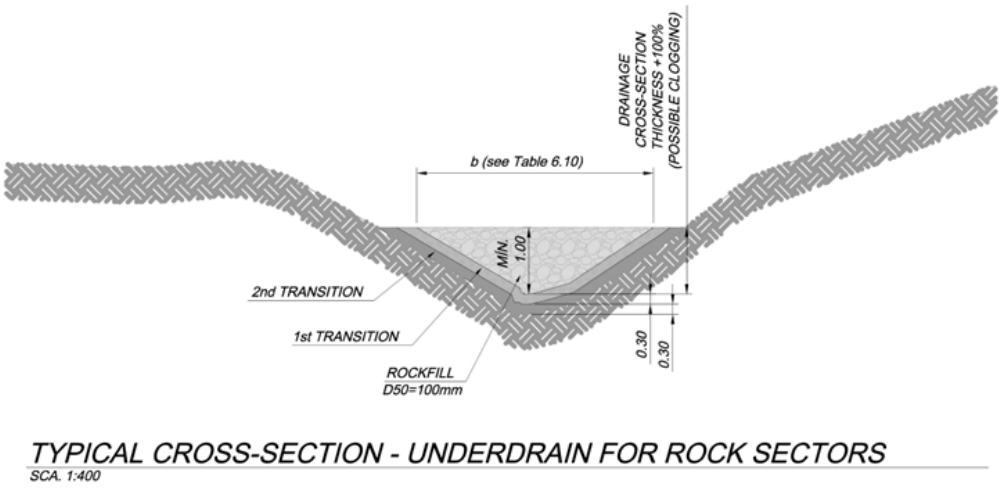


Figure 18.20: WSF 2 Typical Drain Cross-Sections





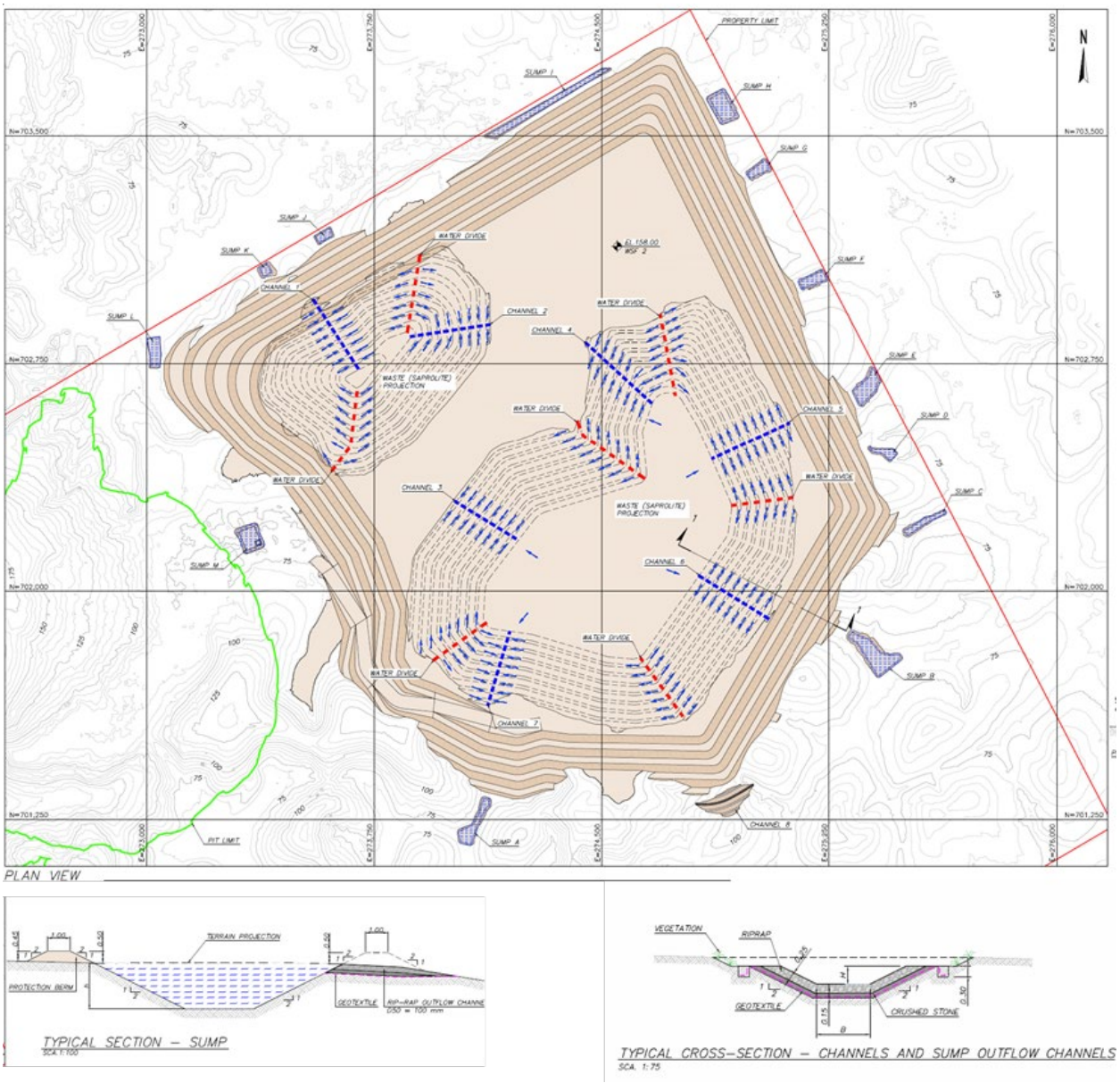


18.6.3.3 Superficial Drainage

The proposed drainage system consists of a diversion channel and a sump to pump runoff from the area between the open pit and the WSF 2. Significant surface flows are not expected in the rockfill disposal zones; therefore, a surface drainage system is unnecessary for the final layout.

The design incorporates rock-lined channels and dissipation basins within the saprolite disposal piles. To capture eroded sediments, twelve (12) sumps excavated in the natural terrain downstream of the pile are planned (Figure 18.21). The sumps will require cleaning at regular intervals (monthly to yearly).

Figure 18.21: WSF 2 Surface Drainage



18.6.3.4 Monitoring

Besides the electric piezometers in the alluvium layers, the proposed instrumentation includes the installation of piezometers in four (4) sections at the west soil structure and five (5) at the east. In addition, a weir should be placed downstream of each internal drainage system to monitor water levels and flow volumes throughout the drains. No deformation monitoring system is considered; however, it is suggested that INSAR be used if this technique is available in Guyana.

18.6.4 Stability Analysis

Slope stability analysis using the Limit Equilibrium Method was conducted on 12 cross-sections (Figure 18.17), encompassing both soil and rock sections. Strength parameters are shown in Table 18.2, and the analysis results are presented in Table 18.4. The acceptance criteria, related to the Factor of Safety (FoS), were as follows:

- Long-term global stability: Minimum Factor of Safety (FoS) of 1.40 - 1.50.
- Critical water level condition (only soil sector) - global stability: Minimum FoS of 1.30.
- Inter-ramp slope stability: Minimum FoS of 1.30.
- Pseudo-static analysis: Minimum FoS of 1.10 - 1.15 (PGA = 0.07 g, for a Return Period of 10,000 years, reduced to 0,035 g for a tolerable displacement of 1.0 m.

Table 18.4: Results of the Stability Analysis of the WSF 2

Cross-Section	Loading Condition	Minimum FoS Required	FoS Obtained
-	Inter-ramp slope stability	1.3	1.64
Cross-Section WA	Long-term global stability	1.5	1.74
	Critical water level global stability	1.3	1.38
	Pseudo-static analysis	1.1	1.56
Cross-Section WB	Long-term global stability	1.5	1.76
	Critical water level global stability	1.3	1.66
	Pseudo-static analysis	1.1	1.58
Cross-Section WC	Long-term global stability	1.5	1.71
	Critical water level global stability	1.3	1.55
	Pseudo-static analysis	1.1	1.10
Cross-Section EA	Long-term global stability	1.5	1.74

Cross-Section	Loading Condition	Minimum FoS Required	FoS Obtained
	Critical water level global stability	1.3	1.66
	Pseudo-static analysis	1.1	1.42
Cross-Section EB	Long-term global stability	1.5	1.70
	Critical water level global stability	1.3	1.55
	Pseudo-static analysis	1.1	1.27
Cross-Section EC	Long-term global stability	1.5	1.67
	Critical water level global stability	1.3	1.56
	Pseudo-static analysis	1.1	1.17
Cross-Section ED	Long-term global stability	1.5	1,75
	Critical water level global stability	1.3	1.60
	Pseudo-static analysis	1.1	1.55
-	Inter-ramp slope stability	1.3	1.37
Cross-Section RA	Long-term global stability	1.5	1.69
	Pseudo-static analysis	1.1	1.10
Cross-Section RB	Long-term global stability	1.5	1.79
	Pseudo-static analysis	1.1	1.29
Cross-Section RC	Long-term global stability	1.5	1.67
	Pseudo-static analysis	1.1	1.12
Cross-Section RD	Long-term global stability	1.5	1.73
	Pseudo-static analysis	1.1	1.30
Cross-Section RE	Long-term global stability	1.5	1.62
	Pseudo-static analysis	1.1	1.46

18.7 Tailings Storage Facility

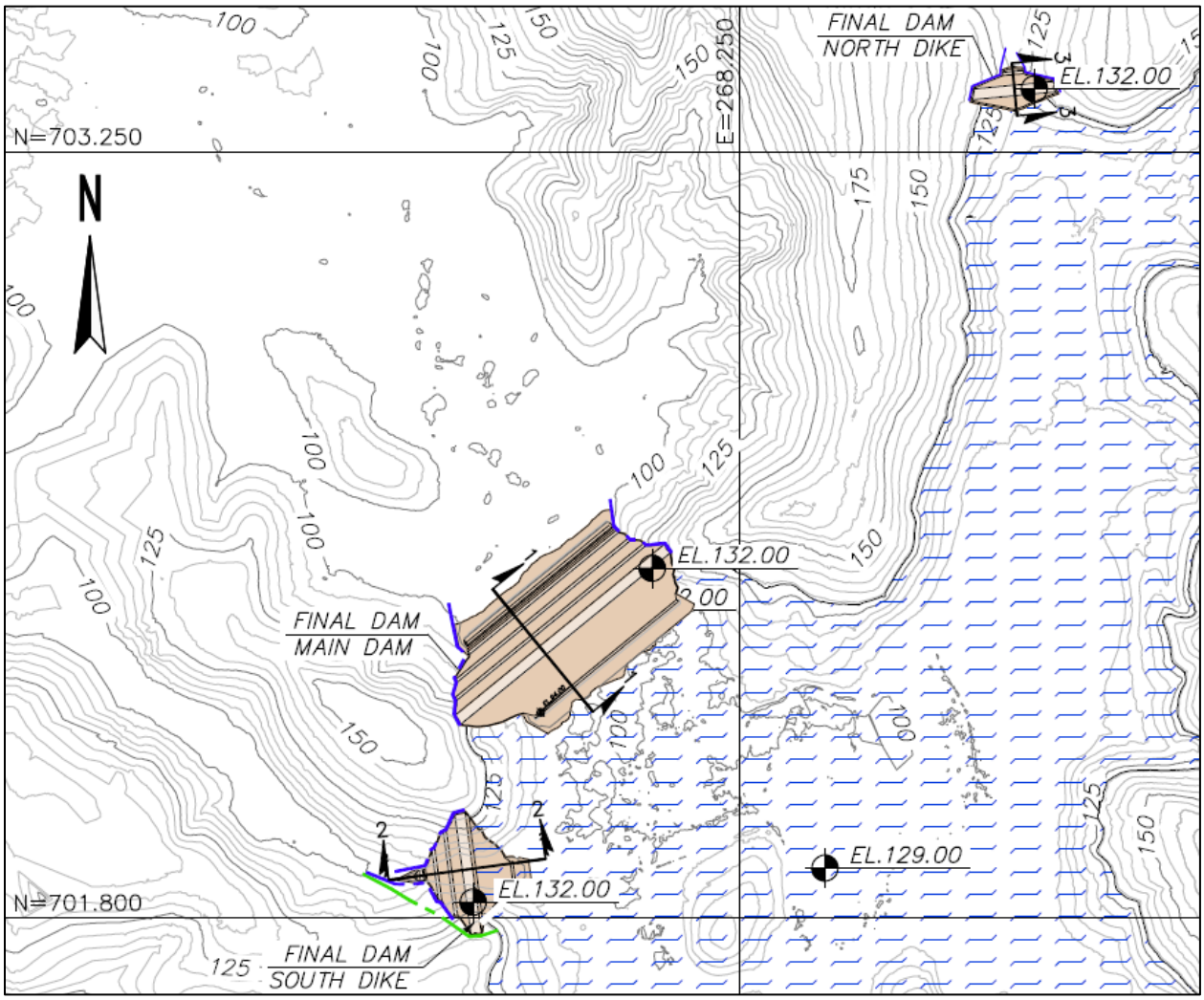
18.7.1 Introduction

The design of the Tailings Storage Facility (TSF) was carried out by TEC3 Geotecnia e Recursos Hídricos Ltda., a Brazilian consulting company specialized in Mining Geotechnics.

TEC3 evaluated locational alternatives for the tailings' storage facility. TSF Option 5 (TSF 5) was selected for its optimal ratio of reservoir to embankment volumes, foundation treatment, alignment with property limits, constructability, and financial assessment.

The total tailings generated during the life of the mine are approximately 58.4 million m^3. To accommodate this volume, the TSF 5, in its final stage, will consist of a main dam and two (2) saddle dikes (Figure 18.22).

Figure 18.22: TSF 5 Layout with Critical Stability Analysis Cross-Section Locations

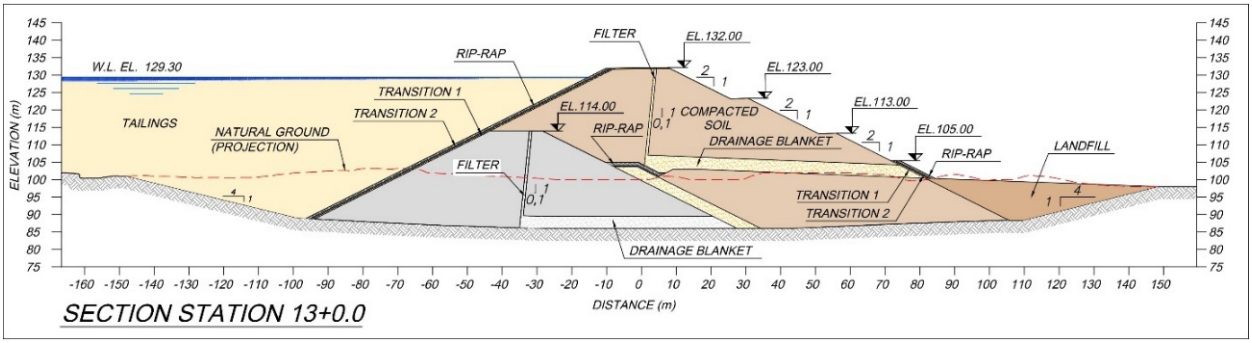


The starter dam will have a crest elevation of 114 m, while the final dam will have a crest elevation of 132 m.

The dam will be built in several stages, downstream raising, considering homogeneous compacted earth fill from adjacent borrow areas, with a chimney and blanket drainage system. Figure 18.23 shows a typical cross-section. The total height of the dam is 42 m, and the dikes will be 12 m high (North dike) and 32 m

high (South dike). As design assumptions, the tailings are considered inert and non-hazardous, following criteria from CFR – Title 40 – Protection of Environmental – Part 260-265 – Hazardous Waste Management. Geochemical tests are running and must be completed before dam construction.

Figure 18.23: Typical Cross-Section of the TSF (Main Dam)



18.7.2 Site Investigation

A geological-geotechnical investigation campaign was carried out to characterize the TSF foundation materials and borrow areas. This investigation campaign included the TSF option 5 and the other TSF alternatives' locations to support the alternative decision.

The campaigns involved rotary drilling with SPT, in situ permeability tests, test pit investigations, and CPTu (piezocone penetration tests). The program also included collecting disturbed and undisturbed soil samples (Shelby tubes and Block Samples) for laboratory testing. Although laboratory tests were planned, they were not completed before the issuance of the design report. The testing program includes natural moisture content (wnat), grain-specific mass (GS), grain size distribution through sedimentation, Atterberg limits, compaction using Standard Proctor energy, dispersibility (crumb test and pinhole tests), triaxial compression test (CIUsat), variable and constant head permeability tests, one-dimensional consolidation test, void ratio, bender element, and cyclic direct simple shear (CDSS). Finishing the test program and reevaluating the results before the dam construction begins is recommended.

The TSF foundation is composed of the Barama Group, which primarily consists of mafic volcanic (MAF), volcanoclastic (VOL), and some granitoid dikes (GRA). Overall, the foundation has an average top layer of 8 m of alluvium / tailings, with four (4) nSPT blow counts in the valley areas, or up to 3 m of colluvium with as many as 14 blow counts in the abutment areas. Beneath these layers lies the in-situ material, which, from top to base, comprises mature residual soils (MRS), young residual soil (YRS), saprolite (SAP), and rock (RO). The thickness and nSPT blow counts vary depending on the original lithological type.

Alluvial soils and soils disturbed by artisanal mining activities occur in lowland areas. The observed thickness reaches up to 15 m near the valley's centerline, decreasing to 6 m near the abutments. This alluvial and tailings material must be removed for dam construction on a competent foundation. The estimated excavation volume in alluvium, tailings, and colluvium is approximately 582,000 m³ for the starter dam, with an additional 341,000 m³ required for the final dam. In the abutments, it is estimated that about 1 m of colluvium, loose material, topsoil, and roots will be removed; the colluvium thickness may increase based on new investigation data.

A summary of the strength parameters of the materials is shown in Table 18.5.

Table 18.5: Strength Parameters

Material	Specific Weight (kN/m³)	Drained Parameters		Undrained Parameters	
		Effective Cohesion (kPa)	Effective Friction Angle (°)	Normal Stress (kPa)	Shear Stress (kPa)
Embankment	19	25	32	0.0	12.0
				182.5	162.4
				600.0	262.3
Young Residual Soil – Mafic Volcanic Rock (YRSM)	19	9	32	0.0	9.0
				245.5	162.4
				800.0	362.6
Young Residual Soil – Granitoid Rock (YRSG)	20	17	32	0.0	17.0
				280.8	192.4
				800.0	423.8
Young Residual Soil – Volcaniclastic Rock (YRSV)	20	22	25	0.0	17.0
				441.7	228.0
				1200.0	446.3
Alluvium	15	0	27	0.0	7.0
				43.3	7.0
				52.6	12.2
				108.6	43.6
Colluvium	17	5	27	s_u/σ'_v = 0.21	-
Filter / Drainage Blanket	20	0	35	-	-
Rip-Rap	20	0	40	-	-

Material	Specific Weight (kN/m³)	Drained Parameters		Undrained Parameters	
		Effective Cohesion (kPa)	Effective Friction Angle (°)	Normal Stress (kPa)	Shear Stress (kPa)
Mafic Volcanic Rock	20	150	35	-	-
Volcaniclastic Rock	20	150	35	-	-
Granitoid Rock	20	150	35	-	-

The potential borrow areas were investigated using auger drilling and sampling for laboratory tests (still not concluded). The regions selected for investigation were those near the TSF alternatives. The pit overburden material was also analyzed, as this area must be excavated; however, due to the hauling distances, nearby borrow pits will be utilized.

The typical gain size distribution has 50% fines content in the borrow areas near the TSF, while the pit overburden material consists of silty, clayey, and sandy components. Generally, both locations serve as suitable sources for the fill. An anticipated problem for construction is that the natural moisture content is, on average, about 6% higher than the optimum level. Dry areas and rehandling of the material will be necessary.

18.7.3 Design Elements

18.7.3.1 Geometry

The design was conducted according to criteria set by international guidelines, such as the Canadian Dam Association technical bulletins, and complies with the Guyana Geology and Mines Commission Codes of Practice. The construction will take place in several stages, including downstream raising. The starter dam was designed to accommodate the first three (3) years of operation. Figure 18.23 shows a typical cross-section, while Table 18.6 summarizes each structure for the starter and final dam phases, respectively; the main design details are presented in sequence.

Table 18.6: Summary of Main Dam and Saddle Dikes at Starter and Final Dam Phases

Structure	Starter Dam Phase		
Main Dam	Crest	Elevation (m)	114
		Width (m)	15
		Length (m)	529
	Compacted Soil Volume (m³)		600,382

Structure	Starter Dam Phase		
	Slope Geometry	Upstream	2H:1V with 1 bench every 10 m (6 m berm width)
		Downstream	2H:1V single slope
South Saddle Dike	Crest	Elevation (m)	114
		Width (m)	15
		Length (m)	117
	Compacted Soil Volume (m³)		36,593
	Slope Geometry	Upstream	2H:1V with one bench every 10 m (6 m berm width)
		Downstream	2H:1V single slope

Structure	Final Dam Phase		
Main Dam	Crest	Elevation (m)	132
		Width (m)	15
		Length (m)	514
	Compacted Soil Volume (m³)		1.140,392
	Slope Geometry	Upstream	2H:1V with one bench every 10 m (6 m berm width)
		Downstream	2H:1V single slope
South Saddle Dike	Crest	Elevation (m)	132
		Width (m)	15
		Length (m)	231
	Compacted Soil Volume (m³)		193,168
	Slope Geometry	Upstream	2H:1V with one bench every 10 m (6 m berm width)
		Downstream	2H:1V single slope
North Saddle Dike	Crest	Elevation (m)	132
		Width (m)	15
		Length (m)	159
	Compacted Soil Volume (m³)		26,627
	Slope Geometry	Upstream	2H:1V with one bench every 10 m (6 m berm width)
		Downstream	2H:1V single slope

The starter main dam will feature a 15 m wide crest at an elevation of 114.00 m, and its construction will utilize 600.382 m³ of compacted soil. The upstream slope angle is 2H:1V with no berms, while the downstream slope has the same angle but includes intermediate berms that are 6.0 metres wide at every 10-metre bench. The internal drainage system includes a vertical drain that is 0.70 m thick, along with a horizontal drainage blanket consisting of a 3.50 m thick sand layer compacted to a minimum relative density of 70%. Only the south dike is necessary in the initial stage (Starter phase), and this dike shares the same geometric characteristics as the main dam.

The final dam will be raised downstream from the starter dam to an elevation of 132 m, maintaining the same geometry. The internal drainage system will connect with the one from the starter dam; a 3.50 m thick sand blanket layer is designed for the internal drainage system, positioned at an approximate elevation of 100 m. The south dike will also be raised to an elevation of 132 m, and the horizontal blanket drain will connect with the existing one. Additionally, a new chimney drain will be constructed up to an elevation of 129.50 m. The northern saddle dike is likewise a homogeneous, compacted earth fill dam, with a crest at an elevation of 132 m. Its geometry and internal drainage are the same as those of the initial stage of the south dike.

18.7.3.2 Internal Drainage

The design of the internal drainage was based on a flow analysis, considering the final configuration of the dams and the maximum reservoir water level. Hydraulic conductivity coefficient (k) values were determined from permeability tests or estimated based on literature data and the designer's experience in similar projects. The sizing of the drains adhered to Darcy's equation, and a major factor of 10 was applied to determine the final geometry. Transition layers are included in the design to retain particle migration and prevent piping. However, the report did not include details and dimensioning for the transition layers, which must be shown for evaluation and construction purposes.

18.7.4 Hydrological and Hydraulic Studies

The hydrological studies and details of the hydraulic dimensioning are presented in this section.

18.7.4.1 Design Storms

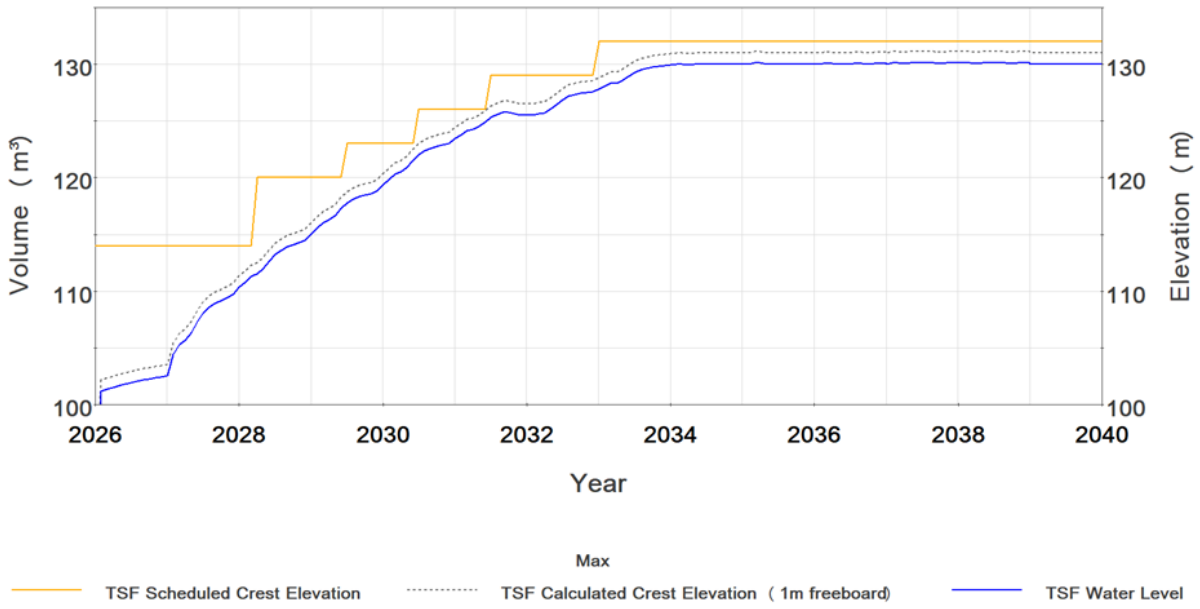
The design incorporates the use of the PMP (probable Maximum Precipitation) and a remaining freeboard of 1.0 m as the design criteria. The Weather Research and Forecast (WRF) model estimated the PMP using data from 2015 to 2023 in the simulation. The report presents the results for several durations, with the seven-day duration indicating a precipitation of 691 mm.

Design floods of the spillway and diversion structures were determined using the Soil Conservation Service (SCS) methodology and the Clark unit Hydrograph with the HEC-HMS (Hydrologic Modelling System) model. Hydrological simulations indicated that the peak flows of the design floods are 27.23 m³/s, considering the PMF event, over a seven-day duration, with the maximum water level at an elevation of 130.99 m and a resulting freeboard of 1.0 m.

18.7.4.2 Water Balance

The water released from tailings and the reservoir's annual water balance will be used at the plant; it is estimated that 80% of the water consumed at the plant will be recirculated from the dam reservoir. A monthly water balance for the TSF, using Goldsim software (Figure 18.24), shows the maximum values of calculated water level in the reservoir, the required elevation for the dam crest, and the planned dam crest elevation. In this situation, there will be no outflow through the spillway.

Figure 18.24: TSF 5 Water Balance



18.7.4.3 Spillway

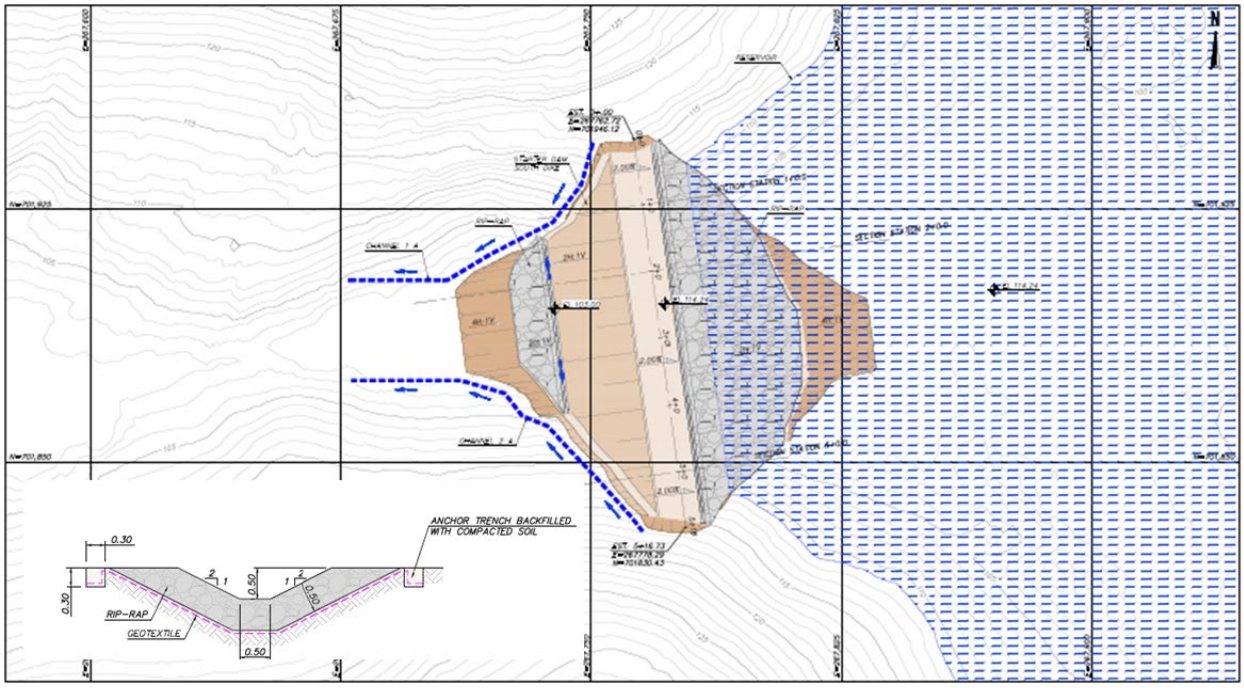
At the end of the mine life, a passive spillway system including a rock-lined (D50 = 800 mm) approach channel and a descent channel with a trapezoidal cross-section will be installed. Additionally, it will feature a stilling basin at its downstream end, discharging inflows into the natural stream. In its final configuration, it will be built at the left abutment of the south dike.

The hydraulic sizing of the spillway and channels used the Manning equation under the assumption of a uniform flow regime. The approach channel has a 6.0 m wide base and a height of 2.7 m; the rapid is 12.0 m wide at the base and 1.0 m high. The stilling base is also 12 m wide at the base and has a height of 1.5 m.

18.7.4.4 Superficial Drainage

The planned drainage system comprises peripheral rock-lined channels designed for a 100-year return period. A total of six (6) channels are scheduled for the main dam and the two (2) dikes. Figure 18.25 shows an example of this system layout for the south dike and a typical cross section of the channels 1A and 2A. Typical dimensions are 0.5 m x 0.5 m, base width *x* height.

Figure 18.25: TSF 5 Surface Drainage Example – South Dike



18.7.4.5 River Diversion for Construction

Two (2) cofferdams are planned to facilitate dam construction upstream of the main dam and the south dike. The cofferdam upstream of the main dam will feature an overflow channel excavated in the ground with a trapezoidal section, a 3.0 m base, slopes of 1.0H:2V, and a longitudinal slope of 0.2%, directing water through the south dike valley. The second cofferdam, which will allow for the construction of the south dike, will function using a 700 m³/h pumping system to accommodate the average reservoir inflow.

18.7.5 Monitoring

The main objective of the instrumentation plan is to monitor post-construction displacement of the embankment, including settlements at the downstream slope and consequent loss of effective freeboard and the effectiveness of the internal drainage. The instrumentation planned for the TSF includes Casagrande and vibrating wire piezometers, monitoring wells, V-notch weirs to monitor pressure and water flows, and prisms for displacement monitoring, installed in three (3) cross-sections in each structure.

No subsurface displacement monitoring is considered; evaluating the need for inclinometers or subsurface vertical extensometers is recommended.

18.7.6 Stability Analysis

Using the Limit Equilibrium Method, slope stability analysis was done in three (3) cross-sections (Figure 18.17), covering the main dam and the dike. The acceptance criteria, based on CDA guidelines, were as follows:

- Long-term global stability: Minimum Factor of Safety (FoS) of 1.50.
- End of construction (upstream and downstream slopes): Minimum FoS of 1.30.
- Pseudo-static analysis: Minimum FoS of 1.10 (PGA = 0.07 g, for a Return Period of 10,000 years reduced to 0,035 g for a tolerable displacement of 1.0 m.

Table 18.7 and Table 18.8 show the stability analysis results for the starter and final dam configurations.

Table 18.7: Results of Stability Analysis of the TSF 5 – Starter Dam

Starter Dam Configuration				Main Dam	South Dike
Section	**Loading Condition**	**Slope**	**Minimum FoS Required**	**FoS Obtained**	**FoS Obtained**
Right Abutment	End of construction before reservoir filling - Upstream	Upstream	1.30	1.37	2.42
	End of construction before reservoir filling - Downstream	Downstream	1.30	2.64	2.44
	Long-term stability	Downstream	1.50	2.86	2.44
	Pseudo-static	Downstream	1.10	2.73	2.00
Central	End of construction before reservoir filling - Upstream	Upstream	1.30	1.79	1.82

Starter Dam Configuration				Main Dam	South Dike
Section	Loading Condition	Slope	Minimum FoS Required	FoS Obtained	FoS Obtained
	End of construction before reservoir filling - Downstream	Downstream	1.30	2.36	2.11
	Long-term stability	Downstream	1.50	2.41	1.98
	Pseudo-static	Downstream	1.10	1.90	1.69
Left Abutment	End of construction before reservoir filling - Upstream	Upstream	1.30	2.46	2.04
	End of construction before reservoir filling - Downstream	Downstream	1.30	2.24	2.41
	Long-term stability	Downstream	1.50	2.08	2.89
	Pseudo-static	Downstream	1.10	1.91	3.83

Table 18.8: Results of Stability Analysis of the TSF 5 – Final Dam

Final Dam Configuration				Main Dam	South Dike	North Dike
Section	Loading Condition	Slope	Minimum FoS Required	FoS Obtained	FoS Obtained	FoS Obtained
Right Abutment	End of construction before reservoir filling - Upstream	Upstream	1.30	1.57	1.85	2.22
	End of construction before reservoir filling - Downstream	Downstream	1.30	1.45	2.06	2.86
	Long-term stability	Downstream	1.50	1.91	1.76	2.72
	Pseudo-static	Downstream	1.10	1.70	1.52	2.13
Central	End of construction before reservoir filling - Upstream	Upstream	1.30	1.96	1.95	2.00
	End of construction before reservoir filling - Downstream	Downstream	1.30	1.73	1.66	2.21
	Long-term stability	Downstream	1.50	1.62	1.59	2.05
	Pseudo-static	Downstream	1.10	1.31	1.35	1.60
Left Abutment	End of construction before reservoir filling - Upstream	Upstream	1.30	1.79	1.83	2.16

Final Dam Configuration				Main Dam	South Dike	North Dike
Section	**Loading Condition**	**Slope**	**Minimum FoS Required**	**FoS Obtained**	**FoS Obtained**	**FoS Obtained**
	End of construction before reservoir filling - Downstream	Downstream	1.30	1.98	2.01	2.94
	Long-term stability	Downstream	1.50	1.97	1.76	2.75
	Pseudo-static	Downstream	1.10	1.65	1.52	2.18

An assessment for liquefaction potential was done using correlation with NSPT values, on the foundation of the dams. FSL (Liquefaction Safety Factor) values were satisfactory in all the analyzed sections.

18.8 Water Management

18.8.1 Industrial / Fire Water

A pumping station will be installed in a non-contact stream within the boundary of the project to supply raw water. Raw water will be treated at the water treatment plant to remove suspended solids and perform pH adjustment and disinfection. The identified treatment is a Greensand filtration, activated carbon filtration and chlorine disinfection. A dedicated tank of 400 m^3 will supply all domestic uses such as bathrooms, showers, safety showers, laundry, laboratory, and industrial uses, such as fire water, service water at the process plant and mining infrastructures.

The industrial and process water required to perform the initial commissioning and startup of the process plant will be pumped from the water accumulated in the TSF impoundment.

A fire protection pumping station will include a diesel pump, jockey pump, piping and instrumentation to feed a buried network connecting to the site hydrants, process plant and mine maintenance facility. Within the processing plant, conveyors and a hydraulic unit will be equipped with a sprinkler system connected to the fire hydrant loop. A fire water pump skip will be adjacent to the fire water tank located at the camp. Manual fire extinguishers will also be installed in all the buildings. A secondary tank of 400 m^3 will be dedicated to fire water after the industrial treatment at the camp complex.

18.8.2 Potable Water

In the early stages of the project, potable water for all facilities on site will be provided by bottles and jugs. Domestic water will be further treated by an osmosis plant to generate potable water for the kitchen and personal consumption. A specific room has been reserved for this future plant in the kitchen.

18.8.3 Sewage Treatment and Oil-Water Separation

A sewage treatment plant is planned to treat sewage from the plant and the camp. Sewage water will be handled by standard septic tank collection systems and treated using natural breakdown bioreactors prior to discharge. Sewage will be treated, separated, and the liquid discharged. Sludge from the sewage treatment plant will be disposed of at an approved location.

Oil-water separation systems with be located adjacent to the likely locations of contamination. Water will be collected from the Maintenance Facility and the diesel fueling station to be redirected to an oil-water separator prior to discharging clean water to the environment. At the wash bay, an oil skimmer will capture any oil before water is reused to clean equipment.

18.8.4 Contact Water Collection Pond

The contact water collection pond is an event pond. It is a critical component of the site's water and environmental management system. It is designed as a multi-purpose containment structure to:

- Capture and settle sediment-laden surface runoff, particularly during heavy rainfall events or site disturbances.
- Serve as a contingency buffer in the event of pipeline failure from either the tailings or reclaim water return systems.
- Temporarily store open pit dewatering effluent prior to treatment or redirection to the environment.

The pond sizing and design of 150,000 m^3 have been guided by site-specific hydrological studies, climate data, geotechnical assessments, and process flow scenarios. Key design inputs include:

- A 24-hour, 100-year return period rainfall event to define peak runoff inflow.
- Assumed instantaneous rupture of tailings or reclaim pipeline at maximum operating flow rate.
- Expected daily dewatering volumes from pit dewatering operations, based on hydrogeological models.

18.8.5 Effluent Treatment Plant

The current water management strategy assumes that natural dilution from precipitation, along with degradation and volatilization processes, will reduce cyanide and ammonia concentrations within the Tailings Storage Facility (TSF) to levels that comply with regulatory discharge criteria. To validate this assumption, all decant water could be fully contained within the TSF during the first year of operations. This period will allow for comprehensive water quality monitoring and assessment to determine if additional treatment measures are necessary.

Should water quality monitoring indicate that effluent treatment is required, the TSF dam would need to be raised during Year 1 to accommodate the design, fabrication, and installation of an effluent treatment plant. This contingency measure ensures that water management strategies remain adaptive to site-specific conditions, regulatory requirements, and environmental best practices.

18.8.6 West Range Diversion Channel

The development of the open pits and underground mine at the project site will require the construction of a water diversion channel, known as the West Range Diversion Channel, to safeguard operations from hydrological risks. This critical infrastructure will be implemented in two (2) phases: an initial temporary diversion to allow for early mining activities, and a final permanent diversion to support full pit development.

18.8.6.1 Phase 1 – Initial Diversion Channel

A temporary diversion channel will be constructed during the early site development phase, prior to the commencement of mining activities within the main open pit. This is essential because a significant creek currently crosses the northern section of the planned pit area.

The initial diversion will be constructed along a temporary mining bench, located between the northern pit and southern pit limits and adjacent surface infrastructure. This temporary channel will reroute the creek safely away from the active mining zone, while maintaining sufficient capacity to handle peak flows based on local hydrological studies. The design will incorporate erosion and sediment control measures, such as:

- Riprap and geotextile reinforcement.
- Settling ponds or sediment traps.
- Perimeter ditches to manage runoff.

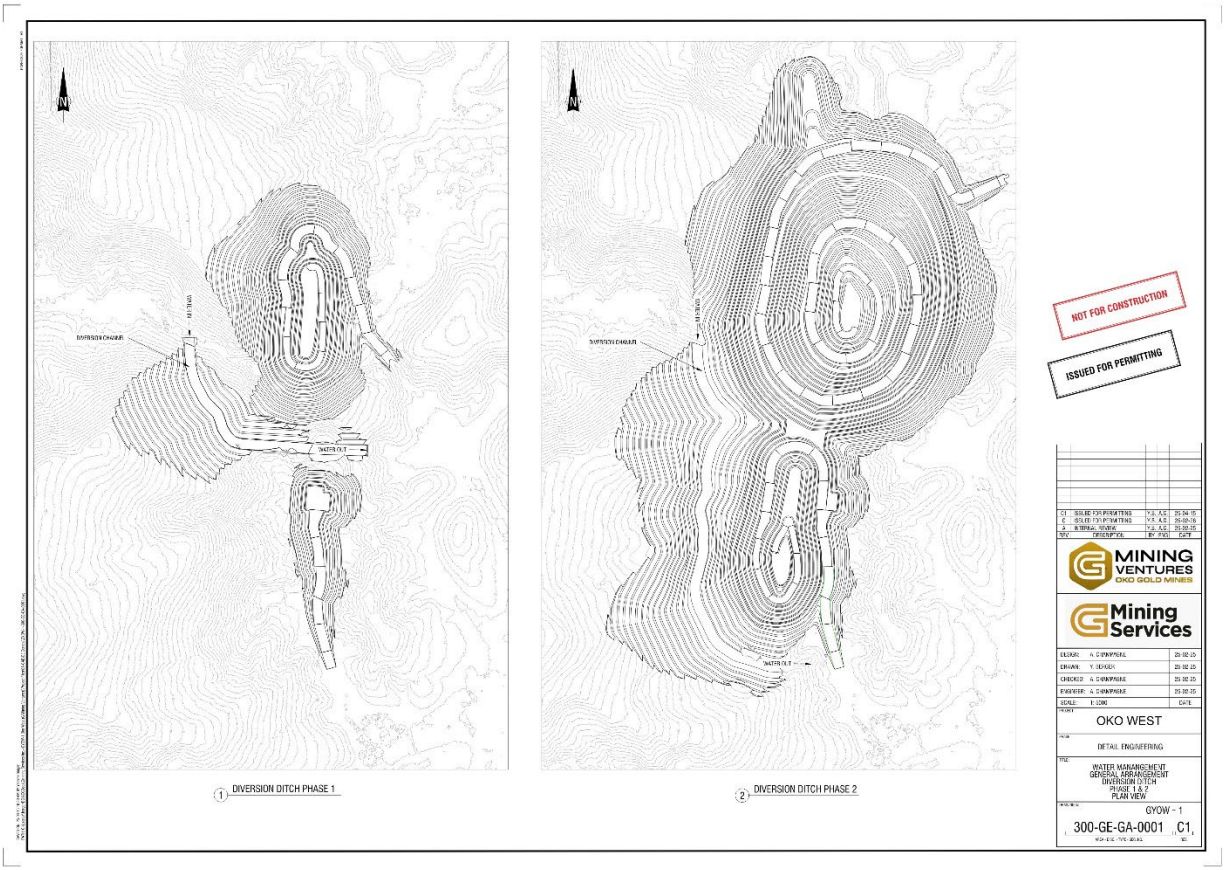
The goal is to ensure safe and controlled water management during the early years of pit development.

18.8.6.2 Phase 2 – Final Diversion Channel

As the pit expands toward its ultimate pit limits, a permanent water diversion will be constructed. This final diversion channel is planned as sustaining capital and will be designed to provide a long-term, environmentally compliant solution for water management throughout the mine life.

Detailed hydrological and geotechnical studies—including flood modelling and slope stability analysis—will be conducted to support the final design. The alignment of the channels has been carefully selected to minimize environmental impact, reduce construction complexity, and ensure the protection of the infrastructure. Figure 18.26 presents the conceptual location of both channels.

Figure 18.26: Diversion Channels



18.9 Fuel Storage and Distribution

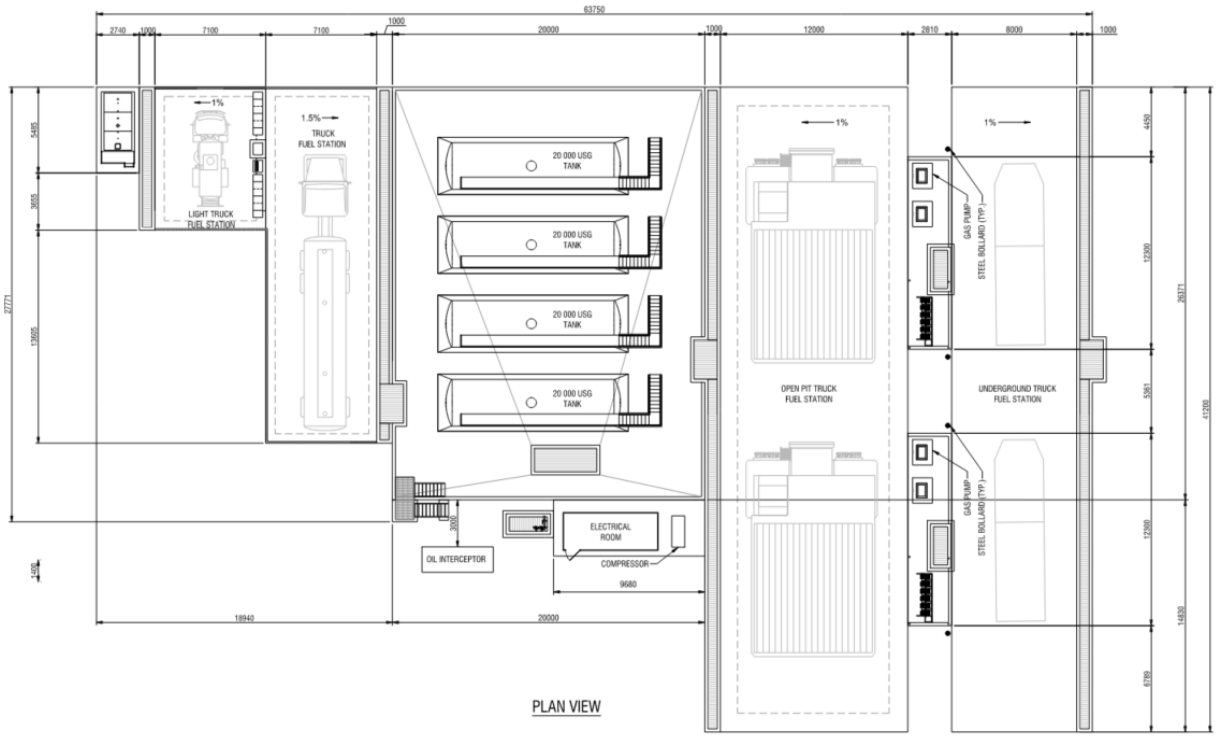
The main fuel storage facility will service the mining and site mobile equipment fleet with four (4) tanks of 220,000 litres of diesel fuel and two (2) tanks of 20,000 litres of diesel fuel for approximately ten (10) days of mobile equipment operations, and a 5,000-litre tank of gasoline for light vehicles. A specific service line

will be used to fill the open pit and underground mining fleet, and another line will service the light vehicle fleet and the tankers (Figure 18.21).

Fuel will also be stored at the power plant; one (1) tank of Heavy Fuel Oil (HFO) dedicated to the permanent gensets of 3,500,000 litres will provide 16 days of autonomy. In the same area, a tank of 700,000 litres of diesel will ensure the supply to the emergency gensets.

Additional fuel storage will be available at the barge landing, where the reception of the supplier's fuel barge will be made and subsequently transported on land with tankers. Two (2) tanks of 800,000 litres will store 16 days of mobile equipment operations. For HFO, two (2) tanks of 1,100,000 litres will provide approximately nine (9) days of autonomy.

Figure 18.27: Fuel Storage



18.10 Power Supply and Distribution

Plant site activities will require an average of 46 MW at its full operation, including the process plant, underground mining, open pit mine and BOP infrastructure. Full power consumption of the plant was benchmarked against similar projects with open pit and underground mining feeding a common processing plant.

The base case scenario for the Project considers the installation of a dedicated HFO-fired power plant. It is anticipated that the power plant will comprise six (6) 9.3 MWe engine generating sets, totalling 55.8 MWe installed capacity and 46.5 MWe running capacity, assuming that one (1) of the generators would be on standby. One (1) additional genset is planned in sustaining capital, assuming that one (1) genset would be out of service for major maintenance.

The power generation infrastructures will be located close to the processing plant.

The power distribution for the Project will be configured at 13.8 kV, 60 Hertz (Hz) and will consist of substations, step-down transformers, switchgears, motor control centres, powerlines and cableways. The electrical loads of the project will be supplied by voltages of 4.16 kV, 480 V or 208/120 V.

The processing plant will have various satellite electrical rooms. These electrical rooms will serve the following areas:

- Crushing area.
- Grinding / gravity.
- Gold room.
- Cyanide detoxification / plant services.
- Pre-leach / leach / CIP / acid wash / elution / carbon regeneration.
- Ore handling.

Power lines will be used to distribute power to other infrastructure, such as:

- Camp / Communication.
- Administration Building / Assay Lab / Gate House.
- Mine Maintenance Facility / Warehouse / Diesel Fuel Storage / Explosives Storage Facility.
- Various Water Management / Treatment Ponds.
- TSF Tailings and Reclaim Water.
- Water Treatment Plant (WTP).
- Sewage Treatment Plant (STP).

18.11 Communications

Off-site fibre optic cable and microwave-based connections with a minimum speed of 1 Gbps will be required to connect the Project. These redundant communications will provide high-speed internet access as required by the early works, construction and operation phases. On-site fibre optic cables will connect all site facilities as the mine is developed, configuring a backbone infrastructure for all computer systems within the Project.

All core network and computer systems will be allocated at a data centre that will require continuous power and air conditioning systems. The data centre should be able to allocate at least three (3) standard equipment racks.

A trunking-based radio system will provide robust radio communication with at least 30 channels that will be mostly used by the various construction sub-teams at the site. A minimum of 250 handheld units and 50 mobile units will be deployed during all stages of construction. Radio and wireless communications for OP and UG development will be required for safety and tracking purposes. Leaky feeder-based technology and LTE options will be evaluated.

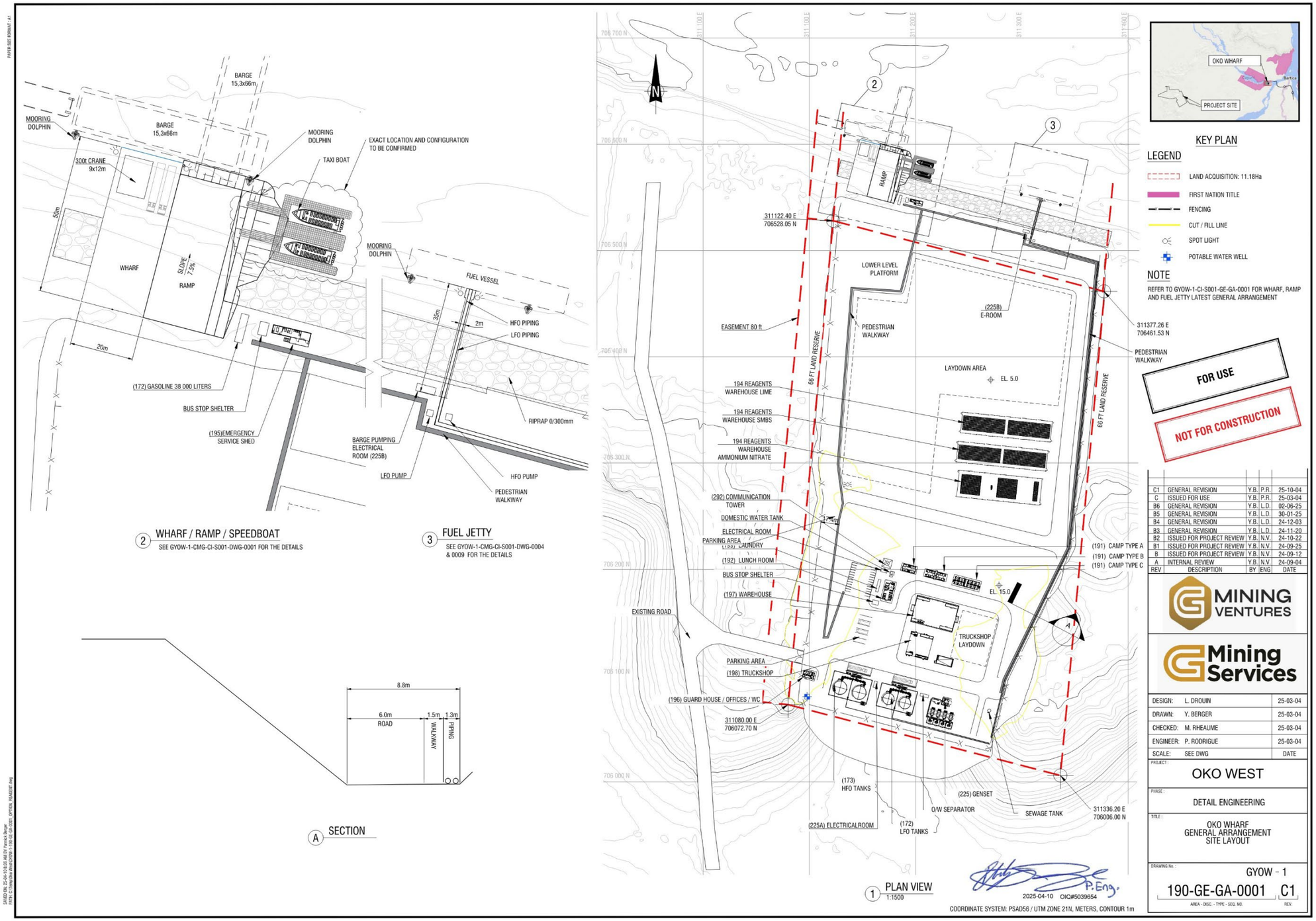
A fibre-based (GPON) network will provide communications to the camp, ensuring communications for all employees and contractors. The fibre will also connect the site to the Oko Wharf.

Mobile coverage will be 4G / LTE to ensure communications. In-country ISPs and mobile communications vendors will be scrutinized to ensure top quality of service and cost efficiency.

18.12 Off-site Infrastructure

The project infrastructure will also need to be supported by off-site infrastructure, as illustrated in Figure 18.28.

Figure 18.28: Offsite Infrastructure Arrangement



The off-site infrastructure critical to the project includes a dedicated wharf and storage facility designed to accommodate the barging of equipment and materials from ocean ships to provide land access and transportation for the rotating workers, both during construction and operations. The facility near the confluence of the Cuyuni and Mazaruni Rivers is strategically positioned to facilitate efficient handling of bulk materials and equipment. It will be designed to accommodate barges for both Roll-on Roll-off and crane unloading.

Adjacent to the wharf, a substantial laydown area will be established to manage and store materials and equipment and consumables prior to their transportation to the project site.

This laydown area is designed to accommodate large volumes of materials. The layout facilitates efficient inventory management and ease of access, with designated zones for different types of materials and equipment. The area will be surrounded by fencing and equipped with security measures to safeguard assets. It will be supported by a range of lifting and mobile equipment capable of lifting heavy and oversized items, as well as mobile forklifts and loaders for smaller material and routine tasks.

The landing area has been strategically designated for the construction of the storage of light fuel oil / diesel (LFO) and heavy fuel oil (HFO). The HFO and LFO storage tanks will accommodate 2,200 m^3 and 1,600 m^3, respectively. This results in nine (9) days of autonomy for HFO and 16 days for LFO. The landing workforce will require a small camp with a kitchen, water services, a small warehouse with a reagent storage area and a maintenance facility. Specific gasoline storage will be available to accommodate the refuelling of the two (2) speed boats with an autonomy of two (2) weeks, with 30,000 litres and with dedicated pumps.

A 50-person camp will be established at the barge landing with one (1) dorm per type (A, B, C) as described previously. All the services to allow a functional camp are present, such as a communication tower, a lunchroom with a small food preparation area, a bus stop shelter, a domestic water treatment plant with fresh water sourced from a water well and a sewage treatment system. Electric power will be generated from four (4) gensets close to the diesel storage. Additionally, the area will be fenced and secured with a guardhouse at the entrance. The camp will accommodate the personnel dedicated to the laydown and warehousing team, and the marine and trucking fleet.

A dedicated reagent storage and handling system has been developed to safely receive, store, and manage the supply of reagents required for processing and mine operations. The key reagents that have a large volume due to their consumption and their difficult logistic to be stored on site include:

- Sodium Metabisulfite (SMBS) 2,500 T equivalent to 4 months of peak consumption and minimum shipment of 2,100 T.

- Lime (CaO) 2,500 T equivalent to 2.5 months of peak consumption and minimum shipment of 1,900 T.
- Ammonium Nitrate (AN) 3,000 T equivalent to 5 months of peak consumption and delivery once every 4 months.

Given the site's remote location and reliance on barge-based transportation, a robust logistical and surge warehousing strategy has been implemented to ensure uninterrupted supply and safe handling throughout the mine life. The biggest storage will be established at the landing to reduce downtime of rehandling between the rented sea containers to be returned.

Finally, a dedicated emergency response shed, strategically located near the wharf and barge landing area, will be constructed as part of the project's health and safety infrastructure. This facility will serve as a first-response centre, providing immediate medical support for personnel and contractors operating in the vicinity of the material handling and logistical zones.

19. MARKET STUDIES AND CONTRACTS

19.1 Gold Market

Gold is a freely traded commodity in a well-established and mature market, renowned as a safe haven for investors. It is sold daily by banks and traders at a spot price for immediate delivery.

The Oko West Project will produce gold in doré bars. Oko West is authorized to export the gold doré bars from Guyana to an internationally recognized refiner. Prices are typically quoted in US dollars per troy ounce.

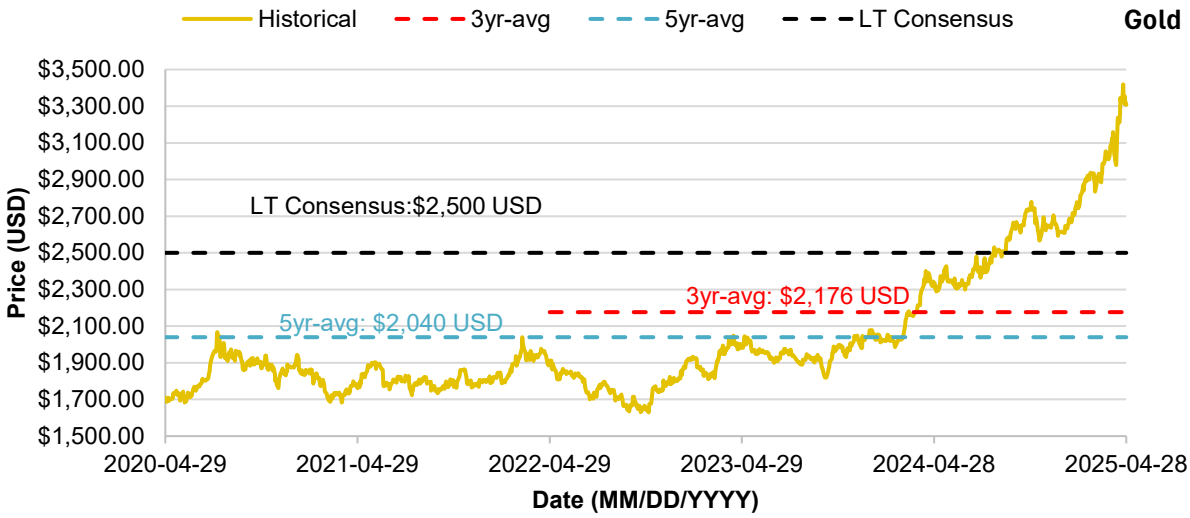
19.2 Metal Price

The price of gold is the primary factor in determining the profitability and cash flow from operations. The project's financial performance is closely linked to the gold price. The gold price was determined based on historical prices and the consensus of long-term estimates from mining analysts.

The long-term consensus price for March 10, 2025, is USD 2,500 per troy ounce *(Source: S&P Capital IQ, a product of S&P Global Market Intelligence. Accessed April 2025. © 2025 S&P Global Market Intelligence. All rights reserved)*. The financial analysis for the Oko West Project was based on the consensus price of USD 2,500 per ounce, which is significantly lower than recent spot prices.

Figure 19.1 shows the historical daily average value of gold for the last three (3) years. As of April 28th, 2025, the five (5)-year average stands at USD 2,040 per troy ounce, while the three (3)-year trailing average is USD 2,176 per troy ounce.

Figure 19.1: Daily Gold Price



Source: World Gold Council

19.3 Contracts

There are no refining agreements or sales contracts currently in place for the Project that are relevant to this Technical Report. The QP expects that terms contained within any sales contract that could be entered into would be typical of and consistent with standard industry practices and be similar to contracts for the supply of gold elsewhere in the world. Transportation and refining contracts for gold doré bars will be negotiated and finalized during the construction phase of the project. This Feasibility Study assumes a 99.95% gold payability factor and a refining, transportation and insurance charge of USD 8.00/oz.

G Mining Ventures has entered into a Master Service Agreement with G Mining Services, a Canadian engineering and mine development firm specializing in the mining sector, for the engineering and construction management of the Oko West Project.

G Mining Ventures has entered into contractual agreements with various suppliers. The current agreements are as follows:

- Open pit mining equipment with Komatsu.
- Process equipment with Metso.
- Permanent camp with Treysan.
- Barge landing camp with Precamp.
- Deforestation contract with Kanhais.

- Earthwork contract with Synergy.
- Earthwork contract with Japart.
- Earthwork contract with SAIF.

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Environmental Resources Management (ERM) is the author of Section 20 of the Technical Report, and Dr Kevin Leahy, Associate Technical Partner at ERM, is the QP responsible for this section. This Section discusses the relevant information on the environmental, permitting, and social or community factors related to the Oko West Project. The Oko West Project is in the advanced stages of exploration and development planning. Therefore, this section focuses on the potential environmental and social risks identified, their mitigation, and the opportunities they present. It also outlines the recommended future work needed to enhance the identification and effective management of potential risks as project design progresses.

In July 2024, G Mining Ventures Corporation (GMIN) and Reunion Gold Corporation announced that a business combination between them had been completed. GMIN has continued to engage the international consultancy Environmental Resource Management (ERM) - which has a registered Guyana entity - to complete their environmental and social impact assessment (ESIA) of the Oko West Project. This was submitted in accordance with local requirements in Guyana in December 2024. It was updated following EPA comments and further baseline results and studies, and was then resubmitted in May 2025. ERM's team included several subject matter experts from the University of Guyana's Centre for Study of Biological Diversity and professionals from other Guyanese environmental consultancies.

The information contained in this Section is informed by the studies completed to date, including: a desktop review of available information provided by GMIN and previous Project owners, Reunion, environmental and social baseline data collection, regulatory permitting, as well as recommended work plans for future environmental and social monitoring.

20.1 Environmental and Social Conditions

GMIN holds a 100 percent interest in the Prospecting Licence (PL) referred to as the "Project" for the purposes of this section. The PL covers approximately 6,700 hectares. G Mining has expressed that it has all of the required permits and approvals for its current stage of development, and these are in good standing. ERM has not undertaken a compliance assessment of the current exploration program.

The Project area has not been identified as a priority area of conservation interest by the Government of Guyana, nor does it fall in or near a Guyana Protected Area, a World Heritage Site, an International Union for Conservation of Nature Key Biodiversity Area or an Alliance for Zero Extinction site. The Government of Guyana has not granted formal title to any Amerindian (or Indigenous) lands within the PL; however, there are Amerindian land titles in very close proximity to Project ancillary facilities.

According to the Guyana Geology and Mines Commission, the PL is surrounded by 13 medium-scale mining and prospecting permits held by various Guyanese title holders and one (1) group of medium-scale mining and prospecting permits controlled by G2 Goldfields. The potential for cumulative environmental and social impacts has been taken into consideration in the ESIA as the Project developed.

20.1.1 Baseline Studies

Physical and biological baseline studies have been undertaken between 2022 and 2025, and social baseline studies between 2023 and 2025, with environmental studies completed during both the dry and wet seasons. The overall purpose of these studies has been to collect relevant data needed to inform Project planning, including the identification of potential issues of concern and recommended actions.

The focus for environmental baseline studies has been the collection of information in the immediate area of the FS Project footprint. Environmental baseline studies of the ancillary facilities, including the power plant, barge landing area and the main access road, were initiated in 2024. Social baseline studies considered the community and other stakeholders who could reasonably be expected to be impacted by the Project and its activities.

The findings of the environmental and social baseline studies are summarized below. Each topic includes a discussion of significant impacts and their mitigation or recommended management.

20.1.1.1 Physical Baseline, Impacts and Mitigation

Ambient Air

ERM conducted an onshore ambient air quality monitoring program in the 2023 and 2024 wet and dry seasons to characterize existing ambient air quality conditions in the Project area.

In the absence of applicable Guyana regulatory guidance, ERM used standard methods for selecting air pollutants to monitor, selecting the monitoring site, and establishing other program parameters. Resulting measurements were compared to the World Health Organization's (WHO) Air Quality Guideline Values (WHO, 2021).

The program was designed and implemented to assess background concentrations of NO_2, SO_2, dustfall, volatile organic compounds (VOC), PM_{10}, and $PM_{2.5}$ during four-week periods during the wet and dry seasons at the proposed mine site and wharf site in both 2023 and 2024. NO_2 or SO_2 results indicated low concentrations at both sample sites, having marginally higher concentrations of both compounds during the

wet season, whilst the wharf samples had marginally higher concentrations of both compounds during the dry season. This was expected as the monitoring location is situated in a community with nearby local emission sources, such as generators. VOC results confirmed low concentrations and non-detects for most compounds, with ethyl acetate and acetone most likely to be found at elevated concentrations. These compounds can occur as a natural byproduct of decomposition and otherwise are frequently found in hand sanitizers and lotions, and from vehicle exhaust emissions.

Monitoring for TSP, PM_{10}, and $PM_{2.5}$ showed background values were generally below the WHO's 2021 Air Quality Guidelines and were representative of expected conditions at the project site. The 2024 dry season results were slightly elevated over the Quality Guideline; this being attributed to global weather condition.

Air quality monitoring and an emissions inventory informed the dispersion modelling (using AERMOD) of a number of airborne pollutants, which predict that NO_2 and dust will be the most significant air quality impacts. The main mitigation for this is the development and implementation of a dust management plan (DMP) to include the monitoring of dust deposition and continuous PM monitoring. The DMP would allow mining activities to be kept under review with regard to dust emissions to assess the need for additional water spraying at the in-pit activities and the haul roads, and to avoid dust-raising activities during dry and windy periods. Upgrading from Tier 2 to Tier 3-compliant engines for haul trucks and engines at the Power Plant would help to mitigate the impacts of NO_2.

GHG and Carbon Stock Analysis

To summarize the baseline carbon stocks, surveys regarding vegetation and soils were conducted during the dry season in 2023. It is important to note that this was not a full baseline and is instead a preliminary report for internal understanding. The Project area was represented by the average of 11 plots established to perform a Forest Inventory. The results of the survey appear consistent with the descriptions of the Forestland in Guyana presented in the REDD+ report (Sampling Design and Implementation Plan for Guyana's REDD+ Forest Carbon Monitoring System (FCMS): Version 2). ERM recommends that this baseline study be followed by a more detailed study considering improvement measures to allow for the design of mitigation and restoration measures for the Project.

Soils

Soil surveys were completed at thirty-eight (38) locations on the Project area during the wet and dry seasons between 2022 and 2024. At each location, soil diagnostic tests were conducted and soil samples collected for chemical analyses. The samples were from undisturbed locations within the mine development area, as well as from areas with evidence of disturbance from activities including exploration and prior

artisanal mining. Due to the lack of data on soil productivity and environmental quality for the Project site, additional soil samples to assess the chemical characteristics of the soils were collected for laboratory analysis.

In general, the soils found in the Project AoI and its vicinity have the following characteristics:

- Deep, well-drained, brown-red, reddish, or yellow-red, gravelly clays, silt loams, or laterite soils, common on the landforms in steep hills and mountains.
- Excessively drained, gravelly, brown, sandy loam and gravelly silt loam, usually associated with landform and relief of sloping, hilly, and steep dissected pediments.
- Well-drained, deep, sandy clay loam and gravelly clay soil, which is common with hilly to steep residual upland landform and relief (ERM 2023).

Despite the often-favourable physical conditions of the soils (e.g., the texture of the soils is generally deep loams intermixed with gravel), the soil fertility appears generally low. The average pH of the soils in the samples collected showed little variability, being acidic with a pH ranging from 4.5 to 5.6. The acidic and well-drained conditions, along with the high-rainfall climate, have leached nutrients from the mineral soils, which has led to the generally low soil fertility observed, typical of tropical lateritic soils.

The reported concentrations for most heavy metals, polycyclic aromatic hydrocarbons (PAHs), total petroleum hydrocarbons (TPHs), and xylene (BTEX) were below the method detection limits. In the few cases where detected, the reported concentrations were very low or below the USEPA screening levels for residential and industrial reference benchmarks. None of the soils sampled had evidence of pollution from artisanal mining (e.g. mercury or petroleum hydrocarbons) or from other anthropogenic impacts.

The soil baseline established provides a useful reference to quality and structure for activities such as revegetating disturbed areas as part of mine closure using stockpiled topsoil, which is the main mitigation for impacts to soil. Other impacts identified were erosion, compaction and pollution of soils, all of which can be mitigated by careful adherence to standard construction and materials handling procedures.

Geochemistry

A thorough geochemical testing program was initiated by ERM in 2023 on waste rock and tailings materials for the Project. This included variability sample selection, as well as static and kinetic testing in the form of humidity cell tests (HCTs). The purpose of the geochemical characterization program was to determine metal-leaching and acid-rock-drainage (ML/ARD) characteristics of materials excavated, produced and stored during the development, operation and ultimately closure phases of the Project (waste rock, tailings,

and overburden). Static and initial kinetic testing results of projected waste rock types and tailings were summarized in a technical report issued to Reunion Gold in July 2024 (ERM, 2024), and the first phase of kinetic testing results were reported to G Mining in March 2025 (ERM, 2025). The program structure was based on global best practice (Price,1997; MEND, 2009) for environmental geochemical characterization. The results have been used to inform water-quality modelling and to develop effective management strategies for these materials throughout the project's lifecycle.

Static tests undertaken on the waste rock, tailings and overburden samples were acid-base accounting (ABA) to determine acid and neutralization potentials (AP and NP, respectively), shake flask extraction (SFE) tests, sulphur- and carbon-speciation analysis, and four-acid ICP-MS multi-element analysis. A total of 75 variability samples were selected to represent the main material types, as represented by multi-element geochemistry (captured by pXRF analysis) and identified by geological logging. The static test results show that ML/ARD risk of the main waste rock units (hanging and footwall granites, and footwall basalts) appears to be low, with the transitional domain and some of the (carbonaceous) sedimentary rocks most at risk to be potentially acid generating (PAG). Similarly, tailings generated from the processing of transitional material are also potentially PAG, but those from other lithologies are non-PAG. Given the relatively small volume of the transitional domain, overall volumes of PAG material are expected to be small. Overburden samples showed low ML/ARD risk, reflecting the deep tropical weathering that produced these soils.

Kinetic testing with HCTs began in October 2023 on 14 waste rock samples and four (4) tailings samples, these being representative of the material to be excavated and processed. HCTs concluded after 59 weeks, and nine (9) of the 14 waste rock samples were found to have stable and circumneutral pH, low sulphate, acidity and metal concentrations in their leachate chemistry and are not of concern for ML/ARD. These nine (9) were from the granites, basalt, volcanics and argillite that make up the bulk of the waste rock. The five (5) other HCT samples were sourced from saprolite, transitional material, and fresh siliciclastics. This included one determined as PAG, and others classified as 'uncertain' in the static tests. They exhibited variously elevated levels of arsenic, sulphate, and phosphorous in their leachate chemistry, but these levels were generally below the considered quality standards. These units represent a relatively low proportion of the overall waste rock to be produced, and so it is considered unlikely that leaching from them will materialize as a significant water quality issue.

The four (4) tailings samples also have shown stable and circumneutral pH, but compared to the waste rocks, had less stable and higher sulphate and lower arsenic in their leachate chemistry, as well as detectable cyanide. The tailings samples obtained for the HCT were from the metallurgical testing program and so were not pre-treated for cyanide, whereas in operations, all tailings will pass through a cyanide destruction circuit prior to tailings deposition. Thus, cyanide leaching is not anticipated to be a significant risk during life of mine. The TSF

pond will be monitored for the influence of leachate from the tailings. Although the potential risk for ML/ARD is considered low, the balance of cyanide leaching versus natural breakdown and dilution by rainfall will be a factor in determining the need for an effluent treatment plant at the TSF that will be constructed if required, as described in Chapter 18.

Three (3) overburden soil samples are currently undergoing HCTs, and following the final conclusion of the extended HCTs, an ML/ARD management plan can subsequently be developed to mitigate the potential adverse effects from waste rock on the receiving environment. The management plan will include periodic static testing undertaken throughout the life of mine to correctly identify PAG / non-PAG material, and the placement of saprolite and transitional waste material in central areas to be encapsulated by the less reactive waste rocks through the course of operations, as described in Chapter 18.

Surface Water Quality and Hydrology

The Project area has four (4) primary watersheds of minor river systems with many sub-basins: the Puriari River, Takutu River, Kairuni River and Oko River. The Puriari River watershed drains the western half of the concession area and flows southwest into the Mazaruni River. The Takutu River watershed drains the southern arm of the concession area and flows southeast into the Mazaruni River. The Kairuni River watershed drains the northeastern portion of the concession area and flows North into the Oko River, approximately 4 km north of the project concession area. The Oko River watershed drains a large area north of the project concession and flows northeast into the Cuyuni River. While all major watersheds in the Project area have been impacted to some degree by artisanal mining, the Oko River is the current site of the most extensive artisanal mining activities.

Two (2) wet and two (2) dry periods were defined in the region of the Project by the United Nations Development Programme (2024) using mean monthly discharge data for the Mazaruni River: the first dry season (February to April), first wet season (May to July), second dry season (August to November), and second wet season (December to January).

In the Project area, nine (9) co-located surface water quality and hydrology monitoring stations were installed, and automated stage monitoring was initiated between July 2023 and July 2024. The 2024 study area covered the mine pit area, airstrip, TSF, WSF and the workers' camp. Downstream locations outside of the Prospect Licence area were monitored from July 2024 to develop an understanding of downstream impacts and existing conditions.

In terms of water flow, there is a consistent streamflow pattern across all stations that highlights the variation between the wet and dry seasons and the region's reliance on rainfall to supply streamflow. In 2023 and

2024, streamflow began to increase in mid-April, and streams maintained elevated flows from May to July, corresponding to the first wet season. A second, comparatively subdued period of flows started in mid-November and continued until the end of January, corresponding to the expected timeframe of the second wet season of December and January.

The surface water flow monitoring has highlighted the historic artisanal mining activities that have significantly altered the responses of some watersheds. Bare ground at one station in the proposed pit area has the effect of being more reactive to precipitation than other stations. At the location of the TSF, data suggest that during the dry seasons, the streams may dry up entirely due to the influence of legacy artisanal pits retaining runoff. Baseline hydrological data suggests surface water scarcity within the project area during the months of the dry seasons. It follows that many streams within the project area may potentially go dry for some periods during a drier-than-average year.

In terms of baseline water quality, samples were collected to characterize existing sediment and water conditions in the Project area in 2022 at 12 stations. This continued in 2023 at 19 new stations, and in April 2024 with three (3) new stations. Sediment quality baseline sampling began in 2023 at 19 stations co-located with surface water sampling stations and continued in 2024 with the addition of five (5) new stations. One such site was also co-located with an aquatic biology site on a natural creek with limited mining impacts to establish a reference site for baseline conditions.

The baseline data collected for sediment and water from 2022 to April 2024 highlights a complex environment with significant spatial and seasonal variation, with elevated metal concentrations. At some stations, these sediment or water concentrations may pose a risk to aquatic life. Several stations are adjacent to roadways and near areas of previous artisanal mining, and these activities may be influencing the results obtained. Impacts of artisanal mining identified were mercury (measured as methylmercury) and cyanide, which were detected in the streams surrounding the project area and also in the Cuyuni River water column and sediments.

Project infrastructure and water diversions will disturb a number of streams and large sections (16 to 33%) of the watersheds, except for the Oko, which is not directly disturbed, but is immediately downstream of the Kairuni catchment. The TSF is entirely within the upper catchment of the Puriari River watershed that drains west and then flows south into the Mazaruni River. The southern part of the pit, the processing facilities and buildings, settling ponds and the air strip are in the upper catchment of the Takutu River watershed that flows southeast into the Mazaruni River. The northern pit and the WSF are located in the small Kairuni River catchment that flows north into the Oko River, approximately 4 km north of the project concession area. This infrastructure will require the diversion of upstream flows around project infrastructure for non-contact water, specifically around the open pit, TSF and WSF. The pit diversion channel will redirect the

uppermost headwaters of the Kairuni to a catchment pond used to supplement operational water supply, and then south and east into the Takutu using a preliminary and then final diversion design to allow operational access and flexibility (see Section 16.3).

Water balance and hydrologic studies by ERM and Newfields have contributed to the drainage design in Section 16.3 and Water Management in Section 18.7. Much of the water from the pit and underground dewatering, mineral processing, TSF and contact water run-off will be recycled for use to supply the process plant, whilst excess water will be diverted to the Takutu. The water quantity results for the model indicate that the water availability onsite under average and reasonably foreseeable extreme conditions is sufficient to meet project needs.

The ERM (2025) model indicates that the largest change in downstream water quantity in the receiving environment is due to the redirection of water from the Kairuni River catchment to the north tributary of the Takutu River. Discharges of used water and contact water from the sediment pond will significantly increase streamflow in the tributary and likely result in the exceedance of the Takutu River north tributary's bankfull discharge capacity. It is important to note that these conclusions are based on numerous assumptions utilized in model development, including the sizing of the sediment pond, and that engineering specifics for some infrastructure components are still under development. Therefore, actual water balance conditions during operations may differ from the current understanding, as mine infrastructure and water use, and treatment are refined.

The TSF water balance described in Section 18.6.2 envisions 80% water recycling from the tailings and the accommodation of a one-in-10,000-year Inflow Design Flood (IDF) 24-hour event with a 1 m safety margin on the freeboard. The geochemical assessment of tailings is described in the sub-section above, along with the criteria for establishing a TSF effluent treatment plant. Water in the TSF will be from the tailings supernatant and the rainfall that enters the impounded area and will be recirculated back into the processing plant or will be lost to evaporation. Water Balance Modelling indicates that excess water requiring discharge from the TSF would only occur in an exceptional run of seven (7) consecutive one-in-15-year wet seasons, and in that case would discharge into the upper catchment of the Puriari River. Towards the end of the life of mine, as ore processing water requirements decline, discharges will be accommodated by a spillway as part of the final closure design that will be subject to monitoring against water quality standards and may require treatment in the long term.

During construction and operation, impacts to surface water quality can occur from erosion and sedimentation, effluent discharge (including treated sewage), seepage from the WSF and TSF, and spillages. These potential impacts can be mitigated by careful adherence to standard construction and materials handling procedures, by the designs and water management processes, and by infrastructure

described above and in more detail in Sections 16.3 and 18.7. An accidental spillage of hazardous material is the impact with the potential for the greatest significance, although the likelihood is considered low. The main mitigation for all these potential impacts is the development of drainage and impoundments for all contact water and for run-off and seepage from the WSF, much of which will be settled and used for process water supply.

The water quality results for the Water Balance Model indicate that during the operational phase, water quality will stay the same in the Kairuni River and Puriari River catchments. For the Takutu, the model indicates that excess water will accumulate in the sedimentation ponds. To protect water quality in downstream receiving environments and reduce sediment loading, the water will need to be tested prior to discharge to ensure it meets the appropriate water quality standards that will be defined in the Environmental Permit to operate. In the current water management design and understanding of geochemical sources, discharge from the sediment pond and flocculant treatment facilities into the north tributary of the Takutu River is expected to increase concentrations of chemical constituents, with some, including but not limited to mercury and arsenic, exceeding IFC water quality guidelines. Further pond sizing or additional water treatment infrastructure may need to be considered in future detailed design phases.

The most affected streams in the area were previously heavily mined by artisanal miners and have little to no community or cultural value, nor are they critical for biodiversity or populations of species of interest, while the quantity of water affected has little influence on downstream regional stream flows. A program of monitoring water quality and flow will be implemented throughout construction and operations that will help inform mitigation strategies and designs in the long term.

The long-term objectives of the mine rehabilitation within the Closure Plan are to establish self-sustaining water conveyance systems that replicate or approximate pre-disturbance flow pathways, watershed characteristics and water quality, taking into account the development of a pit lake in the Kairuni catchment. These are discussed further in Section 20.3.

Groundwater and Hydrogeology

Hydrogeological surveys were conducted during the dry seasons of 2023 and 2024 to expand on the existing groundwater data collection in 2022 as the initial monitoring network and in support of the baseline studies program. Groundwater samples were collected from two (2) sampling points during the wet season and four (4) sampling points during the dry season in the 2022 survey.

A total of 14 monitoring wells were drilled and installed around the Project area during 2023, and a further nine (9) in 2024 to improve spatial coverage at the WSF, TSF and pit. Dataloggers obtained continuous

water level measurements in 14 of the monitoring wells. Hydrogeological testing, including packer and other pumping tests, was undertaken at six (6) key wells to determine aquifer permeability at different depth horizons and establish optimum discharge rates and the well abstraction efficiency.

Three (3) main hydrogeological units have been characterized: the saprolite, the transition zone and the fractured bedrock basement. The saprolite is typically saturated and can extend up to 40 m in depth. High clay content results in low permeability, and the saprolite is therefore considered an aquitard. Conversely, the transitional zone is considered an aquifer and typically consists of a combination of saprolite and smaller boulders or unweathered or fractured granite, several metres thick, up to around 10 m. The fractured bedrock in the Project area is predominantly granites, but other lithologies are present. All have frequent open fractures and occasional fault zones, some with mineralization.

Groundwater is encountered at a range of depths across the monitoring array, from as shallow as 9 m below ground level (mbgl), and up to 105 mbgl. However, it is most commonly between 20 and 45 mbgl, often corresponding with the depth to the transition zone aquifer. There is a correlation between topographic and piezometric (rest-water level) elevation, with shallow groundwater at low topographic elevations and deep groundwater at higher ground, as is common. Rest-water levels are shallowest around the WSF at around 3 mbgl, whilst on higher ground of the Project area, levels are established between 18 and 58 mbgl. Seasonal fluctuations range between 3.5 and 5 m in the wells on higher ground.

Groundwater quality is characterized by near neutral to slightly alkaline pH, and relatively low dissolved solids (34 to 160 mg/L). Dissolved metals were either non-detect or below CONAMA quality guidelines, except for iron and manganese, which widely exceeded the quality standard (typical of lateritic profiles) and zinc in one (1) sample location. There were no other significant detections of organic contaminants, cyanide or mercury.

The results from continued monitoring rounds have been used to inform water quality and numerical groundwater flow and contaminant migration modelling and form the basis for developing effective strategies for managing water inflow into the open pit and underground workings during the life of mine. Modelling has been completed for a range of groundwater inflow scenarios and uncertainties, and these have contributed to the Mining Method design in Section 16.3 and Water Management in Section 18.7, as well as the Water Balance Model (ERM, 2025).

Modelling studies have also been used to validate design assumptions used in the conceptual mine closure plan to mitigate the potential adverse effects to groundwater quality and quantity in the receiving environment (both natural and man-made) from Project activities across construction, operations and into post-closure.

The main impacts on groundwater quality could occur from the seepage of percolating leachate from the WSF and TSF, and whilst the low permeability of the saprolite significantly attenuates seepage, infiltration and limited lateral migration are possible over the long term. Both waste rock and tailings are expected to be predominantly non-PAG, although some sulphate and metal leaching may develop from both over time. Predictive contaminant modelling indicates that small areas close to the TSF dams and the downstream (eastern) margin of the WSF may develop a plume with elevated leachate concentrations over the very long term, in the order of a few hundred metres over one hundred years. However, there are no groundwater wells or users within several kilometres, and baseflow from groundwater to surface water is thought to be minimal due to the saprolite aquitard.

The saprolite aquitard also offers a measure of protection from potential impacts from spillages, all of which can be mitigated by careful adherence to standard construction and materials handling procedures, as well as by the designs and water management processes and infrastructure described above and in more detail in Sections 16.3 and 18.7.

Groundwater quantity is also affected by dewatering during the operational phase, although due to the low permeability saprolite, baseflow contribution to surface waters is considered low, and the impact beyond local groundwater availability is also considered low. Groundwater levels will rebound in closure and reestablish over the long term.

20.1.1.2 Biological Baseline, Impacts and Mitigation

Guyana's biodiversity is characterized by its diverse ecosystems, including tropical rainforests, savannas, wetlands, and mangroves. Located within the Guiana Shield, its stable geological structure contributes to the region’s high levels of endemism and species diversity. Guyana consists of extensive freshwater systems that support a wide array of terrestrial and aquatic life.

The Project Area, though not recognized as a conservation priority by the Government of Guyana, is located within a region of high biodiversity significance. Despite the absence of designated protected areas nearby, the Project Area's ecological value highlights the importance of integrating conservation considerations into project planning and implementation. Understanding the spatial distribution of biodiversity and identifying critical habitats within the Project Area are crucial steps towards responsible environmental stewardship and sustainable development.

The biological baseline for the Project Area was established through a combination of desktop research and in-field surveys conducted between 2022 and 2024, with the objective being to comprehensively assess the presence and distribution of key species and habitats within the study area.

The significant species diversity and abundance observed across terrestrial and aquatic surveys highlight the ecological richness of the Project Area and indicate the regulatory requirements and mitigation measures to ensure protection during project development. Fluctuations in species abundance and richness across seasons suggest the need for adaptive management strategies in planning and mitigation efforts. The presence of species of conservation concern, such as endangered (EN), vulnerable (VU) or near threatened (NT) IUCN Red List species and CITES-listed species, and high biodiversity in the Project Area will require robust environmental management plans.

As part of their support of biodiversity, in the context of local development and social benefit, GMIN plans to set aside the Willems Preserve as a “reserve” in an area south of the Prospect Licence Area. The total area of the proposed Willems Preserve is 16.3 km^2, of which 4.5 km^2 falls within the PL. GMIN is in discussions with the Forestry Commission to allocate this area and confirm its suitability for a reserve.

The baseline highlights the need for a comprehensive conservation strategy that integrates habitat protection, species monitoring, and adaptive management practices. This approach is crucial for minimizing project impacts and ensuring long-term sustainability. The following recommendations are suggested:

- Mitigation strategies should focus on avoiding, minimizing, and compensating for potential impacts on these species and their habitats.
- Implement a Wildlife Sightings & Records Book and database for ongoing monitoring of wildlife sightings and habitat use patterns.
- Establish a Biodiversity Monitoring Program to periodically assess species populations and habitat conditions.
- Conduct detailed assessments to identify critical habitats, especially for species like giant otters, and integrate findings into project planning and mitigation strategies.

By addressing these recommendations and continuing to monitor biodiversity, the Project can proactively manage environmental risks and enhance its conservation outcomes while ensuring compliance with regulatory requirements (as described in Section 20.2.1 below).

Terrestrial Ecology

Terrestrial biodiversity surveys conducted during the dry and wet seasons have provided a comprehensive description of the existing mammal, bird, amphibian and reptile species located in the Project Area. The 2023 surveys significantly expanded both the geographic coverage and methodological approaches compared to the previous year, resulting in a more robust examination of the area. In 2023, additional Auditory Encounter Surveys (AES) and Visual Encounter Surveys (VES) along transects, mist netting, pitfall

traps, camera traps, and environmental DNA (eDNA) sampling ensured a more comprehensive assessment, validating species presence and significantly increasing the understanding of species abundance across all taxonomic groups. In 2024, baseline data collected has expanded to include areas outside of the immediate mine footprint required for planned ancillary infrastructure, such as the barge site, the power line proposed at the time, and the access road.

The mammals identified during the completed wet and dry season surveys offer valuable insights into the biodiversity dynamics within the Project Area. A total abundance of 354 individuals and 23 species were identified during the 2022 to 2024 surveys. In 2022, a total of 81 mammals were recorded, with 34 individuals observed during the dry season and 47 during the wet season. The following year, in 2023, mammal abundance increased to 200 individuals, consisting of 80 mammals observed during the dry season and 120 during the wet season. During 2024, the total number of mammals for the dry season was 73. These observations highlight fluctuations in mammal populations over the three-year period, reflecting influences from seasonal factors, sampling effort, and updated methodology. Several species identified were on the IUCN Red List (EN, VU, or NT) and CITES.

Giant otters (*Pteronura brasiliensis*) are an IUCN red-listed endangered species and were present in the Project area during the baseline field campaign in 2023. In response, more detailed studies were undertaken from July to August 2024, aimed at gaining an understanding of the giant otter population size, distribution and habitat use within the Project Area and surrounding river catchments, including the Puriari River, Takutu River (tributaries of the Mazaruni River), and Oko River (tributary of the Cuyuni River) (collectively referred to as the Regional Survey Zone (RSZ)). Giant otters were observed across all three (3) river systems, confirming their presence within the proposed survey area and allowing a preliminary estimate of population. Further dry season surveys and consideration of giant otters in the Biodiversity Management Plan, including the need for habitat offsets, may be required to be implemented.

Across the wet and dry season bird surveys of 2022 and 2024, there was a noticeable increase in both total abundance and species richness. From 2022 to 2024, surveys identified a total abundance of 4,588 individuals across 274 species. In 2022, 1,187 birds were recorded, with 616 individuals observed in the dry season and 571 in the wet season. The following year (2023) saw an increase in bird abundance to 2,564 individuals, comprising 1,358 birds during the dry season and 1,206 during the wet season. By 2024, bird numbers during the dry season totalled 837. These observations highlight fluctuations in bird populations over the three-year period, influenced by seasonal dynamics, survey efforts, and updated methodologies.

Similar to the mammal survey efforts, the use of different sampling methods across survey seasons offers additional insights into bird species detection and survey efficiency. While AES and VES methods were

consistently used across all seasons, the integration of mist netting alongside AES/VES during the wet seasons of 2023 led to a substantial increase in species detection and identification. The additional survey methods likely contributed to the identification of a higher number of endemic bird species, IUCN Red List species, and CITES-listed species during the wet season of 2023 compared to previous surveys.

In the dry season of 2022, amphibian abundance surpassed 370 individuals with 17 species and 6 families recorded, while reptile counts totalled 121 individuals spanning 9 species across 4 families. Subsequent wet seasons of 2022 and 2023 saw notable increases: amphibian counts exceeded 268 individuals with 15 species and 6 families in 2022, while reptile counts reached 120 individuals across 12 species and 6 families. By the wet season of 2023, amphibian counts exceeded 470 individuals with 26 species observed, and reptile counts rose to 174 individuals spanning 18 species and 9 families. In the 2024 dry season, combined amphibian abundance from transects and pitfall surveys surpassed 452 individuals across 23 species, with reptile surveys identifying 116 individuals across 26 species.

The findings of the surveys highlight the dynamic nature of amphibian and reptile populations across different seasons and note the importance of using diverse sampling methodologies, such as AES/VES and pitfall traps, for comprehensive biodiversity assessments. Some species identified were CITES-listed species. The observed fluctuations in abundance and diversity emphasize the need for continued monitoring and conservation efforts to ensure the preservation of amphibian and reptile species in the Project Area.

The construction and operation of the Project will lead to loss and degradation of vegetation and wildlife habitat of moderate significance due to the permanence of habitat alteration and ongoing degradation, despite mitigation efforts like phased clearing and habitat protection measures. Dust control strategies and continuous monitoring will minimize the impacts of vegetative metabolic distress caused by dust airfall. The sensory disturbances of wildlife can be mitigated by noise and light control practices, whilst injury and mortality of wildlife can be controlled through targeted controls like wildlife relocation and rescue plans and vehicle speed enforcement, which help to minimize wildlife fatalities. During the closure phase, mitigation measures, such as regrading of benches on WSF, revegetation of disturbed areas, long-term water management at the TSF, and capping of decline and ventilation openings, are expected to effectively minimize residual impacts.

Aquatic Ecology

A total of 25 sites were surveyed in the wet and dry seasons in 2022 and 2023 within and surrounding the prospect licence area to establish baselines for macroinvertebrates, fish, and physical habitat conditions.

An additional seven (7) sites were also surveyed in 2024 to include areas that may be impacted by the development of ancillary facilities.

The results of the physical habitat survey indicate widespread impacts of historical and current artisanal gold mining, as well as associated land clearing, road construction, and channel disturbances. Some streams, particularly in the southern portion of the licence area, retain aspects of a natural channel morphology, but nearly all streams within and surrounding the licence area, except for some extreme headwaters, have already been impacted by mining or timbering activities to some degree.

The abundance and diversity of macroinvertebrates varied significantly between the dry and wet seasons, with a greater abundance of terrestrial and aquatic macroinvertebrates in the dry season but greater diversity and evenness in the wet. The majority of sites had better water quality in the wet season than in the dry.

The wet season fish survey identified a total of 66 species of fish within the mine's prospect licence boundary; these species were distributed across five (5) orders, 19 families and 41 genera. The number of species captured across all sites was slightly higher in the dry season than in the wet season (45 species in the dry season versus 41 species in the wet season, or a 10% increase in the dry season over the wet season), but abundance increased by a larger margin (48% increase in the dry season over the wet season).

None of the species recorded in these surveys to date are currently listed as being of conservation concern according to IUCN or CITES. However, several species hold significant ecological and economic value. For example, some species are key food or protein sources, and others are sought after in the aquarium trade. This highlights their potential vulnerability to either fisheries overexploitation and/or habitat loss. Several important fish species recorded for the combined seasons were the *Pseudoplatystoma fasciatum, Hoplias aimara* and *Hoplias malabaricus*, which are at the top of the food chain and can serve as bioindicator species. These species play very important roles in maintaining balance within the food webs by regulating and ensuring that many smaller-sized fish and aquatic animal species do not become dominant. Hence, they maintain the biodiversity and promote stability within these ecosystems. These species also provide an indication of the health of their ecosystems due to their ability to accumulate and bio-magnify environmental contaminants (such as mercury) within their tissues. As bioindicators, monitoring their populations and health can provide a pertinent but early indication of pollution and environmental degradation.

The Project area encompasses a wide range of aquatic habitats, including high-gradient headwater / ephemeral streams, larger perennial bottomland streams, ponds, and swamps. This diversity

in habitat preferences underscores the ecological complexity within these freshwater and headwater systems in tropical watersheds. The historical artisanal mining has led to environmental degradation described above and poses a significant threat to these habitats, potentially impacting the biodiversity and ecological balance of these freshwater streams. Therefore, the Project's contribution to conservation efforts, habitat restoration and continuous monitoring, through the Biodiversity Management Plan, will be important to help maintain the ecological integrity of these freshwater and headwater ecosystems.

Aquatic species and habitats will be affected by minor residual impacts caused by turbidity and sedimentation, and exacerbated by mining activities, despite their adaptation to previous disturbances and implemented mitigation measures. Habitat loss and degradation will moderately affect aquatic species due to permanent alterations, impacting reproduction, behaviour, and migration. Changes in stream hydrology affect small headwater streams and their aquatic populations. Downstream water chemistry alterations are likely to have minor residual impacts on aquatic resources, mitigated by sedimentation controls. During the closure phase, mitigation measures, such as regrading of benches on WSF, revegetation of disturbed areas, long-term water management at the TSF, and capping of decline and ventilation openings, are expected to effectively minimize residual impacts.

20.1.1.3 Social Baseline, Impacts and Mitigation

Socioeconomics

The socioeconomic baseline began in May 2023 to establish an initial understanding of the Project, geographic context, socioeconomic and cultural dynamics, and to define a preliminary Area of Influence (AoI) for a potential future mine. This was followed by longer fieldwork visits in October 2023 and May 2024. All fieldwork trips included a team of international ERM and Guyanese Subject Matter Experts.

Settlements within the preliminary AoI are:

- Georgetown, the capital city.
- Parika, a key port town on the Essequibo River, located west of Georgetown and accessible by road from the capital.
- Bartica, a regional centre located at the confluence of the Cuyuni, Mazaruni, and Essequibo Rivers.
- Agatash and Dagg Point, Amerindian communities that share boundaries and infrastructure with Bartica.
- Batavia, an Amerindian settlement and titled area at the confluence of the Essequibo and Cuyuni Rivers.

- Kartabo (sometimes spelled Karatabo), an Amerindian settlement on the Mazaruni River, in the vicinity of the proposed barge landing facilities. Kartabo is not formally titled.
- Karrau (or Karau), an Amerindian settlement and land title located across the Essequibo River from Bartica.
- Itaballi (or Itaballi Landing, sometimes spelled Itabali), a port community at the northeastern end of the Puruni Road (or Itaballi-Puruni Road).
- Takatu (or Takutu), an informal landing located approximately halfway along the Puruni Road, and currently acting as the land commercial and service centre between Itaballi and the Project.
- Bryan's Gate, a private toll road used to access the Project site between Takatu and the Prospecting Licence Area.
- Oko West Camp and nearby landings known as Sand Hill, Sand Hill 2 and Oko Landing, as well as Blackwater Crusher and Blackwater.
- Puruni (or Puruni Landing), a landing area at the southwestern end of the Puruni Road, on the Puruni River.

Gold and diamond mining is central to livelihood strategies throughout Region 7, specifically artisanal alluvial gold mining in the Project area. To buffer against the risks inherent to boom-bust cycles associated with gold mining, most households practice diversified livelihood strategies, including a mix of other formal and informal activities, such as logging, vending or commerce. Households are typically male-led, and males are the primary—if not often the sole—income earners. Community members, both men and women, also maintain livelihoods as shopkeepers, buying and selling gold, innkeepers and brothel owners, cultivating kitchen gardens, and conducting ad hoc repair and construction work. Interviews with community members suggest formal positions with established mining operations are attractive because salaries tend to be much higher than the national average, but these positions are very difficult to secure.

Mining in Region 7 includes artisanal and small-scale mining, large-scale exploration, and operations. There is currently a moderate to high level of artisanal mining activity in the region and preliminary AoI. Local trails and tracks cross the Project area, although the previous title holder has maintained the PL largely free of artisanal miners. Now, only pork-knockers using metal detectors continue mineral prospecting along roads and paths within the PL. There are active dredging and shaft operations to the north (Blackwater Crusher, Blackwater) and west (towards Puruni) of the Project area. Occupants and operators are granted land usage and extraction rights by legal owners who receive a percentage of mineral earnings.

Primary health concerns in communities throughout the preliminary AoI are malaria, dengue, and typhoid, with access to clean (potable) water a common problem, particularly in landing and backdam areas, as well as smaller interior communities. Food insecurity and nutritional issues are also common within the preliminary AoI, as available cash or spending power is strongly associated with boom-bust mining cycles in the backdams. Correspondingly, successfully harvesting animals (fish and wildlife) for consumption is reported to be more challenging in areas impacted by mining and forestry activities. Access to medical care (including at posts, health centres, and hospitals) is similarly limited in more isolated communities. As these dynamics show, an existing economy underpinned by artisanal and small-scale mining requires key consideration in developing a future mine and is a common risk to projects in many parts of the world, including Guyana.

There are both formally titled and non-titled Amerindian (or Indigenous) land titles in the preliminary AoI, and in proximity to the ancillary facilities Itaballi Landing, including Batavia and Karrau (titled), and Kartabo (untitled). Surface rights on Amerindian-titled lands are ultimately granted by Amerindian communities, with agreements managed and overseen by the Government of Guyana.

Oko Landing, an informal settlement of artisanal miners and those who support them, is located approximately 1 km north of the Project's PL and has experienced a significant reduction in its population. In May 2024, the estimated population was 40 persons and experienced a continued reduction in the rest of the year. There is a possibility that when the Project Mining Licence is granted and artisanal mining is no longer allowed in the Licence area; the population of the Oko Landing will cease to exist and will not influence the operations of the Project. However, some residents had previously noted that Oko Landing has been their primary residence for many years, and finding a new home and livelihood would be difficult.

Bryan's Gate, a privately owned toll gate within the PL, provides access to the landing areas and the Oko West PL area. As the PL undergoes land use changes during the construction phase, the TSF dam will be built close to Bryan's Gate, such that it will no longer be available as a transportation route for the landing communities in the area. This will impact on the movement of individuals living in the region, and alternatives are poorer quality roads through the Puriari River watershed.

It is anticipated that the Project will impact on the following socioeconomic resources / receptors within the Social AoI in terms of positive impacts on employment and business activity, negative impacts on land use and ecosystems services, potential economic displacement, community health, safety, and wellbeing, vulnerable / indigenous peoples, and social infrastructure such as road usage and access and river-based transport. No resettlement is expected, and no current artisanal gold workings are present in the project footprint.

The TSF will eventually impound over 72 million m^3 of tailings and water behind dams up to 32 m high constructed using the downstream raising method, in line with best international practice. Currently, the area downstream from the TSF has approximately 70 artisanal miners working in several small operations, meaning the consequence risk criteria adopted refers to the highest classification, extreme', as per GISTM guidelines, to be confirmed during the next phase of the project. However, once the Mining Licence is granted, which is expected in late 2025, prior to the start of operations, artisanal mining within the Licence Area will cease, and the miners will have to relocate operations elsewhere. For classification purposes, a dam breach analysis will be done for various scenarios and will be shared with the key stakeholders. All artisanal miners and the population living within the flood zone in the event of a dam failure will be mapped annually, as well as trained to take refuge in the self-rescue zones in accordance with our mining dam emergency plan.

Stakeholder engagement, as described in the sub-section below, is a key tool through which social impacts can be mitigated. Positive impacts can be enhanced by employing Guyanese citizens where possible as part of a Local Employment Plan, partnering with local institutions to support workforce development programs targeted to in-demand skillsets and providing Guyanese candidates with appropriate training. Other impacts can be mitigated by careful planning through the development and implementation of a Community Health and Safety Management Plan, Worker Code of Conduct, Grievance Mechanism, Traffic Management Plan, and Emergency Response Plans.

Stakeholder Engagement

Previous owners, Reunion Gold, established strong relationships with some key stakeholders in the Project area, including formal titleholders, organizations, businesses, and/or individuals. Engagement to date has included long-term and informal relationship-building, and public scoping consultation meetings in December 2023, as required by the EPA. There have also been increasingly focused efforts that align with mine planning and development, such as job fairs in nearby population centres like Bartica, as well as community investment and development initiatives in Amerindian communities. Engagement has continued under the current GMIN owners, with the formal EIA Disclosure public meetings and in technical meetings with the Environmental Protection Agency (EPA) in January 2025, as part of the EIA and Environmental Permitting process (see Section 20.2 below).

Engagement planning is a central component of the project's Sustainability Strategy, and a formal Stakeholder Engagement Plan is being finalized that is commensurate with the advancement of mine planning and permitting. Stakeholder 'mapping' conducted in early 2024 as part of the development of a Stakeholder Engagement Plan identified key stakeholders by representative group or category, including:

- Governmental bodies (e.g., Guyana Forest Commission, Ministry of Amerindian Affairs).

- Businesses and private sector organizations (e.g., Willems Timber, Boom Blast Driving, Mekdeci Machinery and Construction).
- Civil society (e.g., Guyana Gold and Diamond Miners Association, Conservation International).
- Local and Indigenous communities and population centres (e.g., Itaballi, Batavia, Bartica).

Feedback and perspectives on the Project collected as part of stakeholder engagement thus far centre on the following themes:

- Livelihood diversification and income-generating opportunities (especially for Indigenous women) with the mine and/or supported by the Oko West Project.
- Education, including programs to support ongoing school attendance and limit secondary school drop-out rates among adolescents who leave school to support their families by finding work in mining.
- Safety, health, and well-being, including concerns about impacts on Amerindian communities as a result of any potential increases in traffic and non-community visitors with mine development (including ancillary facilities).

While the final Stakeholder Engagement Plan is being finalized, GMIN advanced in the last quarter of 2024 with the creation of the Multistakeholder Working Group (MSWG) within Region 7. The group was created to be a platform for open and interactive dialogue amongst the communities and individuals, and all those impacted by the Oko West Project. The group consists of 12 representatives across the Government, regional office, women miners' association, small- and large-scale businesses, regional education and health, Mayor's office, Miners association and villages of Itaballi, Batavia, Karrau and Karatabo. Meetings are held quarterly at Bartica.

GMIN currently invests USD 100K per year in social projects, which are voted on and presented by the MSWG, with USD 1 million per year for the start of production at the Oko West Project. The group is updated on GMIN's current activities and initiatives for the region, which are communicated back to their respective communities.

As part of ongoing work to formalize communications and engagement with all stakeholders, GMIN keeps records of all engagement activities undertaken.

Cultural Heritage

Between September and October 2023, a cultural heritage baseline study that employed background research, archaeological field surveys, and community interviews was conducted within the Project area and surrounding communities to assess what cultural resources (tangible and intangible cultural heritage) may be affected by a potential future mine. Background research did not identify any cultural heritage sites in the study area, but archaeological field surveys documented three (3) cultural resources: one (1) archaeological site and two (2) isolated finds' locations. Three (3) additional cave sites were identified in the Project area, but field surveys in May 2024 did not reveal any cultural materials present and, as such, have been discounted as potential sites. There are no built heritage sites in the current proposed Project area.

In two (2) areas of the project, the TSF area and west of the Pit, a number of colonial-era bottle scatters were observed on the surface of the jungle floor. Investigations did not reveal any buried deposits associated with the bottles, so they are regarded as isolated finds and not significant.

One (1) archaeological site was discovered within the footprint of the planned open pit. The site, a rock shelter (designated Cave-1), had pottery and lithic debitage and stone tools visible on the surface. Interviews with residents of communities adjacent to the Project area or in the broader region indicated that while some people are aware of caves in the area, people are not venturing into them and are not aware of the caves of this region having ever served as habitation or camp locations. In May 2024, a joint team from ERM and the University of Guyana undertook a series of 3D digital surveys and archaeological test excavations at Cave-1 to further understand the nature, extent and significance of the site in order to feed into the regulatory permitting process. This work was under permit from the EPA. A further survey was undertaken by the University of Guyana team in November 2024, including 3D digital surveys, ten (10) archaeological test pit excavations and geoarchaeological surveys and samples.

Significant quantities of archaeological artifacts (circa 700) were recovered from the surface of the cave and from 13 test pits, but there was no evidence of buried features, no funerary activity within the cave or surrounding area – no burials, human bone or votive offerings. There is no evidence of petroglyphs (rock carvings) or pictographs (rock paintings). Post-excavation analysis was undertaken in Georgetown under the direction of Louisa Daggers of the University of Georgetown. While no charcoal was recovered for C14 dating, the artifacts recovered from the site fit the Mabaruma Aruka Phase, suggesting a prehistoric date of 1450 B.P.

The archaeological investigation of parts of Cave-1 undertaken in May and October 2024[1], confirmed the archaeological presence and significance of the site, recovered a significant collection of ceramic and lithic artifacts, and concluded that further archaeological material likely remains in the remaining unexcavated sections of Cave-1.

Cave-1 is an important archaeological site both regionally and in the wider national Guyana context, and it can be classified as 'Non-Replicable Cultural Heritage' as per the IFC Performance Standard for Cultural Heritage (PS8). Given its location within the pit, Cave-1 will be removed if the project proceeds. Preservation in situ is not deemed feasible as it would severely impact the viability of the project. It is deemed that the overall benefits of the project to the surrounding population and Guyana as a whole would outweigh the anticipated cultural heritage loss through the removal of the site. Since preservation is not deemed feasible, a suitable mitigation measure is required under PS8 to minimize this anticipated loss of Cultural Heritage, in line with the Equator Principles, Good International Industry Practice and more specifically, IFC Performance Standard 8 (Cultural Heritage). The EIA has recommended that Preservation by Record (full archaeological excavation of the entire Cave-1 site, followed by an appropriate level of post-excavation analysis) using modern scientific techniques be undertaken as mitigation for the removal of the site. With the removal of only a small portion of the deposit, excavation of the remainder could result in the recovery of additional cultural items and features. However, from the archaeological investigation conducted in October 2024, the University of Guyana report concluded that the most significant cultural remains in Cave-1 have already been removed.

20.1.1.4 Future Environmental and Social Management Plans and Studies

A comprehensive, project-specific baseline study program was initiated in 2022, and information collected to date has helped build an understanding of the local and regional environmental and social conditions that have informed the FS. Baseline surveying was ongoing following the submission of the EIA in January 2025 and was completed prior to the resubmission in May 2025. For many topics, such as surface water and groundwater and biodiversity, baseline studies will be advanced into ongoing regular monitoring programs, as captured in the overarching Environmental and Social Management and Monitoring Plan (ESMMP) as part of the EIA. The ESMMP covers many of the more specific plans and activities described as mitigations of impacts in the section above, such as the Biodiversity Management Plan, a Traffic Management Plan, or a Sustainable Water Resources Management Plan.

[1] *Technical Archaeological Report: Cave-1, Oko West, Guyana (VII-1:2 Oko) – Daggers & Plew, Feb 2025.*

ERM is of the opinion that the baseline information collected to date is sufficient to inform project permitting and has not identified any material issues of concern that would, at this stage, deem a project not viable in this location.

20.2 Environmental Permitting

20.2.1 Guyana Environmental Protection Act

In 1996, the Environmental Protection Act (EP Act) was enacted to implement the environmental provisions of the Constitution. The EP Act is Guyana's most significant piece of environmental legislation because it articulates national policy on important environmental topics, such as pollution control and the requirements for environmental review of projects that could potentially affect the environment. It also provides for the establishment of an environmental trust fund. Most importantly, the EP Act authorized the formation of the Environmental Protection Agency (EPA) and established the EPA as the lead agency on environmental matters in Guyana, including the issuance of environmental authorizations with appropriate conditions. The EP Act mandates the EPA to oversee the effective management, conservation, protection, and improvement of the environment (EPA 2021). It also requires the EPA to take the necessary measures to prevent and control pollution, assess the impact of economic development on the environment and the sustainable use of natural resources.

Regulations on hazardous waste management, water quality, air quality and noise management were established in 2000, pursuant to the EP Act. These pollution management regulations were developed to regulate the activities of development projects during the Construction and Operations stages. The following are regulations applicable to the Project under the EP Act.

20.2.1.1 EPA's Role in EIAs

The EP Act mandated four (4) functions for the EPA, which relate to environmental assessment. The EPA has determined that the development of a mine at this location will likely have significant environmental impacts. Four (4) functions of the EPA are consequently applicable to this Project, which are:

- To take such steps as necessary for the effective management of the natural environment to ensure the conservation, protection and sustainable use of natural resources.
- To promote the participation of members of the public in the process of integrating environmental concerns in planning for development on a sustainable basis.

- To ensure that any development activity which may cause an adverse effect on the natural environment is assessed before such activity is commenced and that such adverse effect is taken into account in deciding whether or not such activity should be authorized.
- To give development consent, which entitles the developer to proceed with the Project.

The EPA is required to implement several environmental management principles as part of this process. These principles are:

- The “polluter pays principle”: The polluter should bear the cost of measures to reduce pollution.
- The “precautionary principle”: Where there are threats of serious or irreversible damage, lack of full scientific certainty shall not be used to postpone measures to prevent environmental degradation.
- The “strict liability” legal principle: Any person who contravenes this Act or regulations shall be liable to the penalties prescribed thereafter.
- The “avoidance” principle: It is preferable to avoid environmental damage, as it can be impossible or more expensive to repair rather than prevent damage.
- The “state of technology” principle: The measures protecting the environment are restricted by what is technologically feasible, and as technology improves, the improved technology should be used.
- To prevent and repair environmental damage.

Prior owners, Reunion Gold, were required to obtain an environmental authorization (also commonly referred to as an Environmental Permit) from the EPA to conduct their current exploration activities. An Application for Environmental Authorization was filed with the EPA on October 17, 2022. After submission and review, the EPA issued a no-objection letter on January 18, 2023, and reconfirmed its decision by letter dated July 4, 2023. GMIN will be required to obtain a new Environmental Permit to eventually develop the Project. In September 2023, the Company filed an initial application and subsequently collaborated with the EPA to establish the Terms and Scope (ToS) of a future environmental impact assessment. As part of this process, the Company conducted meetings with both government agencies and local communities in the last quarter of 2023 to determine the essential elements to be incorporated into the ToS. The approval of the ToS was required for the Company to move forward with work on an environmental and social impact assessment (ESIA).

The EIA was formally submitted to Guyana’s EPA in late November 2024 and remains under review. Public consultation meetings were held in January and February 2025 in local communities, providing critical input for the development of environmental and social programs aligned with regional sustainability priorities. GMIN is currently finalizing responses to all requests for clarification and supplementary information, which

are expected to be submitted to the EPA by mid-May 2025. Final environmental approval and the construction permit are expected in Q2 2025.

Table 20.1 describes the legal framework for national environmental management related to mining operations, and Table 20.2 provides a summary listing of the major anticipated permits which are currently expected to be required based on the mine defined by the FS.

Table 20.1: Legal Framework for National Environmental Management in Relation to Mining

Title	Objectives	Implementing Agency	Comments	Applicability
Environmental Protection Air Quality Regulations	Establishes parameter limits with respect to emission of smoke, solid particles, sulfuric acid mist or sulfuric trioxide, fluoride compounds, hydrogen chloride, chlorine, hydrogen Sulphide, nitric acid or oxides of nitrogen and carbon monoxide.	Environmental Protection Agency	The construction, operation and closure phases of the project may impact air quality.	Environmental Protection Air Quality Regulations
The Environmental Protection Water Quality Regulations	Establishes guidelines to minimize the impacts of discharges on water quality.	Environmental Protection Agency	If the level of contaminants in effluent discharge from the project exceeds tolerable limits, water quality will be impacted. These regulations require registration and environmental authorization by any person whose construction, installation, operation, modification, or extension of any facility causes the discharge of effluents.	The project will entail discharges to surface water. The most stringent of the water quality standards outlined below will be applied during project execution
The Environmental Protection Noise Management Regulations	Ensures effective implementation of health and safety measures during operations and minimizes impacts on receptors in the site vicinity.	Environmental Protection Agency	Under these regulations, operations that emit noise in the execution of various activities such as construction, transport, industry, commerce, and any institution are required to apply to the Agency for an environmental authorization.	The project will generate noise. The regulations are consequently applicable to the project. Further, the most stringent of the following noise regulations will be applied during project execution.

Title	Objectives	Implementing Agency	Comments	Applicability
The Environmental Protection Hazardous Wastes Management Regulations	Ensures that, through the environmental authorization process, all operations that generate, transport, treat, store, and dispose of hazardous wastes are managed in a manner that protects human health and the environment.	Environmental Protection Agency	There are no regulations for the management of hazardous substances. It can be inferred that these regulations will apply to the reagents to be used in the project.	Gold will be beneficiated by cyanidation. This will entail the utilization of cyanide and other hazardous reagents in the processing operations. Petroleum products will be used for power generation and equipment operations. The chemical reagents and petroleum products will have to be managed in conformance with the EPA Hazardous Waste Management Regulations, 2000.
The Environmental Protection Authorizations Regulations	Mandated to keep records of all environmental monitoring (no less than 3 years), sampling procedures, maintenance and calibration procedures and all problems and malfunctions.	Environmental Protection Agency	Before an application for Environmental Authorization is made, an Environmental Impact Assessment or any other relevant document should be submitted to the EPA for approval.	The records should be project-specific and shall include: • Steps that should be followed to avoid, minimize, mitigate, and compensate for impacts. • Records of monitoring information such as date, place, and time of measurement, etc. • Proper maintenance of facilities and appropriate quality assurance procedures. • Establishment of an environmental monitoring programme.
The National Environmental Action Plan (NEAP), developed in 1994.	Aims for sound management of the environment and natural resources.	Environmental Protection Agency		The project must be undertaken based on sound environmental management practices. The National Environmental Action Plan is consequently considered applicable to this project.

Title	Objectives	Implementing Agency	Comments	Applicability
Use of Poisonous Substances	Mandates for the use of cyanide and requires that no operation in which cyanide or any preparation containing cyanide is used in the treatment of gold or other minerals may be commenced until the necessary buildings, structures, rooms, appliances and other arrangements to carry on these operations have been inspected and approved by a mines officer.	Guyana Geology and Mines Commission		The poisonous substances regulations apply primarily to the use of mercury and cyanide in the gold mining arena and stipulate that no person shall use elemental mercury or any form of mercury. Contravention of the regulations results in a Cease Work Order being placed on the operations on the claim or area until cleanup is done to the satisfaction of the Commissioner.
Mining Regulations	Provisions for the sanitary management of human waste and for the burial of animals and any person who expires at the mine.	Guyana Geology and Mines Commission		The Mining (Amendment) Regulations 2005 contain mandates for the following: • Use of Poisonous Substances. • Requirements for Environmental Management for Large and Medium Scale Mining. • Requirements for Environmental Management for Small-Scale Mining. • General Requirements. • Protected Areas. • Pollution Control. • Offences and Penalties.

Title	Objectives	Implementing Agency	Comments	Applicability
Guyana Geology and Mines Commission Act of 1979 and Mining Act of 1989	Provisions for a system of mineral agreements and licences for regulating prospecting. Provisions for the disposal of sanitary waste and the storage of poisonous substances in mining areas.	Guyana Geology and Mines Commission	Gives the Commission the responsibility for establishing regulations for mining and quarrying operations.	A mining licence is required to mine any mineral and is issued at the discretion of the Commissioner of the Guyana Geology and Mines Commission, with the Minister responsible for mining.
Occupational Safety and Health in Mining	Contains provisions applicable to the regulation of health and safety in mines.	Ministry of Labour, Human Services and Social Security, with the assistance of the International Labour Organization (ILO).		Draft OSH Regulation covers duties of employers and self-employed persons, owners of mines, manufacturers and workers. Reporting of dangerous situations and occurrences, administrative requirements such as the preparation, checking and maintenance of mine development plans in advance of mine development, as well as those for mine operations, inspections by mine operators, minimum age of workers and the maximum duration of work shifts, testing of shaft ropes and notice of death or injury are also covered.
Wild Birds Protection Act Chapter 71:01	Ensures the protection of certain wild birds.	Ministry of Legal Affairs		There are no provisions for the protection of wild birds relative to mining operations; however, conservation of biological resources is applicable to the project.

Title	Objectives	Implementing Agency	Comments	Applicability
Town and Country Planning Act Chapter 20:01	Provisions for the orderly and progressive development of land (which is specifically defined), Cities, Towns and other areas be they urban or rural, for their preservation and improvement and other related matters.	The Central Housing and Planning Authority, established under the Housing Act, Chapter 36:20.	This act would regulate any new settlement associated with operations.	
Pesticides and Toxic Chemicals Control Act No. 13 of 2000	The registration of Pesticides and Toxic Chemicals. The issuance of a licence to persons to import or manufacture registered pesticides and toxic chemicals.	Ministry of Agriculture	Regulations for the health and safety of workers against the risk of poisoning by controlled products when working in connection with the use of controlled products or when working on land or in any premises on or in which controlled products have been or are being used, stored or manufactured. Regulations imposing duties on the employers of workers, on the workers themselves and on others using controlled products and controlling and prohibiting the sale, use, disposal and transportation of any toxic chemical or pesticide.	This project will entail the importation of cyanide and other chemical reagents for use in beneficiation, some of which can be classified as toxic chemicals.

Title	Objectives	Implementing Agency	Comments	Applicability
National Biodiversity Action Plan	To integrate the implementation of the Convention on Biological Diversity into national development.	Ministry of Natural Resources and the Environment	The overall goal of the NBAP was “to promote and achieve the conservation of Guyana’s biodiversity, to use its components in a sustainable way, and to encourage the fair and equitable sharing of benefits arising out of the use of Guyana’s biodiversity”.	The mining operation and access roads will be in forested areas and will impinge on biological diversity. Therefore, according to NBAP 1, consideration should be given to the adoption of their four (4) approaches when planning for the management of biodiversity.
Public Health Act Cap. 145	Sanitary control of mining districts or parts thereof, and the health and welfare of the inhabitants.	Ministry of Health		The provisions of the Public Health Act will be factored into the employee health and safety plan for the mine site operations.
Forest Act 2008	Governs all activities that are carried out in the forest, including mining and associated activities.	Guyana Forestry Commission		Under this Act, no person shall engage in the following unless a mineral prospecting or mining licence has been granted under the Mining Act 1989: • Enter and occupy the state forest. • Cut, damage, or take any forest produce or carry on any kind of forest operation. • Carry out any kind of exploratory operation in a state forest.

Title	Objectives	Implementing Agency	Comments	Applicability
Guyana Forestry Commission Act 2007	To provide advice on various forest-related issues and on the formulation of forest policy. To prepare plans, codes of practice, and guidelines for the conservation and management of forests. To research, collate, analyze, prepare, and disseminate data, statistics, and other information about forests and all aspects of forestry, including forest ecology and the use of forest produce. To inspect, certify and accredit services for quality control of forest produce.	Guyana Forestry Commission		The project will entail management and reclamation of forest resources, and this act must therefore be adhered to.
Code of Practice for Timber Harvesting 2002	Provides guidelines and standards on what constitutes satisfactory road building and other related forestry activities.	Guyana Forestry Commission		One important component of this project is the construction of an access road to the mine site. This road will, at a minimum, have to conform to the GFC Road Planning guide as outlined in the Code of Practice for Timber Harvesting.

Title	Objectives	Implementing Agency	Comments	Applicability
Low Carbon Development Strategy 2030	Intends to create a new low-carbon economy in Guyana by establishing incentives which value the world's ecosystem services and promoting these as an essential component of a new model of global development with sustainability at its core. In Guyana's case, harnessing the value of the country's ecosystem services can build a long-term, low-carbon diversification opportunity.	Office of the President	The LCDS 2030 sets out how to accelerate the achievement of these objectives: • Forest Climate Services and other Ecosystem Services. • Stimulate future growth through clean energy and sustainable economic activities. • Protect against climate change. • Aligning with global climate goals in determining how to achieve this and planning for sustainable development is the core principle that guides the LCDS 2030.	The LCDS 2030 will ensure that Guyana creates an ecosystems economy which balances the following: low-impact mining and forestry to enhance employment and income generation opportunities; forest climate services through which the value provided by Guyana's forests to the world is recognised; Guyana's next generation of ecosystem services, such as water management, and biodiversity protection.
Land Permit on Private Lands	The owner of any private lands, granted before the passing of the Mining Ordinance, 1903, shall hold and enjoy all metals other than gold and silver therein or thereon, and may after obtaining a licence or permit under this Act search or mine for them in accordance with this Act and the licence and, when found, take and appropriate them to his own use.	Guyana Geology and Mines Commission	The Commission may, with the approval of the Minister and subject to section 8, grant a licence or permit under this Act authorizing the holder of the licence to enter on private lands and search or mine for, take and appropriate, any minerals.	Before the Mining Ordinance of 1903, private landowners could use any metal, other than gold or silver, found in valuable minerals on their lands. However, they must purchase the metal from the landowner at an agreed price or pay the Commission's stated net value, without deductions for gold or silver or both. The owner must comply with current laws for recording gold or silver extracted from minerals, removing it, and paying royalties.

Title	Objectives	Implementing Agency	Comments	Applicability
Electrical Installation and Inspection Licence	Set up to protect users of electricity against the hazard of unsafe and unsound electrical installations.	Government Electrical Inspectorate	Issuance of Permits to Perform Electrical Installations. Issuance of Certificates for code-compliant electrical installations. Inspection and Testing of Electrical Installations. Licensing of Qualified Electrical Contractors.	The mission of the Government Electrical Inspectorate (GEI) is to promote the awareness of the Electricity Sector Regulations Act (ESRA 2008), the National Electrical Code (NEC) as well as good Electrical safety practices, while providing electrical contractors and other personnel with practical advice in carrying out electrical works in compliance to the relevant codes and/or regulations.
Bulk Fuel Storage Licence / Consumer Installation Licence	Tasked with enforcing the Laws and Regulations governing all activities involving petroleum and petroleum products in Guyana.	Guyana Energy Authority (GEA)	Licences are issued after a site visit is conducted to ensure compliance with the Regulations and approved standards.	The Bulk transportation Licence is issued to a person who owns a vehicle capable of transporting an aggregate quantity of 2,000 litres of Petroleum and petroleum products.
Installation Permit	For the installation of water wells.	Guyana Water Inc., Hydromet (Dept. of the Ministry of Agriculture)		
Wharf Facility Permit	For the utilization or commencement of the construction of a marine structure for the purpose specified.	Maritime Administration Department (Regional Democratic Council)	Renewed annually.	
Tower Installation		Guyana Civil Aviation Authority, Guyana Lands and Survey Commission		

Title	Objectives	Implementing Agency	Comments	Applicability
Explosives - Use Permit	This is covered under the Blasting Operations Act, which states that a person holding a Certificate of Competency granted by the GGMC is deemed suitable by the commission after examination.	Guyana Geology Mines Commission	Blasting Operations Act (Laws of Guyana).	
Radio Frequency Licence (Ground to Aircraft Communication)	Regulation in terms of spectrum management is necessary to ensure that this resource is managed and utilized most efficiently and effectively for the country's benefit.	Telecommunications Agency	Renewed Annually on or before the Anniversary.	
Frequency Allocation Permit (Licence for the Installation and Operation of Radio)		Telecommunication Agency		

Table 20.2: Required Permit Overview

No.	Permit	Regulatory Body*	Legislation
1	Environmental Permit	EPA	Environmental Protection (Authorizations) Regulations 2000
2	Hazardous Waste Permit	EPA	Environmental Protection (Authorizations) Regulations 2000
3	Operation Permit	EPA	Environmental Protection (Authorizations) Regulations 2000
4	Mining Licence	GGMC	Mining (Amendment) Regulations 2005
5	Cyanide Permit	GGMC	Mining (Amendment) Regulations 2005
6	Water Permit (Water Quality Regulation)	EPA	Guyana Water Authority Act (Amendment) 1997
7	Land Permit	GLSC, GGMC	Guyana Mining Act (Amendment 2010) 1989
8	Electrical Installation and Inspection Licence	GEI	Electricity Sector Regulations Act (ESRA 2008)
9	Aerodrome Licence	GCAA	Civil Aviation Act (Cap. 53:01)
10	Export Licence - Gold	Ministry of Business Investment, Guyana Gold Board	Guyana Gold Board Act Cap. 66:01
11	Bulk Fuel Storage Licence / Consumer Installation Licence	Guyana Energy Authority (GEA)	a) Guyana Energy Agency Act 1997 b) Guyana Energy Agency (Amendment) Act 2004
12	Well Installation Permit	GWI, Hydromet (Dept. of Min. of Agriculture)	
13	Wharf Facility Permit	MARAD – (RDC)	
14	Wharf Facility - Lease Land	GLSC	
15	Tower Installation	GCAA. GLSC	
16	Explosives - Use Permit	GGMC	Blasting Operations Act (Laws of Guyana)
17	Radio Frequency Licence (Ground to Aircraft Communication)	Telecommunications Agency	
18	Frequency Allocation Permit (Licence for the Installation and Operation of Radio Equipment)	Telecommunications Agency	
19	Watercourse Diversion Permit	EPA	Environmental Protection (Authorizations) Regulations 2000

No.	Permit	Regulatory Body*	Legislation
20	Radiation Permit	EPA	Environmental Protection (Authorizations) Regulations 2000
21	Transmission Line	Ministry of Natural Resources	A Component of Building the Mine, Covered under the Mineral Agreement
22	Deforestation No Objection (No Permit Needed)	GFC	

**Note(s):*

1) *EPA Environmental Protection Agency*
2) *GCAA Guyana Civil Aviation Authority*
3) *GEA Guyana Energy Authority*
4) *GEI Government Electrical Inspectorate*
5) *GGMC Guyana Geology and Mines Commission*
6) *GLSC Guyana Lands and Surveys Commission*
7) *GWI Guyana Water Inc.*
8) *MARAD Maritime Administration Department*
9) *RDC Regional Democratic Councils.*

20.2.2 Regulatory Timelines

Figure 20.1 is a schematic representation of the EIA process for Guyana, governed by the Environmental Protection Agency (EPA). It is ERM's understanding that regulatory timelines are not legally mandated.

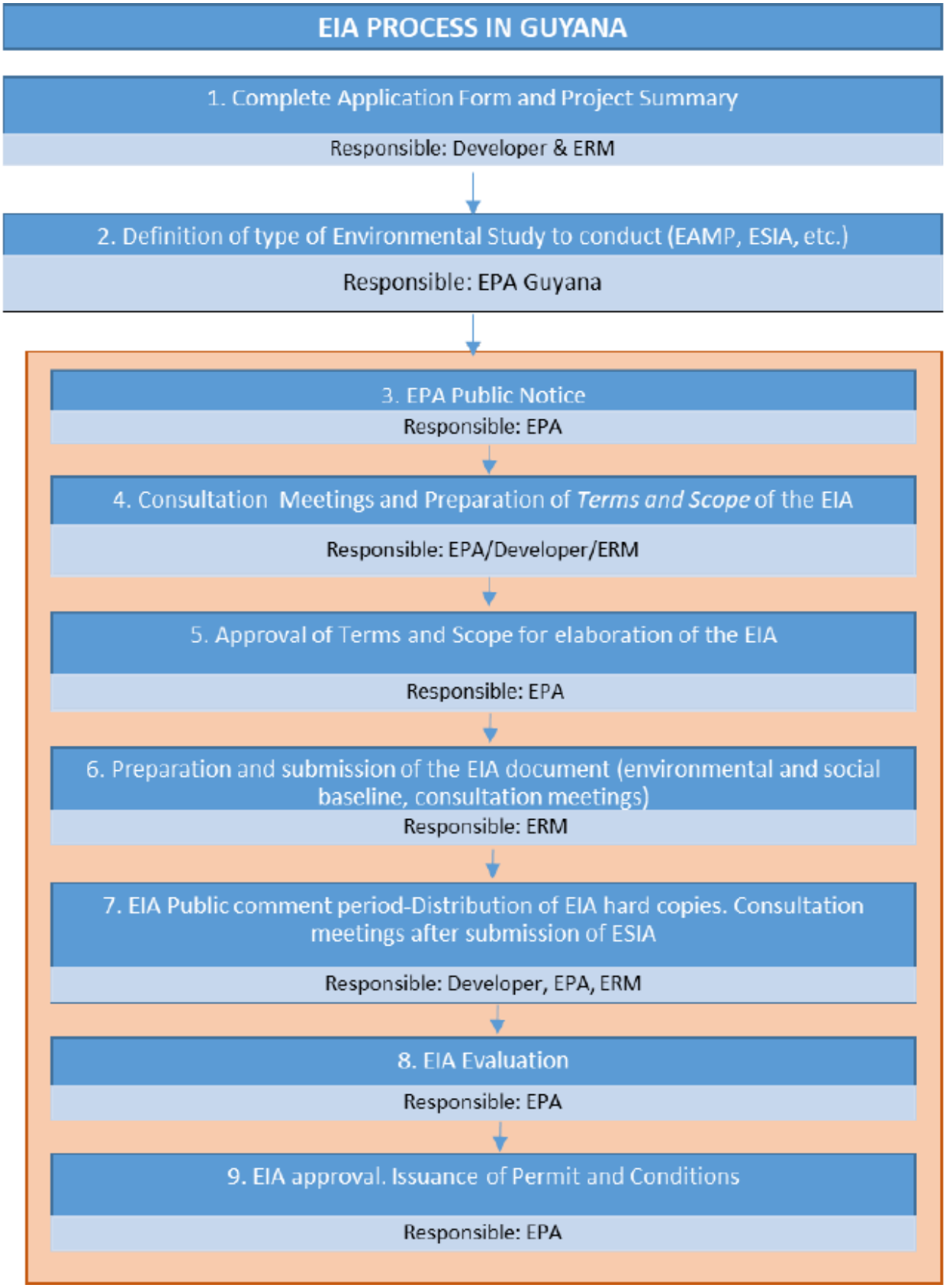
The EIA was submitted to the EPA on November 27, 2024, and public disclosure meetings were held in Bartica, Itaballi, and Puruni in late January 2025, as well as with the EPA in Georgetown. GMIN received an interim environmental permit (IEP) from Guyana's EPA in response to the ESIA submission on January 7, 2025. The EPA provided comments in late March 2025, with the final EIA to be submitted in May 2025. The IEP is valid for one (1) year and, together with the "No Objection" letter received from the Guyana Forestry Commission, GMIN has fulfilled the requirements to initiate early works construction activities on the barge Landing facility, main access road and internal roads, airstrip, permanent camp facility, water and sewage treatment plants, power generation and the communication tower.

At the time of writing, the EPA has completed reviewing and, following the questions and clarifications, a resubmittal will occur in mid-May 2025. The decision that will follow is Stage 9 (final) of the process shown in Figure 20.1, whereupon the Full Environmental Permit will be issued. Following this, GMIN will be able to proceed with the formal construction decision and continuation of detailed engineering as construction works progress.

In parallel, GMIN has initiated applications for other key regulatory authorizations required for the Project's implementation, including the Mining Licence, port operation, permits for fuel use and storage, and approvals for the installation of transmission and telecommunications towers. These complementary

permits will support full-scale construction and operational readiness. All permitting efforts are guided by proactive stakeholder engagement and adherence to international environmental and social performance standards.

Figure 20.1: EIA Process for Guyana



20.3 Mine Closure

Mine Closure Planning at the Feasibility Stage of the Project aims to embed best practice whilst also satisfying national legislation and guidelines, generally following the format outlined in ICMM (2019), incorporating elements from other good international industry practice. The following aspects are covered in this report: Regulatory Setting, Closure Vision, Principles and Objectives, Closure Activities, Closure Criteria and Post-Closure Maintenance and Monitoring. At the Feasibility Stage, the Closure Criteria are yet to be fully developed and should be done in the next iteration of the Closure plan. Closure cost estimates for the plan outlined in this section are provided in Section 21.3.

20.3.1 Regulatory Setting

International good practice for closure planning can be established by reference to various documents, including:

- Sustainability Policy and Standards – (IFC, 2012).
- IFC – Environmental, Health and Safety Guidelines for Mining – (IFC, 2007).
- Integrated Mine Closure Good Practice Guide 2nd Edition (ICMM, 2019).
- Financial Assurance for Mine Closure and Reclamation (ICMM, 2006).
- Tools for Social Performance (ICMM, 2022).
- Mine closure and reclamation planning – Requirements – ISO 21795-1 (International Standard, 2021).
- Mine closure and reclamation planning – Guidance – ISO 21795-2 (International Standard, 2021).

The national legislation underpinning the Project is set out by the Guyana Geology and Mines Commission (GGMC). The 2005 Amended Mining Regulations and the 1989 Mining Act apply to the Project. The GGMC granted Reunion Gold Inc. a Prospecting Licence (PL) for the Project, now combined with and named as G Mining. Once the EIA is approved by the EPA and GGMC, a mining licence would be issued to G Mining to recover gold in the Oko West area. GGMC would be responsible for enforcing the provisions in this regulation.

The GGMC has a Code of Practice for Mine Reclamation and Closure Plans (August 2010).

1. Ensuring that mine sites are restored to a satisfactory condition through eliminating unacceptable health hazards, limiting production and release of environmentally damaging substances, long-term

efforts to reduce maintenance and monitoring, visual restoration of the Site and reclaiming areas where infrastructures are located for future use.

2. Improve the level of awareness and education of all parties on the importance of incorporating the reclamation planning stage of a mine operation.
3. Ensure that reclamation and closure practices are adaptive and dynamic over time.
4. Ensure that reclamation and closure practices are credible and verifiable.

20.3.2 Closure Vision and Principles

At the conceptual level, a preliminary closure vision has been proposed and reads as follows:

"Closure will strive to leave a positive economic, environmental and social legacy, through the integration of biodiversity enhancement with existing socio-economic activity, and create a sustainable future for all stakeholders."

The closure vision will be further refined in future stages of the closure plan development.

Closure Principles are guiding statements for reclamation and closure planning and provide the general framework for Closure Objectives. Suggested principles detailed by ICMM have been adopted for this plan, including (but not limited to): Safety, Physical Stability, Chemical Stability, Ecological Stability, Socio-economic Transition, Risk Limitation, Cost-effectiveness and Long-Term Care.

20.3.3 Closure Objectives

Closure objectives are overarching aims associated with the closure of an operation that expand on components of the vision. At the conceptual stage, preliminary closure objectives to achieve include Physical Stability, Geochemical Stability, Enhance Biodiversity, Health and Safety, Water Quality, Socio-economic Transition and Regulatory.

20.3.4 Closure Activities

Closure activities are the physical tasks undertaken to close an operation that have been determined will achieve the closure objectives and satisfy the success criteria. These activities can either be undertaken during operations (progressive closure) or after operations (final closure and post-closure).

20.3.4.1 Mining Areas

The active mining areas for this project will include an open pit and an underground mining domain. Upon cessation of mining activities, the open pit will be flooded by natural groundwater inflow and surface water run-off. Safety measures such as raising berms, fencing and signage will also be installed around the perimeter of the pit. The lake forming in the pit is envisaged to create a new ecosystem, which could be connected to existing surface water bodies and help promote biodiversity in the area. Geochemical studies will be undertaken throughout the project lifecycle of the mine to help define water quality upon closure and determine whether actions are needed to prevent water from becoming acidic. For the underground mine, the proposed closure activities will involve backfilling the mined-out open stopes with Cement Rock Fill (CRF). At the current stage, the portal and ventilation backfill will require more detailed studies in order to define their closure scenarios. All equipment shall be removed, and an audit of materials, products and plant remaining underground will need to be undertaken prior to closure.

20.3.4.2 Waste Storage Facility (WSF)

The Project's Waste Storage Facility is located directly adjacent to the pit for optimal haulage during the project lifecycle. Geochemical studies undertaken to date indicate that only relatively small volumes of the transitional and sedimentary rocks are potentially acid-generating (PAG) material, and the majority is not acid-generating (see the Geochemistry sub-Section 20.1.1.1). Waste which has the potential to generate acid will be encapsulated within the WSF with appropriate material to avoid potential impacts from acid and metalliferous drainage. The WSF will also host some non-hazardous waste that will be contained in a similar way and will be engineered to minimize infiltration and facilitate leachate collection if needed.

The WSF height will not exceed 178 m of elevation, and waste will be disposed of in 20 m high benches and 10 m wide catch berms, resulting in an overall slope of 1V:2.5H (21.8°). This will be the final overall slope angle, with bench regrading taking place progressively throughout operations to minimize the intermediate slopes that will be captured as operational expenditure. It is expected that five (5) years out from closure, progressive closure will commence and involve the placement, spreading and scarification of topsoil, followed by revegetation. These activities will account for half of the closure effort for the WSF, with the other half taking place throughout the five-year closure period. The closure cost estimate and schedule reflect these actions and the timeline.

20.3.4.3 Tailings Storage Facility (TSF)

The TSF will be designed considering closure slopes, therefore, no regrading of the embankments will be required during closure. Due to the significantly low bearing capacity of the tailings, conventional capping,

covering and revegetation of the TSF surface is not a feasible option. Current planning indicates that the TSF will require perpetual water and geotechnical maintenance and monitoring, as well as perpetual security measures to prevent artisanal mining of the tailings. Continuous alternative closure options assessments will be required to explore optimized solutions for closure of the TSF, and may include transitioning to dry-stack tailings, or complex engineered cover designs such as iteratively layering waste rock and geotextiles to provide bearing capacity.

The current closure concept assumes that an effluent treatment plant (ETP) will not be required, due to there being evidence that the greater proportion of tailings will not be PAG, that the cyanide destruction plant will reduce concentrations to acceptable levels, and that these will be further attenuated by natural degradation processes and diluted by surface water run-off within the large TSF catchment. This assumption will need to be further investigated and tested, and in the event that tailings display ML/ARD characteristics at the time of closure, costs will need to be recalculated to reflect this change.

20.3.4.4 Processing Plant and Supporting Infrastructure

At the current stage of the Project, it is assumed that several pieces of infrastructure will remain in place as part of the post-mining land use (PMLU). The preliminary list of structures that are planned to be retained is the osmosis plant, sewage treatment plant, access road, airstrip, barge landing and the power plant. Buildings and infrastructure that will be demolished will follow a proposed closure base-case procedure. This will consist of all foundations being removed, voids backfilled and covered with topsoil, decontamination of concrete slabs, if necessary, onsite soil used to cover slabs, removal of surface pipelines (unless used for PMLU), removal and reuse of salvageable material, disposal of non-recyclable waste and hazardous material and final grading and revegetation.

20.3.4.5 Water Management Infrastructure

Water management infrastructure will undergo several stages of closure. The main aim of closure for the water management infrastructure is to ensure that contact water is minimized and that long-term maintenance will not be required based on available geochemical information on local hydrology. Water infrastructure related to the cyanide destruction circuit will be demolished. Sediment ponds will be covered and revegetated once water quality has achieved legal requirements, and any potentially hazardous material identified during the closure process will be removed and disposed of in an approved disposal facility. It is assumed at the current stage that the osmosis plant, treated water storage reservoir and sewage treatment plant will remain operational as part of the PMLU.

20.3.4.6 Contaminated Sites

Site investigations should be carried out to identify potentially contaminated sites for decommissioning and restoration, progressively in operations and in closure. These investigations could likely include, but not be limited to, the processing plant and reagent storage areas, process plant, power station, fuel storage areas, explosives storage areas, process plant workshop, warehouse and laboratory, and hazardous waste temporary storage areas.

20.3.4.7 Waste Management

Waste management will focus on reusing and recycling materials where possible. Non-hazardous waste will be segregated as much as possible to facilitate recycling, and a licensed contractor will be hired to handle the final disposal of recyclable and saleable waste. A non-hazardous waste storage area will be constructed with curbs to prevent run-off to permeable areas, and run-off and leachate will be collected and treated prior to discharge. For non-hazardous waste that cannot be recycled or reused, there will be one or more landfills within the project area. These will be created within the WSF areas and will be constructed and operated according to best practices. This includes engineering efforts to minimize infiltration and facilitate leachate collection if needed. Hazardous waste will not be disposed of in the on-site landfill(s). Hazardous waste generated by the Project will be temporarily stored on site pending off-site shipment for treatment and/or disposal at an approved facility by a licensed contractor.

20.3.4.8 Stakeholder Engagement

Stakeholder engagement on closure should continue throughout the entire mining lifecycle, with an approach that empowers the local community in the decision-making process. Key stakeholders to be involved include, but are not limited to, local community members, indigenous peoples, workforce, governmental authorities, industry peers, other sectoral contributors, and possibly academics and the media. A Project-specific engagement strategy will be designed and implemented during the Project's lifetime, noting that local communities must be involved as early as possible in the development of the closure vision and PMLUs.

20.3.5 Post-Closure Maintenance and Monitoring

Closure monitoring and maintenance will encompass physical aspects of the Project, such as landforms, water, and biodiversity. The objective of long-term monitoring is to validate the fulfillment of closure success criteria. Closure monitoring and maintenance will cover many areas of the Project, and the likely domains are described as follows.

Geotechnical Stability will be monitored through a geotechnical monitoring programme. This will be supported by visual inspections, annual site inspection by a qualified professional, and instrumentation. Monitoring points and specific instruments will be installed in each area of geotechnical risk as part of closure.

Reclamation Monitoring involves monitoring the success of reclamation during and after closure, including inspection of the establishment and evolution of vegetative cover. The cover system will be monitored through visual inspections to identify signs of erosion and soil loss. Soil sampling can be undertaken, if necessary, along with quarterly inspections of the final cover.

Water Monitoring will be undertaken through closure and post-closure in line with the water management plan. This will specify monitoring points, parameters and the frequency of measurement. Water quality of the pit will continue to be monitored during the post-closure phase prior to discharge, to ensure that water has achieved the permit's conditions.

Biodiversity Monitoring will be developed through the Biodiversity Management Plan. The program will define monitoring points, parameters and monitoring frequency and will begin during the progressive closure. It will incorporate the results to enhance the closure design and minimize the monitoring program during the post-closure phase.

TSF Monitoring will encompass all the above-mentioned monitoring; however, whilst other site facilities will be monitored to meet closure criteria and result in relinquishment, the TSF will be monitored in perpetuity. This will result in more extensive requirements, such as security measures to prevent artisanal mining, more stringent climate change risk assessments, geotechnical inspections after significant rainfall events, and downstream water monitoring to validate containment of the tailings. Alternative closure concepts will be consistently explored throughout the project lifecycle to explore optimized solutions for the TSF, to reduce the monitoring period and facilitate relinquishment.

20.3.6 Closure Schedule

Three (3) approaches to closure will be undertaken during the project's life. **Progressive Closure** should be undertaken at the earliest opportunity for those domains (area or areas that have distinct and unique characteristics for the purposes of closure) that will end operational life throughout the LOM and for those components whose configuration allows them to be closed progressively while they are still operating. **Final Closure** will start immediately after the cessation of mining operations and is anticipated to take five (5) years. **Post-Closure** refers to the monitoring and maintenance phase after closure has finalized. It

is suggested to have a ten-year monitoring and maintenance period for most domains, and maintenance and monitoring in perpetuity for the TSF.

There will be four (4) phases to closure described below, and as shown in Table 20.3

- Phase 1 – Progressive closure of the WSF and implementation of social engagement and transition plans in the preceding five (5) years to final mining activity.
- Phase 2 – Closure of the underground, open pit, TSF and WSF in the five (5) years following the cessation of mining activity.
- Phase 3 – Closure of processing infrastructure, accommodation camp and haul in the three (3) years following the cessation of processing activity.
- Phase 4 – Maintenance and monitoring activities carried out in perpetuity for the TSF (defined as 30 years for the purposes of costing and scheduling), and for ten (10) years post-closure for all other domains.

Table 20.3: Closure Schedule (Years of Life of Mine)

	1	…	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	…	48
P1																									
P2																									
P3																									
P4																									

21. CAPITAL AND OPERATING COSTS

Life-of-mine project capital costs are estimated to total $1.7 billion, consisting of the following three (3) distinct phases:

- Initial Capital Expenditure – This phase includes all costs to develop the property with a process plant designed to nominally treat a 7 Mtpa blend of saprolite, transition and fresh rock. Initial capital costs total $1,041 million (including $85 million for contingency and excluding $69 million in pre-production revenue), which will be expended over a 34-month engineering, construction, pre-production and commissioning period.
- Sustaining Capital Costs – This phase includes all costs related to the acquisition, replacement, or major overhaul of assets required to develop the underground mine and sustain operations. The plant will be configured to treat 100% fresh rock at 6 Mtpa. Sustaining capital costs are estimated to be $650 million, including indirect costs but excluding a contingency.
- Closure Costs – This phase includes all costs related to the closure, reclamation, and ongoing monitoring of the mine after operations. Closure costs add up to $39 million and include a 20% contingency.
- The capital and sustaining expenditures are summarized in Table 21.1 according to the Level 1 work breakdown structure (“WBS”).

Table 21.1: Initial and Sustaining Capital Expenditures Summary

Capital Expenditures (K$)	Initial Capital Cost	Sustaining Capital Cost	Closure Cost	Total Capital Cost
100 – Infrastructure	105,889	13,404		119,293
200 - Power and Electrical	110,797	28,258		139,054
300 – Water Management	22,882	14,012		36,893
400 – Surface Operations	41,051	-		41,051
500 - Mining	104,078	575,085		679,162
600 - Process Plant	191,639	1,655		193,295
700 - Construction Indirect	134,534	-		134,534
800 - General Services / Owner’s Cost	137,385	-		137,385
900 - Pre-production, Start-up, Comm.	108,173	17,589		125,762
990 - Contingency	84,962	-		84,962

Capital Expenditures (K$)	Initial Capital Cost	Sustaining Capital Cost	Closure Cost	Total Capital Cost
Closure Cost			38,696	38,696
Total	**1,041,389**	**650,003**	38,696	**1,730,088**

The OPEX includes mining, processing, power, general services and administration (G&A), transportation, refining and royalties. Power costs are separated from processing costs. The average OPEX is $958/oz. The all-in sustaining costs (AISC), which include closure, reclamation and sustaining capital costs, average $1,123/oz.

Table 21.2: Operating Costs

Operating Costs	$/t milled	$/oz
Open Pit Mining Cost	13.81	243
Underground Mining Cost	10.34	182
Processing Cost	7.24	128
Power Cost	7.95	140
General & Administration Cost	5.49	97
Refining Cost	0.45	8
Total Site Cost	**45.29**	**798**
Royalty Cost	9.05	160
Total Operating Costs	**54.34**	**958**
Sustaining Capital	8.56	151
Closure Cost	0.51	9
Land Payments	0.29	5
All-In Sustaining Costs (AISC)	**63.70**	**1,123**

21.1 Initial Capital Expenditures

21.1.1 Basis of Estimate

The capital cost estimate is according to AACEI standard Class 3 and is accurate to a -5% / +10% range. The base date of the CAPEX estimate is H1-2025. The initial capital expenditure (“CAPEX”) duration is planned over a period of 34 months, assumed from March 2025 to the end of December 2027, which includes 3 months of commissioning in the last quarter of 2027. The CAPEX estimate is aligned with an owner-managed project delivery model.

The initial CAPEX estimate is presented in US dollars, and no allowance for escalation or exchange rate fluctuation was used. An exchange rate of 1.40 (C$:US$) has been applied as necessary. Expenditures are planned in several other currencies, with the native currencies retained within the overall estimate.

The capital cost estimate is a comprehensive, bottom-up analysis developed by major facilities and disciplines, drawing on G Mining's internal database of completed projects and studies, as well as experience with similar operations. In certain cases, detailed cost breakdowns by cost type, including labour, materials, construction equipment, consumables, and services, were prepared based on material takeoffs from drawings and design concepts. In line with standards defined at the start of the Project, the pricing of equipment, materials, and labour was estimated according to the following guidelines:

- Infrastructures were developed based on approved general arrangement drawings, leading to MTO.
- The concrete and structural costs for these infrastructures were based on executed material take-offs (MTOs), using unit costs per cubic metre of concrete and per tonne of steel, respectively. The cost per cubic metre of concrete was updated using purchase order data and material quotations received in Guyana. For structural steel, budgetary quotations were obtained from experienced companies with whom GMS has previously executed projects. These quotations underwent commercial and technical evaluations to determine pricing by steel type.
- Earthworks quantities and deforestation were based on take-offs from the general arrangement drawings and detailed drawings, with unit costs from firm quotation and benchmarked productivity from executed projects.
- Equipment proposals were specifically specified and quoted for the Project. Prices for minor equipment items, representing less than 8% of the total equipment value, were sourced from recent projects or drawn from GMS's current cost database.
- Specific material prices, such as those for sheet piling and aggregate, were based on firm quotations received directly from suppliers.
- Labour rates were derived from the TZ operation, the GMS expatriate database, and wage surveys conducted in Guyana during the FS study. The labour strategy and construction equipment costs for the Project were developed through separate analyses, specifically for the indirect costs.

21.1.2 Initial CAPEX Summary

The initial CAPEX, excluding pre-production revenue, is estimated at $1,041 million. After accounting for pre-production revenues, the CAPEX is reduced to $972.6 million. The capital expenditure is summarized in Table 21.3.

According to the Level 1 Work Breakdown Structure (WBS), Areas 100 to 600 represent the Project's direct costs, while Areas 700 to 900 encompass indirect costs, owner's costs, and pre-production costs. Pre-production revenues are estimated at approximately $69 million, based on the recovery of 30 koz of gold during the commissioning phase.

The CAPEX includes a contingency of $85M, which is 9% of the total before contingency or 14.7% of the direct costs for WBS areas 100 to 600. The total labour hours for the initial CAPEX phase are estimated at 8.7M hours.

Table 21.3: Initial Capital Expenditures Summary

Level 1 WBS Areas	Initial Capital Cost (K$)
100 – Infrastructure	105,889
200 - Power and Electrical	110,797
300 – Water Management	22,882
400 – Surface Operations	41,051
500 – Mining	104,078
600 - Process Plant	191,639
700 - Construction Indirect	134,534
800 - General Services / Owner's Cost	137,385
900 - Pre-production, Start-up, Comm.	108,173
990 – Contingency	84,962
Total	**1,041,389**

21.1.3 Infrastructures

The CAPEX estimates for WBS 100 – Infrastructure are summarized in Table 21.4. The detailed description of infrastructures is presented in Section 18.

Table 21.4: Infrastructure Capital Expenditures

Capital Expenditures (K$)	
110 – Roads, Bridges and Fencing	**19,257**
111 – General Earthwork (DEFORESTATION)	5,339
112 – Site Roads	6,331
113 – External Site Roads	3,743

Capital Expenditures (K$)	
114 – Bridges	676
115 – Site Drainage & Trenches	808
116 – Fencing	979
117 – Airstrip	1,382
120 – Mine Infrastructure	**22,190**
121 – Mine Dry (Sustaining)	-
122 – Mine Maintenance Facility	15,534
123 – Wash Bay	1,210
124 – Explosive Magazine	1,071
125 – Component Maintenance Facility	850
126 – Emulsion Plant	3,525
130 – Support Infrastructure	**9,624**
131 – Administrative Building	3,619
132 – Site Guard House	262
135 – Laydown	510
136 – Core shack (Sustaining)	-
137 – Assay Lab	5,233
140 – Camp Facilities	**23,943**
141 – Camp Dorms	18,107
142 – Kitchen	3,821
143 – Camp Office / Welcome Centre / Laundry / Recreational Room	1,470
146 – Recycling / Sort Facility	286
148 – Sports Field	259
160 – Process Plant Infrastructure	**6,413**
161 – Process Search House	351
162 – Leach Remote Lab	278
164 – Reagents Storage Building	1,220
165 – Mill Office	2,293
166 – Workshop	1,465
167 – Reagents Warehouse	806
170 – Fuel Systems Storage	**9,852**

Capital Expenditures (K$)	
171 – Heavy and Light Vehicle Fuel Storage	3,760
172 – Oko Wharf Fuel Storage LFO	3,073
173 – Oko Wharf Fuel Storage HFO	3,019
190 – Offsite Infrastructure	**14,609**
191 – Oko Wharf Infrastructure	1,073
192 – Oko Wharf Lunch Room	868
193 – Oko Wharf	4,042
194 – Oko Wharf Laydown / Reagent Storage	4,593
195 – Oko Wharf Emergency Service Shed	214
196 – Oko Wharf Offices / Guard House / Restroom	256
197 – Oko Wharf Warehouse	1,244
198 – Oko Wharf Truck Shop	2,023
199 – Oko Wharf Laundry	296
Total	**105,889**

21.1.4 Power Supply and Electrical

The CAPEX estimates for WBS Area 200 – Power Supply and Electrical are summarized in Table 21.5.

Table 21.5: Power Supply and Communications Capital Expenditures

Capital Expenditures (K$)	
210 – Main Power Generation	**61,217**
220 – Secondary Power Generation	**4,752**
223 – Kitchen Power Generation	105
224 – Emergency Power Generation	3,428
225 – Oko Wharf Power Generation	1,219
230 – Water Management Electrical Room	**1,783**
232 – Water Management E-room	428
233 – Creek Pumping Station	286
234 – Sewage Water E-room	284
235 – Mine Service Event Pond E-room	784
240 – Service Electrical E-room	**155**

Capital Expenditures (K$)	
250 – Mine Electrical Room	**262**
260 – Process Plant Electrical	**31,133**
261 – Crushing E-room	3,419
262 – Grinding E-room	10,769
263 – CIP & Detox E-room	9,466
264 – Lixiviation E-room	1,223
265 – Pumping Station (TSF Barge) E-room	1,516
266 – Refinery E-room	3,245
269 – Crushing E-room	1,494
270 – Medium Voltage (MV) Distribution O/H Line	**4,709**
280 – Automation Network	**820**
281 – Automation Network	620
283 – Instrumentation for Process Plant	200
290 – IT Network & Fire Detection	**5,966**
291 – IT Network	1,633
292 – IT Hardware & System Communication, Fibre	2,032
293 – Fire Detection Network	1,410
294 – Security Network	852
295 – Server Room	39
Total	**110,797**

21.1.5 Water Management

Capital expenditures summary for water management is presented in Table 21.6. Tailings storage facility construction costs are based on a material take-off (MTO) approach, supported by TEC3.

Table 21.6: Water Management Capital Expenditures

Capital Expenditures (K$)	
310 – Fresh Water Intake / Wells	**818**
320 – Water Ponds and Water Management	**4,854**
330 – Domestic Water	**1,980**
340 – Sewage Water	**2,784**
350 – Fire Protection	**4,465**
360 – Effluent Water Treatment	**-**
370 – Tailings Storage Facility (TSF)	**7,982**
371 – Tailings Storage Facility - Bassin Preparation (Foundation)	385
373 – Tailings Storage Facility - Starter Dam	7,359
375 – Tailings Storage Facility - South Dike	238
Total	**22,882**

21.1.6 Surface Equipment

Surface equipment includes site support equipment, construction mobile equipment, a concrete batch plant, and an aggregate plant. A summary of the capital expenditures is presented in Table 21.7. The construction mobile equipment category includes the purchase costs for lifting equipment, utility vehicles, and specialized construction machinery.

Table 21.7: Surface Equipment Capital Expenditures

Capital Expenditures (K$)	
410 - Surface Operations Equipment	**36,605**
411 – Construction Mobile Equipment	14,440
412 – Process Plant Mobile Equipment	38
414 – G&A Mobile Equipment	2,928
415 – Off-Site Surface Ops Equipment	11,260
416 – Marine Equipment	7,939
430 – Concrete Batch Plant	**902**
460 – Borrow Pit	**976**
480 – Aggregate Plant	**2,568**
Total	**41,051**

21.1.7 Mining

The capital costs estimate for the mining areas is presented in Table 21.8. The costs are based on an owner-operated mine fleet.

Table 21.8: Mining Capital Expenditures

Capital Expenditures (K$)	
540 – Mine Infrastructure	**1,478**
541 – Haul Road	1,478
550 – Mine Equipment	**102,600**
551 – Primary Mining Equipment	85,556
554 – Other Mining Support Equipment	9,710
559 – Dispatch / Comm. / Software / Survey	7,334
Total	**104,078**

21.1.8 Process Plant

The estimated capital costs for the processing area are detailed in Table 21.9, with a comprehensive description of the process plant provided in Chapter 17. Direct costs cover all labour, permanent equipment, and materials, excluding mobile equipment and freight, which are accounted for under indirect costs (700s) and G&A costs (800s), respectively.

Each major process area was estimated individually, covering the following disciplines:

- Earthworks (EW): Quantities derived from topographical data.
- Concrete (CI), Structural (ST), and Architectural (AR): Based on 3D models.
- Mechanical (MC): Includes small equipment, platework, tanks, and piping (PI), estimated using equipment quotations and material take-offs (MTOs).
- Electrical (EL) and Instrumentation (IN): Estimated from MTOs developed using the mechanical equipment list, P&IDs, and 3D models.

Table 21.9: Processing Capital Expenditures

Capital Expenditures (K$)	
601 - Site Preparation / Road / Berms	2,684
602 – Pipe Rack	2,627
603 – Buried Services	766
604 – ROM Pad	1,484
605 – Final Grade	649
610 – Comminution	86,448
620 – Gravity & Intensive Leach	5,381
640 – Lixiviation	44,234
650 – Reagents	8,948
660 – Refinery	13,819
680 – Tailings Management	14,099
690 – Process Plant Services	10,499
Total	**191,639**

21.1.9 Construction Indirects

Construction Indirect costs are presented in Table 21.10. They are based on GMS's historical executed project costs and MTOs from first principles. Each single cost code has a detailed explanation.

Table 21.10: Construction Indirect Capital

Capital Expenditures (K$)	
710 – Engineering, CM, PM	**39,557**
711 – Site CM Staff and Consultants	31,738
713 – Surveying	2,878
714 – QA/QC	2,747
715 – Induction	858
716 – Project Control	1,336
720 – Construction Offices, Facilities & Services	**5,683**
721 – Construction Offices / Trailers	749
722 – Temporary Truck Shop	998
723 – Temporary Explosive Magazine	-

Capital Expenditures (K$)	
724 – Temporary Laydown and Tool Crib	458
725 – Camp Construction Temporary Facilities	1,492
727 – Site Toilets / Ablution Units	357
728 – Construction Temporary Power Distribution	1,150
729 – Construction Temp Water and Piping Network	479
730 – Shops	**2,040**
740 – Construction Equipment & Tools	**34,471**
742 – Rental Equipment	6,928
743 – Operation and Maintenance	8,665
744 – Construction Tools	12,780
746 – LOTOTO Team	687
747 – EPP for Construction	2,757
748 – Scaffolding	1,584
749 – IT Equipment	1,070
760 – Energy	**23,361**
780 – Temporary Warehouse	**462**
790 – External Engineering	**28,960**
Total	**134,534**

21.1.10 General Services

Part of General Services includes the basic services listed in Table 21.11.

Table 21.11: List of Departments Included in General Services

General Services	Description
General Management & Corporate Affairs	Site executive, administrative governance, legal compliance
Finance & Accounting Project Insurance	Budgeting, payroll, auditing, cost control, and financial reporting
Information Technology (IT)	IT infrastructure, including hardware, software, data security, and communication systems

General Services	Description
Environment and Sustainability	Monitoring, sustaining commitments included in the ESIA, implementing standards and procedures to minimize impact, managing relationships with local communities, Indigenous groups, and government
Supply Chain	Procurement, contract management, expediting, reception, warehousing
Human Resources & Training	Manages staffing, recruitment, and personnel development, establishes training programs, policy compliance, onboarding processes, labour relations, and performance management
Security	Physical and operational security for the project site and personnel
Health & Safety	Establish programs and procedures, monitor practices, and provide emergency and medical services

The various departments provide support to construction, mining, power and processing, and their efficient operations are essential. In the case of a remote project like Oko West, two (2) additional areas are particularly important:

- **Camp and Site Services:** Site Services encompass camp operations, including kitchen, housing, and laundry, as well as routine site maintenance of all buildings and offices and road maintenance work from wharf to site.
- **Logistics:** This includes the cost of transporting goods from suppliers, whether located domestically or offshore, to the project site. For Oko West, it is anticipated that most materials and equipment will be imported from overseas, shipped by ocean freight to Guyana, transferred onto barges to reach the Oko Wharf landing, and then transported by tractor-trailer for the final leg to the site.
- **Transportation:** This department is responsible for coordinating worker transportation to and from the site in accordance with their rotation schedules.

Capital expenditures for General Services and Administration (G&A), as outlined in Table 21.11, are expected to gradually increase during project development, reach a peak during construction, and then decline to a steady level for the operational phase. Table 21.12 presents the estimated monthly expenditures over the 34 months of the Project.

Table 21.12: General Services Expenditures

Capital Expenditures (K$)	
810 – G&A Departments	**35,680**
820 – Logistics / Taxes / Insurance	**59,526**
821 – Out of Country – Inland Freight	4,666
822 – Sea Freight	46,657
823 – Air Freight	300
824 – In-Country Land Freight	3,495
825 – Customs, Taxes & Duties	2,000
827 – In-Country River Freight	2,408
830 – Camp & Site Services	**42,179**
831 – Camp	28,171
832 – Transportation	12,331
834 – Site Maintenance	1,677
Total	**137,385**

Equipment and materials to be purchased for Sustaining Expenditures carry a 30% allowance for indirects, including logistics.

21.1.11 Pre-Production and Commissioning Expenditures

The initial open-pit operation accounts for the majority of pre-production expenditures, with a smaller portion allocated to Processing for preparation, training and commissioning. The underground operation is expected to be fully operational by the fourth year of commercial production, with its pre-production costs classified under Sustaining Expenditures. Pre-Production and Commissioning expenditures are listed in Table 21.13.

Table 21.13: Pre-Production & Commissioning CAPEX

Capital Expenditures (K$)	
910 – Mining Pre-Prod	**91,556**
911 – Mine Administration	3,777
912 – Mine Engineering	4,525
913 – Mine Geology	3,031
914 – Mine Operations & Maintenance	80,223

Capital Expenditures (K$)	
950 – Process Plant Pre-Production	**14,290**
952 – Process Plant Pre-Production	8,810
953 – Process Plant Commissioning & Construction Support	3,980
954 – Vendor Reps	1,500
960 – First Fill, Spares & Consumables	**2,328**
961 – Spare Capital Parts	1,478
965 – First Fill	850
Total	**108,173**

Pre-Production mining costs will be incurred during the two (2) years prior to the start of commercial production; a tonnage of 2.6 Mt of ore and 47.1 Mt of waste will be mined during this 2-year period.

Pre-Production processing costs include an equivalent of a 5-month period with full processing personnel for training and commissioning. The commissioning period includes one (1) month of wet and waste commissioning. Then, a 2-month period prior to commercial production will include all processing costs required to hot commission the plant, assuming 50% and 60% of nameplate throughput for those two (2) months before declaring commercial production.

The power costs incurred by the site power plant during commissioning are included in the processing pre-production expenditures. It must be noted that commissioning of the process plant and power plant will be done concurrently.

Some three (3) months of construction labour in support of commissioning were included in the commissioning costs.

An estimate of $1,500,000 was also included for vendor representatives and operating specialists.

21.1.12 Contingency

Contingency has been estimated on an area-by-area basis by evaluating the confidence level of both scope definition and cost estimates, followed by a Monte Carlo iteration analysis. The total recommended contingency of $85 million amounts to 9% of direct and indirect expenditures before revenue.

The estimates are grounded in our extensive experience executing similar activities in comparable environments. Our self-perform model also reduces execution risk relative to a traditional EPCM approach

involving multiple contractors. This model has proven effective in managing potential uncertainties and execution challenges. The project's scope and schedule are well-defined and stable, minimizing the risk of significant changes.

By leveraging historical data and lessons learned from similar projects, we have optimized the contingency allocation while maintaining strong risk management and supporting overall project success. Several factors support a slightly lower contingency:

- **Executed Site Work**: Major earthworks and deforestation are already underway, with costs based on firm contracts and actual expenditures. This significantly reduces the uncertainty in these areas.
- **Equipment Procurement**: Procurement of mobile and processing equipment, totalling $126 million, has been completed under fixed-price agreements, minimizing risks associated with cost escalation and supply chain delays.

These developments have improved project definition and lowered exposure to typical execution risks. As a result, a 9% contingency is considered appropriate, providing balanced risk coverage given the current level of project certainty.

21.2 Sustaining Capital

Sustaining capital is presented in Table 21.14. Sustaining capital for the mine includes additional equipment purchases for the open pit operations and the initial and sustaining equipment for the underground operations for an overall mining requirement of $590 million. Significant equipment repairs that entail replacing major components, such as engines, transmissions, and similar parts, are classified as sustaining capital.

Additional work is planned to raise the main dams of the Tailings Storage Facility (TSF) and construct the TSF's North Dike to expand its capacity. The ongoing TSF elevation work will be carried out by the mine operations team using fill material sourced from nearby borrow areas. In Year 2, a mine dry building will be constructed for underground personnel, along with all necessary electrical infrastructure to support underground development. The sustaining capital for these activities is estimated at $60 million.

Table 21.14: Sustaining Cost Summary

Sustaining Cost Summary (K$)	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13+
MINING														
Open Pit & Support Equipment	**77,485**	28,565	9,725	2,315	11,877	5,973	4,840	4,363	2,579	4,821	1,604	488	274	60
Major Repairs	**221,963**	12,919	21,808	18,136	25,446	19,131	21,022	28,888	18,088	24,388	15,858	15,339	922	17
Initial Underground	**67,898**	18,696	49,201											
Sustaining Underground	**222,630**			66,746	46,007	42,988	41,502	5,395	7,738	5,273	4,480	2,440	50	10
Total Mining	**589,976**	**60,181**	**80,734**	**87,197**	**83,330**	**68,093**	**67,364**	**38,646**	**28,406**	**34,482**	**21,942**	**18,267**	**1,246**	**87**
OTHER COSTS														
110 – Deforestation	**10,490**	5,245	0	5245										
100 – Infrastructure	**2,914**	2,914												
250 – Mine Electrical Room & Distribution	**24,541**	14,315	2,888	3,048	2,052	1,707	497	17	17					
271 – OH Line Distribution on Site	**1,485**	**1,485**												
282 – Process Monitoring Advanced Control	**1,841**	**1,841**												
290 – IT Hardware & System	**391**	**391**												
323 – West Range Water Diversion Channel	**2,939**						2,939							
370 – Tailings Storage Facility (TSF)	**11,072**	3,294	0	2,593	0	0	2,593	0	0	2,593				
570 – Mining Portal Services	**2,698**	2,698												
610 – Pebble Crusher	**1,655**	0	0	0	1,655									
Total Other Costs	**60,027**	**32,183**	**2,888**	**10,885**	**3,708**	**1,707**	**6,029**	**17**	**17**	**2,593**				
Grand Total	**650,003**	**92,364**	**83,622**	**98,083**	**87,038**	**69,800**	**73,393**	**38,663**	**28,423**	**37,074**	**21,942**	**18,267**	**1,246**	**87**

21.3 Closure Costs

The closure costs are estimated to be $38.7 million, as summarized in Table 21.14, for each element of mine infrastructure.

Waste Storage Facility (WSF): The closure case for the WSF is to place, spread and scarify topsoil, and revegetate. It is assumed that any potentially acid-generating (PAG) material will be isolated with neutralizing material. The dump height will be 116 m, and the WSF footprint will be 370 ha. The WSF will be constructed to the overall closure slope angle of 2.5H:1V with bench regrading captured as operational expenditure (OPEX). Slopes will be covered with 200 mm of topsoil and revegetated. Progressive closure of the WSF will commence five (5) years before closure and continue until the end of the five-year closure period.

Underground Mine: The closure case for the underground mine is to de-energize and decouple all infrastructure, remove all equipment, cap or plug surface openings and allow the water to naturally fill the voids. Underground closure costs have been estimated based on experience with similar projects.

Open Pit: The closure case for the open pit is to raise safety berms around the pit perimeter, install fencing and signage and allow the pit to flood naturally with both groundwater and surface water.

Tailings Storage Facility: The closure case for the TSF is to leave the surface uncovered, install chain link fencing around the perimeter (20 km) and implement security measures in perpetuity. It is assumed that no embankment regrading will be required.

Processing Infrastructure and Power Plant: The closure case for the processing infrastructure is to decontaminate, decommission and demolish (DD&D) all infrastructure. It is assumed that slabs will be disposed of in the TSF, steel will be sold as scrap, other demolition material hauled to Bartica or Georgetown for recycling / disposal, and footprints covered with topsoil and revegetated. Processing infrastructure and power plant closure costs and salvage value have been estimated based on similar project costs.

Accommodation Camp: The closure case for the accommodation camp is to decontaminate, decommission and demolish all camp infrastructure. It is assumed that steel will be sold as scrap, other demolition material hauled to Bartica or Georgetown for recycling / disposal, and footprints covered with topsoil and revegetated. DD&D costs have been estimated based on similar project costs, and footprint reclamation assumes 6.28 ha covered with topsoil and revegetated.

Service and Haul Roads: The closure case for haul roads is to scarify all roads, cover with topsoil and revegetate. It is assumed that there will be 13.5 km of 30-m-wide haul roads to be covered with 150 mm of topsoil.

Site Transmission Lines: The closure case for the transmission lines is to demolish the lines connecting the power station to site infrastructure. It is assumed that there will be 8.4 km of double-pole power lines.

Access Road, Airstrip and Barge Landing: The closure case for the access road, airstrip, and barge landing is to leave in place and transfer to a new owner or donate to the local community. Currently, this assumes that these domains represent no liability, however, this will need to be discussed and confirmed with key stakeholders throughout the project lifecycle.

Maintenance and Monitoring: A variety of maintenance and monitoring activities have been scheduled for the closure and post-closure periods, including cover / revegetation monitoring, biodiversity monitoring, groundwater and surface water monitoring, revegetation, cap and cover maintenance and TSF-specific monitoring and security measures. These activities vary in frequency and length, with TSF-specific monitoring in perpetuity and all other monitoring having ceased at the end of the ten-year post-closure period.

Social Closure: Social closure will be undertaken through the implementation of stakeholder engagement and social transition plans. It is assumed that the development and execution of these plans will account for 3% of total direct closure costs.

Table 21.15: Closure Cost Summary

Closure Cost	Cost (K$)
Waste Storage Facility	4,580
Underground	250
Open Pit	1,330
TSF	2,979
Processing Infrastructure and Power Plant	5,650
Processing Infrastructure and Power Plant (Scrap Value)	(565)
Accommodation Camp	140
Service and Haul Roads	558
Site Transmission Lines	374
Maintenance and Monitoring	9,180

Closure Cost	Cost (K$)
Sub-Total (Directs)	**24,475**
Social / Communities (3%)	734
Studies and Indirects (15%)	3,671
Project Management (10%)	2,448
Contingency (30%)	7,343
Total	**38,670**

The following assumptions and exclusions should be noted:

- General and administrative (G&A) costs throughout the closure period are excluded.
- It is assumed that the required closure materials and equipment will be available on-site at closure.
- Closure and reclamation of borrow areas are excluded.
- Closure costs for the access road, airstrip and barge landing wharf are excluded, under the assumption that they will all be transferred to a new owner or donated to the local community.

These opportunities, assumptions and exclusions will need to be further assessed and evaluated at the next stage of the study.

21.4 Operating Costs

Operating Costs are summarized in Table 21.16 and presented on a yearly basis in Table 21.17. The operating costs include mining, processing, power, general services and administration ("G&A"), gold transportation and refining and royalties. The average LOM operating cost is $798/oz of gold or $45.29/t milled, excluding royalty costs. The average LOM all-in sustaining cost ("AISC") is $1,123/oz of gold or $63.70/t milled.

Table 21.16: Operating Costs Summary

Cost Summary	Total LOM Cost (M$)	Unit Cost ($/t milled)	Cost per oz ($/oz)
Open Pit Mining Cost	1,049	13.81	243
Underground Mining Cost	785	10.34	182
Processing Cost	550	7.24	128
Power Cost	604	7.95	140

Cost Summary	Total LOM Cost (M$)	Unit Cost ($/t milled)	Cost per oz ($/oz)
General & Administration Cost	417	5.49	97
Refining Cost	35	0.45	8
Total Site Cost	**3,438**	**45.29**	**798**
Royalty Cost	687	9.05	160
Total OPEX Cost	**4,125**	**54.34**	**958**
Sustaining Capital	650	8.56	151
Closure Cost	39	0.51	9
Land Payments	22	0.29	5
All-In Sustaining Costs (AISC)	**4,835**	**63.70**	**1,123**

Table 21.17: Total Operating Costs Summary by Year

Operating Cost Summary	Unit	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13+
Production Highlights															
Tonnage Milled	kt	75,911	6,741	7,000	6,687	6,000	6,016	6,000	6,000	6,039	6,229	6,000	6,000	5,458	1,740
Tonnage Mined MM OP	kt	61,730	6,741	7,000	6,616	5,327	4,748	4,213	4,289	4,316	4,519	4,309	4,405	3,887	1,360
Tonnage Mined MM UG	kt	14,181	-	-	71	673	1,269	1,787	1,711	1,723	1,711	1,691	1,595	1,570	380
Tonnage Mined Waste OP	kt	379,092	37,083	41,289	38,015	38,650	39,275	39,765	38,596	36,315	32,387	14,633	13,359	8,341	1,385
Recovered Gold	koz	4,310	312	317	303	325	354	353	352	368	380	433	386	334	93
Payable Gold	koz	4,307	311	317	303	325	354	353	351	368	380	433	386	334	93
Operating Costs															
Open Pit Mining	M$	1,048.6	77.8	93.4	100.6	97.3	90.5	103.5	105.2	104.9	93.3	66.3	55.4	41.6	18.6
Underground Mining	M$	784.8	4.6	13.0	26.5	60.9	81.4	96.6	83.8	83.5	81.3	77.6	75.4	67.9	32.2
Processing	M$	549.9	42.7	45.0	44.9	43.3	43.6	43.7	43.8	43.9	44.3	43.8	43.7	41.5	25.8
Power	M$	603.6	32.0	39.9	48.5	49.5	50.1	53.3	53.1	53.4	53.1	52.9	52.8	45.2	19.9
General & Administration	M$	416.6	28.1	29.6	34.4	36.5	37.3	37.4	37.2	37.2	37.1	34.2	31.2	27.7	8.6
Refining	M$	34.5	2.5	2.5	2.4	2.6	2.8	2.8	2.8	2.9	3.0	3.5	3.1	2.7	0.7
Total Site Cost	**M$**	**3,437.9**	**187.7**	**223.5**	**257.3**	**290.1**	**305.8**	**337.4**	**325.9**	**325.9**	**312.2**	**278.2**	**261.6**	**226.5**	**105.9**
Royalty Cost on Revenues	M$	687.1	62.1	63.1	59.7	57.8	55.4	48.5	50.0	52.9	55.3	65.8	57.9	45.6	13.2
Total OPEX Cost	**M$**	**4,125.0**	**249.7**	**286.6**	**317.0**	**347.9**	**361.2**	**385.9**	**375.9**	**378.8**	**367.5**	**344.0**	**319.4**	**272.0**	**119.1**
Sustaining Capital	M$	650.0	92.4	83.6	98.1	87.0	69.8	73.4	38.7	28.4	37.1	21.9	18.3	1.2	0.1
Closure Cost	M$	38.7	-	-	-	-	-	-	-	-	-	-	-	-	38.7
Land Payments	M$	21.7	21.7	-	-	-	-	-	-	-	-	-	-	-	-
All-In Sustaining Costs (AISC)	**M$**	**4,835.4**	**363.8**	**370.2**	**415.1**	**434.9**	**431.0**	**459.3**	**414.6**	**407.2**	**404.5**	**366.0**	**337.7**	**273.3**	**157.9**
Unit Operating Costs															
Open Pit Mining Cost	$/t milled	13.81	11.54	13.35	15.04	16.22	15.05	17.25	17.54	17.38	14.98	11.04	9.24	7.62	10.72
Underground Mining Cost	$/t milled	10.34	0.68	1.86	3.97	10.15	13.53	16.10	13.97	13.83	13.05	12.94	12.56	12.44	18.48
Processing Cost	$/t milled	7.24	6.34	6.43	6.71	7.21	7.25	7.29	7.30	7.27	7.11	7.29	7.29	7.60	14.82
Power Cost	$/t milled	7.95	4.75	5.69	7.26	8.24	8.33	8.88	8.85	8.83	8.53	8.81	8.80	8.28	11.46
General & Administration Cost	$/t milled	5.49	4.17	4.24	5.15	6.09	6.19	6.24	6.20	6.17	5.95	5.70	5.20	5.07	4.94
Refining Cost	$/t milled	0.45	0.37	0.36	0.36	0.43	0.47	0.47	0.47	0.49	0.49	0.58	0.51	0.49	0.43
Total Site Cost	**$/t milled**	**45.29**	**27.84**	**31.93**	**38.48**	**48.35**	**50.83**	**56.23**	**54.32**	**53.97**	**50.11**	**46.37**	**43.60**	**41.50**	**60.85**
Royalty Cost	$/t milled	9.05	9.21	9.01	8.92	9.63	9.21	8.08	8.33	8.76	8.88	10.97	9.64	8.35	7.58
Total OPEX Cost	**$/t milled**	**54.34**	**37.05**	**40.94**	**47.40**	**57.98**	**60.04**	**64.31**	**62.65**	**62.72**	**58.99**	**57.34**	**53.24**	**49.84**	**68.44**
Sustaining Capital	$/t milled	8.56	13.70	11.95	14.67	14.51	11.60	12.23	6.44	4.71	5.95	3.66	3.04	0.23	0.05

Operating Cost Summary	Unit	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13+
Closure Cost	$/t milled	0.51	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Land Payments	$/t milled	0.29	3.22	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
All-In Sustaining Costs (AISC)	**$/t milled**	**63.70**	**53.97**	**52.89**	**62.07**	**72.49**	**71.64**	**76.55**	**69.10**	**67.43**	**64.94**	**60.99**	**56.28**	**50.07**	**68.49**

21.4.1 Mining Costs

A detailed mine cost build up was developed from basic cost elements such as remuneration costs, consumable prices, fuel prices and equipment productivities.

Equipment operating costs were estimated for each equipment model, which includes operation and maintenance labour, parts (maintenance and repairs), fuel consumption, lubricant consumption, ground engaging tools or tires if applicable. Equipment operating costs were determined from various sources, including primarily information from the major suppliers and benchmarked costs from operations in similar environments.

The diesel fuel price assumed for estimating mining costs is $0.74/L.

The open pit mining unit cost is $2.39/t mined. The underground mining unit cost is $55.34/t mined. Table 21.18 presents the breakdown of the open pit mining costs by department while Table 21.19 presents the underground mining cost elements.

Table 21.18: OP Mining Cost Summary

Mining Costs (M$)	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Mine Operations	**38.7**	3.1	3.3	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	2.3	1.9	0.9
Mine Geology	**18.5**	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.4	0.6
Mine Maintenance Admin.	**37.1**	2.9	3.0	3.0	3.1	3.2	3.1	3.1	3.0	3.0	2.8	2.7	2.5	1.5
Mine Engineering	**37.3**	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	2.9	2.7	2.3	0.8
Drilling	**91.4**	5.5	7.9	9.2	8.6	8.4	10.6	10.5	9.8	8.1	5.0	4.2	2.9	0.8
Pre-Split Drilling and Blasting	**19.7**	1.0	2.1	2.1	1.3	1.1	2.6	2.3	2.2	1.1	1.5	1.0	1.0	0.2
Blasting	**272.6**	16.4	22.8	26.0	24.7	24.4	30.6	30.5	28.6	23.6	16.3	13.5	10.2	5.2
Loading	**104.3**	10.3	11.2	10.4	10.5	10.4	10.2	10.0	9.6	8.7	5.0	4.3	2.9	0.9
Hauling	**265.3**	19.7	23.4	26.8	26.2	20.9	24.3	26.7	29.7	27.3	16.5	12.5	8.4	2.7
Dewatering	**11.2**	0.4	0.4	0.6	0.8	0.8	1.0	1.0	1.0	1.0	1.2	1.2	0.9	0.9
Dump Maintenance	**39.9**	3.9	3.9	3.9	3.9	3.9	3.9	3.9	3.9	3.2	1.9	1.9	1.3	0.6
Road Maintenance	**46.7**	3.6	4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	3.6	3.0	2.2	1.0
Grade Control	**2.6**	0.3	0.3	0.3	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.1	0.1	0.0
Support Equipment	**52.1**	4.2	4.4	4.4	4.4	4.4	4.4	4.4	4.4	4.3	3.9	3.9	3.2	2.1
UG Backfill and Crushing	**19.5**	1.2	1.3	1.1	1.0	1.2	1.7	1.8	1.7	1.8	1.8	1.8	1.7	1.2
Transfer to Underground	**(13.4)**	-	-	-	-	(0.8)	(1.5)	(1.6)	(1.6)	(1.7)	(1.7)	(1.7)	(1.6)	(1.1)
Rehandling	**5.1**	0.5	0.7	0.6	0.4	0.3	0.3	0.3	0.3	0.3	0.3	0.5	0.3	0.2
Total OP Mining Cost	**1,048.6**	**77.8**	**93.4**	**100.6**	**97.3**	**90.5**	**103.5**	**105.2**	**104.9**	**93.3**	**66.3**	**55.4**	**41.6**	**18.6**
Unit Cost ($/t Mined)	**2.39**	**1.77**	**1.93**	**2.25**	**2.21**	**2.06**	**2.35**	**2.45**	**2.58**	**2.53**	**3.50**	**3.12**	**3.40**	**6.79**

Table 21.19: UG Mining Cost Summary

Mining Costs (M$)	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Diamond Drilling and Geology	**26.1**	-	-	0.1	1.2	2.4	3.4	3.1	3.1	3.0	3.1	3.1	2.9	0.8
Stope Preparation	**91.1**	0.1	0.2	4.2	17.8	17.4	17.4	7.1	8.2	6.6	4.5	3.9	3.7	-
Drilling and Blasting	**57.6**	-	-	-	2.4	5.2	7.0	7.1	7.1	6.4	6.7	6.7	6.5	2.5
Mucking and Hauling	**88.0**	-	-	-	2.3	6.7	10.4	10.4	11.1	11.0	11.3	10.8	10.9	3.1
Backfilling	**141.9**	-	-	-	5.5	13.7	21.1	19.6	17.3	17.8	15.8	14.8	12.2	4.1
Supervision	**50.7**	1.5	2.4	2.8	4.4	4.9	4.9	4.9	4.9	4.9	4.9	4.7	3.5	1.8
Mine Services	**144.9**	0.5	3.3	8.2	12.2	14.9	15.2	14.3	14.4	14.2	14.1	14.1	11.1	8.4
Maintenance Services	**75.3**	0.8	3.1	5.8	6.3	6.5	6.8	6.8	6.8	6.8	6.8	6.8	6.8	5.1
Electrical Services	**33.5**	0.3	1.6	2.7	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	1.6
Technical Services	**62.3**	1.3	2.3	2.8	5.8	5.8	5.8	5.8	5.8	5.8	5.8	5.8	5.8	3.7
Transfer to UG from OP	**13.4**	-	-	-	-	0.8	1.5	1.6	1.6	1.7	1.7	1.7	1.6	1.1
Total OPEX Cost	**784.8**	**4.6**	**13**	**26.5**	**60.9**	**81.4**	**96.6**	**83.8**	**83.5**	**81.3**	**77.6**	**75.4**	**67.9**	**32.2**
Unit Cost ($/t Mined)	**55.3**	**-**	**-**	**371.4**	**90.6**	**64.2**	**54.1**	**49.0**	**48.5**	**47.5**	**45.9**	**47.3**	**43.3**	**84.6**

21.4.2 Processing Costs

The processing flowsheet described in Section 17 is a conventional cyanidation circuit to treat free-milling gold ore. Processing costs will vary with the type of weathered materials fed to the plant (saprolite, transition, hard rock) and their respective proportions. The availability of saprolite and transition material will enable a yearly throughput of 7 Mtpa for the first five (5) years of operations; mill throughput will be limited to 6 Mtpa afterwards when only fresh rock material is fed to the plant. Table 21.20 provides a summary of operating costs. The average processing cost, excluding power cost, is $7.24/t milled.

Table 21.20: Processing OPEX

Cost	Total LOM Cost (M$)	$/t Milled
Labour Process	114	1.46
Primary Crushing	2	0.02
Grinding and Pre-Leach Thickening	23	0.31
Screen	9	0.12
Reagents	314	4.10
Maintenance Costs	71	0.93
Electrowinning and Gold Room	1	0.01
General	5	0.06
Laboratory	2	0.03
Mobile equipment	16	0.21
TOTAL	**557**	**7.24**

The consumption estimates for grinding media and reagents (Table 21.21) are based on metallurgical test work results, modelling and historical data from operations in the Guianas and West Africa. Pricing was compiled from 2025 supplier quotations and adjusted to reflect transportation costs to the project site (Table 21.22). For certain minor reagents, historical prices were adjusted for inflation.

Labour costs were determined using historical staffing levels at similar sites and total compensation packages (including salary, benefits, and incentives), based on labour market surveys conducted in Guyana and recent actual costs for North American expatriates.

Handling costs for saprolite and transition materials at apron feeders using backhoes are estimated at $0.30 per tonne, while rehandling of fresh rock adds a cost of $0.06 per tonne of crusher feed. It is assumed

that transition material will be processed directly through the feeders, although some may be blended with fresh rock and routed through the primary crusher.

A contingency has been included for supplies used in the assay and metallurgical laboratories, as well as for water treatment chemicals to manage potential contaminants not yet identified.

Logistics costs are embedded within the overall costs for consumables, supplies, and maintenance parts. Power costs are addressed in Sub-section 21.4.3.

Table 21.21: Grinding Media and Reagent Consumption

Consumable Rates	Unit	SAP	Transition	OP Rock	UG Rock
Grinding Media & Liners					
SAG Grinding Media	kg/t	0.09	0.17	0.35	0.39
BM Grinding Media	kg/t	0.29	0.29	0.58	0.63
SAG Mill Liners	Sets/y	0.49	0.99	1.98	2.23
Ball Mill	Sets/y	0.49	0.49	0.99	1.06
Reagents					
Cyanide	kg/t	0.29	0.29	0.31	0.37
Lime	kg/t	1.83	1.83	1.36	1.44
Activated Carbon	kg/t	0.04	0.04	0.04	0.04
Flocculant	kg/t	0.06	0.05	0.04	0.04
Caustic	kg/t	0.03	0.03	0.03	0.03
Hydrochloric Acid	t/t	0.10	0.10	0.10	0.10
Sodium Metabisulphite	kg/t	1.19	1.19	1.19	1.19
Copper Sulphate	kg/t	0.14	0.14	0.14	0.14

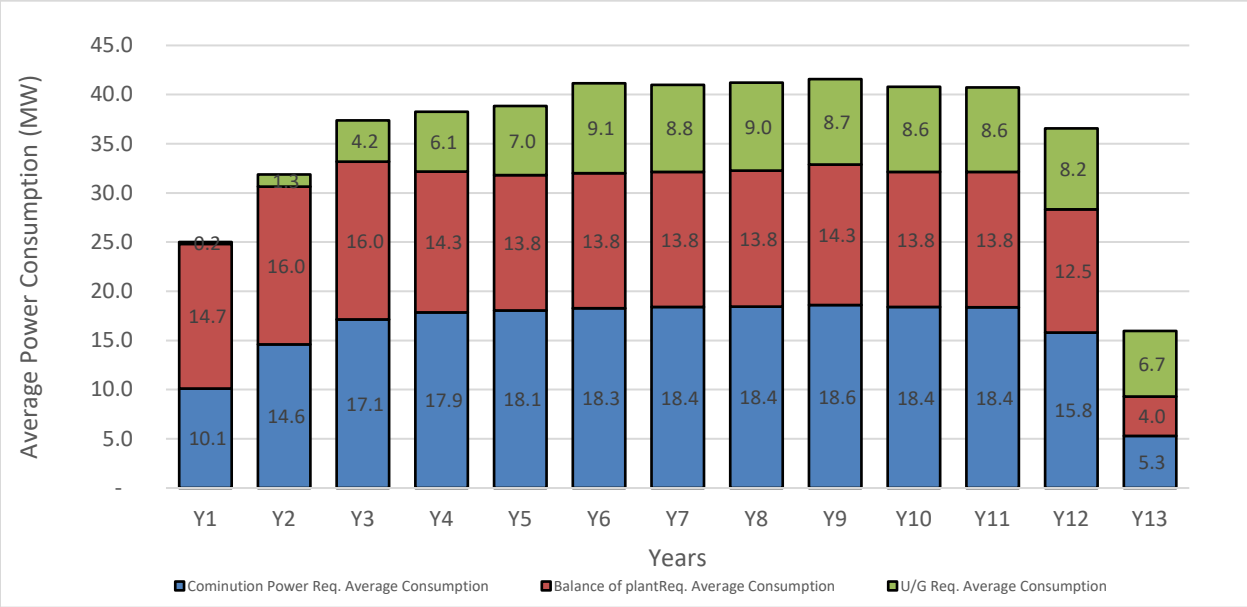
Table 21.22: Consumables Operating Costs

Consumables Operating Costs ($/t Milled)					
	Prices ($/t or Set)	**SAP**	**Transition**	**OP Rock**	**UG Rock**
Grinding Media & Liners					
SAG Grinding Media	1,315	0.11	0.23	0.46	0.52
BM Grinding Media	1,497	0.44	0.44	0.87	0.94
SAG Mill Liners	581,101	0.05	0.10	0.19	0.22
Ball Mill Liners	239,952	0.02	0.02	0.04	0.04
Sub-Total		**0.62**	**0.78**	**1.56**	**1.71**
Reagents – Process					
Cyanide	2,993	0.87	0.87	0.93	1.10
Lime (90% CAO)	318	0.58	0.58	0.43	0.46
Activated Carbon	3,407	0.14	0.14	0.14	0.14
Flocculant	4,943	0.30	0.25	0.20	0.20
Caustic	1,084	0.03	0.03	0.03	0.03
Hydrochloric Acid	472	0.00005	0.000005	0.00005	0.00005
Sub-Total		**1.91**	**1.86**	**1.73**	**1.92**
Reagents – CN Destruction & Other					
Sodium Metabisulphite	562	0.67	0.67	0.67	0.67
Copper Sulphate	3,067	0.42	0.42	0.42	0.42
Sub-Total		**1.08**	**1.08**	**1.08**	**1.08**
Total		**3.62**	**3.73**	**4.37**	**4.72**

21.4.3 Power Costs

Energy costs were estimated using a base price of $0.62 per litre for heavy fuel oil (HFO) and $0.74 per litre for diesel (LFO). The cost estimate is derived from the planned processing production schedule, power requirements for the underground mine, and the mechanical equipment list for surface facilities. It is projected that 2% of total electricity will be generated by diesel generators, with the remaining 98% supplied by HFO-powered generation. Power consumption over time is illustrated in Figure 21.1.

Figure 21.1: Power Consumption



The cost of electricity generation was calculated by including fuel expenses, overhaul costs, and operation and maintenance costs for all generators. The resulting average total site power operating cost is $7.95/t processed, as detailed in Table 21.23.

Table 21.23: Power Cost Breakdown

Power Generation – Power Costs		
Category	**OPEX (M$)**	**Unit Cost ($/t Milled)**
HFO Fuel Costs	513.6	6.77
LFO Fuel Costs	12.3	0.16
Contract Services - Overhaul	17.4	0.23
Overhaul - Parts	16.6	0.22
Operating Labour	13.1	0.17
Maintenance Parts & Services	24.0	0.32
Oil & Lubrication	6.6	0.09
Total	**603.6**	**7.95**

21.4.4 General and Administration Costs

A comprehensive description of General Services is provided in Section 21.1.10. The estimated operating costs for General Services and site maintenance are as follows:

- $19.2 million for Years 1 and 2, with only the open pit mine in operation.
- $22.4 million annually from Years 3 to 9, when both the open-pit and underground mines are in full production.
- $20.0 million in Year 10, $17.6 million in Year 11, $15.2 million in Year 12, and $4.2 million in Year 13, reflecting the gradual reduction in open-pit, underground, and processing activities.

Camp and Site Services, covering camp operations and site maintenance, along with Transportation services, will vary depending on the number of personnel on-site. Travel costs were calculated based on employee origin and rotation schedules.

Logistics expenses will peak during the construction and pre-production phases. During the production phase, transportation costs for consumables, supplies, and spare parts are included in their respective cost estimates.

Table 21.24 provides a summary of the G&A operating costs, and the average cost is $5.49/t milled.

Table 21.24: G&A Cost Breakdown

Cost	Total LOM Cost (M$)	Unit Cost ($/t Milled)
Basic	226	2.97
Camp	99	1.30
Site Maintenance	27	0.35
Transport	66	0.87
TOTAL	**417**	**5.49**

22. ECONOMIC ANALYSES

22.1 Overview

The economic and financial evaluation presented in this Technical Report utilizes a discounted cash flow method on a pre-tax and after-tax basis. The commodity price used in the evaluation was determined in Section 19. The financial model outputs key metrics including Net Present Value (NPV), payback period, and Internal Rate of Return (IRR) for the Oko West Project. The analysis is performed in real terms, excluding the effects of inflation, and is expressed in H1 2025 U.S. dollars. Project financing is not included in the evaluation.

The economic model estimates cash flows on an annual basis for the life of the Oko West Project, based on the level of engineering and design appropriate for a Feasibility Study.

Cash flow projections for the life of the Project are based on revenue from gold sales, operating expenditures (OPEX), capital expenditures (CAPEX), and other cost estimates. CAPEX is estimated in four (4) categories: initial capital, sustaining capital, closure and reclamation cost and working capital. OPEX estimates include labour, consumables, maintenance supplies, services, fuel, and power. Other costs, such as royalties, depreciation, and taxes, are estimated based on the current mine and processing plans.

The economic results are calculated from July 1st, 2025, onwards.

22.2 Cautionary Statements

The results of the economic analyses discussed in this section represent forward-looking information as defined under the Canadian securities law. These results are subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here.

The forward-looking information includes, but is not limited to, the following:

- Assumed prices for gold.
- Cost inflation.
- The proposed mine production plan.
- Assumptions regarding mining dilution and mining recovery.
- The recovery rates of gold in the processing plant.

- Proposed sustaining and operating costs.
- Labour and materials availability.
- Labour and materials costs being approximately consistent with the assumptions in the report.
- Assumptions regarding closure costs.
- Assumptions regarding environmental, social, and licensing risks.
- Changes to tax rates.
- Unexpected variations in the amount of mineralized material and material grade.
- Geotechnical or hydrogeological considerations during mining that differ from the assumptions.
- Ability to maintain social licence to operate.
- Unrecognized environmental risks.
- Unforeseen reclamation expenses.
- Failure of plant, equipment, and processes to operate as anticipated.
- Significant disruptions affecting the development and operation of the Oko West Project.

22.3 Key Assumption

22.3.1 Gold Price

The forecast of gold prices is described in Section 19. The long-term gold price assumption is $2,500/oz, in line with consensus commodity price forecasts from mining analysts.

22.3.2 Fuel Price and Energy

The price for diesel fuel (LFO) used to estimate OPEX is $0.79/L before 2028 and $0.74/L starting in 2028 once efficient logistics are finalized. Diesel fuel is used mostly for mobile equipment.

The price for heavy fuel oil (HFO) is estimated at $0.62/L based on recent quotes. HFO is used entirely at the power plant to supply operations with electricity. The electricity cost for the Oko West Project is based on the cost of HFO delivered to the project plus the operating labor and maintenance. The HFO efficiency is quoted at 5 kWh/L.

Table 22.1: Energy Cost Buildup

Parameters	$/kWh
HFO Fuel Cost	0.1242
LFO Fuel Cost	0.0030
Contract Services - Overhaul	0.0042
Overhaul - Parts	0.0040
Operating Labor	0.0032
Maintenance Parts & Services	0.0058
Oil & Lubrication	0.0016
Total Energy Cost	**0.1460**

22.3.3 Other Assumptions

The other key assumptions used in economic analysis are as follows:

- Discount rate 5%.
- All cost estimates are in constant H1 2025 United States dollars with no inflation or escalation factors considered.

22.4 Metal Production and Revenues

Payable gold over the Project life is 4,338 koz based on an average metallurgical recovery of 93.5% and a payability factor of 99.95%. Payable gold during pre-production is 30.1 koz, generating estimated revenue of $69.0M (net of transportation, refining and royalty costs) which offsets pre-production CAPEX. A total of 4,307 koz of gold will be payable during operations and will generate net revenue of $10,047M (net of transportation, refining and royalty costs). Table 22.2 shows the Mill Production Schedule summary, while Figure 22.1 presents the yearly gold production during LOM. Figure 22.2 shows the split between open pit and underground gold production.

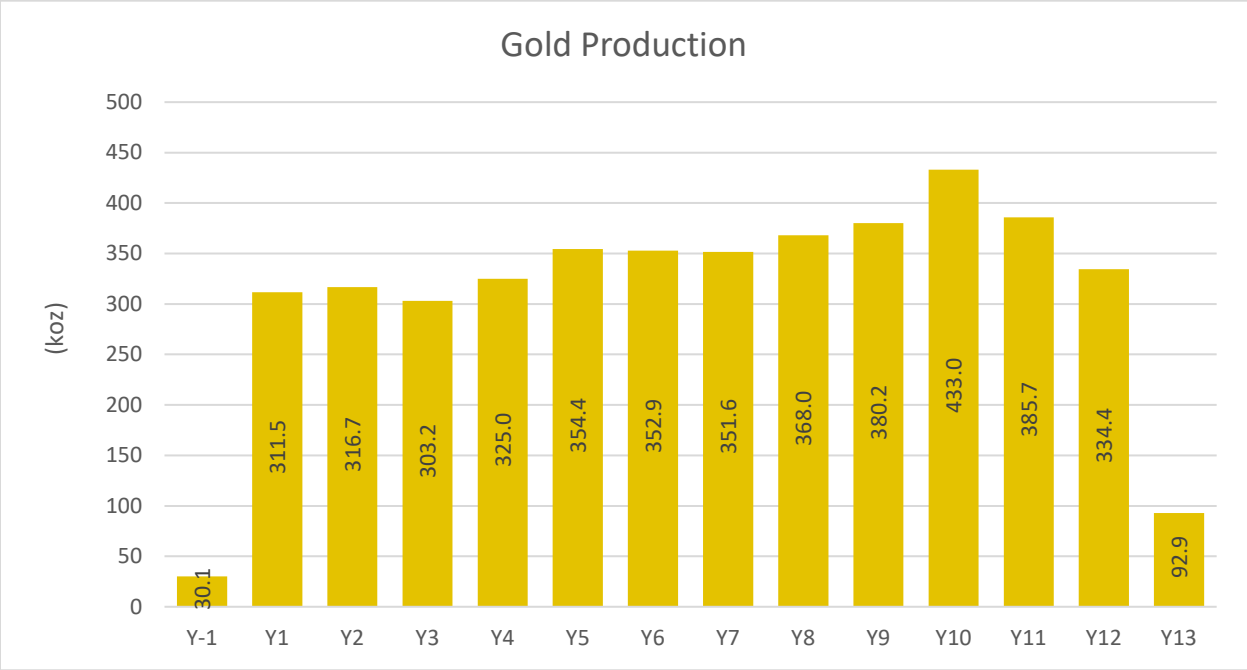
Table 22.2: Milling Production Schedule Summary

Process Plant Schedule[1]		Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Total Ore																	
Total Ore Milled	000 t	**76,555**	-	644	6,741	7,000	6,687	6,000	6,016	6,000	6,000	6,039	6,229	6,000	6,000	5,458	1,740
Head Grade	g/t Au	**1.89**	-	1.68	1.54	1.49	1.50	1.80	1.96	1.96	1.95	2.03	2.03	2.40	2.14	2.04	1.78
Recovered Gold	000 oz	**4,340**	-	30.1	311.5	316.7	303.2	325.0	354.4	352.9	351.6	368.0	380.2	433.0	385.7	334.4	92.9
Saprolite Ore																	
Saprolite Milled	000 t	**7,657**	-	447	2,895	2,382	1,431	121	33	78	1	39	229	-	-	-	-
Head Grade	g/t Au	**1.28**	-	1.47	1.47	1.29	0.99	0.72	0.69	0.69	0.69	0.60	0.73	-	-	-	-
Recovered Gold	000 oz	**295**	-	18.5	128.2	93.9	43.4	2.6	0.7	1.6	0.0	0.7	5.1	-	-	-	-
Transition Ore																	
Transition Milled	000 t	**3,541**	-	181	1,898	683	77	114	104	1	-	5	449	-	-	29	-
Head Grade	g/t Au	**1.43**	-	2.25	1.70	1.16	0.91	1.07	0.83	0.39	-	0.49	0.75	-	-	0.65	-
Recovered Gold	000 oz	**150**	-	11.1	96.0	23.8	2.1	3.7	2.6	0.0	-	0.1	10.1	-	-	0.6	-
Fresh Rock Ore Open Pit																	
Fresh Rock Milled	000 t	**51,176**	-	17	1,948	3,934	5,107	5,092	4,611	4,134	4,288	4,271	3,840	4,309	4,405	3,858	1,360
Head Grade	g/t Au	**1.63**	-	1.17	1.50	1.67	1.63	1.71	1.64	1.41	1.47	1.57	1.73	2.06	1.75	1.41	1.22
Recovered Gold	000 oz	**2,517**	-	0.5	87.3	199.0	251.7	262.8	229.2	176.1	190.3	203.0	200.7	268.7	233.1	164.6	50.2
Underground																	
Fresh Rock Milled	000 t	**14,181**	-	-	-	-	71	673	1,269	1,787	1,711	1,723	1,711	1,691	1,595	1,570	380
Head Grade	g/t Au	**3.26**	-	-	-	-	2.85	2.79	3.23	3.29	3.16	3.20	3.22	3.26	3.21	3.62	3.77
Recovered Gold	000 oz	**1,378**	-	-	-	-	6.1	55.8	121.9	175.2	161.3	164.2	164.3	164.4	152.7	169.2	42.7

**Notes:*

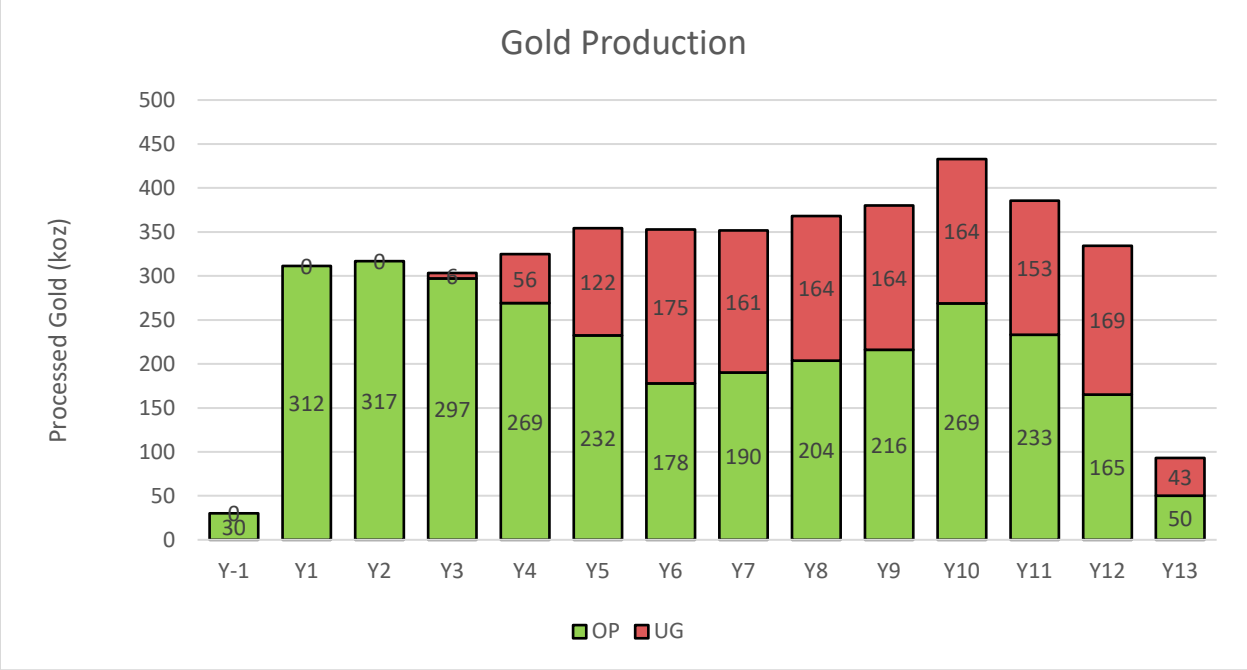
1. *Includes pre-production tonnage.*

Figure 22.1: Gold Production Over LOM



Source: GMS (May 2025)

Figure 22.2: Gold Production Over LOM by Mining Method



Source: GMS (May 2025)

22.5 Royalties

As stated in the Mineral Agreement with the Government of Guyana, a royalty is applicable to the gold production from the property. The applicable royalties for gold produced from open pit mining operation shall be 8% of the net smelter return. The applicable royalty for gold produced from underground mining operation shall be 3% of the net smelter return. A payment of $5 per ounce recovered estimated in the Feasibility Study is payable to the previous owner of the property at the start of commercial production.

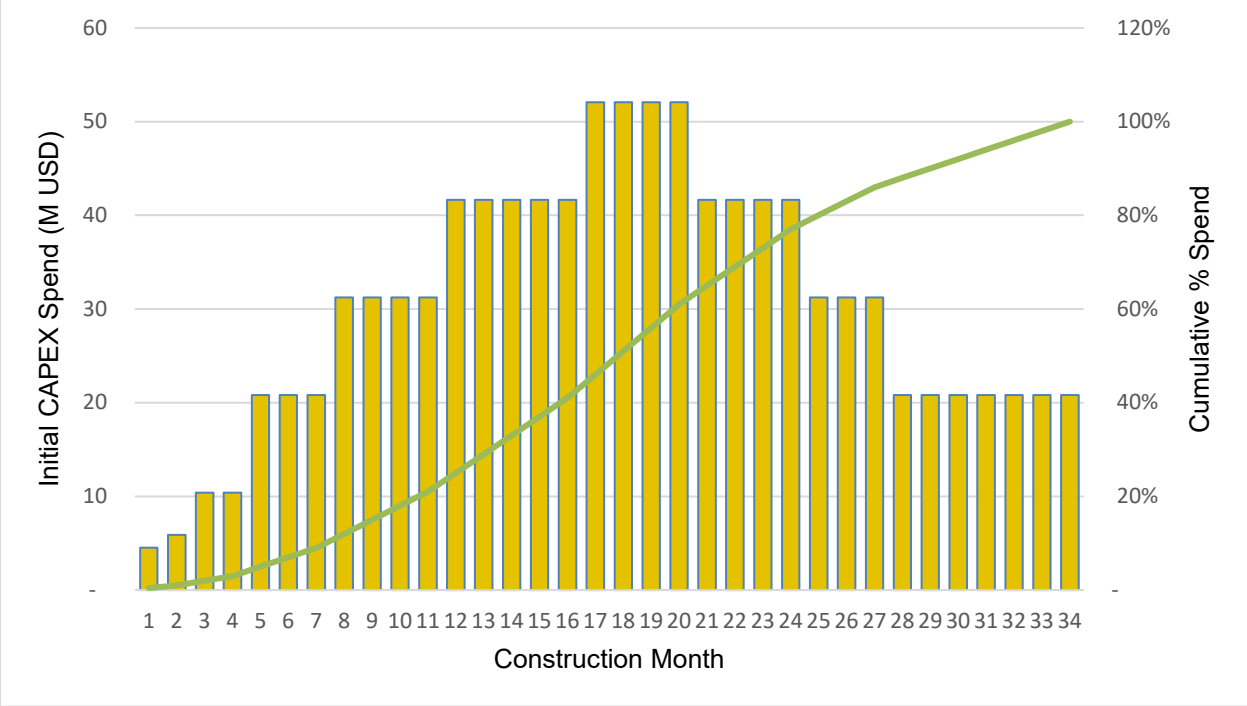
22.6 Capital Expenditures

The capital expenditures include initial capital expenditures as well as the sustaining capital expenditures to be invested after commencement of commercial operations.

22.7 Initial Capital

The initial CAPEX for Project construction, including processing facilities, mine equipment purchases, pre-production activities, infrastructures and other direct and indirect costs, is estimated to be $1,041M before gold credits from pre-production gold sales. The total initial CAPEX includes a contingency of $85M, which is 9% of direct and indirect expenditures. The monthly CAPEX estimate is presented in Figure 22.3.

Figure 22.3: Initial CAPEX by Month



Source: GMS, 2025

22.7.1 Sustaining Capital

Sustaining capital is allocated to specific areas of the project. For the open pit mine, this includes the acquisition of new equipment and major components. In the underground mine, sustaining capital covers all necessary development activities and equipment. Ongoing operations will require further equipment purchases for the underground mine. Additional work will also be undertaken to raise the main dams of the tailings storage facility (TSF), with the mine operations team completing the dam lifts using fill material sourced from the TSF basin. An increase in power demand is anticipated, for which an additional generator will be installed.

The sustaining capital is estimated at $650M and is detailed in Table 22.3.

22.8 Working Capital

Working capital is required to finance supplies in inventory. It is planned to maintain a 30-day inventory of fuels (HFO and LFO). Storage of other main consumables will vary from one (1) month to three (3) months depending on frequency of shipments. This represents a maximum inventory value of $43M. Additionally, it is planned to pay the various operational suppliers within 30 days, and payables are included in the working capital calculation.

22.8.1 Reclamation & Closure Cost

Reclamation and closure costs include infrastructure decommissioning, site shaping and revegetation, maintenance and post closure monitoring. The total reclamation and closure cost is estimated at $39M as discussed in Section 21.

22.8.2 Salvage Value

No residual value was estimated at this stage of the study.

22.9 Operating Cost Summary

Operating costs are presented by year in Table 22.3 and Table 22.4. The operating costs include mining, processing, power, general services and administration (G&A), gold transportation, and refining and royalties. The average LOM operating cost is $798/oz of gold or $45.29/t milled. The average LOM site all-in sustaining cost (AISC) is $1,123/oz of gold or $63.70/t milled.

Table 22.3: Operating Cost Summary (USD)

Operating Cost Summary	Unit	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16
Production Highlights (excl. Pre-Prod.)																		
Tonnage Milled	kt	**75,911**	6,741	7,000	6,687	6,000	6,016	6,000	6,000	6,039	6,229	6,000	6,000	5,458	1,740	-	-	-
Tonnage Mined Ore OP	kt	**61,730**	6,741	7,000	6,616	5,327	4,748	4,213	4,289	4,316	4,519	4,309	4,405	3,887	1,360	-	-	-
Tonnage Mined Ore UG	kt	**14,181**	-	-	71	673	1,269	1,787	1,711	1,723	1,711	1,691	1,595	1,570	380	-	-	-
Tonnage Mined Waste OP	kt	**382,212**	37,126	41,616	38,716	39,282	39,964	40,133	38,715	36,488	32,428	14,649	13,363	8,346	1,385	-	-	-
Recovered Gold	koz	**4,310**	312	317	303	325	354	353	352	368	380	433	386	334	93	-	-	-
Payable Gold	koz	**4,307**	311	317	303	325	354	353	351	368	380	433	386	334	93	-	-	-
Operating Costs (excl. Pre-Prod.)																		
Open Pit Mining	MUSD	**1,048.6**	77.8	93.4	100.6	97.3	90.5	103.5	105.2	104.9	93.3	66.3	55.4	41.6	18.6	-	-	-
Underground Mining	MUSD	**784.8**	4.6	13.0	26.5	60.9	81.4	96.6	83.8	83.5	81.3	77.6	75.4	67.9	32.2	-	-	-
Processing	MUSD	**549.9**	42.7	45.0	44.9	43.3	43.6	43.7	43.8	43.9	44.3	43.8	43.7	41.5	25.8	-	-	-
Power	MUSD	**603.6**	32.0	39.9	48.5	49.5	50.1	53.3	53.1	53.4	53.1	52.9	52.8	45.2	19.9	-	-	-
General & Administration	MUSD	**416.6**	28.1	29.6	34.4	36.5	37.3	37.4	37.2	37.2	37.1	34.2	31.2	27.7	8.6	-	-	-
Refining	MUSD	**34.5**	2.5	2.5	2.4	2.6	2.8	2.8	2.8	2.9	3.0	3.5	3.1	2.7	0.7	-	-	-
Total Site Cost	MUSD	**3,437.9**	**187.7**	**223.5**	**257.3**	**290.1**	**305.8**	**337.4**	**325.9**	**325.9**	**312.2**	**278.2**	**261.6**	**226.5**	**105.9**	**-**	**-**	**-**
Royalty Cost on Revenues	MUSD	**687.1**	62.1	63.1	59.7	57.8	55.4	48.5	50.0	52.9	55.3	65.8	57.9	45.6	13.2	-	-	-
Total OPEX Cost	MUSD	**4,125.0**	**249.7**	**286.6**	**317.0**	**347.9**	**361.2**	**385.9**	**375.9**	**378.8**	**367.5**	**344.0**	**319.4**	**272.0**	**119.1**	**-**	**-**	**-**
Sustaining Capital	MUSD	**650.0**	92.4	83.6	98.1	87.0	69.8	73.4	38.7	28.4	37.1	21.9	18.3	1.2	0.1	-	-	-
Closure Cost	MUSD	**38.7**	-	-	-	-	-	-	-	-	-	-	-	-	-	12.9	12.9	12.9
Land Payments	MUSD	**21.7**	21.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
All-In Site Sustaining Costs (AISC)	MUSD	**4,835.4**	**363.8**	**370.2**	**415.1**	**434.9**	**431.0**	**459.3**	**414.6**	**407.2**	**404.5**	**366.0**	**337.7**	**273.3**	**119.2**	**12.9**	**12.9**	**12.9**

Table 22.4: Operating Cost Summary per Tonne

Operating Costs per Tonnes Mined		Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Open Pit Mining Cost	t/ore + waste mined	**2.36**	1.77	1.92	2.22	2.18	2.02	2.33	2.45	2.57	2.53	3.50	3.12	3.40	6.79
Underground Mining Cost	t/ore mined	**55.34**	0.00	0.00	371.38	90.57	64.16	54.06	48.99	48.46	47.53	45.93	47.25	43.25	84.61
Operating Costs per Tonnes Milled															
Open Pit Mining Cost	t/milled	**13.81**	11.54	13.35	15.04	16.22	15.05	17.25	17.54	17.38	14.98	11.04	9.24	7.62	10.72
Underground Mining Cost	t/milled	**10.34**	0.68	1.86	3.97	10.15	13.53	16.10	13.97	13.83	13.05	12.94	12.56	12.44	18.48
Processing Cost	t/milled	**7.24**	6.34	6.43	6.71	7.21	7.25	7.29	7.30	7.27	7.11	7.29	7.29	7.60	14.82
Power Cost	t/milled	**7.95**	4.75	5.69	7.26	8.24	8.33	8.88	8.85	8.83	8.53	8.81	8.80	8.28	11.46
General & Administration Cost	t/milled	**5.49**	4.17	4.24	5.15	6.09	6.19	6.24	6.20	6.17	5.95	5.70	5.20	5.07	4.94
Refining Cost	t/milled	**0.45**	0.37	0.36	0.36	0.43	0.47	0.47	0.47	0.49	0.49	0.58	0.51	0.49	0.43
Total Site Cost	**t/milled**	**45.29**	**27.84**	**31.93**	**38.48**	**48.35**	**50.83**	**56.23**	**54.32**	**53.97**	**50.11**	**46.37**	**43.60**	**41.50**	**60.85**
Royalty Cost	t/milled	**9.05**	9.21	9.01	8.92	9.63	9.21	8.08	8.33	8.76	8.88	10.97	9.64	8.35	7.58
Total OPEX Cost	**t/milled**	**54.34**	**37.05**	**40.94**	**47.40**	**57.98**	**60.04**	**64.31**	**62.65**	**62.72**	**58.99**	**57.34**	**53.24**	**49.84**	**68.44**
Sustaining Capital	t/milled	**8.56**	13.70	11.95	14.67	14.51	11.60	12.23	6.44	4.71	5.95	3.66	3.04	0.23	0.05
Closure Cost	t/milled	**0.51**	-	-	-	-	-	-	-	-	-	-	-	-	-
Land Payments	t/milled	**0.29**	3.22	-	-	-	-	-	-	-	-	-	-	-	-
All-In Sustaining Costs (AISC)	t/milled	**63.70**	**53.97**	**52.89**	**62.07**	**72.49**	**71.64**	**76.55**	**69.10**	**67.43**	**64.94**	**60.99**	**56.28**	**50.07**	**68.49**

22.10 Taxation

The applicable taxes are included in the economic analysis. The corporate tax rate is 25% and allows for straight line depreciation over five (5) years. An initial tax asset pool of $148.2M was considered in the estimate of taxes to be paid. The total amount of tax payable to the Government of Guyana is $1,193M over the LOM.

22.11 Economic Results

The main economic metrics used to evaluate the Project consist of the net undiscounted after-tax cash flow, the net discounted after-tax cash flow or NPV, IRR and payback period. A 5% discount rate is commonly used as the base case for gold projects.

A summary of the Project economic results is presented in Table 22.5 and the annual Project cash flows are presented in Table 22.6. The total after-tax undiscounted cash flow over the Project life is $3,768M, and the after-tax NPV 5% is $2,163M. The after-tax Project cash flow results in a 2.9-year payback period from the commencement of commercial operations with an IRR of 31.1% before tax and 27.1% after tax.

Table 22.5: Project Economic Results Summary

Technical Report Feasibility Study Update Life-of-Mine Results		
Gold Price – Base Case	$/oz	2,500
Mine Life (operation years)	Mt	12.3
OP Mill Feed Tonnage	Mt	62.37
UG Mill Feed Tonnage	Mt	14.18
Total Ore Mined	***Mt***	***76.56***
Total Waste Mined (OP and UG)	**Mt**	***429.35***
Total Tonnage Mined (OP and UG)	**Mt**	***505.91***
Open Pit Strip Ratio	***waste: ore***	***6.83***
Average Milling Throughput	***Mtpa***	***6.17***
Average Milling Throughput	***tpd***	***16,911***
Gold Head Grade	g/t Au	1.89
OP Head Grade	g/t Au	1.57
UG Head Grade	g/t Au	3.26
Contained Gold	koz	4,642

Technical Report Feasibility Study Update Life-of-Mine Results		
Average Gold Recovery (%)	%	93.5%
Total Gold Production	koz	4,340
Average Annual Gold Production	koz	350
Operating Costs (LOM Average)		
Open Pit Mining Cost	$/t milled	13.81
Underground Mining Cost	$/t milled	10.34
Processing Cost	$/t milled	7.24
Power Cost	$/t milled	7.95
General & Administration Cost	$/t milled	5.49
Refining Cost	$/t milled	0.45
Total Site Costs	**$/t milled**	**45.29**
Total Site Costs	$/oz	798
Government Royalties	$/oz	160
Total Operating Cost	$/oz	958
AISC	**$/oz**	**1,123**
Capital Costs		
Initial Capital Costs	*$ MM*	956
Contingency	*$ MM*	85
Total Initial Capital Cost	***$ MM***	**1,041**
Initial UG Capital Costs (Sustaining Capital)	*$ MM*	68
OP and UG Sustaining Capital	$ MM	522
Life of Mine Sustaining Capital	**$ MM**	**650**
Closure Costs	$ MM	39
Total Capital Costs	**$ MM**	**1,730**
Financial Evaluation		
After-Tax NPV 5%	$ MM	2,163
After-Tax IRR	%	27.1%
Payback	years	2.9

Table 22.6: Project Cash Flow Summary

Cash Flow (M USD)	Total	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16
Gold Price (USD/oz)	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500
Rec. Gold (koz)	4,339.7	-	-	30.1	311.5	316.7	303.2	325.0	354.4	352.9	351.6	368.0	380.2	433.0	385.7	334.4	92.9	-	-	-
Payable Gold - Prod	4,337.6	-	-	30.1	311.4	316.5	303.1	324.8	354.2	352.8	351.4	367.8	380.0	432.8	385.5	334.2	92.9	-	-	-
Gross Revenue	10,809.2	-	-	75.0	775.9	788.7	755.3	809.4	882.7	879.1	875.8	916.6	947.1	1,078.5	960.8	832.8	231.5	-	-	-
Mining OP Cost	(1,140.1)	(4.1)	(32.5)	(54.9)	(77.8)	(93.4)	(100.6)	(97.3)	(90.5)	(103.5)	(105.2)	(104.9)	(93.3)	(66.3)	(55.4)	(41.6)	(18.6)	-	-	-
Mining UG Cost	(784.8)	-	-	-	(4.6)	(13.0)	(26.5)	(60.9)	(81.4)	(96.6)	(83.8)	(83.5)	(81.3)	(77.6)	(75.4)	(67.9)	(32.2)	-	-	-
Processing Cost	(557.0)	-	-	(7.1)	(42.7)	(45.0)	(44.9)	(43.3)	(43.6)	(43.7)	(43.8)	(43.9)	(44.3)	(43.8)	(43.7)	(41.5)	(25.8)	-	-	-
Power Cost	(606.0)	-	-	(2.4)	(32.0)	(39.9)	(48.5)	(49.5)	(50.1)	(53.3)	(53.1)	(53.4)	(53.1)	(52.9)	(52.8)	(45.2)	(19.9)	-	-	-
General & Administration Cost	(416.6)	-	-	-	(28.1)	(29.6)	(34.4)	(36.5)	(37.3)	(37.4)	(37.2)	(37.2)	(37.1)	(34.2)	(31.2)	(27.7)	(8.6)	-	-	-
Refining Cost	(34.7)	-	-	(0.2)	(2.5)	(2.5)	(2.4)	(2.6)	(2.8)	(2.8)	(2.8)	(2.9)	(3.0)	(3.5)	(3.1)	(2.7)	(0.7)	-	-	-
Royalty Cost on Revenues Cost	(693.1)	-	-	(6.0)	(62.1)	(63.1)	(59.7)	(57.8)	(55.4)	(48.5)	(50.0)	(52.9)	(55.3)	(65.8)	(57.9)	(45.6)	(13.2)	-	-	-
Transfer to Capex	101.1	4.1	32.5	64.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Land Payment	21.7	-	-	-	21.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Operating Costs	(4,109.6)	-	-	(6.2)	(228.0)	(286.6)	(317.0)	(347.9)	(361.2)	(385.9)	(375.9)	(378.8)	(367.5)	(344.0)	(319.4)	(272.0)	(119.1)	-	-	-
EBITDA	6,691.0	-	-	69.0	506.9	504.7	440.7	464.2	524.4	496.0	502.7	540.8	582.6	738.0	644.4	563.5	113.1	-	-	-
Initial CAPEX	(1,041.4)	(187.5)	(531.1)	(322.8)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining CAPEX	(650.0)	-	-	-	(92.4)	(83.6)	(98.1)	(87.0)	(69.8)	(73.4)	(38.7)	(28.4)	(37.1)	(21.9)	(18.3)	(1.2)	(0.1)	-	-	-
Closure Costs	(38.7)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(12.9)	(12.9)	(12.9)
Salvage Value	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Capital costs	(1,730.1)	(187.5)	(531.1)	(322.8)	(92.4)	(83.6)	(98.1)	(87.0)	(69.8)	(73.4)	(38.7)	(28.4)	(37.1)	(21.9)	(18.3)	(1.2)	(0.1)	(12.9)	(12.9)	(12.9)
Working Capital	-	(1.1)	(7.5)	(14.3)	(17.4)	(3.3)	(12.3)	(1.3)	(0.5)	(1.5)	0.5	0.0	0.9	2.1	0.9	0.9	3.7	50.1	-	-
Pre-Tax Cash Flow	4,960.9	(188.5)	(538.7)	(268.1)	397.2	417.8	330.3	375.8	454.0	421.2	464.5	512.4	546.5	718.1	627.0	563.1	116.8	37.2	(12.9)	(12.9)
Taxes	(1,192.8)	-	-	-	-	(63.2)	(52.0)	(62.0)	(97.4)	(100.3)	(102.9)	(114.5)	(128.4)	(169.4)	(148.7)	(131.6)	(22.4)	-	-	-
After-Tax Cash Flow	3,768.1	(188.52)	(538.65)	(268.07)	397.23	354.53	278.33	313.84	356.62	320.93	361.65	397.85	418.04	548.68	478.31	431.57	94.41	37.18	(12.90)	(12.90)

22.12 Sensitivity Analysis

The Project financial performance is most sensitive to the gold price and much less to the operating costs and capital expenditures. The results of the sensitivity analysis of the project in terms of NPV, IRR and Payback to the gold price are summarized in Table 22.7, Table 22.8 and Table 22.9 summarize the variation on NPV, IRR and payback to the OPEX and CAPEX. Figure 22.4 to Figure 22.7 show the sensitivity on the After-Tax Total Cashflow, NPV 5%, IRR and Payback.

Table 22.7: Gold Price Sensitivity

Gold Price (USD/oz)	Pre-Tax				After-Tax			
	NPV 0% (M USD)	NPV 5% (M USD)	IRR	Payback Period (y)	NPV 0% (M USD)	NPV 5% (M USD)	IRR	Payback Period (y)
1,400	(2,005)	(1,685)	0%	16.0	(2,005)	(1,685)	0%	16.0
1,600	(1,031)	(1,044)	0%	16.0	(1,031)	(1,044)	0%	16.0
1,800	73	(317)	1%	11.7	73	(317)	1%	11.7
2,000	1,307	495	11%	7.5	1,018	337	9%	7.6
2,200	2,671	1,393	19%	4.5	2,045	1,025	17%	4.7
2,400	4,165	2,377	27%	3.1	3,169	1,768	24%	3.4
2,500	4,961	2,901	31%	2.5	3,768	2,163	27%	2.9
2,600	5,789	3,446	35%	2.1	4,391	2,573	30%	2.5
2,800	7,543	4,600	42%	1.7	5,711	3,443	37%	1.9
3,000	9,426	5,840	50%	1.3	7,129	4,376	43%	1.6
3,200	11,440	7,166	57%	1.1	8,645	5,374	49%	1.3
3,400	13,583	8,577	64%	0.9	10,258	6,436	55%	1.1
3,600	15,856	10,074	71%	0.8	11,969	7,563	61%	1.0

Table 22.8: OPEX Sensitivity

	Pre-Tax				After-Tax			
OPEX	**NPV 0% (M USD)**	**NPV 5% (M USD)**	**IRR**	**Payback Period (y)**	**NPV 0% (M USD)**	**NPV 5% (M USD)**	**IRR**	**Payback Period (y)**
80%	5,645	3,351	34%	2.3	4,281	2,501	30%	2.6
90%	5,303	3,126	33%	2.4	4,025	2,332	28%	2.7
Base Case	4,961	2,901	31%	2.5	3,768	2,163	27%	2.9
110%	4,619	2,675	30%	2.7	3,512	1,994	26%	3.1
120%	4,277	2,450	28%	2.9	3,255	1,824	24%	3.2

Table 22.9: Initial CAPEX Sensitivity

	Pre-Tax				After-Tax			
CAPEX	**NPV 0% (M USD)**	**NPV 5% (M USD)**	**IRR**	**Payback Period (y)**	**NPV 0% (M USD)**	**NPV5% (M USD)**	**IRR**	**Payback Period (y)**
80%	5,169	3,098	37%	1.9	3,924	2,318	32%	2.2
90%	5,065	2,999	34%	2.2	3,846	2,240	30%	2.6
Base Case	4,961	2,901	31%	2.5	3,768	2,163	27%	2.9
110%	4,857	2,802	29%	2.9	3,690	2,085	25%	3.2
120%	4,753	2,703	27%	3.2	3,612	2,007	23%	3.4

Figure 22.4: After-Tax Total Cash Flow Sensitivity

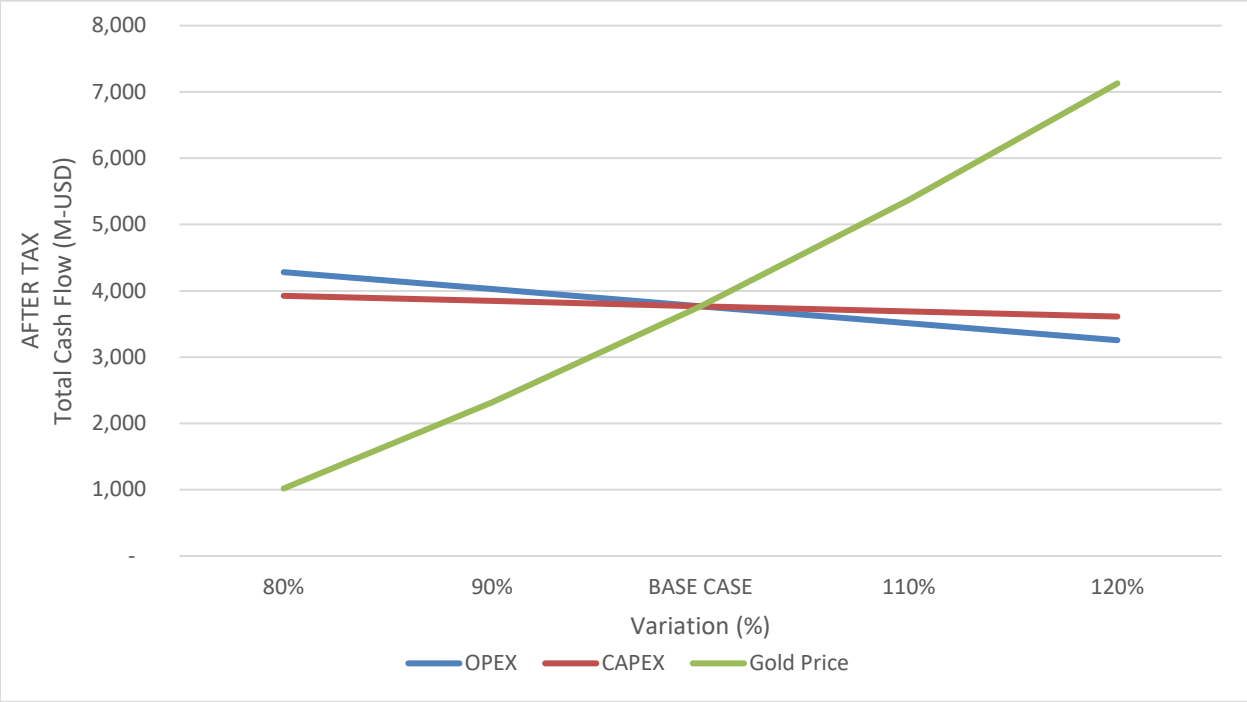


Figure 22.5: After-Tax NPV (5%) Sensitivity

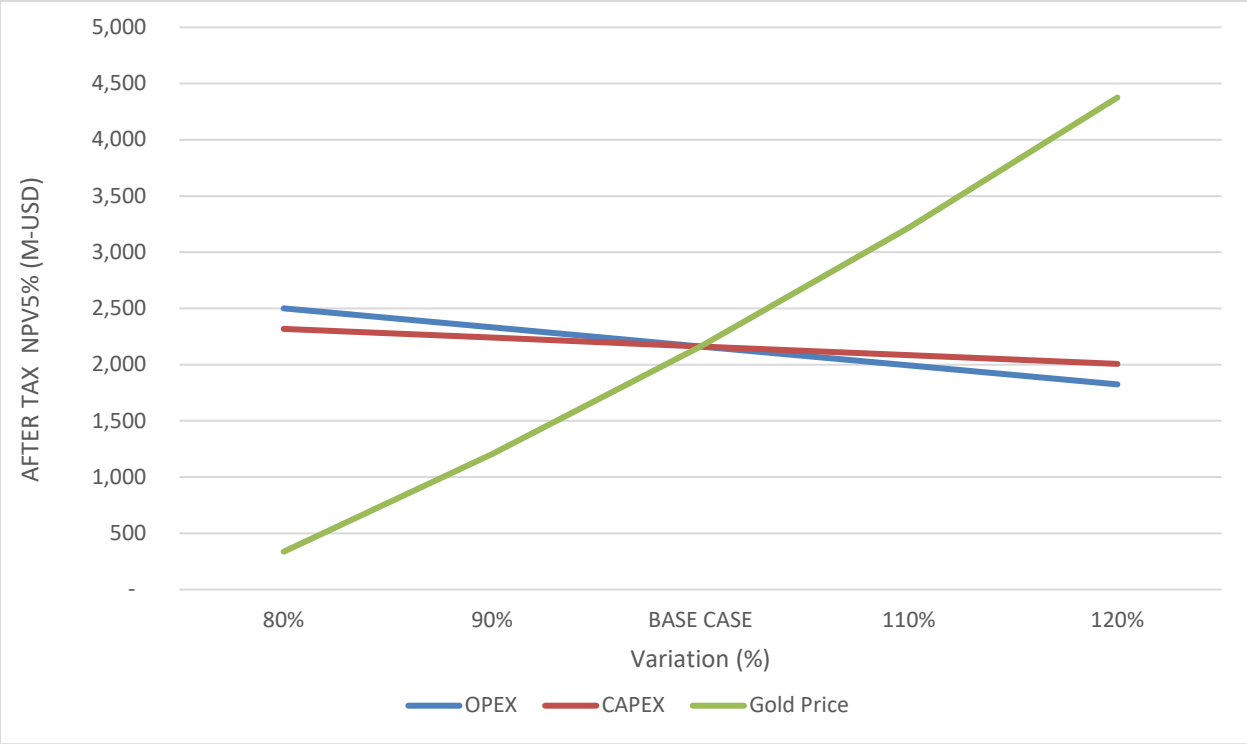


Figure 22.6: After-Tax Internal Rate of Return Sensitivity

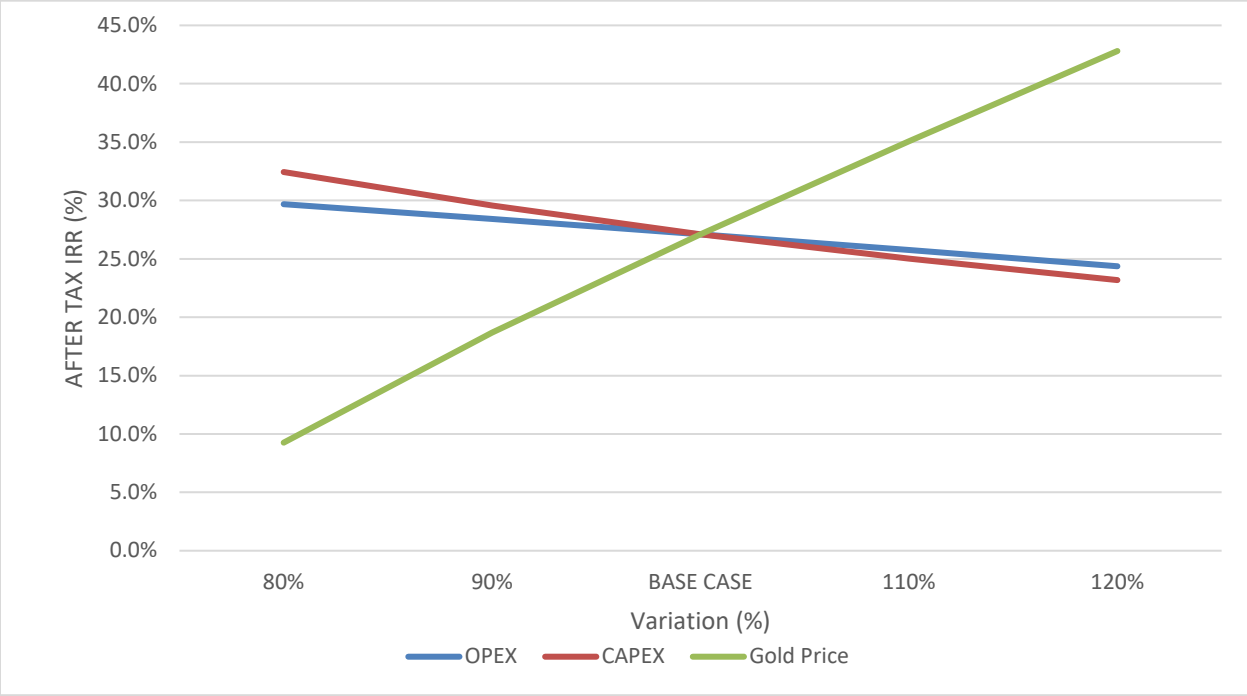
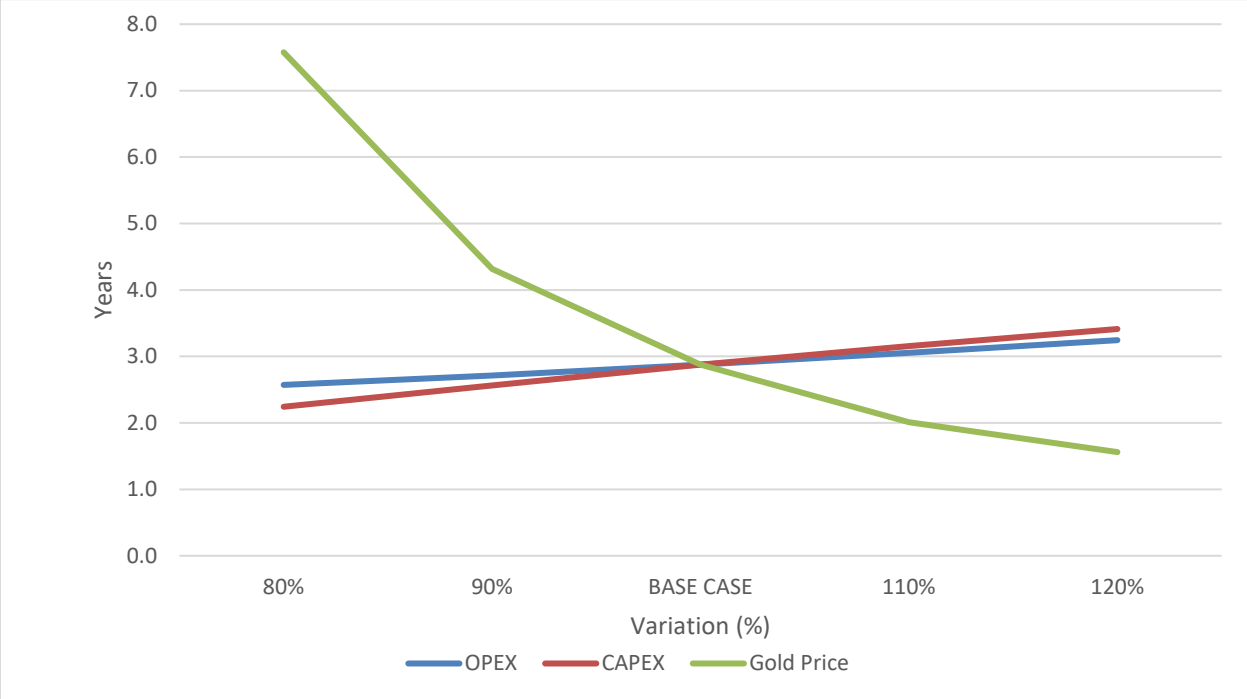


Figure 22.7: After-Tax Payback Period Sensitivity



23. ADJACENT PROPERTIES

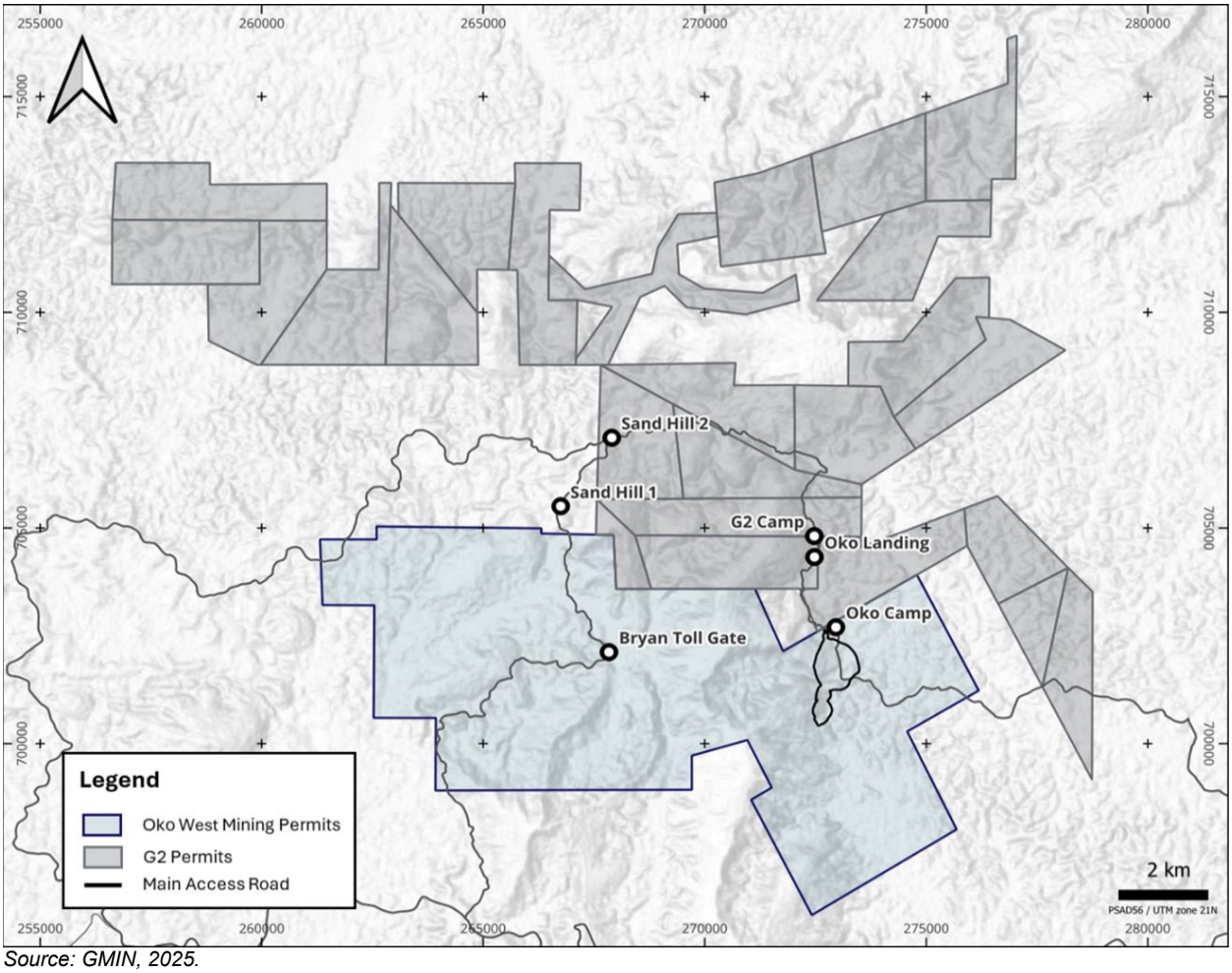
According to the GGMC, the Oko West Prospecting Licence is surrounded by 13 medium-scale mining and prospecting permits held by various Guyanese title holders and one group of medium-scale mining and prospecting permits controlled by G2 Goldfields Inc. (Figure 23.1).

The Company is unaware of any ongoing exploration work on the 13 medium-scale permits held by local entrepreneurs. Significant artisanal mining activity is present on the permits near the community of Sand Hill and within the permits controlled by G2 Goldfields. There is no record of historical artisanal gold production from these permits.

The section below describes the exploration work being done in the permits controlled by G2 Goldfields. A Technical Report in relation to the G2 Goldfields property was filed on SEDAR on April 24, 2025, and is available at www.sedarplus.ca. The information pertaining to the mineral resource update is based on a news release dated March 10, 2025.

The QP has been unable to verify the information on the adjacent property and the information provided herein is not necessarily indicative of the mineralization on the Oko West Project.

Figure 23.1: Oko West Prospecting Licence (PL) and Adjacent Mineral Permits



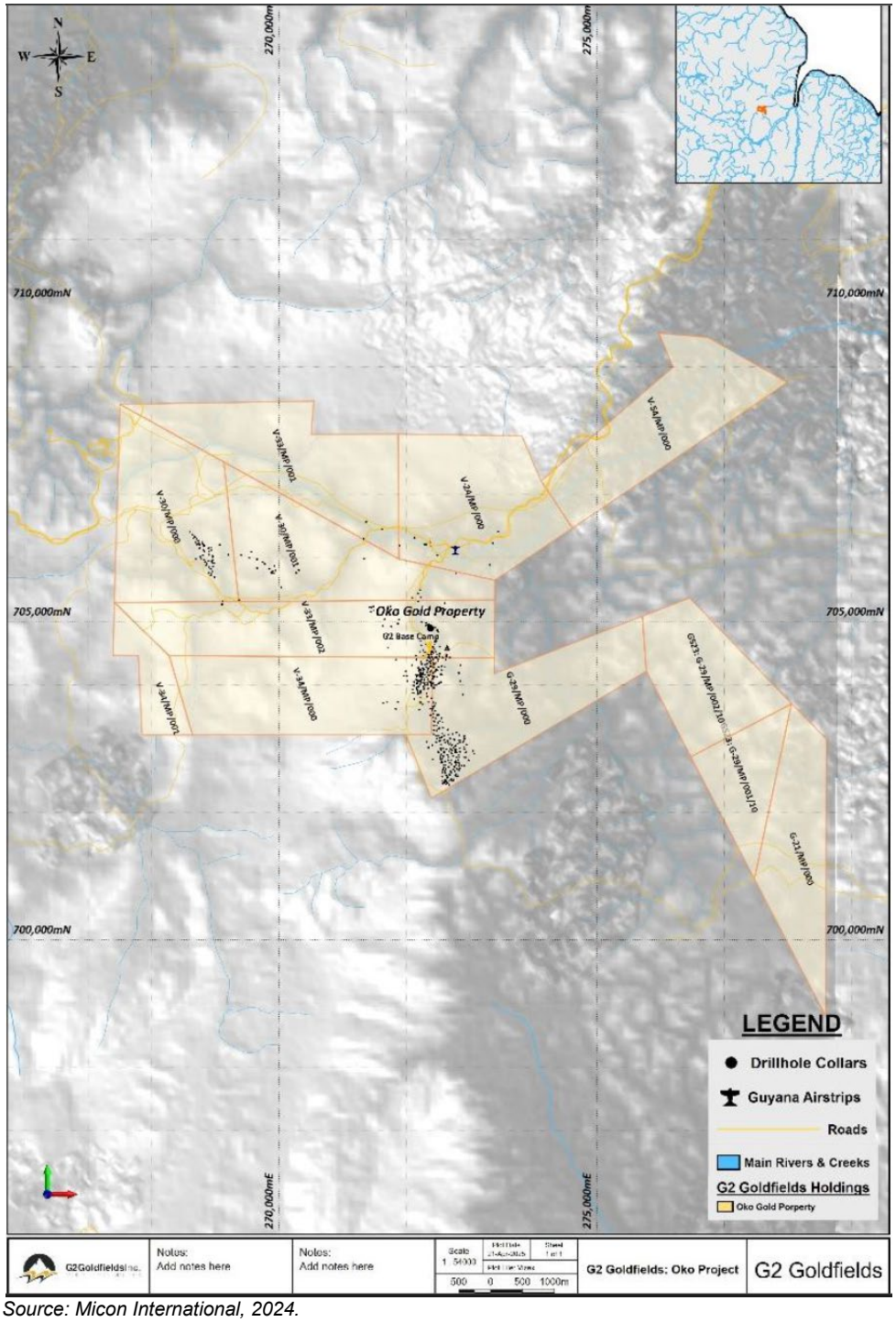
Source: GMIN, 2025.

23.1 G2 Goldfields Inc.

23.1.1 Mineral Rights

In 2016, Sandy Lake Gold Inc. collected grab samples at Crusher Hill, a gold prospect north of Oko West, reporting high gold grades in shaft stockpiles associated with quartz, and quartz-carbonate veins in narrow mineralized zones (Ilieva, 2018). This reconnaissance became the basis for Sandy Lake Gold changing its name to G2 Goldfields (G2), to seek an option agreement with the title holder. In December 2017, G2 acquired an interest in additional medium-scale mining permits. G2's permit portfolio has since increased to 18 medium-scale prospecting and mining permits covering 18,837 acres (7,623 ha) (Figure 23.2). There are several artisanal gold operations located on the permits, including two (2) shafts. Late 2024, G2 announced their intent to undertake a reorganization transaction whereby it would, among other things, spin-out its non-core assets into a new wholly owned subsidiary named G3 Goldfields Inc. (See press releases dated November 29, 2024; December 12, 2024; January 5, 2025, and January 28, 2025).

Figure 23.2: Mineral Title Held by G2 Goldfields Pertaining to the Oko Gold Project



Source: Micon International, 2024.

23.1.2 Exploration Work

In 2018 and 2019, G2 Goldfields completed a soil geochemical survey and geological mapping, and the results were used for outlining soil anomalies and drill-hole targeting. Since September 2019, G2 has

conducted diamond drilling programs targeting areas with known small-scale mining operations, particularly below the Crusher Hill pit. From September 2019 to March 2022, G2 drilled 116 drillholes numbered from OKD 01 to OKD 116 for a total of 28,809 m in three (3) areas called Oko Main zone, Oko Northwest and Oko South (Ghani zone). A second drilling campaign was launched in March 2022 and is still ongoing at the time of writing. This program aims to identify new gold-bearing geological structures and better define known ones. Some 587 surface drillholes for a total of 152,817 m have been drilled between March 2022 and January 2025.

23.1.3 Mineral Resources

G2 Goldfields contracted Micon International Ltd. to undertake an update of the mineral resource estimate of the Oko Gold Project. The mineral resource estimate includes the "Oko Main" deposit, as well as a newly defined area south of the Oko Main called the "Ghanie" zone. The mineral resource estimate is based on the drilling information available up to January 2025 (G2 Press Release, 2025), with an effective date of March 1, 2025. Micon classified the mineral resources at the Oko Gold Project into the Indicated and Inferred categories, assuming both open pit and underground operations. Micon used a gold price of US$2,281/ounce, with a recovery of 85%, a mining cost of US$2.5/t in saprolite, US$2.75/t in fresh rock, and US$75.0/t in underground. The processing costs were US$12/t for saprolite and US$15/t for fresh rock, and the general administration cost was of US$2.5/t. Cut-off grades of 0.27 g/t Au in saprolite, 0.32 g/t Au in fresh rock for open pit operations, and 1.48 g/t Au for underground mining were used. Open pit and Underground Indicated Mineral Resources are estimated at 3,147 kt, grading 7.98g/t Au for 808 koz Au for the Oko Main zone and 10,288 kt, grading 2.01 g/t Au for 663.4 koz Au for the Ghanie zone. Open pit and Underground Inferred Mineral Resources are estimated at 3,473 kt, grading 4.60 g/t Au for 513,5 koz Au for the Oko Main zone and 12,062 kt, grading 2.64 g/t Au for 1,024.5 koz Au for the Ghanie zone. An additional open pit Inferred resource estimated at 4,976 kt, grading 0.61 g/t Au for 97.2 koz Au was defined for the Oko Northwest zone.

24. OTHER RELEVANT DATA AND INFORMATION

This chapter presents other relevant data and information that are pertinent to the understanding and evaluation of the mineral project but are not covered in the preceding sections. The information herein aims to provide a comprehensive overview of the project execution and ensure that all relevant aspects are considered in the assessment of the project.

24.1 Project Execution Plan

An integrated project management team (IPMT) have been created to lead the execution of the Oko West Project using a self-perform approach. The project team will be supplemented by contractors working within the IPMT for both specialized needs and peak workforce requirements. The plan is for the IPMT to lead the project execution and construction of all on-site infrastructure and the process plant. Mine development will also be self-performed by the project team. Off-site infrastructure, including the access road and the Oko Wharf Landing, will be built or upgraded by a contractor under the supervision of the IPMT.

The project team will work in unison to achieve project objectives through the effective use of the Owners' equipment, material and personnel, and to minimize difficulties common to commissioning and start-up. The IPMT will share the responsibility for the planning and execution of the project.

The project team will use a quality assurance / quality control (QA/QC) system in all phases of the project (engineering, procurement, construction, commissioning and start-up). The system will include audits, certification, factory inspection, and destructive and non-destructive testing during construction, as appropriate. It will also provide traceability from origin for each component of the project. The detailed procedures and practices will be developed with the project QA/QC team integrating the project document control process.

The Mine and Mill operations team will be recruited during construction and will work within the IPMT during construction, pre-commissioning, commissioning, start-up, and handover of the facilities. Handover will be a structured and planned process that will require strong coordination between the project personnel and operations team for a gradual transfer of responsibility.

All departments included in General Services will be staffed with owner employees to service the construction efforts. These departments will become fully operational early in the construction period, with a trained workforce and established programs and service providers. As construction is demobilized, the service departments will seamlessly continue into operations.

The Owner mining team will consist of operations, maintenance and technical services and will be a critical component of the overall project team as it will execute a great deal of the mass earth movement.

The project team will be headed by a project director and a general manager who will provide overall project management during construction. The Owner operations team will take full control of the operation after commercial production is achieved. Commercial production is defined as the first day of a period of 30 days when the ore tonnage processed averaged at least 60% of nameplate.

24.2 Project Schedule

Schedule development begins at the management level and drills down through the project / control levels. The management level schedule is used to establish work goals and overall time frames for the scope of work. Project objectives are detailed in Figure 24.1. This Level 1 schedule contains the least amount of detail but provides a high-level tool for management to evaluate and track the main project milestones. The Project schedule actually define the detailed tasks and the duration of each task.

A preliminary Level 2 project schedule is provided in Figure 24.2. It will be further detailed and optimized as the Project goes through the next stages of Project Execution.

As the Project is under pressure by the authorities to be fast tracked, the project schedule started with early works activities in March 2025 followed by the full detailed engineering and execution.

The schedule for the Oko Gold Project spans from late August 2024 to the end of December 2027, covering over three (3) years of multidisciplinary project development. It includes a comprehensive permitting phase, starting with early works authorizations, and progressing to environmental and construction permits through agencies, such as EPA, GGMC, GFC, and MARAD—running mostly through 2025 to 2026. Concurrently, procurement activities and early site infrastructure (roads, camp facilities, wharf, and utilities) are launched. Major infrastructure components such as the process plant, power generation, water and sewage systems, and mining fleet mobilization are gradually implemented through overlapping workstreams during the entire period of 2025. Construction peaks between 2026 and 2027, followed by a commissioning phase starting in Q3 2027 and commercial production starting in January 2028.

Construction activities begin in parallel with early permitting and ramp up significantly through 2025. Key deliverables in 2025 include temporary construction infrastructure, earthwork site preparation, camp construction, OGM landing and external road. In 2026 and 2027, the major milestones include civil and mechanical works for the process plant, power generation and distribution facilities, water management systems, mining support infrastructure and the TSF dams. The schedule demonstrates structured execution

per work packages, including procurement, mobilization of heavy mining equipment, installation of key utilities, and construction of permanent buildings such as assay lab, administrative offices, truck shop. Mining equipment delivery and assembly are aligned with the completion of haul roads and mechanical workshops to support the progressive development of the open-pit and underground operations.

The final phase of the schedule focuses on commissioning and ramp-up. Commissioning is slated to begin in Q3 2027, followed by a full ramp-up throughout 2028, ensuring operational readiness and performance validation of all systems.

Figure 24.1: Oko West Project Schedule – Level 1

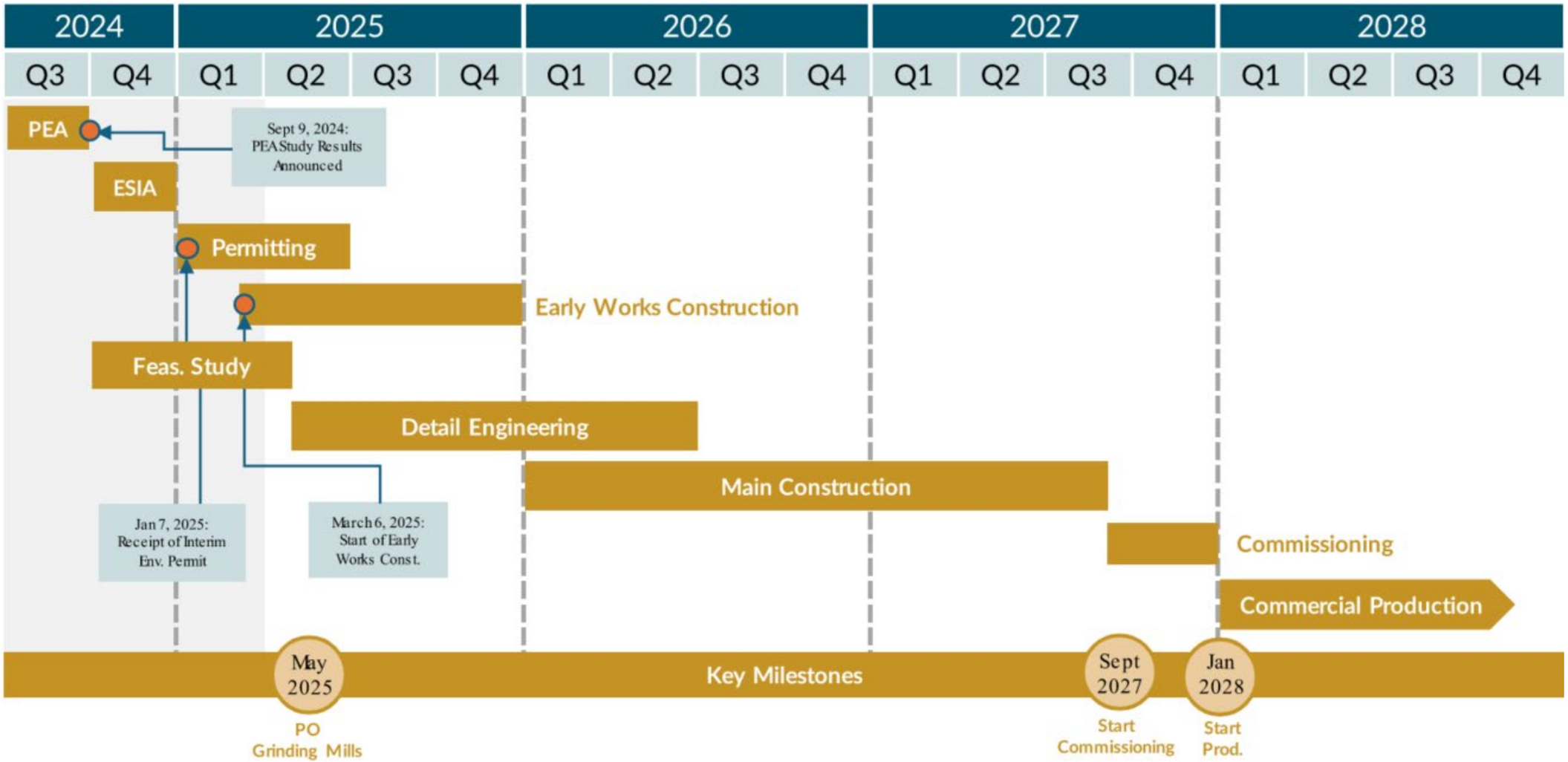
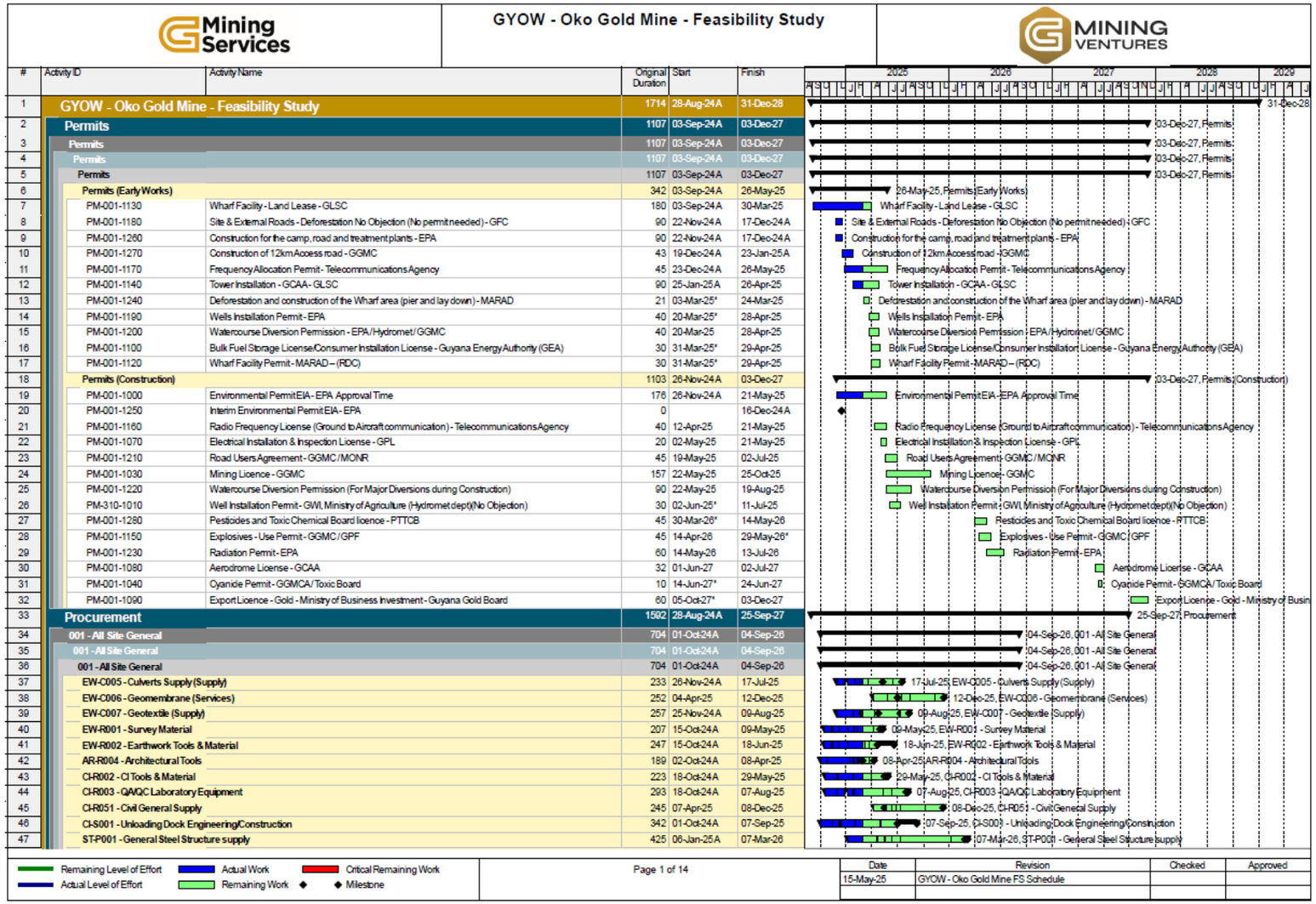
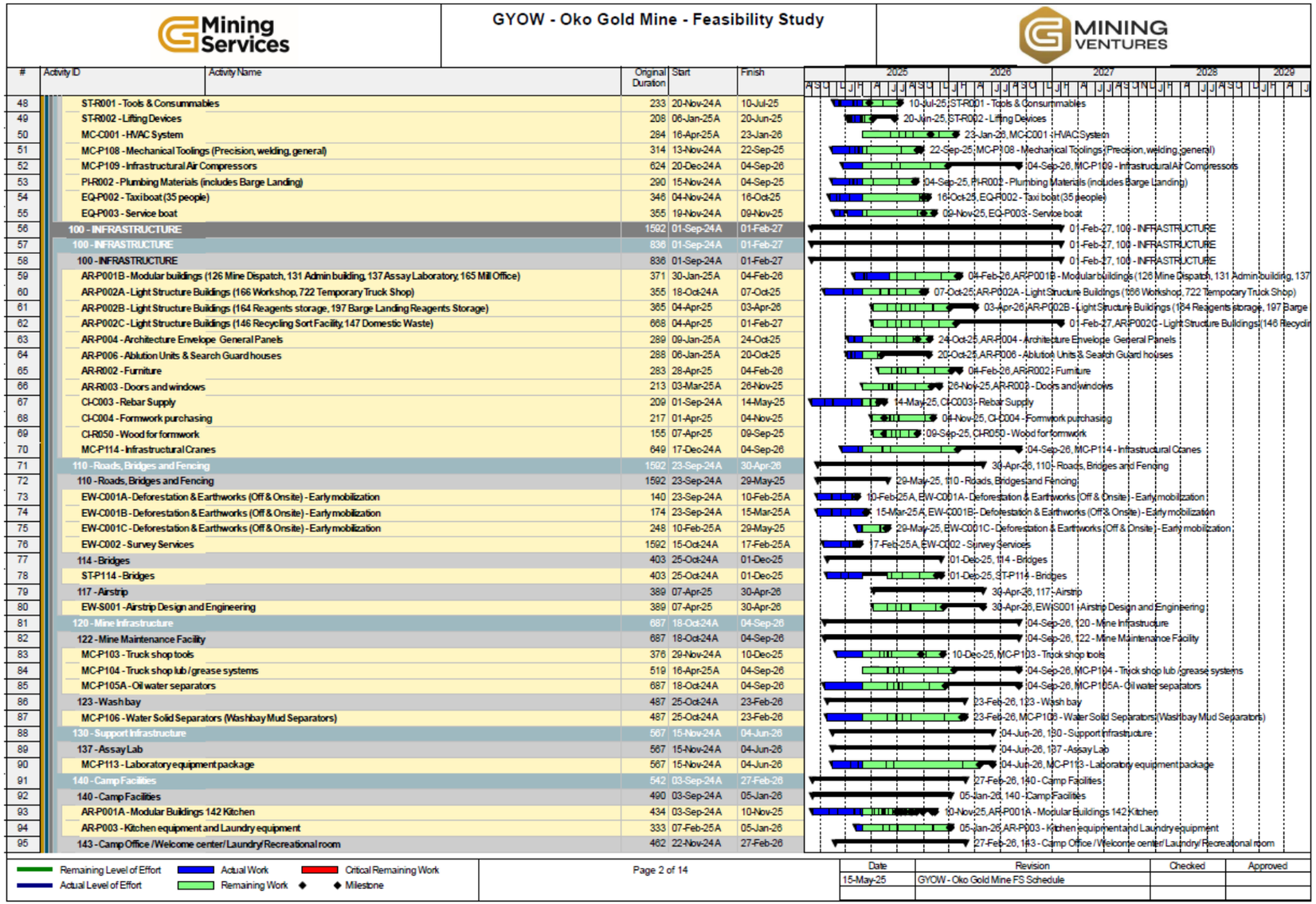


Figure 24.2: Oko West Project Schedule – Level 2

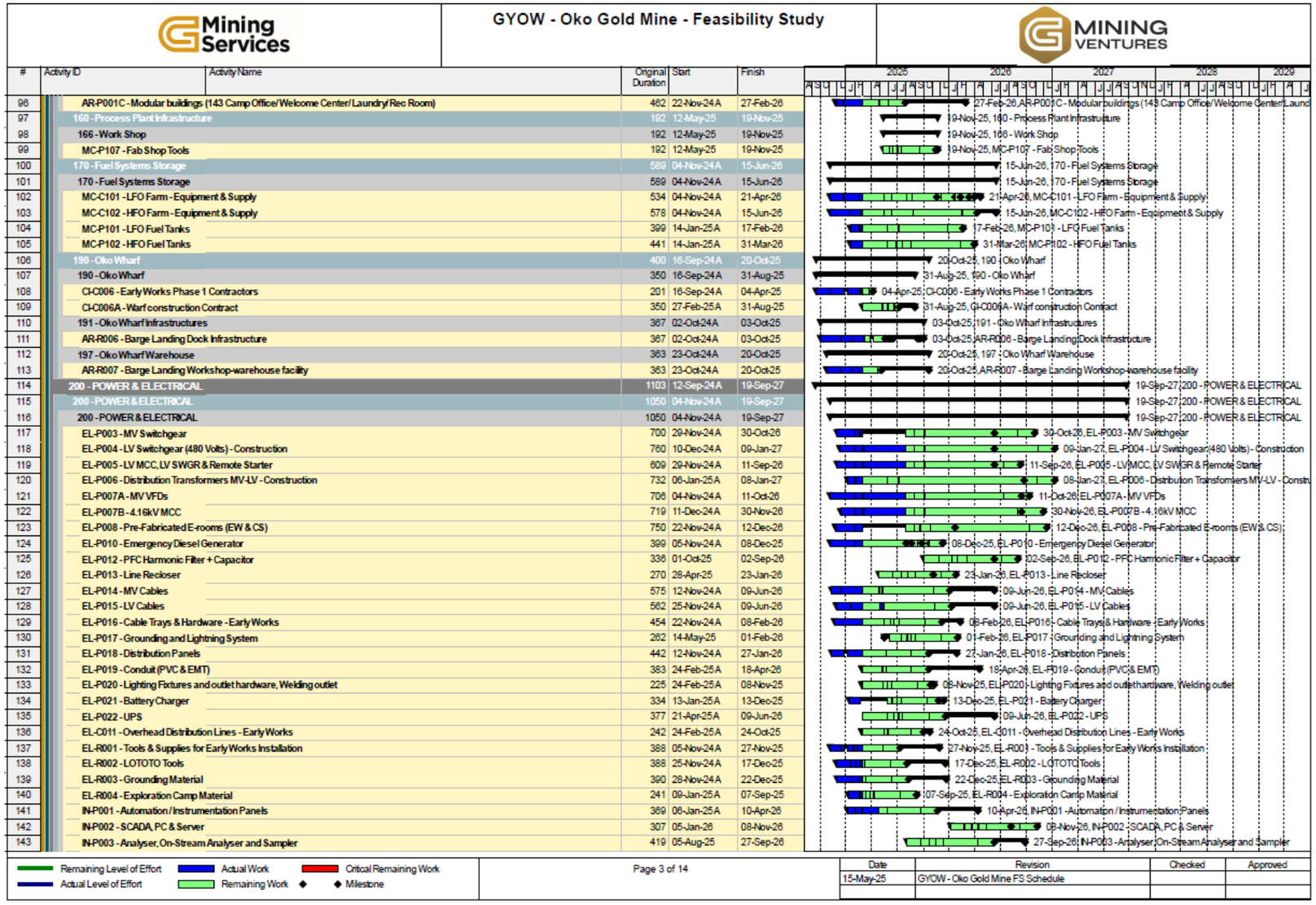


#	Activity ID	Activity Name	Original Duration	Start	Finish
1		GYOW - Oko Gold Mine - Feasibility Study	1714	28-Aug-24 A	31-Dec-28
2		Permits	1107	03-Sep-24 A	03-Dec-27
3		Permits	1107	03-Sep-24 A	03-Dec-27
4		Permits	1107	03-Sep-24 A	03-Dec-27
5		Permits	1107	03-Sep-24 A	03-Dec-27
6		Permits (Early Works)	342	03-Sep-24 A	26-May-25
7	PM-001-1130	Wharf Facility - Land Lease - GLSC	180	03-Sep-24 A	30-Mar-25
8	PM-001-1180	Site & External Roads - Deforestation No Objection (No permit needed) - GFC	90	22-Nov-24 A	17-Dec-24 A
9	PM-001-1260	Construction for the camp, road and treatment plants - EPA	90	22-Nov-24 A	17-Dec-24 A
10	PM-001-1270	Construction of 12km Access road - GGMC	43	19-Dec-24 A	23-Jan-25 A
11	PM-001-1170	Frequency Allocation Permit - Telecommunications Agency	45	23-Dec-24 A	26-May-25
12	PM-001-1140	Tower Installation - GCAA - GLSC	90	25-Jan-25 A	26-Apr-25
13	PM-001-1240	Deforestation and construction of the Wharf area (pier and lay down) - MARAD	21	03-Mar-25*	24-Mar-25
14	PM-001-1190	Wells Installation Permit - EPA	40	20-Mar-25*	28-Apr-25
15	PM-001-1200	Watercourse Diversion Permission - EPA/Hydromet/ GGMC	40	20-Mar-25	28-Apr-25
16	PM-001-1100	Bulk Fuel Storage License/Consumer Installation License - Guyana Energy Authority (GEA)	30	31-Mar-25*	29-Apr-25
17	PM-001-1120	Wharf Facility Permit - MARAD – (RDC)	30	31-Mar-25*	29-Apr-25
18		Permits (Construction)	1103	26-Nov-24 A	03-Dec-27
19	PM-001-1000	Environmental Permit EIA - EPA Approval Time	176	26-Nov-24 A	21-May-25
20	PM-001-1250	Interim Environmental Permit EIA - EPA	0		16-Dec-24 A
21	PM-001-1160	Radio Frequency License (Ground to Aircraft communication) - Telecommunications Agency	40	12-Apr-25	21-May-25
22	PM-001-1070	Electrical Installation & Inspection License - GPL	20	02-May-25	21-May-25
23	PM-001-1210	Road Users Agreement - GGMC / MONR	45	19-May-25	02-Jul-25
24	PM-001-1030	Mining Licence - GGMC	157	22-May-25	25-Oct-25
25	PM-001-1220	Watercourse Diversion Permission (For Major Diversions during Construction)	90	22-May-25	19-Aug-25
26	PM-310-1010	Well Installation Permit - GWI, Ministry of Agriculture (Hydromet dept)(No Objection)	30	02-Jun-25*	11-Jul-25
27	PM-001-1280	Pesticides and Toxic Chemical Board licence - PTTCB	45	30-Mar-26*	14-May-26
28	PM-001-1150	Explosives - Use Permit - GGMC / GPF	45	14-Apr-26	29-May-26*
29	PM-001-1230	Radiation Permit - EPA	60	14-May-26	13-Jul-26
30	PM-001-1080	Aerodrome License - GCAA	32	01-Jun-27	02-Jul-27
31	PM-001-1040	Cyanide Permit - GGMCA/ Toxic Board	10	14-Jun-27*	24-Jun-27
32	PM-001-1090	Export Licence - Gold - Ministry of Business Investment - Guyana Gold Board	60	05-Oct-27*	03-Dec-27
33		Procurement	1502	28-Aug-24 A	25-Sep-27
34		001 - All Site General	704	01-Oct-24 A	04-Sep-26
35		001 - All Site General	704	01-Oct-24 A	04-Sep-26
36		001 - All Site General	704	01-Oct-24 A	04-Sep-26
37		EW-C005 - Culverts Supply (Supply)	233	26-Nov-24 A	17-Jul-25
38		EW-C006 - Geomembrane (Services)	252	04-Apr-25	12-Dec-25
39		EW-C007 - Geotextile (Supply)	257	25-Nov-24 A	09-Aug-25
40		EW-R001 - Survey Material	207	15-Oct-24 A	09-May-25
41		EW-R002 - Earthwork Tools & Material	247	15-Oct-24 A	18-Jun-25
42		AR-R004 - Architectural Tools	189	02-Oct-24 A	08-Apr-25
43		CI-R002 - CI Tools & Material	223	18-Oct-24 A	29-May-25
44		CI-R003 - QA/QC Laboratory Equipment	293	18-Oct-24 A	07-Aug-25
45		CI-R051 - Civil General Supply	245	07-Apr-25	08-Dec-25
46		CI-S001 - Unloading Dock Engineering/Construction	342	01-Oct-24 A	07-Sep-25
47		ST-P001 - General Steel Structure supply	425	06-Jan-25 A	07-Mar-26

Source: GMS (May 2025)



#	Activity ID	Activity Name	Original Duration	Start	Finish
48	ST-R001	Tools & Consummables	233	20-Nov-24 A	10-Jul-25
49	ST-R002	Lifting Devices	208	06-Jan-25 A	20-Jun-25
50	MC-C001	HVAC System	284	16-Apr-25 A	23-Jan-26
51	MC-P108	Mechanical Toolings (Precision, welding, general)	314	13-Nov-24 A	22-Sep-25
52	MC-P109	Infrastructural Air Compressors	624	20-Dec-24 A	04-Sep-26
53	PI-R002	Plumbing Materials (includes Barge Landing)	290	15-Nov-24 A	04-Sep-25
54	EQ-P002	Taxi boat (35 people)	346	04-Nov-24 A	16-Oct-25
55	EQ-P003	Service boat	355	19-Nov-24 A	09-Nov-25
56	100	INFRASTRUCTURE	1592	01-Sep-24 A	01-Feb-27
57	100	INFRASTRUCTURE	836	01-Sep-24 A	01-Feb-27
58	100	INFRASTRUCTURE	836	01-Sep-24 A	01-Feb-27
59	AR-P001B	Modular buildings (126 Mine Dispatch, 131 Admin building, 137 Assay Laboratory, 165 Mill Office)	371	30-Jan-25 A	04-Feb-26
60	AR-P002A	Light Structure Buildings (166 Workshop, 722 Temporary Truck Shop)	355	18-Oct-24 A	07-Oct-25
61	AR-P002B	Light Structure Buildings (164 Reagents storage, 197 Barge Landing Reagents Storage)	365	04-Apr-25	03-Apr-26
62	AR-P002C	Light Structure Buildings (146 Recycling Sort Facility, 147 Domestic Waste)	668	04-Apr-25	01-Feb-27
63	AR-P004	Architecture Envelope General Panels	289	09-Jan-25 A	24-Oct-25
64	AR-P006	Ablution Units & Search Guard houses	288	06-Jan-25 A	20-Oct-25
65	AR-R002	Furniture	283	28-Apr-25	04-Feb-26
66	AR-R003	Doors and windows	213	03-Mar-25 A	26-Nov-25
67	CI-C003	Rebar Supply	209	01-Sep-24 A	14-May-25
68	CI-C004	Formwork purchasing	217	01-Apr-25	04-Nov-25
69	CI-R050	Wood for formwork	155	07-Apr-25	09-Sep-25
70	MC-P114	Infrastructural Cranes	649	17-Dec-24 A	04-Sep-26
71	110	Roads, Bridges and Fencing	1592	23-Sep-24 A	30-Apr-26
72	110	Roads, Bridges and Fencing	1592	23-Sep-24 A	29-May-25
73	EW-C001A	Deforestation & Earthworks (Off & Onsite) - Early mobilization	140	23-Sep-24 A	10-Feb-25 A
74	EW-C001B	Deforestation & Earthworks (Off & Onsite) - Early mobilization	174	23-Sep-24 A	15-Mar-25 A
75	EW-C001C	Deforestation & Earthworks (Off & Onsite) - Early mobilization	248	10-Feb-25 A	29-May-25
76	EW-C002	Survey Services	1592	15-Oct-24 A	17-Feb-25 A
77	114	Bridges	403	25-Oct-24 A	01-Dec-25
78	ST-P114	Bridges	403	25-Oct-24 A	01-Dec-25
79	117	Airstrip	389	07-Apr-25	30-Apr-26
80	EW-S001	Airstrip Design and Engineering	389	07-Apr-25	30-Apr-26
81	120	Mine Infrastructure	687	18-Oct-24 A	04-Sep-26
82	122	Mine Maintenance Facility	687	18-Oct-24 A	04-Sep-26
83	MC-P103	Truck shop tools	376	29-Nov-24 A	10-Dec-25
84	MC-P104	Truck shop lub / grease systems	519	16-Apr-25 A	04-Sep-26
85	MC-P105A	Oil water separators	687	18-Oct-24 A	04-Sep-26
86	123	Wash bay	487	25-Oct-24 A	23-Feb-26
87	MC-P106	Water Solid Separators (Washbay Mud Separators)	487	25-Oct-24 A	23-Feb-26
88	130	Support Infrastructure	567	15-Nov-24 A	04-Jun-26
89	137	Assay Lab	567	15-Nov-24 A	04-Jun-26
90	MC-P113	Laboratory equipment package	567	15-Nov-24 A	04-Jun-26
91	140	Camp Facilities	542	03-Sep-24 A	27-Feb-26
92	140	Camp Facilities	490	03-Sep-24 A	05-Jan-26
93	AR-P001A	Modular Buildings 142 Kitchen	434	03-Sep-24 A	10-Nov-25
94	AR-P003	Kitchen equipment and Laundry equipment	333	07-Feb-25 A	05-Jan-26
95	143	Camp Office / Welcome center/ Laundry/ Recreational room	462	22-Nov-24 A	27-Feb-26



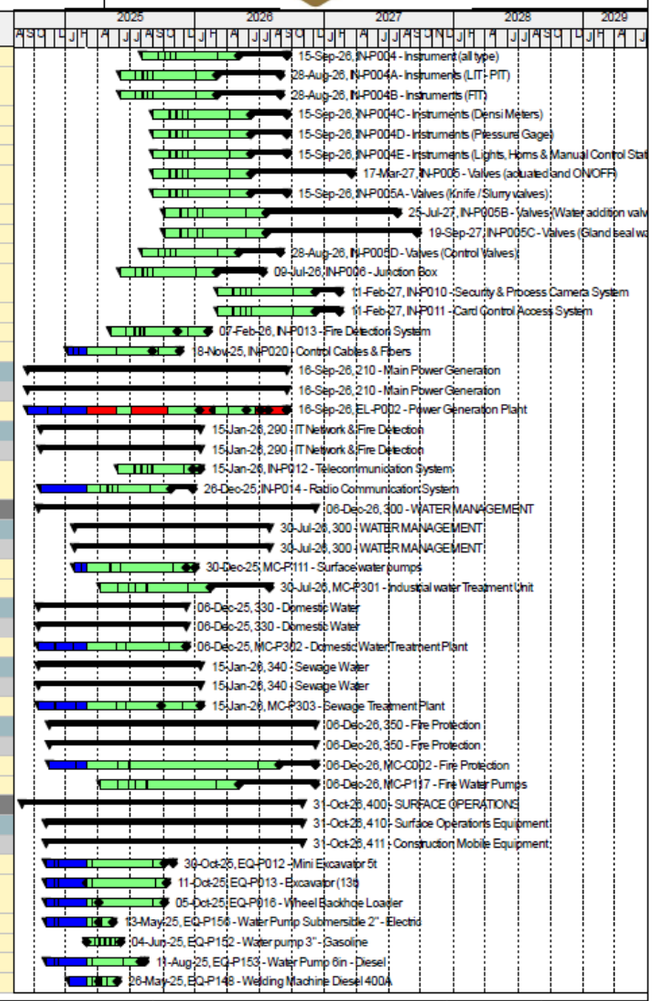
#	Activity ID	Activity Name	Original Duration	Start	Finish
96		AR-P001C - Modular buildings (143 Camp Office/Welcome Center/Laundry/Rec Room)	462	22-Nov-24 A	27-Feb-26
97		160 - Process Plant Infrastructure	192	12-May-25	19-Nov-25
98		166 - Work Shop	192	12-May-25	19-Nov-25
99		MC-P107 - Fab Shop Tools	192	12-May-25	19-Nov-25
100		170 - Fuel Systems Storage	589	04-Nov-24 A	15-Jun-26
101		170 - Fuel Systems Storage	589	04-Nov-24 A	15-Jun-26
102		MC-C101 - LFO Farm - Equipment & Supply	534	04-Nov-24 A	21-Apr-26
103		MC-C102 - HFO Farm - Equipment & Supply	578	04-Nov-24 A	15-Jun-26
104		MC-P101 - LFO Fuel Tanks	399	14-Jan-25 A	17-Feb-26
105		MC-P102 - HFO Fuel Tanks	441	14-Jan-25 A	31-Mar-26
106		190 - Oko Wharf	400	16-Sep-24 A	20-Oct-25
107		190 - Oko Wharf	350	16-Sep-24 A	31-Aug-25
108		CI-C006 - Early Works Phase 1 Contractors	201	16-Sep-24 A	04-Apr-25
109		CI-C006A - Warf construction Contract	350	27-Feb-25 A	31-Aug-25
110		191 - Oko Wharf Infrastructures	367	02-Oct-24 A	03-Oct-25
111		AR-R006 - Barge Landing Dock Infrastructure	367	02-Oct-24 A	03-Oct-25
112		197 - Oko Wharf Warehouse	363	23-Oct-24 A	20-Oct-25
113		AR-R007 - Barge Landing Workshop-warehouse facility	363	23-Oct-24 A	20-Oct-25
114		200 - POWER & ELECTRICAL	1103	12-Sep-24 A	19-Sep-27
115		200 - POWER & ELECTRICAL	1050	04-Nov-24 A	19-Sep-27
116		200 - POWER & ELECTRICAL	1050	04-Nov-24 A	19-Sep-27
117		EL-P003 - MV Switchgear	700	29-Nov-24 A	30-Oct-26
118		EL-P004 - LV Switchgear (480 Volts) - Construction	760	10-Dec-24 A	09-Jan-27
119		EL-P005 - LV MCC, LV SWGR & Remote Starter	609	29-Nov-24 A	11-Sep-26
120		EL-P006 - Distribution Transformers MV-LV - Construction	732	06-Jan-25 A	08-Jan-27
121		EL-P007A - MV VFDs	706	04-Nov-24 A	11-Oct-26
122		EL-P007B - 4.16kV MCC	719	11-Dec-24 A	30-Nov-26
123		EL-P008 - Pre-Fabricated E-rooms (EW & CS)	750	22-Nov-24 A	12-Dec-26
124		EL-P010 - Emergency Diesel Generator	399	05-Nov-24 A	08-Dec-25
125		EL-P012 - PFC Harmonic Filter + Capacitor	336	01-Oct-25	02-Sep-26
126		EL-P013 - Line Recloser	270	28-Apr-25	23-Jan-26
127		EL-P014 - MV Cables	575	12-Nov-24 A	09-Jun-26
128		EL-P015 - LV Cables	562	25-Nov-24 A	09-Jun-26
129		EL-P016 - Cable Trays & Hardware - Early Works	454	22-Nov-24 A	08-Feb-26
130		EL-P017 - Grounding and Lightning System	262	14-May-25	01-Feb-26
131		EL-P018 - Distribution Panels	442	12-Nov-24 A	27-Jan-26
132		EL-P019 - Conduit (PVC & EMT)	383	24-Feb-25 A	18-Apr-26
133		EL-P020 - Lighting Fixtures and outlet hardware, Welding outlet	225	24-Feb-25 A	08-Nov-25
134		EL-P021 - Battery Charger	334	13-Jan-25 A	13-Dec-25
135		EL-P022 - UPS	377	21-Apr-25 A	09-Jun-26
136		EL-C011 - Overhead Distribution Lines - Early Works	242	24-Feb-25 A	24-Oct-25
137		EL-R001 - Tools & Supplies for Early Works Installation	388	05-Nov-24 A	27-Nov-25
138		EL-R002 - LOTOTO Tools	388	25-Nov-24 A	17-Dec-25
139		EL-R003 - Grounding Material	390	28-Nov-24 A	22-Dec-25
140		EL-R004 - Exploration Camp Material	241	09-Jan-25 A	07-Sep-25
141		IN-P001 - Automation / Instrumentation Panels	369	06-Jan-25 A	10-Apr-26
142		IN-P002 - SCADA, PC & Server	307	05-Jan-26	08-Nov-26
143		IN-P003 - Analyser, On-Stream Analyser and Sampler	419	05-Aug-25	27-Sep-26

Mining Services

GYOW - Oko Gold Mine - Feasibility Study

MINING VENTURES

#	Activity ID	Activity Name	Original Duration	Start	Finish
144	IN-P004	Instrument (all type)	407	05-Aug-25	15-Sep-26
145	IN-P004A	Instruments (LIT - PIT)	453	02-Jun-25	28-Aug-26
146	IN-P004B	Instruments (FIT)	453	02-Jun-25	28-Aug-26
147	IN-P004C	Instruments (Densi Meters)	376	05-Sep-25	15-Sep-26
148	IN-P004D	Instruments (Pressure Gage)	376	05-Sep-25	15-Sep-26
149	IN-P004E	Instruments (Lights, Horns & Manual Control Station)	376	05-Sep-25	15-Sep-26
150	IN-P005	Valves (actuated and ON/OFF)	559	05-Sep-25	17-Mar-27
151	IN-P005A	Valves (Knife / Slurry valves)	376	05-Sep-25	15-Sep-26
152	IN-P005B	Valves (Water addition valve)	658	06-Oct-25	25-Jul-27
153	IN-P005C	Valves (Gland seal water valve)	714	06-Oct-25	19-Sep-27
154	IN-P005D	Valves (Control Valves)	390	04-Aug-25	28-Aug-26
155	IN-P006	Junction Box	403	02-Jun-25	09-Jul-26
156	IN-P010	Security & Process Camera System	344	05-Mar-26	11-Feb-27
157	IN-P011	Card Control Access System	344	05-Mar-26	11-Feb-27
158	IN-P013	Fire Detection System	276	07-May-25	07-Feb-26
159	IN-P020	Control Cables & Fibers	225	06-Jan-25 A	18-Nov-25
160	210	Main Power Generation	735	12-Sep-24 A	16-Sep-26
161	210	Main Power Generation	735	12-Sep-24 A	16-Sep-26
162	EL-P002	Power Generation Plant	735	12-Sep-24 A	16-Sep-26
163	290	IT Network & Fire Detection	451	21-Oct-24 A	15-Jan-26
164	290	IT Network & Fire Detection	451	21-Oct-24 A	15-Jan-26
165	IN-P012	Telecommunication System	231	29-May-25	15-Jan-26
166	IN-P014	Radio Communication System	431	21-Oct-24 A	26-Dec-25
167	300	WATER MANAGEMENT	783	15-Oct-24 A	06-Dec-26
168	300	WATER MANAGEMENT	552	24-Jan-25 A	30-Jul-26
169	300	WATER MANAGEMENT	552	24-Jan-25 A	30-Jul-26
170	MC-P111	Surface water pumps	340	24-Jan-25 A	30-Dec-25
171	MC-P301	Industrial water Treatment Unit	479	07-Apr-25	30-Jul-26
172	330	Domestic Water	417	15-Oct-24 A	06-Dec-25
173	330	Domestic Water	417	15-Oct-24 A	06-Dec-25
174	MC-P302	Domestic Water Treatment Plant	417	15-Oct-24 A	06-Dec-25
175	340	Sewage Water	457	15-Oct-24 A	15-Jan-26
176	340	Sewage Water	457	15-Oct-24 A	15-Jan-26
177	MC-P303	Sewage Treatment Plant	457	15-Oct-24 A	15-Jan-26
178	350	Fire Protection	745	15-Nov-24 A	06-Dec-26
179	350	Fire Protection	745	15-Nov-24 A	06-Dec-26
180	MC-C002	Fire Protection	745	15-Nov-24 A	06-Dec-26
181	MC-P117	Fire Water Pumps	609	07-Apr-25	06-Dec-26
182	400	SURFACE OPERATIONS	795	28-Aug-24 A	31-Oct-26
183	410	Surface Operations Equipment	727	04-Nov-24 A	31-Oct-26
184	411	Construction Mobile Equipment	727	04-Nov-24 A	31-Oct-26
185	EQ-P012	Mini Excavator 5t	361	04-Nov-24 A	30-Oct-25
186	EQ-P013	Excavator (13t)	341	04-Nov-24 A	11-Oct-25
187	EQ-P016	Wheel Backhoe Loader	335	04-Nov-24 A	05-Oct-25
188	EQ-P156	Water Pump Submersible 2" - Electric	190	04-Nov-24 A	13-May-25
189	EQ-P152	Water pump 3" - Gasoline	96	28-Feb-25	04-Jun-25
190	EQ-P153	Water Pump 6in - Diesel	280	04-Nov-24 A	11-Aug-25
191	EQ-P148	Welding Machine Diesel 400A	136	10-Jan-25 A	26-May-25



Remaining Level of Effort | Actual Work | Critical Remaining Work
Actual Level of Effort | Remaining Work | Milestone

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Date	Revision	Checked	Approved
15-May-25	GYOW - Oko Gold Mine FS Schedule		

Mining Services

GYOW - Oko Gold Mine - Feasibility Study

MINING VENTURES

#	Activity ID	Activity Name	Original Duration	Start	Finish
192	EQ-P149	Welding Machine Diesel 500A	245	28-Feb-25	30-Oct-25
193	EQ-P147	Welding Machine Electric	245	28-Feb-25	30-Oct-25
194	EQ-P140	Light Plant	160	04-Nov-24 A	13-Apr-25
195	EQ-P143	Mobile Air Compressor 185CFM	311	04-Nov-24 A	11-Sep-25
196	EQ-P075	Pick-up Truck	160	04-Nov-24 A	13-Apr-25
197	EQ-P062	Pick-up Trailers	231	12-Dec-24 A	30-Jul-25
198	EQ-P104	Telehandler 5t	250	04-Nov-24 A	12-Jul-25
199	EQ-P108	Forklift Diesel 2.5t	332	04-Nov-24 A	01-Oct-25
200	EQ-P096	Crane Carry Deck 15t	323	12-Dec-24 A	30-Oct-25
201	EQ-P097	Crane Crawler 230t	359	04-Nov-24 A	29-Oct-25
202	EQ-P098	Crane Rough Terrain 80t	285	04-Nov-24 A	16-Aug-25
203	EQ-P095	Crane Rough Terrain 130t	285	04-Nov-24 A	16-Aug-25
204	EQ-P112	Manlift 45ft	727	04-Nov-24 A	31-Oct-26
205	EQ-P113	Manlift 60ft	314	04-Nov-24 A	14-Sep-25
206	EQ-P114	Manlift 80ft	361	04-Nov-24 A	30-Oct-25
207	EQ-P116	ScissorLift 19ft Electric	361	04-Nov-24 A	30-Oct-25
208	EQ-P117	ScissorLift 26ft Electric	361	04-Nov-24 A	30-Oct-25
209	EQ-P118	ScissorLift 33ft Diesel Rough Terrain	293	04-Nov-24 A	24-Aug-25
210	EQ-P061	Truck Flatbed 5.5t	226	12-Dec-24 A	26-Jul-25
211	EQ-P120	Skid Steer Loader	361	04-Nov-24 A	30-Oct-25
212	EQ-P159	Man Basket	184	28-Feb-25	30-Aug-25
213	EQ-P069	Concrete truck	314	04-Nov-24 A	14-Sep-25
214	414	G&A Mobile Equipment	524	04-Nov-24 A	12-Apr-26
215	EQ-P103	Telehandler 4t	524	04-Nov-24 A	12-Apr-26
216	EQ-P109	Forklift Diesel 5T	175	04-Nov-24 A	28-Apr-25
217	EQ-P058	Knuckle Boom Truck (10t)	166	12-Dec-24 A	27-May-25
218	EQ-P077	Ambulance	166	12-Dec-24 A	27-May-25
219	EQ-P067	Fire Fighting Foam Trailer	245	28-Feb-25	30-Oct-25
220	EQ-P055	Multipurpose Truck - Platform/Waste	172	12-Dec-24 A	01-Jun-25
221	EQ-P158	Waste Containers	111	28-Feb-25	19-Jun-25
222	EQ-P136	Quad	111	28-Feb-25	19-Jun-25
223	EQ-P066	Bus	216	28-Feb-25	02-Oct-25
224	415	Off-Site surface Ops Equipment	361	04-Nov-24 A	30-Oct-25
225	EQ-P105	Telehandler 6t	175	04-Nov-24 A	28-Apr-25
226	EQ-P064	Truck Tractor for trailers	256	12-Dec-24 A	25-Aug-25
227	EQ-P082	53' Flat Bed Trailer	151	12-Dec-24 A	12-May-25
228	EQ-P083	53' Low Bed Trailer	226	12-Dec-24 A	26-Jul-25
229	EQ-P070	Truck Tractor for trailers lowboy	255	12-Dec-24 A	24-Aug-25
230	EQ-P084	Fuel tank trailer (45k l)	256	12-Dec-24 A	25-Aug-25
231	EQ-P021	10 wheel Dump Truck	361	04-Nov-24 A	30-Oct-25
232	430	Concrete Batch Plant	484	28-Aug-24 A	24-Dec-25
233	431	Concrete Batch Plant Construction	484	28-Aug-24 A	24-Dec-25
234	CI-C001	Concrete Pump + Truck	455	26-Sep-24 A	24-Dec-25
235	CI-C002	Concrete Batch Plant	317	28-Aug-24 A	10-Jul-25
236	CI-C005	Concrete QAQC (Services)	269	04-Oct-24 A	29-Jun-25
237	CI-C007	Cement Supply	253	04-Sep-24 A	14-May-25
238	CI-C008	Concrete aggregate (Supply)	224	03-Oct-24 A	14-May-25
239	CI-R001	Concrete Pump + Truck (Rental)	244	26-Sep-24 A	09-Oct-25

2025 2026 2027 2028 2029

Remaining Level of Effort
Actual Level of Effort
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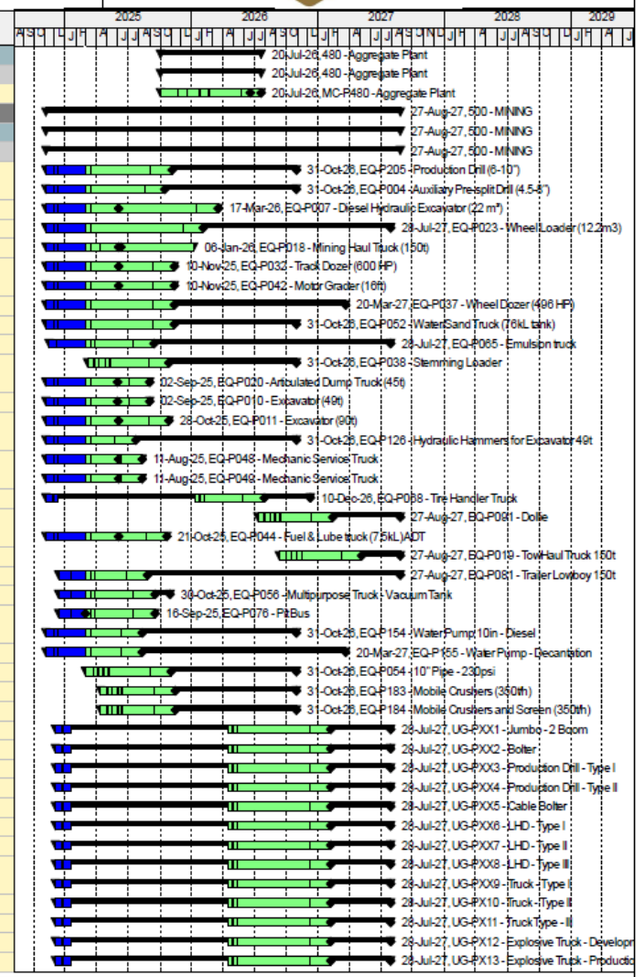
Mining Services

GYOW - Oko Gold Mine - Feasibility Study

MINING VENTURES

#	Activity ID	Activity Name	Original Duration	Start	Finish
240	480 - Aggregate Plant		292	01-Oct-25	20-Jul-26
241	480 - Aggregate Plant		292	01-Oct-25	20-Jul-26
242	MC-P480 - Aggregate Plant		292	01-Oct-25	20-Jul-26
243	500 - MINING		1027	04-Nov-24 A	27-Aug-27
244	500 - MINING		1027	04-Nov-24 A	27-Aug-27
245	500 - MINING		1027	04-Nov-24 A	27-Aug-27
246	EQ-P205 - Production Drill (6-10")		727	04-Nov-24 A	31-Oct-26
247	EQ-P004 - Auxiliary Pre-split Drill (4.5-8")		727	04-Nov-24 A	31-Oct-26
248	EQ-P007 - Diesel Hydraulic Excavator (22 m³)		498	04-Nov-24 A	17-Mar-26
249	EQ-P023 - Wheel Loader (12.2m3)		997	04-Nov-24 A	28-Jul-27
250	EQ-P018 - Mining Haul Truck (150t)		429	04-Nov-24 A	06-Jan-26
251	EQ-P032 - Track Dozer (600 HP)		371	04-Nov-24 A	10-Nov-25
252	EQ-P042 - Motor Grader (16ft)		371	04-Nov-24 A	10-Nov-25
253	EQ-P037 - Wheel Dozer (496 HP)		867	04-Nov-24 A	20-Mar-27
254	EQ-P052 - Water/Sand Truck (76kL tank)		727	04-Nov-24 A	31-Oct-26
255	EQ-P065 - Emulsion truck		986	15-Nov-24 A	28-Jul-27
256	EQ-P038 - Stemming Loader		606	05-Mar-25	31-Oct-26
257	EQ-P020 - Articulated Dump Truck (45t)		302	04-Nov-24 A	02-Sep-25
258	EQ-P010 - Excavator (49t)		302	04-Nov-24 A	02-Sep-25
259	EQ-P011 - Excavator (90t)		358	04-Nov-24 A	28-Oct-25
260	EQ-P126 - Hydraulic Hammers for Excavator 49t		727	04-Nov-24 A	31-Oct-26
261	EQ-P048 - Mechanic Service Truck		280	04-Nov-24 A	11-Aug-25
262	EQ-P049 - Mechanic Service Truck		280	04-Nov-24 A	11-Aug-25
263	EQ-P068 - Tire Handler Truck		365	04-Nov-24 A	10-Dec-26
264	EQ-P091 - Dollie		414	10-Jul-26	27-Aug-27
265	EQ-P044 - Fuel & Lube truck (7.5kL) ADT		351	04-Nov-24 A	21-Oct-25
266	EQ-P019 - TowHaul Truck 150t		352	10-Sep-26	27-Aug-27
267	EQ-P081 - Trailer Lowboy 150t		989	12-Dec-24 A	27-Aug-27
268	EQ-P056 - Multipurpose Truck - Vacuum Tank		323	12-Dec-24 A	30-Oct-25
269	EQ-P076 - Pit Bus		278	12-Dec-24 A	16-Sep-25
270	EQ-P154 - Water Pump 10in - Diesel		727	04-Nov-24 A	31-Oct-26
271	EQ-P155 - Water Pump - Decantation		867	04-Nov-24 A	20-Mar-27
272	EQ-P054 - 10" Pipe - 230psi		611	28-Feb-25	31-Oct-26
273	EQ-P183 - Mobile Crushers (350t/h)		570	10-Apr-25	31-Oct-26
274	EQ-P184 - Mobile Crushers and Screen (350t/h)		570	10-Apr-25	31-Oct-26
275	UG-PXX1 - Jumbo - 2 Boom		969	02-Dec-24 A	28-Jul-27
276	UG-PXX2 - Bolter		969	02-Dec-24 A	28-Jul-27
277	UG-PXX3 - Production Drill - Type I		969	02-Dec-24 A	28-Jul-27
278	UG-PXX4 - Production Drill - Type II		969	02-Dec-24 A	28-Jul-27
279	UG-PXX5 - Cable Bolter		969	02-Dec-24 A	28-Jul-27
280	UG-PXX6 - LHD - Type I		969	02-Dec-24 A	28-Jul-27
281	UG-PXX7 - LHD - Type II		969	02-Dec-24 A	28-Jul-27
282	UG-PXX8 - LHD - Type III		969	02-Dec-24 A	28-Jul-27
283	UG-PXX9 - Truck - Type I		969	02-Dec-24 A	28-Jul-27
284	UG-PX10 - Truck - Type II		969	02-Dec-24 A	28-Jul-27
285	UG-PX11 - Truck Type - III		969	02-Dec-24 A	28-Jul-27
286	UG-PX12 - Explosive Truck - Development		969	02-Dec-24 A	28-Jul-27
287	UG-PX13 - Explosive Truck - Production		969	02-Dec-24 A	28-Jul-27



Remaining Level of Effort | Actual Work | Critical Remaining Work
Actual Level of Effort | Remaining Work | Milestone

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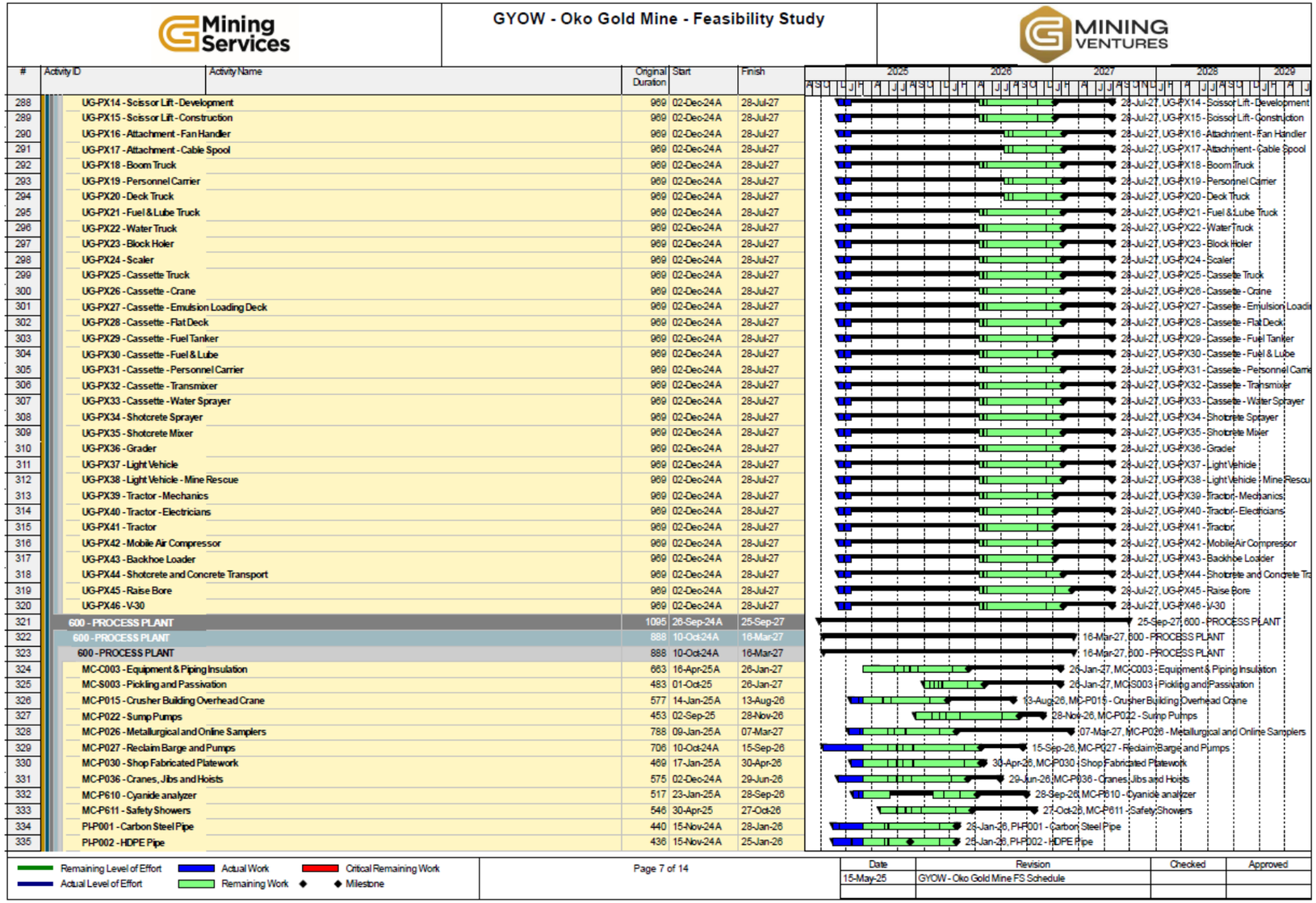
Date	Revision	Checked	Approved
15-May-25	GYOW - Oko Gold Mine FS Schedule		



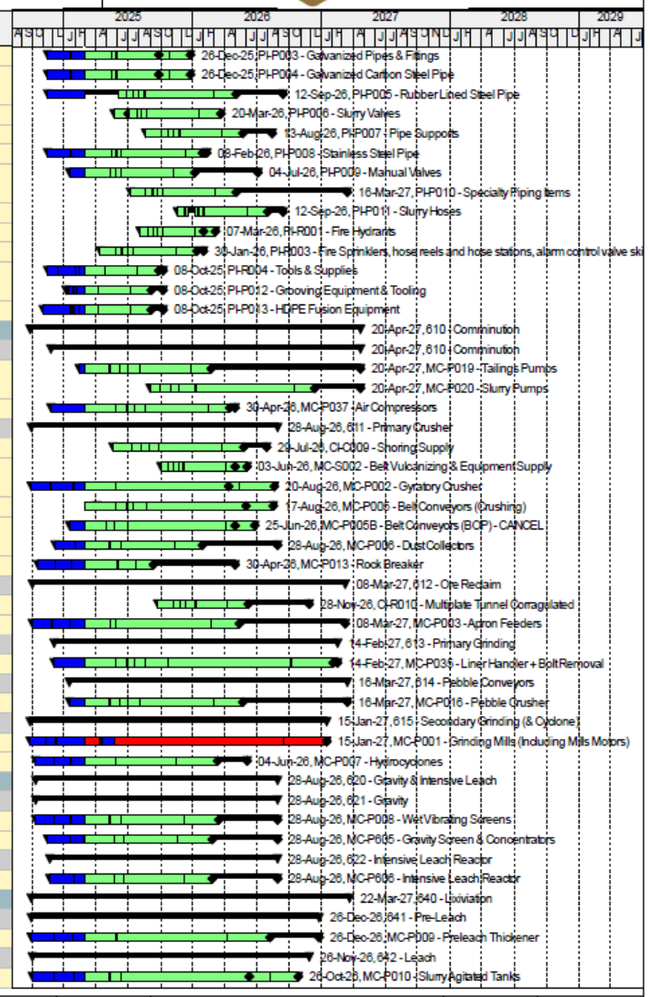
#	Activity ID	Activity Name	Original Duration	Start	Finish
288		UG-PX14 - Scissor Lift - Development	969	02-Dec-24 A	28-Jul-27
289		UG-PX15 - Scissor Lift - Construction	969	02-Dec-24 A	28-Jul-27
290		UG-PX16 - Attachment - Fan Handler	969	02-Dec-24 A	28-Jul-27
291		UG-PX17 - Attachment - Cable Spool	969	02-Dec-24 A	28-Jul-27
292		UG-PX18 - Boom Truck	969	02-Dec-24 A	28-Jul-27
293		UG-PX19 - Personnel Carrier	969	02-Dec-24 A	28-Jul-27
294		UG-PX20 - Deck Truck	969	02-Dec-24 A	28-Jul-27
295		UG-PX21 - Fuel & Lube Truck	969	02-Dec-24 A	28-Jul-27
296		UG-PX22 - Water Truck	969	02-Dec-24 A	28-Jul-27
297		UG-PX23 - Block Holer	969	02-Dec-24 A	28-Jul-27
298		UG-PX24 - Scaler	969	02-Dec-24 A	28-Jul-27
299		UG-PX25 - Cassette Truck	969	02-Dec-24 A	28-Jul-27
300		UG-PX26 - Cassette - Crane	969	02-Dec-24 A	28-Jul-27
301		UG-PX27 - Cassette - Emulsion Loading Deck	969	02-Dec-24 A	28-Jul-27
302		UG-PX28 - Cassette - Flat Deck	969	02-Dec-24 A	28-Jul-27
303		UG-PX29 - Cassette - Fuel Tanker	969	02-Dec-24 A	28-Jul-27
304		UG-PX30 - Cassette - Fuel & Lube	969	02-Dec-24 A	28-Jul-27
305		UG-PX31 - Cassette - Personnel Carrier	969	02-Dec-24 A	28-Jul-27
306		UG-PX32 - Cassette - Transmixer	969	02-Dec-24 A	28-Jul-27
307		UG-PX33 - Cassette - Water Sprayer	969	02-Dec-24 A	28-Jul-27
308		UG-PX34 - Shotcrete Sprayer	969	02-Dec-24 A	28-Jul-27
309		UG-PX35 - Shotcrete Mixer	969	02-Dec-24 A	28-Jul-27
310		UG-PX36 - Grader	969	02-Dec-24 A	28-Jul-27
311		UG-PX37 - Light Vehicle	969	02-Dec-24 A	28-Jul-27
312		UG-PX38 - Light Vehicle - Mine Rescue	969	02-Dec-24 A	28-Jul-27
313		UG-PX39 - Tractor - Mechanics	969	02-Dec-24 A	28-Jul-27
314		UG-PX40 - Tractor - Electricians	969	02-Dec-24 A	28-Jul-27
315		UG-PX41 - Tractor	969	02-Dec-24 A	28-Jul-27
316		UG-PX42 - Mobile Air Compressor	969	02-Dec-24 A	28-Jul-27
317		UG-PX43 - Backhoe Loader	969	02-Dec-24 A	28-Jul-27
318		UG-PX44 - Shotcrete and Concrete Transport	969	02-Dec-24 A	28-Jul-27
319		UG-PX45 - Raise Bore	969	02-Dec-24 A	28-Jul-27
320		UG-PX46 - V-30	969	02-Dec-24 A	28-Jul-27
321	600 - PROCESS PLANT		1095	26-Sep-24 A	25-Sep-27
322	600 - PROCESS PLANT		888	10-Oct-24 A	16-Mar-27
323	600 - PROCESS PLANT		888	10-Oct-24 A	16-Mar-27
324		MC-C003 - Equipment & Piping Insulation	663	16-Apr-25 A	26-Jan-27
325		MC-S003 - Pickling and Passivation	483	01-Oct-25	26-Jan-27
326		MC-P015 - Crusher Building Overhead Crane	577	14-Jan-25 A	13-Aug-26
327		MC-P022 - Sump Pumps	453	02-Sep-25	28-Nov-26
328		MC-P026 - Metallurgical and Online Samplers	788	09-Jan-25 A	07-Mar-27
329		MC-P027 - Reclaim Barge and Pumps	706	10-Oct-24 A	15-Sep-26
330		MC-P030 - Shop Fabricated Platework	469	17-Jan-25 A	30-Apr-26
331		MC-P036 - Cranes, Jibs and Hoists	575	02-Dec-24 A	29-Jun-26
332		MC-P610 - Cyanide analyzer	517	23-Jan-25 A	28-Sep-26
333		MC-P611 - Safety Showers	546	30-Apr-25	27-Oct-26
334		PI-P001 - Carbon Steel Pipe	440	15-Nov-24 A	28-Jan-26
335		PI-P002 - HDPE Pipe	436	15-Nov-24 A	25-Jan-26

Mining Services

GYOW - Oko Gold Mine - Feasibility Study

MINING VENTURES

#	Activity ID	Activity Name	Original Duration	Start	Finish
336	PI-P003	Galvanized Pipes & Fittings	406	15-Nov-24 A	26-Dec-25
337	PI-P004	Galvanized Carbon Steel Pipe	406	15-Nov-24 A	26-Dec-25
338	PI-P005	Rubber Lined Steel Pipe	667	15-Nov-24 A	12-Sep-26
339	PI-P006	Slurry Valves	303	21-May-25	20-Mar-26
340	PI-P007	Pipe Supports	359	20-Aug-25	13-Aug-26
341	PI-P008	Stainless Steel Pipe	450	15-Nov-24 A	08-Feb-26
342	PI-P009	Manual Valves	535	16-Jan-25 A	04-Jul-26
343	PI-P010	Specialty Piping Items	618	07-Jul-25	16-Mar-27
344	PI-P011	Slurry Hoses	300	17-Nov-25	12-Sep-26
345	PI-R001	Fire Hydrants	214	05-Aug-25	07-Mar-26
346	PI-R003	Fire Sprinklers, hose reels and hose stations, alarm control valve skid, post indicator valve, etc.	296	10-Apr-25	30-Jan-26
347	PI-R004	Tools & Supplies	328	15-Nov-24 A	08-Oct-25
348	PI-P012	Grooving Equipment & Tooling	254	06-Jan-25 A	08-Oct-25
349	PI-P013	HDPE Fusion Equipment	342	01-Nov-24 A	08-Oct-25
350	610	Comminution	937	26-Sep-24 A	20-Apr-27
351	610	Comminution	880	22-Nov-24 A	20-Apr-27
352	MC-P019	Tailings Pumps	797	13-Feb-25 A	20-Apr-27
353	MC-P020	Slurry Pumps	596	02-Sep-25	20-Apr-27
354	MC-P037	Air Compressors	525	22-Nov-24 A	30-Apr-26
355	611	Primary Crusher	698	30-Sep-24 A	28-Aug-26
356	CI-C009	Shoring Supply	436	20-May-25	29-Jul-26
357	MC-S002	Belt Vulcanizing & Equipment Supply	245	01-Oct-25	03-Jun-26
358	MC-P002	Gyratory Crusher	689	30-Sep-24 A	20-Aug-26
359	MC-P005	Belt Conveyors (Crushing)	640	03-Apr-25 A	17-Aug-26
360	MC-P005B	Belt Conveyors (BOP) - CANCEL	526	15-Jan-25 A	25-Jun-26
361	MC-P006	Dust Collectors	632	05-Dec-24 A	28-Aug-26
362	MC-P013	Rock Breaker	511	16-Oct-24 A	30-Apr-26
363	612	Ore Reclaim	889	01-Oct-24 A	08-Mar-27
364	CI-R010	Multiplate Tunnel Corragulated	432	23-Sep-25	28-Nov-26
365	MC-P003	Apron Feeders	889	01-Oct-24 A	08-Mar-27
366	613	Primary Grinding	805	02-Dec-24 A	14-Feb-27
367	MC-P035	Liner Handler + Bolt Removal	805	02-Dec-24 A	14-Feb-27
368	614	Pebble Conveyors	791	15-Jan-25 A	16-Mar-27
369	MC-P016	Pebble Crusher	791	15-Jan-25 A	16-Mar-27
370	615	Secondary Grinding (& Cyclone)	842	26-Sep-24 A	15-Jan-27
371	MC-P001	Grinding Mills (Including Mills Motors)	842	26-Sep-24 A	15-Jan-27
372	MC-P007	Hydrocyclones	603	10-Oct-24 A	04-Jun-26
373	620	Gravity & Intensive Leach	688	10-Oct-24 A	28-Aug-26
374	621	Gravity	688	10-Oct-24 A	28-Aug-26
375	MC-P008	Wet Vibrating Screens	688	10-Oct-24 A	28-Aug-26
376	MC-P605	Gravity Screen & Concentrators	652	15-Nov-24 A	28-Aug-26
377	622	Intensive Leach Reactor	647	20-Nov-24 A	28-Aug-26
378	MC-P606	Intensive Leach Reactor	647	20-Nov-24 A	28-Aug-26
379	640	Lixiviation	904	30-Sep-24 A	22-Mar-27
380	641	Pre-Leach	818	30-Sep-24 A	26-Dec-26
381	MC-P009	Preleach Thickener	818	30-Sep-24 A	26-Dec-26
382	642	Leach	787	01-Oct-24 A	26-Nov-26
383	MC-P010	Slurry Agitated Tanks	755	01-Oct-24 A	26-Oct-26



Remaining Level of Effort | Actual Work | Critical Remaining Work
Actual Level of Effort | Remaining Work | Milestone

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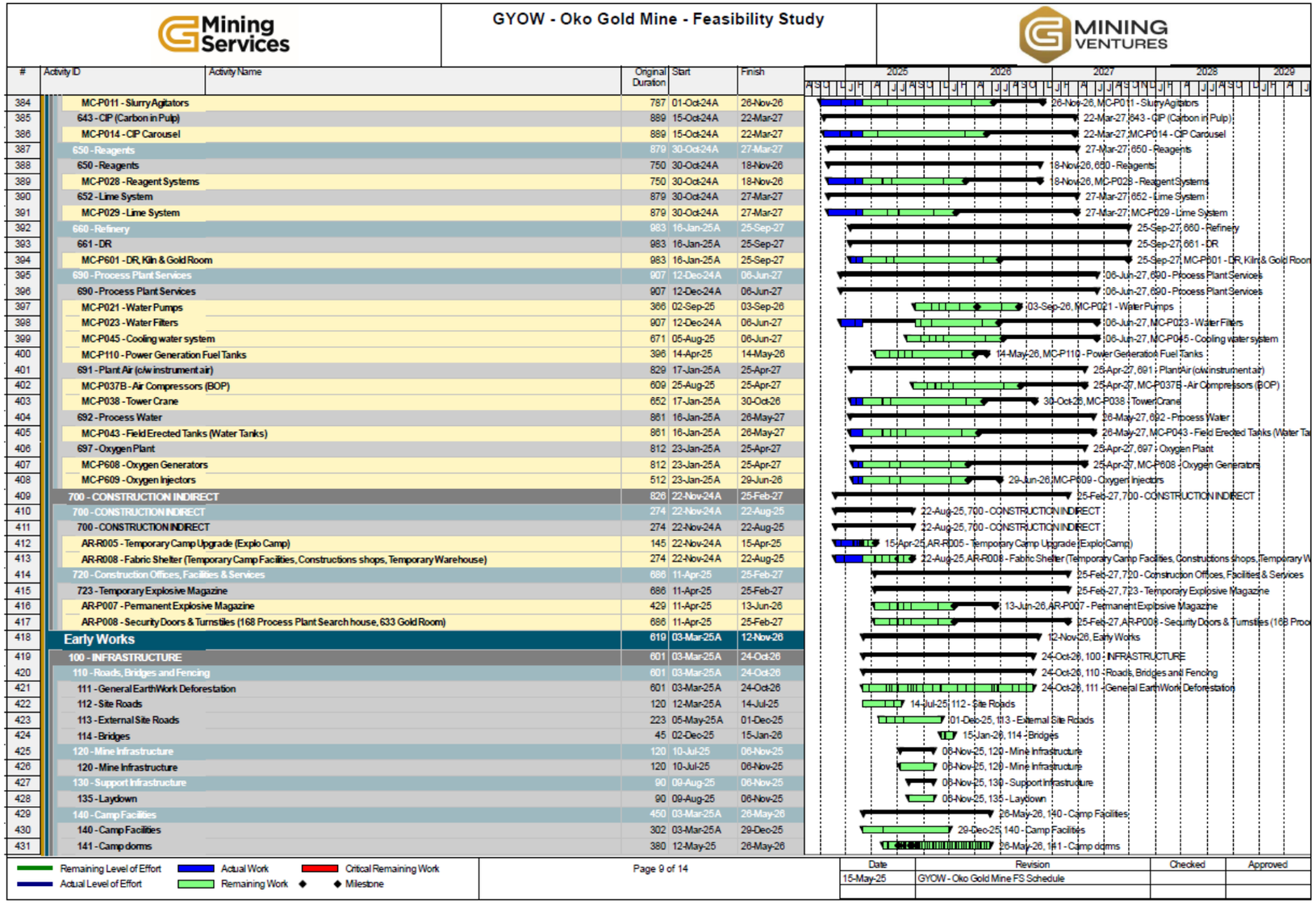
Date	Revision	Checked	Approved
15-May-25	GYOW - Oko Gold Mine FS Schedule		



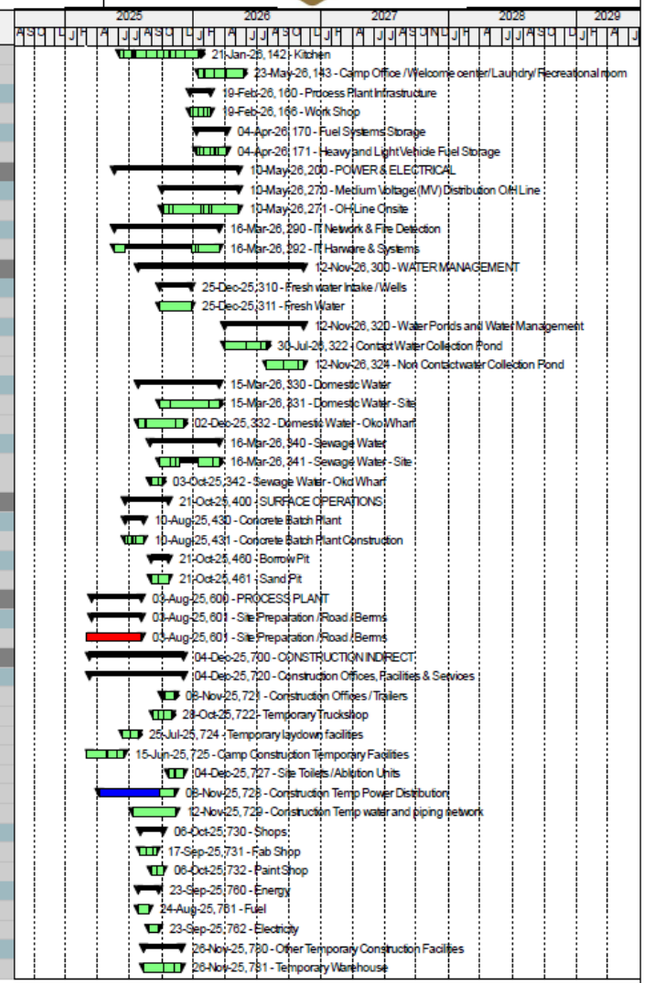
#	Activity ID	Activity Name	Original Duration	Start	Finish
384	MC-P011	Slurry Agitators	787	01-Oct-24 A	26-Nov-26
385	643	CIP (Carbon in Pulp)	889	15-Oct-24 A	22-Mar-27
386	MC-P014	CIP Carousel	889	15-Oct-24 A	22-Mar-27
387	650	Reagents	879	30-Oct-24 A	27-Mar-27
388	650	Reagents	750	30-Oct-24 A	18-Nov-26
389	MC-P028	Reagent Systems	750	30-Oct-24 A	18-Nov-26
390	652	Lime System	879	30-Oct-24 A	27-Mar-27
391	MC-P029	Lime System	879	30-Oct-24 A	27-Mar-27
392	660	Refinery	983	16-Jan-25 A	25-Sep-27
393	661	DR	983	16-Jan-25 A	25-Sep-27
394	MC-P601	DR, Kiln & Gold Room	983	16-Jan-25 A	25-Sep-27
395	690	Process Plant Services	907	12-Dec-24 A	06-Jun-27
396	690	Process Plant Services	907	12-Dec-24 A	06-Jun-27
397	MC-P021	Water Pumps	366	02-Sep-25	03-Sep-26
398	MC-P023	Water Filters	907	12-Dec-24 A	06-Jun-27
399	MC-P045	Cooling water system	671	05-Aug-25	06-Jun-27
400	MC-P110	Power Generation Fuel Tanks	396	14-Apr-25	14-May-26
401	691	Plant Air (c/w instrument air)	829	17-Jan-25 A	25-Apr-27
402	MC-P037B	Air Compressors (BOP)	609	25-Aug-25	25-Apr-27
403	MC-P038	Tower Crane	652	17-Jan-25 A	30-Oct-26
404	692	Process Water	861	16-Jan-25 A	26-May-27
405	MC-P043	Field Erected Tanks (Water Tanks)	861	16-Jan-25 A	26-May-27
406	697	Oxygen Plant	812	23-Jan-25 A	25-Apr-27
407	MC-P608	Oxygen Generators	812	23-Jan-25 A	25-Apr-27
408	MC-P609	Oxygen Injectors	512	23-Jan-25 A	29-Jun-26
409	700	CONSTRUCTION INDIRECT	826	22-Nov-24 A	25-Feb-27
410	700	CONSTRUCTION INDIRECT	274	22-Nov-24 A	22-Aug-25
411	700	CONSTRUCTION INDIRECT	274	22-Nov-24 A	22-Aug-25
412	AR-R005	Temporary Camp Upgrade (Explo Camp)	145	22-Nov-24 A	15-Apr-25
413	AR-R008	Fabric Shelter (Temporary Camp Facilities, Constructions shops, Temporary Warehouse)	274	22-Nov-24 A	22-Aug-25
414	720	Construction Offices, Facilities & Services	686	11-Apr-25	25-Feb-27
415	723	Temporary Explosive Magazine	686	11-Apr-25	25-Feb-27
416	AR-P007	Permanent Explosive Magazine	429	11-Apr-25	13-Jun-26
417	AR-P008	Security Doors & Turnstiles (168 Process Plant Search house, 633 Gold Room)	686	11-Apr-25	25-Feb-27
418	Early Works		619	03-Mar-25 A	12-Nov-26
419	100	INFRASTRUCTURE	601	03-Mar-25 A	24-Oct-26
420	110	Roads, Bridges and Fencing	601	03-Mar-25 A	24-Oct-26
421	111	General EarthWork Deforestation	601	03-Mar-25 A	24-Oct-26
422	112	Site Roads	120	12-Mar-25 A	14-Jul-25
423	113	External Site Roads	223	05-May-25 A	01-Dec-25
424	114	Bridges	45	02-Dec-25	15-Jan-26
425	120	Mine Infrastructure	120	10-Jul-25	06-Nov-25
426	120	Mine Infrastructure	120	10-Jul-25	06-Nov-25
427	130	Support Infrastructure	90	09-Aug-25	06-Nov-25
428	135	Laydown	90	09-Aug-25	06-Nov-25
429	140	Camp Facilities	450	03-Mar-25 A	26-May-26
430	140	Camp Facilities	302	03-Mar-25 A	29-Dec-25
431	141	Camp dorms	380	12-May-25	26-May-26

Mining Services

GYOW - Oko Gold Mine - Feasibility Study

MINING VENTURES

#	Activity ID	Activity Name	Original Duration	Start	Finish
432	142	Kitchen	234	02-Jun-25	21-Jan-26
433	143	Camp Office /Welcome center/ Laundry/ Recreational room	127	16-Jan-26	23-May-26
434	160	Process Plant Infrastructure	60	22-Dec-25	19-Feb-26
435	166	Work Shop	60	22-Dec-25	19-Feb-26
436	170	Fuel Systems Storage	86	09-Jan-26	04-Apr-26
437	171	Heavy and Light Vehicle Fuel Storage	86	09-Jan-26	04-Apr-26
438	200	POWER & ELECTRICAL	356	19-May-25	10-May-26
439	270	Medium Voltage (MV) Distribution O/H Line	220	02-Oct-25	10-May-26
440	271	O/H Line Onsite	220	02-Oct-25	10-May-26
441	290	IT Network & Fire Detection	301	19-May-25	16-Mar-26
442	292	IT Harware & Systems	301	19-May-25	16-Mar-26
443	300	WATER MANAGEMENT	475	25-Jul-25	12-Nov-26
444	310	Fresh water Intake / Wells	90	27-Sep-25	25-Dec-25
445	311	Fresh Water	90	27-Sep-25	25-Dec-25
446	320	Water Ponds and Water Management	225	01-Apr-26	12-Nov-26
447	322	Contact Water Collection Pond	120	01-Apr-26	30-Jul-26
448	324	Non Contact water Collection Pond	105	30-Jul-26	12-Nov-26
449	330	Domestic Water	234	25-Jul-25	15-Mar-26
450	331	Domestic Water - Site	170	27-Sep-25	15-Mar-26
451	332	Domestic Water - Oko Wharf	131	25-Jul-25	02-Dec-25
452	340	Sewage Water	198	30-Aug-25	16-Mar-26
453	341	Sewage Water - Site	170	27-Sep-25	16-Mar-26
454	342	Sewage Water - Oko Wharf	35	30-Aug-25	03-Oct-25
455	400	SURFACE OPERATIONS	125	19-Jun-25	21-Oct-25
456	430	Concrete Batch Plant	53	19-Jun-25	10-Aug-25
457	431	Concrete Batch Plant Construction	53	19-Jun-25	10-Aug-25
458	460	Borrow Pit	51	01-Sep-25	21-Oct-25
459	461	Sand Pit	51	01-Sep-25	21-Oct-25
460	600	PROCESS PLANT	110	16-Mar-25A	03-Aug-25
461	601	Site Preparation / Road / Berms	110	16-Mar-25A	03-Aug-25
462	601	Site Preparation / Road / Berms	110	16-Mar-25A	03-Aug-25
463	700	CONSTRUCTION INDIRECT	272	08-Mar-25A	04-Dec-25
464	720	Construction Offices, Facilities & Services	272	08-Mar-25A	04-Dec-25
465	721	Construction Offices / Trailers	37	03-Oct-25	08-Nov-25
466	722	Temporary Truckshop	51	08-Sep-25	28-Oct-25
467	724	Temporary laydown facilities	45	11-Jun-25	25-Jul-25
468	725	Camp Construction Temporary Facilities	82	08-Mar-25A	15-Jun-25
469	727	Site Toilets / Ablution Units	46	20-Oct-25	04-Dec-25
470	728	Construction Temp Power Distribution	45	07-Apr-25A	08-Nov-25
471	729	Construction Temp water and piping network	125	11-Jul-25	12-Nov-25
472	730	Shops	66	01-Aug-25	06-Oct-25
473	731	Fab Shop	47	01-Aug-25	17-Sep-25
474	732	Paint Shop	35	01-Sep-25	06-Oct-25
475	760	Energy	60	26-Jul-25	23-Sep-25
476	761	Fuel	30	26-Jul-25	24-Aug-25
477	762	Electricity	30	25-Aug-25	23-Sep-25
478	780	Other Temporary Construction Facilities	110	09-Aug-25	26-Nov-25
479	781	Temporary Warehouse	110	09-Aug-25	26-Nov-25



Remaining Level of Effort | Actual Work | Critical Remaining Work
Actual Level of Effort | Remaining Work | Milestone

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Date	Revision	Checked	Approved
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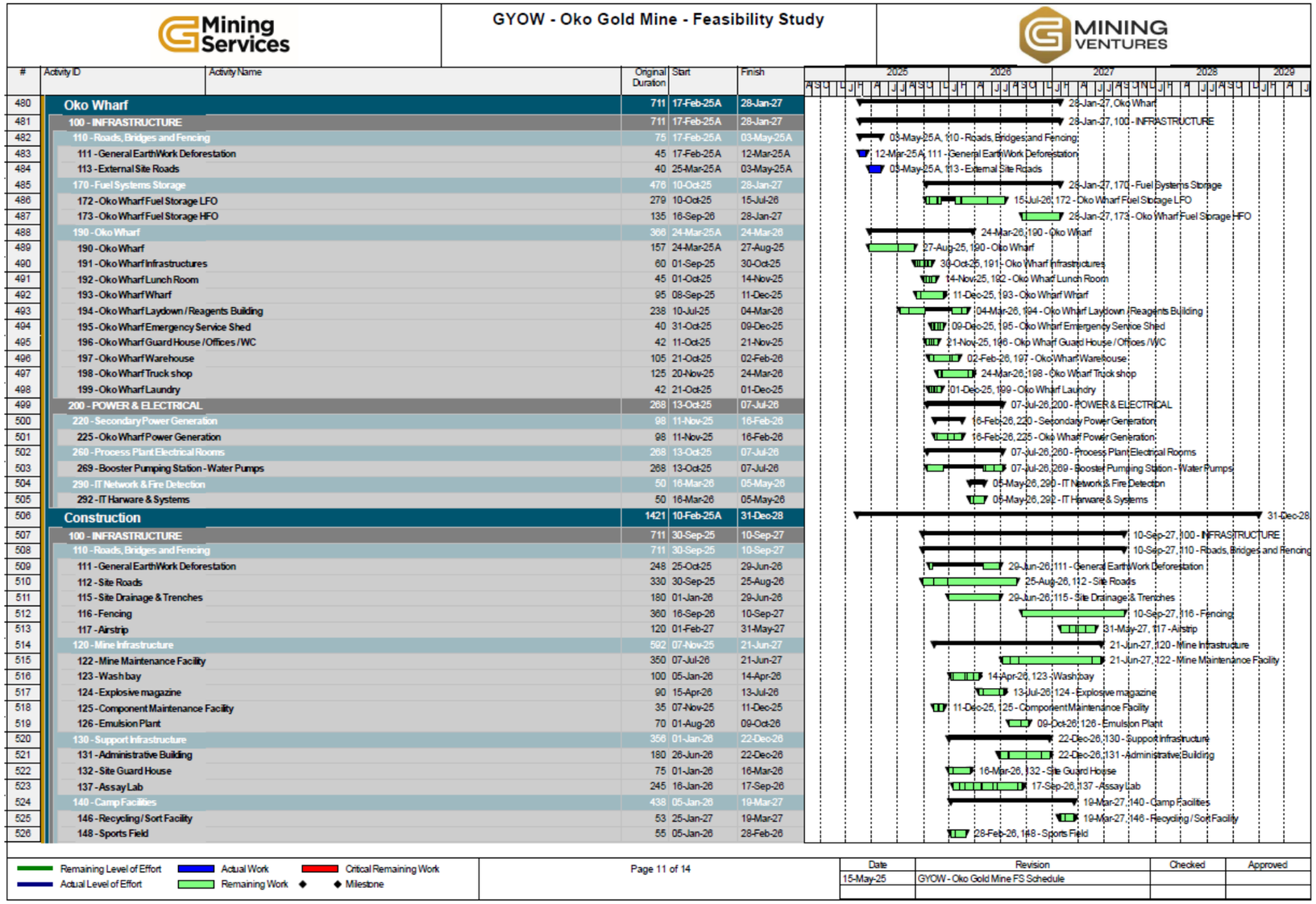


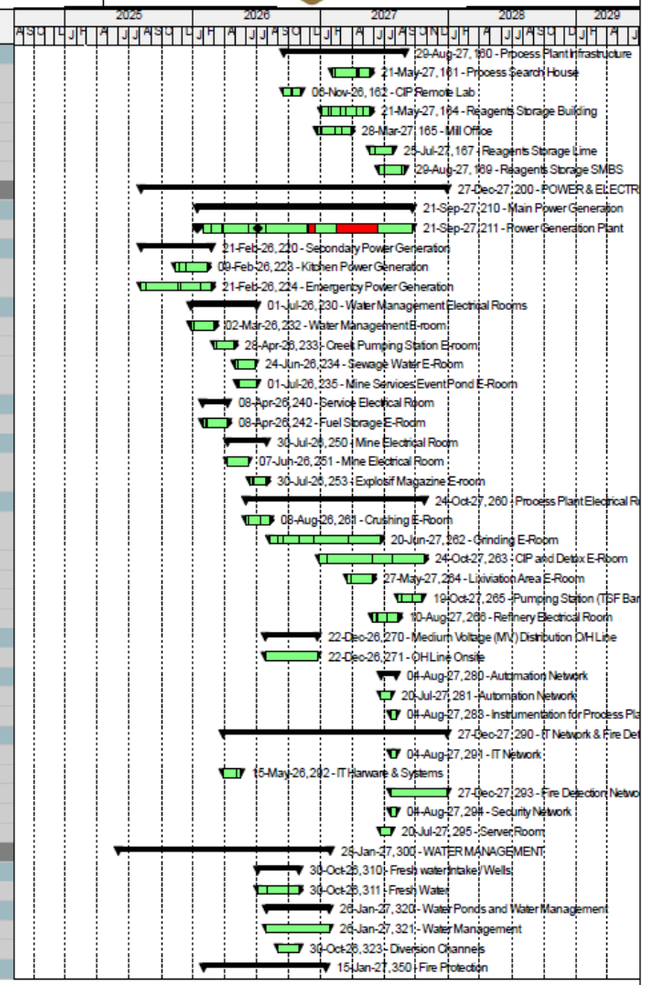
#	Activity ID	Activity Name	Original Duration	Start	Finish
480		Oko Wharf	711	17-Feb-25A	28-Jan-27
481		100 - INFRASTRUCTURE	711	17-Feb-25A	28-Jan-27
482		110 - Roads, Bridges and Fencing	75	17-Feb-25A	03-May-25A
483		111 - General EarthWork Deforestation	45	17-Feb-25A	12-Mar-25A
484		113 - External Site Roads	40	25-Mar-25A	03-May-25A
485		170 - Fuel Systems Storage	476	10-Oct-25	28-Jan-27
486		172 - Oko Wharf Fuel Storage LFO	279	10-Oct-25	15-Jul-26
487		173 - Oko Wharf Fuel Storage HFO	135	16-Sep-26	28-Jan-27
488		190 - Oko Wharf	366	24-Mar-25A	24-Mar-26
489		190 - Oko Wharf	157	24-Mar-25A	27-Aug-25
490		191 - Oko Wharf Infrastructures	60	01-Sep-25	30-Oct-25
491		192 - Oko Wharf Lunch Room	45	01-Oct-25	14-Nov-25
492		193 - Oko Wharf Wharf	95	08-Sep-25	11-Dec-25
493		194 - Oko Wharf Laydown / Reagents Building	238	10-Jul-25	04-Mar-26
494		195 - Oko Wharf Emergency Service Shed	40	31-Oct-25	09-Dec-25
495		196 - Oko Wharf Guard House / Offices / WC	42	11-Oct-25	21-Nov-25
496		197 - Oko Wharf Warehouse	105	21-Oct-25	02-Feb-26
497		198 - Oko Wharf Truck shop	125	20-Nov-25	24-Mar-26
498		199 - Oko Wharf Laundry	42	21-Oct-25	01-Dec-25
499		200 - POWER & ELECTRICAL	268	13-Oct-25	07-Jul-26
500		220 - Secondary Power Generation	98	11-Nov-25	16-Feb-26
501		225 - Oko Wharf Power Generation	98	11-Nov-25	16-Feb-26
502		260 - Process Plant Electrical Rooms	268	13-Oct-25	07-Jul-26
503		269 - Booster Pumping Station - Water Pumps	268	13-Oct-25	07-Jul-26
504		290 - IT Network & Fire Detection	50	16-Mar-26	05-May-26
505		292 - IT Harware & Systems	50	16-Mar-26	05-May-26
506		Construction	1421	10-Feb-25A	31-Dec-28
507		100 - INFRASTRUCTURE	711	30-Sep-25	10-Sep-27
508		110 - Roads, Bridges and Fencing	711	30-Sep-25	10-Sep-27
509		111 - General EarthWork Deforestation	248	25-Oct-25	29-Jun-26
510		112 - Site Roads	330	30-Sep-25	25-Aug-26
511		115 - Site Drainage & Trenches	180	01-Jan-26	29-Jun-26
512		116 - Fencing	360	16-Sep-26	10-Sep-27
513		117 - Airstrip	120	01-Feb-27	31-May-27
514		120 - Mine Infrastructure	592	07-Nov-25	21-Jun-27
515		122 - Mine Maintenance Facility	350	07-Jul-26	21-Jun-27
516		123 - Wash bay	100	05-Jan-26	14-Apr-26
517		124 - Explosive magazine	90	15-Apr-26	13-Jul-26
518		125 - Component Maintenance Facility	35	07-Nov-25	11-Dec-25
519		126 - Emulsion Plant	70	01-Aug-26	09-Oct-26
520		130 - Support Infrastructure	356	01-Jan-26	22-Dec-26
521		131 - Administrative Building	180	26-Jun-26	22-Dec-26
522		132 - Site Guard House	75	01-Jan-26	16-Mar-26
523		137 - Assay Lab	245	16-Jan-26	17-Sep-26
524		140 - Camp Facilities	438	05-Jan-26	19-Mar-27
525		146 - Recycling / Sort Facility	53	25-Jan-27	19-Mar-27
526		148 - Sports Field	55	05-Jan-26	28-Feb-26

Mining Services

GYOW - Oko Gold Mine - Feasibility Study

MINING VENTURES

#	Activity ID	Activity Name	Original Duration	Start	Finish
527	160	Process Plant Infrastructure	346	18-Sep-26	29-Aug-27
528	161	Process Search House	109	02-Feb-27	21-May-27
529	162	CIP Remote Lab	50	18-Sep-26	06-Nov-26
530	164	Reagents Storage Building	140	02-Jan-27	21-May-27
531	165	Mill Office	96	23-Dec-26	28-Mar-27
532	167	Reagents Storage Lime	65	22-May-27	25-Jul-27
533	169	Reagents Storage SMBS	75	16-Jun-27	29-Aug-27
534	200	POWER & ELECTRICAL	876	04-Aug-25	27-Dec-27
535	210	Main Power Generation	619	11-Jan-26	21-Sep-27
536	211	Power Generation Plant	619	11-Jan-26	21-Sep-27
537	220	Secondary Power Generation	201	04-Aug-25	21-Feb-26
538	223	Kitchen Power Generation	90	12-Nov-25	09-Feb-26
539	224	Emergency Power Generation	201	04-Aug-25	21-Feb-26
540	230	Water Management Electrical Rooms	188	26-Dec-25	01-Jul-26
541	232	Water Management E-room	67	26-Dec-25	02-Mar-26
542	233	Creek Pumping Station E-room	57	03-Mar-26	28-Apr-26
543	234	Sewage Water E-Room	57	29-Apr-26	24-Jun-26
544	235	Mine Services Event Pond E-Room	57	06-May-26	01-Jul-26
545	240	Service Electrical Room	70	29-Jan-26	08-Apr-26
546	242	Fuel Storage E-Room	70	29-Jan-26	08-Apr-26
547	250	Mine Electrical Room	113	09-Apr-26	30-Jul-26
548	251	Mine Electrical Room	60	09-Apr-26	07-Jun-26
549	253	Explosif Magazine E-room	47	14-Jun-26	30-Jul-26
550	260	Process Plant Electrical Rooms	512	31-May-26	24-Oct-27
551	261	Crushing E-Room	70	31-May-26	08-Aug-26
552	262	Grinding E-Room	316	09-Aug-26	20-Jun-27
553	263	CIP and Detox E-Room	301	28-Dec-26	24-Oct-27
554	264	Lixiviation Area E-Room	70	19-Mar-27	27-May-27
555	265	Pumping Station (TSF Barge) E-room	70	11-Aug-27	19-Oct-27
556	266	Refinery Electrical Room	75	28-May-27	10-Aug-27
557	270	Medium Voltage (MV) Distribution OH Line	150	26-Jul-26	22-Dec-26
558	271	OH Line Onsite	150	26-Jul-26	22-Dec-26
559	280	Automation Network	45	21-Jun-27	04-Aug-27
560	281	Automation Network	30	21-Jun-27	20-Jul-27
561	283	Instrumentation for Process Plant	15	21-Jul-27	04-Aug-27
562	290	IT Network & Fire Detection	642	26-Mar-26	27-Dec-27
563	291	IT Network	15	21-Jul-27	04-Aug-27
564	292	IT Harware & Systems	50	26-Mar-26	15-May-26
565	293	Fire Detection Network	160	21-Jul-27	27-Dec-27
566	294	Security Network	15	21-Jul-27	04-Aug-27
567	295	Server Room	30	21-Jun-27	20-Jul-27
568	300	WATER MANAGEMENT	607	01-Jun-25	28-Jan-27
569	310	Fresh water Intake / Wells	120	03-Jul-26	30-Oct-26
570	311	Fresh Water	120	03-Jul-26	30-Oct-26
571	320	Water Ponds and Water Management	180	30-Jul-26	26-Jan-27
572	321	Water Management	180	30-Jul-26	26-Jan-27
573	323	Diversion Channels	60	01-Sep-26	30-Oct-26
574	350	Fire Protection	350	31-Jan-26	15-Jan-27



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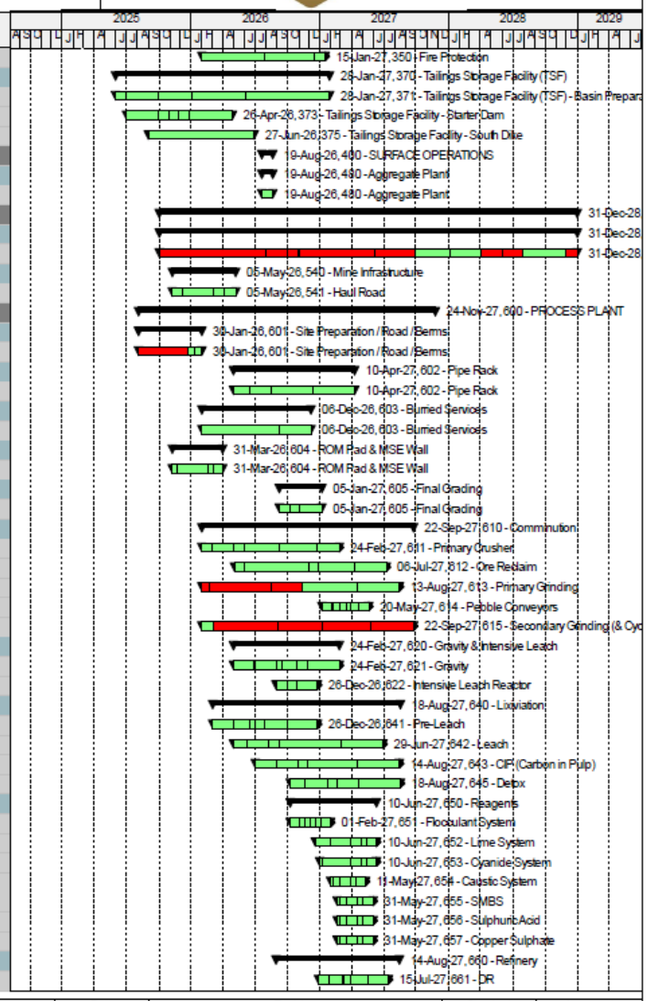
Mining Services

GYOW - Oko Gold Mine - Feasibility Study

MINING VENTURES

#	Activity ID	Activity Name	Original Duration	Start	Finish
575	350	Fire Protection	350	31-Jan-26	15-Jan-27
576	370	Tailings Storage Facility (TSF)	607	01-Jun-25	28-Jan-27
577	371	Tailings Storage Facility (TSF) - Basin Preparation	607	01-Jun-25	28-Jan-27
578	373	Tailings Storage Facility - Starter Dam	300	01-Jul-25	26-Apr-26
579	375	Tailings Storage Facility - South Dike	300	01-Sep-25	27-Jun-26
580	400	SURFACE OPERATIONS	30	20-Jul-26	19-Aug-26
581	480	Aggregate Plant	30	20-Jul-26	19-Aug-26
582	480	Aggregate Plant	30	20-Jul-26	19-Aug-26
583	500	MINING	1188	01-Oct-25	31-Dec-28
584	500	MINING	1188	01-Oct-25	31-Dec-28
585	500	MINING	1188	01-Oct-25	31-Dec-28
586	540	Mine Infrastructure	180	07-Nov-25	05-May-26
587	541	Haul Road	180	07-Nov-25	05-May-26
588	600	PROCESS PLANT	843	04-Aug-25	24-Nov-27
589	601	Site Preparation / Road / Berms	180	04-Aug-25	30-Jan-26
590	601	Site Preparation / Road / Berms	180	04-Aug-25	30-Jan-26
591	602	Pipe Rack	345	01-May-26	10-Apr-27
592	602	Pipe Rack	345	01-May-26	10-Apr-27
593	603	Burried Services	310	31-Jan-26	06-Dec-26
594	603	Burried Services	310	31-Jan-26	06-Dec-26
595	604	ROM Pad & MSE Wall	145	07-Nov-25	31-Mar-26
596	604	ROM Pad & MSE Wall	145	07-Nov-25	31-Mar-26
597	605	Final Grading	120	08-Sep-26	05-Jan-27
598	605	Final Grading	120	08-Sep-26	05-Jan-27
599	610	Comminution	600	31-Jan-26	22-Sep-27
600	611	Primary Crusher	390	31-Jan-26	24-Feb-27
601	612	Ore Reclaim	430	03-May-26	06-Jul-27
602	613	Primary Grinding	560	31-Jan-26	13-Aug-27
603	614	Pebble Conveyors	135	06-Jan-27	20-May-27
604	615	Secondary Grinding (& Cyclone)	600	31-Jan-26	22-Sep-27
605	620	Gravity & Intensive Leach	300	01-May-26	24-Feb-27
606	621	Gravity	300	01-May-26	24-Feb-27
607	622	Intensive Leach Reactor	120	29-Aug-26	26-Dec-26
608	640	Lixiviation	535	02-Mar-26	18-Aug-27
609	641	Pre-Leach	300	02-Mar-26	26-Dec-26
610	642	Leach	425	01-May-26	29-Jun-27
611	643	CIP (Carbon in Pulp)	410	01-Jul-26	14-Aug-27
612	645	Detox	315	08-Oct-26	18-Aug-27
613	650	Reagents	246	08-Oct-26	10-Jun-27
614	651	Flocculant System	117	08-Oct-26	01-Feb-27
615	652	Lime System	175	18-Dec-26	10-Jun-27
616	653	Cyanide System	162	31-Dec-26	10-Jun-27
617	654	Caustic System	102	30-Jan-27	11-May-27
618	655	SMBS	102	19-Feb-27	31-May-27
619	656	Sulphuric Acid	102	19-Feb-27	31-May-27
620	657	Copper Sulphate	102	19-Feb-27	31-May-27
621	660	Refinery	350	30-Aug-26	14-Aug-27
622	661	DR	200	28-Dec-26	15-Jul-27



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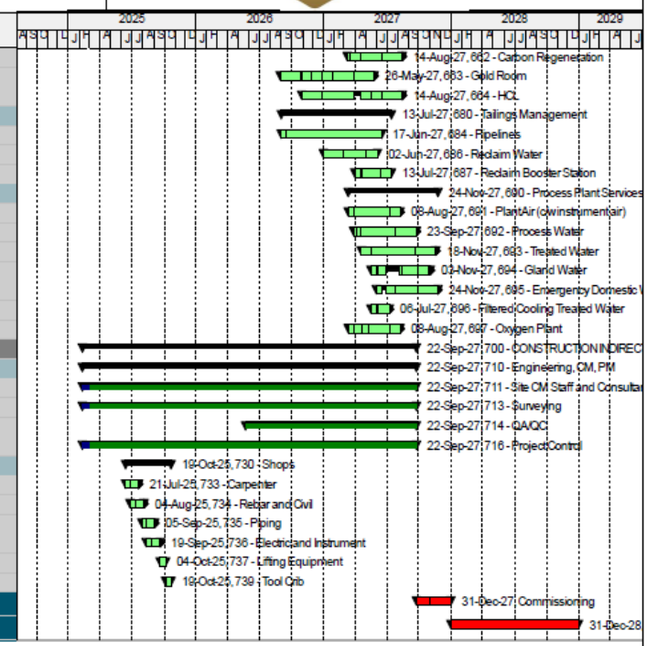
Mining Services

GYOW - Oko Gold Mine - Feasibility Study

MINING VENTURES

#	Activity ID	Activity Name	Original Duration	Start	Finish
623	662	Carbon Regeneration	160	08-Mar-27	14-Aug-27
624	663	Gold Room	270	30-Aug-26	26-May-27
625	664	HCL	290	29-Oct-26	14-Aug-27
626	680	Tailings Management	316	01-Sep-26	13-Jul-27
627	684	Pipelines	290	01-Sep-26	17-Jun-27
628	686	Reclaim Water	155	30-Dec-26	02-Jun-27
629	687	Reclaim Booster Station	106	30-Mar-27	13-Jul-27
630	690	Process Plant Services	258	12-Mar-27	24-Nov-27
631	691	Plant Air (c/w instrument air)	150	12-Mar-27	08-Aug-27
632	692	Process Water	181	27-Mar-27	23-Sep-27
633	693	Treated Water	216	17-Apr-27	18-Nov-27
634	694	Gland Water	171	17-May-27	03-Nov-27
635	695	Emergency Domestic Water	177	01-Jun-27	24-Nov-27
636	696	Filtered Cooling Treated Water	51	17-May-27	06-Jul-27
637	697	Oxygen Plant	150	12-Mar-27	08-Aug-27
638	700	CONSTRUCTION INDIRECT	956	10-Feb-25 A	22-Sep-27
639	710	Engineering, CM, PM	956	10-Feb-25 A	22-Sep-27
640	711	Site CM Staff and Consultants	956	10-Feb-25 A	22-Sep-27
641	713	Surveying	956	10-Feb-25 A	22-Sep-27
642	714	QA/QC	488	23-May-26	22-Sep-27
643	716	Project Control	956	10-Feb-25 A	22-Sep-27
644	730	Shops	129	12-Jun-25	19-Oct-25
645	733	Carpenter	39	12-Jun-25	21-Jul-25
646	734	Rebar and Civil	39	26-Jun-25	04-Aug-25
647	735	Piping	39	28-Jul-25	05-Sep-25
648	736	Electric and Instrument	39	11-Aug-25	19-Sep-25
649	737	Lifting Equipment	15	19-Sep-25	04-Oct-25
650	739	Tool Crib	15	04-Oct-25	19-Oct-25
651	Commissioning		100	23-Sep-27	31-Dec-27
652	Ramp-up		366	01-Jan-28	31-Dec-28



Remaining Level of Effort
Actual Level of Effort
Actual Work
Remaining Work
Critical Remaining Work
Milestone

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Date	Revision	Checked	Approved
15-May-25	GYOW - Oko Gold Mine FS Schedule		

25. INTERPRETATION AND CONCLUSIONS

25.1 Summary

This Feasibility Study (FS) Report is prepared in accordance with the guidelines of the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101") and Form 43-101F1. The objective of this FS Report is the evaluation of the technical feasibility and economic viability of the Project, including the development of an open pit and underground mine with processing facilities and required infrastructures.

The conclusion of the qualified persons ("QPs") is that the Oko West Project, as summarized in this NI 43-101 Feasibility Study (FS) report, contains a sufficient level of detail and information to support the positive economic results presented, and confirms the technical feasibility and economic viability based on a combined open pit and underground mining operation. Table 25.1 shows the Feasibility Study highlights. The Project has an average gold production of 350 koz per year over a 12.3-year life of mine (LOM). It is recommended to advance the Project to the Detail Engineering and Construction and Development phase.

Table 25.1: Oko West Feasibility Highlights

Technical Report Feasibility Study Update Life-of-Mine Results		
Gold Price – Base Case	$/oz	2,500
Mine Life (operation years)	Mt	12.3
OP Mill Feed Tonnage	Mt	62.37
UG Mill Feed Tonnage	Mt	14.18
Total Ore Mined	***Mt***	***76.56***
Total Waste Mined (OP and UG)	**Mt**	***429.35***
Total Tonnage Mined (OP and UG)	**Mt**	***505.91***
Open Pit Strip Ratio	***waste: ore***	***6.83***
Average Milling Throughput	**Mtpa**	**6.17**
Average Milling Throughput	**tpd**	***16,911***
Gold Head Grade	g/t	1.89
OP Head Grade	g/t	1.57
UG Head Grade	g/t	3.26
Contained Gold	koz	4,642
Average Gold Recovery (%)	%	93.5%
Total Gold Production	koz	4,340

Technical Report Feasibility Study Update Life-of-Mine Results		
Average Annual Gold Production	koz	350
Operating Costs (LOM Average)		
Open Pit Mining Cost	$/t milled	13.81
Underground Mining Cost	$/t milled	10.34
Processing Cost	$/t milled	7.24
Power Cost	$/t milled	7.95
General & Administration Cost	$/t milled	5.49
Refining Cost	$/t milled	0.45
Total Site Costs	**$/t milled**	**45.29**
Total Site Costs	$/oz	798
Government Royalties	$/oz	160
Total Operating Cost	$/oz	958
AISC	**$/oz**	**1,123**
Capital Costs		
Initial Capital Costs	*$ MM*	956
Contingency	*$ MM*	85
Total Initial Capital Cost	***$ MM***	**1,041**
UG Capital Costs (Sustaining Capital)	*$ MM*	68
OP and UG Sustaining Capital	$ MM	522
Life of Mine Sustaining Capital	**$ MM**	**650**
Closure Costs	$ MM	39
Total Capital and Closure Costs	**$ MM**	**1,730**
Financial Evaluation		
After-Tax NPV 5%	$ MM	2,163
After-Tax IRR	%	27.1%
Payback	years	2.9

25.2 Geology and Mineral Resources

As of September 15, 2024, sufficient gold mineralization to define a Mineral Resource Estimate (MRE) has been intercepted as part of drilling at the Oko West Project over a strike length of over 3.0 km with a down-dip extent greater than 1,000 m. Additional gold mineralization has been defined in other areas within

the Project extents that warrant further exploration follow up, both along strike and up to 4 km south of the MRE, and in the western portions of the Prospecting Licences (PL). The resource conversion of the Oko West Project has been excellent, with 93% of the Inferred resource converted to the Indicated category so far (GMIN press release, dated April 28, 2025).

Gold mineralization is hosted within a volcano-sedimentary sequence and visually associated with carbonatization-albitization, silicification and sericitic alteration. Elevated gold assays are often visually associated with strong alteration, brittle deformation and shearing and sulfidation of the host rock. The litho structural setting of gold, as well as controls on mineralization have been extensively studied and are generally well understood.

Drilling methods employed at the Oko West Project are typical and follow industry standards used to delineate gold mineralization. Diamond drilling is the principal method, and core recovery is considered excellent. Reverse circulation (RC) drilling is used primarily for scout and reconnaissance drilling. Sampling methods and QA/QC practices are in accordance with industry standards and sufficient controls are in place to ensure a robust drilling database. The drilling database, verified and validated by the QP of this MRE, consists of approximately 106,677 m of assayed diamond drilling (including wedged holes), 24,719 m of assayed RC drilling and 8,080 m of assayed trenches.

Independent sampling reproduced original assay values present in the database within acceptable limits. Subsequent database checks have demonstrated that the drilling database is robust and error-free.

The total pit constrained Indicated Mineral Resource is reported at 73,035 kt grading 2.00 g/t Au, for a total of 4,689 koz of gold. The total pit constrained Inferred Mineral Resource is reported at 1,542 kt grading 1.06 g/t Au, for a total of 52 koz of gold. The underground resources are estimated from zones outside the constrained resources of the open pit. The total constrained underground Indicated Mineral Resource is reported at 7,223 kt grading 3.09 g/t Au, for a total of 718 koz of gold. The total constrained underground Inferred Mineral Resource is reported at 3,585 kt grading 2.93 g/t Au, for a total of 337 koz of gold. Mineral resources are not mineral reserves and have not demonstrated economic viability. Mineral resources are inclusive of mineral reserves. The QP has determined that there are no known factors or issues that could significantly impact the Mineral Resource Estimate (MRE), other than the typical risks associated with mining projects, such as environmental, permitting, taxation, socio-economic, marketing, and political factors, as well as additional risk factors related to indicated and inferred mineral resources.

25.3 Mineral Reserve Estimates

The Mineral Reserves presented in this report are based on Measured and Indicated Mineral Resources and do not include any Inferred Mineral Resources. The Inferred Mineral Resources contained within the mine design are treated as waste at a grade of 0 g/t Au.

The Proven and Probable Reserve for the Oko West project is estimated at 76.6 Mt at an average grade of 1.89 g/t Au for 4.64 million ounces of gold.

Factors that may affect the Mineral Reserve estimates include: geological complexity, geological interpretation, and Mineral Resource block modelling, commodity prices, market conditions and foreign exchange rate assumptions, long-term consumables price assumptions, operating cost assumptions, sustaining capital costs, rock quality and geotechnical constraints, dilution and mining recovery factors, hydrogeological assumptions, metallurgical process recoveries, mining recovery and dilution assumptions

There are no other environmental, legal, title, taxation, socioeconomic, refining, political or other relevant factors known to the QP that would materially affect the estimation of Mineral Reserves that are not discussed in this Report. It is reasonably expected that all necessary government approvals will be issued for the project to proceed.

25.4 Mining Methods

The Oko West Project is planned as a mining operation that integrates both conventional open pit (OP) mining and mechanized long hole open stoping for the underground (UG) mine. The initial milling rate is set at 7 Mtpa with a mix including saprolite, decreasing to 6 Mtpa when fed solely with hard rock. The milling process is designed to operate for 12.3 years, with stockpiles peaking at 2.2 Mt by Year 1 to maintain consistent mill feed

The OP operation will utilize a fleet of diesel-powered equipment, including drills, haul trucks, and hydraulic shovels. The Project consists of a main pit that is deeper and centered on Block 4, with one (1) smaller sub-pit located south of the southern extension to the main pit. The OP operation will be executed in four (4) phases. The OP peak mining rate is 44.0 Mtpa over a Life-of-Mine (LoM) of 13 years. A total of 62.4 Mt of ore material will be mined at an average diluted gold grade of 1.57 g/t Au. A total of 426 Mt of combined waste and overburden will be extracted, resulting in a strip ratio of 6.8 tonnes of waste per tonne of ore material. The primary production equipment includes 22 m^3 diesel-hydraulic shovels paired with 140-t off-highway mining trucks. The mining operation is planned to be owner-operated, with exception for explosives handling and blasting services. Pre-production mining is scheduled over approximately

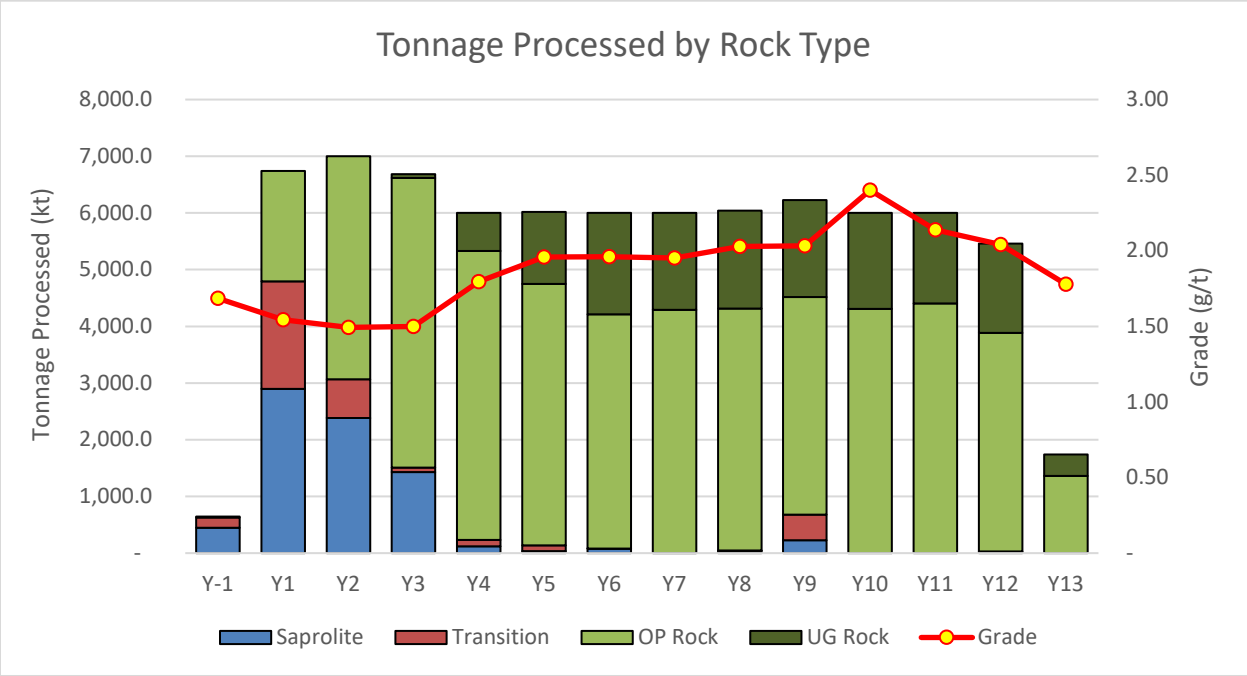
24 months to secure construction material and to remove overburden to allow access to the ore material. A total of 47 Mt of waste as well as 2.6 Mt of ore material will be mined during the preproduction and commissioning period.

The underground (UG) operation comprises a single mine divided into two (2) zones: the main zone and the satellite zone. Both zones are accessible through a common portal located in the south pit and connected via a single decline ramp. The chosen mining method is long hole open stoping (LHOS), with both transverse and longitudinal stoping techniques being employed.

The LoM for the UG mine is expected to be 13 years including construction, development, pre-production, and the full production period. Over this LoM, the UG mine is expected to be in production for 10 years, including a 2-year ramp-up period. A two-year pre-production period is planned to allow sufficient underground development to be completed to sustain full production. The UG mine is expected to achieve an average production rate of 4,625 tpd of mineralized material, with 4,500 tpd derived from stope production and 125 tpd from lateral development. Development of the UG mine includes approximately 51.8 km of lateral and 2.6 km of vertical development to be excavated. A total of 14.2 Mt of ore material is expected to be mined at an average diluted gold grade of 3.26 g/t Au. The primary production equipment includes 21-t diesel-powered load-haul-dump machines (LHD) coupled with 63-t underground mining trucks to handle all mined material.

The combined production for the OP mine and the UG mine is 76.6 Mt of ore at an average grade of 1.89 g/t Au. Figure 25.1 shows the processing plan based on the mineralized material type and origin.

Figure 25.1: Processing by Rock Type



25.5 Metallurgical Testing and Mineral Processing

The development of the Oko West process flowsheet was underpinned by comprehensive metallurgical test work and engineering simulations conducted during the PEA and Feasibility Study phases. The analysis confirms that the selected design and equipment specifications are well-supported by technical evidence and are aligned with the project's operational and recovery objectives.

The crushing and grinding circuits have been rigorously evaluated through simulation using industry-recognized modeling software and expertise. The modeling incorporated key input parameters such as ore blend characteristics, power requirements, and equipment capacity limits. The results demonstrate that the selected mill and crusher sizes are appropriately configured to handle the anticipated ore throughput and variability in feed composition.

Settling and thickening tests both static and dynamic support the design of the pre-leach thickener. These results indicate that the thickener is adequately sized to meet process water recovery requirements while maintaining acceptable underflow density and overflow clarity.

Leaching and gravity recovery test work confirm that the proposed Leach-CIP carousel circuit configuration will maximize gold recovery while effectively minimizing carbon losses to the tailings stream and maintaining a low gold inventory. The gravity test results were instrumental in defining the equipment sizing and

operating strategy for the gravity concentration step, with a clear indication of high gravity-recoverable gold content in the ore.

The cyanide detoxification test work validates the capacity of the selected detoxification circuit to reduce WAD cyanide concentrations to below 1 mg/L, ensuring compliance with environmental discharge standards. This outcome supports the suitability of the detoxification approach for both environmental performance and regulatory compliance.

Ongoing optimization studies are evaluating the opportunity to increase plant throughput during the initial ramp-up period prior to the installation of the pebble crusher. Oxygen uptake testing has shown that the ore has relatively low oxygen demand during leaching, which may reduce operational oxygen supply requirements and associated costs.

Collectively, the metallurgical test work and engineering assessments provide a sound technical basis for the proposed process design. They support the conclusion that the flowsheet is both technically feasible and economically robust, with potential for further optimization during operations.

25.6 Recovery Methods

The proposed process plant design for the Oko West Project is based on a standard metallurgical flowsheet to treat gold bearing ore via cyanidation to produce doré. The flowsheet is based on metallurgical test work, industry standards and conventional unit operations.

The process plant is designed to nominally treat 6 Mtpa of fresh rock (7 Mtpa with a blend of fresh rock and saprolite / transition ore) and includes the following unit operations:

- Comminution circuit (primary gyratory crusher, SAG and ball mill) to produce a primary grind size of P80 of 75 µm.
- Gravity concentration circuit to produce a gold-rich concentrate for intensive leaching and subsequent gold recovery via electrowinning.
- Pre-leach thickening.
- Cyanide leaching and carbon adsorption via a Leach-CIP carousel circuit. Leach residence time of 36 hours to achieve optimal gold extraction.
- Carbon elution via 10-t Split Pressure Zadra circuit.
- Carbon handling and regeneration.

- Electrowinning and smelting to produce doré.
- Cyanide destruction of Leach-CIP tailings using SO_2 / O_2 process to produce weak acid dissociable (WAD) cyanide levels of less than 5 ppm.
- Tailings pumping to a tailings' storage facility.
- Air and oxygen circuits.
- Water systems (domestic water, raw water, gland seal water and process water).

25.7 Project Infrastructure

The Oko West Project requires several infrastructure elements to support the mining and processing operations.

The infrastructure planned for the project includes the following:

- Permanent camp.
- Wharf Site camp.
- Kitchen & Lunchroom.
- Camp Office Welcome Centre, Laundry and Recreation Centre.
- Mine Maintenance Facility & Warehouse area.
- Main Administration Building.
- Mine Dry.
- Explosive Storage.
- Mill Offices.
- Assay Laboratory.
- Reagent Storage.
- Process Plant.
- Industrial / Fire Water.
- Potable Water Treatment.
- Sewage Treatment and Oil-Water Separation.
- Contact water collection pond.

- Diversion Channel.
- Fuel Storage and Distribution.
- HFO-fired power plant.
- 13.8 kV power distribution.
- Communications.
- Wharf and storage facility.
- Waste Storage Facility and Tailings Storage Facility.

25.8 Environmental, Social and Permitting Considerations

A Project specific baseline study program was initiated in 2022 and completed in early 2025. The information collected has built an understanding of the spatial and temporal, local and regional environmental and social conditions that have informed the FS and allowed submission of the preliminary ESIA in December 2024 and final submission in May 2025. The Project area does not overlap with any protected or Indigenous lands and is not located within a priority conservation zone, although it is located within a region of high biodiversity significance.

Baseline data shows an environment already impacted by region-wide and degradation caused by artisanal mining for gold and timbering, with effects especially apparent in river water quality and flow and associated habitats. Artisanal mining is also the primary context for the social and community setting in the Project area and surrounding area, with active mining immediately to the west and north associated with a declining community at Oko Landing.

For many topics, such as surface water and groundwater and biodiversity, baseline studies have advanced into ongoing regular monitoring programs, as captured in the overarching ESMMP part of the EIA, that covers many of the more specific plans and activities described as mitigations of impacts in the section above, such as the Biodiversity Management Plan, a Traffic Management Plan, or a Surface Water and Groundwater Monitoring Plan.

Key findings of the baseline include the presence of Giant Otters and an archaeological cave within the Project area, and the extent to which current artisanal mining is present in the area and has negatively affected the environment. These features represent the main receptors to the environment and social baseline from the Project. Most streams in the area were previously heavily mined by artisanal miners and have little-to-no community or cultural value, nor are they critical for biodiversity or populations of species of interest, while the quantity of water affected has little influence on downstream regional streamflows. The

baseline of other environmental media, such as soil, air, noise, terrestrial habitats and groundwater are relatively undisturbed, especially away from the sites of artisanal mining in stream and river valleys.

A thorough geochemical testing program with both static and kinetic testing was undertaken by ERM between 2023 and 2025 on waste rock and tailings materials for the Project, with overburden (soil) samples still underway. The purpose of the geochemical characterization program was to determine the ML/ARD characteristics of materials excavated, produced and stored during the development, operation and ultimately closure phases of the Project. The majority of materials were non-PAG, with some uncertain and a few marginally PAG – these were from the transitional saprolites and some of the (carbonaceous) sedimentary rocks, which are volumetrically a small proportion of the waste rock and tailings. Otherwise, most of the other materials have some buffering potential. An ML/ARD management plan will be developed to mitigate the potential adverse effects from waste rock on the receiving environment to include periodic static testing undertaken though the life of mine to correctly identify PAG / non-PAG material, and the placement of saprolite and transitional waste material in central areas of the WSF, to be encapsulated by the less reactive waste rocks through the course of operations.

The project will impact environmental and social receptors in a variety of ways, including positive impacts from the socio-economic benefits at a local, regional and national level. Construction and Operation of the Project will result in large magnitude impacts to surface water quantity, quality (in the Takutu River) and associated aquatic habitats and ecology, groundwater quantity, terrestrial habitats, soil and noise. There will be lower magnitude impacts to air, cultural heritage and other social receptors in the area of influence. All of these impacts have mitigations identified to reduce the magnitude such that residual impact significance is rated negligible or minor for most impacts, although some of these are only realised following the end of operations as the Mine Closure Plan is implemented. Key to applying and measuring the effectiveness of mitigation is the ESMMP which collates all the monitoring and management activities and commitments for the different environmental media and manages the relationship with the local community and other stakeholders. Examples plans within the ESMMP are the Biodiversity Management Plan, Dust and Air Quality Management Plan, Health and Safety Management Plan, Stakeholder Engagement Plan, Dam Breach Emergency Response Plan, Sustainable Water Resources Management Plan and the periodically updated Mine Closure Plan.

The Mine Closure Plan for the FS has been updated from that created for the PEA and features progressive closure of the WSF during later operations, a pit lake after operations, and transfer of ownership of the access roads, airstrip and barge landing as useful assets. The closure of the TSF is, at this stage, uncertain and may require security provisions and water treatment in perpetuity. Consequently, a large contingency has been included in the total closure provision of US$ 38.7M.

Key residual impacts and uncertainties at this stage are the site water management and the long term closure of the TSF outlined above. In the current water management design and understanding of geochemical sources, discharge from the sediment pond and flocculant treatment facilities into the north tributary of the Takutu River is expected to increase concentrations for chemical constituents, with some, including but not limited to mercury and arsenic, exceeding IFC water quality guidelines. The discharge volumes are also modelled to exceed capacity in the same tributary. Further pond sizing or additional water treatment infrastructure may need to be considered in future detailed design phases.

Aside from these uncertainties, ERM is of the opinion the baseline information collected is sufficient to inform project permitting and the impact assessment undertaken has not identified any material issues of concern that would deem a project not viable in this location.

25.8.1 Permitting

The EIA was formally submitted to Guyana's EPA in late December 2024 and remains under review. Public consultation meetings were held in January and February 2025 in local communities, providing critical input for the development of environmental and social programs aligned with regional sustainability priorities. GMIN has completed its responses to all clarification and supplementary information requests, which were submitted to the EPA on May 19, 2025. Final environmental approval and the construction permit is expected in Q2 2025.

Also, in mid-December 2024, GMIN received an Interim Environmental Permit ("IEP") from the EPA authorizing the commencement of early works construction activities. The Company is advancing key supporting infrastructures to support full construction, including the installation of water and sewage treatment systems, camp, access roads and wharf area for logistics.

In parallel, GMIN has initiated applications for other key regulatory authorizations required for the Project's implementation, including the Mining Licence, port operation, permits for fuel use and storage, and approvals for the installation of transmission and telecommunications towers. These complementary permits will support full-scale construction and operational readiness. All permitting efforts are guided by proactive stakeholder engagement and adherence to international environmental and social performance standards.

25.9 Capital and Operating Costs

25.9.1 Capital and Operating Costs

Life-of-mine project capital costs are estimated to total $1,730 million consisting of the following three (3) distinct phases (Table 25.2):

- Initial Capital Expenditure – This phase includes all costs to develop the property with a process plant designed to nominally treat 7 Mtpa blend of saprolite, transition and fresh rock. Initial capital costs total $1,041M (including $85 million for contingency and excluding $69 million in pre-production revenue), which will be expended over 34-month of engineering, construction, pre-production and commissioning period.
- Sustaining Capital Costs – This phase includes all costs related to the acquisition, replacement, or major overhaul of assets required to sustain operations and the underground development. The plant will be configured to treat 100% fresh rock at 6 Mtpa. Sustaining capital costs are estimated to be $650 million including indirect costs but excluding contingency.
- Closure Costs – This phase includes all costs related to the closure, reclamation, and ongoing monitoring of the mine after operations. Closure costs total $39 million and include a 20% contingency

The initial and sustaining capital expenditures are summarized in Table 21.1 according to the Level 1 work breakdown structure (“WBS”).

Table 25.2: Initial and Sustaining Capital Expenditures Summary

Capital Expenditures (k USD)	Initial Capital Cost	Sustaining Capital Cost	Closure Cost	Total Capital Cost
100 – Infrastructure	105,889	13,404		119,293
200 - Power and Electrical	110,797	28,258		139,054
300 – Water Management	22,882	14,012		36,893
400 – Surface Operations	41,051	-		41,051
500 - Mining	104,078	575,085		679,162
600 - Process Plant	191,639	1,655		193,295
700 - Construction Indirect	134,534	-		134,534
800 - General Services / Owner’s Cost	137,385	-		137,385
900 - Pre-production, Start-up, Comm.	108,173	17,589		125,762

Capital Expenditures (k USD)	Initial Capital Cost	Sustaining Capital Cost	Closure Cost	Total Capital Cost
990 - Contingency	84,962	-		84,962
Closure Cost			38,696	38,696
Total	**1,041,389**	**650,003**	**38,696**	**1,730,088**

25.9.2 Operating Costs

The operating costs include mining, power, processing, general services and administration ("G&A"), gold transportation and refining and royalties. The average OPEX is $958/oz or $54.34/t milled. The average LOM all-in sustaining cost ("AISC") is $1,123/oz of gold or $63.70/t milled.

25.10 Economic Analysis

The economic analysis is carried out in real terms (without inflation factors) in H12025 United States dollars.

The base case economic analysis using a gold price of $2,500/oz has an after-tax NPV 5% of $2,163M, IRR of 27.1% and payback of 2.9 years after the start of production.

The Project financial performance is most sensitive to the gold price and to a lesser degree to the capital and operating costs.

25.11 Risks and Opportunities

25.11.1 Risks

The main Project risks are:

- Unpredicted local variability of gold distribution, especially in the first phases of the open pit.
- Tension with Venezuela (dispute of land ownership).
- Piracy targeting in-country logistics.
- Limited labor availability, notably on account of competition from the oil and gas industry.
- Limited room available at the Port of Georgetown.
- Delays in procurement.
- Very dry climate limiting barging at wharf facility.

- Water management and discharge.
- Granting permit.
- Inflationary pressure.
- Long supply chains.
- Changes of laws and regulations in country.

25.11.2 Opportunities

The main Project opportunities are:

- Increased Mineral Resources at depth (Block 6) and south (Block 7), and further Mineral Resource conversion from Inferred to Indicated in Block 6.
- An alternate exploration concept involving fault splays currently being tested may increase the in-pit resource potential.
- Exploration potential property wide.
- Increase the size of underground stopes (level spacing).
- Reduce the amount of cement required for the CRF by modifying certain stopes (Avoca method).
- Reduce the size of primary stopes and increasing the size of secondary ones.
- Utilisation of paste backfill.
- Accelerate UG development and production.

26. RECOMMENDATIONS

Based on the results of the technical and financial analyses of this Feasibility Study (FS), which demonstrates positive project economics, GMS recommends proceeding with the detail engineering for the project and start the early works construction as an advanced phase prior to the full construction release once project financing is finalized and the main permits are issued.

Table 26.1 summarizes the proposed budget of $41.6M to advance the project in the detailed engineering stage and perform the recommendations discussed in this section.

Table 26.1: Cost Estimate Associated with Recommendations

Description	Amount ($000)
Geology extensions and splay model testing	8,000
Mining engineering – OP west slope improvement	300
OP design optimization	200
Geotechnical testing and dam break study	100
Environment – biodiversity and cave recovery	200
Detail engineering	29,000
Contingency at 10%	3,780
Total	**41,580**

26.1 Geology and Mineral Resources

Based on the results of the Feasibility Study described in this report, GMIN plans to continue its exploration program in 2025. GMIN intends to pursue exploration property-wide and test a new exploration model to potentially increase mineral resources.

GMS makes the following recommendations to advance the Oko West Project:

- Test the alternate exploration concept involving fault splays and evaluate its impact on the gold distribution.
- Following positive results, update the geological 3D model.
- Prepare a geostatistical study to determine the optimal drill spacing for Measured Mineral Resource.
- Continue drilling for potential mineral resource extensions (Block 7) to the existing deposit.

26.2 Mine Geotechnics

26.2.1 Opportunities

- In Sector 2 (Figure 16.3), which encompasses the saprolite on the west wall, conservative design parameters were adopted due to uncertainties in several key inputs, including the stratigraphic profile, strength variation with depth, and pore water pressure conditions. A targeted investigation campaign to improve data coverage in this area has strong potential to support refined and possibly less conservative slope design criteria.
- In Sectors A and B (Figure 16.3), slope design guidelines have been constrained by the presence of shallow-dipping discontinuity sets. With further geotechnical investigations, these could likely be improved.
- Due to the limited scale of data collection at this stage – primarily based on core logging, which provides a relatively small sample size compared to mapping exposed rock faces – a conservative assumption was made regarding the persistence of these features. As mining progresses and rock slopes are exposed, this assumption can be reassessed. If field observations indicate lower persistence, there may be potential to optimize inter-ramp angles and steepen the slopes.

26.2.2 Limitations

- At the time of the geomechanical drilling campaign, the southern pit wall height was less than 40 m. As a result, this area was not directly investigated, and the design guidelines were extrapolated from geomechanical data obtained from drillholes intersecting the same geotechnical units elsewhere. These assumptions should be verified during the detailed engineering phase, prior to initiating mining in this area.
- Apart from the east-south-east subvertical fault separating Blocks 4 and 5 – about which limited information is available – no structural model incorporating faults or major shear zones has been developed. Establishing such a model would help ensure that no significant structures adversely affect the open pit or underground designs.
- The current slope design guidelines assume effective water management and consistently low pore water pressures throughout the mine life. Continued monitoring and management will be required to maintain these conditions.

26.2.3 Future Works

- Optical and acoustic televiewer surveys should be conducted in Sectors A and B (Figure 16.3) as they have the potential to be steepened.
- Once exposed, LiDAR scans or photogrammetry could be used to review structural orientations, refine the geotechnical model and validate the design criteria.

26.3 Geotechnical and Hydrological Studies

The following is recommended to be completed during the detailed engineering phase of the future surface plant and required infrastructure.

- Complete geotechnical investigation for the plant foundations.
- Balance cut and fill requirements for earthworks.
- Complete test work required to confirm an adequate source of on-site sand for concrete quality requirements.
- Compile hydrology data and maintain monitoring of water streams. Improve the hydrology model.
- Perform in-situ and laboratory geotechnical test work to support the design of the TSF dams and water management.

26.3.1 Waste Storage Facility (WSF)

The design was carried out according to the industry's best practices, presenting no fatal flaws. The slope stability analysis demonstrated satisfactory Factors of Safety.

Some recommendations to ensure the safety of WSF include:

- Remove the vegetation in the interior of the pile for use in future reclamation.
- Run strength parameters laboratory tests, primarily in the residual soils, the more representative foundation material, to confirm design assumptions.
- If waste rock is available before the operation of the WSF 2, load the alluvium layer in the pile's interior to test its consolidation response. If failures occur during the tests, consider removing the alluvium and replacing it with rock fill in the offset of the pile.
- Consider the use of satellite interferometry to monitor displacements in the WSF 2, if this technique is available in Guyana.

26.3.2 Tailings Storage Facility

The design was carried out according to the industry's best practices, presenting no fatal flaws. The slope stability analysis demonstrated satisfactory Factors of Safety.

- The dam raise design considers storage provision and water pumping for safety in all phases before the final dam crest is achieved, when a free overflow spillway is constructed. The evaluation of the benefits of an operational spillway (a channel excavated in soil) to provide additional safety during these intermediate phases is recommended.
- If necessary, confirm that tailings are non-hazardous and inert after the conclusion of geochemical testing.
- Complete the laboratory tests program to confirm design parameters.
- Foundation treatment must consider removing all the alluvium / tailings and colluvium at the foundation and abutments, to assure stability.
- Plan the construction stages in the dry periods due to the high natural moisture content of the construction materials.
- Provide details of the transition layers in the drain design.
- Confirm instrumentation required to monitor the main dam.
- Perform a dam breach analysis.

26.4 Environmental, Social and Permitting Considerations

The development of the Project will necessitate large magnitude impacts to a variety of environmental media (air, soil, water and so on) and social and community receptors, including positive socio-economic benefits across local, regional and national contexts. The ESIA identified numerous negative impacts, and it is recommended that these are managed via the Environmental and Social Management and Monitoring Plan (ESMMP) developed in the ESIA, implemented at various stages of the Project:

- Construction and Pre-Construction Management Plan covering:
  - Air and dust.
  - Noise.
  - Hydrology.
  - Groundwater.
  - Surface water and sediments.

  - Soil erosion and control.
  - Community health and safety.
  - Occupational health and safety.
  - Cultural Heritage (including Chance Finds Procedure).
- Operational Management Plan covering:
  - Air and dust.
  - Noise.
  - Hydrology.
  - Groundwater.
  - Surface water and sediments.
  - Soil erosion and control.
  - Community health and safety.
  - Occupational health and safety.
  - Cultural Heritage (including Chance Finds Procedure).
- Plans covering all Project Phases:
  - Waste Management Plan (including Hazardous Materials Plan).
  - Sustainable Water Resources Management Plan.
  - Cyanide Management Plan.
  - Emergency Response Plan (including Spill Control Plan and Dam Breach Plan).
  - Biodiversity Management Plan.
  - Stakeholder Engagement Plan (including Community Grievance Mechanism).
  - Rehabilitation and Mine Closure Plan (periodically updated, including stakeholder engagement).
  - Local Development Plan.
  - Influx Management Plan.

In terms of other risks, the Water Balance Model indicates that for the current water management design and understanding of geochemical sources, discharge of treated contact water from the sedimentation pond into the north tributary of the Takutu River is expected to increase concentrations for some chemical constituents, including mercury and arsenic, above IFC water quality guidelines. In addition, the discharge

is modelled to likely exceed the Takutu River north tributary's bankfull discharge capacity. It is recommended that these quality and quantity issues be addressed in further pond sizing or additional water treatment studies, undertaken to inform the final detailed design. Moreover, it is recommended that the Water Balance Model be re-run, incorporating any available improvements to input parameters and assumptions, to test proposed water management designs.

The closure of the TSF is, at this stage, uncertain, due to the significantly low bearing capacity of the tailings; conventional capping, covering and revegetation of the TSF surface is not considered a feasible option. Current plans indicate that the TSF will require perpetual water and geotechnical maintenance and monitoring, as well as perpetual security measures to prevent unauthorized and hazardous artisanal mining of the tailings. It is recommended that iterative alternative closure options assessments be undertaken to explore optimized solutions for closure of the TSF during the operational life of the mine, and may include transitioning to dry-stack tailings, or complex engineered cover designs such as iteratively layering waste rock and geotextiles to provide bearing capacity.

The current closure concept assumes that an effluent treatment plant (ETP) will not be required, due to there being evidence that the greater proportion of tailings will not be PAG, that the cyanide destruction plant will reduce concentrations to acceptable levels, and that these will be further attenuated by natural degradation processes and diluted by surface water run-off within the large TSF catchment. It is recommended that this assumption be further investigated and tested, and in the event that tailings display ML/ARD characteristics at the time of closure, water treatment costs will need to be recalculated to reflect this change.

Aside from these uncertainties, ERM is of the opinion that the baseline information collected is sufficient to inform project permitting and the impact assessment undertaken has not identified any material issues of concern that would deem a project not viable in this location, as long as the recommended suite of mitigation activities identified in the ESMMP and Mine Closure Plan are implemented.

26.4.1 Permitting

Table 26.2 presents the complete list of permits required for the construction and operations phases of the Project.

An interim Environmental permit was obtained from the EPA to allow the Early Works (Camp, Roads, Airstrip, Water treatment plant) to proceed. GFC also granted the Deforestation Permit required for Early Works.

A main permit required to fully proceed with the Project is the Environmental Permit to be issued by the EPA. The final ESIA, including comments and requests, was filed with the EPA in May 2025, and the Permit is expected to be issued by the end of June.

Another main permit is the Mining Licence to be issued by the GGMC, and the application will be filed in May 2025.

The permitting process is critical to meet the project execution schedule and estimated initial capital expenditures. GMIN must be responsive and flexible to meet the expectations of the Government of Guyana and all stakeholders. An initial commitment involves a Biodiversity Study. A final recovery of possible artifacts at the Cave is also recommended.

Table 26.2: Current Status Permits (May 2025)

Permit Name	Description / Steps	Agency	Status	Application Deadline	Deadline to be Issued	Comments / Follow-ups
No Objection Authorizations for Early Works	Deforestation of the Wharf area (pier and lay down)	**GFC**	**GRANTED**	**Nov-24**	**Mar-25**	Issued 17th Dec. 2024. No Commercial inventory to be done for the Wharf area.
	Deforestation Permit for all site (camp, road, mining area, mill area and airstrip)	**GFC**	**GRANTED**	**Nov-24**	**Jan-25**	Issued 17th Dec. 2024. Commercial inventory to commence on 20th Jan. 2025.
	Construction for the camp, road and airstrip, and treatment plants	**EPA**	**GRANTED**	**Nov-24**	**Jan-25**	Interim Environmental Permit Granted on 16th Dec. 2024.
	Construction of 14 km Access Road	**GGMC**	**GRANTED**	**Dec-24**	**Q1 25**	No Objection granted for construction of road access on 23 Jan. 2025.
Wharf Facility - Lease Land	Lease ownership of the barge facility and laydown area	**GLSC**	**In Progress**	**Feb-24**	**Q1 25**	Surveying completed. Awaiting the granting of both leases.
Environmental Permit (EPA)	Environmental Authorization for emissions and discharges from the proposed operation.	**EPA**	**In Progress**	**Nov-24**	**May-25**	Submitted to EPA on Nov. 27th. Interim Environmental Permit granted for the Oko Project to cover Early works. Currently reviewing comments from the EPA.
Tower Installation	Authorization to build a communication tower	**Telecom.**	**In Progress**	**Q1 25**	**Jul-25**	Application submitted and being reviewed. Meeting with the Ministry of Housing for discussion of the application.
Frequency Allocation Permit (Licence for the Installation and Operation of Radio Equipment)	Authorization for the installation and operation of Radio equipment	**Telecom.**	**GRANTED**	**Dec-24**	**Q1 25**	Application for 6 frequencies and radios.
Well Installation Permit	Authorization for installing water wells	**Hydromet. / EPA**	**In Progress**	**Q1 25**	**May-25**	Application submitted to Hydromet. Office.
Watercourse Diversion Permission	Authorization for diverting watercourses within the project area	**EPA / Hydrometeorological Office / GGMC**	**To be Applied**	**Q1 25**	**Apr-25**	Awaiting technical details for submission. Presentation will have to be done with GGMC by the Technical team.
Wharf Facility Permit	Authorization to operate the barge landing facility	**MARAD**	**In Progress**	**Q1 25**	**May-25**	Application submitted to the MARAD office.
Wharf Facility Permit	Authorization to operate the barge landing facility	**Sea Defence Board**	**In Progress**	**Q1 25**	**May-25**	Application submitted to the Sea Defence Board for No Objection.
Mining Licence (GGMC)	Authorization for Mining	**GGMC**	**In Progress**	**Q1 25**	**Dec-25**	Application submitted to the GGMC.
Fuel Station	Consumer Installation Licence	**GEA**	**To be Applied**	**Q2 25**	**Jun-25**	Application to be done in April 2025. Some changes in fuel storage capacity at site and the Wharf.
	Petroleum Storage Licence	**Guyana Fire Service**	**To be Applied**	**Q2 25**	**Jun-25**	Awaiting Advice on submission of application.
Radiation Permit	Authorization for the use of Equipment that emits radiation during construction	**EPA / GGMC**	**To be Applied**	**Q2 25**	**Jun-25**	Leeroy and Mathieu are to send details of the equipment to be used. BJ to write GGMC for permission.
Electrical Installation and Inspection Licence (GPL)	To protect against the hazard of unsafe and unsound electrical installations	**GPL**	**To be Applied**	**Q2 25**	**Jun-25**	
Aerodrome Licence (GCAA)	To operate Airstrip	**GCAA**	**To be Applied**	**Q2 25**	**Jul-25**	
Radio Frequency Licence (Ground to Aircraft Communication)	Licence for Ground to aircraft communication	**Telecom.**	**To be Applied**	**Q2 25**	**Jul-25**	
Road Users Agreement	Maintenance of the Puruni Access road	**GGMC / MONR**	**To be Applied**	**Q2 25**	**Jul-25**	
Construction of the Airstrip	Building of the Airstrip at the Oko West project site	**GCAA**	**To be Applied**	**Q4 25**	**Jun-26**	Application yet to be filed. Awaiting technical data from Mathieu.

Permit Name	Description / Steps	Agency	Status	Application Deadline	Deadline to be Issued	Comments / Follow-ups
Explosives - Use Permit	Authorization to use and store explosives	**GGMC / Police**	**To be Applied**	**Q3 25**	**May-26**	Two (2) permits needed, one from the GGMC and one from the Commissioner of the Guyana Police Force.
Pesticides and Toxic Chemical Board Licence	For the use of chemicals at whole site	**Pesticides and Toxic Chemical Control Board (PTTCB)**	**To be Applied**	**Q1 26**	**May-26**	
Cyanide Permit	Authorization for use and storage of Cyanide	**GGDMA / Toxic Board**	**To be Applied**	**Q2 26**	**Aug-27**	Applied for once the Mining Licence is granted. Only for operating and commissioning.
Radiation Permit	Authorization for the use of Equipment that emits radiation during mining	**EPA / GGMC**	**To be Applied**	**Q2 26**	**Apr-27**	
Export Licence - Gold	Authorization to export gold	**GGB**	**To be Applied**	**Q3 26**	**Dec-27**	When commissioning starts.

26.5 Detail Engineering

It is expected that the Detail Engineering will be performed by GMS and several specialized firms for specific facilities such as the OGM wharf, TSF, WSF, water management infrastructure, power plant and camp.

The complete detail engineering programs are estimated at $29M.

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