Feasibility Study – NI 43-101 Technical Report

Tocantinzinho Gold Project

Prepared for:

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Prepared by:

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Thiago Toussaint, MBA, CREA-MG, AMEA, SRK Consulting

Effective Date: December 10, 2021
Issue Date: February 09, 2022

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Feasibility Study – NI 43-101 Technical Report – Tocantinzinho Gold Project

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February 09, 2022

Qualified Persons

Prepared by:

(signed and sealed) "Neil Lincoln" Date: February 09, 2022

Neil Lincoln, P. Eng
President
Lincoln Metallurgical Inc.

(signed and sealed) "Charles Gagnon" Date: February 09, 2022

Charles Gagnon, P. Eng.
President, Owner
CGM Expert

(signed and sealed) "Camila Passos" Date: February 09, 2022

Camila Passos, MSc, PGeo, CREA-SP
Senior Resource Geologist
SRK Consulting

(signed and sealed) "Paulo Ricardo Behrens da Franca" Date: February 09, 2022

Paulo Ricardo Behrens da Franca, P.Eng
President
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(signed and sealed) "Thiago Toussaint" Date: February 09, 2022

Thiago Toussaint, MBA, CREA-MG, AMEA,
Principal Geoenvironmental Engineer
SRK Consulting

Qualified Persons Certificates and Consents

Following 5 pages

CERTIFICATE OF QUALIFIED PERSON

To Accompany the Report entitled:

“Feasibility Study – NI 43-101 Technical Report - Tocantinzinho Gold Project, Pará State, Brazil,”, prepared for G Mining Ventures Corp. effective as of December 10, 2021 and dated February 09, 2022 (the “Technical Report”).

I, Neil Lincoln, P.Eng, do hereby certify that:

1) I am an Independent Metallurgical Consultant and President of Lincoln Metallurgical Inc. located at 1565 Lords Manor Lane, Ottawa, Ontario, K4M 1K3, Canada.
2) I graduated from the University of the Witwatersrand, South Africa, in 1994 with a Bachelor of Science in Metallurgy and Materials Engineering (Minerals Process Engineering) degree.
3) I am a professional engineer in good standing with the Professional Engineers of Ontario (PEO) in Canada (no. 100039153).
4) I have practiced my profession in the mining industry continuously since graduation. I have over 27 years experience as a metallurgist and study manager. I have sufficient relevant experience having worked on numerous projects ranging from scoping studies, prefeasibility and feasibility studies to project implementation related to mineral processing plants. My mineral processing commodity and unit operations experience includes precious metals, base metals and industrial minerals covering metallurgical test work to process plant design. As a result of my experience and qualifications, I am a Qualified Person as defined in NI 43-101. Select gold projects include:
   - Aurizona Gold Mine Expansion Pre-Feasibility Study for Equinox Gold Corp, Maranhão, Brazil
   - Valentine Lake Gold Project (PEA) for Marathon Gold, Canada
   - Natougou Gold Project (Feasibility Study) for Semafo (now Endeavour Mining), Burkina Faso
   - Boto Gold Project (Feasibility Study) for IAMGOLD, Senegal
5) I have read the definition of “qualified person” set out in the National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections: 1.1, 1.2, 1.3, 1.4, 1.11, 1.15, 1.16, 1.18, 1.21, 1.22, 1.23, 2, 3, 4, 5, 6, 13, 17, 18 (except 18.9), 19, 21, 23, 24, 25.1, 25.2, 25.5, 25.6, 25.8, 25.10, 25.11, 26.3, 26.4, 26.5, 27.
7) I have not visited the site property that is the subject of this report.
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading.
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101.
10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 09th day of February 2022.

/signed and sealed/

Neil Lincoln, P.Eng.,
President
Lincoln Metallurgical Inc.

CGM

CERTIFICATE OF QUALIFIED PERSON

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I, Charles Gagnon, do hereby certify that:

1) I am currently Owner and President with CGM Expert with an office located at 1155, avenue des Érables, Québec. QC, G1R 2N4.
2) I have graduated from Laval University with a B.Sc. in Mining Engineering in 2002, and from University Laval, Québec, Canada with a M.Sc. in 2005.
3) I am a Professional Engineer registered with the Ordre des ingénieurs du Québec, (OIQ Licence: 130730).
4) I have practiced my profession continuously in the mining industry since my graduation from university. I have been involved in mining operations, teaching (Mining engineering department, Laval University, Québec city), engineering and financial evaluations for 17 years, including (Eleonore (Goldcorp), Perseverance (Xstrata-Zinc), Bracemac-Mcleod (Glenncore)), pit optimization, surface mine design, mineral reserve estimations and mine scheduling.
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections: 1.13, 1.14, 1.19, 1.20, 15, 16, 21, 22, 25.4, 25.9, 26.2.
7) I have not visited the site property that is the subject of this report
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading.
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Dated this 09th day of February 2022

/signed and sealed/

Charles Gagnon, P.Eng.,
Owner and President
CGM Expert

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I, Camila Passos, do hereby certify that:

1) I am currently employed as Senior Resource Geologist with SRK Consulting in an office located at Rua Gonçalves Dias, 89 – 10° Andar, Belo Horizonte, Minas Gerais, Brazil.
2) I have graduated from the University of São Paulo, São Paulo, Brazil with a MSc in Geochemistry in 2005. I have practiced my profession continuously since 2005.
3) I am a Professional Geoscientist registered with the Association of Professional Geoscientists of the province of Ontario (APGO #2431).
4) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
5) I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections: 7, 8, 9, 10, 11, 12, 14.
6) I have visited the site property that is the subject of this report from November 21 to 24, 2020. The purpose of the visit was to review the digitalization of the exploration database and validation procedures, review exploration procedures, define geological modelling procedures, examine drill core, interview project personnel, and collect all relevant information for the preparation of a revised mineral resource model and the compilation of a technical report.
7) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading.
8) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101.
9) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 09th day of February 2022

/signed and sealed/

____

Camila Passos, MSc, PGeo, CREA-SP.
Senior Resource Geologist
SRK Consulting

CONSULTORIA E PROJETOS

CERTIFICATE OF QUALIFIED PERSON

To Accompany the Report entitled:

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I, Paulo Ricardo Behrens da Franca, do hereby certify that:

1) I am a Professional Civil and Geological Engineer, employed as Director, of F&Z Consultoria e Projetos, Brazil and residing at Rua Desembargador Penna 131, Belo Horizonte, Minas Gerais, 30.320-220, Brazil.
2) I have graduated from f the Universidade Federal de Ouro Preto (Brazil) and hold a Geological Engineer title (1986), a graduate of the Escola de Engenharia Kennedy (Brazil) and hold a Civil Engineer title (1989). I also hold a Master of Science degree in Mining Engineering, from Queen´s University (Canada), obtained in 1995.
3) I am a practicing geotechnical engineer and a member of the Australasian Institute of Mining and Metallurgy (AusIMM, No. 311775). I also hold an AusIMM Chartered Professional Accreditation under the Discipline of Geotechnics.
4) I have practiced my profession continuously in the mining industry since 1986.
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections: 18.9.
7) I have visited the site property that is the subject of this report from February 21 to 23, 2019. The purpose of the visit was to inspect the dam site.
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading.
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101.
10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 09th day of February 2022

/signed and sealed/

____

Paulo Ricardo Behrens da Franca, P.Eng
Director
F&Z Consultoria e Projetos

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I, Thiago Toussaint Marcelino Moreira, do hereby certify that:

1) I am currently employed as Principal Geoenvironmental Engineer with SRK Consulting in an office located at Rua Gonçalves Dias 89, 10º Floor, Funcionários, Belo Horizonte/MG, Brazil, CEP 30.140-090.
2) I have graduated from FUMEC University, Brazil with a B.Sc. in Environmental Engineering in 2008.
3) I am a Professional Engineer registered with the Conselho Regional de Engenharia e Agronomia de Minas Gerais, (Licence: 106495/D), and Chartered Professional registered with Australasian Institute of Mining and Metallurgy as Competent Person (MAusIMM(CP) #335799).
4) I have practiced my profession continuously in the mining industry since my graduation from university. I have been involved in mining operations, engineering and financial evaluations for 14 years, including environmental site assessment of contaminated areas for inorganic compounds, water management, surface and ground water, soil and sediments quality assessment, environmental monitoring programs, QA/QC, Technical Reports and environmental Audits/Due Diligences.
5) I have read the definition of “qualified person” set out in the National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections: 1.21, and 20.
7) I have visited the site property that is the subject of this report from November 21 to November 24, 2020. The purpose of the visit was to understand the environmental and social context, actions and plans of the project site.
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading.
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101.
10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 09th day of February 2022

/signed and sealed/

Thiago Toussaint, MBA, CREA-MG, MAusIMM(CP) #335799
Principal Geoenvironmental Engineer
SRK Consulting

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1 SUMMARY

1.1 Introduction

G Mining Ventures Corp. ("GMIN") mandated G Mining Services Inc. ("GMS") and SRK Consulting Canada Inc. ("SRK") as lead consultants along with other engineering consultants to prepare an updated Feasibility Study ("FS") under the supervision of the QPs for the Tocantinzinho Gold Project ("TZ Project" or "Project") located in Pará State, Brazil.

GMIN completed the acquisition of the Project from Eldorado Gold Corp. ("Eldorado") through the purchase of all the issued and outstanding shares of *Brazauro Recursos Minerais S.A.* ("Brazauro" or "BRM") on October 27, 2021.

This Technical Report is prepared in accordance with the guidelines of the Canadian Securities Administrators' 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 mine thereat, including processing facilities and related infrastructure. This Report builds on previous Feasibility level engineering performed by Eldorado in the past with various updates to mineral resources, mineral reserves, project engineering with associated operating and capital cost estimates. The mineral resource statement reported herein was prepared in conformity with generally accepted CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines.

The Report and FS responsibilities of the engineering consultants are as follows:

- GMS: overall Report and FS coordination, property description and location, accessibility, history, mineral processing and metallurgical testing, mineral reserve estimation, mining methods, recovery methods, project infrastructures, operating costs, capital costs, economic analysis and project execution plan.
- SRK: geological setting, deposit type, exploration, drilling, sample preparation, data verification, mineral resource estimation, environmental studies, permitting and adjacent properties.

1.2 Property Description and Location

The TZ Project is an advanced-stage development gold project located in Pará State, Brazil, 200 km south-southwest of the city of Itaituba, 108 km from the Morais de Almeida district, and 1,150 km southwest of Belém, capital of Pará State.

The land tenure is comprised of two (2) mining concessions, covering an area of 12,889 hectares, 23 exploration licences covering an area of 76,116 hectares, and two (2) applications for exploration licences covering 10,569 hectares. The mineral resources reported herein for the Tocantinzinho gold deposit are located within the mining concessions.

1.3 Accessibility, Climate, Local Resources & Physiography

The Project is accessible by road via a 72-km municipal dirt road connecting to the Transgarimpeira State Road which connects to the BR-166 Cuiaba-Santarem paved highway. The travel time to Morais Almeida along the highway is approximately 3 hours. The municipal road will require ongoing maintenance by the project.

Along the access road, there is a barge to cross the Jamanxim River at Jardim do Ouro that is operated by a commercial operator whose contract is awarded by the Municipality of Itaituba. The barge has a capacity of 402 tonnes with three lanes of vehicles onboard.

Small aircrafts such as a Cessna Grand Caravan with 10 to 14 passengers can be chartered from Itaituba or other nearby cities to the airstrip located on the property. This airstrip will primarily be used to supply the site with personnel, supplies and export bullion. The flight takes approximately one hour from Itaituba and is weather-dependent.

At the Project site there is an existing exploration camp with a capacity of about 90 beds complete with kitchen, recreation room, clinic, fuel storage, core shacks, and office space.

The climate in northwestern Brazil is tropical, with a rainy season from January to April and a dry season extending from June to December. The average annual precipitation is approximately 1,957 mm.

The area is hilly with areas that are rocky or with saprolite cover. In the study area three forest typologies were observed: secondary forest, dense alluvial forest, and submontane dense ombrophilous forest.

1.4 History

Historical mining activity in the Tapajós Province region was mostly related to gold mineralization. Artisanal miners discovered gold in the region through small-scale mining activities in the 1950's. Although there are no published records to support the timing and amount of production, gold extraction in the Tocantinzinho area is thought to be initiated in 1970 with intense small-scale mining activity in the mid-eighties to mid-nineties.

- An exploration license was originally granted to Mineração Aurífera Limitada in 1979. The license expired in 1986 and the property files were archived by the DNPM in 1992.
- Renison Goldfields from Australia and Altoro formed a Joint Venture (“JV”) in 1997 to explore for gold and acquired the property. In 1998 the JV with Renison Goldfields was terminated and Altoro inherited all projects and data acquired.
- In 2000, Altoro was acquired by Solitario Resources Corporation and terminated the Tocantinzinho Project a year later due to a drop in gold price.
- In 2003, the land over the Tocantinzinho Project mineralization was acquired by Brazauro Resources Corporation, through its Brazilian subsidiary Jaguar Resources do Brazil Ltda.
- In July 2008, Eldorado made an acquisition agreement with Brazauro Resources Corporation, which involved the continued exploration and development the Tocantinzinho Project. In July 2010, Eldorado completed the arrangement to acquire all the issued and outstanding securities for ownership of Brazauro Resources Corporation.

1.5 Geological Setting & Mineralization

The Tocantinzinho deposit is underlain by igneous rocks of older magmatic arcs of the Tapajós (Cuiú-Cuiú/Creporizão). Textural evidence and contact relationships suggest that the host granitic rocks at Tocantinzinho intruded as elongate bodies along a northwest-striking fault zone that cut through more regionally extensive quartz monzonites and granites. The granitoids were likely emplaced synchronous with faulting, and both intrusive contact and vein orientations suggest the host fault zone was active during this period as a sinistral, dominantly strike-slip feature. The presence of abundant aplites, miarolitic cavities, and blebby quartz textures implies that the host granitic intrusions represent late, volatile-rich components of the parent magma. Vein textures suggest that at least some of the veins, and possibly gold mineralization, were introduced during or just after solidification of the host rocks. Mineralized granites at Tocantinzinho are visually divided into two sub-units by granite type, alteration mineralogy and colour: these are locally named smoky and salami. The smoky unit is a true granite with quartz, alkali feldspar and plagioclase whereas salami is an alkali feldspar granite comprising quartz, K-feldspar and albite. Contacts are diffuse and a complete gradation exists between the two units. The mineralized granites are intruded by an andesitic body outcropping along the axis of the deposit.

The Tocantinzinho deposit forms a sub-vertical, northwest-trending elongate body approximately 900 m long by 150-200 m wide. It has been drilled to approximately 450 m depth and remains open below this depth. Within the mineralized granite, gold grades are remarkably consistent and are associated primarily with pyrite in sheeted veins and veinlets.

1.6 Deposit Types

The Tocantinzinho deposit is best classified as a granite-hosted, intrusion-related gold system (IRGS). It has many of the characteristics of an IRGS, including a fractionated granite host rock package, mineralized magmatic-hydrothermal transition textures and alteration assemblages with early potassic-sodic feldspar through to silicification and pervasive to vein controlled quartz-sericite-chlorite-calcite.

However, some features of the Tocantinzinho gold deposit are not typical of an IRGS, such as the moderately oxidized state of the intrusion, and the unusual age (Paleoproterozoic, ~2,007 Ma) of the deposit, suggesting that Tocantinzinho may be one of the oldest examples of this deposit type.

Tocantinzinho does not appear to fit the orogenic classification for several reasons despite the deposit and related intrusion occurring in a major regional structure. At a regional scale, the deposit is not hosted in a metamorphic terrane and at the deposit scale, the mineralization is controlled by granite-facies development and related veins and alteration rather than fault-related features.

1.7 Exploration

The exploration work at the Project completed to date can be separated into two distinct periods: Brazauro Resources (2004 – 2008), and Eldorado Gold Corp. (2008 – 2021). Early exploration by Brazauro was focused on regional-scale geological mapping, channel and chip sampling, soil and stream sediment geochemical surveys, auger drilling, and core drilling. Geophysical surveys were completed during the period, including a ground-magnetic survey covering the TZ Deposit.

Since the acquisition of the property by Eldorado in July 2010, soil and channel sampling continued with the goal of discovering extensions and/or parallel trends to the TZ deposit. A topographic aerial laser survey of the project site was carried out in September 2010 by Geoid Ltda. A total area of 53 km^2 was surveyed including the deposit, possible tailings dam areas and the future plant site. In late 2010, Eldorado completed an induced polarization (IP) geophysical survey of 45 km line, covering the areas along the Tocantinzinho trend to the northwest and southeast of the deposit. In 2011, aerial and ground magnetic survey data collected in 2005 were re-interpreted.

1.8 Drilling

The earliest known drilling on the property was undertaken by Altoro in 1998 and 1999, but only consisted of regional-scale power auger drilling for which few details exist.

Brazauro and Eldorado drilled a combined total of 296 core boreholes (approximately 82,805 m) on the Tocantinzinho Gold Project between 2004 and 2021. Several auger and reverse circulation boreholes have also been drilled in the Tocantinzinho Gold Project; however, as this drilling data was not used for the resource estimation, they are not discussed in detail in this section.

Early core drilling by Brazauro involved drilling saprolite and the weathered rock using NTW diameter, which was reduced to BTW diameter when fresh rock was reached. Brazauro completed down hole surveys at intervals that varied from 15 to 376 m using a Reflex FlexIt tool. The collar was surveyed using a total station instrument.

Subsequent drilling by Eldorado used more powerful drill rigs, making it possible to drill deeper and produce wider varieties of core in addition to BTW, including HQ and NTW sizes. Borehole collar surveys were carried out using a total station instrument. Downhole surveys were completed in intervals of 50 or 60 metres using ReflexIt and Reflex EZ shot instruments.

Reserve circulation drilling methods were employed in exploration adjacent to the TZ deposit but was not part of the resource definition drilling campaigns.

Drilling in the tailings was undertaken using core drilling, vary from 2 to 40 metres in length and were only surveyed using handheld GPS at the time of drilling. In 2021, GMS contracted Geotec Projetos e Serviços from Divinópolis, Minas Gerais to resurvey the tailing collars boreholes using an RTK (Real-Time Kinematic) equipment. GMS was able to locate 74 collars tailing boreholes, which represents around 48 percent of the tailing borehole total.

1.9 Sample Preparation, Analyses and Security

Diamond drilling is the principal sample collection method at the TZ Deposit, and the mineral resource has been built exclusively using this data. All diamond drilling in Tocantinzinho was done with wire line core rigs and mostly of HQ size (with minor amounts of BTW and NTW core diameters). The entire lengths of the diamond drill holes were sampled with sample intervals ranging from 0.5 to 2.0 m, usually at two-metre-long intervals producing samples weighing between 2 to 3 kg. Core was logged and samples handled in accordance with industry best practices. Sample analyses were initially performed by SGS Geosol, and then subsequently by ALS Chemex, and ACME laboratories for the majority of the drilling.

Samples submitted by Brazauro were analyzed using a fire-assay method with a 50 g sample weight. Eldorado made modifications to the sample analysis procedure in 2010, using a 30 g sample weight and a gravimetric finish was performed on fire assays returning more than 10 g/t gold. Samples with visible gold were submitted to a metallic screen analysis.

Sample batches were arranged to contain regularly inserted control samples by both Brazauro and Eldorado. Eldorado inserted a standard reference material (SRM) and a blank into the sample stream at every 10th to 40th sample, and a duplicate at every 15th sample. The duplicates are used to monitor precision, the blank sample can indicate sample contamination or sample mix-ups, and the SRM is used to monitor accuracy of the assay results. The SRMs varied between 0.89 g Au/t and 13.64 g Au/t. Monitoring of the quality control samples showed all data were within control throughout the preparation and analytical processes.

Specific Gravity measurements were taken by Eldorado regularly using a standard weight in water/weight in air methodology on the unweathered core over complete sample intervals. The weathered samples were wrapped in PVC film and weighed before being submerged in a beaker in water. The total water displacement is measured and recorded. The sample was then dried at 250 degrees Celsius for 90 minutes and weighted again.

Drill core is securely stored in the core sheds at Tocantinzinho Gold Project exploration camp.

1.10 Data Verification

To comply with National Instrument 43-101 guidelines, a site visit to the Tocantinzinho gold deposit was completed between November 21 to 24, 2020 by Ms. Camila Passos and Mr. Thiago Toussaint from the SRK Brazil office. Core drilling was not ongoing while SRK was on the site. SRK reviewed drill core from Tocantinzinho deposit, Santa Patricia and the KRB target, and inspected core storage facilities, and reviewed field procedures. In addition, a selection of drill collar coordinates was verified.

Both Brazauro and Eldorado undertook database verifications and quality assurance and quality control programs, including sending sample pulps to a secondary umpire laboratory, and external conducting external database verifications.

1.11 Mineral Processing & Metallurgical Testing

A significant amount of metallurgical test work has been completed on granite ore, saprolite and artisanal mining (garimpeiros) tailings samples and composites related to the Project including:

- Ore variability in terms of lithology, gold head grade, sulfur head grade, depth, and sample blending

- Metallurgical test work for primary sulfide ore, gold bearing soil, saprolite, transitional and garimpeiros tailings
- Detailed chemical analyses of ore feeds, flotation concentrates and flotation tailings
- Ore mineralogy and characteristics assessment
- Comminution testing including Bond crushing, rod milling, and ball milling indices; SMC index, and abrasion index
- Whole ore cyanide leach and cyanide leach of flotation concentrates
- Flotation including batch rougher and cleaner, locked cycle, and pilot plant
- Gravity recoverable gold
- Thickening testing of ore feed, flotation concentrate, leached residue and flotation tailing
- Cyanide detoxification (several methods) and aging test work on tailings and effluent
- Environmental and geotechnical testing of residue

Pilot plant flotation and cyanide test work were completed by Wardell Armstrong International in UK. Gravity test work was completed by FLS Knelson in Canada and the tailings and cyanide destruction test work was latest carried out by SGS Mineral Services in Canada. A new metallurgical test work program was initiated end of 2021 to complete confirmatory test work, but the results were not available at the time of writing this technical report.

The average annual plant head grade is 1.32 g Au/t for granitic ore (main source of ore), 1.03 g Au/t for saprolite, and 1.11 g Au/t for garimpeiros tailings. The combined average annual plant feed grade is 1.31 g Au/t with a maximum peak of 1.70 g Au/t in Year 7.

There are two types of gold association with sulfide minerals; the first association occurs with pyrite, while the second association exists with pyrite, chalcopyrite, galena and sphalerite.

From a comminution point of view, granite ore samples can be characterized as:

- Medium to hard ore in terms of crushing, with work index varying from 10.1 to 15.5 kWh/t.
- Moderately soft to medium hardness ore with respect to resistance to impact breakage (A x b) based on SMC test results. The (A x b) value varies from 51.5 to 59.3.
- Hard ore with respect to ball milling, with work index varying from 16.8 to 18.5 kWh/t.
- Highly abrasive ore, with abrasion index varying from 0.418 to 0.717 g.

The gravity recovery test work was performed using a Knelson concentrator which obtained moderate high recoveries for gold and was therefore included in the process design.

Batch, lock cycle and pilot plant flotation test work demonstrated the following:

- A two-stage (rougher/scavenger and cleaner) flotation circuit is the optimal circuit to produce a gold concentrate for subsequent cyanidation leaching.
- A circuit feed size of P80 of 125 µm results in the optimum flotation gold recovery.
- Cleaner mass pull for 4.5% for granite, 7.7% for saprolite and 2.8% for garimpeiros tailings.
- SIBX collector, DF-250 frother and copper sulphate as a promoter are required flotation reagents.

Overall concentrate leaching test work results demonstrated the following:

- Leaching at concentrate (flotation feed size; P80 = 125 µm) achieved recoveries in excess of 94%.
- Re-grinding or leaching at a finer particle size (P80 = 85 µm) improved recoveries (~97%).
- Leaching at increased cyanide concentrations (2 g/L vs 5 g/L) did not display improved gold recoveries but increased cyanide consumption.
- Leaching kinetics began to plateau after 32 hours.

Cyanide destruction test work at SGS was able to successfully use the SO_2 / Air process to reduce the total cyanide concentration to below target (< 0.2 mg/L CN_{WAD}) and less than 0.2 mg/L CN_{TOTAL} (limit is 1.0 mg/L CN_{TOTAL}) after aging.

Gold balances were completed, and gold overall recoveries were estimated for granite, saprolite and garimpeiros tailings feed material based on historical metallurgical test results as follows.

Table 1.1: Gold Recoveries

Feed Material	Feed Grade g Au/t	Gravity Stage Recovery	Flotation Stage Recovery	CIL Stage Recovery	Overall Recovery
Granite	1.32	24%	93% 4.5% mass pull	95%	90.9%
Saprolite	1.03	14%	71% 7.7% mass pull	93%	70.8%
Garimpeiros Tailings	1.11	14%	86.4% 2.8% mass	96%	85.4%

1.12 Mineral Resource Estimate

SRK was commissioned to audit a deposit (rock) mineral resource model prepared by Eldorado (2019) and a tailings mineral resource model prepared by GMS (2021) in terms of international mineral resource estimation and reporting guidelines and to assume independent Qualified Person responsibility for these mineral resource models in this study. The reported Mineral Resource Statement represents the third mineral resource evaluation for the Tocantinzinho Gold Project in accordance with the Canadian Securities Administrators' National Instrument 43-101.

The deposit mineral resource model considers 78 core boreholes (22,134 m) drilled during February 2004 to September 2008 by Brazauro and 74 core boreholes (22,030 m) drilled during September 2008 to December 2010 by Eldorado. In addition, some 155 tailing boreholes (1,594.04 m) drilled by Eldorado in 2011 and 2014 were considered for the tailings mineral resource model.

The SRK audit involved the review of the following aspects of the Tocantinzinho resource models:

- Database compilation and verification.
- Construction of wireframe models for the boundaries of the gold mineralization.
- Definition of resource domains.
- Data conditioning (capping and compositing) for geostatistical analysis and variography.
- Grade interpolation in a 3D block model.
- Model validation and resource classification.
- Assessment of "reasonable prospects for eventual economic extraction" and selection of appropriate cut-off grades.
- Preparation of the Mineral Resource Statement.

The Mineral Resource Statement for saprolite, rock and tailings material tabulated in Table 1.2 was prepared by Camila Passos, P.Geo (APGO#2431). The overall process was reviewed by Dr. Oy Leuangthong, P.Eng (PEO#90563867) for rock material and by Glen Cole, P.Geo (APGO#1416) for the tailings material. Ms. Passos is an independent Qualified Person as this term is defined in National Instrument 43-101. The effective date of the Mineral Resource Statement is December 10, 2021.

Table 1.2: Mineral Resource Statement*, Tocantinzinho Gold Project, Brazil, SRK Consulting (Canada) Inc., December 10, 2021

Domain	Classification	Cut-off Grade Au (g/t)	Quantity Tonnes (000's)	Grade Gold (g/t)	Contained Metal Gold Ounces (000's)
	Measured				
Saprolite		0.30	375	1.40	17
Rock		0.30	17,234	1.49	824
	Total Measured	0.30	17,609	1.49	841
	Indicated				
Saprolite		0.30	1,085	1.01	35
Rock		0.30	27,988	1.31	1,176
	Total Indicated	0.30	29,073	1.30	1,211
	M + I				
Saprolite		0.30	1,460	1.11	52
Rock		0.30	45,222	1.38	2,000
	Total M + I	**0.30**	**46,682**	**1.37**	**2,052**
	Inferred				
Saprolite		0.30	59	0.66	1
Rock		0.30	732	0.92	22
	Total Inferred	0.30	791	0.90	23
Tailings**	Measured	0.30	-	-	-
	Indicated	0.30	1,432	1.10	50
	Total M+I	**0.30**	**1,432**	**1.10**	**50**
	Inferred	0.30	789	1.07	27

** Mineral resources are not mineral reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimates. Assays were capped where appropriate. Open pit mineral resources are reported at a cut-off grade of 0.30 g/t gold. The cut-off grades are based on a gold price of USD 1,600 per troy ounce and metallurgical recoveries of 78% and 90% for gold in saprolite rock respectively.*
*** Mineral resources are not mineral reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimates. Assays were capped where appropriate. Open pit mineral resources are reported at a cut-off grade of 0.30 g/t gold. The cut-off grades are based on a gold price of USD 1,600 per troy ounce and metallurgical recoveries of 82% for gold.*

In the opinion of the SRK qualified person, the resource evaluation reported herein is a reasonable representation of the global gold mineral resources found in the Tocantinzinho Gold Project at the current level of sampling. The mineral resources have been estimated in conformity with generally accepted CIM

"Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines and are reported in accordance with the Canadian Securities Administrators' National Instrument 43-101. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve.

1.13 Mineral Reserve Estimate

The Proven and Probable Ore Reserve for the TZ Project is estimated at 48.7 Mt at an average grade of 1.31 g/t of gold for 2,042 koz of contained gold as summarized in Table 15.1. The contained gold in the proven category represents 41% of the total ore reserve estimate.

Table 1.3: Tocantinzinho Project Ore Reserve Estimate (December 10, 2021)

Mineral Reserve by Category	Tonnage kt	Grade G Au/t	Contained Gold koz
Proven	17,973	1.46	842
Probable	30,703	1.22	1,200
Proven and Probable	**48,676**	**1.31**	**2,042**

Notes:
1. *CIM definitions were followed for mineral reserves.*
2. *Effective date of the estimate is December 10, 2021.*
3. *Mineral reserves are estimated for a gold price of USD 1,400/oz.*
4. *Mineral reserve cut-off grade is 0.36 g Au/t for all materials.*
5. *A dilution skin width of 1 m was considered resulting in an average mining dilution of 5.5%.*
6. *Bulk density of ore is variable with an average of 2.67 t/m^3.*
7. *The average strip ratio is 3.36:1.*
8. *Numbers may not add due to rounding.*

The open pit mine design and ore reserve estimate have been prepared by GMS to a level appropriate for a Feasibility Study. The mineral reserve stated herein is consistent with the CIM definitions and is suitable for public reporting. As such, the mineral reserves are based solely on measured and indicated mineral resources with applicable modifying factors and therefore exclude any inferred mineral resources. The inferred mineral resources contained within the mine design are classified as waste for reporting purposes.

The parameters used for optimization were updated from previous work done on the TZ Project as well as benchmarking on similar projects. A long-term metal price assumption of USD 1,400/oz was used for calculating cut-off grades and estimating ore reserves.

A feasibility level pit slope design study was carried out by Golder Associates. The conclusions of this study have been used as an input to the pit optimization and design process.

1.14 Mining Methods

Mining will be carried out using conventional open pit techniques with 10 m benches. An Owner mined open pit operation is planned with hydraulic shovels and mining trucks, including outsourcing of certain support activities such as blasting.

Production drilling of the 10 m benches will be by blast hole drill rigs with both rotary and down-the-hole ("DTH") drilling capability. Blast holes are loaded with bulk emulsion. The majority of the loading in the pit will be carried out by two 16.5 m^3 hydraulic face shovels, one 17 m^3 hydraulic excavator, and one 10.5 m^3 front-end wheel loader. The shovels and loader will be matched with a fleet of 92.5 t payload mine trucks.

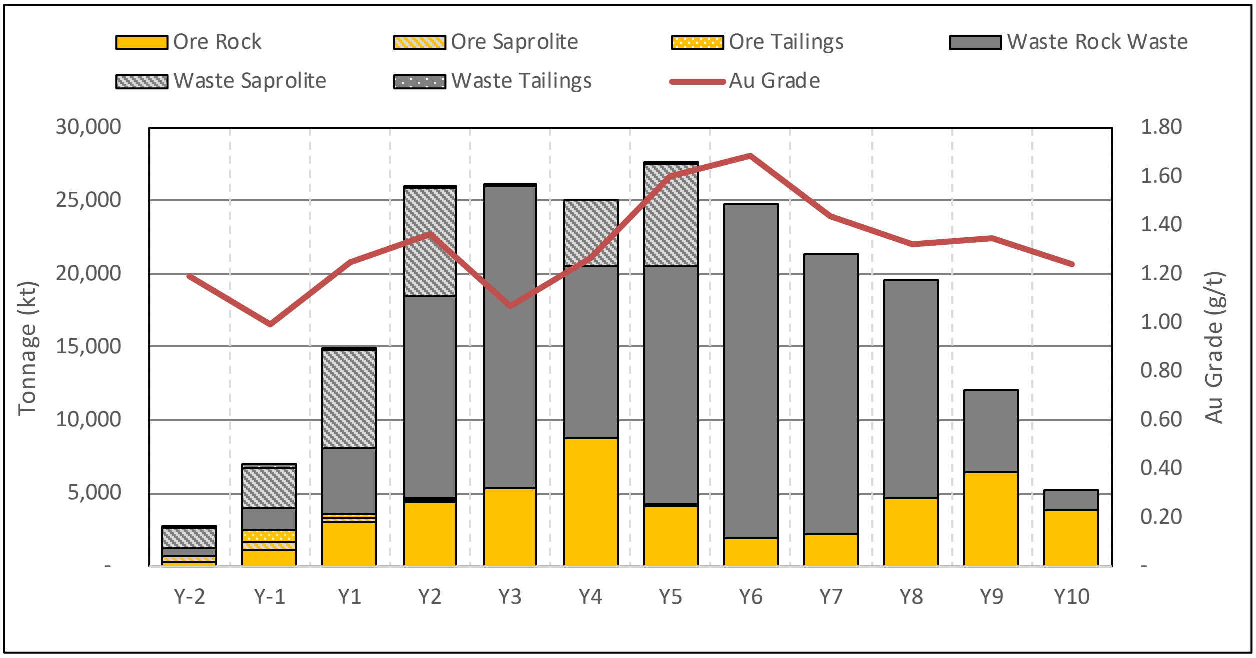
Mining of the Tocantinzinho main pit will occur in three main phases preceded by a starter pit. Waste rock will be disposed of in two distinct waste dumps. The main waste dump will be located near the pit and the other waste dump will be located downstream of the flotation tailings dam. This second dump is an option to increase long-term safety factor of the dam. The open pit generates 163.4 Mt of waste rock and 48.7 Mt of ore, inclusive of historic tailings, over the life of mine ("LOM") for an average LOM strip ratio of 3.36:1.

Mining activities are planned over a duration of 11 years which includes two (2) years of pre-production mining. Once the open pit is depleted and mining activities are stopped stockpile reclaim continues for another 1.5 years to continue feeding the mill. The mining rate reaches a peak of 28.3 Mtpy in Year 5 of commercial production. Figure 1.1 presents the mining schedule by material type and gold grade.

The mill schedule includes two (2) months of commissioning with ore with the second month planned to achieve 60% of nameplate throughput after which commercial production is achieved with 10.5 years of operation.

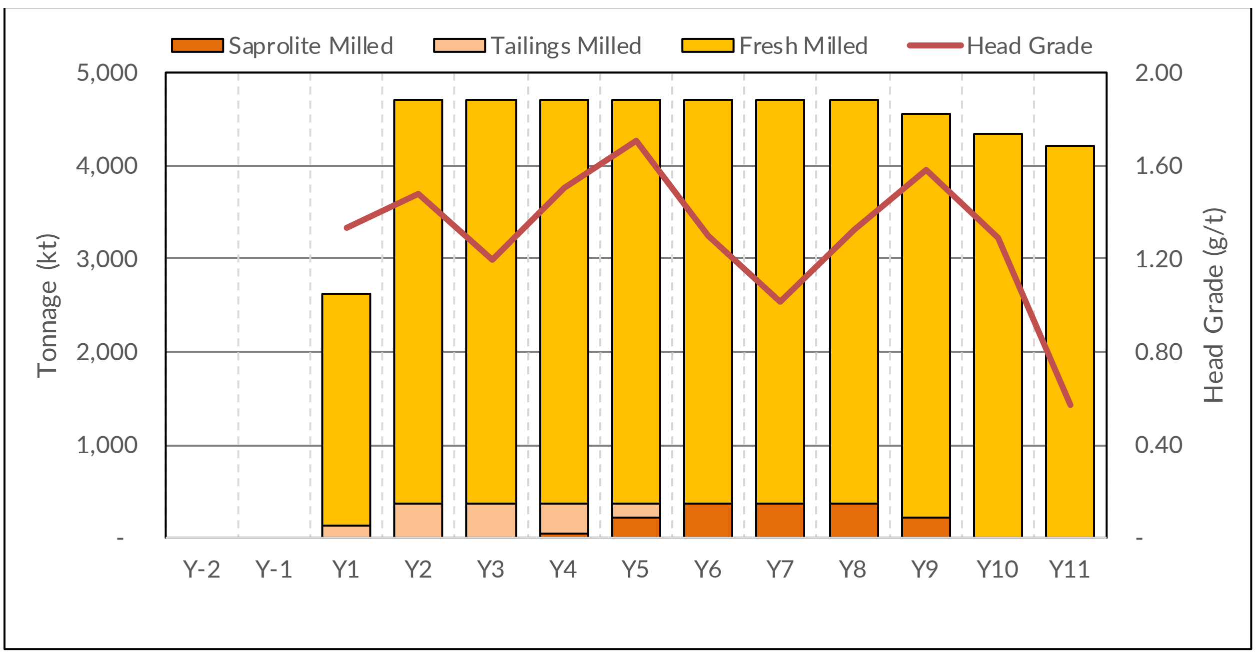
The peak milling capacity is 4,705 kt/y or 12,890 t/d of nominal throughput and is maintained for the first 7.5 years while softer saprolite and tailings material is available as "supplemental" mill feed at a rate of 1,000 t/d in addition to the fresh rock. Fresh rock represents 94% of the total mill feed with saprolite and tailings representing only 6%. Mill feed is maximized with direct feed from the pit and rehandled stockpiled material. Figure 1.2 presents the mill feed by rock type and corresponding grade.

Figure 1.1: Annual Mine Production



Source: GMS, 2021

Figure 1.2: Annual Mill Production



Source: GMS, 2021

1.15 Recovery Methods

The process plant is designed to treat a nominal throughput of 4.34 Mt/y of granite ore and up to 4.70 Mt/y when saprolite and garimpeiros tailings are available in the blend which is limited to 1,000 t/d. The process plant will consist of comminution, gravity concentration, gold flotation, cyanide leach and adsorption of the gold concentrate via carbon-in-leach (CIL), carbon elution and gold recovery circuits. CIL tailings will be treated in a cyanide destruction circuit and dewatered to produce a tailings slurry for storage onsite. The process plant feed will consist of run of mine (ROM) granite ore, along with minor amounts of saprolite and garimpeiros tailings. Figure 1.3 presents the overall flowsheet and Figure 1.4 presents the overall layout for the Tocantinzinho process plant.

The proposed process plant will consist of the following unit operations:

- Primary crushing of ROM.
- Coarse ore stockpile and reclaim.
- Grinding consisting of semi-autogenous (SAG) mill and ball mill with hydrocyclones producing a final product P80 of 125 µm.
- Gravity concentration to produce a gold-rich concentrate for intensive leaching and subsequent gold recovery via electrowinning.
- Two-stage flotation circuit with an average mass pull of 4.5% to produce sulphide concentrate for cyanide leaching.
- Pre-leach, cyanide leaching, and carbon adsorption via a CIL circuit with 36 hours residence time to achieve optimal gold extraction.
- Carbon elution via a 3t Pressure Zadra circuit.
- Carbon handling and regeneration.
- Electrowinning and smelting to produce doré.
- Cyanide destruction of CIL tailings using SO_2 / air process.
- Tailings – Flotation tailings and concentrate cyanidation tailings (i.e., CIL tailings) are stored in separate tailings storage facilities.
- Air and oxygen circuits.
- Water systems (potable water, raw water, gland seal water and process water).

Figure 1.3: Process Plant Flowsheet

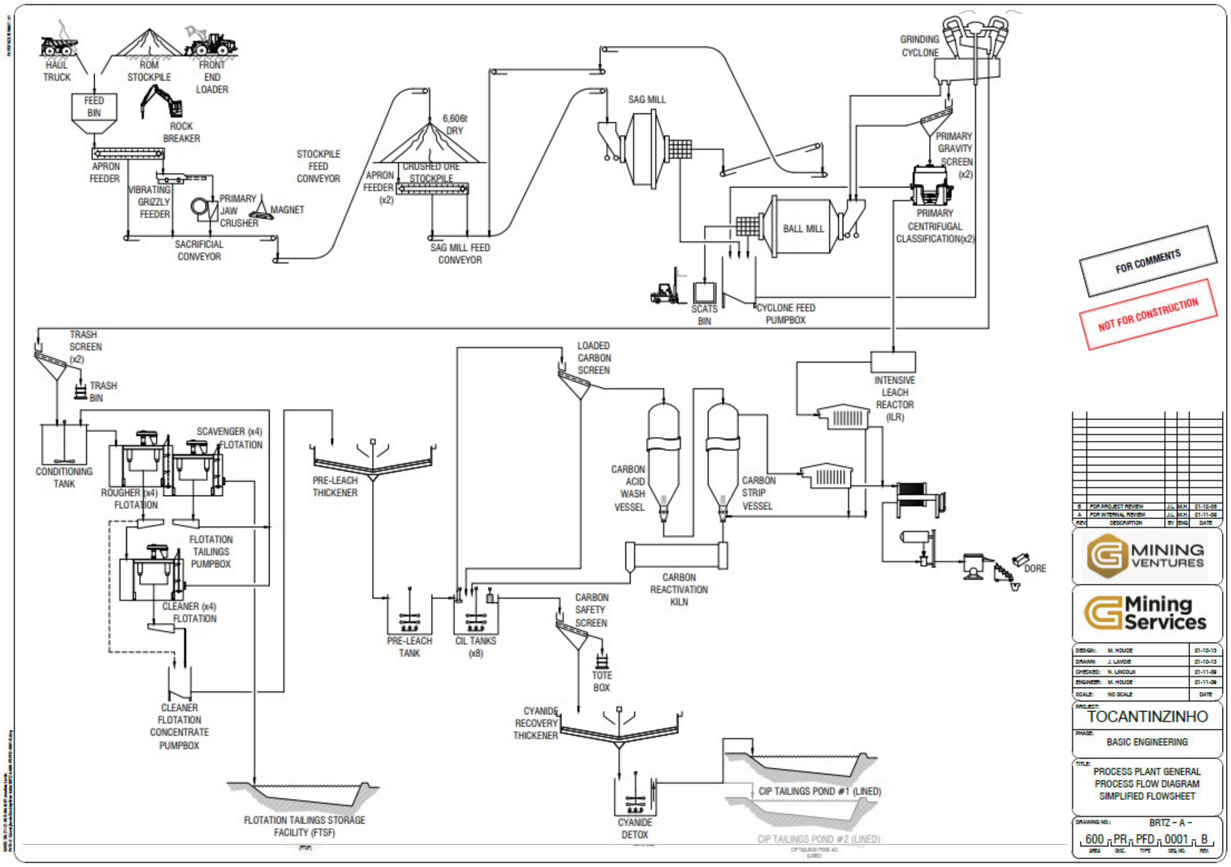
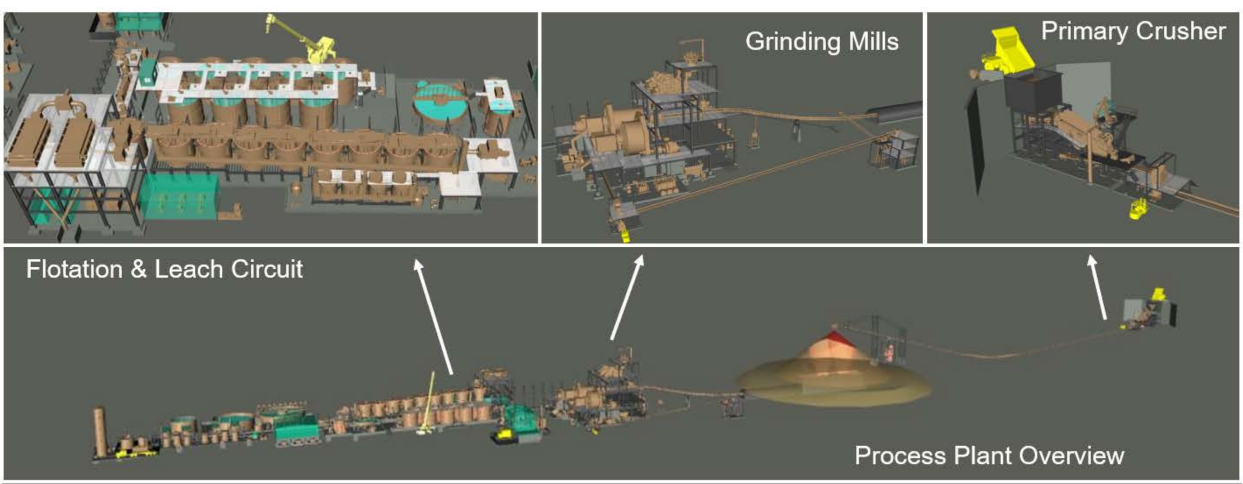


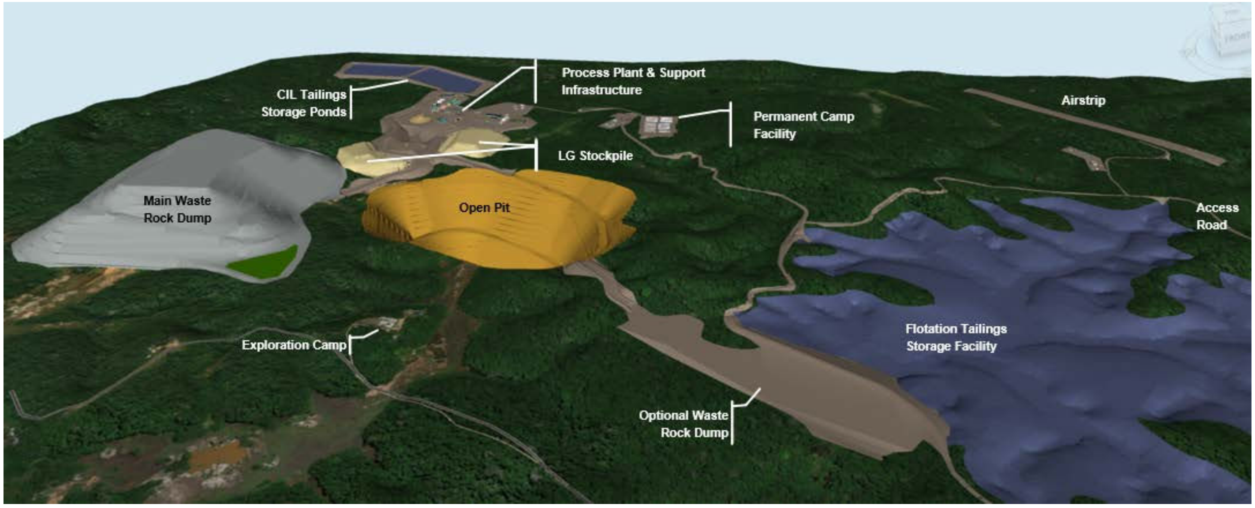
Figure 1.4: Process Plant Design Overview



1.16 Project Infrastructure

The site layout has been optimized to reduce footprint and make the site as compact as possible.

Figure 1.5: Tocantinzinho Project Site 3D View



The Project requires several infrastructures to support mining and processing operations (Figure 1.5) as summarized below.

- Access
  - Road access is via an existing 108km dirt road connecting the Project to the BR-163 Federal highway in Moraes Almeida. Upgrades to certain segments are planned to improve gradients, surfacing and drainage. An existing radio communication system with four repeater towers allows for continuous communications along the access road.
  - A gate and guardhouse will be located at the property entrance along the access road.
  - Service roads of 14.8 km will connect the various infrastructures located on the property, notably the airstrip, explosives storage facility, tailings storage facilities, operations site and camp site.
  - Air access is via an existing 775 m long airstrip that will be used during construction. A new 1,300 m long airstrip, rated category 2, capable of landing larger planes is planned. This airstrip will be used for personnel, supplies, medical emergencies and exporting gold.

- Power supply and distribution
  - Primary power supply is via a 200 km long 138 kV transmission line from the switching station in Novo Progresso to the substation at the mine site near the process plant. Average power consumption is estimated at 21 MW with a peak requirement of 25 MW.
  - The main substation will consist of two 20/25 MVA 138-13.8 kV transformers.
  - Site power distribution is planned with two 13.8 kV overhead lines.
  - Secondary power supply (i.e., emergency power) consists of 6.2 MW from four generators (3x 1.8 MW and 1x 0.8 MW).

- Process plant support buildings
  - Mill office (500 m^2) for plant staff and will be located within the fenced off area of the process plant. A process plant search house will be located at the mill entrance to access this secured area.
  - The reagent storage building (1,300 m^2) will have ample storage capacity sized according to supply chain considerations.

- Mine site
  - The mine includes a main open pit with two waste storage facilities to store a total of 163.4 Mt of material and an ore stockpile area to contain up to 8.9 Mt of ore.
  - Mine haul roads, 4.2 km in total, connect the open pit to the dumps, FTSF, leach ponds and mine support facilities. The open pit will be a source of waste fill material for various construction activities.

- Mine support infrastructures
  - Facilities located south-east of process plant and allow easy access for heavy equipment while a safe traffic is ensured by segregation of light and heavy traffic.
  - Permanent mine maintenance facility will have five heavy duty bays serviced by 30/5 t overhead crane with an additional two light duty bays. The maintenance facility will include office space for the maintenance staff, tool crib and lube storage.
  - Temporary maintenance facility (475 m^2) to be constructed at the onset will have 4 bays. This facility will be built with containers and a fabric top to be used during construction and subsequently repurposed as a welding bay.
  - Warehouse (815 m^2 area) close to the maintenance facility will be used to store parts and supplies.
  - Fuel storage will have a total capacity of 420 kL equivalent to 12 days of consumption.
  - 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.
  - Explosive storage facility is designed for a capacity of 160 t of emulsion using 40 t skid mounted tanks, 18 t of explosives products in a magazine with another magazine for accessories. Storage capacity is sufficient for 30 days at peak consumption.
  - Assay laboratory (784 m^2) is configured to process up to 350 samples per day for mine grade control, exploration and metallurgical samples.

- Administration and general infrastructure
  - Administration complex (1,700 m^2) for mine management, engineering, geology and G&A functions. Included in this complex is a clinic and security office to monitor all cameras on the property.
  - Communications will be provided by an existing network of interconnected telecom towers with the bandwidth to be increased with demand.
  - A greenhouse and nursery (200 m^2) will be established to cultivate plants for future revegetation activities.
  - A recycling and sorting facility will be set up to sort waste materials. Inert solid waste will be disposed in a landfill.

- Camp facilities
  - Temporary camp (i.e., for exploration and construction) will be expanded from the current 100 beds to 200 beds. This camp facility consists of soft-shell buildings and includes kitchen and dining hall with required services such as power, water and sewage. The temporary camp will serve during early works and peak requirements during construction.
  - Permanent camp facility will have a 1,200-person capacity during construction with three types of camp modules. The camp will easily service the approximate 800-person capacity during operations. The permanent camp facility includes kitchen and dining area for 320 people, recreation facilities, camp office and laundry facilities with associated water and sewage services.

- Tailings and water management
  - Flotation tailings storage facility for inert material will require one main dam. The starter dam has a height of 29 m and provides storage capacity for three years. Subsequent raises to the final dam height of 44 m will provide a storage capacity of 29.3 Mm^3. A total volume of 1.49 Mm^3 of fill material consisting of a saprolite core and compacted rockfill on the downstream slope is required to construct the main dam. A small saddle dyke will be required for the final facility.
  - A reclaim water barge will be installed in the FTSF to recycle water back to the process plant or for discharge to the environment at a rate of 401 m^3/h.
  - The leach tailings will report to two ponds lined with HDPE geomembrane. The first pond will be built as part of the initial project and the second pond during the first year of operations for water management purposes.

- Effluent treatment plant for excess water from the leach tailings ponds to be constructed during the first year of operations. This facility will have a capacity of 100 m^3/h principally to treat for dissolved copper by hydroxide precipitation via lime addition.
- A vertical pumping station in Veados Creek will provide up to 200 m^3/h of freshwater. An industrial tank will provision 108 m^3 capacity in the bottom for fire water storage. Water from Veados Creek will be treated for industrial consumption with further treatment for potable water.

1.17 Environmental Studies, Permitting, and Social

1.17.1 Environmental Studies

In 2011, Brazauro completed an Environmental Impact Assessment ("EIA"), carried out by Brandt Amazônia, with the objective of obtaining its first Preliminary License for the Project, in accordance with the legal requirements in Brazil for permitting purposes.

Environmental baseline studies were completed on the Project including flora and fauna studies, hydrology and hydrogeology monitoring and studies, archeological surveys, geochemistry analysis and geotechnical studies.

Systematic monitoring of hydrological, and hydrogeological variables was executed within the Project site and the monitored data provided background information on several variables of interest for the Project. The quantity and quality of the data obtained made it possible to establish several correlations between rainfall, runoff flows and water levels.

1.17.2 Geochemistry

The chemical species formed by the weathering of rocky materials are largely stored in sediments and soils. Due to the wide artisanal mining activity in the past, extensive sampling of soils and sediments were completed to determine the concentrations of chemical parameters and to establish geochemical background for the site. The information also provided data for the evaluation of possible environmental interferences resulting from future mining activity in the area.

Brazauro conducted two campaigns of static testing to evaluate the potential for acid rock drainage ("ARD") and metal leaching ("ML") of its ore and waste materials. The first campaign was carried out in 2010 by the company VOGBR and the second campaign by CLAM Engenharia Hidrocnese ("CLAM") in 2012, which report consolidated both campaigns. The results of the metal leaching tests in both campaigns showed that there is no release of any excess elements to the limits established in Annex F to ABNT NBR 10004:2004, which allowed to conclude that the materials do not fall within the category of hazardous waste (class I A). The analysis of the results of static tests obtained from both campaigns showed that the waste materials of the Project, especially quartz-monzonite, have high PN/PA ratios due to the presence of carbonate in all materials and consequently have low to no potential for acid generation.

The process plant tailings were not available for ARD potential testing, only for ML. The ML tests performed on the flotation tailings by SGS Geosol Laboratories in 2016 according to the standards of ABNT NBR 10004, determined that the flotation tailings samples were classified as Class II B (Not hazardous - Inert).

Water Management

The Project is within an area with hydrological surplus, resulting from annual precipitation of 2,248 mm and average pan evaporation of 834 mm as measured at Itaituba from 2006 to 2020.

The project has an estimated raw water demand of 200 m^3/h for the process plant and multiple industrial uses. CLAM's report suggests that the Veados creek has enough water availability to provide the 200 m^3/h. Veados creek has a total available flow of 387 m^3/h, almost double the project demand, with limited upstream or downstream potential usage given the remote location. Water from the creek will be filtered before being stored and distributed as domestic water throughout the plant and camp. Some of the domestic water will be further treated to qualify as potable water. To maximize water conversation, process water required for the process plant will be recirculated from the flotation tailings storage facility and process water for cyanide destruction will use decanted water from the CIL tailings ponds.

During construction, an initial dam will be built to create a reservoir to store flotation tailings discharged from the process plant. The reservoir (FTSF) will initially store sufficient rainwater to enable the start of process plant operations. Tailings will be initially spigotted on the upstream face of the main dam as per best practice to keep decant water away from the dam and improve the dry density of deposited tailings. A barge holding sufficient pumping capacity will be located in a bay northeast of the main dam to optimize tailings sedimentation; the pumping arrangement will be designed to recycle water to the process plant and discharge surplus water in the environment once operations have reached steady state.

Some low drainage basins will receive contact water from the mine operations. It is planned to line all slopes with waste rock from pit operations to minimize erosion and addition of solids in suspension to the existing drainage. If required, small dams with spillways may be required to allow sedimentation of solids before discharge in the environment.

The project considers two tailings streams from the process plant: tailings from the flotation circuit and tailings from the leach/CIL circuit.

1.17.3 Tailings Management

The flotation tailings storage facility ("FTSF") will receive approximately 95% of the tailings from the process plant and will require the construction of a main dam that will be phased over the life of the Project as described earlier. Given the tailings are classified as non-hazardous and inert, there is no requirement for a liner system. The effluent from the FTSF will be discharged without any treatment since the tailings are inert. Sufficient settling time will ensure respect of the solids in suspension criteria. The FTSF will have a total volume capacity of 29.8M m^3. If additional storage was required due to increase in reserves, it would be possible to increase the height of the main dam and add saddle berms or used the mined-out pit for storage.

The CIL circuit tailings representing the remaining 5% of the process plant tailings will be stored in a separate storage facility ("CTSF") which will consist of containment provided by two ponds. The CIL ponds #1 and #2 are designed with storage capacity for the life-of-mine solid tailings from cyanide leaching and CIL gold recovery from the gold concentrates. The effluents from the CIL circuit will be treated in a cyanide detoxification circuit using the conventional SO_2/air process before deposition in the ponds. The CIL Pond #2 is planned to be constructed in Year 1 of the mine life for use in Year 2. The two ponds will have a total capacity of 1.58M m^3. To avoid contamination of the groundwater the whole internal face and the bottom of the pond will be lined with a layer of geomembrane liner to guarantee impervious ponds

Cyanide destruction testing by SGS Canada in 2017 of the Leach and CIL tailings confirmed that the tailings are classified as hazardous and potentially acid generating because of the high sulphide content. A minimum one meter of water will be maintained above the deposited tailings to control any oxidation of the tailings. Analysis of the quality of the water after the cyanide destruction process was obtained from laboratory testing. Based on two-stage cyanide destruction test work at 40% solids, the detox product achieved the following results for water contaminants:

- 0.41 mg/L CN_{TOTAL} (below 1 mg/L effluent limit)
- 0.2 mg/L CN_{WAD} (on effluent limit of 0.2 mg/L)
- 4.28 mg/L Cu (above 1 mg/L effluent limit)
- <0.05 mg/L Fe
- 23.9 mg/L N (above 20 mg/L effluent limit for total ammoniacal nitrogen as N)

It is currently assumed that dilution from rainwater, natural degradation and volatilization will bring the cyanide and ammonia concentrations in the CIL tailings ponds into compliance with discharge criteria. The only parameter for which treatment should clearly be anticipated is copper. No removal of dissolved copper is expected as a result of aging in the ponds. It is currently assumed that it will be necessary to treat and remove copper by hydroxide precipitation via lime addition. The resulting solids will be recovered by ballasted clarification.

All decant water will be contained in the two ponds for the first two years of operations. This provides enough time to assess water quality and adjust the design of the effluent treatment in the first year of operations, construct the facility and put it in use for the third year of operations.

1.17.4 Permitting Process

In parallel with securing mining concessions and undertaking development and mining construction activities, environmental licenses are required.

The Brazilian National Environmental Policy (Federal Law No. 6.938/1981) requires that all potentially or effectively polluting activities be subject to the environmental licensing process. Applicable rules regarding the licensing procedure were established by Resolution No. 237 of the National Council for Environment ("CONAMA") dated December 19, 1997. The Federal Complementary Law No. 140/2011, in turn, describes the criteria establishing jurisdiction for environmental licensing by the union, the states, the federal district and the municipalities.

By means of the licensing process (which licenses will be issued by the competent environmental authority), the issuing agency determines the conditions, limits, and measures for control and use of natural resources, and allows the installation and implementation of an activity. Usually, the environmental licensing process follows three steps:

1. a Preliminary License ("LP"), granted during the preliminary stage of planning the facility or activity, which approves the location and the project conception.
2. an Installation License ("LI"), authorizing the facility or activity setting up in accordance with approved plans, programs and designs. and
3. an Operation License ("LO"), authorizing the operation of the facility or activity, after actual compliance with the prior licenses.

In 2011, Brazauro completed its EIA, carried out by Brandt Amazônia, with the objective of obtaining its initial LP for the Project. That EIA was approved in September 2012, with the granting of the relevant LP, which covers two main structures: the site, including activities related to mining and ore processing and the access road to the Project. During public hearing, discussions were focused on employment of the local population and opportunities for local businesses.

In January 2016, an Installation License ("LI") for the Project was requested, was subsequently granted in April 2017 with additional modifications granted in August 2017

1.17.5 Environmental Compensation

Conservation Units ("UCs") are specially protected areas created and managed by federal, state, or municipal governments with relevant natural characteristics legally instituted by the relevant authority, with conservation purposes and defined limits, under a special administration regime.

Applicable law requires BRM to compensate unmitigated negative impacts identified during the environmental licensing proceeding.

BRM signed terms of commitment with IDEFLOR-Bio and with ICMBio (environmental authorities responsible for the management of UCs at the state and federal levels, respectively) for environmental compensation. The total compensation amount is BRL 9,720,456.06 (historical value) representing 0,9721% of the estimated costs to implement the Project. From this total amount only BRL 972,047 remain to be paid in October 2022.

1.17.6 Future Community Engagement

Several social programs relating to community development were created and all of them are scheduled to be implemented during the Project installation. They include:

- Social communication program and relationship with stakeholders in the Project that includes a plan for development and implementation of the Project's website. The main objective is to contribute to the strengthening of the social dialogue between the community and Brazauro, to give greater support to all activities that involve execution.
- Local development promotion program, which includes a rural economy promotion program.
- Training, qualification, and improvement of the workforce, which includes an action plan for labour mobilization.
- Occupational health and safety program.
- Public management support program.
- Environmental education program.

1.17.7 Closure and Reclamation

The closure plan was established to identify environmental, social and economical risks after production will terminate and to determine measures to be implemented during construction, operation and closure. It will be continuously updated and implemented prior to the shutdown of the Project's operations.

The current closure strategy is as follows:

- Open Pit: equipment and infrastructure will be removed from the pit which will fill with water. All mine-influenced water that is not suitable for discharge to the environment will be treated. Testing and studies will be carried out to predict the future water quality in the pit.
- Waste Rock Pile: the pile was designed with gentle slope angles so no further sloping will be required to accommodate topsoil placement. Rock capping of the slopes will minimize erosion. Tests for acid rock drainage prediction (ARD) were completed and most of the tests of waste have shown elevated PN/PA ratio in excess of 2 and mostly in excess of four and are considered non-acid rock generating. This can be explained by the significant carbonate content in the ore and waste rocks.
- Tailings Storage Facility: Depending on the results of further geochemical investigations, a permanent wet cover could be implemented in the CTSF to avoid ARD; otherwise an impermeable soil cover will be placed.

  For the FTSF, vegetation will be implemented and drainage through a permanent spillway will control water accumulation.
- Process Plant, Camp, Onsite Infrastructure, Onsite Roads, Onsite Power Line: concepts for closure will depend on future land use. Equipment will be evaluated for potential reuse. Non-reusable equipment and metallic structures will be segregated from other materials to be sold as scrap. Hazardous waste generated during demolition will be segregated and disposed of properly. The areas will be reclaimed by revegetation of native species.
- Monitoring and Maintenance: Monitoring and maintenance will be necessary in the post-closure period to ensure proper revegetation and repair any erosion that may occur, if applicable.

1.18 Capital Cost Estimate

The capital cost estimate is according to AACEI standard Class 3 and is accurate to a -10% / +15% range. The base date of the CAPEX estimate is Q1-2022. The initial capital expenditure ("CAPEX") duration is planned over a period of 29 months, assumed from February 2022 to end of June 2024. The initial CAPEX estimate is aligned with an owner-managed project delivery model. Expenditures are planned in several currencies with the native currencies retained as part of the estimate. The initial CAPEX estimated is presented in US dollars using an exchanges rate of 5.20 BRL/USD

The initial CAPEX estimate was based on material take-offs from feasibility level designs for all aspects of the Project. The mine mobile equipment fleet is based on firm pricing with certain units the subject of purchase orders. The remaining equipment and material costs are based on budgetary bid processes, quotes, consultant's historical data and in-house databases or benchmarked from previous projects. Labour unit rates were developed from first principles based on budgetary quotations and direct installation hours based on a combination of firm price proposals, budgetary quotes and feasibility study estimates, benchmarked against previous projects and reviewed by experienced construction personnel.

A tax analysis was performed by L&M Advisory, a tax specialist in Brazil, and consisted of the following:

- Operating costs ("OPEX") and initial CAPEX: Analysis and application of tax incidences on operating cost items; ICMS balance projection highlighting OPEX and initial CAPEX credits and tax balances by period over the life of the project; simulation of the Drawback tax benefit.
- CAPEX: Tax review of the main commercial proposals, calculation of factors and application of tax incidences on initial CAPEX items. This step consisted of a complete analysis of taxation including the basic incidence, tax calculations, application of benefits and tax compensation provided for in the law.

The initial CAPEX is estimated at USD 457.8M net of recoverable taxes and tax credits of $17.9M with approximately 59% planned to be spent in local BRL currency. The capital expenditure is summarized in Table 1.4 according to the Level 1 work breakdown structure ("WBS"). This amount includes pre-production revenues of approximately USD 5.5M for 3.79 koz of gold recovered during commissioning.

The CAPEX includes a contingency of USD 38.3M, which is 9.1% of the total before contingency or 10.3% of the total excluding major mining equipment.

Table 1.4: Initial Capital Expenditure Estimate

Capital Expenditures (k USD)	Total Pre-Tax	Total Taxes Payable	Total Post-Tax	Recoverable Taxes and Tax Credits	Total Net of RT and Credits
100 – Infrastructure	38,472	5,263	**43,735**	2,822	**40,913**
200 - Power and Electrical	57,666	8,575	**66,241**	4,382	**61,859**
300 – Water Management	12,216	1,750	**13,966**	902	**13,064**
400 – Surface Operations	10,695	1,739	**12,434**	407	**12,026**
500 - Mining	42,811	4,403	**47,214**	189	**47,025**
600 - Process Plant	78,530	10,067	**88,597**	2,561	**86,036**
700 - Construction Indirect	52,676	5,319	**57,995**	2,640	**55,355**
800 - General Services / Owner's Cost	54,611	3,568	**58,179**	1,193	**56,986**
900 - Pre-production, Start-up, Comm.	41,074	8,020	**49,094**	2,808	**46,286**
990 - Contingency	38,295	0	**38,295**	0	**38,295**
Total	**427,046**	**48,704**	**475,750**	**17,904**	**457,845**

Sustaining capital is presented in Table 1.5. Sustaining capital for the mine includes additional equipment purchases for a total of USD 50.0M. Major equipment repairs were kept in the operating costs. Additional work is required for raising the main embankment of the flotation tailings storage facility and the construction of the second pond as part of the CIL tailings storage facility. The continued raising of the FTSF will be completed by the mine operations team with fill material from the open pit mine. An effluent water treatment plant will be constructed in Year 2 to treat water from the CTSF prior to discharge. Tailings and water management sustaining capital is estimated at USD 16.7M.

The total salvage value is estimated at USD 12.6M and includes mining equipment purchased during operations that will not have been utilized to its useful life and a residual value for some of the process plant major equipment.

Reclamation and closure costs include infrastructure decommissioning, site preparation and revegetation, maintenance, and post closure monitoring. The reclamation costs begin before commercial operations end when the mine will be depleted, and low-grade stockpile will be reclaimed and will continue for four years after. The total reclamation and closure cost is estimated to be USD 23.5M.

Table 1.5: Sustaining Capital Estimate

Cost by Area (k USD)	Years	1	2	3	4	5	6	7	8	9	10	11
Mobile Equipment	**50,042**	**19,531**	**10,994**	**5,350**	**1,133**	**1,122**	**3,851**	**2,114**	**5,146**	**724**	**65**	**12**
Primary	**29,222**	12,470	10,020	5,049	-	-	1,683	-	-	-	-	-
Secondary	**7,603**	1,044	-	-	768	588	1,044	-	3,774	384	-	-
Ancillary	**3,523**	1,052	-	-	-	-	-	2,001	471	-	-	-
Others	**5,512**	1,942	216	188	365	284	1,086	113	901	340	65	12
Fleet Mgmt. System	**4,181**	3,023	758	113	-	250	38	-	-	-	-	-
Tailings & Water Mgmt	**16,659**	**3,554**	**7,648**	**2,351**	**-**	**3,106**	**-**	**-**	**-**	**-**	**-**	**-**
Deforestation	**2,587**	-	2,587	-	-	-	-	-	-	-	-	-
Tailings FTSF	**6,246**	-	1,881	1,881	-	2,484	-	-	-	-	-	-
Tailings CTSF	**2,136**	2,136	-	-	-	-	-	-	-	-	-	-
Effluent Treatment Plant	**2,359**	708	1,651	-	-	-	-	-	-	-	-	-
Construction Indirects	**3,332**	711	1,530	470	-	621	-	-	-	-	-	-
Process Plant												
Process Plant Allowance	**4,683**	-	837	481	481	481	481	481	481	481	481	-
Taxes												
Non-recoverable Taxes	**11,535**	4,502	2,534	1,233	261	259	888	487	1,186	167	15	3
Total Incl. NRT	**82,919**	**27,588**	**22,013**	**9,415**	**1,875**	**4,967**	**5,220**	**3,082**	**6,813**	**1,372**	**561**	**14**

1.19 Operating Cost Estimate

OPEX are summarized in Table 1.6. The OPEX includes mining, processing, general services and administration ("G&A"), transportation and refining, and royalties. Power costs are included in processing costs. The average OPEX is USD 623/oz Au or USD 23.68/t milled over the LOM. The all-in sustaining cost ("AISC") which includes closure, reclamation and sustaining capital costs averages USD 681/oz Au or USD 25.88/t milled.

Table 1.6: Total Operating Costs Summary by Year

Operating Cost Summary	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13+
Production Highlights														
Tonnage milled (kt)	**48,284**	2,235	4,705	4,705	4,705	4,705	4,705	4,705	4,705	4,552	4,340	4,222		
Tonnage mined (kt)	**194,939**	7,475	25,918	26,000	25,000	27,500	24,782	21,387	19,625	12,000	5,253	-		
Recovered gold (koz)	**1,834**	93	203	163	206	233	175	137	180	209	163	70		
Operating Costs (M USD)														
Mining	**459.37**	17.05	47.41	57.56	50.55	56.68	56.23	57.61	53.32	37.47	19.88	5.61	-	-
Processing	**426.55**	19.65	41.16	41.16	41.16	41.16	41.16	41.16	41.16	40.49	39.55	38.75	-	-
G&A	**151.18**	7.89	15.74	15.71	15.73	15.74	15.74	15.76	15.49	14.32	10.26	8.80	-	-
Transport & Refining	**18.34**	0.93	2.03	1.63	2.06	2.33	1.75	1.37	1.80	2.09	1.63	0.70	-	-
Private Royalty	**43.75**	2.22	4.85	3.89	4.91	5.56	4.19	3.28	4.29	5.00	3.89	1.68	-	-
Govt. Royalty	**44.02**	2.24	4.88	3.92	4.94	5.59	4.21	3.30	4.32	5.03	3.92	1.69	-	-
Total Operating Cost	**1,143.21**	**49.98**	**116.05**	**123.87**	**119.35**	**127.05**	**123.28**	**122.48**	**120.37**	**104.40**	**79.13**	**57.24**	**-**	**-**
Sustaining Capital	**82.92**	27.59	22.01	9.42	1.87	4.97	5.22	3.08	6.81	1.37	0.56	0.01	-	-
Closure Cost	**23.53**	-	-	-	-	-	-	-	-	-	1.25	2.53	9.08	10.67
AISC	**1,249.66**	**77.57**	**138.07**	**133.29**	**121.22**	**132.02**	**128.50**	**125.57**	**127.19**	**105.77**	**80.94**	**59.78**	**9.08**	**10.67**
Unit Operating Costs														
Mining Cost / t mined	**2.36**	2.28	1.83	2.21	2.02	2.06	2.27	2.69	2.72	3.12	3.78	-		
Process Cost / t milled	**8.83**	8.79	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.89	9.11	9.18		
Total OPEX / t milled	**23.68**	22.36	24.67	26.33	25.37	27.00	26.20	26.03	25.58	22.93	18.23	13.56		
Total OPEX / oz	**623**	537	571	759	579	545	703	891	669	498	485	812		
AISC / oz	**681**	833	679	817	588	567	732	914	707	505	496	848		

1.20 Economic Analysis

The base case economic model has been developed using a long-term gold price assumption of USD 1,600/oz and an exchange rate of BRL/USD 5.20. The Project economic results are summarized in Table 1.7

Gold production over the LOM is 1,834 koz based on an average processing recovery of 90%. Gold production begins during the pre-production period (3.8 koz) and is treated as revenue partially offsetting pre-production costs.

The economic model excludes any Project debt financing but includes equipment financing. The Project funding is assumed to be through equity for the purposes of the Report. The economic results are calculated as of the start of the pre-production CAPEX phase at start of Q2 of Year -2 which includes the remaining detailed engineering and all procurement.

The before-tax Project cash flow over the Project life is estimated at USD 1,232M. The Project before-tax net present value ("NPV") at a discount rate of 5% is estimated to be USD 752M with a before-tax internal rate of return ("IRR") of 27.3%.

The total after-tax cash flow over the Project life is estimated to be USD 1,043M. The Project after-tax NPV at a discount rate of 5% is estimated to be USD 622M. The after-tax Project cash flow results in a 3.2-year payback period from the commencement of commercial operations with an after-tax IRR of 24.2%.

Table 1.7: Project Economic Results Summary

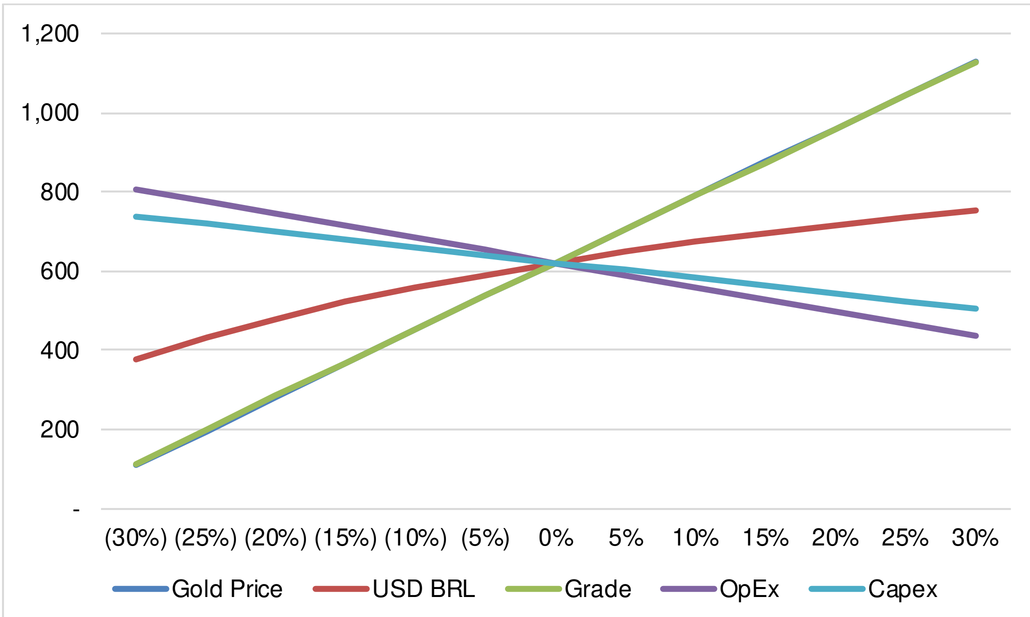
Project Economics Results		Base Case
Production Summary[1]		
Tonnage Mined	Mt	194.94
Ore Milled	Mt	48.28
Head Grade	g Au/t	1.31
Gold Processed	koz	2,036
Recovery	%	90%
Gold Production	koz	1,834
Cash Flow Summary		
Gross Revenue	M USD	2,935
Mining Costs (incl. rehandle)	M USD	(459)
Processing Costs	M USD	(427)
G&A Costs	M USD	(151)
Transport & Refining Costs	M USD	(18)
Royalty Costs	M USD	(88)
Total Operating Costs	M USD	(1,143)
Operating Cash Flow Before Taxes	M USD	1,792
Initial CAPEX[2]	M USD	(458)
Sustaining CAPEX	M USD	(83)
Closure Costs	M USD	(24)
Salvage Value	M USD	13
Total CAPEX	M USD	(564)
Royalty Buy-Back	M USD	(4)
Before-Tax Results		
Before-Tax Undiscounted Cash Flow	M USD	1,232
NPV 5% Before-Tax	M USD	752
Project Before-Tax Payback Period	years	3.1
Project Before-Tax IRR	%	27.3
After-Tax Results		
After-Tax Undiscounted Cash Flow	M USD	1,043
NPV 5% After-Tax	M USD	622
Project After-Tax Payback Period	years	3.2
Project After-Tax IRR	%	24.2

Note 1: Production period only, 2: USD 4.1M in tax credits netted from Iniital CAPEX.

A sensitivity analysis was performed for ±30% variations for gold price, exchange rate, grade, operating costs and initial capital expenditure Figure 1.6.

The Project is most sensitive to gold price and grade followed by exchange rate, initial capital costs and operating costs.

Figure 1.6: Sensitivity Analysis – After-Tax NPV 5%



Source: GMS (2022)

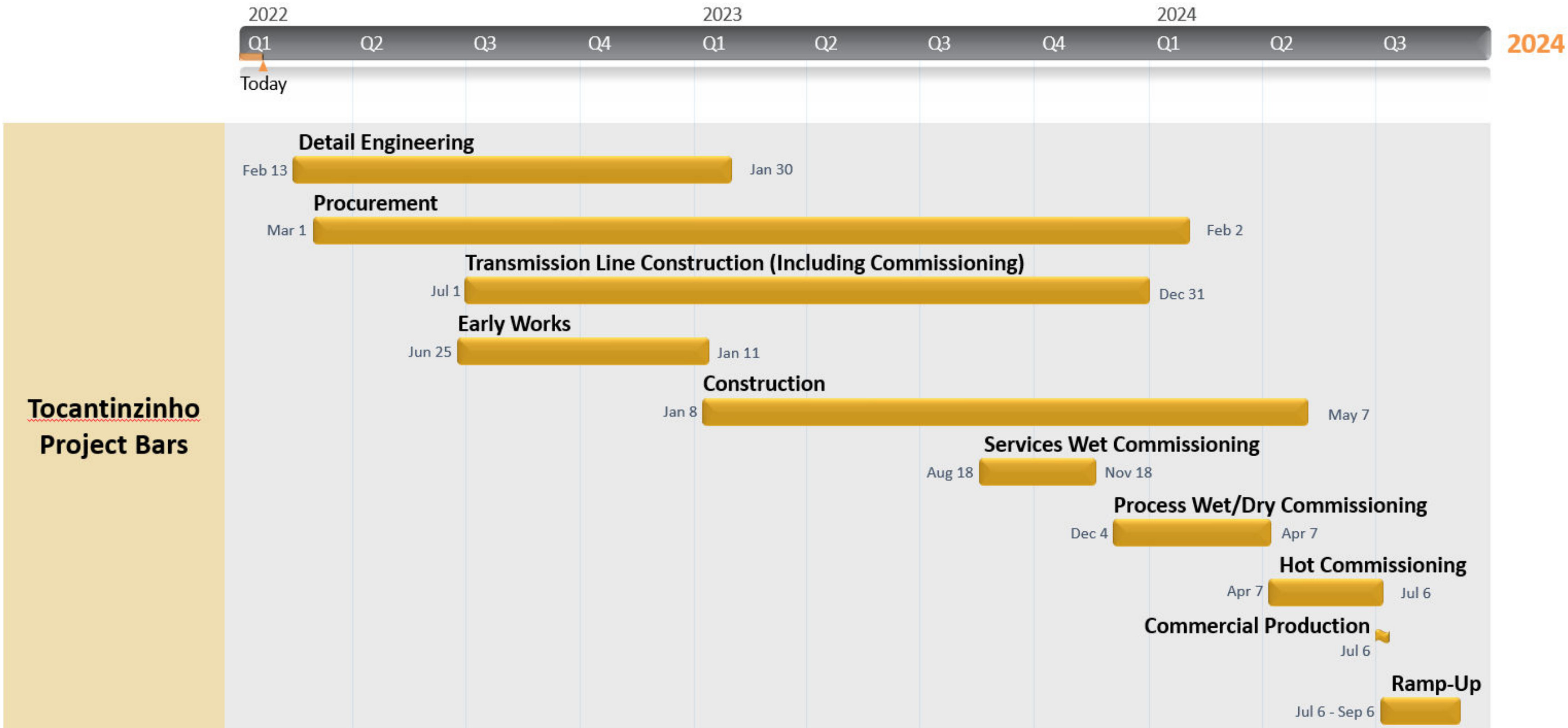
After-Tax NPV5%					
% Change	**Gold Price**	**BRL/USD**	**Grade**	**OPEX**	**CAPEX**
(30%)	111	378	113	806	738
(25%)	198	432	199	775	718
(20%)	283	479	284	745	699
(15%)	368	521	369	714	680
(10%)	453	558	453	683	660
(5%)	537	592	537	652	641
0%	622	622	622	622	622
5%	706	649	706	591	602
10%	791	674	790	560	583
15%	875	696	874	529	564
20%	960	717	959	499	544
25%	1,044	736	1,043	468	525
30%	1,129	753	1,127	437	506

1.21 Other Relevant Data

An integrated project management team (IPMT) will be created to lead the execution of the Project using a self-perform approach. The project team will be supplemented by contractors working within the IPMT for both specialized needs and peak manpower 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 Tocantinzinho mine team. Off-site infrastructure, including the access road and powerline, will be built or upgraded by a contractor under the supervision of the IPMT. The initial project construction period is scheduled over a period of 29 months. A level 1 project execution schedule is provided in Figure 1.7.

Figure 1.7: Tocantinzinho Project Schedule – Level 1



1.22 Risks and Opportunities

The main Project risks are:

- Completion of permit updates for the revised footprint
- Finalization of a power agreement contract
- Manpower availability in remote location
- Impact of foreign exchange on capital cost estimate
- Continued inflationary pressure
- Availability of goods and services in a remote location
- Lengthening of lead times for equipment and materials in general and specifically due to supply chain issues related to COVID
- Longer construction period due to COVID outbreaks affecting employees and contractors on site.

The main Project opportunities are:

- Increased Resources and Reserves at depth
- Exploration success in large surrounding exploration land package
- Optimization of comminution circuit following additional testwork
- Improved gold recovery of about 2% with fine grinding of sulphide concentrate
- Silver revenues expected as evidenced by metallurgical test work but not quantified in the resource model.

1.23 Recommendations

Based on the robust economic results of this Feasibility Study it is recommended that GMIN progress the Tocantinzinho Gold Project to detailed engineering and construction.

During the course of the Feasibility Study, items were identified as requiring additional information to further improve precision and information as part of the detailed engineering. Certain risks were also identified that require significant initiatives and continuous monitoring.

2 INTRODUCTION

The Tocantinzinho Gold Project (the "Project") is an advanced-stage development gold project located in Pará State, Brazil, 200 km south-southwest of the city of Itaituba, 108 km from the Morais de Almeida district, and 1,150 km southwest of Belém, capital city of Pará State.

GMIN completed the acquisition of the Project from Eldorado Gold Corp. ("Eldorado") through the purchase of all the issued and outstanding shares of *Brazauro Recursos Minerais S.A.* ("Brazauro" or "BRM") on October 27, 2021. In connection with that purchase and pursuant to the applicable securities legislation in Canada, GMIN must file an updated independent Technical Report under the National Instrument 43-101 Standards for Disclosure for Mineral Projects ("NI 43-101") in respect of the Project (this "Report") within 180 days of the transaction announcement (*i.e.*, the signing of a Share Purchase Agreement with subsidiaries of Eldorado) on August 9, 2021.

GMIN commissioned GMS and SRK, in each case under the supervision of independent qualified persons, as lead consultants along with other engineering consultants to prepare an updated Feasibility Study ("FS") in accordance with the guidelines of the Canadian Securities Administrators' NI 43-101 and Form 43-101F1, which FS will be this Report's subject matter.

The objective of this Technical Report and the FS is the evaluation of the technical feasibility and economic viability of the Project, notably the development of an open pit mine thereat, including processing facilities and related infrastructure. The mineral resource statement reported herein was prepared in conformity with generally accepted CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines.

2.1 Scope of Work

The Technical Report and FS responsibilities of the engineering consultants are as follows:

- GMS: overall Report and FS coordination, property description and location, accessibility, history, mineral processing and metallurgical testing, mineral reserve estimation, mining methods, recovery methods, project infrastructures, operating costs, capital costs, economic analysis, adjacent properties and project execution plan.
- SRK: geological setting, deposit type, exploration, drilling, sample preparation, data verification, mineral resource estimation, environmental studies and permitting.

A summary of the qualified persons ("QP") responsible for each section of the Report is detailed in Table 2.1.

Table 2.1: Summary of Qualified Persons

Qualified Person	Company	Title	Report Sections
Neil Lincoln, P.Eng.	Independent QP(1) for GMS Lincoln Metallurgical Inc	Metallurgical Consultant, President	1.1, 1.2, 1.3, 1.4, 1.11, 1.15, 1.16, 1.18, 1.21, 1.22, 1.23, 2, 3, 4, 5, 6, 13, 17, 18 (except 18.9), 19, 21, 23, 24, 25.1, 25.2, 25.5, 25.6, 25.8, 25.10, 25.11, 26.3, 26.4, 26.5, 27 Overall Responsibility for the Report
Charles Gagnon, P.Eng.	Independent QP(1) for GMS CGM Expert	Mining Consultant Owner and President	1.13, 1.14, 1.19, 1.20, 15, 16, 21, 22, 25.4, 25.9, 26.2
Camila Passos, MSc, PGeo, CREA-SP	SRK Consulting	Senior Resource Geologist	1.5, 1.6, 1.7, 1.8, 1.9, 1.10, 1.12, 7, 8, 9, 10, 11, 12, 14, 25.3, 26.1
Paulo Ricardo Behrens da Franca, P.Eng.	F&Z Consultoria e Projetos	Director	18.9
Thiago Toussaint, MBA, CREA-MG, AMEA	SRK Consulting	Principal Geoenvironmental Engineer	1.17, 20, 25.7, 26.6

(1) Engaged as independent contractors by GMS.

2.2 Sources of Information and Data

Unless otherwise stated, all the information and data contained in the Report or used in its preparation has been provided by GMIN or Brazauro; its wholly-owned local Brazilian subsidiary since October 27, 2021. The above-named QPs have no reason to doubt the reliability of the information provided.

Sources of information include:

- Discussions with GMIN and BRM personnel,
- Inspection of the Tocantinzinho Gold Project area, including outcrop and drill core,
- Review of exploration data collected by BRM,
- Additional information from public domain sources,
- Data and information from the 2019 Technical Report by Eldorado, and
- All figures and tables cited using references in Section 27.

The QPs who prepared the Report used information provided by the following sources who are not QPs for this Report:

- Golder Associates Brazil, a geotechnical engineering firm, who carried out in May 2012 a study of geomechanical modeling and provided pit slope design recommendations for the Project.
- Orway Mineral Consultants, a mineral processing consultancy, who performed crushing and comminution modelling and preliminary crusher and mill sizing in November 2021 as part of the process plant design.
- TEC3, a geotechnical consultancy, who completed revised tailings storage facility designs, deposition plans and stability analyses for the Project.

All currencies in this Report are expressed in United-States dollars (USD) unless otherwise stated.

2.3 Site Visits and Scope of Personal Inspection

Table 2.2: Site Visit Dates of Qualified Persons

Qualified Person	Site Visit Scope	Dates
Neil Lincoln, P. Eng.	Did not visit the site	N/A
Charles Gagnon, P. Eng.	Did not visit the site	N/A
Camila Passos, MSc, PGeo, CREA-SP	Geology/Mineral Resources	November 21 to 24, 2020
Paulo Ricardo Behrens da Franca, P. Eng	Tailings Infrastructure	February 21 to 23, 2019
Thiago Toussaint, MBA, CREA-MG, AMEA	Environment and Social	November 21 to 24, 2020

Several members of the GMS team, but who are not QPs, have performed visits to the Project site and elsewhere in Brazil during the month of October 2021 to collect information and meet vendors and service providers that were solicited for budgetary pricing and service proposals.

2.4 Units of Measure, Abbreviations and Nomenclature

The units of measure presented in this Report, unless noted otherwise are in the metric system. A list of the main abbreviations and terms used throughout this Report is presented in Table 2.3.

Table 2.3: List of Main Abbreviations

Abbreviations	Full Description
Ag	Silver
As	Arsenic
Au	Gold
BRM	Brazauro Recursos Minerais S.A.
C	Carbon
CAD	Canadian Dollar
CIL	Carbon-in-leach
CoG	Cut-off Grade
Cu	Copper
DD	Diamond Drilling
DGPS	Differential Global Positioning System
EGL	Effective Grinding Length
ETP	Effluent Treatment Plant
F	Degrees Fahrenheit
FA	Fire Assay
Fe	Iron
FS	Feasibility Study
G	Giga – (000,000,000's)
g	Gram
g Au/t	Grams of gold per tonne
gpt or g/t	Grams per tonne
g/L	Gram per litre
G&A	General & Administration
GMS	G Mining Services Inc.
gpm	Gallons per minute (US)
GPS	Global Positioning System

Abbreviations	Full Description
ha	Hectares
h	Hour
h/d	Hours per day
h/y	Hours per year
h/wk	Hours per week
HDPE	High-Density Polyethylene
hp	Horsepower
Hz	Hertz
IRR	Internal Rate of Return
ISO	International Organization for Standardization
k	Kilo – (000's)
kg	Kilograms
kg/t	Kilograms per tonne
kV	Kilovolts
km	Kilometre
km/h	Kilometre per hour
kPa	Kilopascal
kW	Kilowatts
kWh	Kilowatts per hour
L	Litre
LOM	Life of Mine
M	Mega or Millions (000,000's)
masl	Metres above sea level
m	Metre
m/min	Metre per minute
m/s	Metre per second
m^2	Square metre

Abbreviations	Full Description
m^3	Cubic metre
mg	Milligram
mg/L	Milligram per litre
mm	Millimetre
µm	Micron
ml	Millilitre
min	Minute
Mo	Month
Mt	Million tonnes
Mtpd or Mt/d	Metric tonne per day
Mtpy or Mt/y or Mt/a	Metric tonne per year
MVA	Megavolt-ampere
MW	Megawatt
NI 43-101	National Instrument 43-101Standards of Disclosure for Mineral Projects
NPI	Net Profit Interest
NPV	Net Present Value
NQ	Drill Core Diameter (47.6 mm)
NSR royalty	Net Smelter Return Royalty
Ø	Diameter
OK	Ordinary Kriging Methodology
OPEX	Operating Expenditures
oz	Troy Ounce (31.10348 grams)
OCR	Off-Channel Reservoir
PEA	Preliminary Economic Assessment
PFS	Pre-feasibility Study
Pb	Lead
PLC	Programmable Logic Controller

Abbreviations	Full Description
ppb	Parts per Billion
ppm	Parts per Million
psi	Pounds per square inch
PV	Present Value
RC	Reverse Circulation
RoM / ROM	Run-of-mine
rpm	Revolutions per minute
S	Sulphur
SAG	Semi-Autogenous
Sec	Second (time)
SRK	SRK Consulting Canada Inc./ SRK Consulting Brasil
STP	Sewage Treatment Plant
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
TRS	Tailings Reclaim Sump
TSF	Tailings Storage Facility
TTP	Thickened Tailings Plant
TWSP	Treated Water Storage Pond
USD	United States Dollar
V	Volt
VAT	Value Added Tax
VSD	Variable Speed Drive
WAD	Weak Acid Dissociable
wk	Week

Abbreviations	Full Description
XRF	X-ray Fluorescence
y	Year

3 RELIANCE ON OTHER EXPERTS

This Technical Report was prepared by GMS and SRK, 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 or its local Brazilian subsidiary, Brazauro Recursos Minerais S.A. ("Brazauro" or "BRM") and other third-party sources.

In preparing this Report, the QPs have fully relied upon certain work, opinions and statements of experts concerning environmental, legal, political or tax matters. The authors consider the reliance on other experts, as described in this section, as being reasonable based on their knowledge, experience and qualifications. The following companies and consultants have been retained by GMIN or BRM, to prepare various reports for the Project and have been relied upon in preparation of this Technical Report. The companies and their involvements are listed below:

- Mattos Filho, a law firm qualified to practice law in Brazil, performed a title opinion dated October 21, 2021, in connection with the acquisition of BRM from a subsidiary of Eldorado Gold Corp. ("Eldorado") by GMIN. The title opinion concluded that the Mineral Rights are currently in force and in good standing in all material aspects and is relied upon for Section 4 (Property Description and Location).
- Grebler Advogados, a law firm qualified to practice law in Brazil, reviewed Section 4 (Property Description and Location).
- Brandt Meio Ambiente, a consultancy in sustainability with a broad scope of environmental services, completed various baseline studies along with the ESIA for the Project dated July 2011, and is relied upon for various site descriptions, climate information and environmental and social impacts related to the Project.
- L&M Advisory, a Brazilian tax specialist, provided calculations of taxes used in the financial analysis of the Project, received by GMIN on January 30, 2022.
- CLAM Meio Ambiente, an engineering and environmental consultancy, conducted surface and ground water monitoring for the Project and estimated pit dewatering requirements and is relied upon for the relevant portions of Section 20 (Environmental Studies, Permitting and Social or Community Impact).

GMS believes the information provided to be reliable, but does not guarantee the accuracy of conclusions, opinions, or estimates that rely on third party sources for information that is outside the area of technical expertise of GMS. As such, responsibilities for the various components of the Summary, Conclusions and Recommendations are dependent on the associated sections of the Report from which those components were developed.

This Report is intended to be used by GMIN as a Technical Report with Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes contemplated under provincial securities laws, any other use of this Report by any third party is at such party's sole risk.

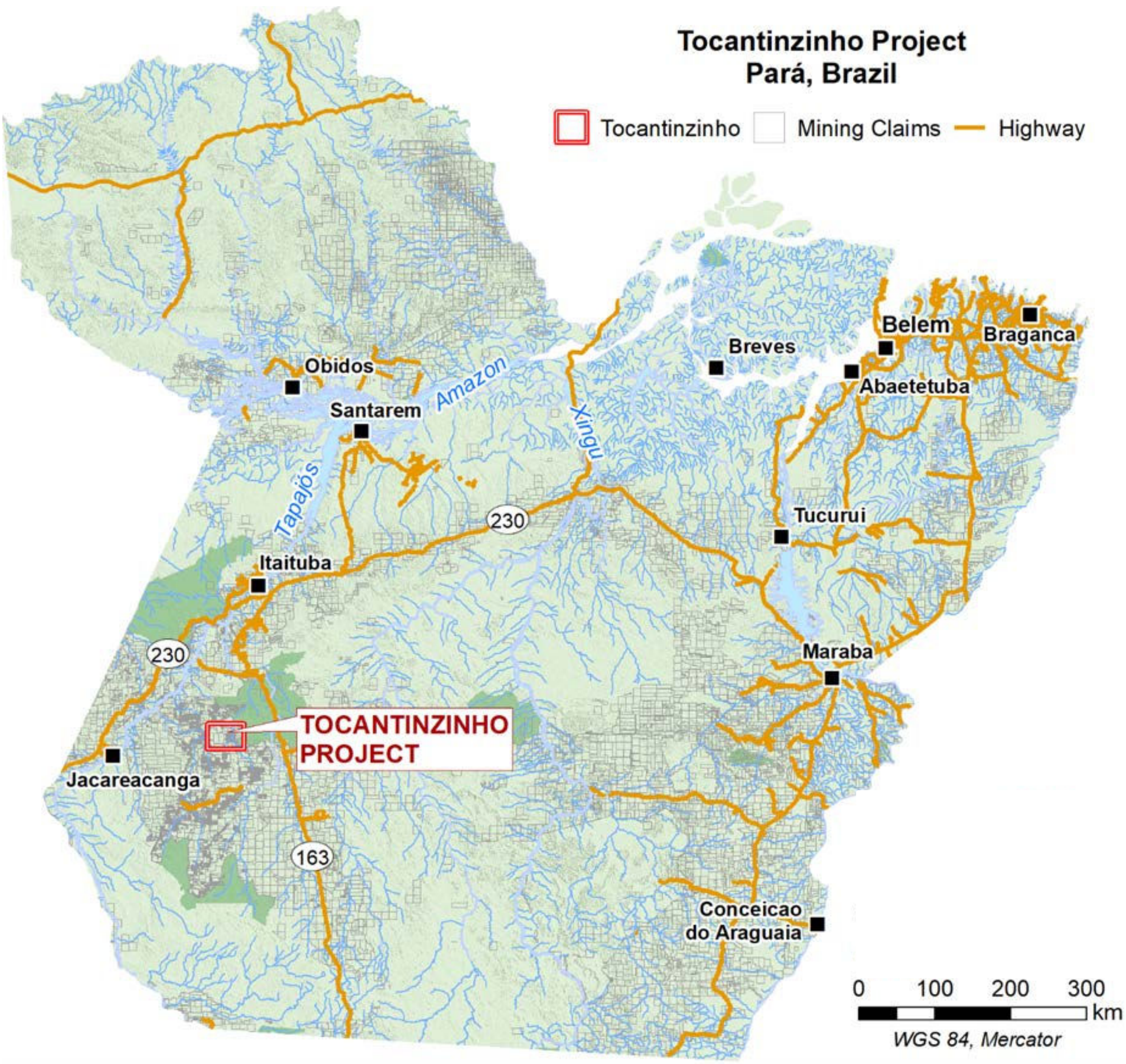
Permission is given to use portions of this Report to prepare advertising, press releases and publicity material, provided such advertising, press releases and publicity material does not impose any additional obligations upon, or create liability for GMS, except as required under provincial securities laws.

4 PROPERTY DESCRIPTION AND LOCATION

The Project area is located in the Tapajós Gold Province, approximately 200 km south-southwest of the city of Itaituba, 108 km from the Morais de Almeida district, and 1,150 km southwest of Belém, the capital city of Pará State, located along the north seacoast of Brazil, at the mouth of the Amazon River (Figure 4.1).

Itaituba (Figure 4.1) is the local center for services and supplies, and it is accessible by the BR-163 highway. A 72-km municipal road connects the Project site to the *Transgarimpeira* state road. There is a 775-m airstrip at the site.

Figure 4.1: Tocantinzinho Gold Project Location



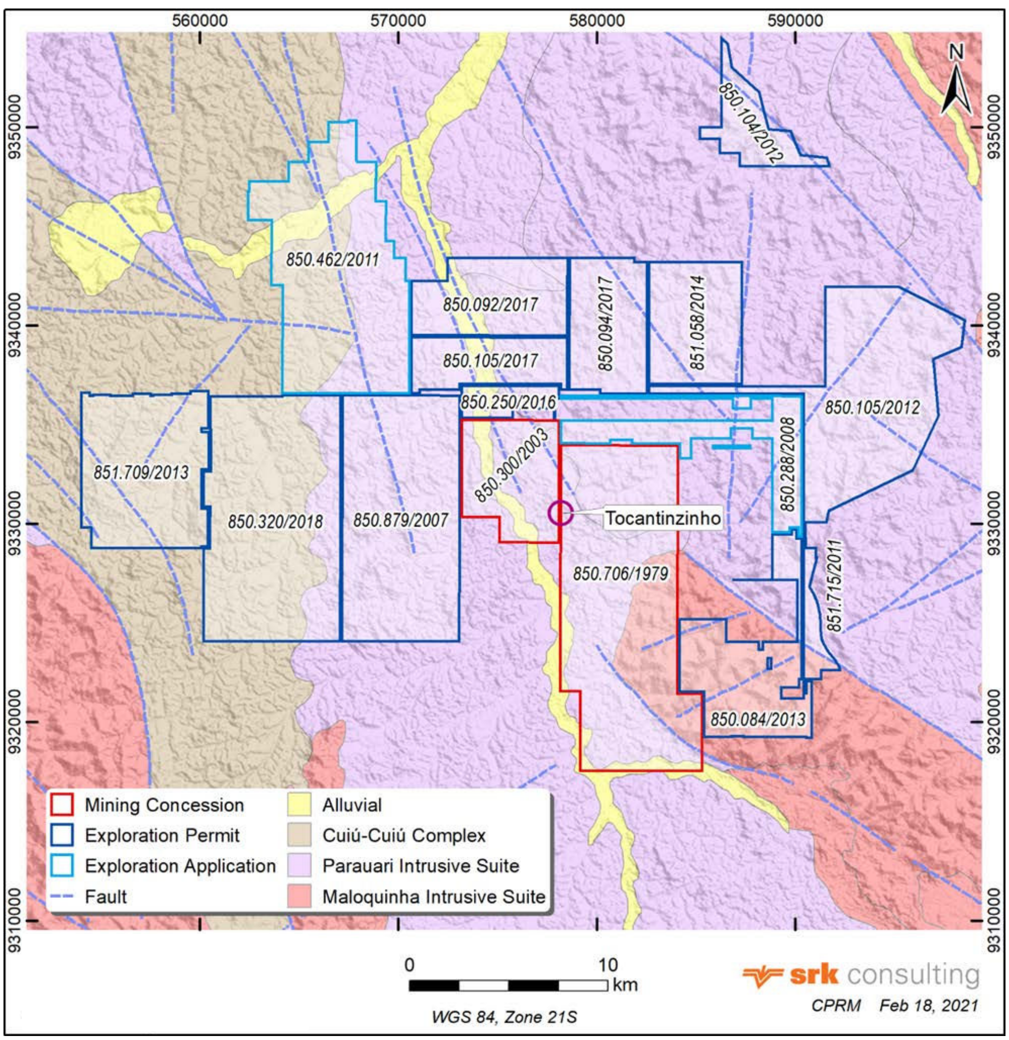
Source: SRK, 2019

4.1 Mineral Tenure

Under Brazilian law, the property of mineral resources differs from the property of the surface as the former is of exclusive ownership of the Federal Government. Therefore, to explore and exploit mineral resources, one must obtain an exploration licence and a mining concession, respectively. Since the granting of a mining concession depends on the results of prior exploration, the first step in obtaining a mining concession is actually to apply for an exploration licence from the National Mining Agency (*Agência Nacional de Mineração* or "ANM"); exploration licences are granted on a first-come, first-served basis.

Effective October 27, 2021, GMIN has a 100% stake in the Tocantinzinho Project through its wholly-owned subsidiary Brazauro Recursos Minerais S.A ("BRM"). It is comprised of two (2) mining concessions covering an area of 12,889 hectares, 23 exploration licences covering an area of 76,116 hectares, and two (2) applications for exploration licences covering 10,569 hectares (Figure 4.2).

Figure 4.2: Land Tenure Map



Exploration applications by public tender process are for those lands which have been held by other landholders, but which have been allowed to lapse. Table 4.1 summarizes the mineral tenure information for the mining concessions, exploration permits and exploration applications.

The National Department of Mineral Production (*Departamento Nacional de Produção Mineral* or "DNPM") was replaced in December 2017 by the ANM, which is the authority responsible for the enforcement and application of mineral legislation in Brazil, under the control of the Ministry of Mines and Energy ("MME"). While exploration licences do not have physical boundaries, they are issued based on digital geographic map staking, that is, they are not required to be legally surveyed and are subject to an annual rental fee.

BRM has requested the suspension of activities on mining concessions for an indefinite period pending a decision to undertake development and construction works, or a transaction regarding the Project. Such request is still under analysis at the ANM. Even though some exploration licences have already expired, the mineral rights attaching thereto remain fully valid and in force during the analysis period (of the relevant extension applications) by the ANM.

The mineral resources reported herein for the Tocantinzinho gold deposit are located within the mining concessions 850.300 / 2003 and 850.706 / 1979.

Table 4.1: Mineral Tenure Information

Phase	ANM-ID	Company	Status	Granting Date (dd/mm/yyyy)	Expiry Date (dd/mm/yyyy)	Area (ha)
Mining Concessions	850.300/2003	BRM	Suspension of activities (1)	18/05/2018	Depletion of the mineral deposit	2,888.69
	850.706/1979	BRM	Suspension of activities (1)	18/05/2018	Depletion of the mineral deposit	10,000.00
Subtotal						**12,888.69**
Exploration Licenses	851.709/2013	BRM	July 31, 2018: filing of a request for extension and a partial exploration report – under ANM analysis	12/11/2015	Three years after publication of Exploration License extension in the Official Gazette, which has yet to occur.	5,001.13
	850.320/2018	BRM	-	04/10/2018	October 4, 2021 (automatically extended to April 16, 2023, due to the pandemic)	8,537.43

Phase	ANM-ID	Company	Status	Granting Date (dd/mm/yyyy)	Expiry Date (dd/mm/yyyy)	Area (ha)
Exploration Licenses	850.879/2007	BRM	October 11, 2016: filing of a request for extension and a partial exploration report – under ANM analysis	17/12/2013	Three years after publication of Exploration Licence extension in the Official Gazette, which has yet to occur.	7,497.75
	850.105/2017	BRM	April 1, 2020: filing of a request for extension and a partial exploration report – under ANM analysis	06/06/2017	Three years after publication of Exploration Licence extension in the Official Gazette, which has yet to occur.	2,043.52
	850.092/2017	BRM	April 1, 2020: filing of a request for extension and a partial exploration report – under ANM analysis.	06/06/2017	Three years after publication of Exploration Licence extension in the Official Gazette, which has yet to occur.	2,979.17
	850.094/2017	BRM	April 1, 2020: filing of a request for extension and a partial exploration report – under ANM analysis	06/06/2017	Three years after publication of Exploration Licence extension in the Official Gazette, which has yet to occur.	2,734.56
	851.058/2014	BRM	July 31, 2018: filing of a request for extension and a partial exploration report – under ANM analysis	24/12/2015	Three years after publication of Exploration Licence extension in the Official Gazette, which has yet to occur.	2,988.53
	850.105/2012	BRM	July 31, 2018: filing of a request for extension and a partial exploration report – under ANM analysis	12/11/2015	Three years after publication of Exploration Licence extension in the Official Gazette, which has yet to occur.	7,003.77

Phase	ANM-ID	Company	Status	Granting Date (dd/mm/yyyy)	Expiry Date (dd/mm/yyyy)	Area (ha)
	851.715/2011	BRM	July 31, 2018: filing of a request for extension and a partial exploration report – under ANM analysis	02/10/2015	Three years after publication of Exploration Licence extension in the Official Gazette, which has yet to occur.	661.58
	850.084/2013	BRM	July 31, 2018: filing of a request for extension and a partial exploration report – under ANM analysis	11/12/2015	Three years after publication of Exploration Licence extension in the Official Gazette, which has yet to occur.	3,645.74
Exploration Licenses	850.104/2012	BRM	July 31, 2018: filing of a request for extension and a partial exploration report – under ANM analysis	26/10/2015	Three years after publication of Exploration Licence extension in the Official Gazette, which has yet to occur.	1,507.74
	851.691/2011	BRM	July 31, 2018: filing of a request for extension and a partial exploration report – under ANM analysis	02/10/2015	Three years after publication of Exploration Licence extension in the Official Gazette, which has yet to occur.	5,727.81
	851.695/2011	BRM	July 31, 2018: filing of a request for extension and a partial exploration report – under ANM analysis	02/10/2015	Three years after publication of Exploration Licence extension in the Official Gazette, which has yet to occur.	915.95
	851.696/2011	BRM	July 31, 2018: filing of a request for extension and a partial exploration report – under ANM analysis	02/10/2015	Three years after publication of Exploration Licence extension in the Official Gazette, which has yet to occur.	1,573.51

Phase	ANM-ID	Company	Status	Granting Date (dd/mm/yyyy)	Expiry Date (dd/mm/yyyy)	Area (ha)
	851.697/2011	BRM	July 31, 2018: filing of a request for extension and a partial exploration report – under ANM analysis	02/10/2015	Three years after publication of Exploration Licence extension in the Official Gazette, which has yet to occur.	732.39
	851.698/2011	BRM	July 31, 2018: filing of a request for extension and a partial exploration report – under ANM analysis	02/10/2015	Three years after publication of Exploration Licence extension in the Official Gazette, which has yet to occur.	4,329.53
	851.708/2011	BRM	July 31, 2018: filing of a request for extension and a partial exploration report – under ANM analysis	02/10/2015	Three years after publication of Exploration Licence extension in the Official Gazette, which has yet to occur.	2,602.79
	851.709/2011	BRM	July 31, 2018: filing of a request for extension and a partial exploration report – under ANM analysis	02/10/2015	Three years after publication of Exploration Licence extension in the Official Gazette, which has yet to occur.	1,552.69
Exploration Licenses	851.710/2011	BRM	July 31, 2018: filing of a request for extension and a partial exploration report – under ANM analysis	02/10/2015	Three years after publication of Exploration Licence extension in the Official Gazette, which has yet to occur.	1,615.24
	851.714/2011	BRM	July 31, 2018: filing of a request for extension and a partial exploration report – under ANM analysis	02/10/2015	Three years after publication of Exploration Licence extension in the Official Gazette, which has yet to occur.	4,032.22

Phase	ANM-ID	Company	Status	Granting Date (dd/mm/yyyy)	Expiry Date (dd/mm/yyyy)	Area (ha)
	851.779/2011	BRM	July 31, 2018: filing of a request for extension and a partial exploration report – under ANM analysis	02/10/2015	Three years after publication of Exploration Licence extension in the Official Gazette, which has yet to occur.	4,156.42
	850.096/2012	BRM	July 31, 2018: filing of a request for extension and a partial exploration report – under ANM analysis	26/10/2015	Three years after publication of Exploration Licence extension in the Official Gazette, which has yet to occur.	1,126.78
	851.710/2013	BRM	July 31, 2018: filing of a request for extension and a partial exploration report – under ANM analysis	24/12/2015	Three years after publication of Exploration Licence extension in the Official Gazette, which has yet to occur.	3,150.08
Subtotal						**76,116.43**
Exploration Applications	850.462/2011	André Luiz de Deus Maciel	On December 5, 2019: appeal against ANM's decision of October 29, 2015, which had rejected the extension of exploration licence and authorised a public tender process. The appeal has not been analyzed by ANM yet.	–	–	7,895.87
	850.288/2008	BRM	Application	–	–	2,673.37
Subtotal						**10,569.24**

Note: 1. On October 26, 2018, BRM informed ANM about the commencement of the mining activities. On June 3, 2020, BRM requested the suspension of the mining activities.
2. The Corporation has 60 days to appeal from ANM's decision to deny extension and intends to do so shortly.

4.2 Underlying Agreements

As mentioned above, GMIN has 100% ownership in the Project through its wholly-owned subsidiary, BRM, since October 27, 2021. Exploration licences in Brazil are subject to an annual tax to the Federal Government. The annual tax per hectare is R$3.70 / USD 0.68 (as of March 2021) for the first three (3) years of exploration, which increases to R$5.56 / USD 1.02 (as of March 2021) for the next three (3) years of exploration.

The surface area of the Project is located on a property of the Federal Government known as *Gleba Sumaúma* and registered with the Real Estate Registry Office of the Municipality of Itaituba, State of Pará. The Project's two (2) mining concessions covering an area of 12,888.69 hectares are within the perimeter of such property.

As BRM did not execute a specific agreement with the Federal Government to occupy the Project area, it possesses surface area based on (i) its rights deriving from the mining concessions (BRM may implement mining servitudes in due course) and (ii) agreements negotiated with land occupiers.

In September 2009, the National Institute of Colonization and Agrarian Reform ("INCRA") issued an Official Letter stating that, up to that date, there has been no indigenous land, traditional *quilombola* communities or settlements (*assentamentos*) in the aforementioned area which is part of *Gleba Sumaúma*.

BRM identified several occupants in the area who had requested, via administrative proceedings, the recognition of possession rights before the Federal Government. By means of a private instrument of assignment of possession rights (*Instrumento de Cessão de Direitos de Posse para fins de Pesquisa e Lavra de Minérios*), executed on July 27, 2011 (the "Assignment Agreement"), BRM concluded negotiations with the occupants for their rights over an area of 9,278 hectares, reduced to 6,670 hectares by means of an amendment to the Assignment Agreement executed on July 28, 2017. The relevant indemnifications, as set forth in the Mining Code, have been duly paid.

These acquired hectares are sufficient for the Project, including all areas required for the pit, waste dump, process plant, tailings dam and ponds, camping and administrative buildings.

The Assignment Agreement, as amended, includes terms of resignation (*termos de renúncia*) from all assignors / occupants. These terms were addressed to INCRA and registered with the third Notary Office of Santarém / PA, on August 1st, 2017. By these terms, the assignors waive any possession rights they could have over the aforesaid areas (*i.e.*, the possession rights in respect of which they were seeking recognition by the federal authorities). There are no outstanding indemnities to be paid to the assignors / occupants.

In addition to the abovementioned assignors / occupants, parts of the aforementioned area were also occupied by squatters, artisanal miners (*garimpeiros*), small merchants and other occupants without any title to the land, and BRM negotiated with a view to relocating and/or indemnifying them. As a result, BRM received 44 Terms of Commitment to Permanently Leave the Area (*termos de compromisso de desocupação definitiva de área*) executed with such occupants, who irrevocably undertook to completely vacate the areas, waiving any right they may have in respect thereof.

Most occupants indemnified between 2016 and 2019 have left the area. Five (5) of those occupants, although indemnified, did not leave the area and filed lawsuits against BRM. Another lawsuit against BRM was filed by Everaldo Cassiano (a.k.a. Baixinho do Peixe). He was not recognized by BRM as an occupant and, therefore, did not receive indemnification. BRM is pursuing a court settlement or a final court decision in respect of such disputes to complete this relocation / indemnification process. None of those occupants are located within the Project's footprint, except Francisco Soares dos Santos (a.k.a. "Chiquinho da Burra"), whose area is located close to the projected pit. In this case, the occupant was granted the right to stay in the area by an interim judicial order, which has been contested in court by BRM. BRM has also filed another lawsuit against *Chiquinho da Burra* based on the Mining Code, seeking an interim order for his relocation. The central issue around those cases is essentially the compensation quantum.

In addition, illegal artisanal mining activities occur on parts of the abovementioned area. There are seven (7) illegal artisanal mining areas within the perimeter of mining concession 850.300 / 2003, and two (2) of such areas within the perimeter of mining concession 850.706 / 1979. BRM reported such irregular activities to the mining authority, but there were no developments in that regard.

There are no other underlying agreements of significance currently affecting the Project, save for royalty agreements (please see below).

4.3 Environmental Licensing

Mineral rights in the exploration phase are granted exclusively by the ANM. However, in parallel with securing mining concessions and undertaking development and mining construction activities, environmental licences are required. More detailed information regarding all relevant aspects of environmental licensing is set out in Section 20 – Environmental Studies, Permitting and Social or Community Impact.

4.3.1 Use of Water

Interfering with the quality and the quantity of a specific waterbody by using its resources or discharging of sewage or other effluents requires a special permit that shall be granted by the National Water Agency (for projects that impact water bodies under federal jurisdiction) or by the state agency (for projects that impact water bodies under local jurisdiction). Regarding the Project, the relevant authority is currently SEMAS and there are currently three (3) active water permits:

- Groundwater extraction through one (1) well.
- Extraction of superficial water in *Água Branca*'s stream affluent and other water bodies in eleven specific locations, for paving works and concrete mass formation, in relation to the access road.
- Groundwater extraction through another well.

And there are currently two (2) water permits in respect of which a request for suspension was made to SEMAS:

- Installation of twelve culverts on the Tocantins River's sub-affluent, which is related to crossings (drainage).
- Flow regularization by means of a tailings dam.

4.3.2 Vegetation Suppression and Fauna Management

Relevant permits for vegetation suppression and fauna management expired as no activities being the subject matter of such permits are carried out currently. New permits must be requested before construction begins.

4.3.3 Conservation Units

Conservation Units ("UCs") are specially protected areas created and managed by federal, state, or municipal governments – more detailed information regarding all relevant aspects of UCs are set out in Section 20 – Environmental Studies, Permitting and Social or Community Impact.

4.3.4 Forest Code

The main environmental aspects regarding rural properties are foreseen in the Forest Code and relate to:

- CAR (Environmental Rural Registry): owners or possessors of rural properties must register them in the CAR, carried out through the federal or state Rural Environmental Registration System. There are a few CARs relating to the Project; they have been originally issued in the name of land occupiers, who granted consent to BRM for rectification of said registries for the name of BRM.

- RL (Legal Reserve): all rural properties must maintain an area of native vegetation covered, as RL, which varies between 20% and 80% of total area. There is an ongoing discussion in Pará State regarding the percentage to be set as RL for the rural properties located therein.
- APP: specially protected areas.

4.3.5 Waste Management

Management of solid waste generated in BRM's activities must comply with the National Policy on Waste Management ("PNRS") and with applicable state and municipal regulations. Among the PNRS obligations, companies generating mining waste (extraction or processing) must submit a Solid Waste Management Plan ("PGRS") under the environmental licensing procedure, providing information on generation, treatment, packing, transportation and final disposal of solid waste.

While there is a PGRS for the Project (presented in the licensing procedure), it has yet to be implemented due to the low quantities of waste currently generated.

4.3.6 Tailings Dams

The National Dams Safety Policy ("PNSB") provides for the main rules regarding the construction, operation and maintenance of dams for accumulation of water for any use, final or temporary disposal of tailings and the accumulation of industrial waste. In addition, mining companies must comply with the Brazilian Mining Code, ANM's regulations and other applicable technical standards regarding dams erected for mining activities.

Mining companies operating dams must submit to ANM and other relevant environmental authorities, updated technical information about the dams, and their Emergency Action Plan (*Plano de Ação Emergencial*).

4.4 Environmental Considerations

4.4.1 Past Mining Activities

Garimpeiros (*i.e.*, illegal, artisanal miners) have been working within the Project area for decades with continued activity at the site and it would be expected to find environment contamination related to the past activities, since the primary method of gold extraction used by the *garimpeiros involved* using mercury. However, although there is wide dissemination of mercury in areas of the site, the magnitude found is not enough cause for concern. However, during pre-stripping of the mine, monitoring will take place to discover any potential areas with higher-than-expected concentrations.

In letters sent to SEMAS, in 2013, 2018, and 2019, BRM informed that (i) prior to its presence in the area, in 2008, there were already two (2) areas being used by *garimpeiros*; (ii) BRM implemented a project to monitor the area with its security team so that such events do not occur within the area occupied by BRM; (iii) BRM's activities cannot be confused with the illegal activities developed by the *garimpeiros*; and (iv) BRM reported the occurrence to the ANM. No actions have been taken by SEMAS or ANM in this regard and BRM have not received any responses to the letters sent.

4.4.2 Environmental Studies

Environmental studies have been completed to support the environmental assessment and enable LI granting related to relevant structures of the Project (mining site, beneficiation plant, tailings dam, transmission lines, fuel station, concrete batch plant and landfill). The LIs are currently suspended but will be reinstated soon. The environmental studies necessary for the obtainment of those environmental licences were carried out by environmental engineering groups familiar with the Federal and State Government regulations for mining projects.

4.4.3 Closure and Site Remediation Planning

A high-level closure strategy, consistent with industry practice, has been developed and will be further defined upon commencing the Project's execution. More detailed information regarding all relevant aspects of closure and reclamation are set out in Section 20 – Environmental Studies, Permitting and Social or Community Impact.

4.5 Mining Rights in Brazil

As mentioned above, exploration licences in Brazil are issued based on digital geographic map staking and are not required to be legally surveyed.

Any Brazilian or Foreign Company properly registered in Brazil in accordance with Brazilian laws, as well as any Brazilian born citizen, can own mineral rights in Brazil.

Applications to obtain mineral rights must be filed at the ANM local office for the relevant mineral commodity, with precise reference to the land extents. Once the application is accepted, an exploration licence will be granted, normally for a period of three (3) years from the date of its publication in the Official Gazette (*Diário*

Oficial da União). An additional three-year extension can be requested by filing a report that details the completed exploration work and a proposed exploration program. At ANM's discretion, this period can be extended once under specific circumstances. During the exploration licence's term, its holder shall pay an annual fee per hectare (*Taxa Anual por Hectare*).

At the end of the three-year extension period, a final report must be filed with ANM to demonstrate the delineation of reserves or resources supported by drill results. Once the final report is approved by the ANM, the licence holder has exclusivity to apply for a mining concession to MME within one (1) year of such approval and has to complete the equivalent of a feasibility study, named a PAE (*Plano de Aproveitamento Econômico*) during that year. This can also be renewed for another one-year period, if appropriately justified. During such one-year period, the concession applicant must also apply for the necessary environmental licences. Once the licenses are granted and the PAE report is approved the concession owner has a period of six (6) months to commence mining operations.

The granted concession for mining provides the miner a title that warrants the use of the mineral resource (*Portaria de Lavra*). This title can only be achieved through definition of an economic mineral reserve through mineral exploration, which is presented in the PAE.

Upon granting of the mining concession, the holder must start mine development plan within six (6) months. In case of breach of that obligation, or if the titleholder does not apply for an extension of such term, ANM may apply administrative penalties. A mining concession is currently granted for an indefinite term, but it does consider the reserves evidenced in the mine development plan which may be reviewed and amended from time to time, and which is valid until the depletion of the mineral deposit.

The holder of a mining concession may institute a mineral servitude for the use of an area within or outside the limits of its Mining Concession, as may be required for the operations (*e.g.*, for opening of transport routes and transmission lines, collection and supply of water, passage opening for ventilation and dump areas).

Land occupiers affected by the servitude are entitled to compensation and rent (guidelines are provided in the Brazilian Mining Code). To establish a servitude, a request must be filed with ANM. Such request is not mandatory and is mainly in the interest of the mining titleholder who does not own the property on which it intends to build the relevant facilities. BRM made a request to create a mineral servitude in respect of the area covering the mining concessions to ANM, which has yet to be analyzed.

4.6 Power Line

In December 2017, BRM obtained the permission for the construction of a 200 km, 138 kV electrical transmission line to transport 18 MW power to the Project site.

To construct this line, BRM sought to create easements along a land strip with 192 km length per 25 m width. Easement agreements (*contratos de constituição de servidão*) basically will guarantee the passage of the power line over the area, the right to access the area whenever needed and that no constructions or activities that may affect the power line will be developed underneath it. Negotiations of these easements involved 148 land occupiers and, as of February 2019, 117 easement agreements were executed.

Among the outstanding negotiations with the remaining land occupiers, five (5) of them have accepted BRM's proposals and are in the process of executing relevant documentation, five (5) are negotiating with BRM, and 20 occupiers refused to sign any agreement. In the latter cases, BRM may need to revert to courts to implement the easement judicially. Reverting to courts, for all practical intents and purposes, means debating about the amount of compensation to be paid to the relevant occupiers. The possibility of never obtaining easements is highly remote.

4.7 Royalties

Under Brazilian law, the Brazilian government charges a statutory royalty known as the Financial Compensation for the Exploitation of Mineral Resources (*Compensação Financeira pela Exploração de Recursos Minerais* – "CFEM"). Any revenue resulting from the sale of mineral products is subject to CFEM, which is due and payable to the Federal Government and distributed among the Federal Government, the State, certain Municipalities, and other public administration departments.

CFEM is assessed (i) upon revenues from sales, considering gross revenues, which are calculated as being the amount resulting from sales of mineral product after deduction of the taxes levied on the sales, or (ii) for those cases in which the mining operator uses the mineral production in its production process, based on the market price (either international, national, regional or local) or based on the reference value to be defined upon completion of the beneficiation process. The current CFEM rate for gold is 1.5%.

In addition to the foregoing, GMIN has entered into an agreement pursuant to which it assumes the obligation to pay royalties of 2.75% and 0.75% to Osisko Gold Royalties Ltd. ("Osisko") and Metalla Royalty & Streaming Ltd. respectively, on future revenues derived from the sale of gold mined from the Project. The area covered by these royalties substantially correspond to that covered by the two (2) mining concessions. It is worth noting that, under Brazilian law, mining royalties have a contractual nature and cannot be registered against mineral rights.

As disclosed by GMIN in a press release dated November 24, 2021, GMIN exercised the first of its two (2) Buy-Down Rights (as defined in the relevant agreement) to reduce Osisko's royalty by 1% to 1.75%, and paid a cash consideration of USD 2.0 million. GMIN's second Buy-Down Right to further reduce Osisko's royalty by an additional 1% to 0.75% can be exercised within 30 days of a construction decision in respect of the Project (for a cash consideration of USD 3.5 million).

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

5.1.1 Access by Road

The Tocantinzinho Gold Project is located approximately 200 kilometres southwest of the city of Itaituba, which is the local centre for services and supplies. The Transgarimpeira Highway is located 50 kilometers south of the property and connects with BR-163 at the town of Morais Almeida, approximately 22 kilometres east of the Jamanxim River barge crossing at Jardim do Ouro (Eldorado 2019). A 72-kilometre-long gravel road connects the Tocantinzinho Project to the Transgarimpeira Road, which requires ongoing maintenance to facilitate traffic for the construction of the Project.

The Cuiabá-Santarém highway (BR-163) extends south from east of Itaituba to Cuiabá in Mato Grosso state, passing approximately 70 kilometres east of the project. This highway provides a link to other national highways in southern Brazil. The Cuiabá-Santarém highway extends north to ports in Itaituba and Santarém along the Tapajós River. The Tapajós River is a major tributary of the Amazonas River, which is navigable by barge to Belém, the capital and largest city of the state of Pará. Belém, in the country's north, is the gateway to the Amazonas River with a busy port and international airport. The BR-163 highway combined with the Tapajós River is in fact an extensive multimodal logistics corridor for the transportation of agricultural commodities from the north of Mato Grosso to the Atlantic.

Table 5.1: Road Access from Various Cities of Interest

Route	Distance (km)	Road Type	Travel Time
São Paulo – Moraes Almeida	2,704	Paved highways	36 h
Belo Horizonte – Moraes Almeida	2,752	Paved highways	38 h
Belém – Moraes Almeida	1,547	Mostly paved, with 502 km of dirt road	25 h
Itaituba – Moraes Almeida	303	Paved highways	4:39 h
Novo Progresso – Moraes Almeida	100	Paved highways	1:19 h
Moraes Almeida – Jardim do Ouro	22	Dirt road	32 min
Moraes Almeida – Tocantinzinho	108	Dirt road	2:45 h

5.1.2 Access by Air

Small aircrafts such as a Cessna Grand Caravan with 10 to 14 passengers can be chartered from Itaituba, or other nearby cities, to the airstrip located on the property. The Pista Nações Unidas airstrip which serves the Project is 775 m long and is situated 2 km south of the camp. This airstrip will primarily be used to supply the site with personnel and supplies. The flight takes approximately one (1) hour from Itaituba and is weather-dependent. Utilization is limited to daylight as the runway has no signals or lighting.

Air transportation routes and distances are summarized Table 5.2.

Table 5.2: Air Travel from Various Cities of Interest

Route	Flight	Travel Time
São Paulo – Belém	Commercial	4 h
Belo Horizonte – Belém	Commercial	3 h
Belém – Itaituba	Commercial	3 h
Itaituba – Tocantinzinho	Charter	1 h

5.2 Climate

The mean annual temperature in the region is approximately 28ºC. In general, the temperature amplitudes are small with a gradual increase during winter. The mean absolute values are in range of 23ºC to 37ºC. The relative humidity averages above 86% throughout the year.

The climate in northwestern Brazil is tropical, with a rainy season from January to April and a dry season extending from June to December. The average annual precipitation is approximately 1,957 mm. The rainiest trimester contributes about 40% of total annual rainfall, corresponding to the months of February, March and April. The driest trimester, corresponding to the months of July, August and September, contribute less than 15% of total annual rainfall.

To verify the climatic characteristics of the region, data was collected in the Climatological Station of Itaituba municipality. The climate normals and averages were used to summarize and describe the average climatic conditions of the project area. The data were compiled and are presented in Table 5.3.

Table 5.3: Itaituba Climate Normals Station Data

Month	Min	Avg.	Max	Max. Absolute	Min. Absolute	Relative Humidity %	Total Insolation hours	Total Rainfall mm	Average Evaporation mm
	Temperature °C								
Jan	24.51	26.48	28.6	34.52	19.21	86.33	139	226	58
Feb	24.25	26.2	28.42	33.48	19.35	89.76	109	291	47
Mar	24.8	26.34	28.14	33.2	19.21	89.71	119	302	52
Apr	25.05	26.55	28.15	33.76	19.22	89.44	148	243	49
May	25.16	26.66	27.78	34.1	19.8	89.68	170	186	51
Jun	24.65	26.74	28.09	34.27	18.82	87.81	213	93	61
Jul	23.95	26.66	28.02	35.33	16.96	85.48	246	65	76
Aug	25.01	27.28	29.12	35.94	17.18	82.97	241	57	91
Sep	25.19	27.86	29.47	36.58	19.68	82.07	218	79	95
Oct	24.9	27.97	29.5	36.25	20.17	81.63	193	99	97
Nov	24.82	27.71	30.22	36.34	19.55	81.21	171	144	84
Dec	25.07	27.2	29.21	34.86	19.35	86.22	145	173	73
Annual	24.78	26.97	28.73	34.88	19.04	86.03	176	1,957	834

Annual average wind speeds vary from 0 m/s to 7 m/s (light to moderate breeze), rarely exceeding 7 m/s (moderate breeze). The highest average wind speed is 4 m/s, and lowest average 2 m/s. Maximum gust speed recorded in 2015 was 37 m/s. The wind is most often out of the east (15% of the time).

5.3 Local Resources

The town of Itaituba is the major local center for services and supplies, located at the crossing of the Tapajós River and the Trans-Amazonica Highway. The Cuiabá-Santarém Highway BR-163, extending northward from the state of Mato Grosso, reaches Itaituba via a ferry crossing of the Tapajós River. Most heavy equipment and supplies reach Itaituba by smaller ships, which move along the Amazon River and Tapajós River.

Power will be supplied from the Novo Progresso substation to the south, which will require the construction of approximately 198 kilometres of 138 kV transmission line and a substation at the site. Power infrastructure to upgrade and supply additional power to Novo Progresso was constructed by the Brazilian utility in 2020.

Fresh water is accessible by the Veados Creek, which will be stored in a water tank for processing activities. A portion of the water within the tank will be dedicated as fire water storage. Potable water will be sourced from wells and will be treated prior to use. Water reclaimed from ponds will be recycled for use in the processing plant.

5.4 Infrastructure

The access road connecting the Project to the Transgarimpeira Highway was constructed by Eldorado and is considered a Municipal Road of the Itaituba Municipality. The Project is responsible for maintenance of this road but is used by the public. Some improvements are required to improve drainage and gradients for certain segments.

Along the access road, there is a barge to cross the Jamanxim River at Jardim do Ouro that is operated by a commercial operator whose contract is awarded by the Municipality of Itaituba. The barge has a capacity of 402 tonnes with three (3) lanes of vehicles onboard. There is spare tug on location to assure reliability of service.

At the Project site there is an existing exploration camp with a capacity of about 90 beds complete with kitchen, recreation room, clinic, fuel storage, core shacks, and office space. The camp has two (2) wells for potable water supply and treatment capacity of 1,000 L/d, sewage treatment and two 40 kVA power generators. The exploration camp will be expanded to approximately 200 beds to support the initial phases of construction during which the permanent camp will be constructed.

The data and internet system of the Tocantinzinho Site is carried by a network of interconnected telecommunication towers beginning in Moraes Almeida. The internet link has a capacity of 40 Mbp/s, with the ability to upgrade. A radio communications system with four repeater towers along the access road allows for continuous communications.

5.5 Physiography

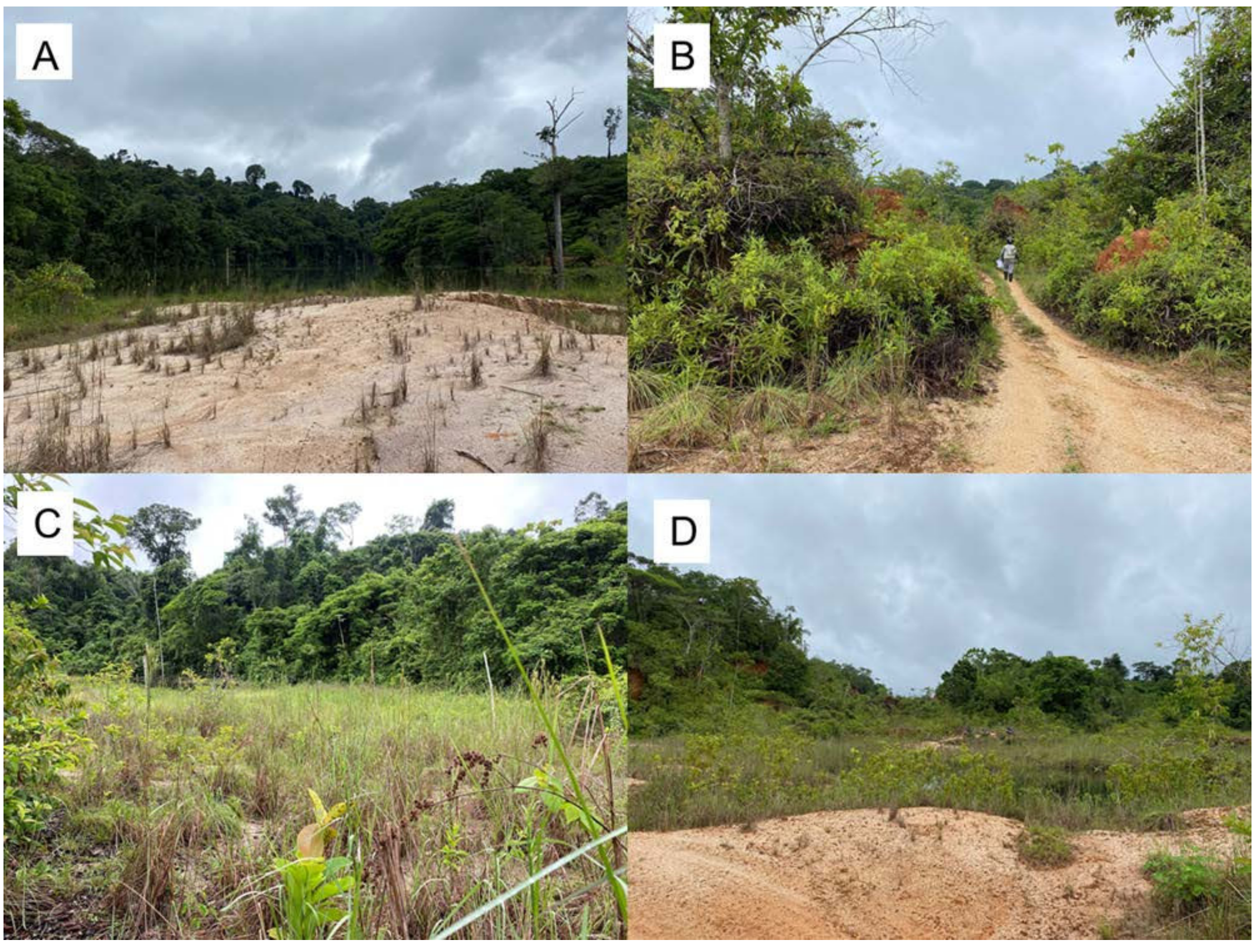
The Project property is located in the north region of Brazil, which is humid tropical forest. In the study area, three (3) forest typologies were observed: secondary forest, dense alluvial forest, and submontane dense ombrophilous forest.

The area is hilly with areas that are rocky or with saprolite cover. The local geology is represented by three (3) lithostratigraphic units: Parauari Intrusive Suite, Maloquinha Intrusive Suite and Quaternary Alluvial Reservoirs. The most significant rocky outcrops occur in the sector where artisanal mining activities were concentrated. The rocks are altered and the different saprolites present a distinction of color and lithological material.

The site area is part of the hydrographic sub-basin of the Jamanxim River that integrates the Tapajós River basin. This basin has the most of its extension protected by conservation units. The site area is influenced by Jamanxim and Tocantinzinho rivers, besides its main tributaries, Veados and Teodorão creeks.

General illustrations of the physiography of the project area are shown in Figure 5.1

Figure 5.1: Typical Landscape in the Project Area
(Images A-D represent a mosaic of landscape images from within the project area)



Source: GMIN (2021)

6 HISTORY

6.1 Prior Ownership and Changes

Historical mining activity in the Tapajós Gold Province region was primarily related to gold mineralization. Artisanal miners discovered gold in the region through small-scale mining activities in the 1950s. It was not until the 1980s that the area became a significant gold producer. Historical production involving primitive artisanal methods amounted to between 200,000 and 1 million ounces of gold per year, and was estimated at approximately 16 million ounces by the 1990s.

Although there are no published records to support the timing and amount of production, gold extraction in the Tocantinzinho area is thought to have initiated in 1970 with intense small-scale mining activity in the mid-eighties to mid-nineties. An exploration license was granted to Mineração Aurífera Limitada in 1979 by the Departamento Nacional de Produção Mineral ("DNPM"), known today as the Agência Nacional de Mineração ("ANM") over the Tocantinzinho Project area. The license expired in 1986 and the property files were archived by the DNPM in 1992.

Renison Goldfields from Australia and Altoro formed a Joint Venture ("JV") in 1997 to explore for gold in Brazil. An air charter pilot brought the Tocantinzinho area to the JV's attention, for which Altoro was the operator of exploration activities. A geologist from the JV visited the Project and collected channel samples from various artisanal pits, which returned favourable results and prompted the acquisition of the property. In 1998, the JV with Renison Goldfields was terminated and Altoro inherited all projects and data acquired.

In 2000, Altoro was acquired by Solitario Resources Corporation and terminated the Tocantinzinho Project a year later due to a drop in gold price.

In 2003, the land over the Tocantinzinho Project mineralization was acquired by Brazauro Resources Corporation, through its Brazilian subsidiary Jaguar Resources do Brazil Ltda.

In July 2008, Eldorado made an acquisition agreement with Brazauro Resources Corporation, which involved the continued exploration and development of the Tocantinzinho Project through access to Eldorado's exploration and project development expertise in Brazil. In July 2010, Eldorado completed the arrangement to acquire all the issued and outstanding securities for ownership of Brazauro Resources Corporation, for a total consideration of approximately CAD 122.4 million.

Eldorado expanded the tenement from 33,979 ha to 68,803.6 ha between 2012 and 2018.

On August 9, 2021, GMIN made the announcement to acquire the Tocantinzinho Gold Project from Eldorado through the acquisition of all the issued and outstanding shares of Brazauro Recursos Minerais S.A. The acquisition was completed on October 27, 2021, for an aggregate consideration of $115 million made up of an upfront and deferred considerations as follows:

- An initial consideration of $55 million:
  - $20 million cash payment paid at closing.
  - 46,926,372 shares of GMIN valued at $35 million. Following the completion of the acquisition, Eldorado owns a 19.9% direct equity interest in GMIN.
- A deferred consideration of $60 million:
  - Payable at GMIN's option anytime from closing until the first anniversary of Project achieving commercial production.
  - GMIN, at its option, may defer 50% of the deferred consideration for 12 months subject to a $5 million premium payable on the second anniversary of the Project achieving commercial production.

6.2 Previous Exploration Work

6.2.1 Altoro Gold Corporation (1997-1999)

Altoro's exploration program was carried out from 1998 to early 2000 and consisted of soil geochemistry, auger drilling, geological mapping and a ground magnetic survey.

Channel Sampling

Altoro completed a total of 476 channel samples of saprolite from the various artisanal pits. These involved 4-m long horizontal samples from walls and floors of the artisanal miners pits and a few vertical channels collected from the walls. Various samples were collected from artisanal miners working faces with little to no systematic approach to sample collection locations. All channel samples collected by Altoro were assayed for gold by Bondar Clegg laboratory.

Soil Sampling

Altoro collected over 700 soil samples between 1997 and 1999. Sampling was conducted along lines and grids extending up to 2 km around the main garimpeiro workings. Within the main gride over the mineralized zone, soil lines were spaced 100 metres apart and samples were taken at 40-m intervals along the lines using a hand auger.

A survey was conducted at the start of the program to determine the optimum depth for soil sample collection. In this study, samples were collected from several locations involving 50-cm increments. The results showed slight gold enrichment in the top 50 cm and relatively consistent enrichment below. As a result of this survey, all samples were collected at a depth of between 0.5 and 1.0 m. The soil samples were not assayed, but instead gold grade was calculated based on the number of gold grains counted after panning the samples.

All of the soil sampled areas are now covered by either sandy tailings or water.

Power Auger Drilling

Altoro completed a total of 87 power auger boreholes (1,318 m) in 1998, followed by 58 boreholes (503 m) in 1999 (Table 6.1). The power augers were acquired from a manufacturer in Belo Horizonte and were designed specifically for sampling laterite / saprolite to a depth of 30 m with perforation rates of 15 to 20 m per day. Most of the boreholes were drilled to 20 m.

Table 6.1: Summary of Drilling Completed by Altoro on the Eldorado Gold Project

Company	Period	Type	Target Area	Goal	No. Boreholes	Total Length (m)
Altoro	1998	Auger	TZ	Exploration	87	1,318
	1999	Auger	TZ	Exploration	58	503
Total					**145**	**1,821**

Sampling occurred in 1-m intervals for the first 10 power auger boreholes, and 2-m intervals for subsequent boreholes. Boreholes were logged by a geologist for primary lithologies interpreted for saprolite. The average saprolite intersection power auger drilling was 9.1 m with an average grade of 1.00 g Au/t.

Geophysical Surveys

Altoro completed collected 6 km of magnetic data using one magnetometer along 10 established grid lines, spaced 50 m apart.

6.3 Historical Mineral Resources Estimates

A first mineral resource estimate for the Tocantinzinho deposit was prepared by Pincock, Allen & Holt (PAH) on November 27, 2006, documented in a National Instrument 43-101 technical report. The deposit was estimated at 794,000 ounces of indicated mineral resources of gold with an average grade of 1.48 grams of gold per tonne (g Au/t) and 833,000 ounces of inferred mineral resources of gold with an average grade of 1.34 grams of gold per tonne (g Au/t) using a cut-off of 0.1 g Au/t (Table 6.2).

Table 6.2: Mineral Resource Statement* Tocantinzinho Gold Deposit, Pará, Brazil, Pinckock, Allen & Holt, November 27, 2006

Category	Quantity ('000 tonnes)	Gold Grade (g/t)	Contained Gold ('000 ounces)
Indicated	16,646	1.48	794
Inferred	19,384	1.34	833

**Note: Reported at a cut-off grade of 0.1 grams of gold per tonne. All figures rounded.*

In 2007, NCL Brasil Ltda (NCL) prepared a Preliminary Assessment of the Tocantinzinho Gold Project, documented in a National Instrument 43-101 technical report. The deposit was estimated at 1,053,000 ounces of indicated mineral resources of gold with an average grade of 1.33 g Au/t and 1,048,000 ounces of inferred mineral resources of gold with an average grade of 1.18 g Au/t using a cut-off of 0.2 g Au/t (Table 6.3).

Table 6.3: Mineral Resource Statement* Tocantinzinho Gold Deposit, Pará, Brazil, NCL Brasil Ltda, December 2007

Category	Quantity ('000 tonnes)	Gold Grade (g/t)	Contained Gold ('000 ounces)
Indicated	24,597	1.33	1,053
Inferred	27,704	1.18	1,048

**Note: Reported at a cut-off grade of 0.2 g Au/t.*

In 2010, NCL updated the Preliminary Economic Assessment of the Tocantinzinho Gold Project, documented in a National Instrument 43-101 technical report. The deposit was estimated at 1,892,000 ounces of measured and indicated mineral resources of gold with an average grade of 1.14 g Au/t and 389,000 ounces of inferred mineral resources of gold with an average grade of 0.98 g Au/t using a cut-off of 0.254 g Au/t and an open pit (Table 6.4).

Table 6.4: Mineral Resource Statement* Tocantinzinho Gold Deposit, Pará, Brazil, NCL Brazil, March 2010

Category	Quantity ('000 tonnes)	Gold Grade (g/t)	Contained Gold ('000 ounces)
Measured	15,834	1.23	629
Indicated	35,572	1.10	1,264
Measured and Indicated	**51,406**	**1.14**	**1,892**
Inferred	12,400	0.98	389

**Note: Reported at a cut-off grade of 0.254 g Au/t based on an open pit mining scenario, metal prices of USD 1,200 per ounce of gold, metallurgical recovery of 80% for gold in oxide and 93% in sulphide material.*

In early 2011, Eldorado published a NI 43-101 technical report. The deposit was estimated at 2,394,000 ounces of measured and indicated mineral resources of gold with an average grade of 1.06 g Au/t and 147,000 ounces of inferred mineral resources of gold with an average grade of 0.66 g Au/t using a cut-off of 0.3 g Au/t (Table 6.5).

Table 6.5: Mineral Resource Statement* Tocantinzinho Gold Deposit, Pará, Brazil, Eldorado Gold Corp., December 31, 2010

Category	Quantity ('000 tonnes)	Gold Grade (g/t)	Contained Gold ('000 ounces)
Measured	19,777	1.29	820
Indicated	50,457	0.97	1,574
Measured and Indicated	**70,234**	**1.06**	**2,394**
Inferred	6,950	0.66	147

**Note: Reported at a cut-off grade of 0.3 g Au/t.*

The last mineral resource statement for the Tocantinzinho Gold Project was prepared by Eldorado and published in a NI 43-101 technical report in the middle of 2019. Eldorado estimated the deposit at 2,115 thousand ounces of measured and indicated mineral resources of gold with an average grade of 1.35 g Au/t and 69,000 ounces of inferred mineral resources of gold with an average grade of 0.90 g Au/t using a cut-off of 0.3 g Au/t (Table 6.6).

Table 6.6: Historical Mineral Resource Statement* Tocantinzinho Gold Deposit, Pará, Brazil, Eldorado Gold Corp., September 30, 2018

Category	Quantity ('000 tonnes)	Gold Grade (g/t)	Contained Gold ('000 ounces)
Measured	17,530	1.51	851
Indicated	31,202	1.26	1,264
Measured and Indicated	**48,732**	**1.35**	**2,115**
Inferred	2,395	0.90	69

**Note: Reported at a cut-off grade of 0.3 g Au/t.*

6.4 Historical Production

Many small alluvial gold deposits of the Tocantinzinho Gold Project have been mined by artisanal miners. There is no available information regarding the average grade of the material, or the quantity of gold extracted from these small-scale mines.

GMIN is not aware of any artisanal or irregular small-scale miners presently operating within the project area.

7 GEOLOGICAL HISTORY AND MINERALIZATION

This section is modified from Eldorado Gold Corporation (2019).

7.1 Regional Geology

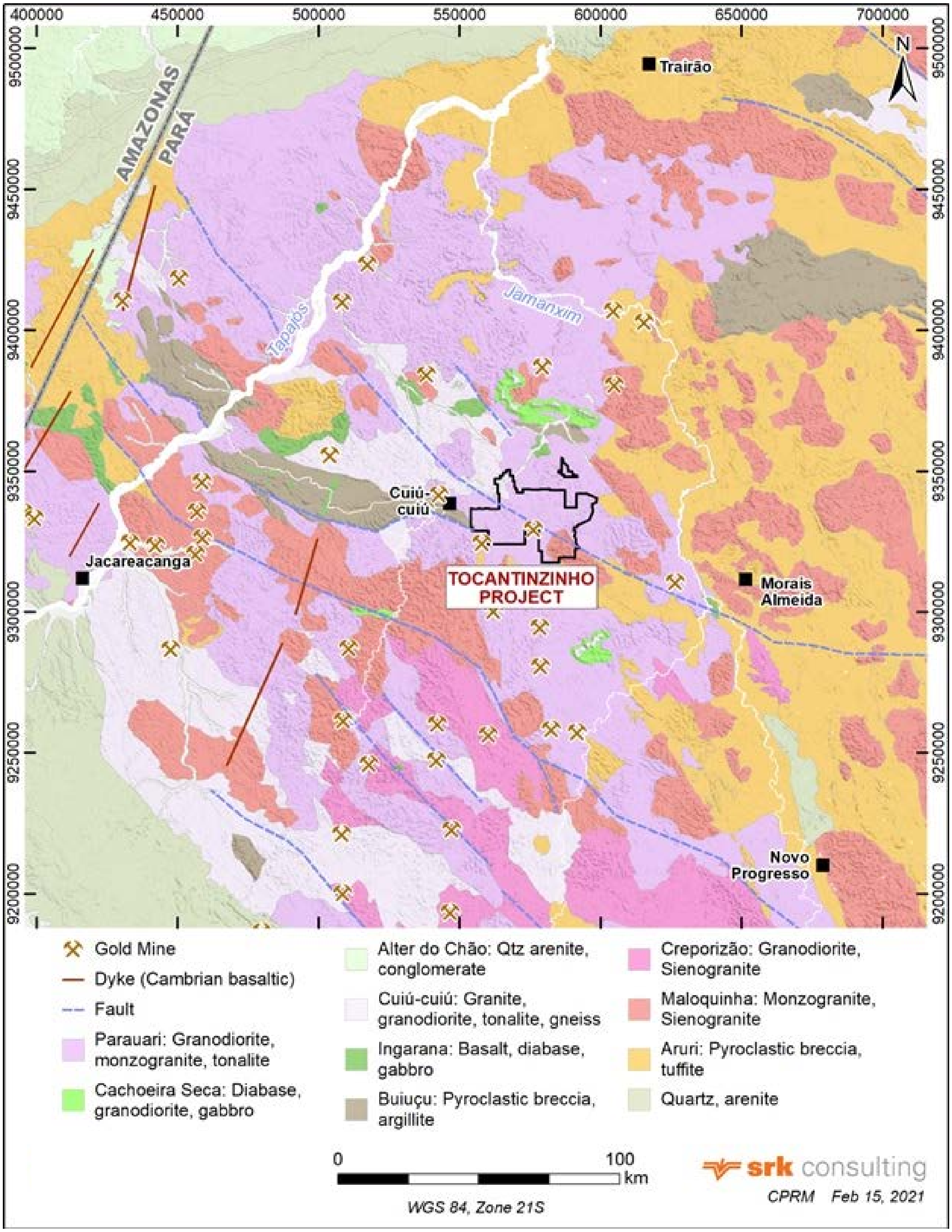
The Tocantinzinho Gold Project is located in the Tapajós Gold Province in the central southern portion of the Amazon craton and part of the Venturi-Tapajós (Tassinari and Macambira, 1999) or Tapajós-Parima (Santos et al., 2001) geochronological-tectonic province (Figure 7.1).

The oldest rocks of the Tapajós district form the basement rocks in the region and comprise gneisses, schists, and metagranites of the Cuiú- Cuiú complex (2,033 – 2,011 Ma; Santos et al., 2001). The Cuiú-Cuiú complex was intruded by granites and granodiorites of the Creporizão Suite (1,974 ± 6 to 1,957 ± 6 Ma), tonalites, diorites and granodiorites of the Tropas Suite (1,909 – 1,895 Ma) and granites and granodiorites of the Parauari Suite (1,898 – 1,880 Ma; Santos et al., 2001). The rocks of the Parauari, Tropas and Creporizão suites are interpreted to represent the roots of magmatic arc systems. All units are cut or overlain by coeval intrusive and extrusive rhyolite, dacite, and andesite of the Bom Jardim and Salustiano formations (1,900 – 1,853 Ma) and volcaniclastic rocks of the Aruri Formation (1,893 – 1,853 Ma). All intrusive units are subsequently intruded by the Maloquinha Suite alkaline granites (1,882 – 1,870 Ma).

The dominant rock types of the Tapajós district are the Parauari granites in the central-northwest, Creporizão granites in the southeast, and Salustiano and Aruri volcanic sequences in the east. The Maloquinha granite is widespread throughout the district.

Gold mineralization occurs regionally throughout the Tapajós district in almost all the rock types mentioned above. Most of the mineralized intrusive rock types are associated with significant artisanal mining in this district which align along a north-northwest trending lineament known as the Chico Torres Megashear or Tocantinzinho Trend (Santiago et al., 2013; Borgo et al., 2017; Biondi et al., 2018). This structural trend is represented by distinct topographic lineament on satellite images and is also visible on regional aeromagnetic maps as a linear magnetic anomaly.

Figure 7.1: Regional Geology Setting of the Tocantinzinho Gold Project



7.2 Property Geology

The Project occurs in predominantly multiphase granitic igneous rocks aged between 2007 to 1,979 Ma (Borgo et al. 2017), which are likely part of the older magmatic arcs of the Cuiú-Cuiú Complex (~2010 Ma) or the Creporizão Suite (~1,970 Ma; Santos et al. 2001). At the Tocantinzinho main deposit, quartz monzonite hosts mineralized phases of granitic intrusives, which are concentrated along a northwest-striking structure (Figure 7.2).

7.2.1 Quartz Monzonite

Fine- to medium-grained, greyish green to reddish quartz-monzonite comprises the country rock hosting the main granitic intrusive phases. The quartz monzonite is generally magnetic, and epidote often fills millimetre- scale fractures throughout. Fine grained, disseminated pyrite is common, but is not associated with gold mineralization.

7.2.2 Granitic Rocks

The Tocantinzinho deposit is hosted within complexly textured, strongly veined and altered granitic rocks that are localized within the northwest-striking fault zone that crosscuts the earlier quartz monzonite country rocks. These granitic rocks were likely emplaced during active sinistral, strike-slip faulting, as supported by their intrusive contacts and vein orientations (Juras et al., 2011; Borgo et al., 2017; Biondi et al., 2018).

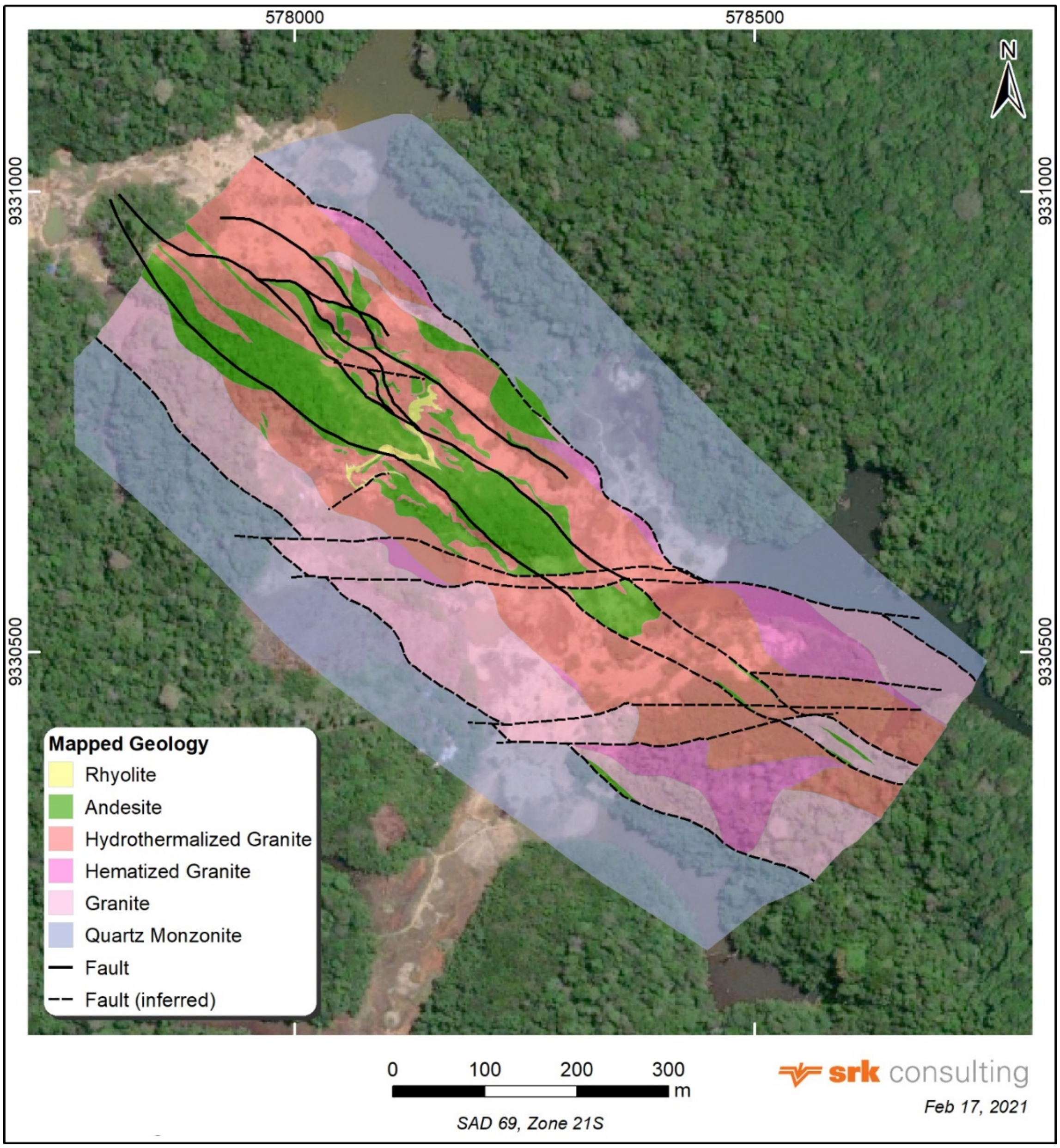
The granitic rocks are divided into three distinctly textured, fractured and hydrothermally altered granites within the main Tocantinzinho deposit area. The majority of gold mineralization is contained within two of these granite types, locally named the Salami and Smoky granites. The complexly textured mineralized granite host rocks are in contact with a barren granite with hematite, all hosted within a greater quartz monzonite intrusion.

The Salami granite is an alkali feldspar granite dominantly composed of potassium feldspar, quartz and albite. The Smoky granite instead has plagioclase feldspar present and is considered a granite sensu stricto. The plagioclase within the Smoky granite commonly displays crystal zoning differentiated by sericite with lesser chlorite and calcite alteration, particularly in the feldspar core. This likely reflects original crystal zoning in the plagioclase, indicating sodium-rich rims and calcium-rich cores. The Salami and Smoky granites have gradational contacts and locally transition into syenite and quartz syenite.

The mineralized granitic rocks contain highly variable textures including pegmatite and aplite, and textures indicative of the magmatic-hydrothermal transition such as miarolitic cavities, interconnected miarolitic cavities and unidirectional solidification textures.

The barren granite is a hematite-bearing, pink to red medium- to coarse-grained equigranular phase. The plagioclase feldspar within this unit is commonly altered to sericite. Any primary biotite is generally strongly chloritized, and the hematite present is usually replacing primary magnetite. The granite displays no sign of penetrative deformation or brecciation.

Figure 7.2: Local Geology Setting of the Tocantinzinho Gold Project



7.2.3 Andesite

A large andesite body intrudes the mineralized zone. This unit forms an upward flaring cap over the main mineralized zone. It varies from 50 to 80 m wide at surface and has a vertical dimension of approximately 50 m. Below this are a series of narrow andesite dykes interpreted as feeder dykes to the larger andesite body. The unit is strongly altered overall, with intense carbonate, chlorite and sericite alteration throughout. Carbonate ± chlorite commonly fill millimeter-scale fractures.

Gold mineralization is generally absent in the andesite, however anomalous gold values are associated with quartz-sulphide veins along its contacts.

7.2.4 Rhyolite Dykes

Rhyolite dykes occur in the central portion of the deposit, both exposed at surface and in the drill core. Exposures indicate that the rhyolite truncates all other rocks, including the andesite. The rhyolite is composed of a cream to light green coloured aphanitic groundmass with rare millimetre-scale quartz and potassium-feldspar phenocrysts. The dykes are typically 1 to 5 m wide and are relatively barren with respect to gold mineralization, however anomalous values ranging from 0.1 to 0.2 g Au/t have been detected in zones of intense veining.

7.3 Mineralization

The Tocantinzinho deposit forms a sub-vertical, northwest-trending elongate body approximately 900 m long by 150 to 200 m wide. It has been drilled to approximately 450 m depth and remains open at depth.

The gold mineralization is bound by two structural zones which mark the contact with the surrounding barren granite and monzonite rocks. The structural corridor represents an outer geological constraint on the mineralization. The andesitic intrusive body close to surface is also largely unmineralized and is therefore an internal constraint on the mineralization.

The grade distribution is similar in the both the Smoky and Salami granitic subunits. The green-grey colour of the Smoky granite is due the alteration of plagioclase to sericite with lesser chlorite, calcite and pyrite particularly in the core of the feldspar. Salami mineralized granites are distinctively bright red due to hematite alteration (Figure 7.3). Staining for potassium feldspar using cobaltinitrite has revealed that both potassic and sodic feldspar are host to the hematitic dusting in feldspar and therefore the red-pink colour is not indicative of potassic alteration. Anastomosing veinlets are common and similar in scale to those in the Smoky granite but are generally filled with a distinct black chlorite and lesser sericite-calcite-pyrite-quartz. Quartz textures in the Salami and Smoky granites commonly have curvilinear grain boundaries with feldspar. Chlorite-sericite-calcite veins are typically wormy rather than planar and appear to cut feldspar but develop along the margins of the quartz grains suggesting this was a relatively late magmatic-hydrothermal phase. Contacts are diffuse between smoky and salami granites and a complete gradation exists between the two units.

Figure 7.3: Gold Mineralization at the Tocantinzinho Deposit

Drill-hole TOC 186: Smoky granite from 158.83 to 162.22 m and Salami granite from 162.22 to 164.50 m; sample intervals were assayed at more than 1 g Au/t.
Source: SRK (2020)

Pyrite is the main sulfide phase at the Tocantinzinho deposit and commonly contains inclusions of chalcopyrite and pyrrhotite. The presence of pyrrhotite is genetically significant because it indicates that the mineralization may have at least formed locally under reduced conditions despite the abundance of hematite in the outer red granite. Gold grains were observed in close association with pyrite along grain boundaries, within fractures and locally as inclusions within pyrite. A white-grey mineral was also observed associated with gold and bismuthinite or bismuth. Multi-element data for samples with anomalous gold (> 0.1 g/t) show a strong correlation pair between gold and bismuth (0.87 correlation coefficient). Figure 7.4 shows Smoky granite with mineralization associated with quartz veins, sulphide and chlorite alteration.

Figure 7.4: Tocantinzinho Mineralization in Smoky Granite

Drill Hole TOC 08-93, 238.50 to 240.50 m: granite with Smoky texture, mineralization associated with quartz vein, sulphide and chlorite alteration; sample interval was assayed at 11.028 g Au/t.
Source: SRK (2020)

7.4 Exploration Targets

7.4.1 Santa Patricia

The Santa Patricia target occurs 2.5 km west of the main Tocantinzinho deposit and is defined as an over 8-km northwest striking copper anomaly (> 20 ppm) based on soil geochemistry that coincides with a regional magnetic lineament. This anomaly was drill tested by 16 widely spaced core boreholes in 2012 (4 boreholes) and 2014 (12 boreholes) to depths of between approximately 200 and 380 m.

The Santa Patrícia target geology is mainly comprised of intrusive rocks, including mafic to intermediate and alkali-granitoid plutonic suites, of which are transected by late-stage fine-grained andesite and rhyolite dykes.

The mafic to intermediate suite is composed of medium-grained equigranular to porphyritic granodiorite and quartz monzonite, lesser diorite to monzodiorite and localized gabbro. The alkali-granitoid suite consists of granite to alkali feldspar granite transitional to Salami-textured syenite and quartz syenite. Late phases of this suite include micro-granite, aplite and pegmatite. The contacts between the two suites vary from gradational to crosscutting with the fractionated aplite-pegmatite phases typically as the youngest. Evidence of magma mingling and mixing textures between granite, granodiorite and diorite suggests that the two suites may be coeval.

Copper-molybdenum mineralization occurs within veins that are hosted by all of the above igneous rocks, including the late andesite. The highest grades occur in zones of strong stockwork vein and veinlet intensity, with increased copper grades with depth (as in boreholes TOC272 and TOC275).

A paragenetic framework has been established that comprises six main vein and alteration stages, including:

- The earliest phase as white quartz ± potassium-feldspar stockwork veining that are typically planar, and less commonly irregular, with associated pink-red potassium-feldspar alteration occurring as vein halos to wider pervasive zones.
- The main copper-molybdenum stage associated with second generation veins composed of magnetite-muscovite-quartz-chalcopyrite-pyrite-molybdenite±hematite within a strong pervasive coarse muscovite alteration, which is greisen-like in places.
- Localized laminated quartz-molybdenite veins with silicification.
- Pyrite-sericite-calcite veinlets and alteration.
- Distal propylitic epidote-calcite-chlorite-pyrite veins and alteration.
- Late K-feldspar-calcite veinlets that crosscut all vein stages.

The Santa Patricia copper molybdenum system is mostly devoid of gold. Where localized gold grades occur (approximately 0.15 g Au/t), the mineralization style is more similar to that within the Salami granite style of Tocantinzinho.

7.4.2 KRB

Rock types at the KRB target are comprised of a variety of intrusive rocks from alkali feldspar granite, syenogranite and monzogranite to quartz monzonite, granodiorite and monzonite. These sequences are intruded by microgranite, pegmatite, aplite, as well as late dykes of felsic to intermediate composition.

The understanding of mineralization controls at KRB is at a conceptual stage considering the limited exploration campaign carried out at this target to date. Gold appears to be associated with quartz veining and pyrite along the margins of felsic to intermediate dykes.

8 DEPOSIT TYPES

The following subsections are extracted from Eldorado Gold Corporation (2019).

8.1 Tocantinzinho Deposit

The Tocantinzinho deposit is best classified as a granite-hosted, intrusion-related gold deposit. Intrusion related gold deposits were defined by Thompson et al. (1998) with the following characteristics:

- Deposits hosted within or zoned proximal to intermediate to felsic granitic rocks.
- Intrusions are typically moderately reduced to moderately oxidized (ilmenite through to magnetite series) I-type granitic rocks.
- The associated pathfinder elements are typically Bi, Te, Mo, W in the core of the intrusion system, zoning outward to distal As, Sb, Pb and Zn.
- The deposits have a range in styles including sheeted, breccia, stockwork, flat-vein and disseminated to greisen that is controlled by proximity to intrusions, depth of emplacement and structural controls on intrusions.
- Mineralization is coeval with the related intrusions demonstrated through zonation with respect to the causative intrusion and mineralized magmatic-hydrothermal transition textures. These may include miarolitic cavities, vein dykes, unidirectional solidification textures, brain rock, and granite-facies control on gold distribution. In addition, age dating in global examples has confirmed synchroneity between intrusions and mineralization.
- Mineralization is typically characterized by reduced, low sulfide (< 2%) ore assemblages including pyrite, pyrrhotite and arsenopyrite with magnetite less common and hematite rare. Proximal gold is typically high fineness and paragenetically related to Bi ± Te.
- Alteration is usually more limited than in typical porphyry environments, and characterized by early, high temperature quartz-feldspar (both potassic and sodic) alteration intimately associated with magmatic-hydrothermal transition textures that evolve to lower temperature white mica / sericite-carbonate-chlorite-quartz alteration and veining typically associated with the main mineralization stage.
- The mineralization formed from H_2O-CO_2 ± salt magmatic fluids.
- The majority of known deposits are Phanerozoic in age and formed in continental arc to back arc settings.

The Tocantinzinho gold deposit has many of the characteristics above including:

- Fractionated granite host rock package (quartz monzonite, syenite, alkali feldspar granite, granite and aplite).
- Mineralized magmatic-hydrothermal transition textures including unidirectional solidification textures, interconnected miarolitic textures, rapid grain size variations from pegmatite to aplite and vein dykes, and granite facies control on gold-distribution (Salami and Smoky).
- Alteration assemblages with early (transitional magmatic-hydrothermal) potassic-sodic feldspar through to silicification and pervasive to vein controlled quartz-sericite-chlorite-calcite.
- Fluid inclusions contain H_2O-CO_2 ± salt.

However, some features of the Tocantinzinho gold deposit are not typical of an intrusion related gold system including:

- Intrusion oxidation state: the distinctly red hematitic dusting to the feldspar suggests that the intrusions are more oxidized than typical intrusion related gold systems. However, the exact timing of the hematite is debatable. Late pink-red feldspar veins are common and strong hematitic alteration is associated with late stage breccias and fractures. Primary magnetite was clearly replaced by hematite in the hematitic granite on the margins of the deposit, and primary titanite-ilmenite is commonly observed in the Tocantinzinho granitic rocks indicative of moderately oxidized to moderately reduced primary magma compositions.
- Oxidation state of the mineralization: the red-hematitic nature of the feldspar suggests more oxidized conditions for mineralization formation; however, the presence of pyrrhotite inclusions within pyrite and the Au-Bi association is not compatible with a strongly oxidized ore fluid.
- Age and timing: Borgo et al. (2017) presented U-Pb zircon dating on the Tocantinzinho intrusions and surrounding host rocks. The former is older (~2,007 to 1,997 Ma) than the main Tocantinzinho granitic suite, although in part overlap with error. The Tocantinzinho granitic rocks and mineralization are cut by late andesite and rhyolite dykes, and age data on the dykes (~1,996 to 1,992 Ma) overlap and are within error of the main host granite phases (~1,993 to 1,979 Ma). The age of the intrusions, and by inference the mineralization, is Paleoproterozoic which is unusual for intrusion related gold deposits and suggests that Tocantinzinho may be one of the oldest examples of this deposit type.

Biondi et al. (2018) further discuss the classification of Tocantinzinho comparing it to both reduced intrusion related gold and oxidized intrusion related gold (cf. Robert et al. 2007), and to Au-rich porphyry Cu deposits and orogenic Au deposits. The lack of Cu and extensive alteration associated with porphyry systems is inconsistent with that deposit group; however, Santa Patricia certainly has characteristics of porphyry Cu-Mo systems.

There has been much debate regarding the orogenic and intrusion related gold classification (Goldfarb et al., 2005). Tocantinzinho does not appear to fit an orogenic classification for several reasons despite the deposit and related intrusion occurring in a major regional structure. At a regional scale, the deposit is not hosted in a metamorphic terrane and at the deposit scale, the mineralization is controlled by granite-facies development and related veins and alteration rather than fault-related features. Furthermore, textural timing relationships and age data support a coeval timing for intrusions and mineralization.

8.2 Santa Patricia Deposit

Santa Patricia is best described as a porphyry Cu-Mo system based on the following features:

- Highest Cu-Mo grades are associated with the most intense, multi-stage stockwork vein zones.
- The veins and alteration evolve from high (potassium K-feldspar-muscovite) to low temperature (sericite-calcite).
- The alteration is broadly zoned from K-feldspar and muscovite-rich alteration associated with the most intense stockwork centers through to an outer pyrite-sericite alteration to a distal propylitic alteration.

A porphyry exploration model offers potential upside to Santa Patricia. It is likely that the centre(s) of the potential porphyry system(s) have not been discovered. Evidence for this includes the lack of coeval porphyry related intrusions and the absence of abundant, early, irregular A-veins. Additionally, in the well mineralized zones, Cu grades appear to increase with depth, and drilling to date has been limited to the upper 300 m.

9 EXPLORATION

The following information is largely summarized from Pincock Allen and Holt (2006) and Eldorado Gold Corporation (2019).

9.1 Brazauro Resources Corporation (2004-2008)

In 2003, Brazauro acquired the properties covering the Tocantinzinho mineralization through its Brazilian subsidiary, Jaguar Resources do Brasil Ltda. Between 2004 and 2008, work by Brazauro included channel and soil sampling, and core and power auger drilling.

9.1.1 Channel and Soil Sampling

Over 500 channel / chip and soil samples were collected by BRM across the Eldorado Gold Project.

9.1.2 Geophysical Surveys

In 2005, BRM completed a ground magnetic survey including a regional pass with survey lines spaced 400 metres apart over the broader Project and an infill pass spaced at 100 metres centred on Tocantinzinho.

BRM contracted Reconsult Geofísica (Reconsult) from São Paulo, Brazil to process and interpret the raw ground magnetic data and as well as other geophysical airborne data collected by FUGRO company over the Tocantinzinho area.

It was suggested by Reconsult that the gold mineralization is likely associated with the main Tocantinzinho trend structure with an azimuth of 300 degrees and is truncated by magnetic rock to the southwest. Based on the magnetic and radiometric data, Reconsult considered that there is a strong potential for continuation of mineralization to the NW.

9.2 Eldorado Gold Corporation (2008 to 2021)

In September 2008, Eldorado became the operator of a joint venture with Brazauro and continued exploration with a focus on the main Tocantinzinho area until 2010. Exploration activities between 2004 and 2010 established Tocantinzinho as a significant gold discovery and prompted the acquisition of Brazauro by Eldorado. Exploration activities were put on standby from April 2010 onwards, until Eldorado completed the acquisition of all the issued and outstanding securities of Brazauro, which was finalized in July 2010.

9.2.1 Channel and Soil Sampling

Eldorado conducted soil sampling campaigns from mid-2009 to early 2015. The initial focus between 2009 and 2011 was on the extensions of the main deposit area trend and adjacent areas. Further extensions of the main mineralized trend as well as the testing of parallel trends were carried out between 2012 and 2015.

Between 2009 and 2010, Eldorado collected 2,604 soil, 46 channel / chip and 100 dump samples. The surficial volume tested by these early surveys has now been almost completed excavated by the garimpeiros.

Soil samples were collected at 50 m intervals, using a hand auger with half-metre depth. Line spacing varied by area and campaign. Targets along-strike southeast of the Tocantinzinho deposit were tested with 100 m spaced lines. Targets along-strike northwest of the deposit was covered with 200 m spaced lines. The far southeast extension was tested with 400 m spaced lines. Other areas were initially sampled on 800 m spaced lines, progressively infilled to 100 m in case of positive results. The latter were generally considered to be spatially consistent values above 100 ppb gold.

9.2.2 Topographic Surveys

In 2010, Eldorado carried out a detailed topographic survey of the Tocantinzinho deposit area, covering 2.5 km^2. This included the main pit area and other areas adjacent to the mineralized zone. The coordinate system was calculated based on one official and five implemented geodesic points.

In September 2010, a topographic aerial laser survey was completed of the Project area by Geoid Ltda. (Geoid), from Belo Horizonte, Brazil. This covered an area of 53 km^2, including the main deposit and potential tailings dam and plant site areas. The survey was completed with a contour interval of 1 metre and an accuracy of 0.15 centimetres in both horizontal and vertical directions.

9.2.3 Geophysical Surveys

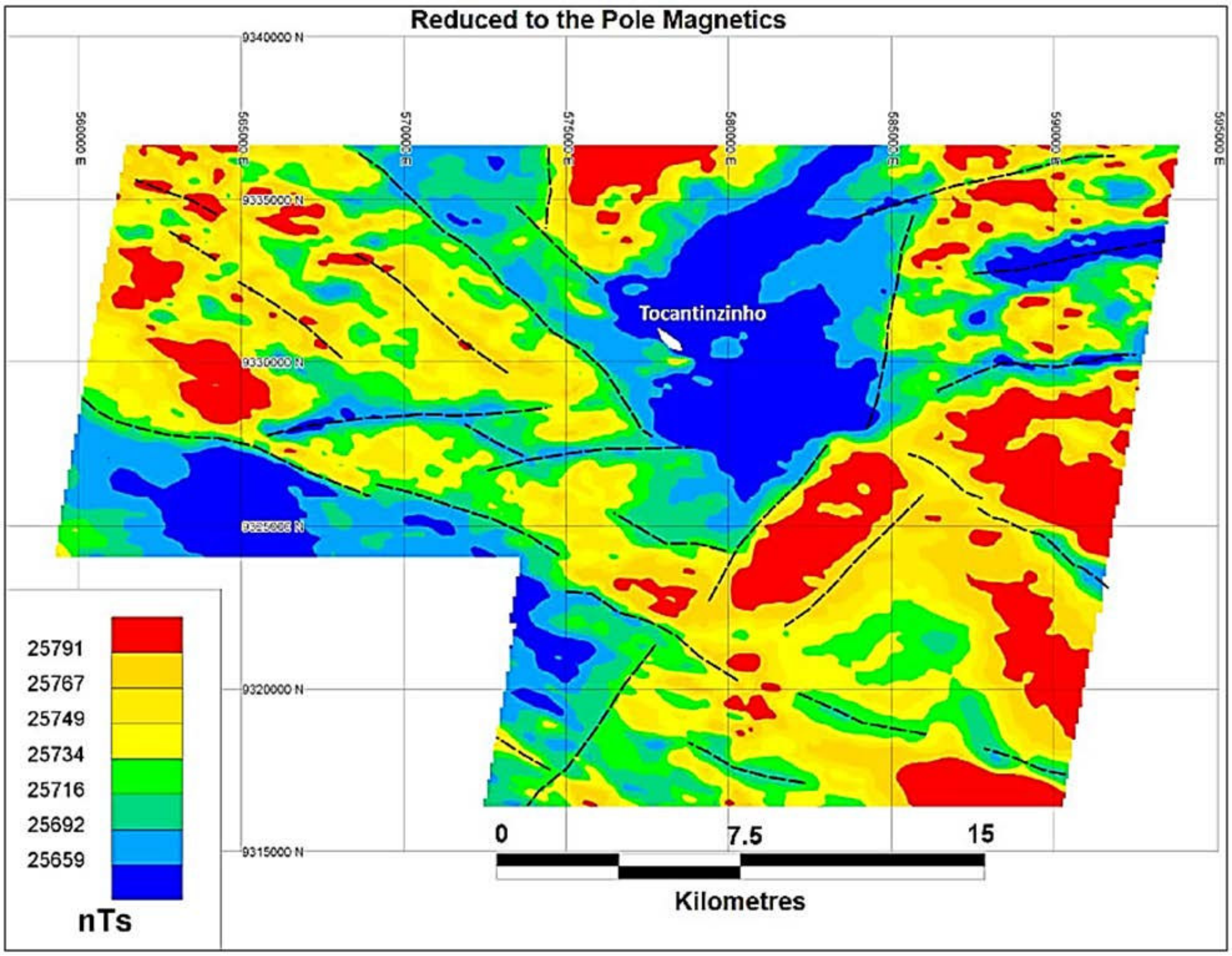
9.2.3.1 Induced Polarization Survey

During late 2010, Eldorado completed an induced polarization (IP) geophysical survey involving a total of 45 line-kilometres covering areas along the Tocantinzinho trend to the northwest and southeast of the deposit. Results of the IP survey showed that geology and structural breaks are both defined well on the chargeability and resistivity maps, however these methods were not as effective in detecting Tocantinzinho-style gold mineralization.

9.2.3.2 Ground Magnetic Survey

In 2011, Eldorado contracted S.J. Geophysics (S.J.) to review the magnetic airborne data collected by Brazauro in 2005. The interpretation focused on establishing northwest and northeast structural trends and positioned Tocantinzinho inside a zone of low magnetic susceptibility (Figure 9.1).

Figure 9.1: Reduced to Pole Magnetic Survey Interpretation Completed by S.J. Geophysics (2011)



Source: Eldorado Gold Corporation (2019)

10 DRILLING

Drilling completed on the Tocantinzinho Gold Project after 1998 is tabulated in Table 10.1. Brazauro and Eldorado drilled a combined total of 296 core boreholes (approximately 82,805 m) on the Tocantinzinho Gold Project between 2004 and 2021. Several auger and reverse circulation boreholes have also been drilled in the Tocantinzinho Gold Project; however, as this drilling data was not used for the resource estimation they are not discussed in detail in this section.

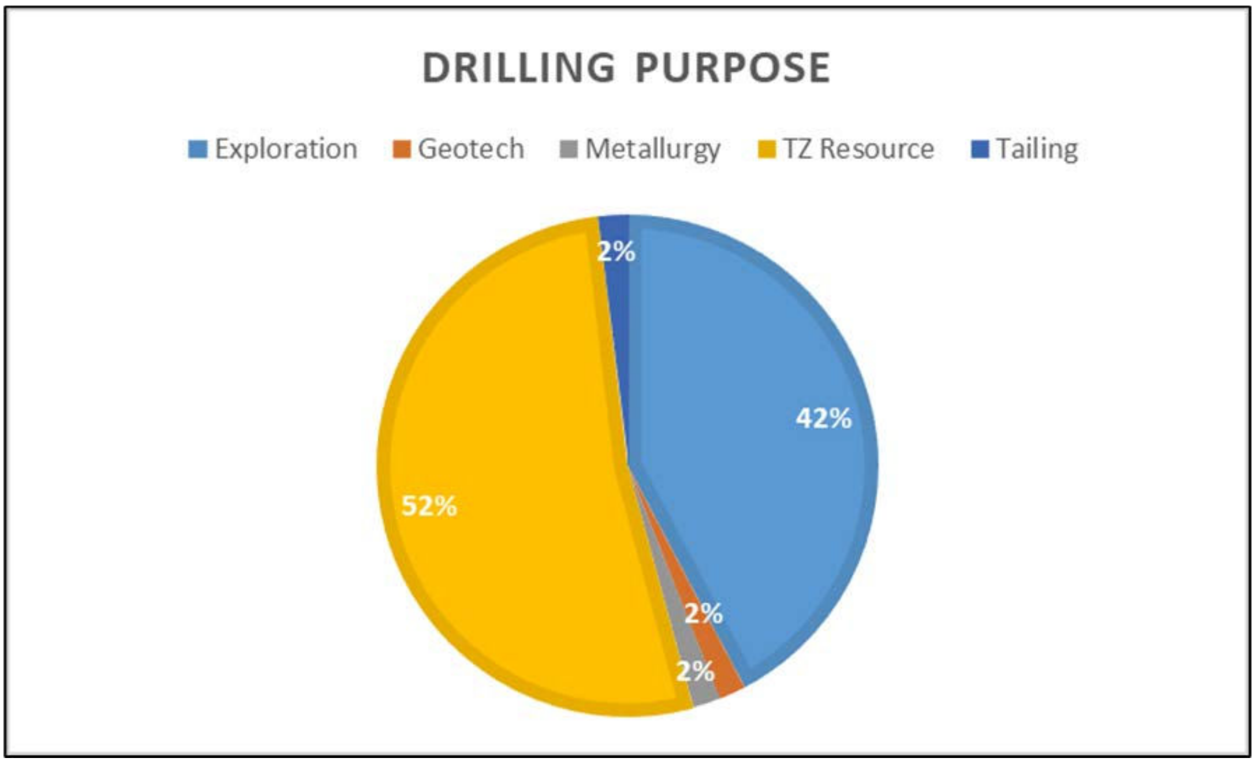
The distribution of drilling by purpose is shown in Figure 10.1. A map showing the distribution of core drilling collars is provided in Figure 10.2.

Table 10.1: Summary of Drilling Completed on the Tocantinzinho Gold Project

Company	Period	Target Area	Purpose	Power Auger		Reverse Circulation		Core		Tailing	
				No. of Holes	Length (m)	No. of Holes	Length (m)	No. of Holes	Length (m)	No. of Holes	Length (m)
Altoro	1998	TZ	Exploration	87	1,318	-	-	-	-		
	1999	TZ	Exploration	58	503	-	-	-	-		
Brazauro	2004	TZ	Mineral Resources	-	-	-	-	17	4,197		
		TZ	Exploration	-	-	-	-	3	496		
	2005	TZ	Mineral Resources	-	-	-	-	14	3,759		
	2006	TZ	Exploration	-	-	-	-	2	489		
		TZ	Mineral Resources	-	-	-	-	10	2,533		
	2007	TZ	Exploration	-	-	-	-	14	2,516		
		TZ	Mineral Resources	-	-	-	-	11	3,247		
	2008	TZ	Mineral Resources	-	-	-	-	26	8,398		
	2008	TZ	Exploration	106	934	-	-		-		
Eldorado	2008	TZ	Mineral Resources	-	-	-	-	11	3,518		
	2009	TZ	Mineral Resources	-	-	-	-	47	14,633		
		TZ	Metallurgy	-	-	-	-	6	1,490		
		TZ	Exploration	112	416	11	4,693	-	-		
	2010	TZ	Mineral Resources	-	-	-	-	22	4,754		
		TZ	Exploration	-	-	8	3,759	21	5,705		

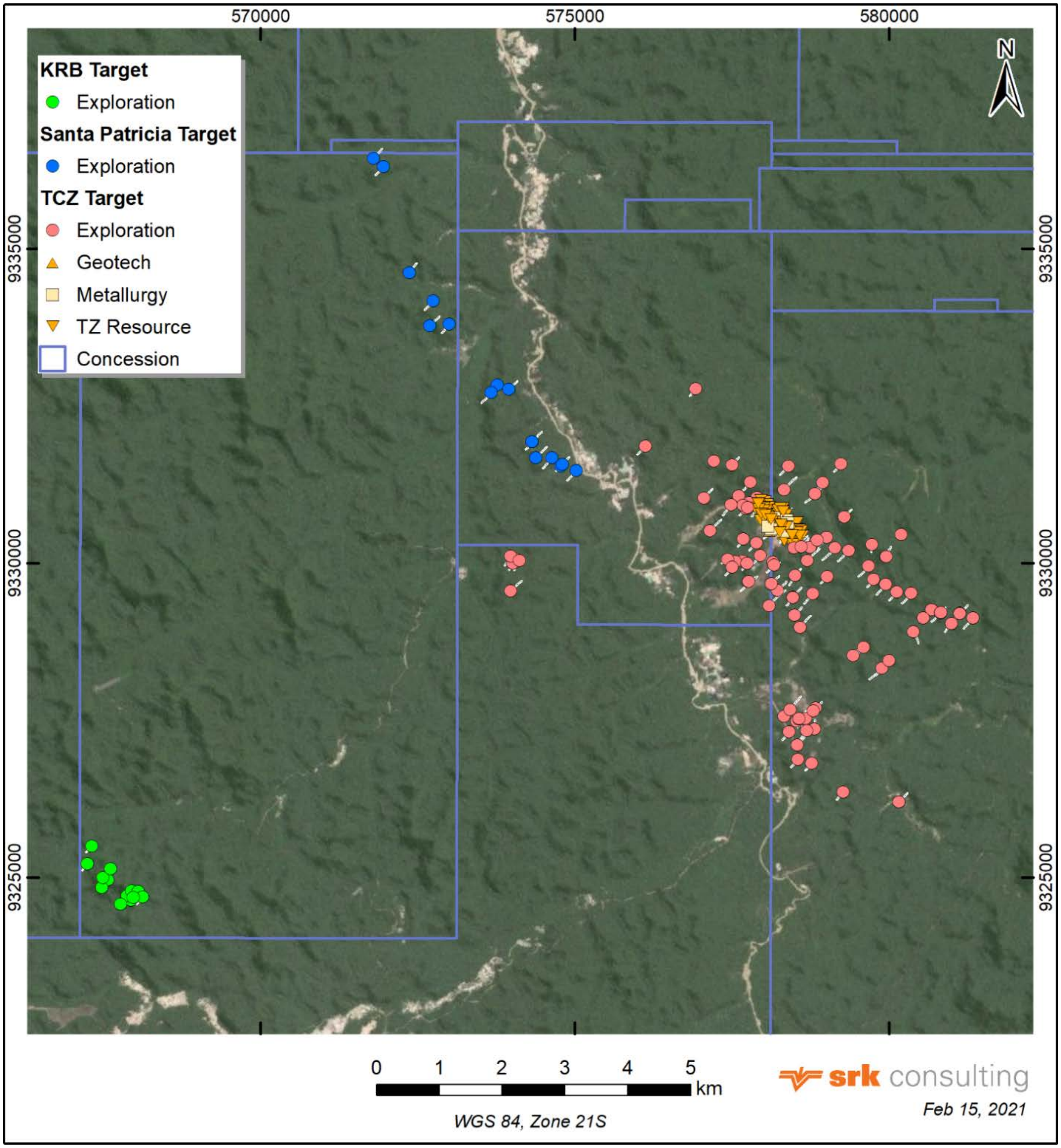
Company	Period	Target Area	Purpose	Power Auger		Reverse Circulation		Core		Tailing	
				No. of Holes	Length (m)	No. of Holes	Length (m)	No. of Holes	Length (m)	No. of Holes	Length (m)
Eldorado	2010	Santa Patrícia	Exploration	-	-	-	-	4	1,311		
		TZ	Geotechnical	-	-	-	-	6	1,784		
	2011	TZ	Exploration	-	-	-	-	56	17,316		
		TZ	Geotechnical							75	946
	2014	Santa Patrícia	Exploration	-	-	-	-	12	3,594		
		TZ	Tailing							80	648
	2015	KRB	Exploration	-	-	-	-	14	3,065		
Total				**363**	**3,171**	**19**	**8,452**	**296**	**82,806**	**155**	**1,594**

Figure 10.1: Drilling Data (Length) by Purpose



Source: SRK (2021)

Figure 10.2: Map Showing Core Drilling Collar Locations at the Tocantinzinho Gold Project



10.1 Brazauro Resources Corporation (2004-2008)

Based on results of geochemical soil sampling in early 2004, Brazauro (through its Brazilian subsidiary Jaguar Resources do Brazil Ltda.) initiated an exploratory drilling program of 20 core boreholes with an average length of 227 m per borehole. Drilling was continued until 2008, with a total of 25,635 m on 97 core boreholes completed on the Project. Additionally, a total of 106 power auger boreholes (934 m) were completed in 2008 for exploration purposes.

10.1.1 Core Drilling

Brazauro commissioned Kluane International Drilling, Inc. (Kluane) to undertake core drilling. Kluane used a light weight portable Hydrocore Gopher all hydraulic drill rig utilizing wire-line methodology for this purpose. During the core drilling, the saprolite and the weathered rock were drilled using NTW diameter which was reduced to BTW diameter when fresh rock was reached.

Borehole collar surveys were carried out using a total station instrument. Brazauro completed down hole surveys at intervals that varied from 15 to 376 m using a Reflex FlexIt tool. Only Boreholes TOC 07-47 (341 m) and TOC 07-48 (329 m) which were considered for resource estimation, did not have downhole surveys information.

After each shift, the drill core was retrieved from the drill site and brought to the camp. The core was placed on a tow box vertical stand, photographed and logged for geotechnical information including percent recovery, rock quality designation (RQD), joint frequency and condition, degree of breakage and weathering / alteration.

A geologist then completed a summary log of the core including lithology, alteration type and grade, texture, structure, presence of sulphides as well as a brief description of important features. During logging procedure, the geologist measured and marked the intervals to be sampled. Samples were typically 2 m in length, respecting the geological contacts.

The core was marked with a continuous linear cutting line before being sawn in half lengthwise. Both halves of the core were placed back into the core-box where the geologist recorded a detailed description of each sample including lithology, veining, alteration and mineralogy. Sample intervals which contained visible gold were marked to be analyzed by screened metallic assay at the laboratory. The rock saw was flushed regularly with fresh water to minimize contamination.

Trained geotechnicians then placed half of the core into sample bags and clearly marked the sample identification on the core box, including the interval footages and sample number. The samples were consistently taken from the same side of the cut line. The bagged sample was marked, tagged and sealed for shipping to the laboratory. Groups of bagged samples were placed in larger sacks that were marked by batch, indicating sample numbers.

All information was recorded on handwritten logs, and all logged information was entered into an Acquire digital database. When analytical results were received, they were typed into the core log sheets, at which points the logs were completed.

10.2 Eldorado Gold Corporation (2008 to 2021)

Eldorado drilled a total of 199 core boreholes (57,170 m) at the Tocantinzinho Gold Project between 2008 and 2015, since becoming the project operator. Additional to the core drilling, Eldorado also completed 112 power auger (416 m) and 11 reverse circulation (4,693 m) in 2009, and 8 reverse circulation boreholes (3,759 m) focused on exploration outside of the main deposit area in 2010. In August 2010, Eldorado commissioned 6 oriented core boreholes (1,784 m) drilled for geotechnical purposes, for which Golder and Associates Inc. (Golder) were contracted to complete the geotechnical logging.

In 2011, Eldorado also drilled 75 boreholes in the tailings dam (946 m) and in 2014, another drill campaign in the tailings dam for 80 boreholes (648 m) was undertaken.

Eldorado has not undertaken any further drilling on the Tocantinzinho Project since 2015.

10.2.1 Drilling Procedures and Methodology

10.2.1.1 Core Drilling

The drilling procedures utilized by Eldorado largely mirrored that of Brazauro. Core drilling was undertaken by Kluane using wire-line methodology by a light weight portable Hydrocore Gopher all hydraulic drill rig. Advancements in drilling equipment allowed for Eldorado to use more powerful drill rigs, making it possible to drill deeper and produce wider varieties of core in addition to BTW, including HQ and NTW sizes.

Borehole collar surveys were carried out using a total station instrument. Downhole surveys were completed in intervals of 50 or 60 m using ReflexIt and Relfex EZ shot instruments.

Boreholes TOC 09-111 (230 m) and TOC 09-135 (64 m) were considered for resource estimation but were not surveyed. Geotechnical core boreholes were oriented using an ACT Reflex instrument.

10.2.1.2 Tailing Drilling

Borehole collar surveys were not carried out for the tailing boreholes. These boreholes vary from 2 to 40 m in length and thus were not surveyed

10.2.1.3 Reverse Circulation Drilling

The following information is sourced from the Eldorado Report (2011).

From October 2009 to February 2010, a reverse circulation drilling program was carried out with 19 holes totaling 8,452 m. All reverse circulation boreholes were drilled for exploration purposes in areas adjacent to and surrounding the Tocantinzinho main deposit.

No deviation measurements were made for reverse circulation boreholes. Reverse circulation chip samples were logged in a similar manner as the core and stored at the project core house. The samples were collected in 2 metre intervals and split to get a 4.0 kg sample. These data were not used for resource estimation.

10.2.1.4 Power Auger Drilling

Eldorado completed a power auger drilling program in the Tocantinzinho main pit area involving 112 boreholes (416 m). This drilling method was used due to the easy mobility, operation and maintenance by local personnel. Power auger holes were logged according to standard operating procedures established for the Tocantinzinho Project and sampled in 1 metre intervals. Boreholes were stopped upon reaching the water level, with an average hole length of 3.7 m.

Power auger boreholes can only be drilled vertically, which is subparallel to the geological features of the project. Therefore, samples obtained through power auger drilling were not able to be collected across geological features. The depth limitation of the power augers also did not allow for the sampling of fresh rock.

10.2.2 Exploration Targets

The following has been modified from Eldorado Gold Corporation (2019).

On July 2013, Eldorado presented the Economic Exploitation Plan to the DNPM (now called ANM) to apply for the Mining Concession and Easement Concession at Tocantinzinho project. According to Brazilian mining regulations, service or exploration drilling is restricted inside the Mining Concession Application while the application is in process. As a result, drilling at the project shifted to tenement-wide exploration in the period between 2014 and 2015. The exploration drilling followed up on the spatially significant and coherent Cu-Mo soil anomaly at Santa Patrícia target and the gold anomaly at KRB target. The drilling restriction at Tocantinzinho was lifted with the granting of the Mining Concession in May 2018.

10.2.2.1 Santa Patricia

Diamond drilling at Santa Patricia was conducted over two phases: An initial phase in 2012 comprised 4 boreholes (1,311 m) and a second phase in 2014 consisting of 12 boreholes (3,594 m). These two (2) phases were purposed to test the 8-kilometre-long copper and molybdenum soil anomalies divided into four (4) clusters. The northern edge of the soil anomaly remains untested by drilling.

Grid spacing within the clustered drilling was approximately 200 m (Figure 10.2). True mineralization thickness is unknown. In the southern cluster, drill holes TOC266, TOC267, TOC272 and TOC275 had significant copper mineralization. In the two latter drill holes, the trend of increasing copper grades downhole is noteworthy. The southern cluster is approximately 3 km west-northwest of the main Tocantinzinho deposit.

Table 10.2: Santa Patricia Drilling Locations

Hole ID	Location X	Location Y	Location Z	Length (m)
TOC266	574,387	9,331,675	179.08	384.00
TOC267	574,638	9,331,674	169.26	354.00
TOC268	573,775	9,332,835	163.85	207.00
TOC269	573,675	9,332,713	171.03	366.00
TOC270	574,326	9,331,929	170.00	337.66
TOC271	574,326	9,331,929	170.00	250.87
TOC272	574,785	9,331,544	148.00	309.56
TOC273	574,808	9,331,571	148.00	231.78
TOC274	573,010	9,333,800	140.00	301.37
TOC275	575,030	9,331,472	150.00	344.65
TOC276	573,960	9,332,760	160.00	286.70
TOC277	572,755	9,334,172	150.00	306.53
TOC278	572,370	9,334,620	180.00	294.84
TOC279	572,697	9,333,770	180.00	330.43
TOC280	571,956	9,336,309	157.00	299.06
TOC281	571,804	9,336,437	192.00	300.43

10.2.2.2 KRB

Core drilling at KRB target was carried out in 2015 with a total of 14 boreholes (3,065 m) drilled. This campaign was designed to test the 2-kilometre-long soil gold anomaly with values above 0.08 g/t gold. This soil anomaly is located approximately 12 km southwest of the main Tocantinzinho deposit and is oriented in a parallel northwest-southeast trend.

Drilling focused on the zones of higher gold in soils and employed scissor-style drill hole orientations. True mineralization thickness is unknown. Drilling intersected two (2) separate NW-SE corridors, but grade continuity is poor over the 100 m spacing between sections. The northern edge of the soil anomaly remains untested by drilling.

10.3 G Mining Services (2021)

10.3.1 Tailing Drilling

In 2021, GMS contracted Geotec Projetos e Serviços from Divinópolis, Minas Gerais to resurvey the tailing collars boreholes using an RTK (Real-Time Kinematic) equipment. GMS was able to locate 74 collars tailing boreholes, which represents around 48 percent of the tailing borehole total.

10.4 Qualified Person Comments

The qualified person referenced information in the Tocantinzinho reports (Pincock Allen and Holt 2006, NCL Brasil Ltda 2007 and 2010, Eldorado Gold Corporation 2011 and 2019) provided by GMS and also on observation made during the site visit in 2021 to review and document the drilling procedures undertaken by Brazauro and Eldorado. Core and reverse drilling, core handling, logging and the maintenance of the database were well managed by the respective operators and undertaken in accordance with well-defined procedures that meet generally industry standards. These procedures, which encompass every aspect of the exploration cycle from drilling to database management are considered appropriate for the project. The qualified person is not aware of any drilling, sampling or other factors that could materially impact the accuracy and reliability of the results discussed herein.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

Preparation of the power auger samples for Altoro was undertaken at Companhia de Pesquisa de Recursos Minerais (CPRM), a Brazilian state-owned geological survey company preparation facility in Itaituba, and then analyzed at Bondar Clegg's laboratory in Luiziânia, Goiás.

Between 2004 and 2005, Brazauro sent drilling samples to SGS Geosol Laboratory (SGS) in Itaituba, Pará. Prior to this, all core samples collected by Brazauro were shipped to the SGS sample preparation facility in Parauapebas town located in the Carajás District of Pará State.

Since the beginning of April 2006 and up to hole TOC-09-123, all samples were prepared at SGS Geosol in Itaituba, and then analyzed at the SGS analytical laboratory in Belo Horizonte, Minas Gerais State (NCL Brasil Ltda, 2007). Commencing from borehole TOC-09-124, samples were prepared and analyzed at ALS-Chemex Laboratory (ALS) in Belo Horizonte.

From borehole TOC-270 to TOC-281, Eldorado sent samples for preparation by ACME Laboratory (ACME) at Itaituba, and then analyzed at ACME in Santiago, Chile. From May 2009 to December 2011 (boreholes KRB-01 to KRB-14), samples were prepared and analyzed at ALS in Vespasiano. From July 2011 to January 2015, all assays were performed at ACME Analytical Laboratory in Santiago, Chile.

11.1 <u>Sample Preparation and Analyses During 1997 -2015</u>

11.1.1 <u>Altoro Gold Corporation (1997-1999)</u>

Sampling was performed by Altoro personnel. Samples were transported from the core shack to the sample preparation laboratory in Itaituba, Pará.

Power auger samples were transported to a processing facility near camp where a 2-kilogram assay sample, along with a 2-kilogram duplicate, were split by cone and quartering. Further sample preparation was completed at CPRM in Itaituba, from which 200 grams of sample pulp material were sent to Bondar Clegg's laboratory in Luiziânia, Goiás for analysis.

Bondar Clegg assayed all power auger samples. For the first 35 holes, the remaining material was panned and gold grains counted to estimate gold grade. The panning and counting technique were discontinued to primarily prevent the garimpeiros from locating anomalous areas of gold. Additionally, it was found that the counted grades compared to the assayed grades were generally underestimating the grade of the sample.

There was no information available regarding the preparation procedures for the soil samples. Sample pulps were subsequently shipped to Bondar Clegg laboratory in Vancouver, Canada for analysis where 50-grams pulp samples were assayed using an aqua regia digestion and inductively coupled argon plasma spectroscopy (ICP).

11.1.2 <u>Brazauro Resources Corporation (2004-2008)</u>

Core samples collected between 2004 and 2005 were bagged and sent by either bush plane or a combination of boat and truck transportation to SGS in Itaituba (NCL Brasil Ltda, 2007). Prior to April 2006, all core samples were shipped by truck from Itaituba to the SGS sample preparation facility in Parauapebas in the Carajás District of Pará State. In April 2006, SGS opened a preparation facility in Itaituba (NCL Brasil Ltda, 2007), and since then all core samples were sent there for preparation.

Core samples typically weighed between 2 to 3 kilograms. Samples were dried to 110 degrees Celsius before being crushed to less than 2 millimetres. A single kilogram subsample was split by a Jones splitter, and ground and passed through a 150-mesh screen, from which a 125-gram homogenized fraction was removed for analysis.

Prepared sample pulps were sent from Parauapebas to the SGS analytical laboratory in Belo Horizonte, Minas Gerais State (NCL Brasil Ltda, 2007). Samples were usually analyzed for gold only, with earlier geochemical analytical results that showed that there were insignificant concentrations of other metals.

The sample was analyzed for gold by fire assay with atomic absorption finish (SGS code FAA505) on a 50-gram aliquot. The remaining 75 grams (pulp reject) sample was stored in a marked envelope for reference.

Select core samples marked for visible gold were additionally analyzed by Inductively Coupled Plasma (ICP) multielement analysis. Both analyses were reported separately and a weighted average of the two results is assigned to the sample interval.

11.1.3 <u>Eldorado Gold Corporation (2008-2015)</u>

Until April 2009, Eldorado adopted the same sample preparation and analyses procedures as Brazauro.

From May 2009 to December 2011, samples were air shipped to the ALS in Vespasiano, Brazil. The sample preparation procedure ALS involved weighing and drying the sample before being crushed to 70% passing 2 millimetres. The sample was then riffle split to 1-kilogram and pulverised to 85% passing 75 µm (200-mesh screen).

Chemical analysis was undertaken at ALS Lima, Peru using 30 g sample for fire assay analysis (ALS Code Au-AA23). Select samples were also assayed for a suite of trace elements using aqua-regia digestion and inductively coupled plasma-emission spectroscopy (ICP-ES, ALS code ME-ICP41). A gravimetric finish was performed on fire assays returning more than 10 g/t gold. Samples with visible gold were submitted to a metallic screen analysis under ALS protocols (ALS Code: SCR21).

From July 2011 to January 2015 assays were undertaken at ACME Analytical Laboratory in Santiago, Chile. Sample preparation was completed at ACME Analytical Laboratory in Itaituba, Brazil, where they were crushed, split and pulverized until passing a 200-microns mesh. Gold was analyzed by fire assay with an atomic absorption finish (ACME Code G6/FA430). Samples were also assayed for a suite of trace elements using an aqua regia digestion and ICP-ES (ACME Code: D01/AQ300).

11.2 Specific Gravity Data

No specific gravity measurements were undertaken by Brazauro.

Eldorado undertook specific gravity determinations during the 2009 drilling program using a standard weight in water/weight in air methodology on the unweathered core over complete sample intervals. Specific gravity for weathered material was measured using the water displacement method. The database contains 734 measurements for mineralized and waste core intervals.

Unweathered sampled intervals were weighed before being submerged. In some cases, the rock sample was coated in paraffin and weighed again before submerging. The lithology and the depth of the selected piece of core was recorded in a spreadsheet with weights in air and water.

The weathered samples were wrapped in PVC film and weighed before being submerged in a beaker in water. The total water displacement is measured and recorded. The sample was then dried at 250 degrees Celsius for 90 minutes and weighted again. The lithology and the depth of the selected piece of core was recoded in a spreadsheet with the respective weights in air and in water. The lithology and the depth of the selected piece of core was recorded in a spreadsheet with wet and dry weights and total water displacement.

11.3 Sample Storage

Drill core is securely stored in the core sheds at Tocantinzinho Gold Project exploration camp.

11.4 Quality Assurance and Quality Control Programs

Quality control measures are typically set in place to ensure the reliability and trustworthiness of the exploration data. These measures include written field procedures and independent verifications of aspects such as drilling, surveying, sampling and assaying, data management and database integrity. Appropriate documentation of quality control measures and regular analysis of quality control data are important as a safeguard for project data and form the basis for the quality assurance program implemented during exploration.

Analytical control measures typically involve internal and external laboratory control measures implemented to monitor the precision and accuracy of the sampling, preparation and assaying process. They are also important to prevent sample mixing and to monitor the voluntary or inadvertent contamination of samples.

Assaying protocols typically involve regularly duplicating and replicating assays and inserting quality control samples to monitor the reliability of assaying results throughout the sampling and assaying process. Check assaying is normally performed as an additional test of the reliability of assaying results; it generally involves re-assaying a set number of sample rejects and pulps at a secondary umpire laboratory.

SRK analyzed the analytical quality control data accumulated by Brazauro and Eldorado for the Tocantinzinho gold deposit during the core drilling conducted between February 2004 and November 2015. Results of the quality control program are discussed in Section 12.3.2 and presented in Appendix A.

11.4.1 Brazauro Resources Corporation

Brazauro implemented external analytical control measures comprising of inserting field blanks and certified reference materials in all core sample batches submitted for assaying (Table 11.1).

The source of the field blank material used by Brazauro is unconfirmed. Brazauro used two commercially certified reference materials sourced from Rocklabs Ltd. (Rocklabs) in New Zealand (Table 11.1) (NCL Brasil Ltda, 2007). According to NCL Brasil Ltda (2007), control samples were inserted at a rate of 1 in 10 samples. In each batch of 50 samples, a low standard was inserted at the 10th and 20th sample, a blank at the 30th and one high standard at the 40th sample. Duplicate samples were not part of the quality control protocols used by Brazauro.

Table 11.1: Specifications of Control Samples Used by Brazauro Between April 2004 and October 2008

Reference Material	Source	Expected Value (g/t)	Std. Dev. (g/t)	Inserts	Comments
StdL	Rocklabs	1.805	-	495	Certificates not available
StdH	Rocklabs	8.367	-	148	
Total				**643**	

Considering the compiled database provided for SRK it seems that three (3) different types of low-grade standards were used for Brazauro, however the previous reports available mention just one. The certificates and commercial names for these standards were not made available to SRK.

11.4.2 Eldorado Gold Corporation

Eldorado implemented external analytical quality control measures comprising the regular insertion of control samples in all sample batches submitted for assaying, including field blanks and certified reference materials.

Field blank material was sourced locally from the Zé Bomba quarry located in the municipality of Itaituba, Pará, Brazil. On October 17, 2008, a total of 10 blank material samples were submitted to SGS laboratory for gold analysis, which consistently returned values below the detection limit (0.005 ppm). On March 31, 2009, Eldorado submitted another seven samples of the field blank material to SGS Geosol. On June 12, 2009, a total of 30 samples of the field blank material were collected and assayed at ALS to return values below the detection limit (0.005 ppm).

Eight commercially certified reference materials across a variety of gold grades were used by Eldorado on the Tocantinzinho Project sourced from Geostats Pty Ltd. (Geostats) of Australia and Rocklabs (Table 11.2).

According to NCL Brasil Ltda (2010) at every 10th sample one standard was inserted. Blanks were used at the rate of one blank at each 40 samples. Field duplicates (one quarter of a core) were inserted at each 15th sample or less. According to NCL Brasil Ltda (2010) coarse duplicates were submitted to the laboratory after return of the rejects of quartering but this information was not provided to SRK and could not be verified.

Table 11.2: Specifications of Eldorado Control Samples – September 2008 to June 2015

Reference Material	Source	Expected Value (g/t)	Std. Dev. (g/t)	Inserts
G901-13	Geostats	1.18	0.05	168
G903-7	Geostats	13.64	0.42	6
G903-9	Geostats	11.26	0.41	7
G907-2	Geostats	0.89	0.06	612
G907-6	Geostats	7.25	0.29	270
G907-8	Geostats	6.78	0.27	270
Si-42	RockLabs	1.761	0.054	146
Si-54	RockLabs	1.780	0.034	228
Total				**1,707**

11.5 Qualified Person Comments

In the opinion of the qualified person, Brazauro and Eldorado personnel used care in the collection and management of field and assaying procedures for exploration data. The sampling preparation, security and analytical procedures used by Brazauro and Eldorado are consistent with generally accepted industry best practices and are therefore adequate. The qualified person considers that the exploration data collected by Eldorado and Brazauro are of sufficient quality to support mineral resource evaluation.

12 DATA VERIFICATION

12.1 Verifications by Brazauro

Brazauro has undertaken database verifications and quality assurance and quality control programs (NCL Brasil Ltda, 2010).

Quality control measures implemented by Brazauro included independent verifications of assaying that involve external quality control measures on all sampling activities. Assaying protocols involve inserting certified quality control samples (field blanks and standards). Regular analysis of quality control data was undertaken by Brazauro. The criteria used for Brazauro accepting or not a batch is not available.

According to NCL Brasil Ltda (2007) in the first half of 2005, scoping level metallurgical tests were undertaken on four composite samples prepared from 58 individual samples of mineralized drill core. Each sample was fire assayed by SGS Lakefield Laboratory in Belo Horizonte, Brazil. Coarse rejects from each sample were then shipped to Lakefield Research Ltd. of Lakefield, Ontario, Canada and the assaying of each sample was repeated. Fire assay results from the two laboratories returned nearly identical values (NCL Brasil Ltda, 2007).

12.2 Verifications by Eldorado

Eldorado has undertaken database verifications and quality assurance and quality control programs (Eldorado Gold Corporation, 2019). The Tocantinzinho assay and geology database underwent a detailed comparison to the original source data and the core library by Eldorado staff. Eldorado re-surveyed collar data for older Brazauro boreholes to rectify incorrect topographic information. Discrepancies and inconsistencies in the geology data were checked against the core and corrected and incorporated into the current resource database (Eldorado, 2019).

Quality control measures were implemented by Eldorado which included independent verifications of assays at a secondary umpire laboratory and external quality control measures on all sampling activities. Assaying protocols involved replicating assays (field duplicates) and inserting certified quality control samples (field blanks and standards). Eldorado re-assayed a total of 676 samples collected by Brazauro between 2004 to 2008.

Regular analysis of quality control data was undertaken by Eldorado and used the following criteria for implementing corrective action:

- Automatic batch failure if the standard result is greater than the round-robin limit of three (3) standard deviations.
- Automatic batch failure if two (2) consecutive standard results are greater than two (2) standard deviations on the same side of the mean.
- Automatic batch failure if the field blank result is greater than 0.05 g/t gold.

If a batch failed based on the above criteria, the entire batch was re-assayed until it passed.

12.3 Verifications by Qualified Persons (SRK)

12.3.1 Site Visit

To comply with National Instrument 43-101 guidelines, a site visit to the Tocantinzinho gold deposit was completed between November 21 to 24, 2020 by Ms. Camila Passos and Mr. Thiago Toussaint from the SRK Brazil office, accompanied by Mr. Rafael Gradim and Mr. Danilo Ferreira from Brazauro.

The purpose of the site visit was to inspect the property and assess logistical aspects relating to conducting exploration work in the area. SRK was given full access to project data. While on site, SRK interviewed project personnel regarding the exploration strategy and field exploration procedures used by Brazauro and Eldorado.

Core drilling was not ongoing while SRK was on the site. However, SRK reviewed 3 boreholes from Tocantinzinho deposit (TOC 05-21, TOC 08-93 and TOC 186), one (1) borehole from Santa Patrícia target (TOC 275) and one (1) from KRB target (KRB01). SRK also inspected core storage facilities, and reviewed field procedures. Finally, SRK verified the collar location of ten (10) core boreholes using a handheld GPS receiver (Table 12.1). Typically, most of the results were within 4 metres in location East and within 9 metres in location South determined by Eldorado using a Total Station tool, suggesting that collar coordinates determined by Eldorado are suitable for resource estimation purposes.

Table 12.1: Verification Checks of Selected Collar Locations against the Project Database

Borehole ID	Eldorado Database		SRK Handheld GPS		Difference	
	Easting*	Southing*	Easting*	Southing*	Δ East	Δ South
TOC 04-11	578,380	9,330,679	578,380	9,330,675	0	-5
TOC 06-43	578,362	9,330,655	578,364	9,330,649	2	-6
TOC 07-66	577,905	9,331,038	577,903	9,331,036	-2	-2
TOC 08-75	578,327	9,330,502	578,329	9,330,494	2	-8
TOC 08-77	578,303	9,330,516	578,303	9,330,510	1	-6

Borehole ID	Eldorado Database		SRK Handheld GPS		Difference	
	Easting*	Southing*	Easting*	Southing*	Δ East	Δ South
TOC 09-140	578,300	9,330,521	578,300	9,330,515	-0	-6
PK08	577,918	9,331,025	577,905	9,331,034	-13	9
TOC 09-48TW	578,399	9,330,638	578,397	9,330,631	-2	-7
TOC 183	578,372	9,330,572	578,368	9,330,563	-4	-9
TOC 184	578,370	9,330,570	578,371	9,330,567	1	-3

**Note: SAD69 datum, Zone 21S.*

12.3.2 Database Verifications

The qualified person conducted a series of routine verifications to ensure the reliability of the electronic data used for resource estimation. This included checking the digital data against original assay certificates. More than 10% of the assay data were audited for accuracy against assay certificates, totalling 5,054 assay results from all diamond drill holes campaigns. Less than 0.12% (6 samples) presented inconsistencies.

SRK also checked all digital tailings assays against original assay certificates and found no errors.

12.3.3 Verifications of Analytical Quality Control Data

For this study, the qualified person analyzed all the analytical quality control data accumulated by Brazauro and Eldorado for the Tocantinzinho Project for all core and tailing drilling conducted between 2004 and 2015.

The qualified person was provided with an external analytical control dataset containing the assay results for the quality control samples for the Project in Microsoft Excel spreadsheets, accompanied by original laboratory pdf-format certificates. Control samples (field blanks and certified reference materials) were charted and summarized on time series plots to highlight the performance by the qualified person. Paired data (field duplicates) were charted and analyzed using bias charts, quantile-quantile, and relative precision plots.

The external analytical quality control data produced for the Tocantinzinho Gold Project are summarized in Table 12.2. The external quality control data generated for this project represent approximately 15% of the total number of samples assayed.

Table 12.2: Summary of Analytical Quality Control Data Generated by Brazauro and Eldorado on the Tocantinzinho Gold Project (2004-2015)

		Brazauro		Eldorado				Tota l	(%)
		Core	(%)	Core	(%)	Tailing s	(%)		
Sample Count		**13,545**		**28,625**		**1,434**		**43,604**	
Standards		**643**	**4.7%**	**1,707**	**6.0%**	**83**	**5.8%**	**2,433**	**5.6%**
	StdL	495						495	
	StdH	148						148	
	G901-13			168				168	
	G903-7			6				6	
	G903-9			7				7	
	G907-2			612		36		648	
	G907-6			270		5		275	
	G907-8			270		6		276	
	Si-42			146				146	
	Si-54			228		36		264	
Duplicates		**–**	**–**	**2,753**	**9.6%**	**48**	**3.3%**	**2,801**	**6.4%**
	Field Duplicates			2,753	9.6%	48		2,801	
Blanks		**548**	**4.0%**	**830**	**2.9%**	**42**	**2.9%**	**1,378**	**3.3%**
	Blank	548		830		42		1,378	
Total QC Samples		**1,191**	**8.8%**	**5,290**	**18.5 %**	**173**	**12.1%**	**6,481**	**15.4%**

In general, analyses of blank samples consistently yielded gold values near or below the detection limit of the laboratory used. The performance of blank samples assayed at SGS laboratory between 2004 and 2009 is acceptable with little to no sample contamination detected and less than 3% returning values above 10 times the detection limit. The performance of blank samples assayed at ALS and ACME laboratories between 2009 and 2015 is acceptable with no sample contamination detected.

According to NCL Brasil Ltda (2007), Brazauro used a total of two (2) certified standard reference material types, one (1) high grade standard and one (1) low grade standard. The qualified person verified the compiled database provided by GM and noted that three (3) different low-grade standards were used by Brazauro. The standards certificates were not available for review. Overall, the performance of these materials is acceptable with failure rates ranging from 5% to 12%.

Eldorado used a total of 8 certified standard reference material types. Overall, the performance of these materials is acceptable with failure rates ranging from 0% to 11%. Some standard materials less than 10 inserts due to discontinued use, and therefore the statistical performance of these material may be unevolved or undetermined.

Brazauro did not used duplicates in their analytical quality control program. When Eldorado became the operator of the Project, a total of 676 field duplicates samples from 2004 to 2008 were re-assayed. Paired assay data examined by Eldorado considering the Brazauro historical drilling shows that approximately 57% of the samples had HARD below 30%. Reproducibility of core assays from field material show a correlation coefficient of 0.80, indicating moderate to high variability between samples.

Eldorado also instituted the regular insertion of field duplicates for core drilling campaigns carried out between 2008 and 2015. Approximately 73% and 71% of the samples had HARD values below 30% with correlation coefficients of 0.66 and 0.93 for samples assayed in SGS laboratory in 2008 and 2009, respectively. For samples assayed in ALS laboratory in 2009 and 2010, between 67% and 69% of the samples had HARD values below 30% with correlation coefficients of 0.67 and 0.77, respectively.

The qualified person also reviewed the field duplicate results from 2011 to 2015 for ALS and ACME. For ALS, 76% to 78% of paired data had HARD values below 30%, with coefficients of variation varying from 0.69 to 0.95. For ACME, 72 to 80% of paired data had HARD values below 30%, and the coefficient of variation varies from 0.55 to 0.96.

There is no obvious evidence of systematic bias for the results of all paired field duplicate data. The poor reproducibility for paired field data indicates a moderate to high variability between duplicate samples and may be related to a moderate to strong nugget effect, which is not unexpected for this type of deposit.

12.4 SRK Comments

The qualified person undertook a detailed quality control review including the review of analytical quality control programs carried out by Brazauro from 2004 to 2008 and Eldorado from 2008 to 2015. The objective of this review was to verify the reliability of exploration data generated during this period to be used in the mineral resource estimation process.

Based on previous Tocantinzinho reports (NCL Brasil Ltda, 2007; NCL Brasil Ltda, 2010; Eldorado Gold Corporation, 2011; Eldorado Gold Corporation, 2019) and on the site visit undertaken by the qualified person, the qualified person believes that drilling, logging, core handling, core storage, and analytical quality control protocols used by Brazauro and Eldorado meet generally accepted industry best practices.

Overall, the qualified person considers analytical results from core sampling conducted between 2004 and 2015 at the Tocantinzinho Project to be globally sufficiently reliable for the purpose of mineral resource estimation. The data examined by the qualified person does not show any obvious evidence of analytical bias. Overall, SRK considers analytical results from core sampling conducted between 2004 and 2015 at the Tocantinzinho Project are globally sufficiently reliable for the purpose of resource estimation. The data examined by SRK do not show any obvious evidence of analytical bias.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

This section has largely been reproduced from the previous technical report on this property documented in Eldorado et al (2019) and provides a description of metallurgical test work, analysis and interpretation of the test work results completed from 2001 to 2017. A new metallurgical test work program was initiated at the end of 2021 to complete confirmatory test work but the results were not available at the time of writing this technical report. Previous test work results have been used for the process plant design criteria.

The following metallurgical test facilities were involved with previous test work programs:

- Hazen Research, Golden, CO, USA
- SGS Mineral Services, Lakefield, ON, Canada
- SGS Geosol Laboratories, Belo Horizonte, Brazil
- Wardell Armstrong International, Cornwall, UK
- FLSmidth Knelson, Langley, BC, Canada
- FLSmidth UK, United Kingdom
- Bruce Geller, Advanced Geologic Services, Lakewood, ON, Canada
- CyPlus GmbH, Hanau-Wolfgang, Germany
- Ralph Meyertons Consulting, Nederland, CO, USA
- Federal University of Minas Gerais, Minas Gerais, Brazil

13.2 Metallurgical Test Work

Metallurgical test work completed for the Project included the following tests:

- Ore variability in terms of lithology, gold head grade, sulfur head grade, depth, and sample blending
- Metallurgical test work for primary sulfide ore, gold bearing soil, saprolite, transitional and artisanal mining (garimpeiros) tailings
- Detailed chemical analyses of ore feeds, flotation concentrates and flotation tailings
- Ore mineralogy and characteristics assessment

- Comminution testing including Bond crushing, rod milling, and ball milling indices; SMC index, and abrasion index
- Whole ore cyanide leach and cyanide leach of flotation concentrates
- Flotation including batch rougher and cleaner, locked cycle, and pilot plant
- Gravity recoverable gold
- Thickening testing of ore feed, flotation concentrate, leached residue and flotation tailing
- Cyanide detoxification (several methods) and aging test work on tailings and effluent
- Environmental and geotechnical testing of residue

Pilot plant flotation and cyanide test work were completed by Wardell Armstrong International in UK. Gravity test work was completed by FLS Knelson in Canada and the tailings and cyanide destruction test work was latest carried out by SGS Mineral Services in Canada.

13.3 Composite and Sample Selection

The description of the samples as per the profile strata for individual and composite testing are summarized in Table 13.1.

Table 13.1: Description of Samples in Test Work Program

Sample Description	Feature
SMKG	Smoky Granite
SMIG	Salami Granite
TOP	Top Half Orebody
BOT	Bottom Half Orebody
ALL	Overall Composite
S1[1]	Top 100 m
S2	Middle 100 m
S3	Bottom 100 m
S4	Top 100 m
S5	Middle 100 m
S6	Bottom 100 m

Sample Description	Feature
S7	Low Gold Grade
S8	Average Gold Grade
S9	High Gold Grade
S10	Low Sulfur Grade
S11	Average Sulfur Grade
S12	High Sulfur Grade
S13	Life-of-Mine Average
S1[2]	Soil
SP1	Saprolite
TST1	Transitional
T1	Garimpeiros Tailing
T2	Garimpeiros Tailing

Note 1: Sample S1 was sent to SGS for ore characteristic test work in 2007.
Note 2: Sample S1 was sent to Wardell Armstrong for gravity, whole ore cyanide leach and flotation test work in 2009.

Composites SMKG and SMIG were used to form two composites:

- Blend A: Consists of 75% SMKG and 25% SMIG
- Blend B: Consists of 25% SMKG and 75% SMIG

Samples S1 to S3 were used for hardness and abrasion testing by SGS Lakefield Research in Canada. Samples S4 to S13 were used for flotation and cyanide leach of flotation concentrates by Hazen Research in USA. Sample ALL was the overall composite that was sent to Wardell Armstrong for flotation batch tests and pilot plant test work. The drill holes that characterize the composites for ore characterization and pilot plant test work are identified in Table 13.2, Figure 13.1 and Figure 13.2.

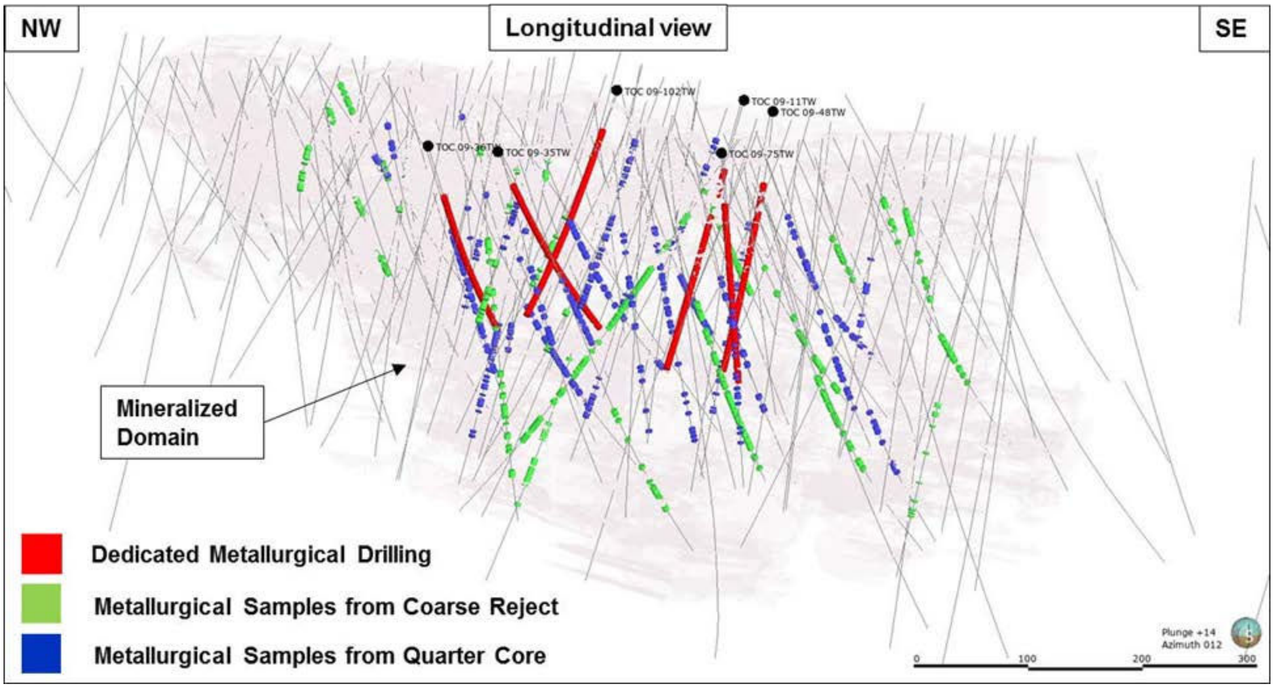
Table 13.2: Drill Hole and Composite Description

Sample	Description	Drill Hole ID		Test Work	Contractor
SMKG	Smoky Granite	TOC-09-11TW	TOC-09-75TW	Ore Characteristics (SMC Ore Hardness, CWI,	SGS (2009) and Wardell Armstrong (2010)
		TOC-09-35TW	TOC-09-102TW		

Sample	Description	Drill Hole ID		Test Work	Contractor
		TOC-09-48TW	-	UCS, RWI, BWI, Abrasion)	
SMIG	Salami Granite	TOC-09-11TW	TOC-09-48TW	Ore Characteristics (SMC Ore Hardness, CWI, UCS, RWI, BWI, Abrasion)	SGS (2009) and Wardell Armstrong (2010)
		TOC-09-35TW	TOC-09-75TW		
		TOC-09-36TW	TOC-09-102TW		
TOP	Top half orebody	TOC-09-11TW	TOC-09-102TW	Ore Characteristics (SMC Ore Hardness, CWI, RWI, BWI,	SGS (2009) and Wardell Armstrong (2010)
		TOC-09-36TW	TOC-09-102TW		
BOT	Bottom half orebody	TOC-09-75TW	TOC-09-48TW	Ore Characteristics (SMC Ore Hardness, CWI, RWI, BWI,	SGS (2009) and Wardell Armstrong (2010)
		TOC-09-36TW	TOC-09-48TW		
ALL	Overall composite	TOC-09-11TW	TOC-09-48TW	Pilot Plant	Wardell Armstrong (2010)
		TOC-09-35TW	TOC-09-75TW		
		TOC-09-36TW	TOC-09-102TW		
S1	Top 100 m	TOC-04-03	TOC-05-32	Ore Characteristics (RWI, BWI, Abrasion)	SGS (2007)
		TOC-04-04	TOC-06-37		
		TOC-04-05	TOC-06-44		
		TOC-04-06	TOC-06-45		
		TOC-04-12	TOC-07-47		
		TOC-04-15	TOC-07-48		
		TOC-04-17	TOC-07-55		
		TOC-04-19	TOC-07-57		
		TOC-04-20	TOC-07-60		
		TOC-05-21	TOC-07-61		
		TOC-05-27	TOC-07-62		
		TOC-05-29	TOC-07-66		

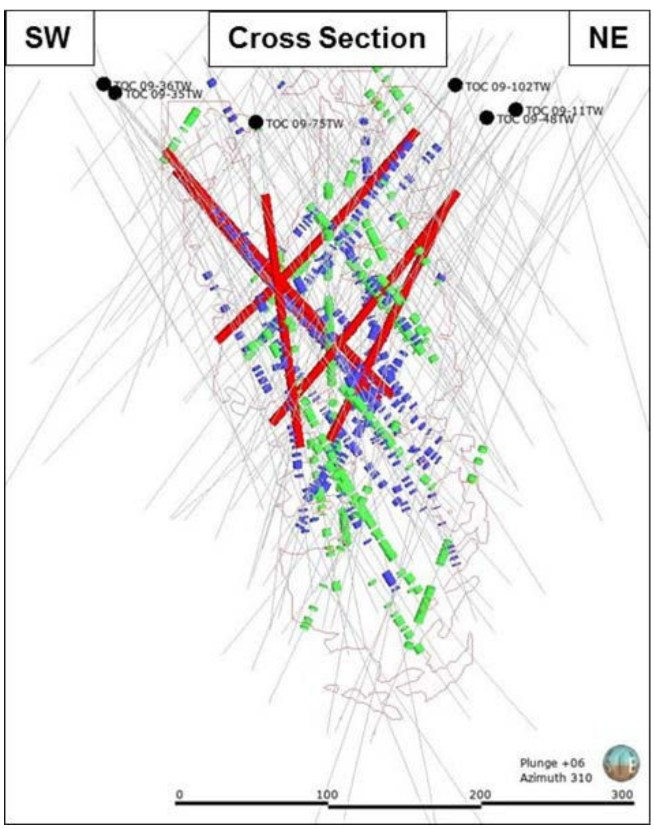
Sample	Description	Drill Hole ID		Test Work	Contractor
S2	Middle 100 m	TOC-04-01	TOC-05-28	Ore Characteristics (RWI, BWI, Abrasion)	SGS (2007)
		TOC-04-04	TOC-05-30		
		TOC-04-06	TOC-05-31		
		TOC-04-10	TOC-05-32		
		TOC-04-11	TOC-05-34		
		TOC-04-16	TOC-06-36		
		TOC-04-17	TOC-06-39		
		TOC-04-18	TOC-06-43		
		TOC-05-21	TOC-06-45		
		TOC-05-22	TOC-07-47		
		TOC-05-24	TOC-07-57		
		TOC-05-25	TOC-07-62		
		TOC-05-27	-		
S3	Bottom 100 m	TOC-04-01	TOC-06-35	Ore Characteristics (RWI, BWI, Abrasion)	SGS (2007)
		TOC-04-11	TOC-06-36		
		TOC-04-16	TOC-06-36		
		TOC-04-17	TOC-07-47		
		TOC-04-19	TOC-07-48		
		TOC-05-21	TOC-07-52		
		TOC-05-22	TOC-07-56		
		TOC-05-24	TOC-07-60		
		TOC-05-30	TOC-07-61		
		TOC-05-32	TOC-07-62		
		TOC-05-34	-		

Figure 13.1: Drill Holes for Metallurgical Testing – Longitudinal View



Source: Eldorado (2019)

Figure 13.2: Drill Holes for Metallurgical Testing – Cross Section



Source: Eldorado (2019)

13.4 Ore Composition

The Tocantinzinho mill feed during the LOM plan will come from three main sources. Granite representing approximately 94% of the LOM feed, saprolite representing approximately 3% of the LOM feed, and the *garimpeiros* tailings that were produced from previous artisanal mining, representing the remaining 3%.

The average annual plant head grade is 1.32 g Au/t for granitic ore, 1.03 g Au/t for saprolite, and 1.11 g Au/t for *garimpeiros* tailings. The combined average annual plant feed grade is 1.31 g Au/t with a maximum peak of 1.70 g Au/t in Year 7.

Other elements including carbon, carbonate, arsenic, mercury, copper, zinc, etc. were assayed and are briefly described below:

- Silver in the orebody ranges from 0.03 to 1.11 ppm with an average of 0.66 ppm. Silver has not been considered in the economic evaluation of the Project.
- Arsenic in the orebody ranges from 1.7 to 13 ppm with an average of 3.9 ppm.
- Mercury is in the order of 0.01 ppm.
- There are no other carbon elements other than carbonate.
- Average sulphide content of 0.27%.
- Carbonate (CO_3) level averages 0.65% and seems adequate in neutralizing any acid potentially generated from the oxidation of sulfide minerals within the ore body.
- Average levels for cadmium and chromium are 2.2 ppm and 83 ppm, respectively.
- Average levels for nickel, cobalt, copper, lead and zinc are 19 ppm, 3.6 ppm, 61 ppm, 115 ppm, and 104 ppm respectively.

13.5 Mineralization

There are two types of gold association with sulfide minerals; the first association occurs with pyrite (FeS_2), while the second association exists with pyrite (FeS_2), chalcopyrite ($CuFeS_2$), galena (PbS) and sphalerite (ZnS). Gold occurs as filling in the fractures and as inclusions (minor) in the following sulfide minerals:

- Disseminated pyrite
- Veinlets (in millimetre) of quartz, chlorite and pyrite (sheeted veins)
- Veins (in centimetre) of quartz, chlorite, carbonate, pyrite, chalcopyrite, galena and sphalerite

Pyrite occurs as liberated angular particles ranging in size from 30 to 400 µm with an average between 100 and 200 µm. Pyrite also occurs as intergrowth with galena, chalcopyrite, and rutile.

Chalcopyrite occurs as irregularly shaped particles ranging in size from 50 to 100 µm. The majority of chalcopyrite is liberated but may occur as inclusions in pyrite or intergrowth with galena and rutile.

The majority of gold is free at grain size ranging from 5 to 100 µm with an average range between 30 and 50 µm. The shape of gold grains is rounded, irregular shape, or elongated.

Pyrite in a disseminated form and in the sheeted veinlets hosts a bulk of gold mineralization. High-grade mineralization is often intimately associated with base metal veins. Gold also occurs less frequently as liberated particles of irregular shape with a size ranging from 5 to 250 µm with an average between 100 and 200 µm.

13.6 Comminution

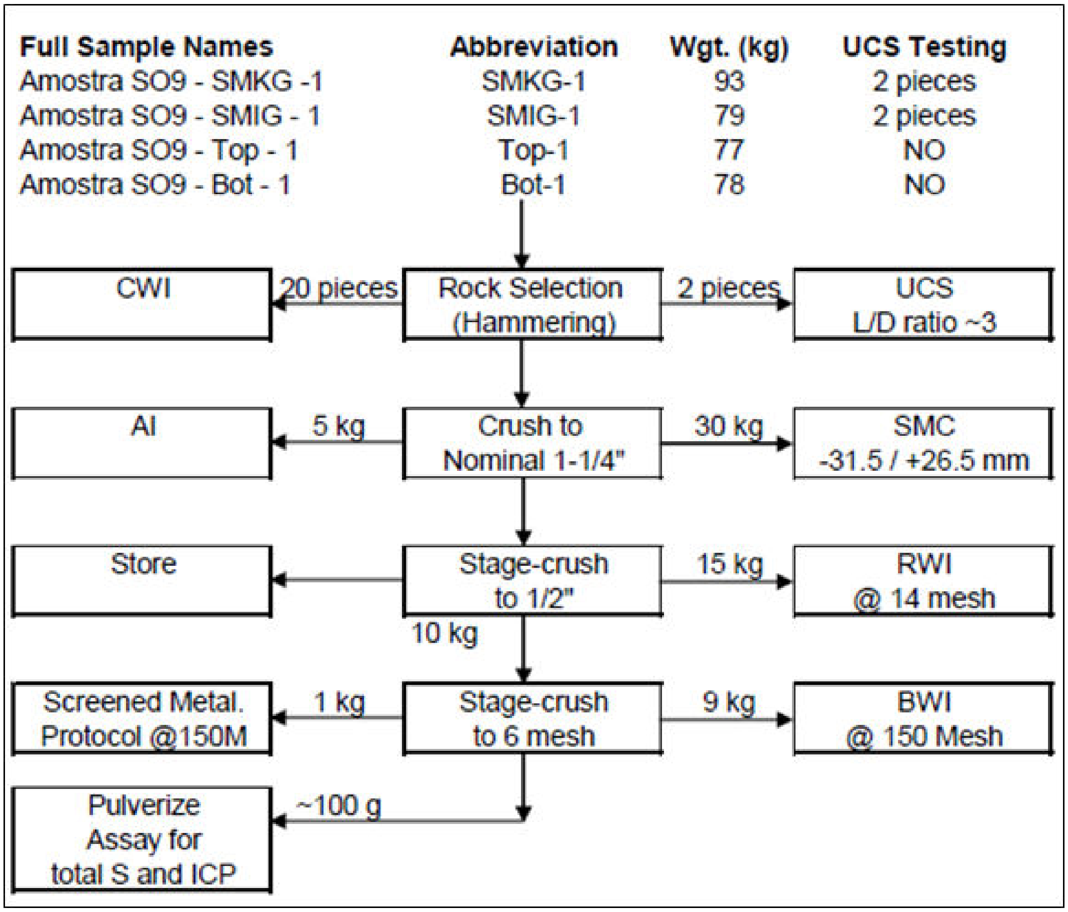
Seven samples were submitted to SGS Mineral Services in Canada for Bond Crushing Work index (CWi), Bond Rod Mill Work index (RWi), Bond Ball Mill Work index (BWi), SMC index (SMC is a shortened version of the standard JKTech drop-weight testing), Bond abrasion index and unconfined compressive strength (UCS).

Original comminution tests were carried out on three composites (S1, S2 and S3) by SGS in 2007.

Additional comminution tests were carried out on four composites (SMKG, SMIG, Top and Bot) by SGS in 2009.

The sample were prepared according to Figure 13.3.

Figure 13.3: Sample Preparation Diagram for Comminution Test Work



Source: SGS (2009)

From the test work, the ore samples can be characterized:

- Medium to hard ore in terms of crushing, with work index varying from 10.1 to 15.5 kWh/t.
- UCS test results ranged from 44.4 MPa to 114.7 MPa.
- Moderately soft to medium hardness ore with respect to resistance to impact breakage (A x b) based on SMC test results. The (A x b) value varies from 51.5 to 59.3.
- Medium ore in terms of rod milling, with work index varying from 13.2 to 14.7 kWh/t.
- Hard ore with respect to ball milling, with work index varying from 16.8 to 18.5 kWh/t.
- Highly abrasive ore, with abrasion index varying from 0.418 to 0.717 g.

Eleven ore samples were selected for specific gravity test work. The value of specific gravity varies from 2.60 to 2.82 t/m^3 with an average of 2.67 t/m^3. The results are summarized in Table 13.3.

Table 13.3: Comminution Test Work Summary

<table>
<tr><th rowspan="2">Sample</th><th colspan="5">SMC</th><th colspan="3">Bond Indices kWh/t</th><th>AI</th><th>UCS</th><th>SGS Report</th></tr>
<tr><th>A x b</th><th>A</th><th>b</th><th>ta</th><th>DWI</th><th>CWI</th><th>RWI</th><th>BWI</th><th>g</th><th>MPa</th><th>Year</th></tr>
<tr><td>S1</td><td></td><td></td><td></td><td></td><td></td><td></td><td>14.1</td><td>17.6</td><td>0.695</td><td></td><td>2007</td></tr>
<tr><td>S2</td><td></td><td></td><td></td><td></td><td></td><td></td><td>14</td><td>17.6</td><td>0.717</td><td></td><td>2007</td></tr>
<tr><td>S3</td><td></td><td></td><td></td><td></td><td></td><td></td><td>14.7</td><td>18.2</td><td>0.612</td><td></td><td>2007</td></tr>
<tr><td rowspan="2">Amostra S09 - SMKG - 1</td><td>59.345</td><td>71.5</td><td>0.83</td><td>0.58</td><td>4.5</td><td>10.08</td><td>13.2</td><td>18.2</td><td>0.5009</td><td>44.4</td><td rowspan="2">2009</td></tr>
<tr><td></td><td></td><td></td><td></td><td></td><td></td><td></td><td></td><td></td><td>114.7</td></tr>
<tr><td rowspan="2">Amostra S09 - SMIG - 1</td><td>58.065</td><td>73.5</td><td>0.79</td><td>0.57</td><td>4.5</td><td>12.88</td><td>13.5</td><td>17.6</td><td>0.4501</td><td>104.4</td><td rowspan="2">2009</td></tr>
<tr><td></td><td></td><td></td><td></td><td></td><td></td><td></td><td></td><td></td><td>75.4</td></tr>
<tr><td>Amostra S09 - Top - 1</td><td>51.544</td><td>75.8</td><td>0.68</td><td>0.5</td><td>5.2</td><td>15.48</td><td>13.5</td><td>18.5</td><td>0.5578</td><td></td><td>2009</td></tr>
<tr><td>Amostra S09 - Bot - 1</td><td>53.436</td><td>73.2</td><td>0.73</td><td>0.52</td><td>5</td><td>15.31</td><td>14.1</td><td>16.8</td><td>0.4182</td><td></td><td>2009</td></tr>
<tr><td>Max</td><td>59.3</td><td>75.8</td><td>0.8</td><td>0.6</td><td>5.2</td><td>15.5</td><td>14.7</td><td>18.5</td><td>0.72</td><td>114.7</td><td></td></tr>
<tr><td>Min</td><td>51.5</td><td>71.5</td><td>0.7</td><td>0.5</td><td>4.5</td><td>10.1</td><td>13.2</td><td>16.8</td><td>0.42</td><td>44.4</td><td></td></tr>
<tr><td>Average</td><td>55.6</td><td>73.5</td><td>0.8</td><td>0.5</td><td>4.8</td><td>13.4</td><td>13.9</td><td>17.8</td><td>0.56</td><td>84.7</td><td></td></tr>
<tr><td>85th Percentile</td><td>52.4</td><td></td><td></td><td>0.6</td><td>5.1</td><td>15.4</td><td>14.2</td><td>18.2</td><td>0.70</td><td>110.1</td><td></td></tr>
</table>

The comminution values derived for the design criteria are summarized in Table 13.4.

- Unconfined Compressive Strength (UCS): 85th percentile out of the UCS test work used for the process design criteria (PDC).
- Maximum hardness (lowest value of 51.5) from the SMC Test work was used for the PDC to ensure the SAG mill can withstand variability of the ore.
- Bond Indices:
  - Crusher Work Index (CWI): 85th percentile out of the CWI test work was used for the design criteria.
  - Rod Work Index (RWI): 85th percentile out of the RWI test work was used for the design criteria.
  - Bond Work Index (BWI): 85th percentile out of the BWI test work was used for the design criteria.

- Abrasion Index: Maximum abrasion value obtained from the available test work was used for the PDC.
- Specific Gravity: Average value obtained from the available test work was used for the PDC.

Table 13.4: Design Criteria – Comminution Characteristics

Criterion	Units	Design	Notes
Unconfined Compressive Strength (UCS)	MPa	87.8	85^{th} Percentile
JK Breakage Parameters			
A x b	-	51.5	Maximum Hardness from 4 Available Data
DWi	-	5.2	Maximum Hardness from 4 Available Data
ta	-	0.5	Maximum Hardness from 4 Available Data
Work Index			
Impact Crushing Work Index (CWI)	kWh/t	15.4	85^{th} Percentile
Bond Rod Work Index (RWI)	kWh/t	14.2	85^{th} Percentile
Bond Ball Mill Work Index (BWI)	kWh/t	18.2	85^{th} Percentile
Abrasion Index	g	0.72	Maximum Value from All Available Data
Specific Gravity	-	2.67	Average Value from All Available Data

Calculations and comminution modeling simulations for the design and sizing of the comminution circuit was based on the values established in the design criteria. The target product grind size was 80% passing 125 microns based on pilot plant test work. A SAG Mill/Ball Mill (SAB) configuration was chosen, and the size of the mills are summarized below:

- One (1) 8.5 m diameter by 5.0 m long (Effective Grinding Length) SAG Mill with an installed power of 6,500 kW.
- One (1) 6.4 m diameter by 9.15 m long (Effective Grinding Length) Ball Mill with an installed power of 6,500 kW.

13.7 Gravity Concentration

The gravity recovery test work was performed using a Knelson concentrator which obtained moderate high recoveries for gold and was therefore included in the process design.

The gravity concentration test work program was evaluated four times using the Knelson concentrator.

The first test work program was carried out by SGS Mineral Services in Canada on four composite ore samples (2005 report). Gold recovery increased with head grade, reaching 41.9% for a high-grade (12.3 g/t) ore sample. The concentrate was upgraded using a Mozley table and obtained a much higher concentrate grade.

The second test work program on gravity concentration was conducted by Wardell Armstrong International in UK on three composite ore samples (2009 report). Four stages of gravity concentration were applied consecutively to each ore sample as the grind size became finer. All three ore samples achieved high recoveries ranging from 73% to 90% with a mass pull of 4.2% and a concentrate grade ranging 26 -36 g Au/t.

The third test work program on gravity concentration was conducted by Wardell Armstrong International in 2009. Seven (7) composites were tested and obtained moderate-high gold recoveries, ranging from 53% to 70%, with an average mass yield of 4.4%. The average recovery of the seven composites was used to support gravity recovery values in the design criteria (see Table 13.5).

Table 13.5: Gravity Test Work on Composites by Wardell Armstrong (2009)

Sample			SMKG	SMIG	Blend A	Blend B	BOT	TOP	ALL	Avg.
Calculated Head Grade		g/t	0.49	1.81	1.05	1.71	1.63	1.89	1.48	1.44
Gravity Concentrate	Mass	%	5.03	4.48	4.14	4.46	4.07	4.50	4.28	4.42
	Grade	g/t	6.01	21.5	15.5	23.0	25.1	29.5	18.7	19.9
	Recovery	%	62.2	53.0	61.0	60.2	62.4	70.3	54.1	60.5

The fourth test program was a single test conducted by FLSmidth (2012). The test consisted of sequential liberation and recovery stages. It obtained gold recovery of 65% at a 1.1% mass yield. The final concentrate grade was 75.9 g Au/t. This test was used as the basis for gravity modeling and selection of the gravity concentrator.

A summary of the gravity test work is shown in Table 13.6.

The gravity recovery circuit will consist of two centrifugal gravity concentrators and an intensive cyanidation unit (ICU). The gravity concentrate will be leached in batches and assumed to have a leach recovery of

94.5% for the design criteria. The leach recovery is based on the same cyanide leach recovery for flotation concentrate as per the pilot plant test work.

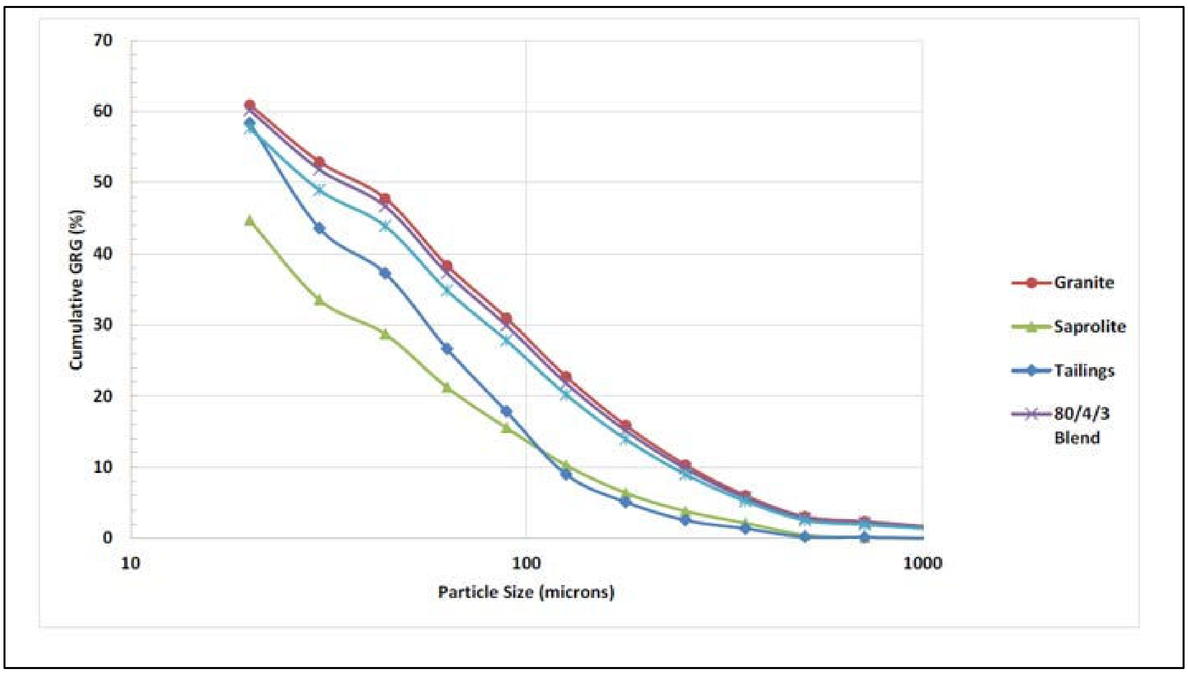
Table 13.6: Summary of Gravity Test Work Results

Gravity Test Program	Sample ID	Calc Head Grade	Grind Size 80% passing	Mass Pull	Concentrate Grade	Au Recovery	Concentrator Passes	Batch
		g Au/t	µm	%	g Au/t	%		
1st Gravity Test Program - SGS (2005)	Comp A	1.33	66	0.057	351	15	1 pass	1 kg
1st Gravity Test Program - SGS (2005)	Comp B	1.02	75	0.121	239	28	1 pass	1 kg
1st Gravity Test Program - SGS (2005)	Comp C	3.55	65	0.117	1178	39	1 pass	1 kg
1st Gravity Test Program - SGS (2005)	Comp D	9.89	45	0.08	5171	42	1 pass	1 kg
2nd Gravity Test Program - Wardell Armstrong (2009)	TOP	1.61	125	4.2	31.5	83	3 passes	11 kg
2nd Gravity Test Program - Wardell Armstrong (2009)	BOT	1.72	125	4.23	36.5	90	3 passes	11 kg
2nd Gravity Test Program - Wardell Armstrong (2009)	ALL	1.56	125	4.21	26.2	73	3 passes	11 kg
3rd Gravity Test Program - Wardell Armstrong (2009)	SMKG	0.49	125	5.03	6.01	62	1 pass	2 kg
3rd Gravity Test Program - Wardell Armstrong (2009)	SMIG	1.81	125	4.48	21.5	53	1 pass	2 kg
3rd Gravity Test Program - Wardell Armstrong (2009)	Blend A	1.05	125	4.14	15.5	61	1 pass	2 kg
3rd Gravity Test Program - Wardell Armstrong (2009)	Blend B	1.71	125	4.46	23	60	1 pass	2 kg
3rd Gravity Test Program - Wardell Armstrong (2009)	BOT	1.63	125	4.07	25.1	62	1 pass	2 kg
3rd Gravity Test Program - Wardell Armstrong (2009)	TOP	1.89	125	4.5	29.5	70	1 pass	2 kg

Gravity Test Program	Sample ID	Calc Head Grade	Grind Size 80% passing	Mass Pull	Concentrate Grade	Au Recovery	Concentrator Passes	Batch
		g Au/t	µm	%	g Au/t	%		
3rd Gravity Test Program - Wardell Armstrong (2009)	ALL	1.48	125	4.28	18.7	54	1 pass	2 kg
4th Gravity Test Program - FLS Knelson (2012)	Tocantinzinho	1.3	70	1.1	75.9	65	3 passes	20 kg

FLSmidth subsequently completed additional gravity recoverable gold (GRG) modeling in 2021 based on historical gravity test work (refer to Figure 13.4). GRG for the mill feed is summarized in Table 13.7.

Figure 13.4: Cumulative GRG as a Function of Particle Size



Source: FLSmidth (2021)

Table 13.7: GRG for Gold Recovery Calculations

	Granite	Saprolite	Tailings
GRG	25%	15%	15%

13.8 Flotation

Significant flotation test work has been completed since 2007. Test programs included batch rougher / scavenger, batch cleaner, locked cycle and continuous pilot plant to examine flotation kinetics, reagent selection, grind size and metallurgical performance of various ore types.

13.8.1 Preliminary Flotation Tests

Grind size was evaluated initially by Ralph Meyertons (2007). PAX and Aeroflot were used as collectors, and mineral oil and Aerofroth 65 were used as frother. A summary of the test work is shown in Table 13.8.

Overall gold recovery averaged 92.4% with a concentrate mass pull of 2.4%. Grind size correlation to gold recovery could not be established as the best two gold recovery results were achieved with the coarsest and the finest samples (Test 1E and 3A, respectively). The calculated head grade was also found to be quite variable.

Table 13.8: Flotation Test Work Results from Ralph Meyertons (2007)

Test ID			1D	1E	2A	2B	3A	3B	3C	Average
Calculated Head Grade		g/t	1.67	1.70	1.89	1.35	2.07	1.06	0.93	1.52
Grind Size	Passing 150 µm	%	86.0	86.0	~100	99.5	~100	95.4	99.5	-
	Passing 100 µm		-	-	98.8	94.5	98.8	74.1	93.5	-
	Passing 75 µm		-	-	88.9	75.5	88.9	54.0	75.5	-
Final Concentrate	Mass Pull	%	3.5	5.0	2.0	2.0	1.4	1.1	1.5	2.4
	Grade	g/t	44.7	32.8	87.8	62.5	137.0	88.1	55.9	72.7
	Recovery	%	90.5	96.1	93.4	92.9	95.9	88.5	89.6	92.4

**Note: Natural pH, 32% pulp density, 10 minutes conditioning time, 30 g/t PAX, 30 g/t Aeroflot 31, 30 g/t mineral oil, 20 ~ 40 g/t Aerofroth 65, 20 ~ 25 minutes flotation time, 1.46 g Au/t assay head grade.*

13.8.2 Initial Reagent Screening

Batch test work flotation tests were carried out by Hazen starting in 2007. Initial tests were done on composite samples to examine reagent selection, operating conditions and upgradeability with scavenger and cleaning stages. Additional variability and confirmatory test work were carried out by Hazen again in 2009. The reagents that were tested is listed below:

- Collectors/Promoters:
  - ORFOM CO100 (Mercaptan)
  - PAX (Potassium amyl xanthate)
  - Aero 5688 (Monothiophosphate salt)
  - Aerofloat 31
  - S-8474
  - Mineral Oil
  - CMC (Carboxymethyl cellulose)
- Frothers:
  - Aerofroth 65
  - DF-250
  - MIBC (Methyl isobutyl carbinol)

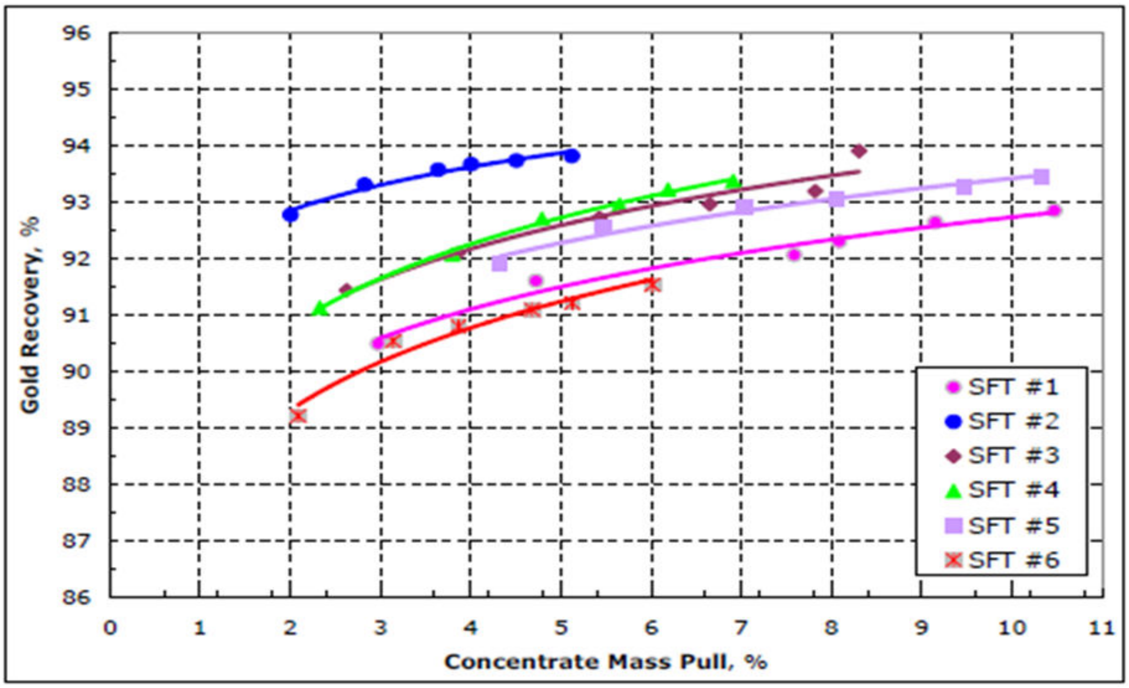
A summary of the reagent variability test work is shown in Table 13.9 and Figure 13.5.

Table 13.9: Reagent Variability Flotation Test Work Summary

Test Number			SF1	SF2	SF3	SF4	SF5	SF6
Ore Material			Composite					
Calculated Feed Grade		g Au/t	1.3	1.5	1.5	1.4	1.4	1.1
		S %	0.40	0.44	0.40	0.40	0.37	0.35
Grind Size, 80% Passing		µm	~75 µm					
Concentrate Mass Pull		%	10.5	5.1	8.3	6.9	10.3	6.0
Concentrate Grade		g Au/t	11.1	28.1	17.0	19.0	12.4	16.9
Au Recovery		%	92.9	93.8	93.9	93.4	93.5	91.5
Collector/ Promoters	CO100 (Grind)	g/t	-	-	100	100	0	30
	PAX	g/t	45	45	30	30	45	45
	Aero 5688	g/t	-	-	60	60	90	90
	Aerofloat 31	g/t	45	-	-	-	-	-
	S-8474	g/t	-	45	-	-	-	-
	Mineral Oil	g/t	45	-	-	-	-	-

Test Number			SF1	SF2	SF3	SF4	SF5	SF6
Ore Material			Composite					
	CMC	g/t	-	100	-	-	-	-
Frothers	Aerofroth 65	g/t	28	-	-	-	-	-
	DF250	g/t	-	-	33	37	37	46
	MIBC	g/t	-	74	-	-	-	-

Figure 13.5: Gold Recovery vs Concentrate Mass Pull for Reagent Variability Test Work



Source: Eldorado (2019)

Among these six tests, SFT #2 obtained the best performance, achieving nearly 94% recovery at 5.1% mass pull.

The operating conditions for SFT#2 are listed below:

- Grinding – 80% passing 75 µm, 15 g/t PAX, 15 g/t S-8474, 100 g/t CMC.
- Flotation – natural pH, 30-35% pulp density, 48 minutes total flotation time, 30 g/t PAX, 30 g/t S-8474.

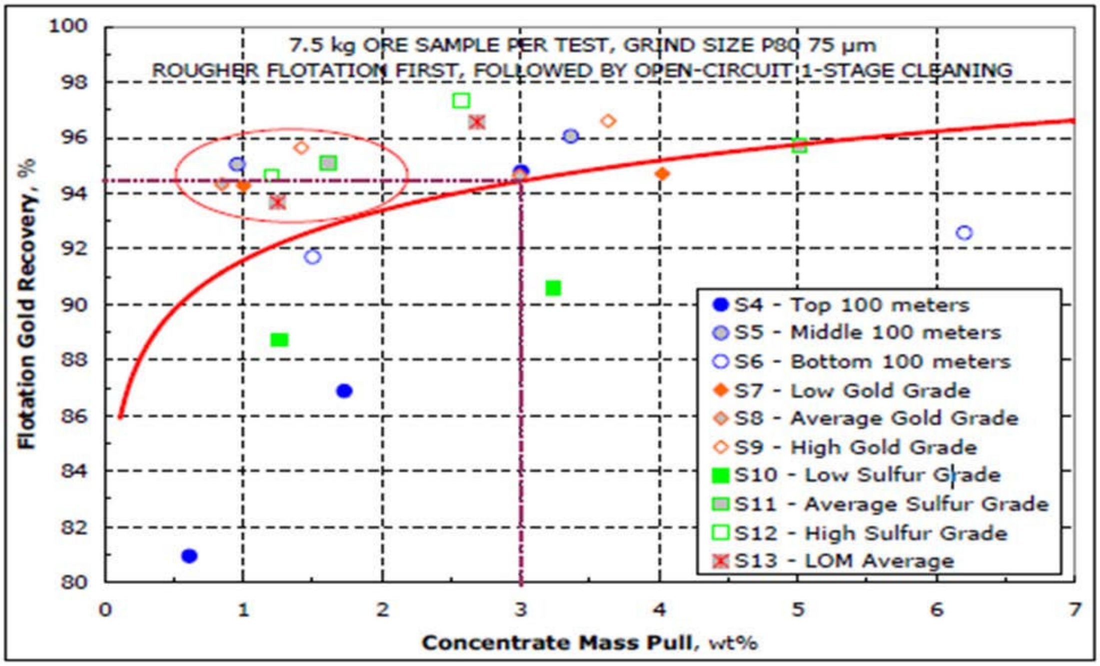
13.8.3 Initial Ore Variability Assessment

Following the operating conditions for SFT #2, Hazen completed testing of 10 different ore types in 2009. These results are presented in Table 13.10 and Figure 13.6.

Table 13.10: Ore Variability Flotation Test Work Summary

Test Number			LF1	LF2	LF3	LF4	LF5	LF6	LF7	LF8	LF9	LF1
Ore Material			**S4**	**S5**	**S6**	**S7**	**S8**	**S9**	**S10**	**S11**	**S12**	**S13**
Calculated Feed Grade		Au g/t	0.24	0.93	0.92	0.56	1.17	1.85	0.96	0.89	0.89	1.35
Grind Size, 80% Passing		µm	~75 µm									
Concentrate Mass Pull		%	1.7	3.4	6.2	4.0	3.0	3.6	3.2	5.0	2.6	2.7
Concentrate Grade		Au g/t	50.1	26.4	13.7	13.1	37.1	49.2	27.1	17.1	66.1	48.6
Au Recovery		%	86.8	96.0	92.6	94.7	94.6	96.6	90.7	95.7	97.3	96.6
Collector/ Promoters	CO100 (Grind)	g/t	-	-	-	-	-	-	-	100	100	100
	PAX	g/t	60	60	60	60	60	60	60	40	40	40
	Aero 5688	g/t	-	-	-	-	-	-	-	80	80	80
	Aerofloat 31	g/t	-	-	-	-	-	-	-	-	-	-
	S-8474	g/t	60	60	60	60	60	60	60	-	-	-
	Mineral Oil	g/t	-	-	-	-	-	-	-	-	-	-
	CMC	g/t	100	100	100	100	100	100	100	-	-	-
Frothers	Aerofroth 65	g/t	-	1	1	1	1	1	1	-	-	-
	DF250	g/t	21	-	-	-	-	-	-	4	3	3
	MIBC	g/t	-	-	-	-	-	-	-	-	-	-

Figure 13.6: Gold Recovery vs Concentrate Mass Pull for Ore Variability Test Work



Source: Eldorado (2019)

Flotation performance for these ten different composite samples demonstrate:

- Materials near the surface containing a lower sulfur level did not float as well
- Ore with higher gold grade or sulfur grade floated better
- Mass pull at 3%, gold recovery is expected to be ~94.5%
- Mass pull at 5%, gold recovery is expected to be ~96% The results are in good agreement with the previous test work.

13.8.4 Additional Variability Test: Rougher Flotation

Additional variability test work, including rougher, cleaner and locked cycle tests, were completed in 2010 by Wardell Armstrong on 7 composites:

- SMKG, SMIG, BOT, TOP, ALL
- Blend A: Consists of 75% SMKG and 25% SMIG
- Blend B: Consists of 25% SMKG and 75% SMIG

The reagents that were tested includes:

- Collector: SIBX (Sodium isobutyl xanthate)

- Activator: Copper sulfate
- Frother: DF-250

Two (2) grind sizes were tested for all 7 composites:

- 80% passing 75 µm
- 80% passing 125 µm

Additional testing at 80% passing 100 µm and 150 µm was done on ALL Composite.

A summary of the tests on the ore sample designated ALL can be found in Table 13.11. This ore sample represents a global proportion of ore types within the orebody.

Table 13.11: Rougher Flotation Test Work for the ALL (Global Composite) Ore Sample

Test	Conditions	Mass Pull %	Recovery %	
			Au	STot
FT1	Grind 75 µm	3.5	96.3	87.7
FT2	Grind I00 µm	3.8	96.0	88.2
FT3	Grind 125 µm	3.8	96.7	80.4
FT4	Grind 150 µm	3.8	95.2	87.3
FT5	As FT3 but with no $CuSO_4$	7.2	94.2	82.8
FT6	As FT3 but with reduced $CuSO_4$	3.1	94.8	88.4
FT7	As FT3 but with MIBC	3.4	92.1	84.8
FT8	As FT3 but with reduced SIBX	4.5	92.1	87.2
FT9	As FT3 but with increased SIBX	4.3	92.3	85.5
FT10	As FT3 but with SIBX+A5688	5.3	95.0	83.7
FT11	As FT10 but with increased SIBX+A5689	7.9	94.9	84.3
FT12	Hazen Conditions from Previous study	10.5	87.6	88.4
FT13	As FT3 but with PAX	2.4	92.2	78.7
FT14	As FT3 but with SIBX+S-7151	3.1	75.6	80.9
FT15	As FT3 but with SIBX+A8761	3.9	94.2	82.9
FT16	As FT3 but with 50% Increase in DF250	4.1	88.7	81.7
FT17	As FT3 but with 50% decrease in DF250	2.7	93.1	77.0

The results from the ALL rougher flotation test work demonstrate:

- Grind size at 125 µm resulted in similar to slightly better results when compared to finer grind sizes.
- Flotation performance without copper sulfate is adversely affected (lower recoveries, higher mass pull).
- Varying dosage levels of SIBX and DF-250 were tested from Tests FT7 to FT17. Compared with the baseline reagent suite (Test FT3: 50 g/t $CuSO_4{\cdot}5H_2O$, 80 g/t SIBX, 40 g/t DF-250), other conditions and alternative reagents did not generate better flotation performance.

- Test FT3 obtained the highest recovery and was chosen as the basis for further test work including the pilot plant.
- The design criteria for retention time and reagent addition is based on the parameters for Test FT3 (Table 13.13).
- Downstream test work such as cleaner, locked cycle and pilot plant test work used the parameters from Test FT3 for rougher flotation conditions.

13.8.5 Additional Variability Test: Cleaner Flotation

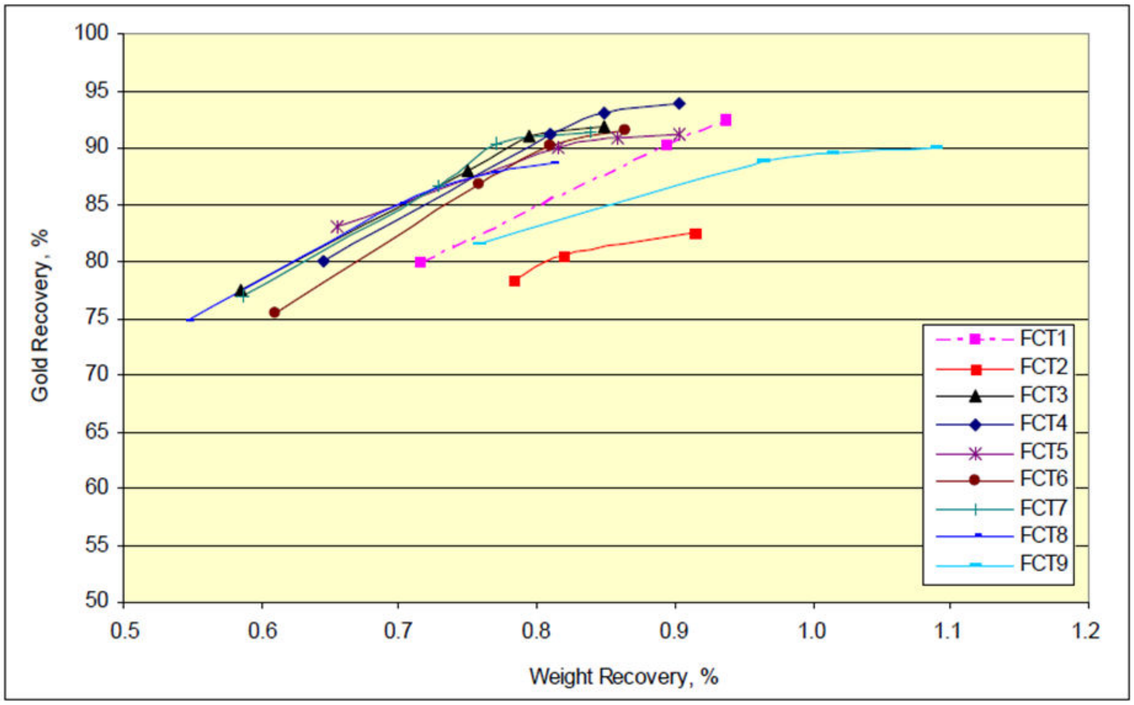
A more comprehensive cleaner flotation test was done on the global composite (ALL) and is summarized in Table 13.12 and Figure 13.7.

Rougher flotation was carried out based on the conditions in Test FT3 (See Section 13.8.4) and only one stage of cleaning was tested.

Table 13.12: Cleaner Flotation Test Work for ALL (Global Composite) Ore Sample

Test	Conditions	Mass Pull %	Recovery %	
			Au	STot
FCT1	One cleaner stage	3.0	94.1	82.4
FCT2	Scalp and clean, rougher scavenger	0.9	83.0	69.7
FCT3	As FT1 but increase float time	2.4	92.7	78.7
FCT4	As FT1 but no DF250 or SIBX	2.5	95.2	82.6
FCT5	As FT1 but increase SIBX	2.2	91.7	77.3
FCT6	As FT1 but $CuSO_4$·added	2.4	92.3	85.3
FCT7	Airflow rate 1 l/min	2.3	92.3	90.4
FCT8	Airflow rate 0.5 l/min	2.4	90.3	85.4
FCT9	Airflow rate 1.5 l/min	2.6	90.8	86.6

Figure 13.7: Gold Recovery vs Concentrate Mass Pull for Cleaner Flotation Test Work



Source: Wardell Armstrong International (2009b)

Test FCT4 had a much higher feed grade which contributed to significant higher recovery and was therefore considered an outlier.

Test FCT3 obtained the best mass pull to recovery ratio and was used to develop the design criteria.

Combining the best conditions from rougher and cleaner test work (Test FT3 from Section 13.8.4 and Test FCT3 from Section 13.8.5), the flotation duration and the flotation reagent design criteria can be summarized in Table 13.13.

Table 13.13: Design Criteria - Flotation Retention Time and Reagent Addition

Flotation Duration Criterion	Units	Design	Notes
Conditioning	min	5.0	
Rougher Flotation - Lab Retention Time	min	6.5	Batch test work rougher and rougher scavenger total at 13 min (Based on Test FT3)
Rougher-Scavenger Flotation - Lab Retention Time	min	6.5	
Cleaner Flotation - Lab Retention Time	min	7.0	Cleaner batch test work at 7 min (Based on Test FCT3)

Flotation Duration Criterion	Units	Design	Notes
Flotation Reagent Addition Criterion	**Units**	**Design**	**Notes**
Collector: SIBX Addition Rate			Based on Wardell Armstrong Test Work FCT3 (same as Pilot Plant)
Rougher Flotation Conditioning Addition	g/t Mill Feed	60.0	
Rougher Flotation SIBX Addition	g/t Mill Feed	20.0	
Cleaner Flotation SIBX Addition	g/t Mill Feed	10.0	
Total SIBX Addition	g/t Mill Feed	90.0	
Frother: DF-250 Addition Rate			Based on Wardell Armstrong Test FCT3 (same as Pilot Plant)
Rougher Flotation Addition	g/t Mill Feed	30.0	
Rougher-Scavenger Flotation Addition	g/t Mill Feed	10.0	
Cleaner Flotation Addition	g/t Mill Feed	10.0	
Total Frother Addition	g/t Mill Feed	50.0	
Promoter: Copper Sulphate Addition Rate			Based on Wardell Armstrong Test FCT3 (same as Pilot Plant)
Grinding Addition	g/t Mill Feed	25.0	
Rougher Flotation Conditioning Addition	g/t Mill Feed	25.0	
Total Promoter Addition for Flotation	g/t Mill Feed	50.0	

13.8.6 Additional Variability Test: Locked Cycle Test Work

Wardell Armstrong completed locked cycle tests on seven composites in 2010. A summary of the results is presented in Table 13.14.

A simple reagent suite based on SIBX as collector, copper sulfate as activator and DF-250 as frother was used.

Table 13.14: Locked Cycle Flotation Test Work Summary

Ore Sample			SMKG	SMIG	Blend A	Blend B	TOP	BOT	All		
Calculated Head Grade	Gold	g/t	0.59	1.91	0.85	1.48	1.49	1.66	1.53	1.43	1.34
	Sulfur	%	0.34	0.35	0.28	0.26	0.44	0.48	0.33	0.32	0.29
Grind Size, 80% Passing		µm	125						125	100	125
Rougher Condition	SIBX	g/t	80						80	80	120
	DF250	g/t	40						40	40	60
Cleaner Condition	SIBX	g/t	20						10	10	15
	DF250	g/t	20						10	10	15
Concentrate Mass Pull		%	0.79	0.78	0.87	0.92	0.98	1.35	0.80	1.29	1.57
Concentrate Grade	Gold	g/t	69.6	221	83.8	146	144	118	175	101	79.7
	Sulfur	%	32.3	37.1	23.2	23.0	40.4	31.0	33.2	21.2	15.6
Recovery	Gold	%	92.5	89.9	86.0	91.3	94.3	95.5	91.6	91.7	93.4
	Sulfur	%	75.1	82.9	71.9	81.1	88.9	87.6	81.8	84.7	84.7

**Note: 32% pulp density for rougher, natural pH (8.4 ~ 9.3) for rougher and cleaner, 50 g/t $CuSO_4 \cdot 5H_2O$, rougher flotation time 13 to 15 minutes, cleaner time 6 to 7 minutes. Average results from 5th and 6th cycles.*

Composite ore samples, namely TOP, BOT and ALL, outperformed slightly individual ore types, SMKG and SMIG. The average gold recovery was 93.3% with an average concentrate mass pull of 1.20%.

13.8.7 <u>Pilot Plant Flotation</u>

Pilot plant flotation test work was undertaken by Wardell Armstrong International in 2010-2011. The composite ore sample designated ALL, which represents a global proportion of ore types within the orebody, was used for this test work.

A total of 3,100 kg material was processed at a rate of 65 ~ 80 kg/h through crushing, grinding, rougher flotation and cleaner flotation. The purpose of the pilot plant campaign was to verify gold recovery and concentrate mass pull, and to generate tailing materials for other test work.

The reagent suite and dosage rates for the pilot plant test work were based on bench scale flotation test FT3 and FCT3 (See Section 13.8.4 and 13.8.5).

The pilot plant test work was tested at two primary grind sizes; 80% passing 125 µm and 85 µm, and the results are summarized in Table 13.15.

Table 13.15: Pilot Plant Test Work Summary

Day	Grind µm	Feed Wt			Wt Pull, %	Recovery %
		kg	%	%wrt Grind		
Day 1 - 6th	87	525.90	16.89	35.13	5.20	83.93
Day 2-1 - 7th	85	387.40	12.44	25.88	3.44	90.69
Day 2-2 - 7th	85	237.80	7.64	15.88	4.30	87.94
Day 3 - 8th	90	346.06	11.11	23.11	6.16	95.77
Fine Grind (Av. assay)		-	-	-	4.49	89.36
Fine Grind (Wt Av.)		1,497.16	48.08	100.00	4.82	89.05
Day 4 - 9th	115	528.37	16.97	32.68	4.74	91.51
Day 5 - 10th	120	537.48	17.26	33.24	6.88	95.77
Day 6 - 11th	125	551.04	17.70	34.08	1.98	93.10
Coarse Grind (Av. Assay)		-	-	-	3.62	93.52
Coarse Grind (Wt Av.)		1,616.89	51.92	100.00	4.51	93.46
Total		3,114.1	100.00	-	4.66	91.34

Copper sulphate solution was added to the first conditioner at a rate of 50 g/t $CuSO_4{\cdot}5H_2O$ and SIBX and DF-250 solution was added to the second conditioner at a rate of 90 g/t and 50 g/t respectively. (See Section 13.8.4 and 13.8.5)

At the coarser grind sizes (P80 from 115 to 125 µm), gold recovery achieved an average of 93.5% at a concentrate mass pull of 4.51%. The coarser grind was used as the flotation design criteria (Table 13.16) due to a higher recovery to mass yield ratio.

13.8.7.1 Grind Size

Pilot plant utilized two (2) types of grind sizes. A finer target with P80 from 85 to 90 µm and a coarser target with P80 from 115 to 125 µm were chosen based on optimum results from batch variability tests (refer to Section 13.8.4). Based on the results from the batch test work and the pilot plant test program, a 125 µm P80 was selected for the design criteria.

Table 13.16: Design Criteria - Flotation Recovery (Granite)

Criterion	Units	Design	Notes
Flotation feed size, P80	µm	125	Based on batch and pilot plant test work
Concentrate recovery, Au, relative to mill fresh feed	%	93.5	Flotation pilot plant test work
Gravity recovery (Design), relative to mill fresh feed	%	25	Based on GRG simulation
Cleaner mass pull	%	4.5	Average from pilot plant test work

13.8.8 Additional Flotation Test Work

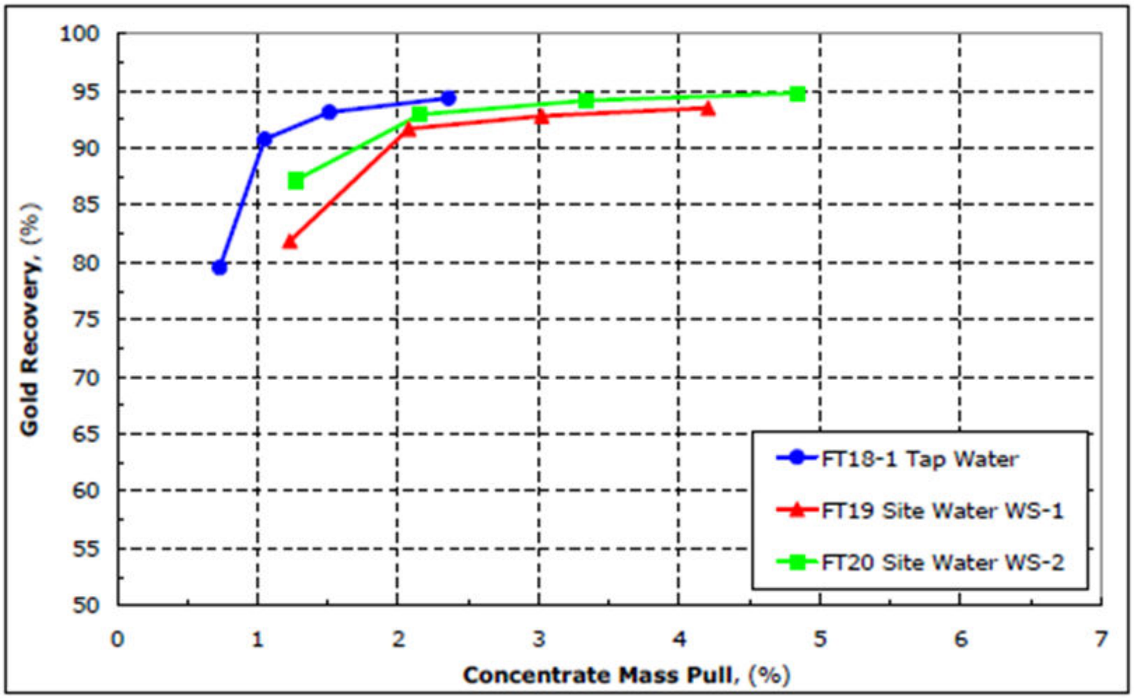
Impact of process water on flotation:

- Recycling process water marginally increased gold recovery but primarily impacted concentrate mass pull.
- Concentrate mass pull increased from 2.4% (FT18-1) to 6.6% (FT18-3) after process water was recycled twice.
- Reagent requirement during plant operation was not tested but is expected to drop significantly. Normally one third in comparison with what consumed in the laboratory where only fresh water is used.
- Impact of site make-up water on flotation.
- There was no significant impact on gold recovery despite slightly slower flotation kinetics. At a concentrate mass pull of 4.51%, similar gold recovery is expected in comparison with clean tap water.

Table 13.17: Water Test Conditions and Results

Test ID	Conditions	Concentrate Mass Pull	Gold Recovery	Sulfur Recovery	Calculated Head Grade	
		%	%	%	g Au/t	%S
FT18-1	Tap water	2.4	94.4	87.2	1.74	0.30
FT19	Water from the drill hole TOC-91 (WS-1)	4.2	93.5	79.2	1.77	0.37
FT20	Water from the Teodorao River (WS-2)	4.8	94.8	81.3	1.28	0.30

Figure 13.8: Gold Recovery and Water Test Conditions



Source: Eldorado (2019)

13.8.9 Composition of Flotation Concentrate

Ten cleaner flotation concentrates were analyzed to obtain a general impression of the composition. Due to variations in ore composition and concentrate mass pull during flotation, concentration compositions vary. The average flotation concentrate assay of notable elements below was primarily from the locked cycle tests in the Wardell Armstrong International pilot plant test program in 2010 (Mass Pull ~1%).

- Silver = 89 ppm
- Aluminum = 4.2%
- Arsenic = 176 ppm
- Carbonate carbon = 0.26%
- Gold = 113 ppm
- Calcium = 0.9%
- Copper = 3,628 ppm
- Iron = 23%
- Mercury = 0.8 ppm
- Potassium = 2.0%
- Lead = 6,396 ppm
- Magnesium = 0.8%
- Zinc = 3,423 ppm
- Sodium = 0.9%
- Total sulfur = 24.7%

The cleaner concentrate grade used for design was calculated from a 4.5% mass yield and flotation recoveries from pilot plant test work with gravity taken into consideration (Table 13.18).

Table 13.18: Design Criteria – Flotation Concentrate Grade (Granite)

Criterion	Units	Nominal		Design
Cleaner Mass Pull	%	4.5		Flotation Pilot Plant Test Work
Gold Concentrate Grade	g/t	21.6	24.4	Design grade based on 5% mass yield and same recoveries

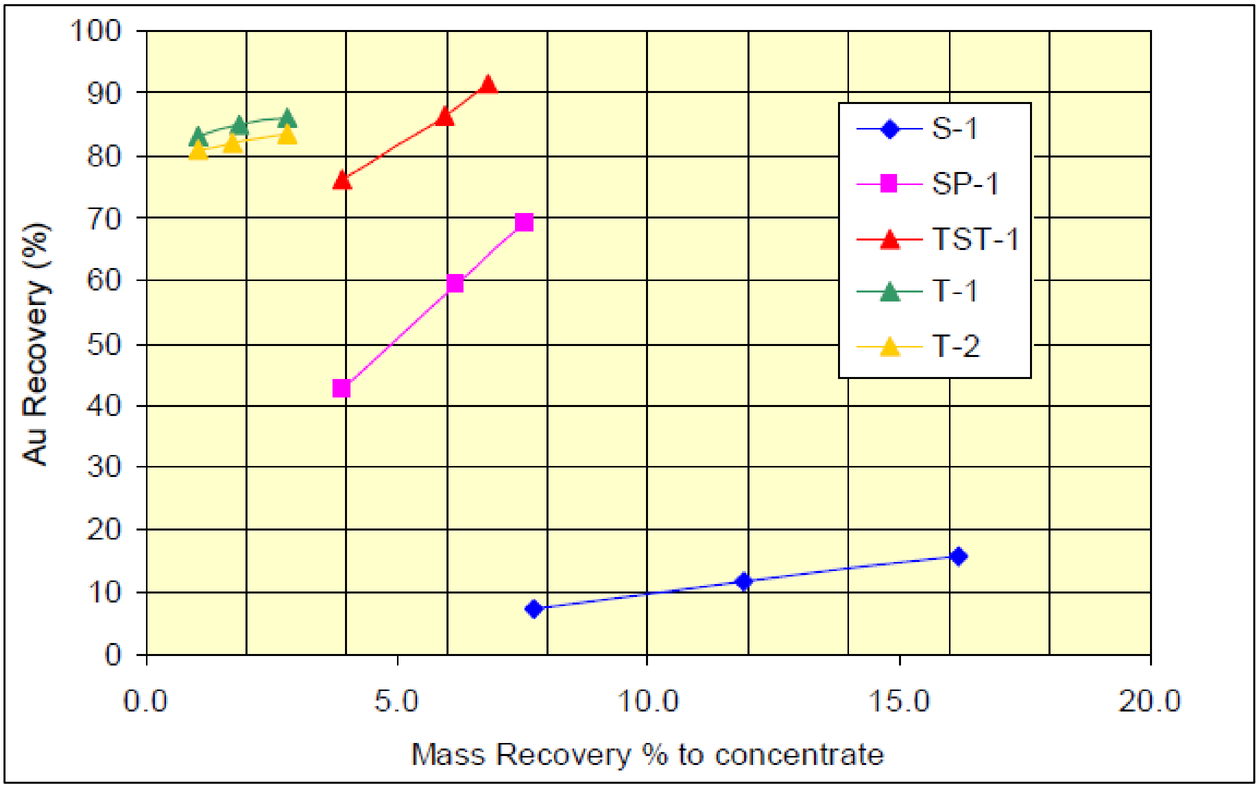
13.8.10 Saprolite and Tailings Flotation Test Work

In 2009, Wardell Armstrong International completed metallurgical test work on soil, saprolite, transition zone and tailings samples and included gravity tests, whole of ore leach tests and flotation tests. The samples tested are not considered representative.

Single bench flotation tests were completed on samples at a grind size of P80 of 75 µm using a Denver D12 laboratory flotation cell and at a pulp density of 30% w/w. All tests were kinetic type with concentrates being taken over timed intervals and assayed separately. All test products were assayed for gold and total sulphur. Copper sulphate was used to activate sulphide minerals and lime added to increase the pH to neutral. Guar was added as a gangue depressant and potassium amyl xanthate as the collector. An oxide float was also undertaken using sodium hydrogen sulphide (NaSH) and the Cytec promoter A412.

The gold recovery results are provided in Figure 13.9. The two tailings samples gave a similar flotation response with gold recoveries of 86.2% for T1 and 83.6% for T2. Mass recoveries were low for both samples at 2.8%. The TST transition sample gave the highest gold recovery of 91.5% and the gold recovery versus mass recovery trend suggests that longer flotation times would increase recovery. The saprolite SP1 sample gave a recovery of 69.1% at a mass pull of 7.6%. The soil sample S1 gave the poorest flotation response with a recovery of 15.7%, despite the high mass pull of 16.2%.

Figure 13.9: Gold Flotation Recovery Results of Saprolite and Tailings Samples



Source: Wardell Armstrong International (2009a)

13.9 Cyanide Leaching

Initial cyanide leaching test work was carried out by Hazen in 2009 to determine the leach kinetics of different concentrate samples generated from previous flotation test work. Eleven concentrates, without additional grinding were leached for 90 hours. All samples, excluding one outlier, achieved excellent gold recovery with an average of 98%. The leach rate was reasonable as most of the gold recovery begin to plateau within 48 hours.

In 2010, Wardell Armstrong International obtained seven different concentrate samples for cyanide leaching. The test program compared oxygen/air sparging, cyanide concentration and concentrate regrind. The tests were leached for 48 hours before activated carbon was added and continued for an additional six hours.

The results of the batch and pilot plant test work are summarized in Table 13.19 with the description of the test work conditions listed below:

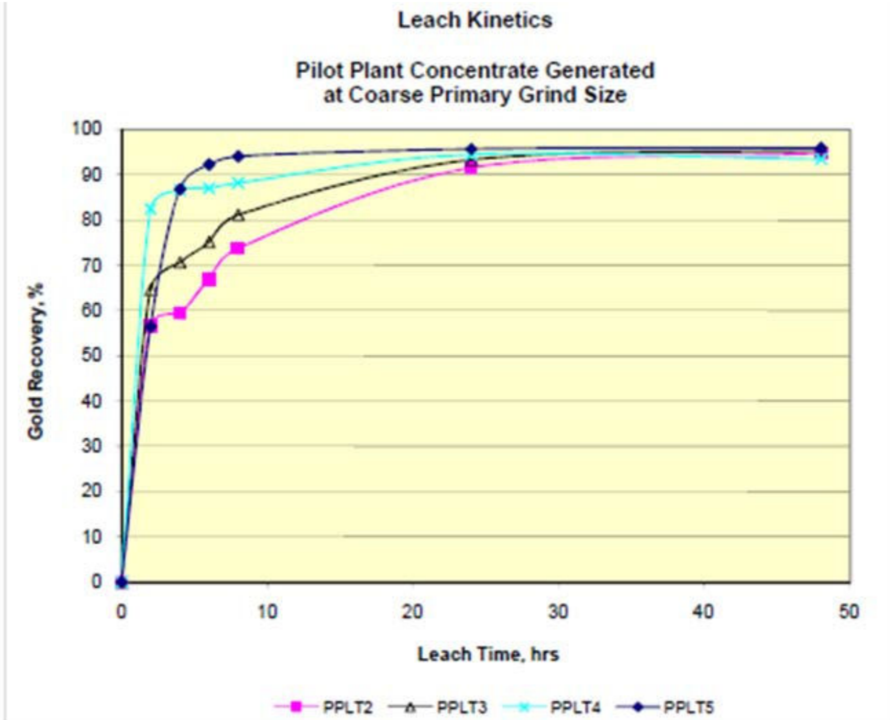
- PP-LT2: Concentrate P80 = 125 µm, maintain cyanide at 2 g/L
- PP-LT3: Concentrate P80 = 125 µm; re-ground to P80 = 18 µm, maintain cyanide at 1 g/L

- PP-LT4: Concentrate P80 = 125 µm, maintain cyanide at 1 g/L and use oxygen for aeration
- PP-LT5: Concentrate P80 = 125 µm, maintain cyanide at 5 g/L
- PP-LT6: Concentrate P80 = 85 µm, maintain cyanide at 5 g/L
- PP-LT7: Concentrate P80 = 85 µm, maintain cyanide at 2 g/L

Table 13.19: Cyanide Leach on Flotation Concentrate Test Work (Batch and Pilot Test)

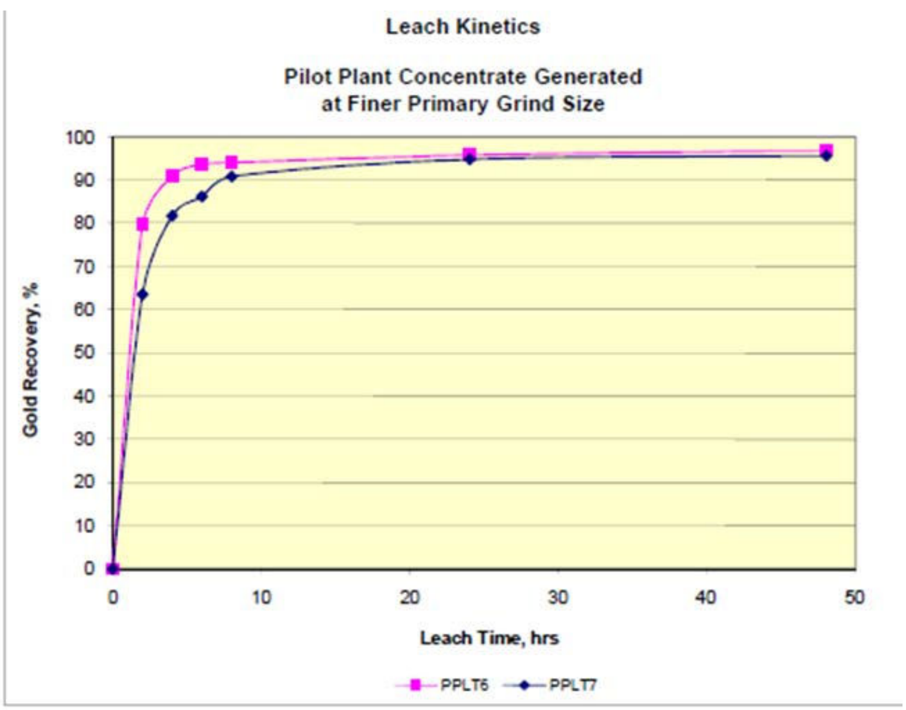
Sample ID	Grind Size 80% Passing	Pulp Density	Cyanide Concentration		Concentrate Regrinding	Sparging	Gold Recovery	Lime Consumed	Cyanide Consumed	Calculated Head Grade
			Initial	Maintained						
	µm	%	g/L NaCN				%	kg/t	kg/t NaCN	g Au/t
SMKG	125	4.3	2.0	2.0	no	air	95.7	0.44	32.8	69
SMIG		3.6					91.1	0.54	92.3	281
Blend A		5.4					97.0	0.35	44.7	190
Blend B		7.2					974	0.26	21.3	120
TOP		4.5					96.8	0.42	34.1	101
BOT		4.5					97.1	0.42	42.1	142
Average	/	/	/	/	/	/	**95.9**	**0.41**	**44.6**	/
ALL - PPLT2	125	27	4.9	2.0	No	air	95.0	0.23	6.34	30.6
ALL - PPLT3		27	4.9	1.0	yes- 18 µm	air	97.8	0.33	4.82	29.7
ALL- PPLT4		27	4.9	1.0	no	oxygen	94.0	0.22	3.89	32.6
ALL- PPLT5		34	10.3	5.0	no	air	96.3	0.15	16.4	30.0
ALL - PPLT6	85	35	10.0	5.0	no	air	96.8	0.20	12.6	13.0
ALL - PPLT7		35	4.0	2.0	no	air	97.2	0.20	5.81	13.9
Average	/	/	/	/	/	/	**96.2**	**0.22**	**8.3**	**25.0**

Figure 13.10: Leach Kinetics of Pilot Plant Concentrate Leach Test Work at Coarser Feed Size (P80 = 125 µm)



Source: Eldorado (2019)

Figure 13.11: Leach Kinetics of Pilot Plant Concentrate Leach Test Work at Finer Grind Size (P80 = 85 µm)



Source: Eldorado (2019)

The pilot plant test work results are summarized below:

- Leaching of concentrate (flotation feed size; P_{80} = 125 µm) achieved recoveries in excess of 94%.
- Re-grinding or leaching at a finer particle size (P_{80} = 85 µm) improved recoveries (~97%).
- Leaching at increased cyanide concentrations (2 g/L vs 5 g/L) did not display improved gold recoveries but increased cyanide consumption.
- Leaching kinetics began to plateau after 32 hours.

In 2017, SGS completed additional leaching test work to test pre-aeration, dissolved oxygen, leach concentration and reagent dosage. The tests were leached over a 32-hour period with a finer concentrate (P_{80} = 22 µm – 51 µm). The test results are summarized in Table 13.20.

The SGS additional test work can be summarized below:

- Extraction of gold was greater than 94% in all tests.
- Increasing the cyanide concentration from 1 g/L NaCN to 2 g/L NaCN resulted in a slight increase in gold extraction from 95.0% to 95.7% and improved leach kinetics.
- Increasing the cyanide concentration to 3 g/L NaCN did not affect the gold recovery.

The design criteria for leaching conditions and recovery are best represented in the pilot plant test work PP-LT2 and PP-T4 as pilot testing results are more reliable and since re-grind or a finer grind size was not chosen for this process. The average gold recovery of these two tests were used for the design criteria. Since the recovery of the pilot cyanide leach tests were calculated after carbon assays, the recovery used for the design criteria has already taken into account for gold loss in solution or carbon.

The design criteria for cyanide leaching is summarized in Table 13.21.

Table 13.20: SGS Leach Kinetics Test Work Summary

CN Test No.	Grind P80 µm	Actual DO mg/L		Preaer. Sol'n mg/L		Leach NaCN g/L	Reagents, kg/t of CN feed				Au Extr'n%	CN Residue g/t		CIP Recovery %		CIP Residue g/t		Head (calc) g/t	
							Added		Consumed										
		Preaer	CN	S_2O_3	Au		NaCN	CaO	NaCN	CaO		Au	Ag	Au	Ag	Au	Ag	Au	Ag
CN-1	51	3-5	14-26	58	<0.01	1	2.53	1.66	1.70	1.66	95.6	1.86	13.4	95.0	76.4	1.70	12.4	35.2	52.5
CN-2	51	3-4	9-27	65	<0.01	2	3.99	1.60	2.09	1.60	95.9	1.70	<10	95.7	83.6	1.51	7.5	34.0	45.6
CN-4	51	4-5	11-13	55	<0.01	2*	3.56	1.08	2.32	1.06	95.8	1.62	9.3	95.8	86.2	1.45	6.7	35.5	48.5
CN-3	51	4-5	7-28	20	<0.01	3	5.52	1.53	2.54	1.53	96.2	1.57	<10	95.8	86.3	1.51	6.4	34.6	46.7
CN-5	33	4	7-13	120	<0.01	3	5.61	1.79	2.78	1.72	97.6	0.97	7.9	97.1	86.9	0.97	6.4	34.3	48.8
CN-6	22	3	9-14	250	0.05	3	6.16	1.71	3.44	1.66	98.3	0.68	6.7	98.3	88.9	0.58	5.7	34.9	51.2
CN-7	51	4	6-8	52	<0.01	3	6.36	1.33	4.01	1.21	97.9	-	7.2	94.7	85.4	1.89	7.6	36.6	52.2

**Note: CN concentration maintained until 8h only*

Table 13.21: Design Criteria – Cyanide Leaching

Criterion	Units	Design	Notes
Flotation Feed Grind Size, 80% passing	µm	125	Potential opportunity to reduce feed size
Cyanide Concentration	g/L	2.0	Based on pilot plant recoveries and reagent consumption
Leach Retention Time	h	36	Leaching plateau after 32 hours
pH	-	11.0 - 11.6	Based on pilot plant test work
Solids Density	% wt	45	Selected for design
Gold Extraction	%	94.5	Most representative pilot plant test work (PPLT4 at 24 h)
Cyanide Consumption	kg/t leach feed	5.0	Based on pilot plant test work and client recommendations
Lime Consumption	kg/t leach feed	1.0	Based on pilot plant test work and client recommendations

13.9.1 <u>Cycle Test</u>

SGS conducted a single test in 2017 which recycled CIP barren solution for three cycles to determine the effects on recovery. The cycle test was based on the conditions of CN-3 (see Table 13.19). The leach was maintained at 3 g/L NaCN without any regrind. The results are presented in Table 13.22.

Key points from the cycle test work:

- Recoveries were unaffected by recycling
- Requirement for fresh cyanide was reduced by ~ 1.5 kg/t NaCN through recycling.

Table 13.22: Cycle Test Summary

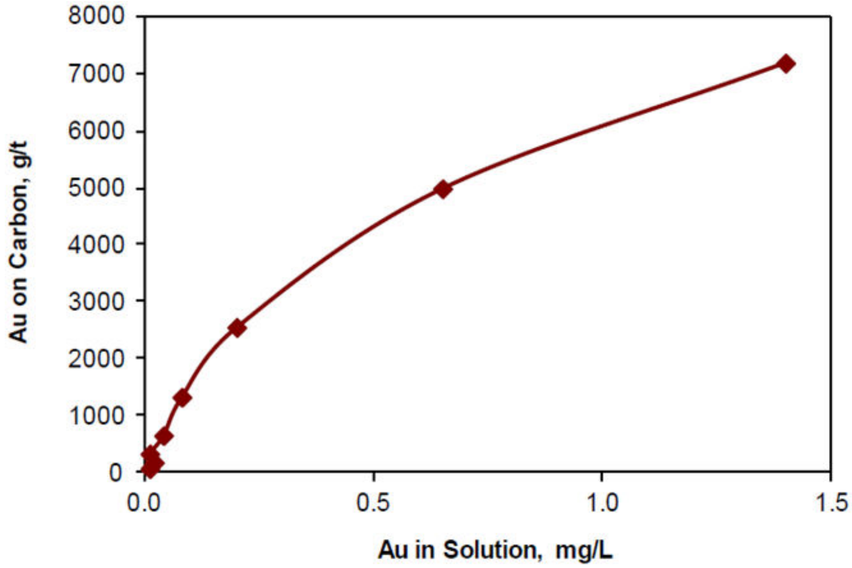
CN Test No.	Actual DO mg/L		Reagents, kg/t of CN feed					Au Extr'n %	CN Residue g/t		CIP %		CIP Residue g/t		Head (calc) g/t	
			Added			Cons.										
	Preaer	CN	NaCN	NaCN*	CaO	NaCN	CaO		Au	Ag	Au	Ag	Au	Ag	Au	Ag
CN-3 (baseline)	4-5	7-28	5.52		1.53	2.54	1.53	96.2	1.57	<10	95.8	86.3	1.51	6.4	34.6	46.7
CN-8A (cycle 1)	5	8-18	5.69		1.09	2.85	1.02	96.0	1.58	8.0	95.6	86.2	1.54	7.1	35.5	51.5
CN-8B (cycle 2)	5	7-12	5.41	4.12	1.29	2.31	1.16	96.2	1.50**	13.4	96.3	87.7	1.28	6.6	35.1	53.7
CN-8C (cycle 3)	3	10-17	5.40	4.18	1.42	2.60	1.35	96.3	1.53	7.6	95.3	86.9	1.64	6.7	36.3	51.2

**Note: fresh cyanide only*
***Note: estimated value, actual assay was 5.28 g Au/t*

13.9.2 Gold and Silver Loading

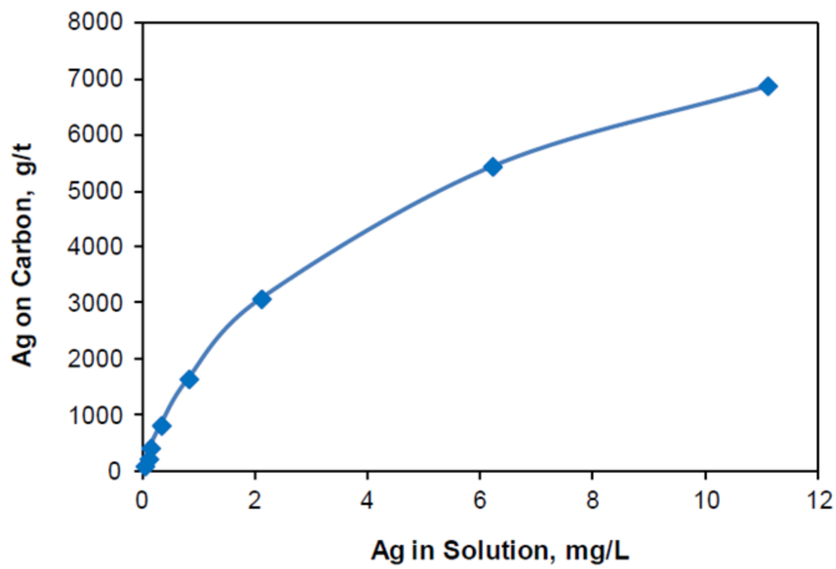
Gold and silver loading isotherm tests were completed by SGS in 2017. The pregnant solution was obtained from following leach test CN-3 (see Table 13.20). The leach was maintained at 3 g/L NaCN without any regrind. Carbon was added for 72 hours for stabilization. The results for the isotherm tests are plotted in Figure 13.12 and Figure 13.13.

Figure 13.12: Equilibrium Loading of Gold on Carbon



Source: SGS (2017a)

Figure 13.13: Equilibrium Loading of Silver on Carbon



Source: SGS (2017a)

13.10 Cyanide Destruction

Various cyanide destruction treatment options including Cold Caro's Acid, CombinOx, a combination of Cold Caro's Acid / CombinOx method, sodium hypochlorite, SO_2 / air process, activated carbon, and Caroate method were tested.

The target of the pulp treatment program was to reduce the total cyanide concentration to less than 1 mg/L total cyanide (CN_T) and less than 0.2 mg/L for free cyanide (weak acid dissociable) (CN_{WAD}).

Based on effluent release standards based on CONAMA resolution 430 (May 13, 2011), the Brazilian federal requirement, for cyanide concentration in effluents was:

- Total cyanide 1.0 mg/L CN
- Free cyanide (weak acid dissociable) 0.2 mg/L CN

SO_2 / Air process followed by aging was found to be the preferred cyanide detoxification method as the cyanide levels could be significantly reduced after the process (to 0.41 CN_T mg/l and 0.2 CN_{WAD} mg/l) and achieve < 0.2 mg/l for both CN_T and CN_{WAD} after only 10 days of aging (Table 13.25 and Table 13.26 provide more details). In addition, this process is a conventional method of treatment with low operating costs.

13.10.1 Initial Cyanide Destruction Test Work

Initial and preliminary cyanide destruction test work was conducted by Wardell Armstrong in 2010 but was not successful in reaching the target levels.

CyPlus performed various cyanide destruction tests in 2011 including the SO_2 / air process. Following the SO_2 / Air process, CyPlus was able to reduce the pulp to total cyanide levels below 2 mg/L. However, the aging test on the supernatant was unable to achieve the final supernatant target of 0.2 mg/L CNwad even after a twentyfold dilution.

CyPlus also conducted aging tests by treating both leach tailings and flotation tailings together. The cyanide concentration did not decrease after dilution but increased (attributed to the presence of ferrocyanide in the effluent). This prompted the decision to treat leach tailings and flotation tailings separately.

13.10.2 Batch and Continuous Cyanide Destruction Test Work

SGS performed batch and continuous cyanide destruction test work in 2013 focusing on the SO_2 / Air process and aging tests. Preparation for cyanide destruction test work included pre-aeration and leaching the flotation concentrate at 5.0 g/L NaCN for 48 hours with a continuous supply of air. 10 g/L of pre-attrition activated carbon (CIP) was added to the pregnant leach pulp to recover gold.

Batch and continuous cyanide destruction tests (SO_2 / Air process) were performed on the barren leached products. Batch tests were conducted to optimize reagent dosage and retention time. The results from the test work are summarized in Table 13.23.

Table 13.23: Cyanide Destruction Test Work Results by SGS (2013)

Test No	Mode	Retention Time, min	Feed/ Product	pH	FeCl3 Addition in Feed, mg/L	FeCl3 Addition in Product, mg/L	Assays, mg/L				Cumulative Reagent Addition					
											g/g CNWAD			g/L Pulp		
							CNT	CNWAD	Cu	Fe	SO_2 eq.	Lime	Cu	SO_2 eq.	Lime	Cu
CND B1	Batch	120	Feed	10.7	-	-	2550	1520	361	274	6.61	2.88	0.22	8.51	3.71	0.28
			Product	8.6			**1967**	**347**	462	211						
CND B2	Batch	150	Feed	10.7	-	-	2550	1520	361	274	19.3	11.0	0.53	24.9	14.2	0.68
			Product	8.6			**1.08**	**0.20**	32.3	0.20						
CND B3	Batch	150	Feed	10.2	-	-	2550	1520	361	274	23.1	13.4	0.66	29.8	17.3	0.85
			Product	8.6			**30.9**	**1.63**	1.03	0.03						
CND C1	Continuous	84	Feed	10.2	-	15.0	2550	1520	361	274	20.8	12.0	0.20	26.8	15.5	0.26
			Product	8.6			**45.5**	**1.14**	103	85.8						
CND B4	Batch	180	Feed	10.2	15.0	-	2550	1520	361	274	8.50	6.15	0.66	10.9	7.93	0.85
			Product	8.5			**1.75**	**0.84**	-	-						
CND C2	Continuous	200	Feed	10.2	15.0	-	2550	1520	361	274	25.6	20.8	1.26	33.0	26.8	1.62
			Product	8.6			**1.50**	**0.69**	0.37	0.34						
CND B5	Batch	180	Feed	10.2	15.0	-	2550	1520	361	274	11.0	6.81	0.66	14.2	8.77	0.85
			Product	8.6			**0.86**	**0.76**	-	-						
CND C3	Continuous	186	Feed	10.2	15.0	-	2550	1520	361	274	23.0	15.6	0.88	29.6	20.1	1.13
			Product	8.6			**0.34**	**0.46**	1.31	0.2						

Aging tests were tested on cyanide destruction product itself (Aging-01) and a blend of flotation tailings and cyanide detoxed product (Aging-02). The tests were run for 55 days at room temperature (~25˚C) to determine if environmental disposal can be applicable. The results from the aging test work are summarized in Table 13.24.

Table 13.24: Aging Test Work by SGS (2013)

Time	CN Species, mg/L	Aging -01	Aging -02
Day-01	CN_T	0.86	14.9
	CN_{WAD}	0.76	0.51
Day-07	CN_T	0.25	9.05
	CN_{WAD}	0.17	0.20
Day-15	CN_T	0.20	3.33
	CN_{WAD}	0.16	0.21
Day-33	CN_T	0.24	046
	CN_{WAD}	0.23	0.20
Day-45	CN_T	32.7	0.45
	CN_{WAD}	1.22	0.07
Day-55	CN_T	N/A	0.28
	CN_{WAD}	N/A	0.08

**Note: N/A=Not Applicable*

The detoxified leached flotation concentrate contained less than 1.0 mg/L total cyanide by using the SO_2/Air process. The cyanide detox product was then aged for 55 days to lower CN_T below 0.3 ppm. It was unable to reach the target of 0.2 ppm CN_{WAD} required for environmental discharge.

Aging with flotation tailings show higher cyanide levels associated with insoluble precipitation with iron or copper. This agreed with previous CyPlus test work which suggests separating flotation tailings from cyanide tailings.

13.10.3 Confirmation Cyanide Destruction Test Work

Additional cyanide destruction test work using the SO_2/Air process was completed by SGS in 2017. The flotation concentrate received was much finer (P_{80} = 51 µm) but was leached at cyanide concentration levels more representative to the design criteria. The concentrate was pre-aerated and leached for 32 hours with oxygen being sparged. 15 g/L of pre-attrition activated carbon (CIP) was added to the pregnant leach pulp to recover gold and silver.

The test conditions were based on previous cyanide destruction test work conducted by CyPlus in 2011 which included three stages. Copper sulfate and air were added at the start of the cyanide destruction test. For the first stage, copper sulfate and sodium metabisulfite was added continuously until a low residual cyanide level was reached (target < 1 mg/L CNT). The overflow would enter the second stage, where additional copper sulfate and air was added. The third stage would include adding ferric chloride. Each stage had a retention time of approximately one hour. The results from the test work are summarized in Table 13.25.

Table 13.25: Confirmatory Aging Test Work by SGS (2017)

Aging Time days	Solution Analysis			
	CNT mg/L	CNWAD mg/L	Cu mg/L	Fe mg/L
1	< 0.1	< 0.1	3.11	< 0.05
3	0.16	< 0.1	2.23	< 0.05
7	0.29	0.2	1.84	< 0.05
10	0.12	< 0.1	1.28	< 0.05
14	0.16	0.15	1.25	< 0.05

The barren pulp from the most representative cyanide destruction test work (Test CND1-3) was used for an aging test. The results are shown in Table 13.26.

Tests CND1-1 and CND 1-3 were able to reduce the total cyanide concentration to less than 1 mg/L in two stages.

Test CND1-2 was terminated earlier since the CN_T levels did not reach this target. This may be due to the lack of copper addition in the first stage.

Test CND1-3 is identical to CND1-1 without the third stage and confirms that the target cyanide concentration can be reached without ferric chloride. The solution from this test was used for aging test and was able to stabilize and reach the environmental discharge limit (< 1.0 mg/L CN_T and < 0.2 mg/L CN_{WAD}) after 14 days. Test CND1-3 was used as the basis for the cyanide destruction design criteria.

Table 13.26: Confirmatory Cyanide Destruction Test Work Results by SGS (2017)

Feed/Test	Total Test Time min.	Retention Time min per stage	Stage	Test Conditions								Test Results Solution Analysis					
				pH	emf mV	DO mg/L	Reagent Addition g/g CN_{WAD}			Reagent Addition mg/L solution							
							SO_2 Equiv.	Lime	Cu	Cu	$FeCl_3$	CN_T mg/L	CN_{WAD} mg/L	Picric* mg/L	Cu mg/L	Fe mg/L	NH_3+NH_4 as N, mg/L
Feed:				*9.6*								*1050*	*799*		*420*	*104*	
CND1-1	300	54	1	8.5	174	3.2	5.50	3.63	0.02	19	19	<0.1	< 0.1	5.8	14.0	<0.2	
			2	8.4	153	8.0	0	0	0.03	21	21	<0.1	< 0.1	0.6	4.2	<0.2	
			3	8.2	165		0	0	0	0	5	<0.1	< 0.1	0.5	3.6	<0.2	
Feed:												*1100*	*649*				
CND1-2	150	64	1	8.7	100	3.3	7 08	4.1	0	0	0	0.56	0.1	9.2	18.5	0.1	
			2	8.2	166	8.0	0	0	0	23	0	0.25	0.1	2.2	2.9	0.1	
Feed:												*998*	*674*				
CND1-3	360	60	1	8.4	188	2.4	6.00	4.7	0.03	22	0	0.13	<0.1	0.5	6.39	<0.05	25.7
			2	8.2	174	8.8	0	0	0.03	21	0	0.41	0.2	0.8	4.28	<0.05	23.9

**Note: CN_{WAD} analysis by picric acid method.*

13.10.4 Cyanide Destruction Design Criteria

A summary of the cyanide destruction test work is shown in Table 13.27 for comparison.

Table 13.27: Cyanide Destruction Test Work Summary

Test Work	Leach Cyanide Concentration	Feed CNT	Feed CNWAD	SO2	Product CNWAD	After Aging CNT	After Aging CNWAD
	g/L	mg/L	mg/L	g/g CNWAD	mg/L	mg/L	mg/L
SGS 2017/CND1-3	3	998	674	6.0	0.20	0.16	0.15
SGS 2013/CNDC3	5	2550	1520	23	0.46	0.28	0.08
CyPlus 2011/T25	-	988	302	5.3	0.60	>10	>10

Test CND1-3 from the SGS report in 2017 was able to successfully use the SO_2 / Air process to reduce the total cyanide concentration to below target (< 1 mg/L CN_T) and less than 0.2 mg/L CN_T after aging.

The design conditions and reagent addition for cyanide destruction is summarized in Table 13.28.

Table 13.28: Design Criteria – Reagent Addition for Cyanide Destruction

Reagent	Units	Design	Notes
SMBS	kg/t Mill Feed	0.203	Assumed 50% recovery from thickener
Copper Sulphate	kg/t Mill Feed	0.001	Assumed 50% recovery from thickener
Lime	kg/t Mill Feed	0.107	Assumed 50% recovery from thickener

13.11 Environmental Test Work

Environmental test work was carried out by SGS in 2017 to assess contaminant release potential associated with CIL tailings from the SO_2 / air process. The test work carried out on the barren pulp generated from Test CND 1-3 (See Section 13.10.3) was able to reduce CN_T < 1 ppm and was used as the design basis for cyanide destruction.

The effluent discharge must comply with the Effluent Discharge for Para State, and Federal standards (CONAMA). Assumptions and considerations for the water treatment of effluent are summarized below:

- The dissolved copper in the residue decant was 3.55 mg/L which is above the Federal limit of 1.0 mg/L total copper in solution, requiring copper pre-treatment in the water treatment plant
- A higher level of ammonia was identified in the post-detox effluent. The concentration of total unionized ammonia was 23.9 mg/L, whereas the required maximum level is 20 mg/L. No aging test was performed to measure the ammonia degradation.
- It is expected that the metals concentration level will decline due to dilution from rainfall precipitation and natural degradation in the tailings pond.
- The decant solution contains very high levels of sulphate. Although there are no sulphate limits, this will impact treatment selection.
- Currently, there is no discharge limit for thiocyanate; however, thiocyanate concentration in the residue decant is high (Analysis 3: 520 mg/L) and may require further treatment if thiocyanate limits and/or toxicity tests are imposed. There is currently no treatment allowed for thiocyanate.

A bench-scale test should be completed with aged effluent to confirm design basis for water treatment plant. Feasibility level design was not completed for the water treatment system, as the design basis is not fully developed at this stage.

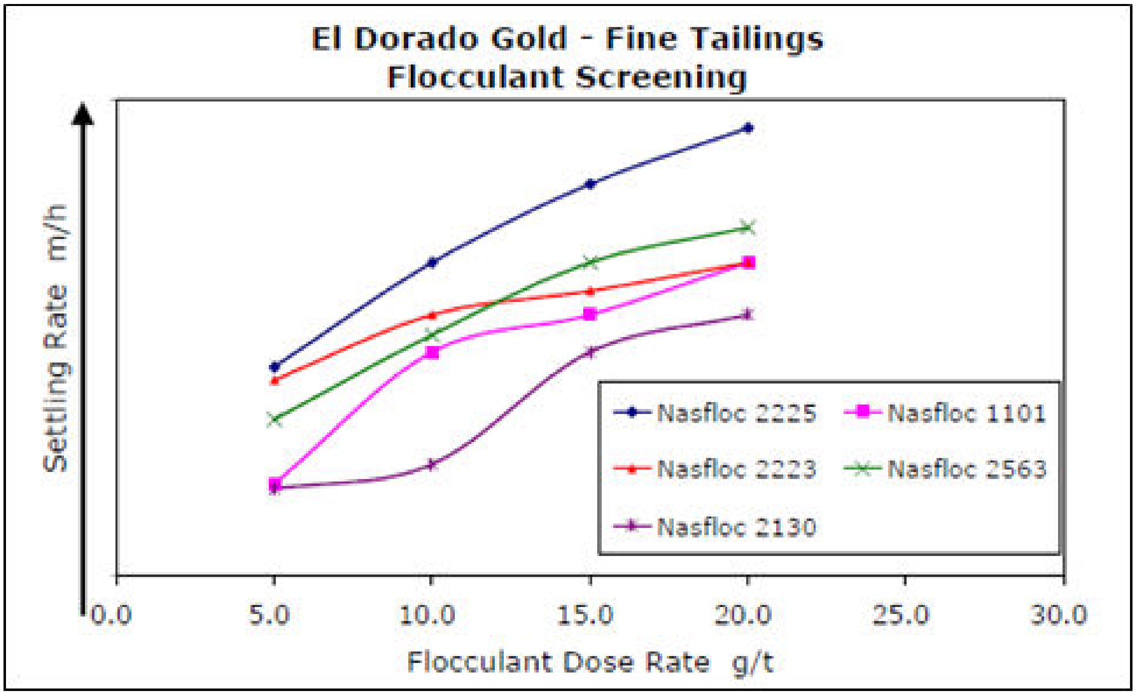
13.12 Thickener Testwork

Thickener and flocculant test work was completed by FLSmidth in 2010. Samples were generated from the Wardell Armstrong pilot plant and locked cycle test work.

Fine (P80 = 85 µm), coarse (P80 = 125 µm), TOP and BOT tailings were tested to determine flocculant dosage and settling rates. The results are shown in Figure 13.14 to Figure 13.17 and summarized in Source: FLSmidth (2010)

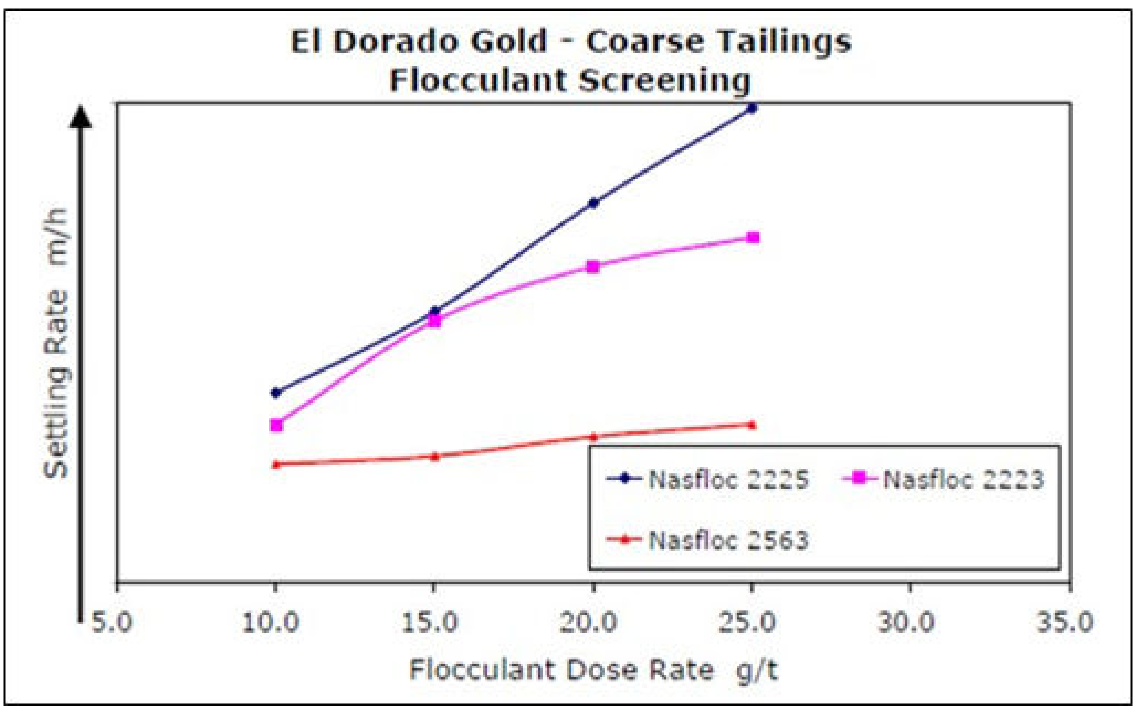
Table 13.29 and Table 13.30 are used in sizing for a High Rate Thickener (HRT).

Figure 13.14: Flocculant Test Work for Fine Tailings



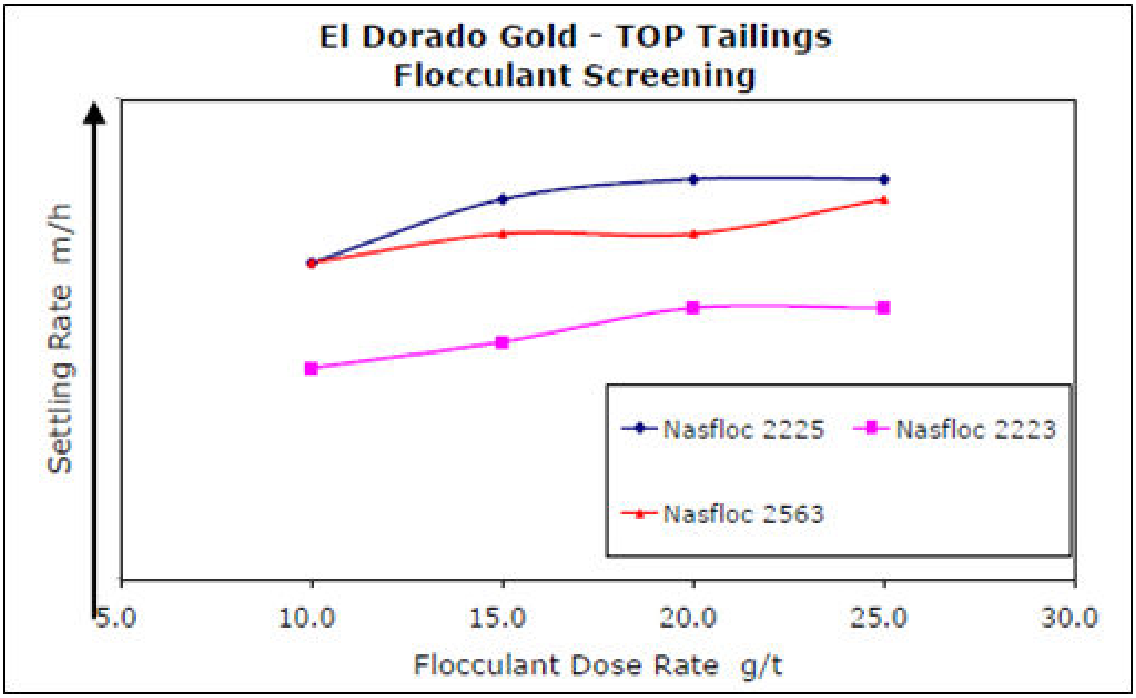
Source: FLSmidth (2010)

Figure 13.15: Flocculant Test Work for Coarse Tailings



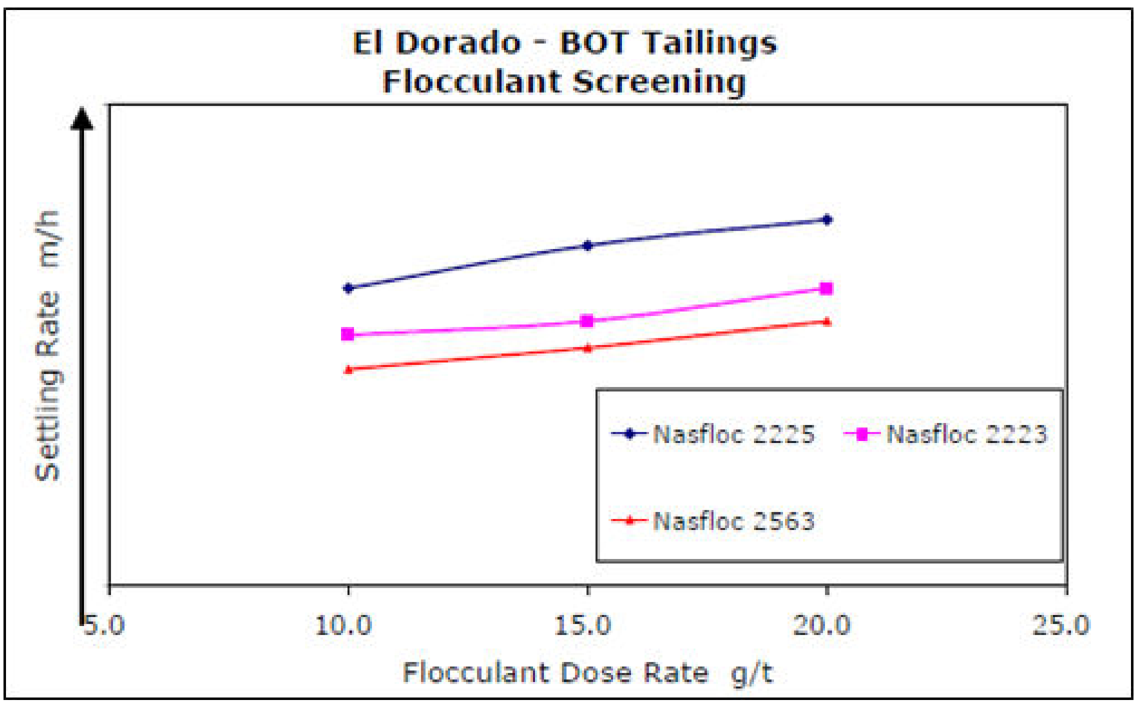
Source: FLSmidth (2010)

Figure 13.16: Flocculant Test Work for TOP Tailings



Source: FLSmidth (2010)

Figure 13.17: Flocculant Test Work for BOT Tailings



Source: FLSmidth (2010)

Table 13.29: Thickener Settling Test Summary on Flotation Concentrate

Concentrate – Thickener Based on Coarse Concentrate	HRT
Feed Solids Concentration Required (wt%)	3
Recommended Flocculant Dose (g/mt)	35
Recommended Minimum Unit Area (m^2/mtpd)	0.348
Minimum Recommended Diameter (m)	16
Design Underflow Density (wt%)	35
Retention Time Required (hours)	3.5
Yield Stress at Design Underflow (Pa)	<20

Table 13.30: Thickener Settling Test Summary on Leached Residue

CN Residue – Thickener Based on Coarse CN Residue	HRT
Feed Solids Concentration Required (wt%)	3
Recommended Flocculant Dose (g/mt)	50
Recommended Minimum Unit Area (m^2/mtpd)	0.087
Minimum Recommended Diameter (m)	8
Design Underflow Density (wt%)	35
Retention Time Required (hours)	1.25
Yield Stress at Design Underflow (Pa)	<20

Nasfloc 2225 provided the best flocculant settling rates. However, it was reported that the overflow clarity was found to be poor.

The settling rate for the leached residue was unexpectedly much faster than the flotation concentrate. It is recommended to keep the slower settling rate (higher value) for the determination of unit area since it is unusual to see such a big difference between similar materials.

The design criteria used for flocculant dosage and thickener sizing is summarized in Table 13.31.

Table 13.31: Design Criteria – Thickener

Flotation Concentrate Thickener	Units	Design	Notes
Solid Loading, Feed Rate	m^2/tpd	0.348	
Flocculant Dosage	g/t	35	
Leached Residue Thickener			
Type of Thickener	-	HRT	
Solid Loading, Feed Rate	m^2/tpd	0.348	Used same rate from flotation concentrate thickener
Flocculant Dosage	g/t	50	

13.13 Gold Recovery

Gold balances were completed, and gold overall recoveries were estimated for granite, saprolite and garimpeiros tailings feed material based on historical metallurgical test results as follows.

Table 13.32: Gold Recoveries

Feed Material	Feed Grade g Au/t	Gravity Stage Recovery	Flotation Stage Recovery	CIL Stage Recovery	Overall Recovery
Granite	1.32	24%	93% 4.5% mass pull	95%	90.9%
Saprolite	1.03	14%	71% 7.7% mass pull	93%	70.8%
Garimpeiros Tailings	1.11	14%	86.4% 2.8% mass	96%	85.4%

13.14 Recommendations

- It is recommended to complete additional metallurgical test work which has been initiated at SGS Geosol:
    - To assess the variability of ore hardness to adequately size the SAG and Ball mills. This includes completing SMC and BWI tests on 20 representative granite samples and SPI tests on five of those samples.
    - To confirm gravity recovery results. This includes completing gravity separating tests via a laboratory gravity centrifuge on 20 representative granite samples.
    - To confirm flotation recovery results at P80 pf 125 µm. This includes completing rougher and cleaner flotation tests on gravity tailings samples generated from the gravity tests above.
    - To confirm leach gold extraction results. This includes completing bottle roll tests encompassing intensive leaching, cyanidation kinetics, CIL, gold-to-carbon adsorption curves and Freundlich isotherms.
    - To investigate the potential of increasing gold extraction by regrinding the flotation concentrate. This includes completing regrind/leach tests and establishing a regrind curve for mill sizing.
    - To confirm and optimize cyanide destruction parameters. This includes completing cyanide destruction tests using SO_2/air and Caro's acid methods.
    - To complete additional geochemical testing of the CIL tailings.
- Following the completion of the additional metallurgical test work, it is recommended to confirm and/or update the process design criteria, gold recoveries and confirm equipment sizing.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

SRK was commissioned to audit a deposit (rock) mineral resource model prepared by Eldorado (2019) and a tailings mineral resource model prepared by GMS (2021) in terms of international mineral resource estimation and reporting guidelines and to assume independent Qualified Person responsibility for these mineral resource models in this study. The Mineral Resource Statement presented herein represents the third mineral resource evaluation for the Tocantinzinho gold project in accordance with the Canadian Securities Administrators' National Instrument 43-101.

The deposit mineral resource model considers 78 core boreholes (22,134 metres) drilled during February 2004 to September 2008 by Brazauro and 74 core boreholes (22,030 metres) drilled during September 2008 to December 2010 by Eldorado. In addition, some 155 tailing boreholes (1,594.04 metres) drilled by Eldorado in 2011 and 2014 were considered for the tailings mineral resource model. An audited Mineral Resource Statement was prepared by Ms. Camila Passos, PGeo (APGO#2413) with geostatistical support from Dr. Oy Leuangthong, PEng (PEO#90563867) supervised by Mr. Glen Cole, PGeo (APGO#1416). All are full-time employees of SRK. By virtue of her education, relevant work experience, and affiliation to a recognized professional association, Ms. Passos is the independent Qualified Person as this term is defined in National Instrument 43-101, for the Mineral Resource Statement presented herein. The effective date of the Mineral Resource Statement is December 10, 2021. In this section, the term "SRK' refers to the Qualified Person for the Mineral Resource Statement.

A feasibility study on the Tocantinzinho gold project was initially completed by Eldorado in 2015, and an updated technical report was published on June 21, 2019, with the effective date of the reported mineral resource being September 30, 2018. No additional exploration data has been acquired since that time. SRK's audit of the Eldorado (2019) mineral resource model is described in this section.

The database used to estimate the Tocantinzinho gold project mineral resources was audited by SRK. SRK is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries for gold mineralization and that the assay data are reliable to support mineral resource estimation.

Minesight® was used to construct the geological solids, prepare assay data for geostatistical analysis, construct the block model, estimate gold grades, and tabulate mineral resources for the rock material. For tailings material, Leapfrog™ version 5.1.2 was used to construct the solid and GEMS™ version 6.8.2 to prepare assay data for geostatistical analysis, construct the block model, estimate gold grades, and tabulate mineral resources. SRK used a combination of GEMS™ version 6.8.4, Leapfrog Geo™ version 5.1.2 and GSLib (Geostatistical Software Library) software for the audit.

This section describes the mineral resource estimation methodology and summarizes the key assumptions considered to prepare the geology and the mineral resource model and the results of the SRK audit. In the opinion of SRK, the resource evaluation reported herein is a reasonable representation of the global gold mineral resources found in the Tocantinzinho gold project at the current level of sampling. The mineral resources have been estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines and are reported in accordance with the Canadian Securities Administrators' National Instrument 43-101. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve.

14.2 Resource Estimation Methodology

The SRK audit involved the review of the following aspects of the Tocantinzinho resource models:

- Database compilation and verification.
- Construction of wireframe models for the boundaries of the gold mineralization.
- Definition of resource domains.
- Data conditioning (capping and compositing) for geostatistical analysis and variography.
- Grade interpolation in a 3D block model.
- Model validation and resource classification.
- Assessment of "reasonable prospects for eventual economic extraction" and selection of appropriate cut-off grades.
- Preparation of the Mineral Resource Statement.

14.2.1 Resource Database

The resource database comprises samples from core and tailings boreholes drilled from surface. The header, down-hole survey, lithology intervals and assay results were received on October 14, 2020. SRK was provided with a database comprising 296 core boreholes (82,805 metres); 152 (44,163 metres) of which are considered resource boreholes, 133 (35,665 metres) of which are considered exploration boreholes, 5 (1,488 metres) are considered geotechnical boreholes and 6 (1,490 metres) are considered metallurgical boreholes. The tailings database received comprises 155 boreholes (1,594.04 metres). A re-survey of a large proportion of the tailings collars were undertaken in 2021 to improve data confidence.

The final database used for the resource model comprises 78 (22,134 metres) boreholes drilled by Brazauro and includes 11,752 samples assayed for gold and 74 (22,030 metres) boreholes drilled by Eldorado and includes 11,439 samples assayed for gold and 734 specific gravity samples (Table 14.1). The tailings database considered 155 boreholes (1,594.04 metres) and 1,434 samples assayed for gold (

Table 14.2).

Table 14.1: Summary of Rock Drilling Data Considered for Resource Modelling

Company	Period	Type	No of Holes	Total Length (m)	Assay
Brazauro	2004	DDH	17	4,196.79	2,132
	2005	DDH	14	3,758.88	1,844
	2006	DDH	10	2,532.94	1,414
	2007	DDH	11	3,247.48	1,751
	2008	DDH	26	8,397.63	4,611
SubTotal			78	22,133.72	11,752
Eldorado	2008	DDH	11	3,517.95	1,955
	2009	DDH	47	14,633.22	7,487
	2010	DDH	16	3,878.35	1,997
SubTotal			74	22,029.52	11,439
Total			152	44,163.24	23,191

Table 14.2: Summary of Tailing Drilling Data Considered for Resource Modelling

Company	Period	Type	No of Holes	Total Length (m)	Assay
Eldorado	2011	DDH	75	946.15	850
	2014	DDH	80	647.89	584
Total			**155**	**1,594.04**	**1,434**

All borehole collars were surveyed according to UTM coordinates (SAD69 datum, Zone 21S). SRK was also provided with a high-resolution topographic surface for geological modelling. Brazauro completed down hole surveys at different intervals that varied from 15 to 376 metres using a FlexIt tool. Boreholes TOC 07-47 (341.37 metres length) and TOC 07-48 (328.57 metres length) were not surveyed. Eldorado down hole surveys were completed at different intervals mainly varying each 50 or 60 m using different tools (Reflex FlexIt and Reflex EZ Shot Instrument). Boreholes TOC 09-111 (230.12 m length) and TOC 09-135 (64 m length) were not surveyed. SRK considered recovery of the boreholes acceptable based on observations of the boreholes that were checked during the site visit. Less than one (1) percent intervals in the tailings recovery table presented inconsistencies showing recovery length greater than the length interval. Recovery from tailings varied from 0 to 100 percent with an average recovery of 97 percent.

Ms. Camila Passos, PGeo visited the Tocantinzinho gold project during November 21 to 24, 2020 to review all inputs to the exploration database. SRK is satisfied that the exploration work carried out by Brazauro and Eldorado was conducted in a manner consistent with industry best practices and, therefore, the exploration data and the drilling database are sufficiently reliable to support a mineral resource evaluation.

14.2.2 Domain Modelling

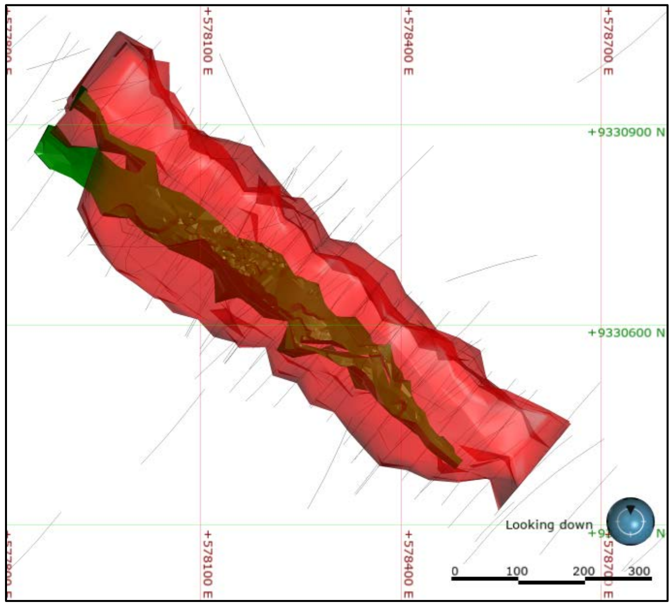
14.2.2.1 Rock

The gold mineralization at the Tocantinzinho gold project occurs inside a granite intrusion in a northwest-southeast structural corridor with a sub-vertical dip. Gold occurs up to 900 m along strike, up to 350 m wide, and from the surface to approximately 400 m in depth. The mineralized zone is associated with hydrothermal alteration characterized by low sulphidation (predominantly pyrite), chloritization and carbonation.

A combination of lithology and assays were used to define the boundaries of the gold mineralization in the Tocantinzinho deposit resulting in three wireframes (Figure 14.1):

- A northwest-southeast trending envelope representing the granite mineralization, limited on the footwall and hanging wall by unmineralized hematite granite and quartz monzonite, respectively.
- A barren andesite dyke that intrudes the granite.
- A weathering surface separating the saprolite from the fresh rock.

Figure 14.1: Plan View of the Granite Envelope (red) and Andesite Dyke (Green)



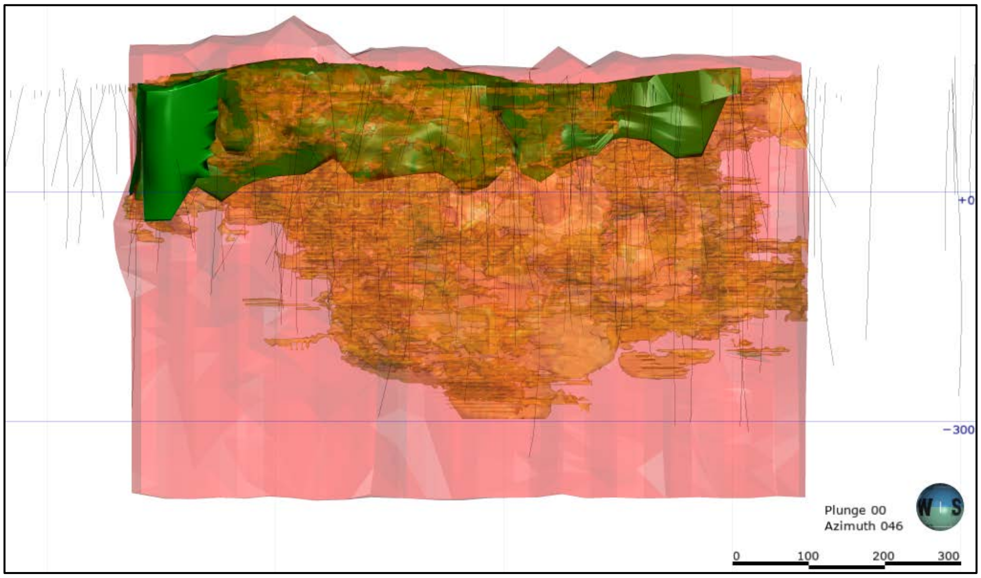
Source: SRK (2021)

To construct a resource domain inside the mineralization envelope, an indicator shell at a grade threshold of 0.3 g/t gold and contoured a domain at 0.42 probability threshold was constructed using a tool called Pack in Minesight software. The following steps were applied:

- The threshold value of 0.30 g/t gold was determined using histograms and probability curves as well as indicator variography.
- Assay samples were converted into a binary indicator code equal to one if the composited assay is equal to or higher than 0.30 g/t gold or zero if the assay is less than 0.30 g/t.
- The indicators were estimated into a block model generating a range of probability values between zero and one.
- Shell outline selection was done by checking contoured probability values. The 42% probability value was determined visually.
- The 42% shell was then clipped to be bound inside the mineralization corridor and edited on plan and section views to remove artifacts and isolated portions without 3D continuity.
- Assays samples were selected inside the final shell and outside the modelled andesitic intrusion.

Figure 14.2 shows the indicator shell inside the mineralization envelope.

Figure 14.2: Longitudinal View of the Indicator Shell inside the Mineralized Envelope

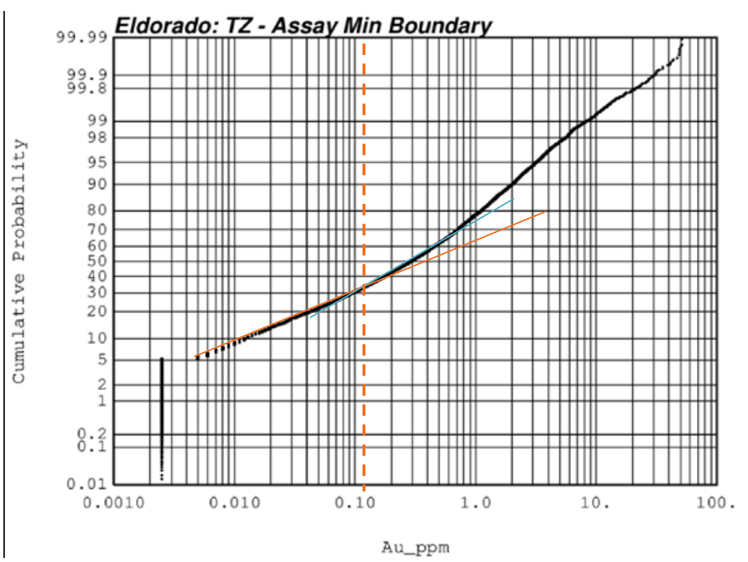


Red – Mineralized Envelope, Orange – Indicator Shell, Green – Andesite dyke.
Source: SRK (2021)

To assess the reasonableness of the mineral resource domain, SRK performed the following verification steps:

- Review the grade threshold used for modelling the resource domain (indicator shell) and assessed the appropriateness of the probability threshold based on indicator kriging theory.
- SRK checked the assay samples in a probability plot and consider a grade threshold of approximately 0.15 g/t gold reasonable for domaining inside the mineralized envelope (Figure 14.3).

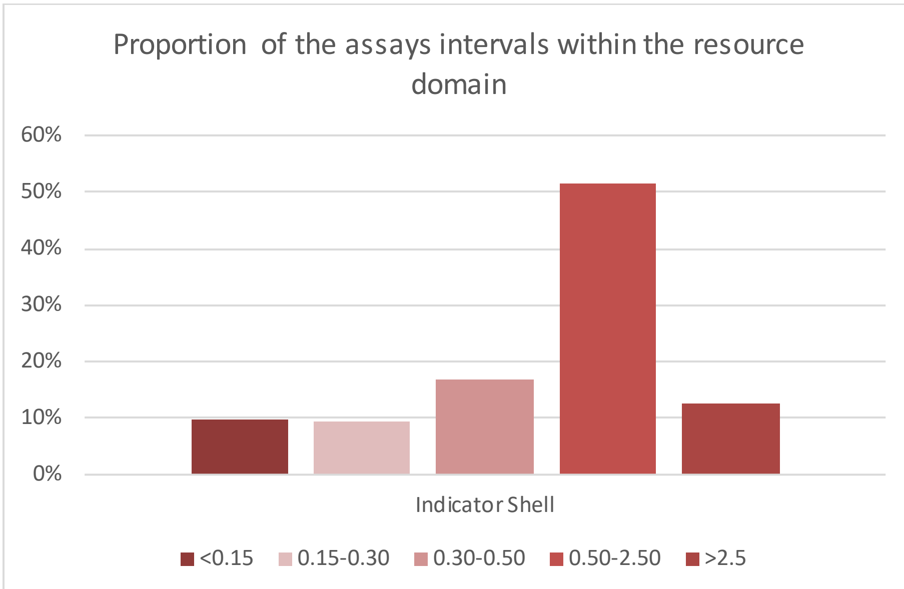
Figure 14.3: Probability Plot from Assays Inside the Mineralized Envelope



Source: SRK (2021)

- SRK coded the assay intervals into groups based on the grade and their location inside/outside of the indicator shell and analyzed the length distribution of the intervals. The results of this comparison for the indicator shell are presented in Figure 14.4. Within the 0.3 g/t gold shell 19% internal dilution was found suggesting acceptable adherence to the threshold chosen.

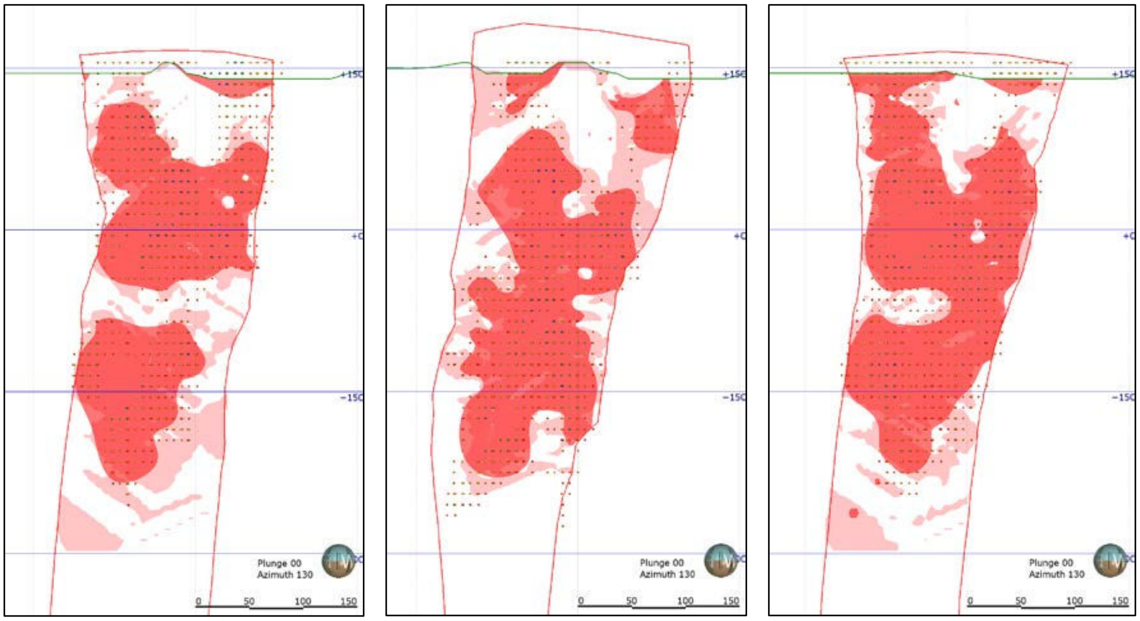
Figure 14.4: Length Distribution of the Coded Assays within Indicator Shell by Grade Thresholds



Source: SRK (2021)

- SRK reviewed the indicator variogram and orientation used for modeling and assessed the sensitivity of the domain to the orientation.
- SRK checked the declustered proportion of intervals above the 0.3 g/t gold threshold (60% less than 0.3 g/t gold or 40% greater than 0.3 g/t gold), which should generally correspond to the iso-probability shell selected for domaining.
- Using alternate orientations, SRK performed an independent indicator kriging (using different estimation parameters) and exported the estimated blocks below 40% probability for comparison.
- SRK considered an alternate, implicit-based approach using the Leapfrog intrusion modelling approach.
- SRK visually compared the alternate models (points and Leapfrog shell) against the modeled shell (Figure 14.5).

Figure 14.5: Example of Vertical Sections Comparing Alternate Grade Shells

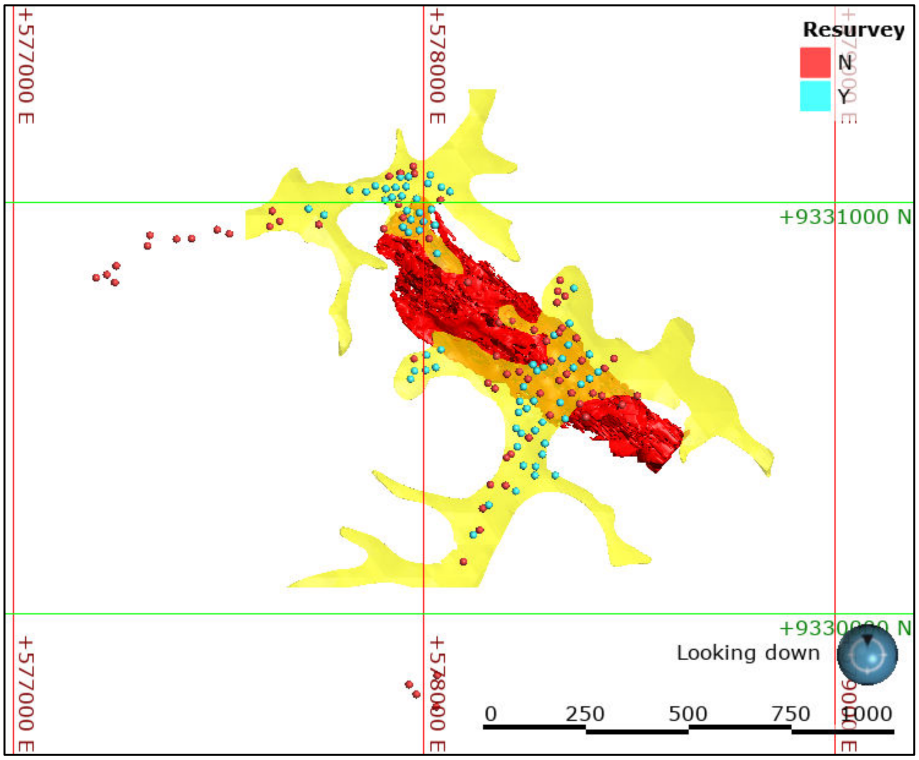


Light Red – Eldorado Shell, Points – SRK Indicator probability plots, Red – Shell Generate in Leapfrog
Source: SRK (2021)

14.2.2.2 Tailings

The logging for tailings boreholes was used to generate a wireframe in Leapfrog™ version 5.1.2 using an implicit-based approach (Figure 14.6).

Figure 14.6: Plain View Showing Rock Mineralization (red) and Tailings (yellow). Collars in blue were re-surveyed by GMS



Source: SRK (2021)

SRK checked the tailings resource model visually comparing the logging against the tailings' wireframe shell provided.

14.2.2.3 SRK Comments

SRK reviewed the domaining methodology adopted, the various input parameters (threshold for grade and probability, and associated variogram and estimation parameters) and considered an alternate implicit approach. The alternative modelling resulted in a 'mineralized region' that is similar to that defined in the original model.

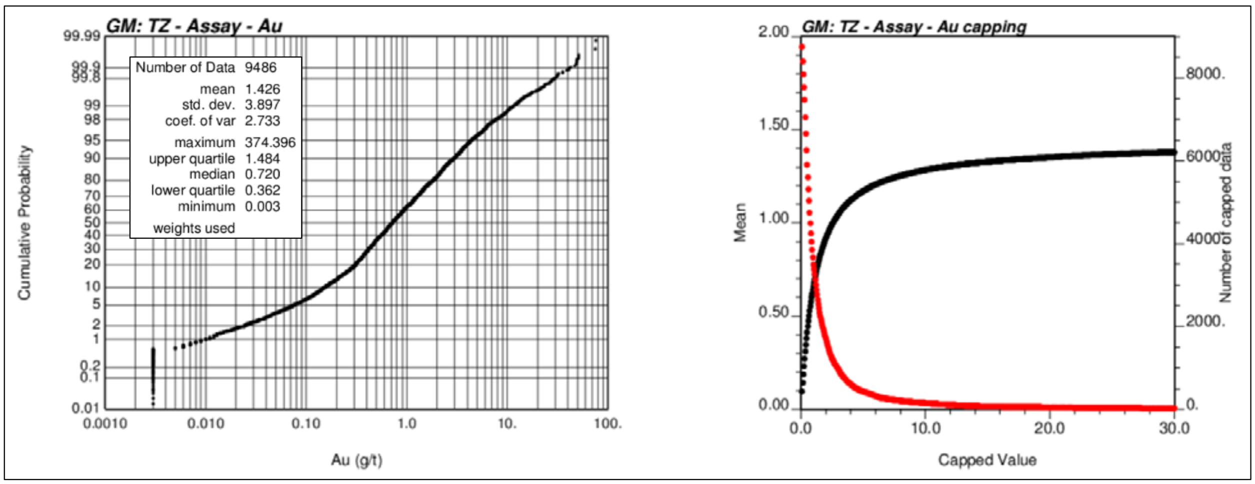
While some enhancements can be made to the methodology and to various inputs, SRK considers that resource domain reasonably captures the mineralization volume and is adequate for mineral resource estimation purposes.

14.2.3 Evaluation of Outliers

14.2.3.1 Rock

To further limit the influence of high gold grade outliers during grade estimation, assays were capped before compositing. Capping was performed for the assays inside the indicator shell. The selected capping value was 25 g/t gold (Figure 14.7). A total number of 35 assays were capped. SRK used a combination of probability plots and capping sensitivity plots to check the selected capping grade. Grade inflections in the probability plot support the selected capping value.

Figure 14.7: Grade Probability Plot (Left) and Capping Sensitivity Curve (Right) for Rock

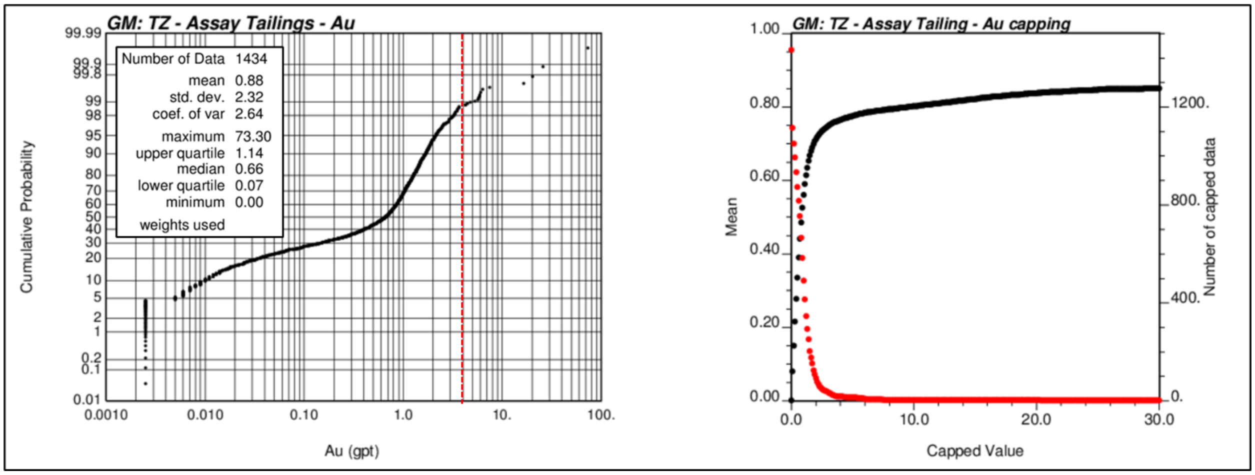


Source: SRK (2021)

14.2.3.2 Tailings

The influence of high gold grade outliers was also constrained for the tailings assays. Capping was performed for the assays inside the tailings wireframe. The selected capping value was 4 g/t gold. A total number of four (4) assays were capped. SRK relied on a combination of probability plots and capping sensitivity plots to check the selected capping grade (Figure 14.8). Grade inflections in the probability plot support the selected capping value.

Figure 14.8: Grade Probability Plot (Left) and Capping Sensitivity Curve (Right) for Tailings



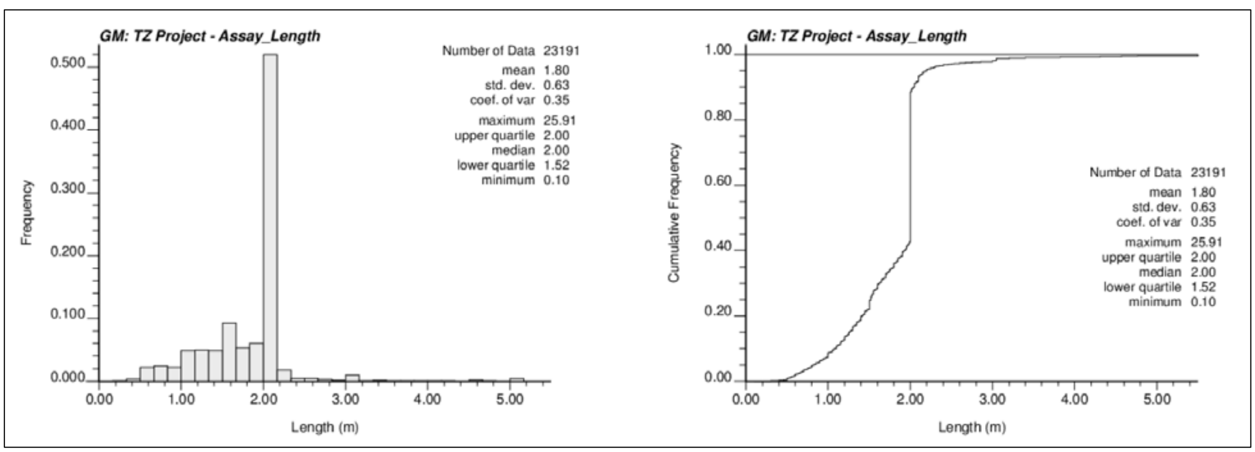
Source: SRK (2021)

14.2.4 <u>Compositing</u>

14.2.4.1 <u>Rock</u>

All assay intervals within the resource domain were composited to a length of 2.0 m. The cumulative plots for all samples considered during the resource estimation show that 90% of assays are sampled at 2 m or less, supporting the composite length chosen by Eldorado (Figure 14.9).

Figure 14.9: Rock Assay Lengths Distributions



Source: SRK (2021)

The selected capped values, along with the uncapped assays and capped assays and composites statistics are provided in Table 14.3. Overall, the global mean gold grade dropped by 5%, and the coefficient of variation reduced by 49%.

Table 14.3: Summary Basic Statistics for Raw Sample, Capped Assay and Capped Composite Data (SD = standard deviation, CV = coefficient of variation)

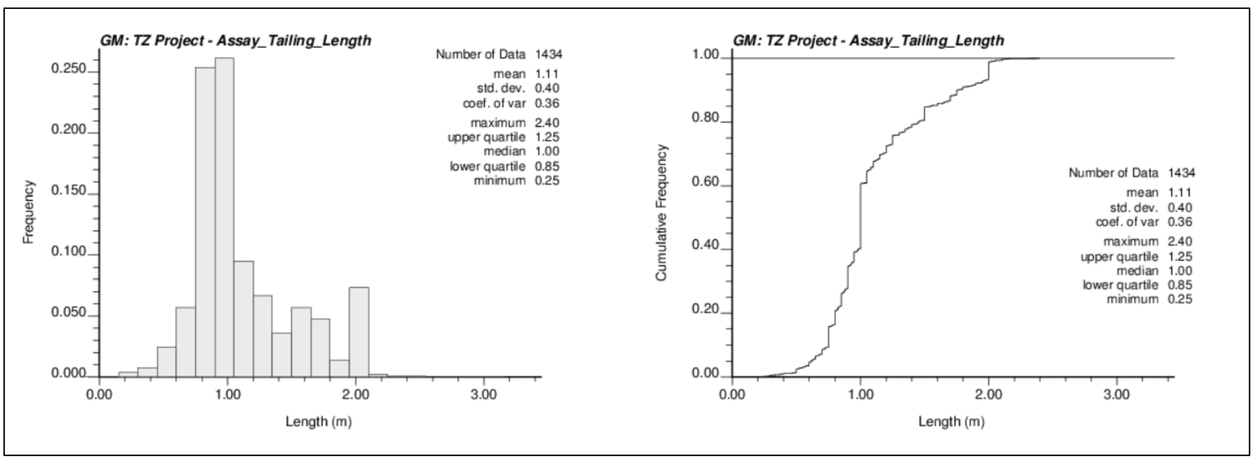
Data	Count	Mean	Std Dev.	CV	Min	Max	Capping Impact		Compositing Impact	
							Mean	CV	Mean	CV
Assays*	9,486	1.43	3.90	2.73	0.00	374.40				
Capped Assays	9,486	1.37	2.29	1.67	0.00	25.00	-4%	-39%		
Composites	8,571	1.36	1.97	1.45	0.00	25.00			-1%	-13%

** Length weighted*

14.2.4.2 Tailings

All tailings assay intervals within the resource domain were composited to a length of 3.0 m. The cumulative plots for all samples considered during the resource estimation show that 100% of assays are sampled at 3 m or less, supporting the composite length chosen by GMS (Figure 14.10).

Figure 14.10: Tailings Assay Lengths Distributions



Source: SRK (2021)

The selected capped values, along with the uncapped assays and capped assays and composites statistics are provided in Table 14.4. Overall, the global mean gold grade dropped by 2%, and the coefficient of variation reduced by 40%.

Table 14.4: Summary Basic Statistics for Raw Sample, Capped Assay and Capped Composite Data (SD = standard deviation, CV = coefficient of variation)

Data	Count	Mean	Std Dev.	CV	Min	Max	Capping Impact		Compositing Impact	
							Mean	CV	Mean	CV
Assays*	1,019	1.03	1.04	1.01	0.00	16.55				
Capped Assays	1,019	1.00	0.73	0.73	0.00	4.00	-4%	-28%		
Composites	353	1.02	0.61	0.60	0.01	4.00			2%	-16%

** Length weighted*

14.2.4.3 SRK Comments

SRK reviewed the applied capping methodology and considers it reasonable for the Tocantinzinho gold project.

14.2.5 Variography

Minesight mining software was used to model the spatial continuity of gold for the Tocantinzinho gold project. The variogram was modeled with two spherical structures (Table 14.5).

Table 14.5: Summary of Tocantinzinho Variogram Model Parameters

Minesight Angles			Variogram Model						
Z	X	Z	Nugget	Str. No.*	CC*	Type	X (m)	Y (m)	Z (m)
-60	-9	97	0.2	1	0.78	Spherical	10.6	8.2	7.2
-90	-5	-15		2	0.02	Spherical	594	102.6	417.3

** Str. No. = structure number, CC = variance contribution*

14.2.5.1 SRK Comments

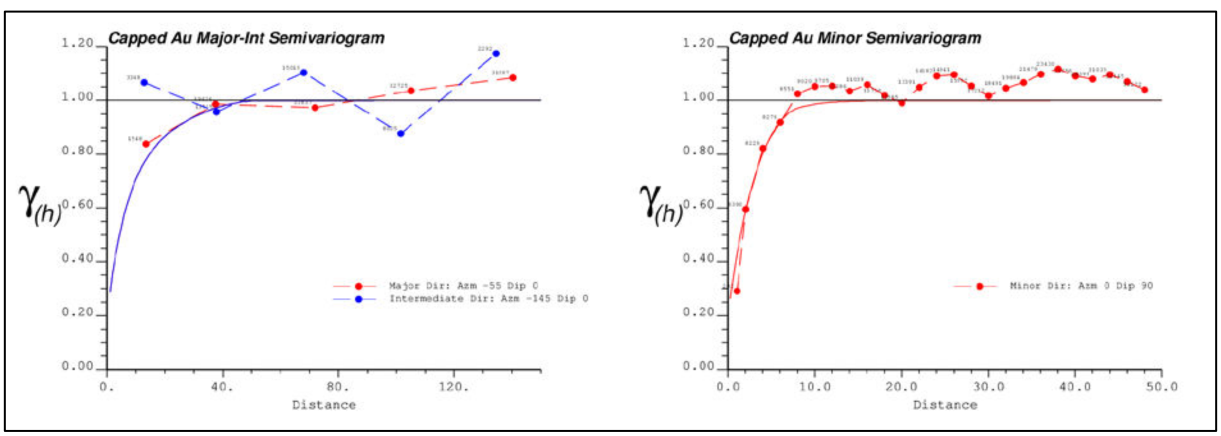
The first structure is applied to only one block because of the small ranges applied. The second structure of the variogram is in the east-west direction, different from the strike of the mineralization that is northwest-southeast. The second structure represents only 2% variability of the variogram.

SRK audited the variogram models by reviewing the detailed parameters used to generate the variograms, and independently calculated gold variograms using Geostatistical Software Library (GSLib, Deutsch and Journel, 1998) (Table 14.6). SRK assessed two different spatial metrics: (1) traditional semivariogram of original gold, and (2) correlogram of original gold. SRK verified the first structure of the variogram that represents almost 98% variability of Eldorado's variogram Finally, SRK also modeled a variogram following the main direction of the mineralization (SRK variogram) (Figure 14.11). Downhole variograms were calculated to determine the nugget effect.

Table 14.6: Gold Variogram Parameters for the Resource Domain

Variograms	Minesight Angles			GSLIB Angles			Variogram Model						
	Z	X	Z	ANG1	ANG2	ANG3	Nugget	Str. No.*	CC*	Type	Strike Range (X)	Dip Range (Y)	Vert Range (Y)
Eldorado Variogram	-60	-9	97				0.2	1	0.779	Spherical	10.6	8.2	7.2
	-90	-5	-15					2	0.021	Spherical	594	102.6	417.3
SRK Fit Variogram				-60	-9	-79	0.2	1	0.5	Spherical	15	10	7
				-90	-9	-79		2	0.3	Spherical	40	35	8
SRK Variogram				-55	0	0	0.2	1	0.57	Spherical	20	20	8
				-55	0	0		2	0.23	Spherical	53	53	8

Figure 14.11: SRK Gold Variogram for the Resource Domain



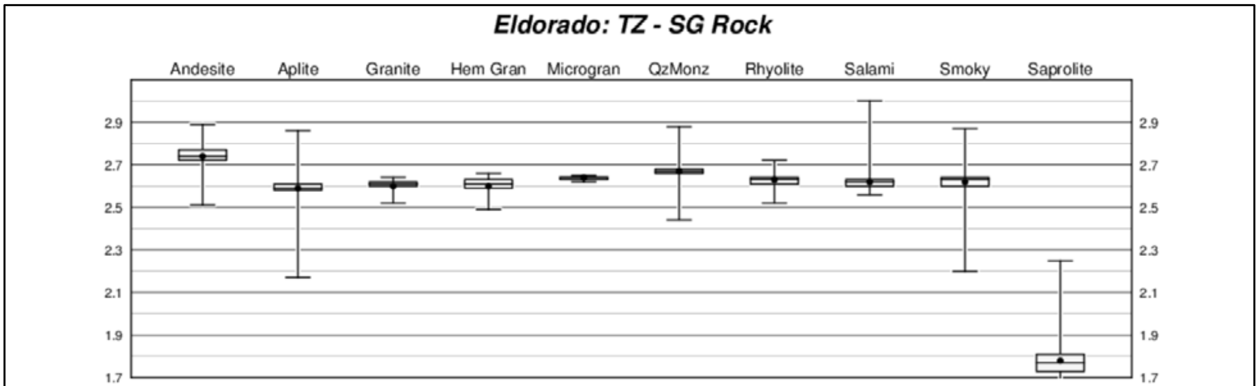
Source: SRK (2021)

14.2.6 Specific Gravity

Specific gravity was measured using a standard weight in water/weight in air methodology on the unweathered core for complete sample intervals. Specific gravity for weathered material was measured using the displacement volume methodology. A total of 678 and 56 specific gravity measurements were taken for unweathered and weathered material, respectively (Figure 14.12). SRK has reviewed the specific gravity database and agrees that the application of an average specific gravity is appropriate for mineral resource estimation.

For the mineral resource model, a specific gravity value of 1.78 was applied for saprolite. For the mineralized granite, a mean specific gravity of 2.62 based on the mineralized granites with salami and smoky textures was applied.

Figure 14.12: Boxplot of Specific Gravity by Lithology



	Andesite	Aplite	Granite	Hem Gran	Microgran	QzMonz	Rhyolite	Salami	Smoky	Saprolite	
Number of data	73	75	20	88	4	52	34	173	159	56	Number of data
Mean	2.74	2.59	2.6	2.6	2.64	2.67	2.63	2.62	2.62	1.78	Mean
Std. Dev.	0.05	0.08	0.03	0.03	0.01	0.05	0.04	0.05	0.05	0.06	Std. Dev.
Coef. of Var.	0.02	0.03	0.01	0.01	0.0	0.02	0.01	0.02	0.02	0.03	Coef. of Var.
Maximum	2.89	2.86	2.64	2.66	2.65	2.88	2.72	3.0	2.87	2.25	Maximum
Upper quartile	2.77	2.61	2.62	2.63	2.64	2.68	2.64	2.63	2.64	1.81	Upper quartile
Median	2.74	2.59	2.61	2.61	2.64	2.67	2.63	2.62	2.63	1.77	Median
Lower quartile	2.72	2.58	2.6	2.59	2.63	2.66	2.61	2.6	2.6	1.73	Lower quartile
Minimum	2.51	2.17	2.52	2.49	2.62	2.44	2.52	2.56	2.2	1.66	Minimum

(Hem Gran – Hematite Granite; Microgran – Microgranite; QzMonz – Quartz Monzonite; Salami – Mineralized Granite with Salami Texture; Smoky – Mineralized Granite with Smoky Texture)
Source: SRK (2021)

No specific measurements were made in the tailings material. A specific gravity of 1.60 was applied in the block model. The tailings material is moderately well compacted and presents specific gravity higher than sand which suggests that the value chosen reasonable. SRK however recommend that additional specific gravity measurements in the tailings material should be generated for an accurate assessment of tailings specific gravity.

14.2.7 Block Model Parameters

The selection of the block size considered the borehole spacing, composite length, the geometry of the modelled zone and the anticipated mining method. An unrotated percent block-model was generated to better reflect the shape of the mineralization domain. The block model coordinates are based on the local UTM grid (SAD 69 datum, Zone 21S). The block size is 10 by 10 by 10 metres. The block model definition is summarized in Table 14.7. The block model was assigned with the domain codes from the wireframes.

For the tailings material , Gems™ software version 6.8.2 was used to build the block model using the same parameters as applied for the rock block model.

Table 14.7: Tocantinzinho Block Model Specifications

Axis	Block Size (m)	Origin*	Number of Cells	Rotation
X	10	577,565	137	0
Y	10	9,330,066	121	0
Z	10	260	56	0

** (SAD 69 datum)*

14.2.8 Grade Estimation and Validation

14.2.8.1 Rock

Gold grades were estimated using ordinary kriging in three estimation runs summarized in Table 14.8. All passes used an ellipsoidal search. SRK observes that the search ellipsoid applied has a different rotation to the variogram model, and that a smaller search ellipsoid was used in the second pass than in the first pass. The search ellipse is oriented in accordance with the main direction of the mineralization. SRK understands that these parameters were based on the geological interpretation, data and variogram analyses. To limit the influence of high-grade composites, an outlier restriction for all composites higher than 20 g/t gold within 25 m was applied.

An average specific gravity of 2.62 and 1.78 was assigned to blocks corresponding to the unweathered and saprolite zones, respectively.

Table 14.8: Eldorado Estimation Parameters for Gold in the Rock Material

Method	Data	Pass	Minesight Rotation			No. Composites			Range (m)			Search Type	Comments
			Z	X	Y	Max Comps per Borehole	Min.	Max.	X	Y	Z		
OK	Capped	1	-55	0	0	3	4	9	65	35	60	Ellipsoid	Outlier restriction 20 g/t in 25 m
		2	-55	0	0	3	4	9	35	20	35	Ellipsoid	
		3	-55	0	0	2	2	6	80	45	75	Ellipsoid	

Block model estimates were validated by visually comparing block grades and nearby composite grades; statistically comparing ordinary kriged and nearest neighborhood estimates at no cut-off grade; and using swath plots in northing, easting and elevation to compare the block model against the nearest neighbour (declustered) model; and using a change-of-support method.

14.2.8.2 Tailings

Gold grades in the tailings were estimated using an inverse distance to a power of two methodology using two estimation runs summarized in Table 14.9. All passes used an ellipsoidal search.

Table 14.9: Estimation Parameters for Gold in the Tailings Material

Method	Data	Pass	Gems Rotation			No. Composites			Range (m)			Search Type
			Z	X	Y	Max Comps per Borehole	Min.	Max.	X	Y	Z	
ID2	Capped	1	0	0	0	3	4	15	60	60	12	Ellipsoid
		2	0	0	0	3	2	15	80	80	12	Ellipsoid

An average specific gravity of 1.60 was assigned to all tailings blocks by GMS.

14.2.9 SRK Audit and Comments

14.2.9.1 Rock

The SRK resource model audit was conducted in two parts. Firstly, SRK verified the mineral resources using the same estimation parameters the original model. This check yielded results within 3% of reported ounces in the original model. SRK considers that this difference in in-situ metal is acceptable and may be attributed to slight implementation differences between Minesight (original) and GEMS (SRK).

Secondly, SRK assessed the sensitivity of the mineral resource estimate to the estimation strategy by varying some parameters. Parameters assessed included:

- Maximum number of boreholes for a block estimate.
- Minimum and maximum number of composites.
- Effect of not using the outlier restriction.
- Type of search.

This independent analysis undertaken by SRK used the parameters tabulated in Table 14.10:

- Base Case (BC) – original estimation parameters.
- BC2 – original estimation parameters; using SRK Fit variogram where SRK tried to reproduce original variogram using the same direction in both structures.
- SRK 1 – same as Base Case with no outlier restriction to assess the impact of this tool.
- SRK 2 – same estimation parameters as original; using SRK variogram.
- SRK 3 – alternate estimation parameters.
- SRK 4 – Same parameters as SRK 3 with no outlier restriction.
- SRK 5 – Same parameters as SRK 4 and different search in the first pass.

SRK sensitivities that involved the SRK variogram used search ellipsoid ranges based on 1, 2 and 3 times the variogram range.

SRK considers that the SRK-selected inputs are sufficiently different from Eldorado's to present an impact assessment on the resource estimate.

Table 14.10: SRK Sensitivity Analysis Parameters

Case	Pass		No. Composites		Range (m)			Search	Comments
		Max Comps / BH	Min.	Max.	X	Y	Z	Type	
Base Case	1	3	4	9	65	35	60	Ellipsoid	Outlier restriction
	2	3	4	9	35	20	35	Ellipsoid	
	3	2	2	6	80	45	75	Ellipsoid	
Eldorado (SRK Fit Eld Variogram)	1	3	4	9	65	35	60	Ellipsoid	Outlier restriction
	2	3	4	9	35	20	35	Ellipsoid	
	3	2	2	6	80	45	75	Ellipsoid	
SRK 1	1	3	4	9	65	35	60	Ellipsoid	
	2	3	4	9	35	20	35	Ellipsoid	
	3	2	2	6	80	45	75	Ellipsoid	
SRK 2	1	3	4	9	65	35	60	Ellipsoid	Outlier restriction
	2	3	4	9	35	20	35	Ellipsoid	
	3	2	2	6	80	45	75	Ellipsoid	
SRK 3	1	5	6	10	50	50	10	Ellipsoid	Outlier restriction
	2	5	6	15	100	100	20	Ellipsoid	
	3	0	2	15	150	150	30	Ellipsoid	
SRK 4	1	5	6	10	50	50	10	Ellipsoid	
	2	5	6	15	100	100	20	Ellipsoid	
	3	0	2	15	150	150	30	Ellipsoid	
SRK 5	1	5	6	10	50	50	10	Octant*	* min 3 oct, max of 5 comps/oct
	2	5	6	15	100	100	20	Ellipsoid	
	3	0	2	15	150	150	30	Ellipsoid	

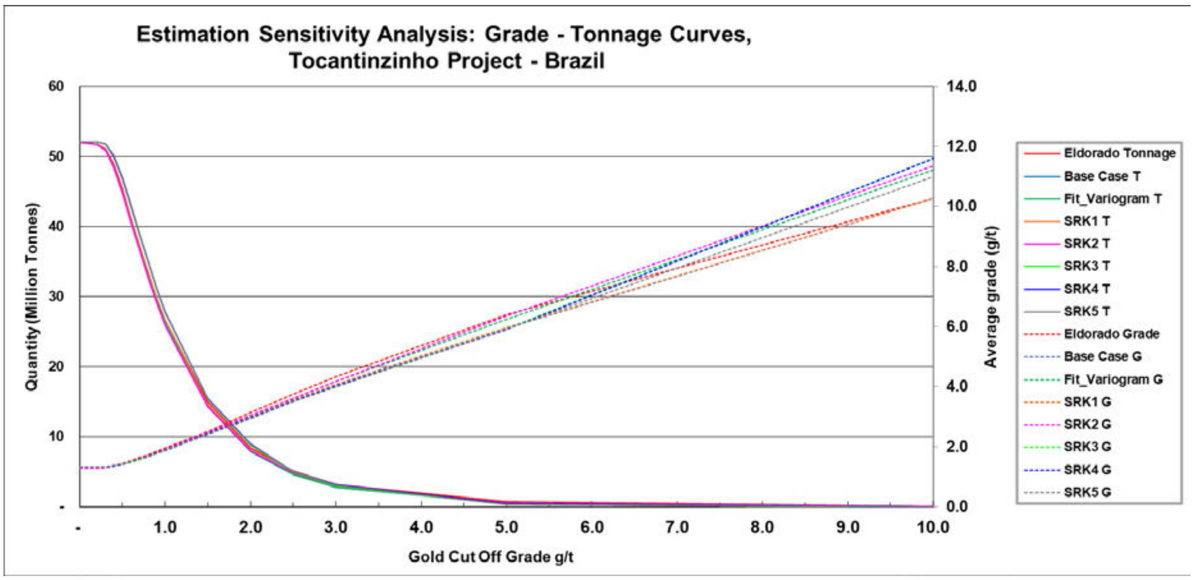
A summary of this analysis is provided in global grade and tonnage plots (Figure 14.13).

SRK noted that the use of outlier restriction (SRK 2 and SRK 3) impact less then 1% grade/metal content at lower cut-off grades. The use of different variograms (SRK Fit and SRK 2) and different search (SRK 5) also show no material impact. The biggest impact observed was due to applying different data parameters (SRK 3) where an increase of 4% in metal was observed.

In all cases, the grade-tonnage comparisons show that the mineral resource estimates are relatively insensitive to these slight differences.

Based on the results of this analysis, SRK concludes that there is minimal sensitivity to changes in estimation parameters, and therefore, the parameters used in the audited mineral resource model are reasonably robust.

Figure 14.13: Estimation Sensitivities Analysis for the Tocantinzinho Gold Project



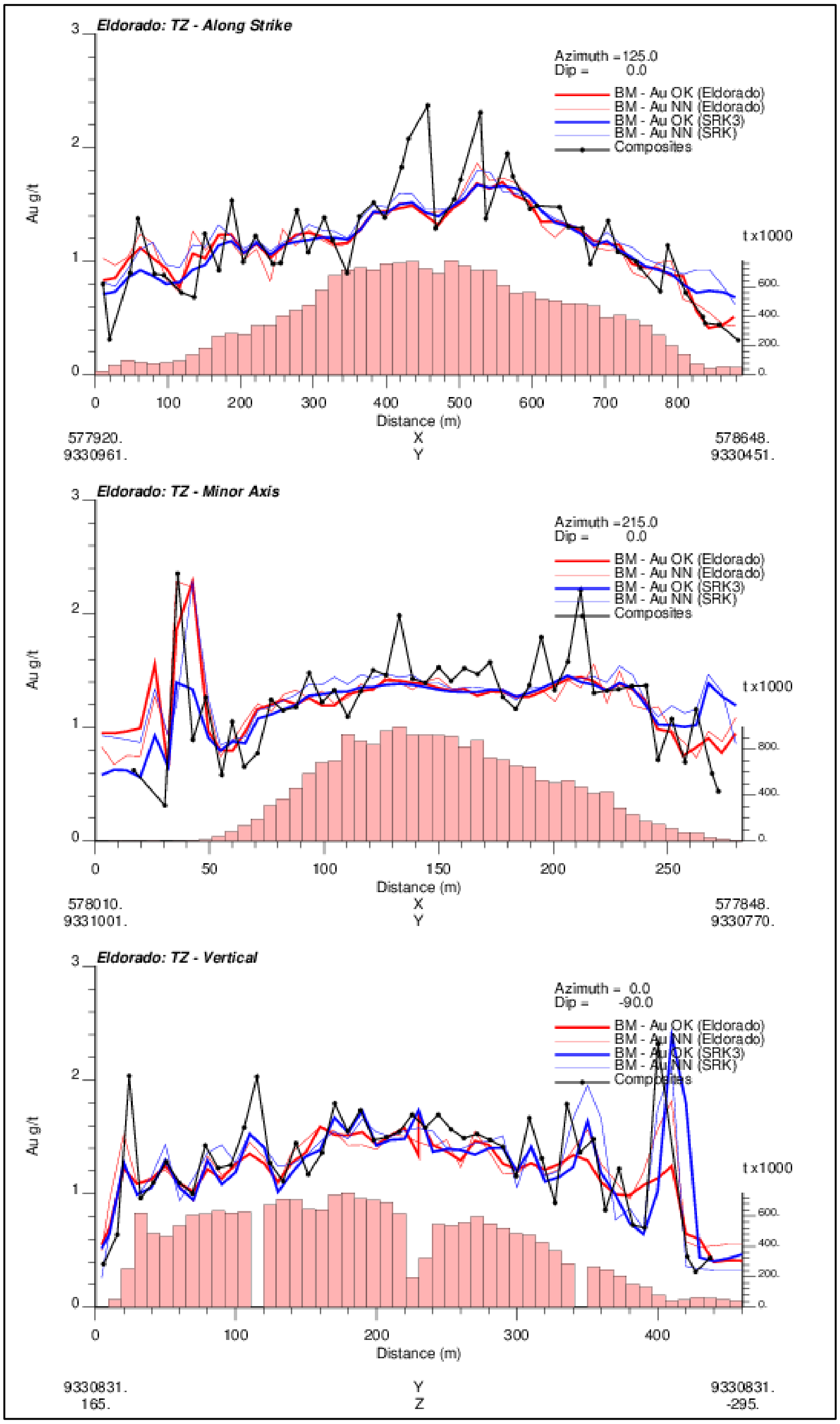
Source: SRK (2021)

Following SRK's analysis, it was decided to adopt the SRK 3 model for this study.

SRK validated the block model (SRK 3) using visual comparison of block estimates and informing composites, and statistical comparisons between composites and block model distributions at zero cut-off grade. No significant deviations between the block model and informing data were found. SRK also validated the block model using swath plots along strike, in the minor axis and by elevation to compare the block model against the informing composites and the nearest neighbours (declustered) model (Figure 14.14). This shows generally good agreement between the various block models and the nearest neighbours declustered data. As expected, clustered composite data is more variable than all other cases.

Based on these checks, SRK confirms the block estimates for the Tocantinzinho deposit is appropriate, and reasonably reflect the underlying borehole sampling data.

Figure 14.14: Swath Plot Along Strike (top), Across Minor Axis (middle) and Vertical (bottom)

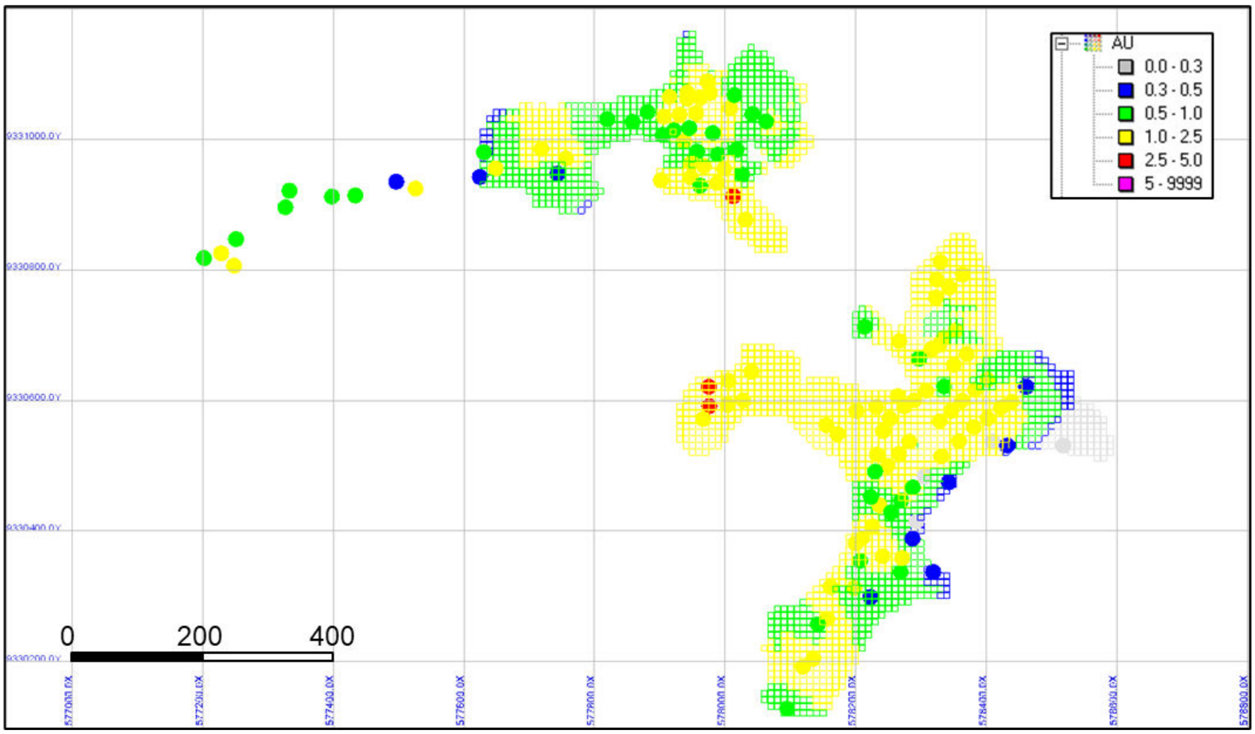


Source: SRK (2021)

14.2.9.2 Tailings

SRK validated the tailings block model undertaken by GMS using visual comparison of block estimates and informing composites (Figure 14.15), and statistical comparisons between composites and block model distributions at zero cut-off grade. No significant deviations between the block model and informing data were found. SRK also validated the block model using swath plots in east and north directions to compare the block model against the nearest neighbours (declustered) model. This shows generally good agreement between the block model and the nearest neighbours declustered data.

Figure 14.15: Planview Showing the Block Estimates and the Informing Composites



Source: SRK (2021)

Based on these checks, SRK confirms the block estimates for the tailings in the Tocantinzinho deposit is appropriate, and reasonably reflect the underlying borehole sampling data.

14.2.10 Mineral Resource Classification

Mineral resource classification is typically a subjective concept. Industry best practices suggest that resource classification should consider the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates, and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating these concepts to delineate regular areas at similar resource classification.

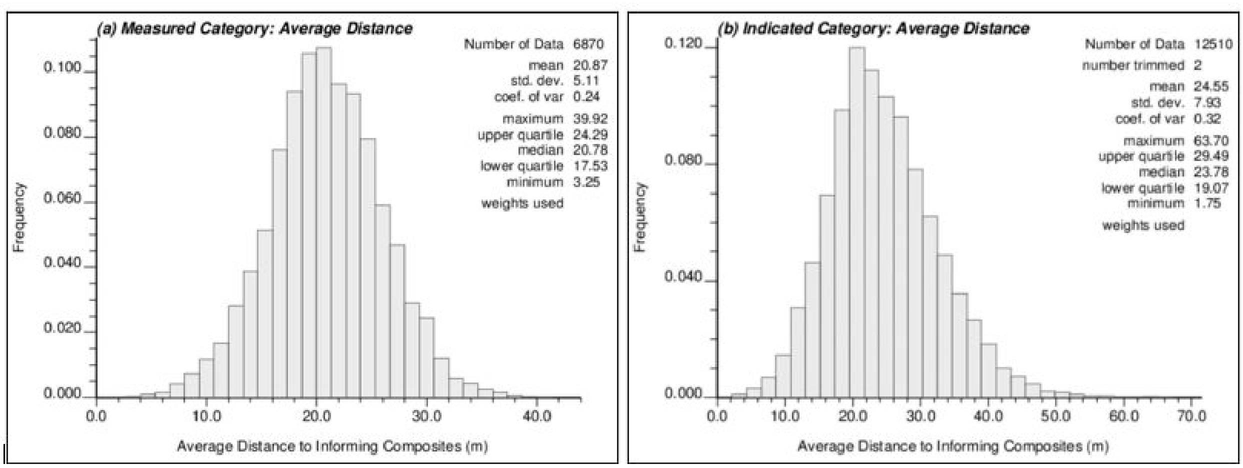
14.2.10.1 Rock

The mineral resource classification strategy considers borehole spacing, geologic confidence and continuity of category. The adopted classification parameters considered a geometric approach applied only to the mineralized zone, wherein:

- Measured blocks require three (3) boreholes located within 45 metres radii. This normally corresponds to a maximum drillhole spacing of 63 metres.
- Indicated blocks require two (2) boreholes located within 45 metres radii.
- Inferred – all other blocks estimated within the mineralized zone.

SRK validated the classification by applying a different block model created solely to assist with block classification using an estimation run that accounts for the geometric configuration of the available boreholes and found for the measured blocks an average mean distance of the nearest three (3) boreholes of 21 m (see Figure 14.16a) and for the indicated blocks an average mean distance to the nearest two (2) boreholes of 24 m (see Figure 14.16b). The distances found are within reported specifications. SRK considers the adopted mineral resource classification to be reasonable.

Figure 14.16: Average Distance to Nearest 3 Holes in (a) Measured Blocks and to Nearest 2 Holes in (b) Indicated Blocks



Source: SRK (2021)

14.2.10.2 Tailings

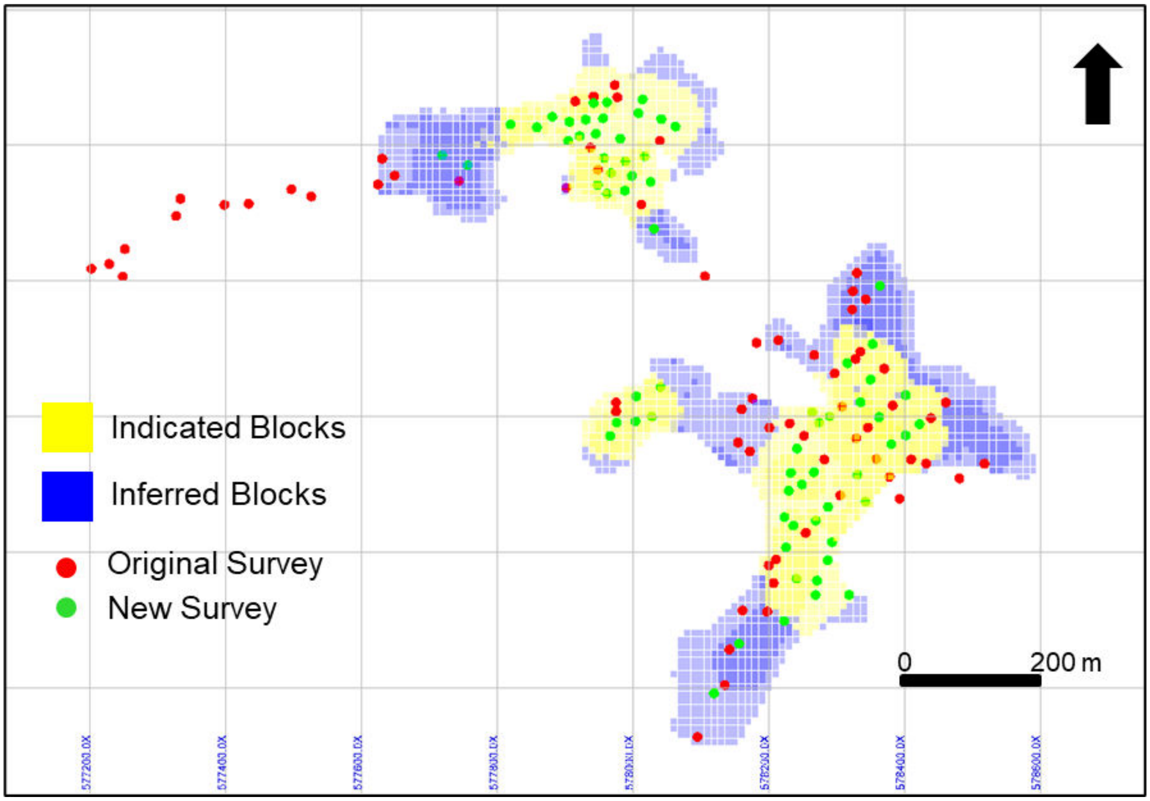
A geometric classification approach was adopted wherein:

- Indicated blocks were classified in the first estimation pass considering only the resurveyed boreholes.

- Inferred blocks were classified in the second estimation pass considering all boreholes.

Figure 14.17 shows a plan distribution of the classification for the tailings bock model.

Figure 14.17: Classification of Tailings Block Model



Source: SRK (2021)

14.2.10.3 SRK Comments

SRK examined the classification visually by inspecting sections and plans throughout the mineral resource models. SRK considers that the parameters used to define Measured blocks reasonably reflect estimates at a high confidence level, material classified as Indicated reflect estimates made with a moderate level of confidence within the meaning of *CIM Definition Standards for Mineral Resources and Mineral Reserves* (May 2014), and all other material is estimated at a lower confidence level.

14.3 Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) defines a mineral resource as:

> "*A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.*"

The "reasonable prospects for eventual economic extraction" requirement generally implies that quantity and grade estimates meet certain economic thresholds and that mineral resources are reported at an appropriate cut-off grade that takes into account extraction scenarios and processing recovery. SRK considers that the Tocantinzinho deposit is amenable for open pit extraction.

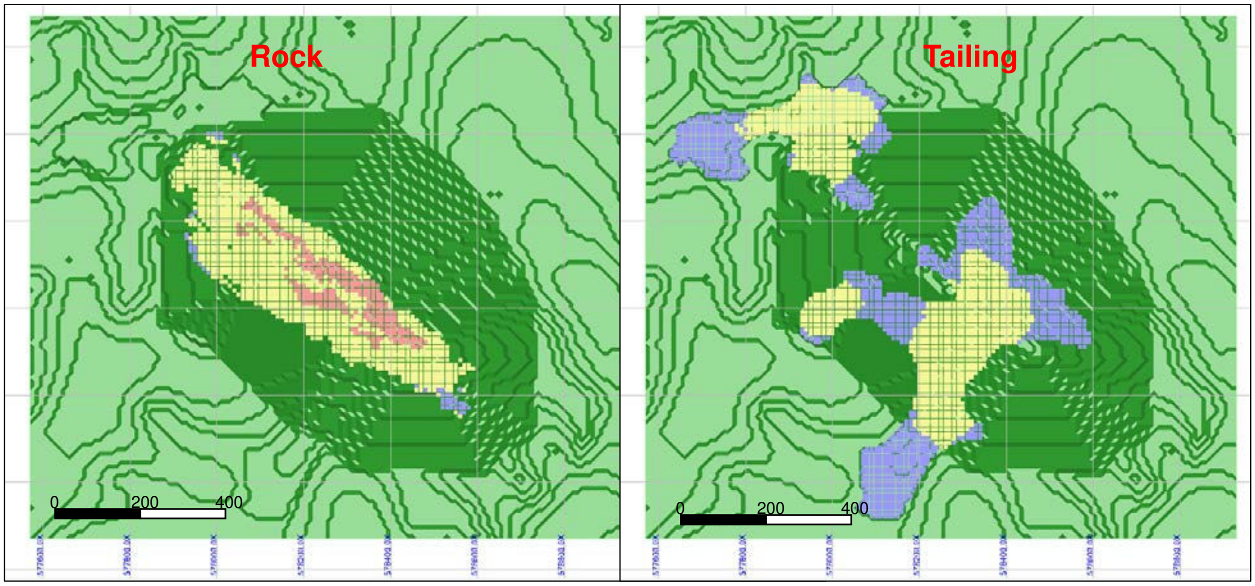
To assist with determining which portions of the gold deposit show reasonable prospect for economic extraction and to assist with selecting a reasonable reporting cut-off grade, a conceptual open pit shell using the assumptions described in Table 14.11 was developed and used to constrain mineral resource reporting.

Table 14.11: Resource Pit Shell Optimization Parameters

Economic Parameters	Unit	Value		
Exchange Rate	BRL/USD	5		
Discount Rate	%	6%		
Gold Price	USD/oz	1,600		
Transport & Refining Cost	USD/oz	15		
Royalty Rate	% NSR	3.00%		
Royalty Cost	USD/oz	47.55		
Net Gold Value	USD/oz	1,537		
Optimization Parameters		**Rock**	**Saprolite**	**Tailings**
Recovery & Dilution Factors				
Metallurgical Recovery	%	90.00%	78.00%	82.00%
Mining Dilution	%	0.00%	0.00%	0.00%
Ore Based Costs				
Power Cost	USD/t milled	2.21	1.02	1.02
Consumables (Reagents & Media)	USD/t milled	5.1	5.1	5.1
Plant Maintenance	USD/t milled	0.55	0.55	0.55
Plant Labour	USD/t milled	1.19	1.19	1.19
Sub-total Processing Costs	USD/t milled	9.05	7.86	7.86
Ore Feed Rehandle	USD/t milled	0	0	0
Incremental Ore Haulage	USD/t milled	0	0	0
General & Administration Costs	USD/t milled	3	3	3
Sustaining Capital & Closure	USD/t milled	1.3	1.3	1.3
Sub-total Other Ore Based Costs	USD/t milled	4.3	4.3	4.3
Total Ore Based Cost	**USD/t milled**	**13.35**	**12.16**	**12.16**
Cut-off Grade	**g Au/t**	**0.3**	**0.3**	**0.3**
Mining Costs				
Total Mining Reference Cost	USD/t mined	2.05	2.05	2.05
Incremental Bench Cost	USD/10 m bench	0.035	0.035	0.035

SRK considers that the blocks located within the conceptual pit envelope show "reasonable prospects for eventual economic extraction" and can be reported as a mineral resource (Figure 14.18).

Figure 14.18: Plan Showing Estimated Blocks in Rock (left) and Tailings (right) Within the Tocantinzinho Deposit Relative to the Conceptual Pit



Source: SRK (2021)

SRK is satisfied that the mineral resources were estimated in conformity with the generally accepted CIM *Estimation of Mineral Resource and Mineral Reserve Best Practices Guidelines* (November 2019). The mineral resources may be affected by further infill and exploration drilling that may result in increases or decreases in subsequent resource estimates. The mineral resources may also be affected by subsequent assessments of mining, environmental, processing, permitting, taxation, socio-economic, and other factors.

The Mineral Resource Statement for rock and tailings material tabulated in Table 14.12 was prepared by Camila Passos, P.Geo (APGO#2431). The overall process was reviewed by Dr. Oy Leuangthong, P.Eng (PEO#90563867) for rock material and by Glen Cole, P.Geo (APGO#1416) for tailings material. Ms. Passos is an independent Qualified Person as this term is defined in National Instrument 43-101. The effective date of the Mineral Resource Statement is December 10, 2021.

Table 14.12: Mineral Resource Statement*, Tocantinzinho Gold Project, Brazil, SRK Consulting (Canada) Inc., December 10, 2021

Domain	Classification	Cut-off Grade Au (g/t)	Quantity Tonnes (000's)	Grade Gold (g/t)	Contained Metal Gold Ounces (000's)
	Measured				
Saprolite		0.30	375	1.40	17
Rock		0.30	17,234	1.49	824
	Total Measured	0.30	17,609	1.49	841
	Indicated				
Saprolite		0.30	1,085	1.01	35
Rock		0.30	27,988	1.31	1,176
	Total Indicated	0.30	29,073	1.30	1,211
	M + I				
Saprolite		0.30	1,460	1.11	52
Rock		0.30	45,222	1.38	2,000
	Total M + I	**0.30**	**46,682**	**1.37**	**2,052**
	Inferred				
Saprolite		0.30	59	0.66	1
Rock		0.30	732	0.92	22
	Total Inferred	0.30	791	0.90	23
Tailings**	Measured	0.30	-	-	-
	Indicated	0.30	1,432	1.10	50
	Total M+I	**0.30**	**1,432**	**1.10**	**50**
	Inferred	0.30	789	1.07	27

** Mineral resources are not mineral reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimates. Assays were capped where appropriate. Open pit mineral resources are reported at a cut-off grade of 0.30 g/t gold. The cut-off grades are based on a gold price of USD 1,600 per troy ounce and metallurgical recoveries of 78% and 90% for gold in saprolite rock respectively.*
*** Mineral resources are not mineral reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimates. Assays were capped where appropriate. Open pit mineral resources are reported at a cut-off grade of 0.30 g/t gold. The cut-off grades are based on a gold price of USD 1,600 per troy ounce and metallurgical recoveries of 82% for gold.*

14.3.1 Grade Sensitivity Analysis

Tocantinzinho Project mineral resources are sensitive to the selection of the reporting cut-off grade. To illustrate this sensitivity, the global model quantities and grade estimates within the optimized pit are presented in Table 14.13 and Table 14.14 at different cut-off grades for rock and tailings material respectively. The reader is cautioned that the figures presented in these table should not be misconstrued with a Mineral Resource Statement. These figures are only presented to show the sensitivity of the mineral resource model estimates to the selection of cut-off grade. Figure 14.19 and Figure 14.20 presents this sensitivity as grade tonnage curves for rock and tailings material respectively.

Table 14.13: Global Block Model Quantities and Grade Estimates* for Rock at Various cut-off Grades Inside Pit, Tocantinzinho Gold Project, Brazil

Cut-off	Measured			Indicated			Measured+Indicated			Inferred		
Grade Au (gpt)	Quantity (000't)	Grade Au(gpt)	Metal Content (000'oz)	Quantity (000't)	Grade Au (gpt)	Metal Content (000'oz)	Quantity (000't)	Grade Au (gpt)	Metal Content (000'oz)	Quantity (000't)	Grade Au (gpt)	Metal Content (000'oz)
0.20	17,628	1.48	841	29,270	1.29	1,213	46,898	1.36	2,054	842	0.86	23
0.30	**17,610**	**1.49**	**841**	**29,073**	**1.30**	**1,212**	**46,682**	**1.37**	**2,052**	**791**	**0.90**	**23**
0.40	17,316	1.50	837	28,144	1.33	1,201	45,460	1.39	2,038	735	0.94	22
0.50	16,573	1.55	827	26,449	1.38	1,176	43,022	1.45	2,003	687	0.97	21
0.60	15,542	1.62	808	24,269	1.46	1,138	39,811	1.52	1,946	603	1.03	20
0.70	14,394	1.69	784	21,729	1.55	1,085	36,123	1.61	1,869	462	1.15	17
0.80	13,181	1.78	755	19,435	1.65	1,029	32,616	1.70	1,784	384	1.23	15
0.90	12,011	1.87	723	17,236	1.75	969	29,247	1.80	1,692	332	1.29	14
1.00	10,894	1.97	689	15,207	1.86	907	26,101	1.90	1,596	280	1.35	12
1.50	6,361	2.50	510	8,365	2.38	639	14,725	2.43	1,149	77	1.79	4

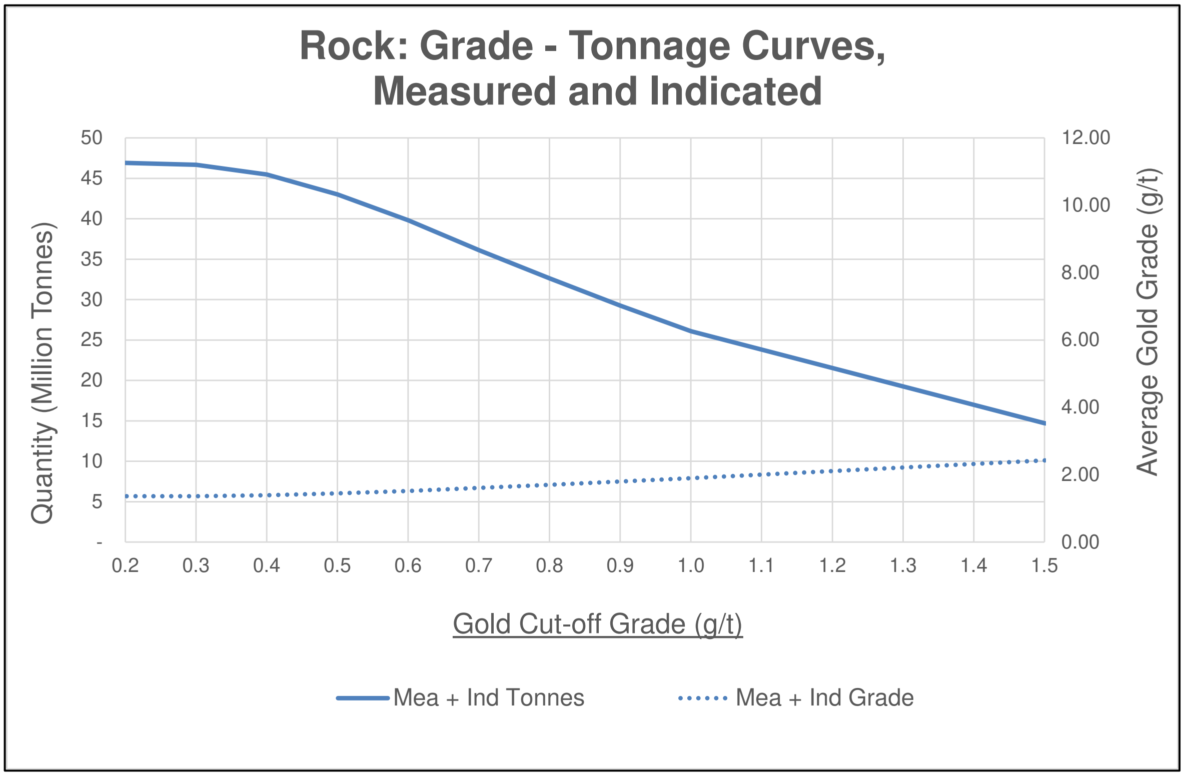
** The reader is cautioned that the figures in this table should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of a cut-off grade. All figures have been rounded to reflect the relative accuracy of the estimates.*

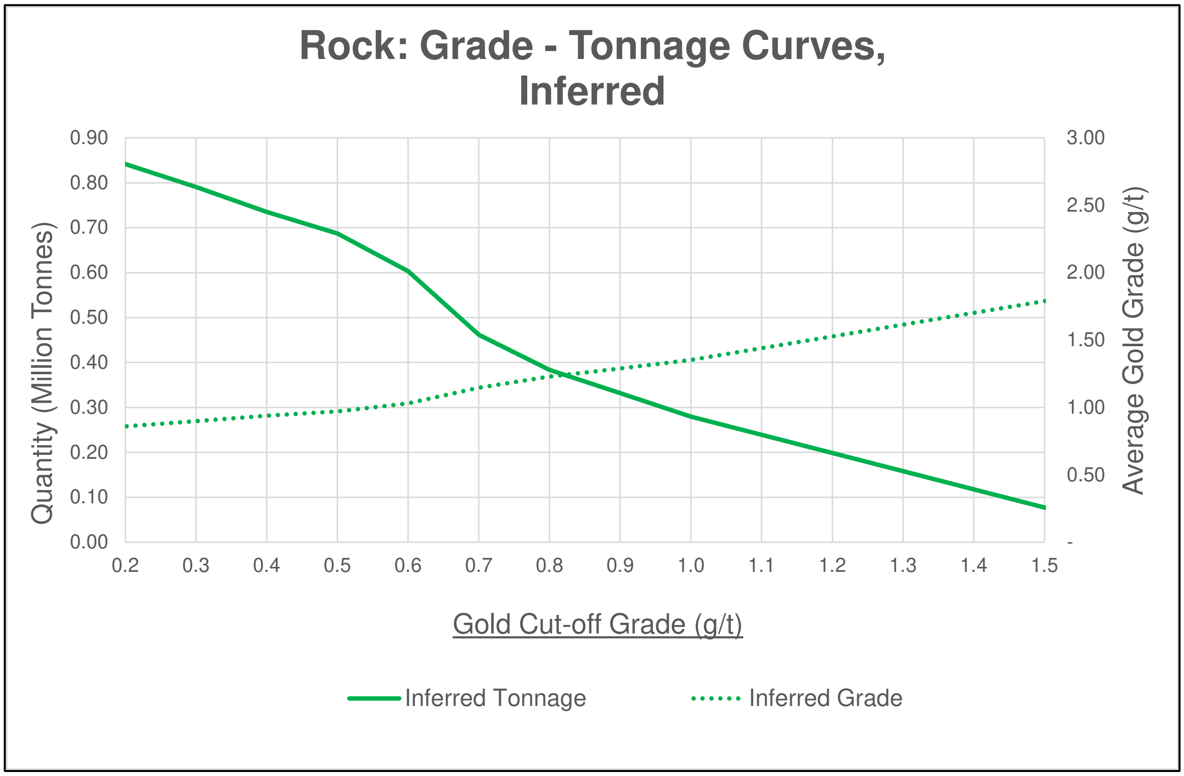
Table 14.14: Global Block Model Quantities and Grade Estimates* for Tailings at Various cut-off Grades Inside Pit, Tocantinzinho Gold Project, Brazil

Cut-off Grade Au (gpt)	Indicated			Inferred		
	Quantity (000't)	Grade Au(gpt)	Metal Content (000'oz)	Quantity (000't)	Grade Au (gpt)	Metal Content (000'oz)
0.10	1,432	1.10	50	847	1.02	28
0.20	1,432	1.10	50	847	1.02	28
0.30	**1,432**	**1.10**	**50**	**789**	**1.07**	**27**
0.40	1,429	1.10	50	773	1.09	27
0.50	1,421	1.10	50	762	1.09	27
0.60	1,411	1.11	50	737	1.11	26
0.70	1,359	1.12	49	710	1.13	26
0.80	1,274	1.15	47	652	1.16	24
0.90	1,125	1.19	43	568	1.21	22
1.00	904	1.24	36	461	1.27	19

** The reader is cautioned that the figures in this table should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of a cut-off grade. All figures have been rounded to reflect the relative accuracy of the estimates.*

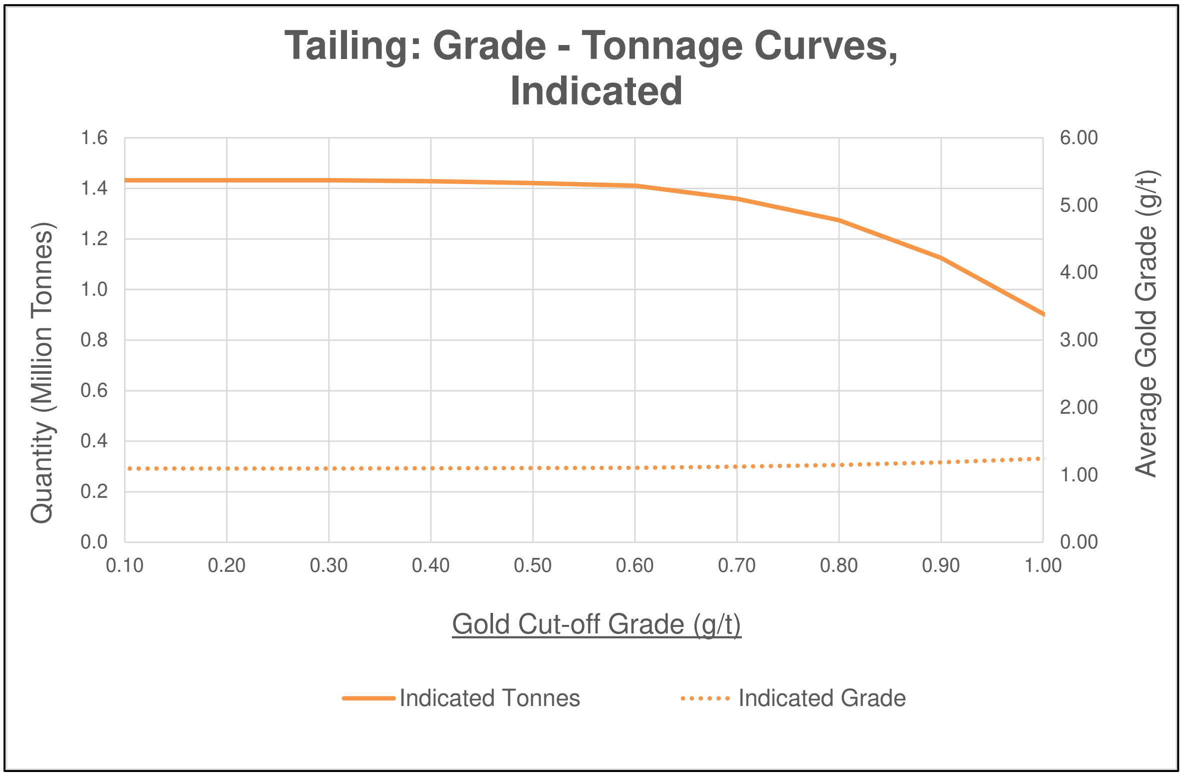
Figure 14.19: Global Grade Tonnage Curves for Rock (Measured + Indicated, above and Inferred below)

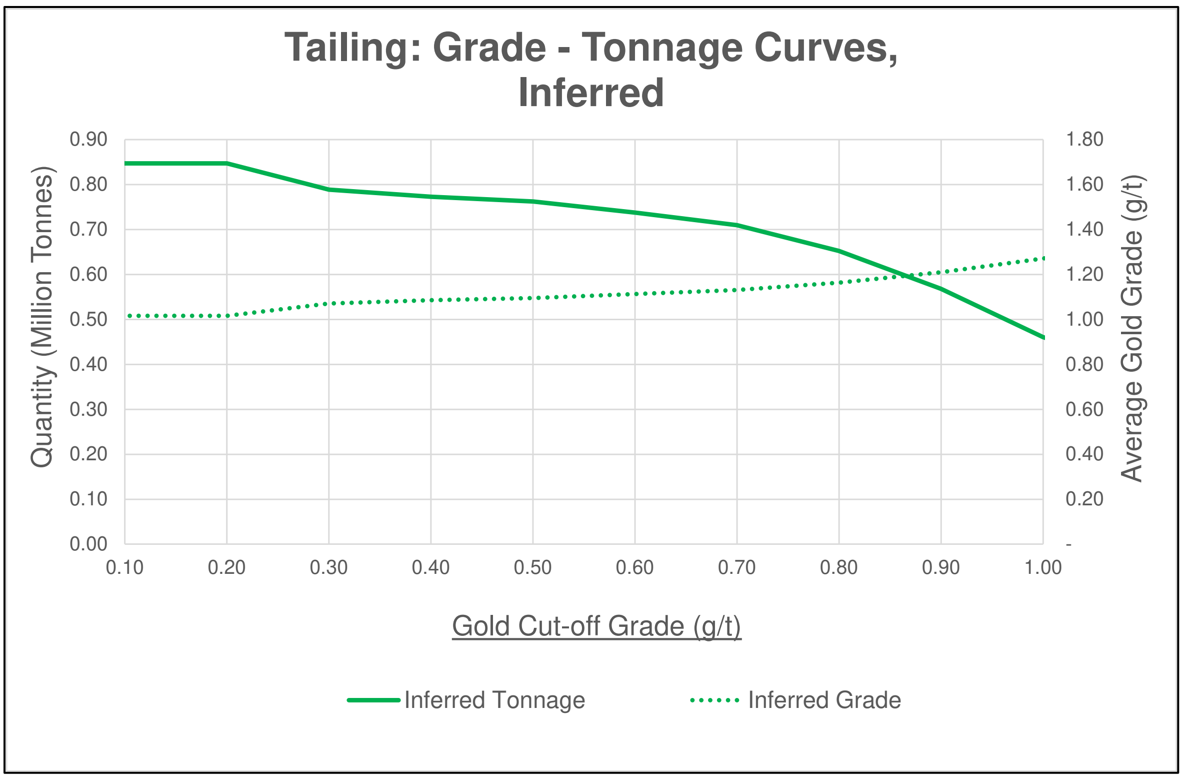




Source: SRK (2021)

Figure 14.20: Global Grade Tonnage Curves for Tailings (Indicated, above and Inferred below)





Source: SRK (2021)

14.3.2 Comparison with Previous Mineral Resource Estimates

A comparison between the Mineral Resource Statement prepared by Eldorado (June 21, 2019) and the December 10, 2021 Mineral Resource Statement prepared herewith by SRK is shown in Table 14.15 and Table 14.16. The reader is cautioned that the 2019 Mineral Resource Statement was reported at a 0.30 g/t cut-off grade and not constrained by a conceptual shell while the December 2021 Mineral Resource Statement is reported at the same cut-off but inside a conceptual shell. No tailings material was reported in the June 2019 Mineral Resource Statement.

Table 14.15: Comparison Between June 2019 and December 2021 Mineral Resource Statements for Rock Material

Classification	MRS - June 2019			MRS - December 2021			Differences		
	Quantity Tonnes (000's)	Grade Gold (g/t)	Contained Metal Gold Ounces (000's)	Quantity Tonnes (000's)	Grade Gold (g/t)	Contained Metal Gold Ounces (000's)	Quantity Tonnes (000's)	Grade Gold (g/t)	Contained Metal Gold Ounces (000's)
Measured	17,530	1.51	851	17,610	1.49	841	0%	-2%	-1%
Indicated	31,202	1.26	1,264	29,073	1.30	1,212	-7%	3%	-4%
Total M+I	**48,732**	**1.35**	**2,115**	**46,682**	**1.37**	**2,052**	**-4%**	**1%**	**-3%**
Inferred	2,395	0.90	69	791	0.90	23	-67%	0%	-67%

** 0.30 g Au/t cut-off grade - without a conceptual pit shell*
*** 0.30 g Au/t cut-off grade - inside a conceptual pit shell*

Table 14.16: Comparison Between June 2019 and December 2021 Mineral Resource Statements for Tailings Material

Classification	MRS - June 2019			MRS - December 2021			Differences		
	Quantity Tonnes (000's)	Grade Gold (g/t)	Contained Metal Gold Ounces (000's)	Quantity Tonnes (000's)	Grade Gold (g/t)	Contained Metal Gold Ounces (000's)	Quantity Tonnes (000's)	Grade Gold (g/t)	Contained Metal Gold Ounces (000's)
Measured	-	-	-	-	-	-	-	-	-
Indicated	-	-	-	1,432	1.10	50	-	-	-
Total M+I	**-**	**-**	**-**	**1,432**	**1.10**	**50**	**-**	**-**	**-**
Inferred	-	-	-	789	1.07	27	-	-	-

** 0.30 g Au/t cut-off grade - without a conceptual pit shell*
*** 0.30 g Au/t cut-off grade - inside a conceptual pit shell*

The differences between the two mineral resource statements are attributed to a number of factors. As mentioned above the main difference in the two statements is that the 2019 statement reflects quantities that are unconstrained, while the 2021 quantities are constrained within a mineral resource pit shell.

The overall impact of these differences is similar tonnage for Measured and Indicated Mineral Resources and reduced tonnage for Inferred Mineral Resources, with similar average grade contributing to an overall reduction in metal content.

15 MINERAL RESERVE ESTIMATES

15.1 Summary

The Proven and Probable Ore Reserve for the Tocantinzinho Project ("TZ Project") is estimated at 48.7 Mt at an average grade of 1.31 g Au/t for 2,042 koz of contained gold as summarized in Table 15.1. The contained gold in the proven category represents 41% of the total ore reserve estimate.

Table 15.1: Tocantinzinho Project Ore Reserve Estimate (December 10, 2021)

Mineral Reserve by Category	Tonnage kt	Grade g Au/t	Contained Gold koz
Proven	17,973	1.46	842
Probable	30,703	1.22	1,200
Proven and Probable	**48,676**	**1.31**	**2,042**

Notes:
1. *CIM definitions were followed for mineral reserves.*
2. *Effective date of the estimate is December 10, 2021.*
3. *Mineral reserves are estimated for a gold price of USD 1,400/oz.*
4. *Mineral reserve cut-off grade is 0.36 g Au/t for all materials.*
5. *A dilution skin width of 1m was considered resulting in an average mining dilution of 5.5%.*
6. *Bulk density of ore is variable with an average of 2.67 t/m^3.*
7. *The average strip ratio is 3.36:1.*
8. *Numbers may not add due to rounding.*

The open pit mine design and ore reserve estimate have been prepared by GMS to a level appropriate for a Feasibility Study. The mineral reserve stated herein is consistent with the CIM definitions and is suitable for public reporting. As such, the mineral reserves are based solely on measured and indicated mineral resources with applicable modifying factors and therefore exclude any inferred mineral resources. The inferred mineral resources contained within the mine design are classified as waste for reporting purposes.

15.2 Resource Block Model

The resource block model was completed by SRK. It was imported to the Deswik.CAD™ software as multiple block models, with artisanal miner tailings in one model and the rest of the ore body in the other. For mine planning purposes, GMS built a sub blocked model for the tailings and the contact between the models using a SMU block size of 1 m x 1 m x 1 m and the remainder of the orebody using a SMU block size of 10 m x 10 m x 10 m.

15.3 Pit Optimization

Open pit optimization was conducted in GEOVIA Whittle™ to determine the optimal economic shape of the open pit to guide the pit design process. This task was undertaken using the Whittle software which utilizes the Lerchs-Grossmann algorithm. The method works on a block model of the ore body, and progressively constructs 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 in the software. This section describes all the parameters used to calculate block values in Whittle™.

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 and excluded inferred blocks.

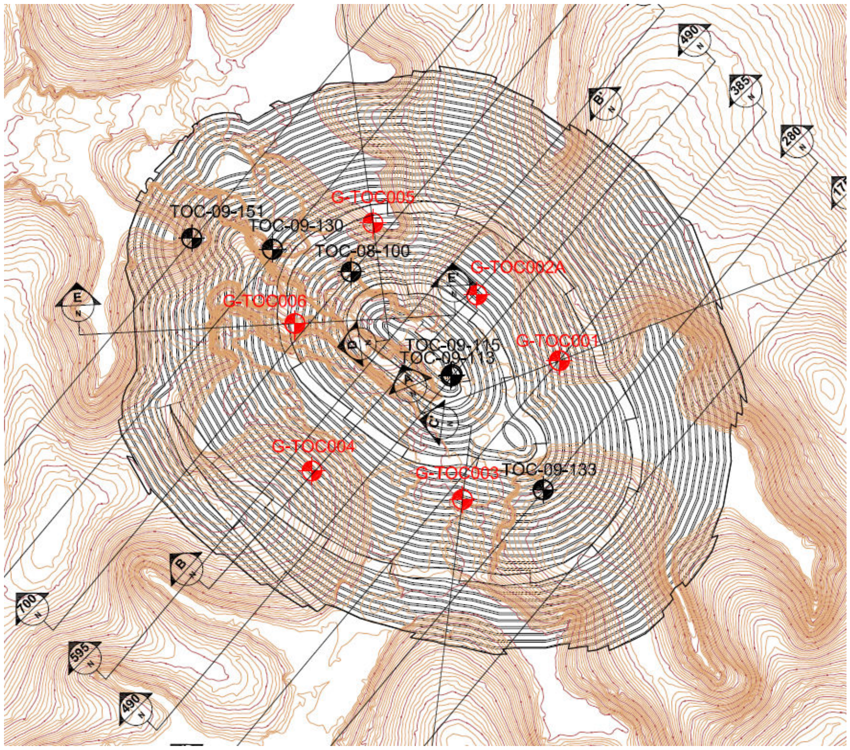
15.3.1 Pit Slope Geotechnical Assessment

In 2010, Golder Associates carried out a study of geomechanical modeling and slope design considering the estimated strength values of the materials that will make up the slopes of the final pit. Work carried out by Golder Associates is detailed in the following sections.

15.3.1.1 Data Collection

A program of geotechnical data collection and analysis was conducted to form the basis for slope design used in the open pit. This program was carried out by Golder and consisted of drilling and logging six oriented core holes (see Figure 15.1). The strike and dip of all fractures were recorded and the summarized in stereonet plots.

Figure 15.1: Locations of Oriented Geotechnical Drill Holes and Exploration Drill Holes



Source: Golder Associates, 2012

Besides the six oriented drill cores, an additional 11 exploration holes were logged for lithology, RQD and rock mass classification. In total, 17 core holes were logged, and intervals were assigned a geo-mechanical class designation of either Class II/I, Class III, Class IV and Class V. Generally, Class V corresponds to saprolite material, while Class II/I is fresh un-weathered rock. Classes III and IV represent a rock mass which is either slightly weathered or highly fractured. A series of geomechanical sections were created from the classification.

15.3.1.2 <u>Geotechnical Analysis</u>

Golder also analyzed the geotechnical data performing kinematic failure analysis and overall slope failure analysis.

Kinematic Analysis

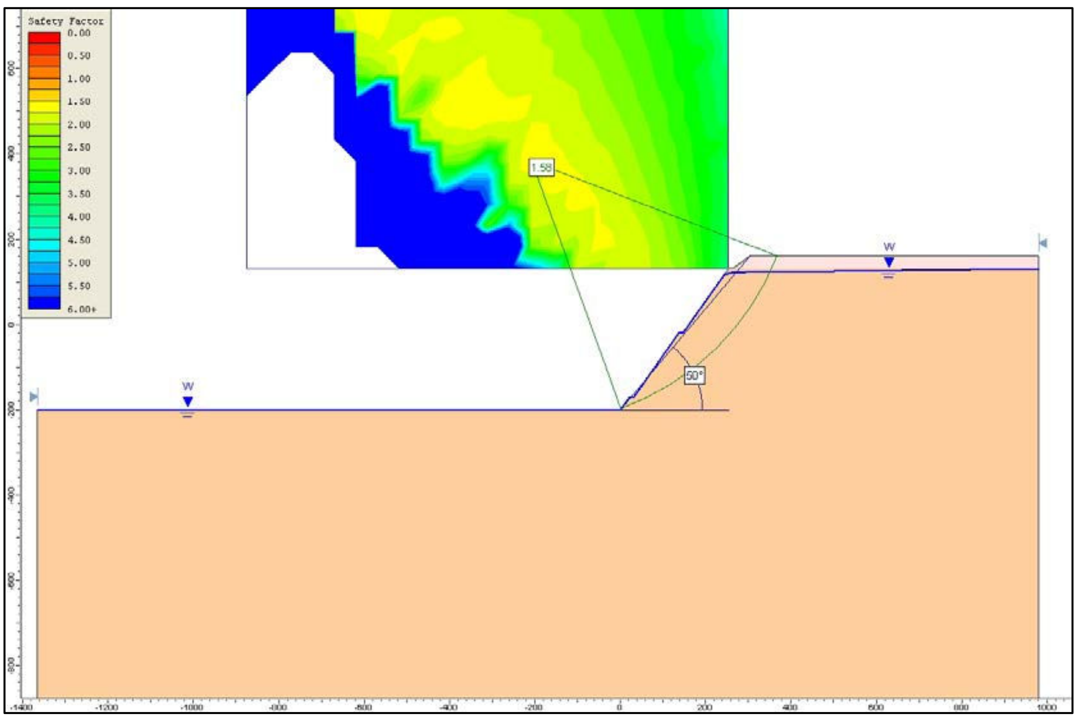
Kinematic failure analysis was used to determine the likely mode of failure on a given bench orientation. For each geotechnical hole and corresponding stereonet, Golder took the orientation of the pit wall and conducted a graphical analysis to determine which fracture planes are able to slip in a planar mode of failure and which fracture plane combinations are able to slip in a wedge plane type of failure. This analysis was used in recommending the slope design.

Slope Failure Analysis

An analysis of overall slope failure using an industry standard software product based on a limit equilibrium numerical method was conducted. The input parameters were estimated from Golder's experience with other projects.

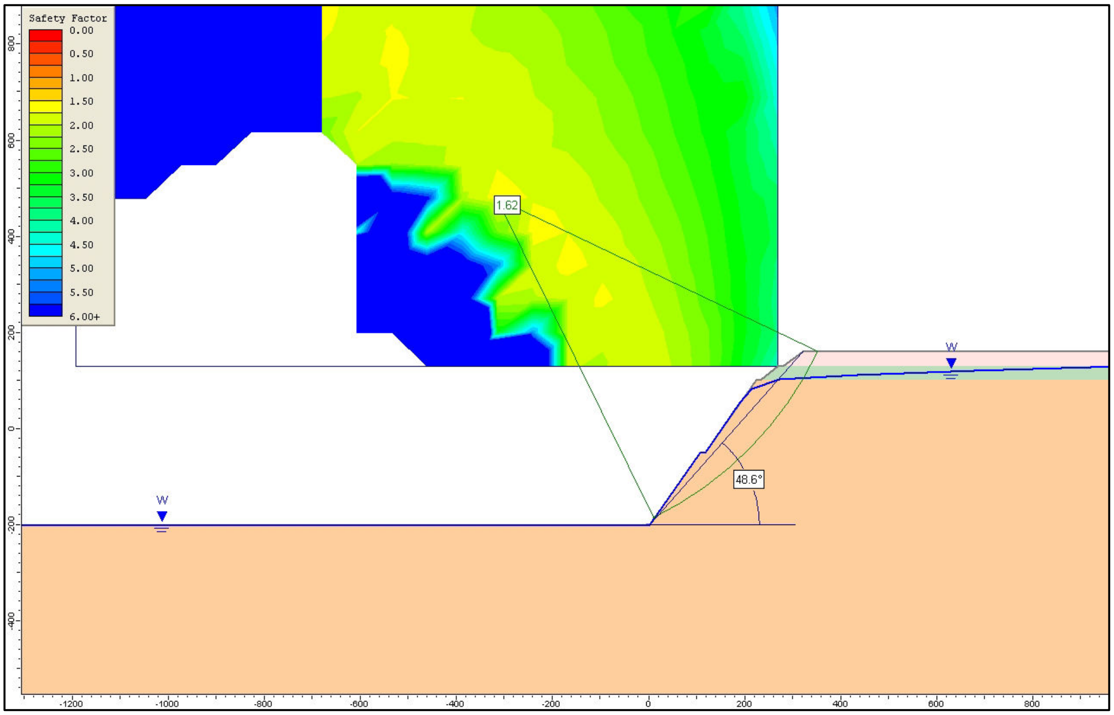
The results indicate that a 50-degree overall slope with a 360 m height will have a factor of safety of 1.58 while a 48.5 degree slope of the same height will have a factor of safety of 1.62 (Figure 15.2 and Figure 15.3 respectively) according to the *Morgenstern-Price* limit equilibrium method.

Figure 15.2: Results of Overall Slope Failure Analysis (FS=1.58)



Source: Golder Associates, 2012

Figure 15.3 Results of Overall Slope Failure Analysis (FS=1.62)

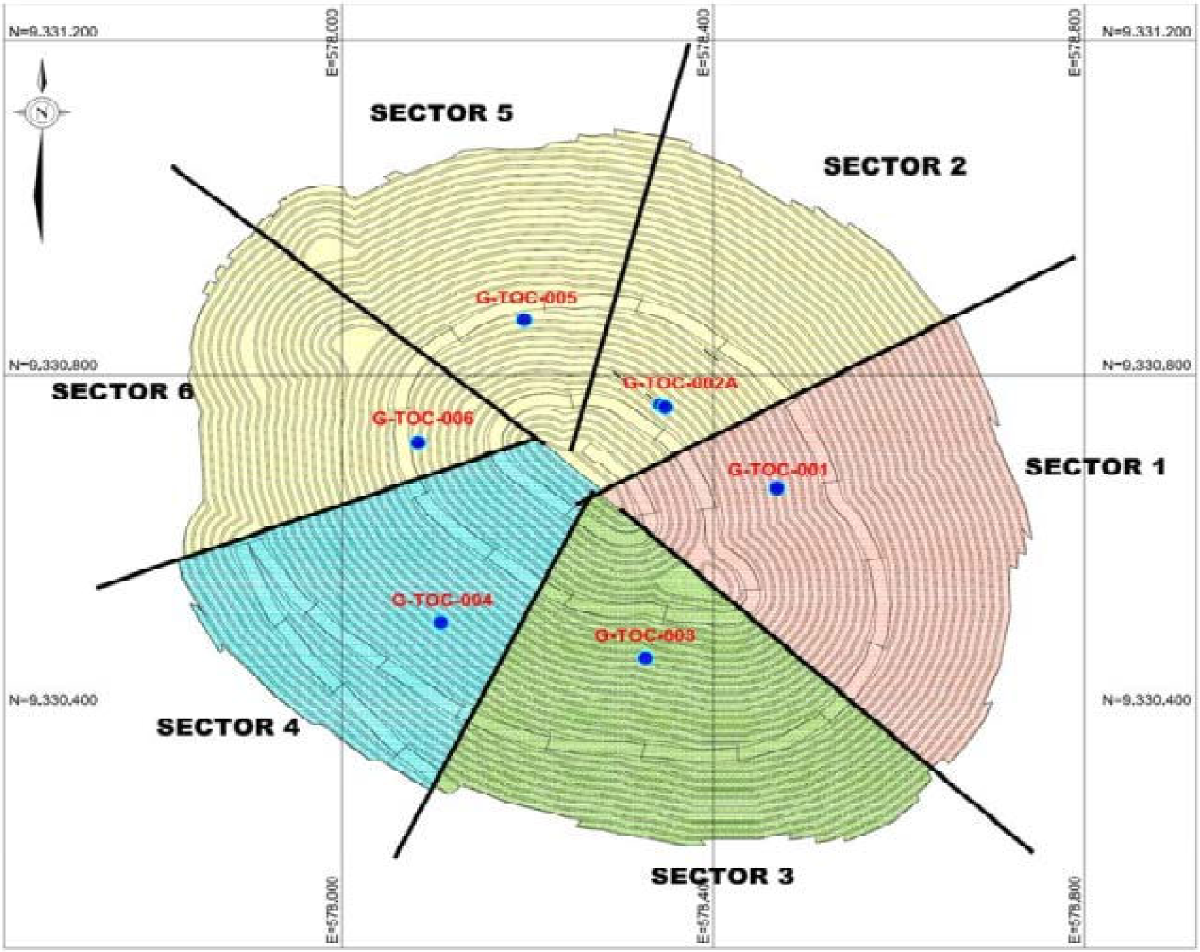


Source: Golder Associates, 2012

15.3.1.3 <u>Golder Conclusions and Slope Recommendations</u>

Golder concluded that overall slope failure would not be the most critical type of failure in the open pit. They also concluded that six geotechnical sectors were relevant with wedge failures being most likely in sector 4 and with planar failures being the most likely in sectors 1 and 3 (see Figure 15.4). Table 15.2 summarizes the slope design recommendations.

Figure 15.4: Golder Associates Slope Design Sectors



Source: Golder Associates, 2012

Table 15.2: Golder Associates Detailed Slope Design Parameters

Rock Type	Rock	Rock	Rock	Rock	Saprolite
Sector	**1**	**2, 5, 6**	**4**	**3**	**All**
Final Vertical Bench Height	20.0	20.0	20.0	20.0	10.0
Bench Face Angle	70.0	65.0	75.0	65.0	55.0
Avg. Catch Berm Width	8.00	8.00	8.50	8.00	6.50
Inter-Ramp (Crest-To-Crest)	52.6	49.1	55.3	49.1	36.5

Maximum interramp height of 140 m (7 benches). One 10.5 m wide berm each 7 benches if needed.

15.3.1.4 Pit Optimization Overall Slope Angles

From the Golder recommendations, overall slope angles were determined considering the number of ramp segments in walls. The overall slope angles utilized for pit optimization are presented in Table 15.3.

Table 15.3: Overall Slope Angles

Sector	Overall Slope Angle (Deg) Inclusive of Ramps
Sector 1	50.0
Sector 2	46.0
Sector 3	46.0
Sector 4	52.0
Sector 5	46.0
Sector 6	46.0
Saprolite	36.5

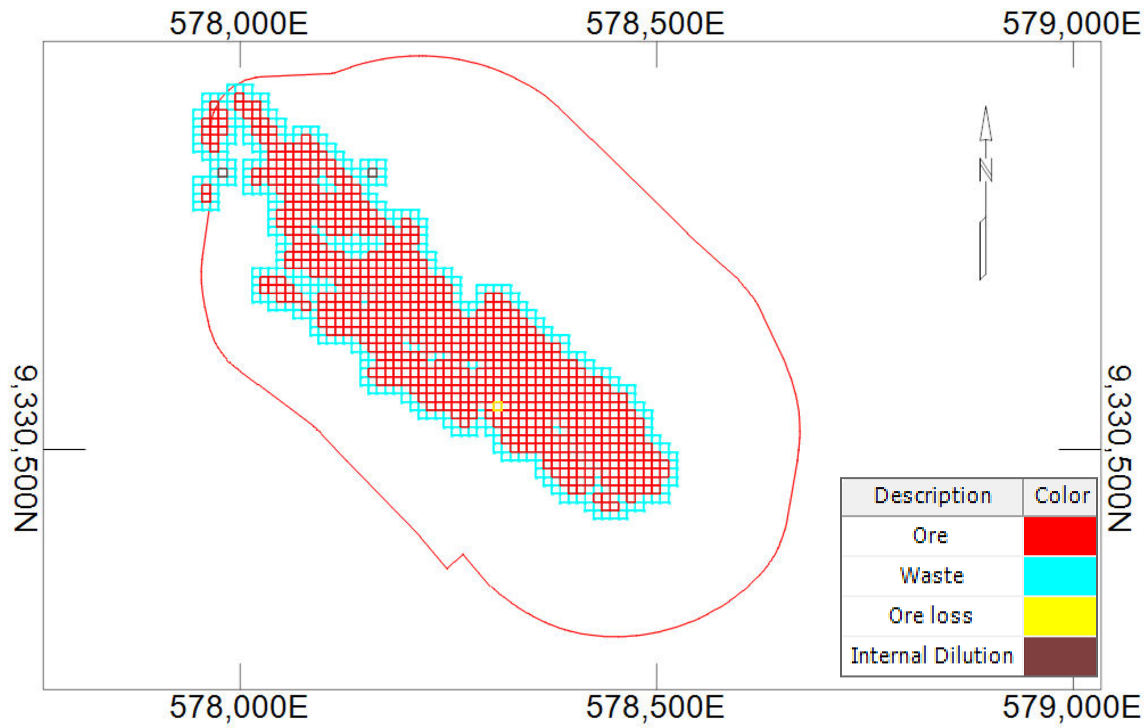
15.4 Mining Dilution and Ore Loss

A mining dilution assessment was made by evaluating the number of contacts for blocks above an economic cut-off grade (“CoG”). The block contacts are then used to estimate a dilution skin around ore blocks to estimate an expected dilution during mining. The dilution skin consists of 1.0 m of material in a north-south direction and 1.0 m in an east-west direction.

For each mineralized block in the resource model diluted grades and a new density are calculated by considering the in-situ grades and in-situ density of the surrounding blocks.

Although dilution skin thickness is conservative, the average external mining dilution result is low at 5.5%. This is explained by the fact that the ore body is generally massive in nature and continuous in the middle of the pit resulting in minimal ore-waste contacts (Figure 15.5).

Figure 15.5: Ore Body (RL 40)



Source: GMS, 2021

15.5 Pit Optimization Parameters and Cut-Off Grade

A summary of the open pit optimization parameters is presented in Table 15.4 and Table 15.5. The parameters used for optimization were updated from previous work done on the TZ Project as well as benchmarking on similar projects. A long-term metal price assumption of USD 1,400/oz was used. The mining reference cost (i.e., for a block near surface) is USD 2.05/t with an incremental cost of USD 0.035/t per 10 m bench added to account for the additional haulage cycle time.

The total ore-based cost varies from USD 12.16/t to USD 13.35/t depending on the ore type fed to the plant. The variation in cost is mainly due to the variation in power consumption at the grinding stage. Power cost varies form USD 1.02/t to USD 2.21/t. The ore-based cost is based on a nominal throughput of 4.34 Mtpy.

The estimated CoG varies by material type. For simplicity, a single CoG of 0.36 g/t was set for the project. CoG information is detailed in Table 15.5. The highest CoG is with saprolite material and is explained by the fact that although the cost is lower, the recovery is also low.

Table 15.4: Economic Optimization Parameters

Economic Parameters		
Discount rate	%	6%
Gold price	USD/oz	1,400
Transport & refining cost	USD/oz	15.00
Private Royalty Rate	% NSR	1.5%
Brazilian Govt. Mining Royalty (CFEM)	% GOR	1.5%

CFEM = Financial Compensation for Mineral Resources Exploration – CFEM

Table 15.5: Economic Optimization Parameters by Rock Type

Optimization Parameters		Rock	Saprolite	Tailings
Recovery & Dilution Factors				
Metallurgical Recovery	%	90.0%	78.0%	82.0%
Mining Dilution	%	5.0%	5.0%	5.0%
Ore Based Costs				
Power Cost	USD/t milled	2.21	1.02	1.02
Consumables (Reagents & Media)	USD/t milled	5.10	5.10	5.10
Plant Maintenance	USD/t milled	0.55	0.55	0.55
Plant Labour	USD/t milled	1.19	1.19	1.19
Sub-Total Processing Costs	USD/t milled	9.05	7.86	7.86
General & Administration Costs	USD/t milled	3.00	3.00	3.00
Sustaining Capital & Closure	USD/t milled	1.30	1.30	1.30
Sub-Total Other Ore Based Costs	USD/t milled	4.30	4.30	4.30
Total Ore Based Cost	USD/t milled	13.35	12.16	12.16
Cut-Off Grade	g Au/t	**0.36**	**0.38**	**0.36**
Mining Costs				
Total Mining Reference Cost	USD/t mined	2.05	2.05	2.05
Incremental Bench Cost	USD/10 m bench	0.035	0.035	0.035

15.6 Pit Optimization Results

The Whittle nested shell results are presented in Table 15.6 using only the measured and indicated mineral resource, applying a 5% dilution within Whittle. The nested shells are generated using revenue factors to scale up and down from the base case the selling price.

Table 15.6: M&I Whittle Shell Results @ USD 1,400/oz

Pit Shell	Best Case Disc. @ 6% (M$)	Specified Disc. @ 6% (M$)	Worst Case Disc. @ 6% (M$)	Total Tonnage (kt)	Ore Tonnage (kt)	Au Grade (g/t)	In-Situ Gold (koz)	Strip Ratio (W:O)
2	180	181	180	8,824	4,331	1.63	227	1.0
3	278	277	276	15,125	7,138	1.56	359	1.1
4	542	538	532	36,390	16,533	1.46	774	1.2
5	646	639	630	47,262	20,304	1.46	951	1.3
6	696	686	676	53,380	22,288	1.46	1,043	1.4
7	746	733	722	60,108	24,360	1.45	1,137	1.5
8	760	746	735	62,202	25,046	1.45	1,166	1.5
9	783	768	755	65,776	26,256	1.44	1,216	1.5
10	798	782	769	68,448	27,055	1.43	1,248	1.5
11	806	789	775	70,147	27,403	1.43	1,264	1.6
12	868	843	822	83,556	31,988	1.38	1,421	1.6
13	903	873	848	93,589	34,317	1.37	1,514	1.7
14	904	874	849	93,823	34,435	1.37	1,518	1.7
15	949	915	887	109,615	37,417	1.37	1,646	1.9
16	963	927	897	115,255	38,464	1.37	1,689	2.0
17	973	936	903	119,414	39,438	1.36	1,724	2.0
18	975	938	904	120,176	39,592	1.36	1,730	2.0
19	990	950	914	127,681	40,891	1.35	1,781	2.1
20	995	955	916	131,069	41,444	1.35	1,802	2.2
21	1,007	963	918	138,855	42,696	1.35	1,850	2.3

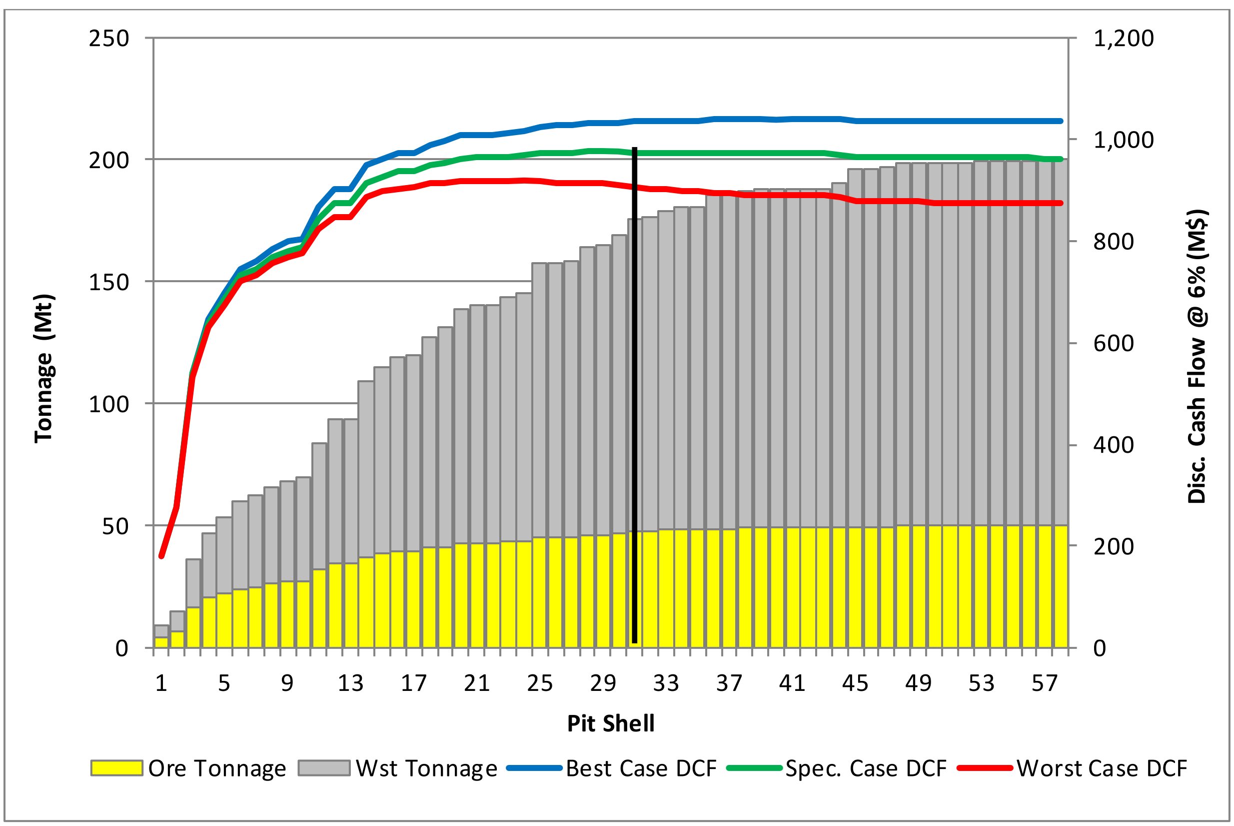
Pit Shell	Best Case Disc. @ 6% (M$)	Specified Disc. @ 6% (M$)	Worst Case Disc. @ 6% (M$)	Total Tonnage (kt)	Ore Tonnage (kt)	Au Grade (g/t)	In-Situ Gold (koz)	Strip Ratio (W:O)
22	1,009	965	919	140,364	42,891	1.35	1,859	2.3
23	1,009	965	919	140,525	42,933	1.35	1,860	2.3
24	1,013	967	919	143,848	43,422	1.35	1,878	2.3
25	1,015	969	919	145,656	43,708	1.34	1,888	2.3
26	1,026	975	916	157,520	45,273	1.34	1,946	2.5
27	1,026	975	916	157,957	45,405	1.34	1,949	2.5
28	1,027	975	916	158,420	45,491	1.33	1,952	2.5
29	1,031	976	912	164,459	46,195	1.33	1,978	2.6
30	1,031	976	912	164,750	46,254	1.33	1,980	2.6
31	1,033	976	910	168,905	46,942	1.33	2,000	2.6
32	1,036	975	905	175,896	47,737	1.32	2,028	2.7
33	1,037	975	903	176,919	47,940	1.32	2,033	2.7
34	1,038	974	902	179,318	48,204	1.32	2,042	2.7
35	1,038	974	900	180,306	48,322	1.32	2,046	2.7
36	1,038	974	900	180,571	48,364	1.32	2,047	2.7
37	1,039	972	893	185,803	48,917	1.31	2,065	2.8
38	1,039	972	893	186,111	48,956	1.31	2,066	2.8
39	1,039	972	891	187,573	49,091	1.31	2,071	2.8
40	1,039	972	891	187,799	49,155	1.31	2,072	2.8
41	1,039	972	891	187,821	49,161	1.31	2,072	2.8

The shell selection is presented in Table 15.7 and Figure 15.6. Pit shell 31 was selected as the optimum final pit shell which corresponds to a USD 1,150/oz pit shell (i.e., revenue factor 0.82). This shell has a total tonnage of 168.9 Mt including 46.9 Mt of ore. This is the smallest shell that achieves close to maximum value using a practical phasing approach.

Table 15.7: M&I Pit Shell Selection

Shell Selection	Best	Spec.	Worst	Selection
Shell Number	41	31	25	31
Shell RF	1.00	0.82	0.71	0.82
Shell Price	1400	1150	1000	1150
Total Tonnage (kt)	187,821	168,905	145,656	168,905
Waste Tonnage (kt)	138,660	121,963	101,948	121,963
Strip Ratio (W:O)	2.8	2.6	2.3	2.6
Ore Tonnage (kt)	49,161	46,942	43,708	46,942
Grade (g Au/t)	1.31	1.33	1.34	1.33
In-situ Gold (koz)	2,072	2,000	1,888	2,000

Figure 15.6: M&I Pit by Pit Graph @ USD 1,400/oz



Souurce: GMS, 2021

15.7 Mine Design

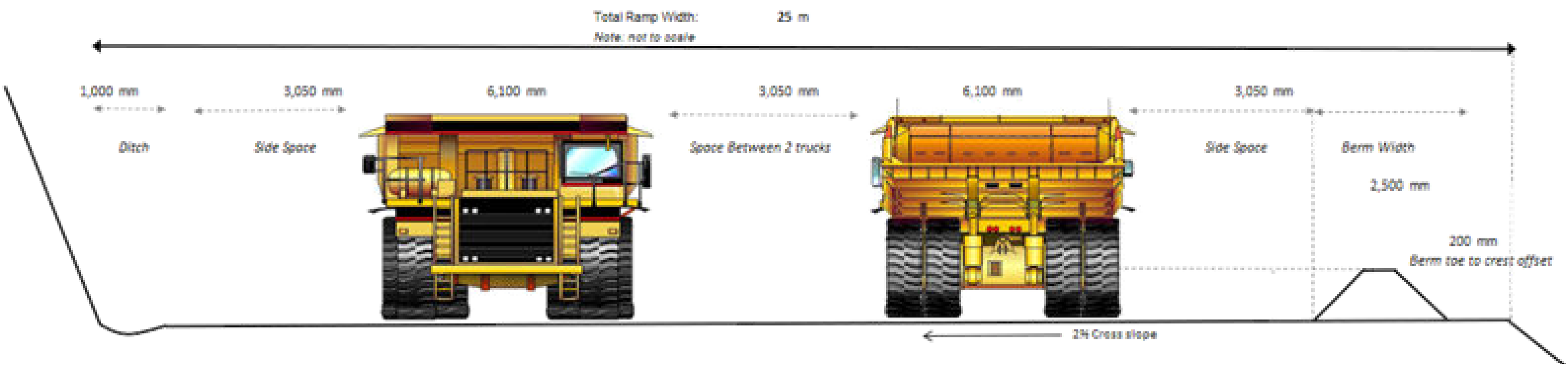
15.7.1 Ramp Design Criteria

The ramps and haul roads are designed for the largest equipment being a CAT 777E haul truck with a canopy width of 6.2 m. For double lane traffic, industry best-practice recommends designing a travelling surface of at least three times the width of the largest vehicle. Ramp gradients are established at 10%.

A shoulder barrier or safety berm on the outside edge will be constructed of 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 1.35 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 assure proper drainage of the running surface. The ditch will be 1.0 m wide. To facilitate drainage of the roadway, a 2% cross slope on the ramp is planned.

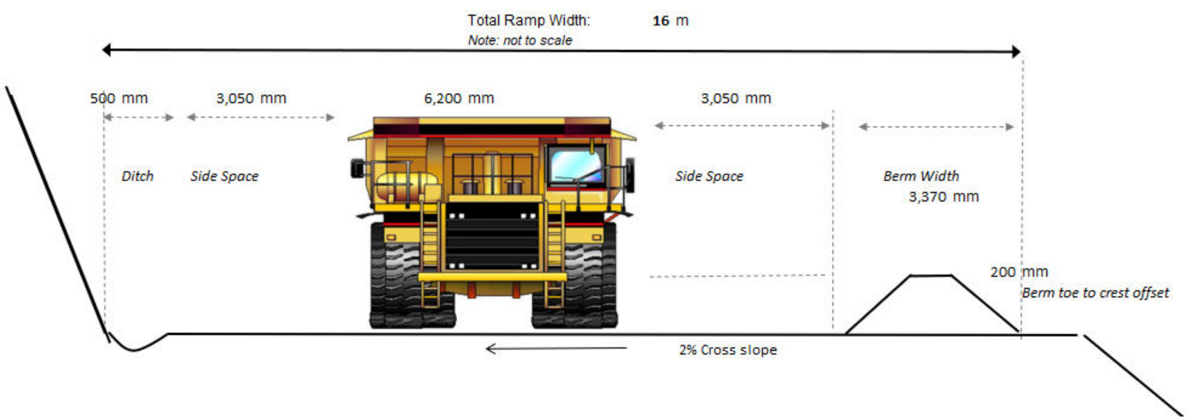
The double lane ramp width is 25 m wide, and the single lane ramp is 16 m wide. Single-lane ramps are introduced in the pit bottom when the benches start narrowing and when the mining rates will be significantly reduced. Double and single-lane ramp configurations are shown in Figure 15.7 and Figure 15.8.

Figure 15.7: Double-Lane Ramp Design Criteria



Source: GMS, 2021

Figure 15.8: Single-Lane Ramp Design Criteria

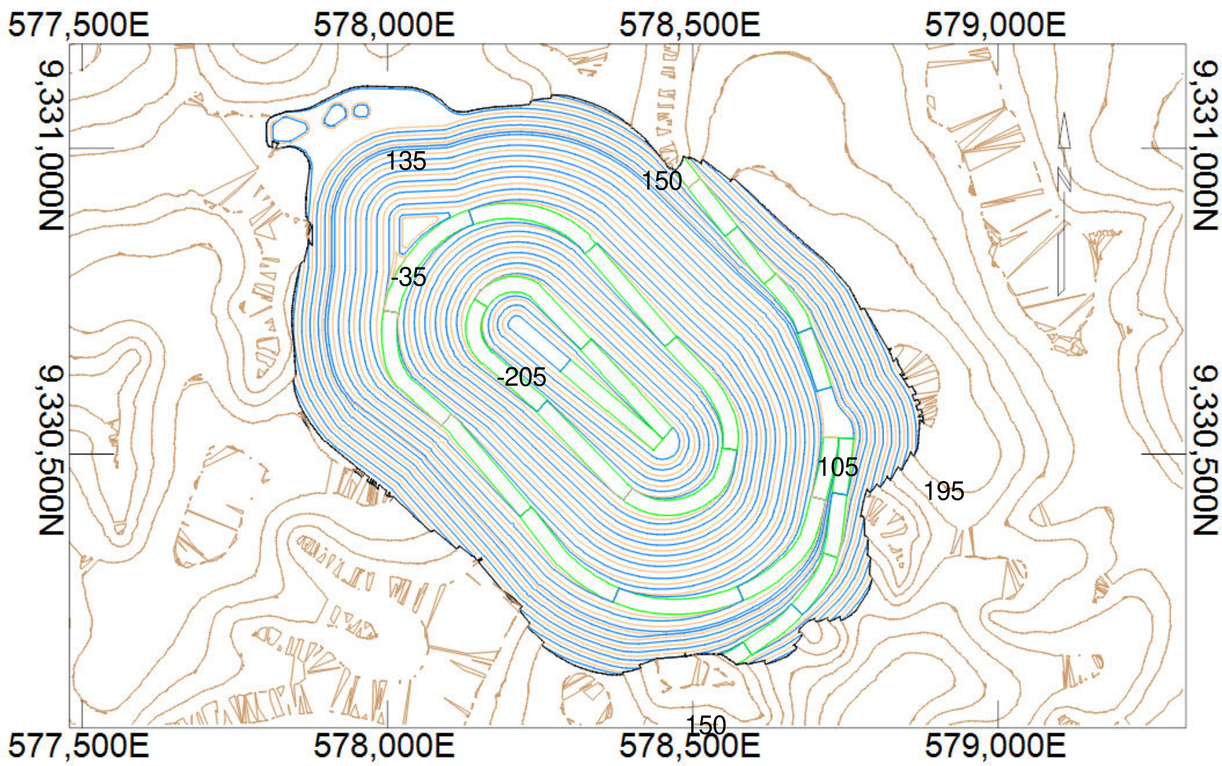


Source: GMS 2021

15.7.2 Open Pit Mine Design Result

The Tocantinzinho deposit is mined as a single pit as presented in Figure 15.9. The pit has a roughly ellipsoidal shape with a NW-SE orientation. It is approximately 1,250 m long by 800 m wide and reaches an average depth of 350 m. The final pit design has one exit to the north to facilitate the access to the crusher and the primary waste dump and another to the south to reach the TSF. The ramp system avoids switchbacks and limits the stack height; due to this, only one 10.5 m wide geotechnical berm was required in the north-east of the pit at elevation 15. The final design ramp is double lane to the very bottom of the pit.

Figure 15.9: Final Pit Design Plan View



Source: GMS, 2021

15.8 Mineral Reserves Statement

The ore reserve and stripping estimates are based on the final pit design presented in Section 15.7. The Proven and Probable ore reserves are inclusive of mining dilution and ore loss. The total ore tonnage before external mining dilution and ore loss is estimated at 46.2 Mt at an average grade of 1.37 g Au/t.

The external mining dilution around the ore blocks results in a dilution tonnage of 2.6 Mt @ 0.11 g/t. The dilution tonnage represents 5.5% of the ore tonnage before dilution and the dilution grade is estimated from the block model and corresponds to the average grade of the dilution skin. Table 15.8 presents a mineral resource to ore reserve reconciliation.

Table 15.8 Mineral Resource to Ore Reserve Reconciliation

Resource to Reserve Reconciliation	Tonnage (kt)	Gold Grade (g/t)
Ore Before Dilution	46,219	1.37
Less: Ore Loss (Isolated Blocks)	94	0.79
Ore Before Mining Dilution	46,125	1.37
Add: Mining Dilution	2,551	0.11
Proven & Probable Mineral Reserve	48,676	1.31

The Proven mineral reserves total is 18.0 Mt at an average grade of 1.46 g Au/t. The Probable mineral reserves total is 30.7 Mt at an average grade of 1.22 g Au/t. The total Proven and Probable mineral reserve is 48.7 Mt an average gold grade of 1.31 g/t for 2,042 koz contained gold. The total tonnage to be mined is estimated at 212.1 Mt for an average strip ratio of 3.36 (Table 15.9).

The saprolite and artisanal miner tailings represent only 4.9% of the ore reserve contained gold (or 5.9% of tonnage) with the granite fresh rock being the main material type at 95.1% of contained gold (or 94.1% of tonnage).

Table 15.9: Tocantinzinho Project Ore Reserve Estimate by Material Type (December 10, 2021)

Material	Category	Tonnage kt	Grade g Au/t	Contained Gold (koz)
Tailings	Proven	0	0.00	0
	Probable	1,308	1.11	47
	Total P&P	**1,308**	**1.11**	**47**
Saprolite	Proven	399	1.35	17
	Probable	1,182	0.93	35
	Total P&P	**1,581**	**1.03**	**53**
Rock	Proven	17,574	1.46	825
	Probable	28,213	1.23	1,118
	Total P&P	**45,787**	**1.32**	**1,943**
Total	Proven	17,973	1.46	842
	Probable	30,703	1.22	1,200
	Total P&P	**48,676**	**1.31**	**2,042**

Notes:
1. *CIM definitions were followed for mineral reserves.*
2. *Effective date of the estimate is December 10, 2021.*
3. *Mineral reserves are estimated for a gold price of USD 1,400/oz.*
4. *Mineral reserve cut-off of grade of 0.36 g/t.*
5. *A dilution skin width of 1 m was considered resulting in an average mining dilution of 5.5%.*
6. *Bulk density of ore is variable with an average of 2.67 t/m^3.*
7. *The average strip ratio is 3.36:1.*
8. *Numbers may not add due to rounding.*

16 MINING METHODS

16.1 Introduction

Mining of the Tocantinzinho deposit is planned as a conventional open pit operation using 17 m^3 hydraulic excavators and 92 t class haul trucks. A bulk mining approach is well suited for the massive ore body with mining to take place on 10 m-high benches. The mine is planned as an owner mining operation with blasting activities to be outsourced.

The mine consists of a single open pit that will be developed in four (4) phases which allows for deferral of waste stripping over the mine life and maximizing mill feed grade during the earlier years with an objective of optimizing the production schedule and resulting economics. Pre-production mining will take place over a period of two years with a total of 17.1 Mt mined which provides for waste fill material for construction purposes and exposes higher grade ore prior to commercial production.

The mine plan was developed to feed the process plant at a rate of 4.7 Mtpy. To achieve this, a peak mining rate of 28.3 Mt was determined which allows for stockpiling of lower grade material for processing at the end of the mine life. Once the open pit is depleted, processing activities will continue for another 18 months from reclaiming low grade stockpile.

Over the mine life, the Project will produce 48.7 Mt of ore and 163.4 Mt of waste for an overall stripping ratio of 3.36 (W:O). The mine will produce 1,838 koz of gold based on an average process recovery of 90%. The average gold production during the 10.5 years of commercial production is 175 koz/y.

16.2 Open Pit Designs

16.2.1 Mine Design Parameters

The open pit mine designs were guided using optimal Whittle shells and pit slope and ramp design criteria outlined in Section 15.

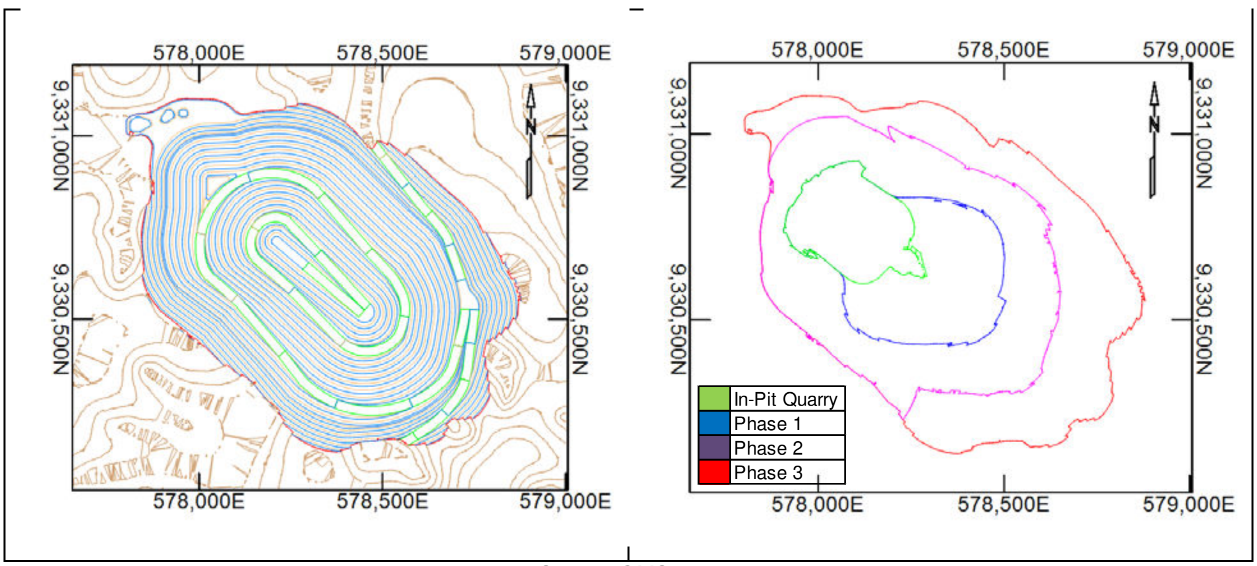
Mining of the Tocantinzinho open pit is planned with four phases within one nested pit with some common pit walls. The mining physicals of each of the mining phases are summarized in Table 16.1 and final configuration of the pit is presented in Figure 16.1. The internal phases result in a lower stripping ratio then the subsequent phases.

Table 16.1: Mining Physicals Summary by Phase

Summary by Mining Phase		Total	Phase 0	Phase 1	Phase 2	Phase 3
Total Tonnage	kt	**212,067**	5,273	16,220	84,166	106,407
Waste Tonnage	kt	**163,391**	3,576	9,135	60,788	89,891
Rock Tonnage	kt	**133,185**	2,021	5,237	47,513	78,415
Saprolite Tonnage	kt	**29,715**	1,474	3,644	13,122	11,475
Tailings Tonnage	kt	**491**	81	254	153	2
Strip Ratio	W:O	**3.36**	2.11	1.29	2.60	5.44
Ore Tonnage	kt	**48,676**	1,697	7,085	23,378	16,516
Gold Grade	g Au/t	**1.31**	1.04	1.41	1.30	1.30
Contained Gold	k oz	**2,042**	55	320	979	688

A mining width of 80 m was maintained between pushbacks to allow for efficient and safe operations. This cutback width allows for loading operations, circulation on the bench and drilling activities. This targeted mining width is in excess of the 40 m typical minimum width for this size of equipment.

Figure 16.1: End of LOM Pit Layout and Phase Limits



Source: GMS, 2021

16.2.2 Pit Phases

The final pit design is presented in Figure 16.1 along with the surface expression of the four mining phases. Ramps for the pits are designed to exit to the north for better access to the primary dump and the primary crusher / ROM pad. Phase 3 has an additional exit to the south for building of the tailings storage facility ("TSF") dam and the associated downstream rock dump.

The Phase 0 mine design is presented in Figure 16.2 and is essentially an in-pit quarry designed specifically to produce construction materials required for platforms, site roads, pads, and the tailings dam. The in-pit quarry has a 40 m depth and is approximately 400 m by 300 m. The specific location was chosen as there was minimal ore and easy access to the andesite rock beneath a shallow 30 m layer of saprolite. The ore mined during pre-production from this phase will be stockpiled. The ramp exits to the north where most of the construction work is located.

Phase 1 (Figure 16.3) is the first phase of the mine targeting ore production. Its goal is to quickly access shallow lying ore and minimize waste stripping at the start of commercial production. Phase 1 has the lowest stripping ratio of all the pit phases. The pit is an extension to the in-pit quarry expanding the footprint in the southeastern direction. The in-pit quarry helps strip some of the waste material, further improving the stripping ratio of Phase 1. Single lane ramps are present at the bottom two benches to access additional high-grade ore. The ramp exits on the east side facing north to easily access the processing plant and the primary waste dump. Some material will head south to the dam, the east ramp offering a compromise to the various material destinations. Phase 1 has a depth of 140 m and is approximately 650 m at its longest and 480 m wide.

Phase 2 (Figure 16.4) shares a final wall with Phase 3 to the west. This final wall is shared until level -5 where the pit separates keeping an 80 m minimum mining distance from the final pit wall. This is done to allow Phase 2 to reach additional depths to access ore and pre-strip some of the Phase 3 material. Phase 2 has two switchbacks. The ramp exits to the north for access to the processing plant and the primary waste dump. Phase 2 has a depth of 255 m and is approximately 950 m at its longest and 660 m wide.

Phase 3 (Figure 16.5) is the final phase achieving the ultimate pit limits. The design is a large clockwise ramping circular pit. An additional ramp from level 105 exits to the south to better access the optional tailings dam waste dump and the tailings dam. Like other phases, the primary ramp exits to the north to access the processing plant and the primary dump. A double-lane ramp was used to slot down to the bottom of the pit. Phase 3 has a depth of 355 m and is approximately 1,250 m along strike and 860 m wide.

Figure 16.2: Phase 0 – In-pit Quarry Mine Design

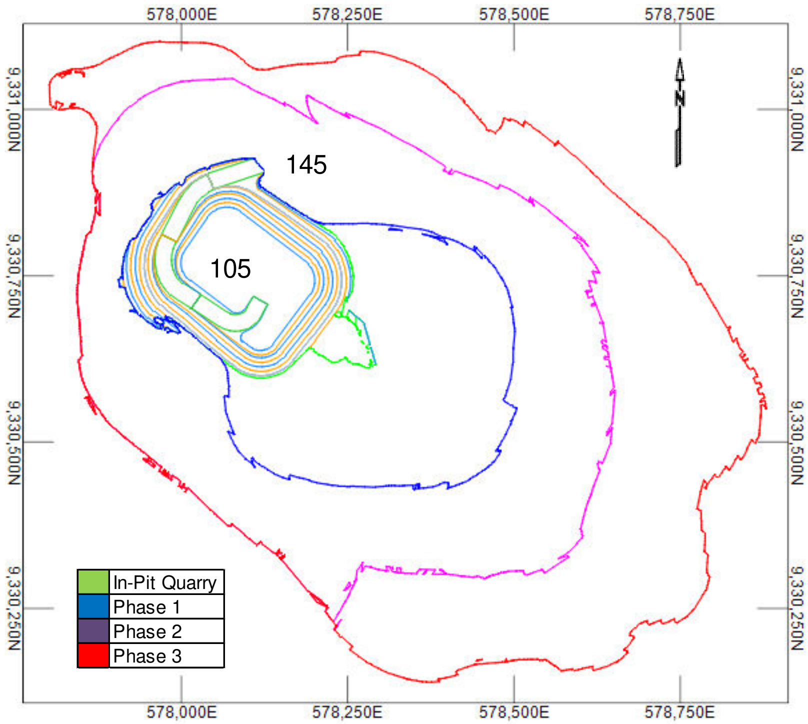
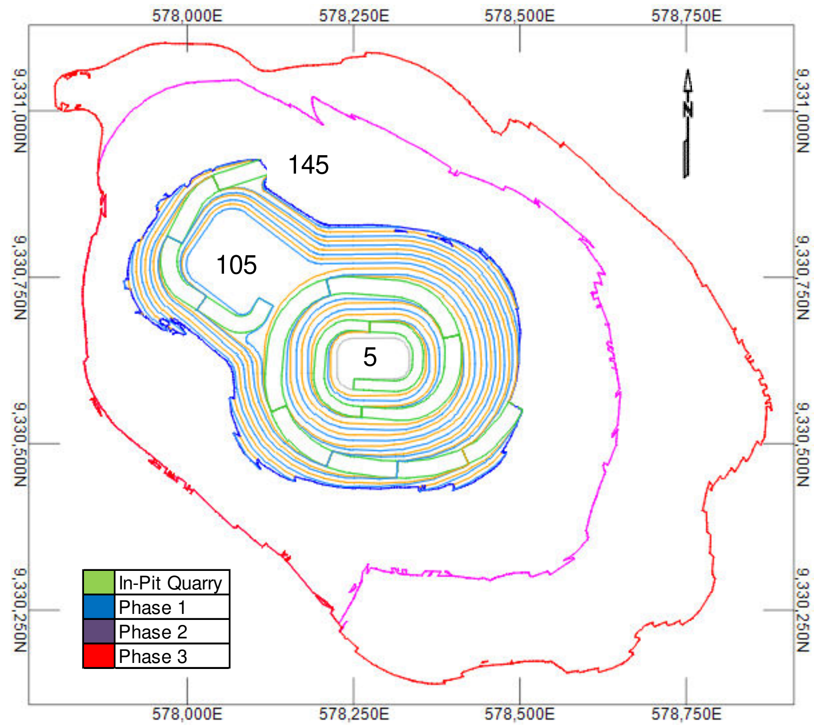


Figure 16.3: Phase 1 Design



Source: GMS, 2021

Figure 16.4: Phase 2 Design

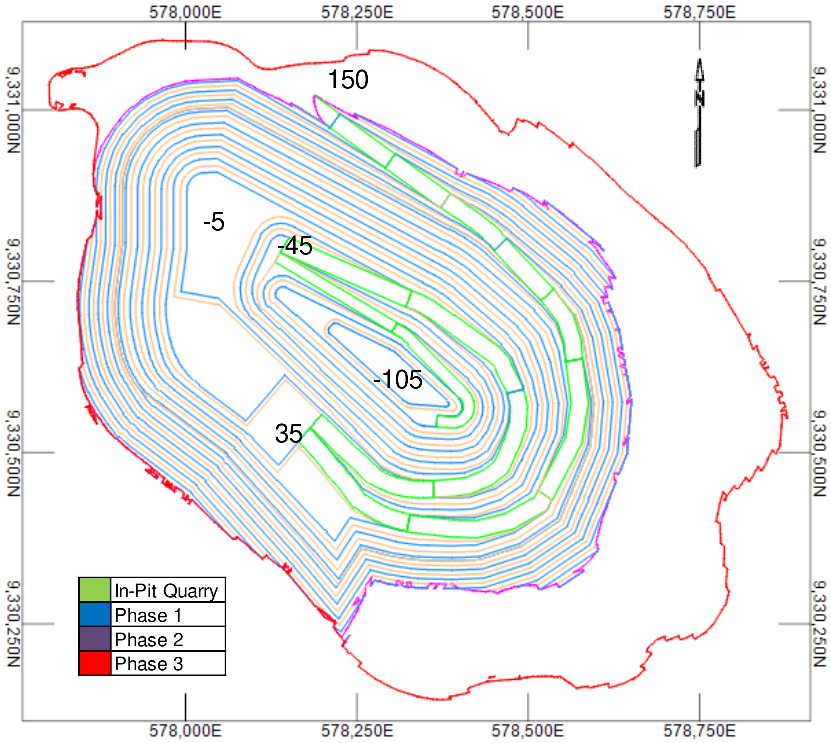
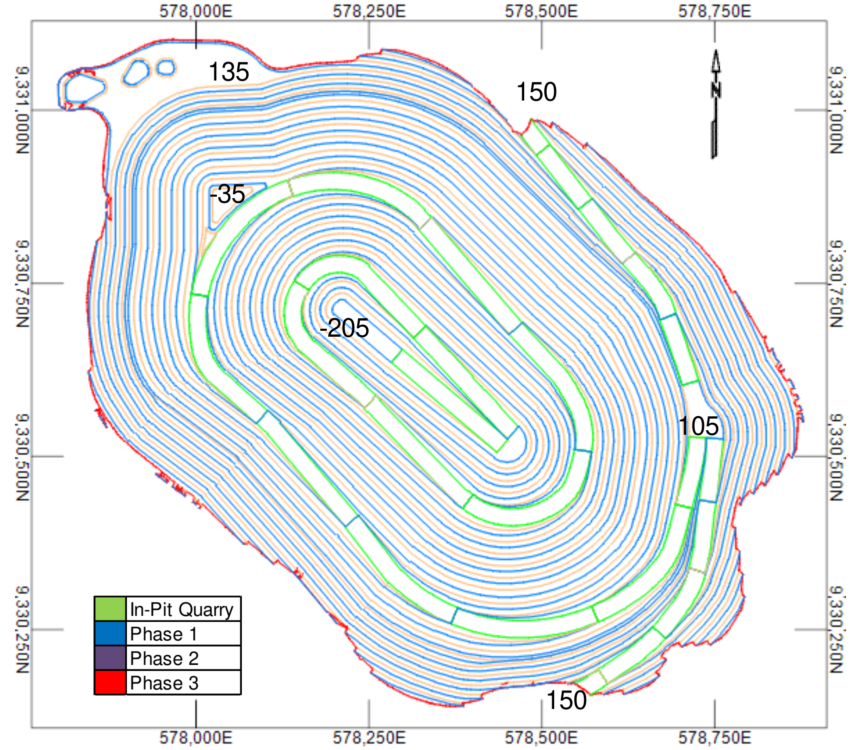


Figure 16.5: Phase 3 Design (Final Pit)



Source: GMS, 2021

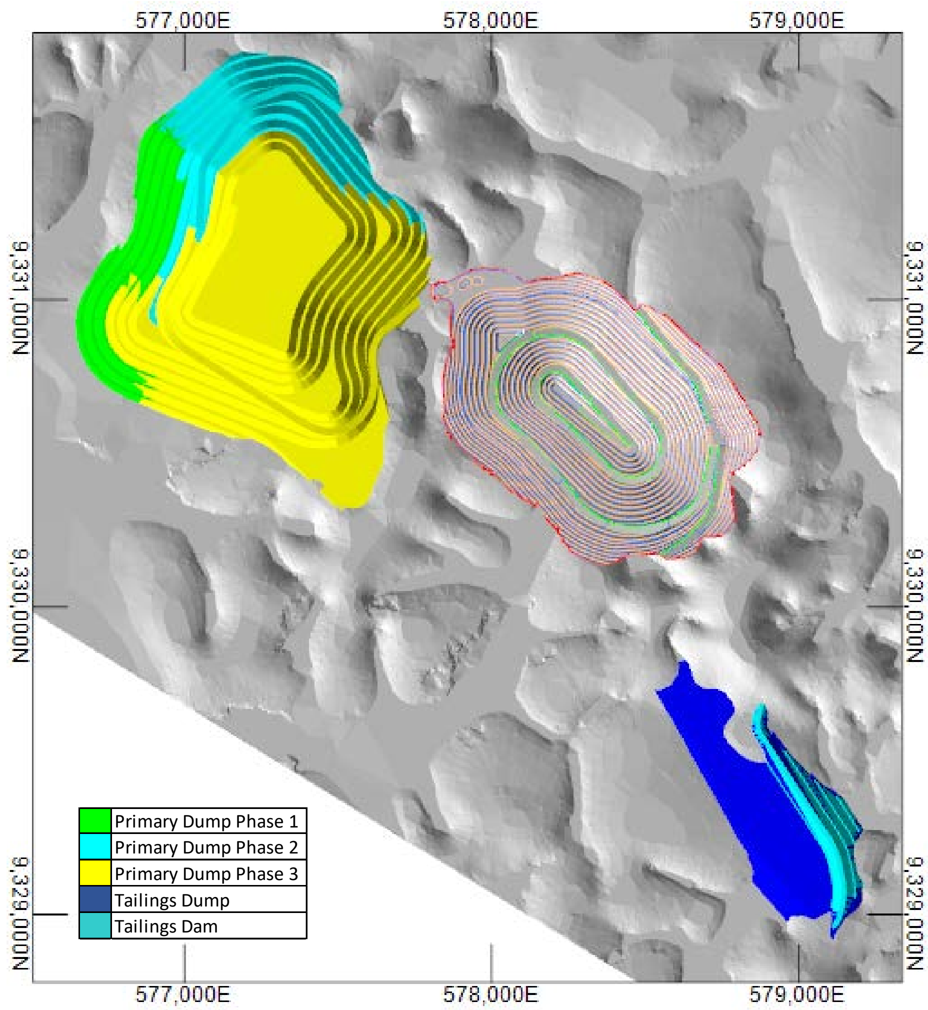
16.3 Waste Storage Facilities

A total 163.4 Mt of waste material is produced over the mine life. This material can be sent to one (1) of two (2) waste storage facilities; the primary waste dump is located to the northwest of the pit and the optional tailings dump would be downstream of the tailings dam (Figure 16.6).

The primary waste dump is built up in three (3) phases or stages to reduce the footprint in early years. Each phase has a dedicated ramp for access.

The tailings dump is located against the tailings dam and is nestled in a valley allowing radial dump from the top. The tailings dump is an optional dump to provide additional long-term stability to the flotation dam and is not expected to be used before Year 4.

Figure 16.6: Waste Storage Facilities

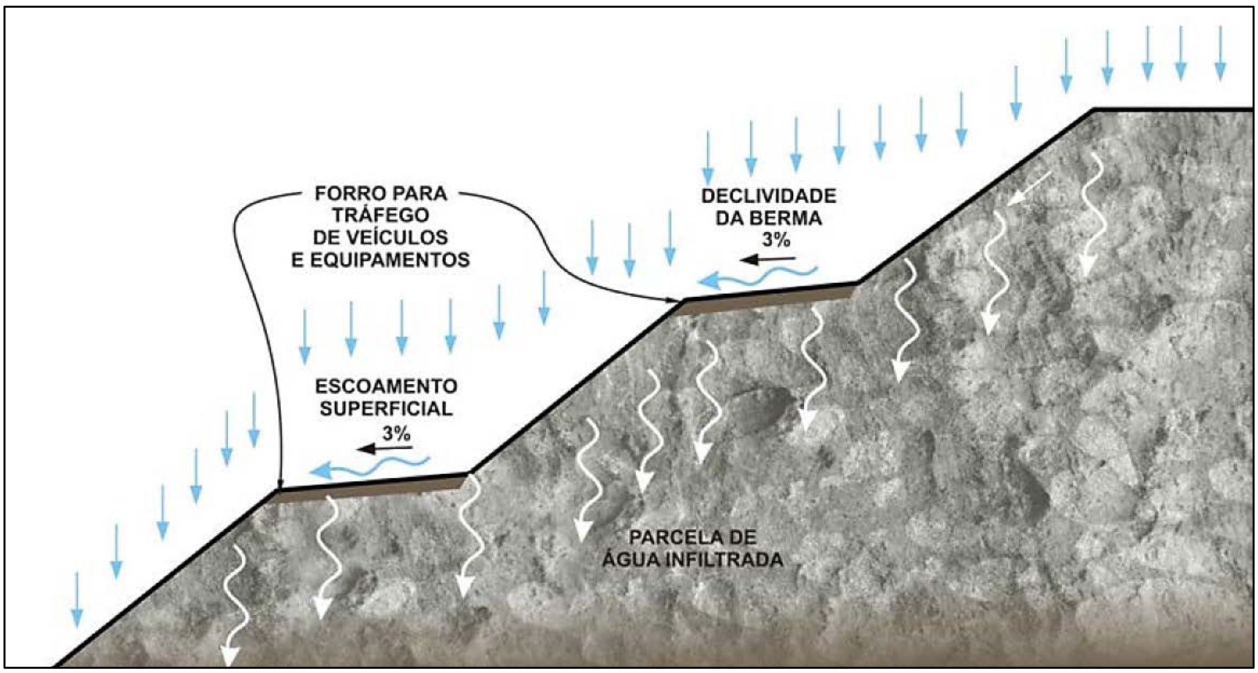


Source: GMS, 2021

The waste storage facility design criteria are as follows:

- A total storage capacity requirement of 163.4 Mt or 81.7 M m^3 based on a density of 2.0 t/m^3.
- Primary dump with a maximum height of 140 m (i.e., elevation 270) for a total capacity of 79.2 M m^3 on a footprint of 107 ha.
- Tailings dump with a maximum height of 30 m (i.e., elevation 165) for a total capacity of 4.5 M m^3 which could be increased. The TSF dump currently has a footprint of 45 ha.
- Lift height of 20 m with an angle of repose of 37 degrees.
- Berm width of 10 m resulting in an overall slope angle of 2.1H:1V (or 25 degrees).
- An offset distance of 300 m horizontally and 5 m vertically from the Tocantinzinho River.
- The berm surfaces will have a 3% slope to allow for surface drainage (see Figure 16.7).

Figure 16.7: Inclined Berms

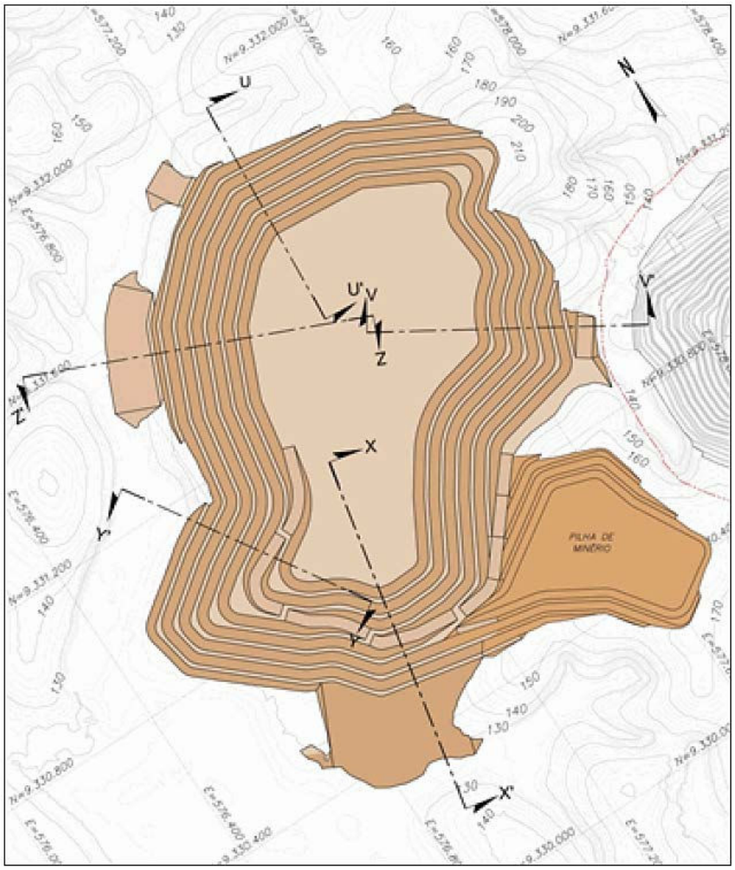


Source: Tec3, 2016

Detailed stability analyses were undertaken for the previous design of the waste storage facility, under drained and undrained conditions. This design and the location of the analyses are shown in Figure 16.8. The analyses show that the calculated factor of safety exceeds minimum standards in all cases (see Source: Tec3, 2016

Table 16.2). Considering that the waste storage facility is located in the same area, and it has more conservative design criteria, these analyses are deemed to still be valid.

Figure 16.8: Previous Waste Storage Facility Design and Section Lines of Stability Analysis



Source: Tec3, 2016

Table 16.2: Primary Dump Stability Analysis Results

Condition	Location	Saturation Setting	Minimum Factor of Safety	Factor of Safety Obtained
-	Between Berm	-	1.5	1.63
Non-Drained	Section U-U’	Nominal Angle of Operation	1.3	1.53
	Section V-V’			1.50
	Section X-X’			1.83
	Section Y-Y’			1.50
	Section Z-Z’			1.66
Final	Section U-U’	Nominal Angle of Operation	1.5	1.53
	Section V-V’			1.50
	Section X-X’			1.97
	Section Y-Y’			1.50
	Section Z-Z’			1.72

Topsoil material recoverable at surface from cleared footprints will be placed in the topsoil storage area to be used for progressive reclamation activities.

16.4 Ore Stockpile

The ore stockpiles are located directly north of the pits, near the pit exits and adjacent to the primary crusher. The ROM pad will also be able to store ore feed which can be rehandled with a front-end loader. Material from the primary stockpiles will be rehandled into trucks and taken to the crusher.

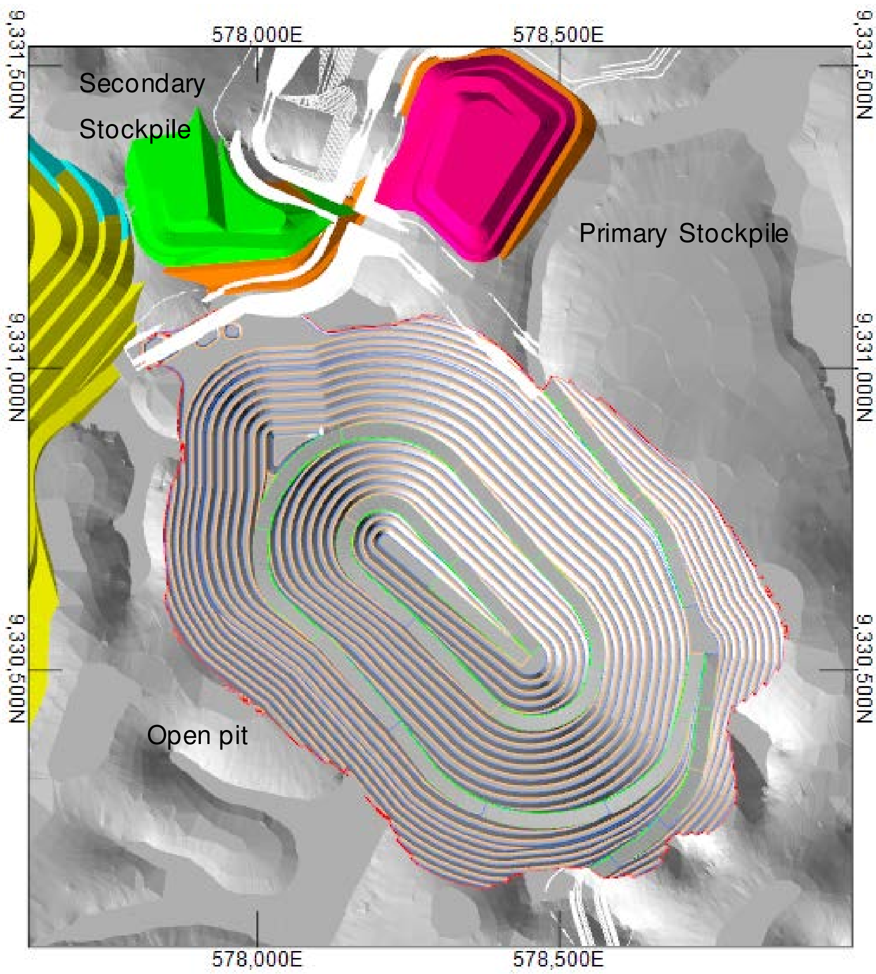
A maximum 8.9 Mt of stockpiled ore is planned at peak capacity. This material is stockpiled to cover periods of increased stripping and to match blending requirements for the mill. The stockpiled ore is split into five primary categories: low grade fresh, medium grade fresh, high grade fresh, saprolite and tailings. The fresh rock ore grade bins are as follows:

- Low grade: 0.36 to 0.80 g/t
- Medium grade: 0.80 to 1.50 g/t
- High grade: > 1.50 g/t

The primary stockpiles will vary greatly in size and shape. The low-grade ore will be built over time and mostly reclaimed at the end of the mine life with other piles more actively used for leveling mine and mill production capacities. The saprolite and tailings stockpiles are drawn from according to throughput limitation of 1,000 tonnes per day of plant feed for these two materials. Stockpile design criteria consist of the following:

- Lift height of 20 m and angle of repose of 37 degrees
- Berm width of 10 m for an overall slope angle of 2.1H: 1V
- Maximum height of 50 m

Figure 16.9: Stockpile Layout



Source: GMS, 2021

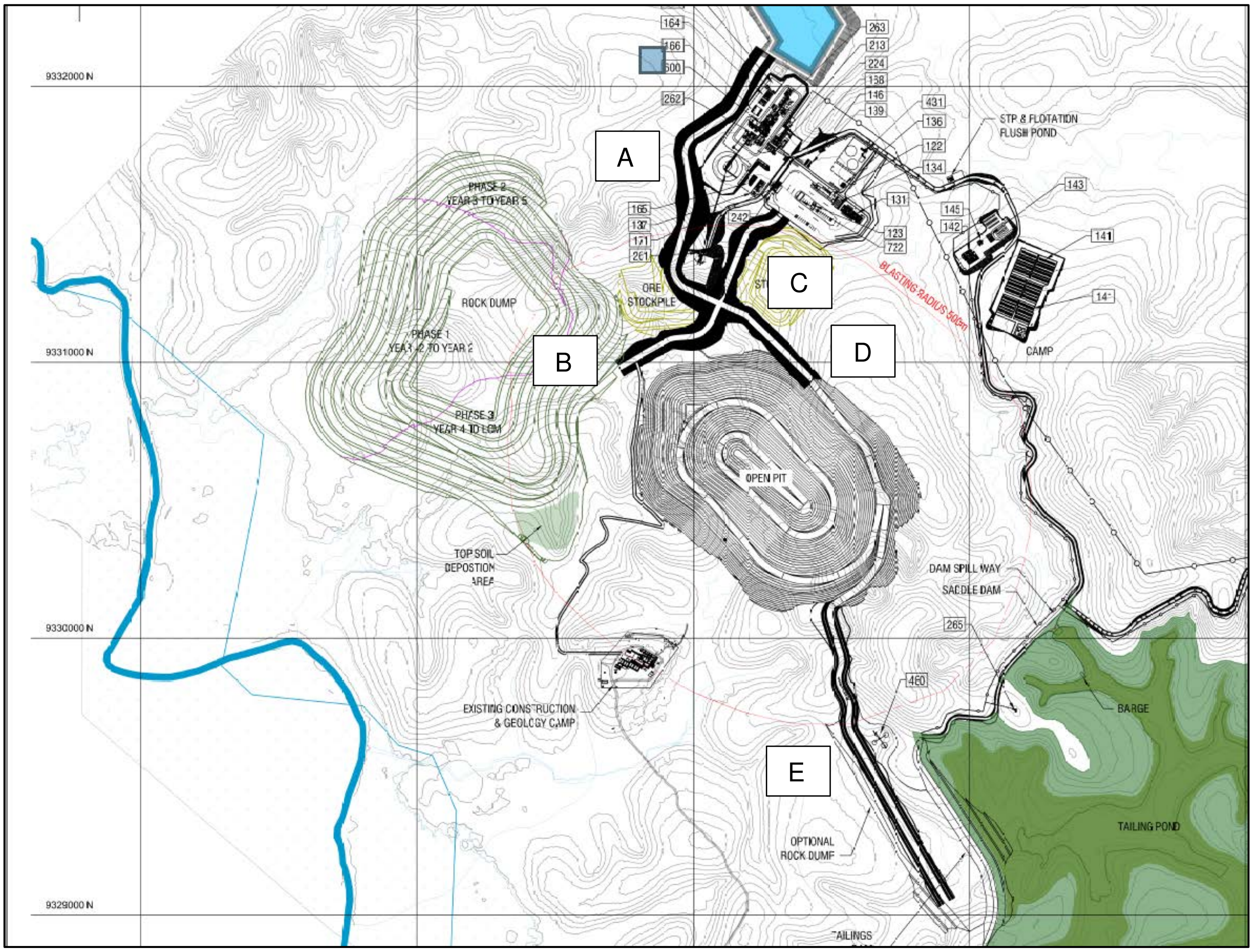
16.5 Mine Haul Roads

The mine site layout is compact requiring only 4.2 km of mine haul roads to connect the open pit, waste dumps, ore stockpiles, truck shop, CIL Ponds and TSF dam (see Figure 16.10, Table 16.3). The ex-pit roads have a rolling surface of 22 m and safety berms as required downslope of the road (Figure 16.11).

Table 16.3: Ex-pit Roads

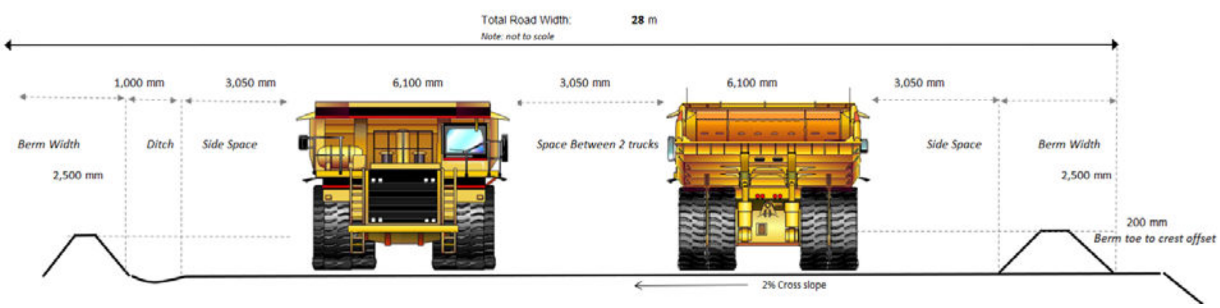
Road Segment	ID	Length (km)
Junction to CIL Pond	A	1.4
Junction to Waste Dump	B	0.5
Junction to Truck Shop	C	0.5
North pit exit to Junction	D	0.5
South pit exit to TSF	E	1.3
Total		**4.2**

Figure 16.10: Mine Haul Road Layouts



Source: GMS, 2021

Figure 16.11: Ex-Pit Haul Road Design Criteria



Source: GMS, 2021

16.6 Production Schedule

The life-of-mine production schedule was optimized using Minemax Scheduler which is an industry leading schedule optimizer using best in class CPLEX technology. Minemax Scheduler is an automated mine scheduling tool which leverages multi-period optimization to determine maximum 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 on an annual basis were then further detailed monthly using Deswik to track material movements, stockpile inventory, mill blending, waste movements and equipment usage / movements.

16.6.1 Mining Schedule

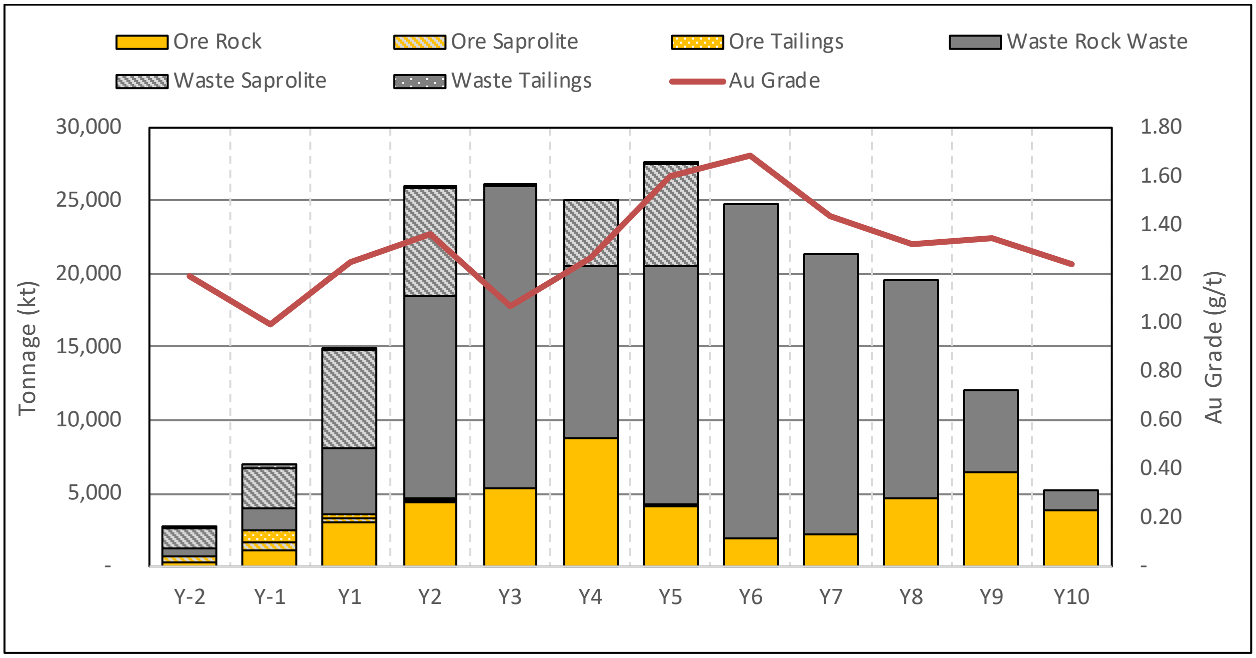
Mining activities are planned over a duration of 11 years which includes two (2) years of pre-production mining. Once the open pit is depleted and mining activities are stopped stockpile reclaim continues for another 1.5 years to continue feeding the mill. The mining rate reaches a peak of 27.5 Mt/y in Year 5 of commercial production. Figure 16.12 presents the mining schedule by material type and gold grade.

Figure 16.13 presents the tonnage mined from the various pit phases over the mine life. In any given year there are up to two (2) active mining phases at once where generally one phase is the primary source of ore and the other is being stripped. The mine plan kept a maximum sinking rate of 60 m for the first five years and increased to 70 m (or 7 benches) from Year 6 onwards. This maximum bench turnover rate limits the mining rate from Year 7 onwards at which point only the final phase remains to be mined.

Mine production details showing mined grades and material movement are presented in Table 16.4.

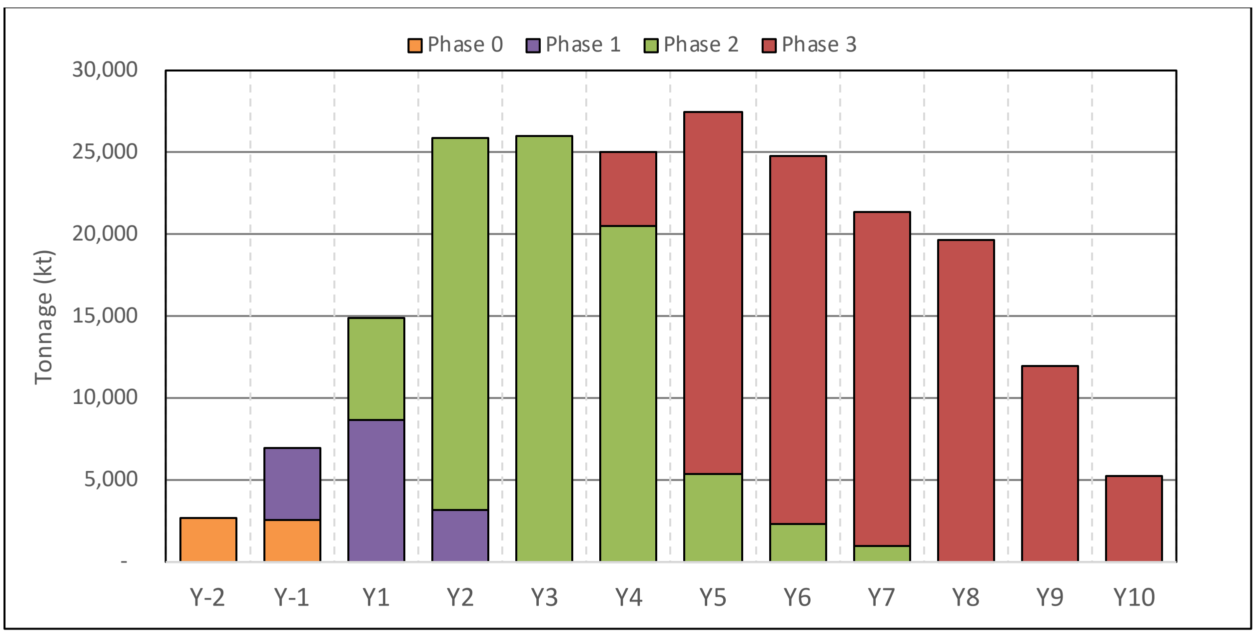
Figure 16.14 to Figure 16.20 show the pit progression for selected years from the end of preproduction to the end of activity in the open pit (Y10).

Figure 16.12: Mine Production Schedule by Material Type



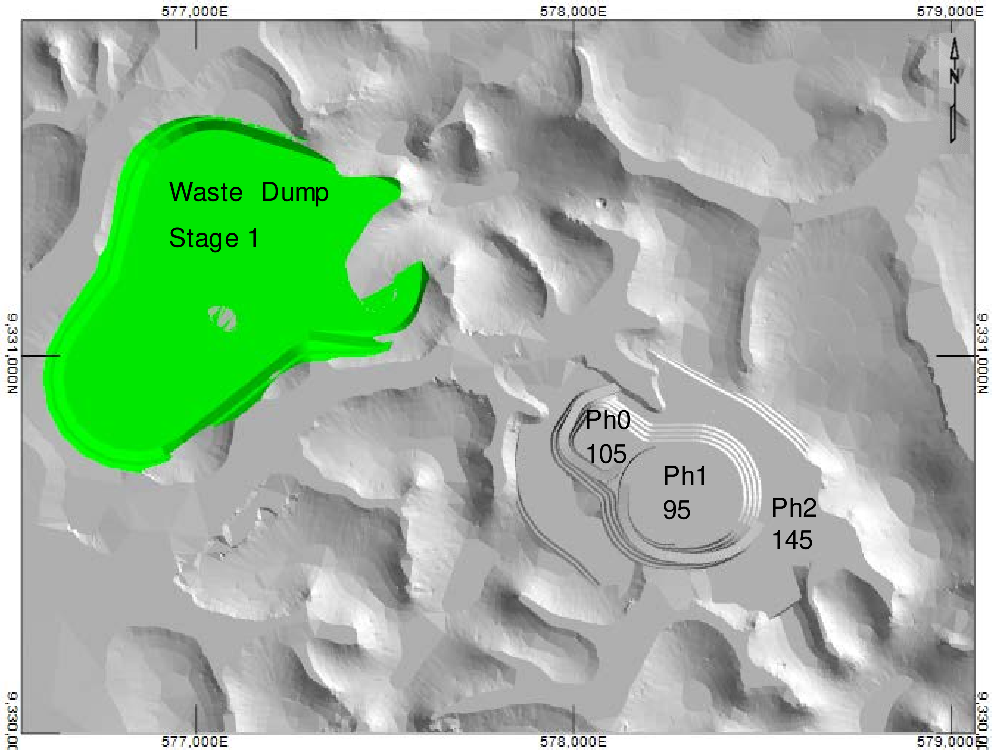
Source: GMS, 2021

Figure 16.13: Mine Production Schedule by Mining Phase



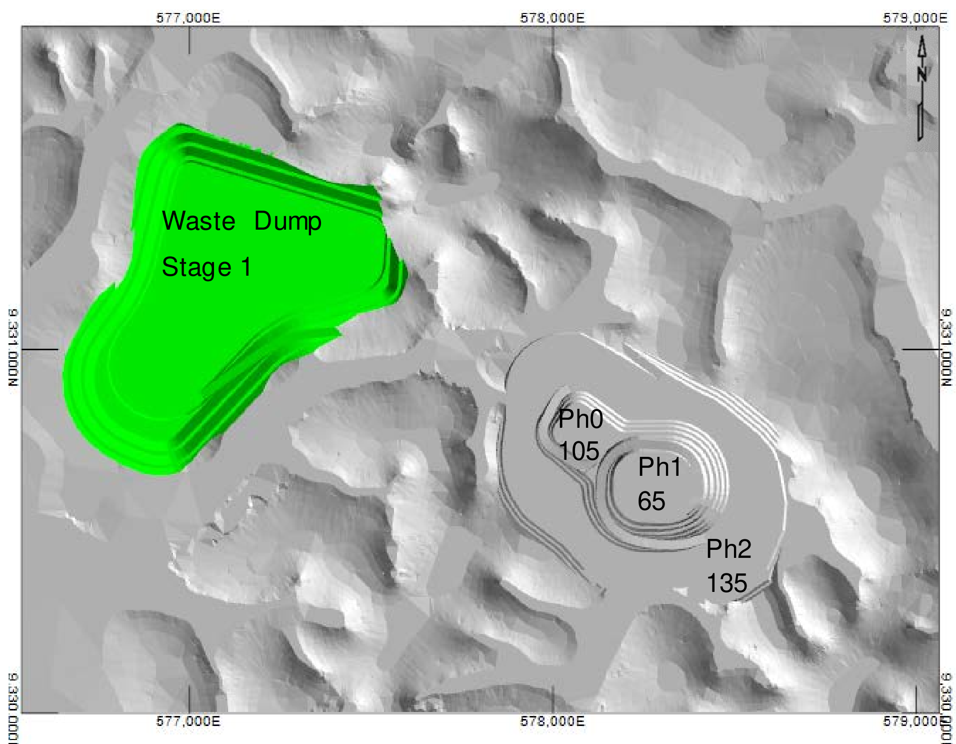
Source: GMS, 2021

Figure 16.14: Mine Development – End of Pre-Production



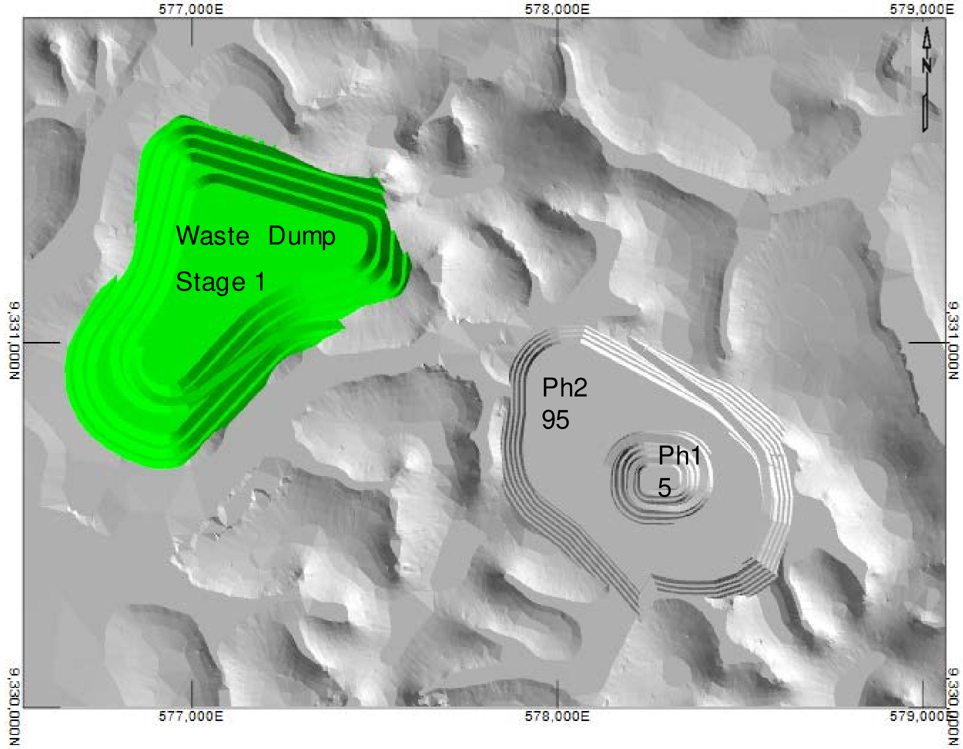
Source: GMS, 2021

Figure 16.15: Mine Development – End of Year 1



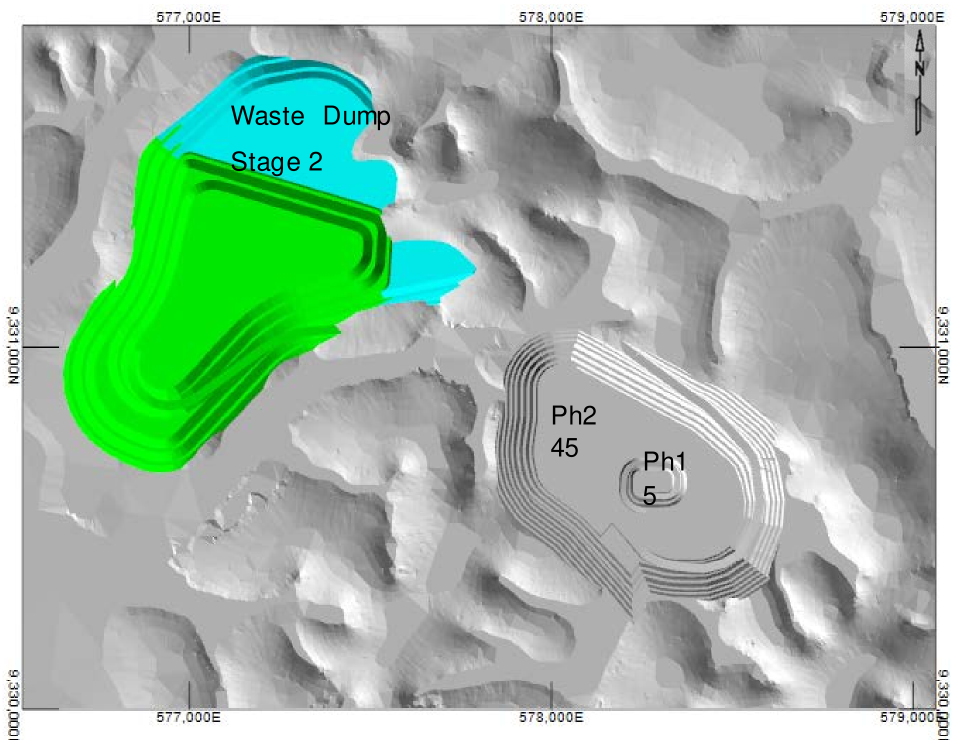
Source: GMS, 2021

Figure 16.16: Mine Development – End of Year 2



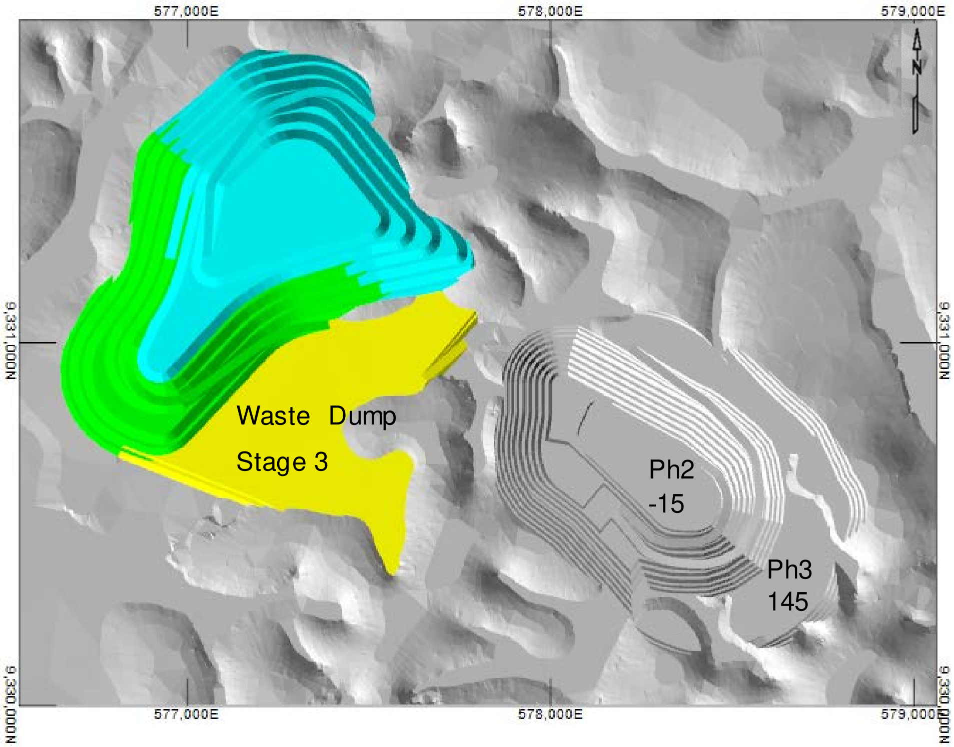
Source: GMS, 2021

Figure 16.17: Mine Development – End of Year 3



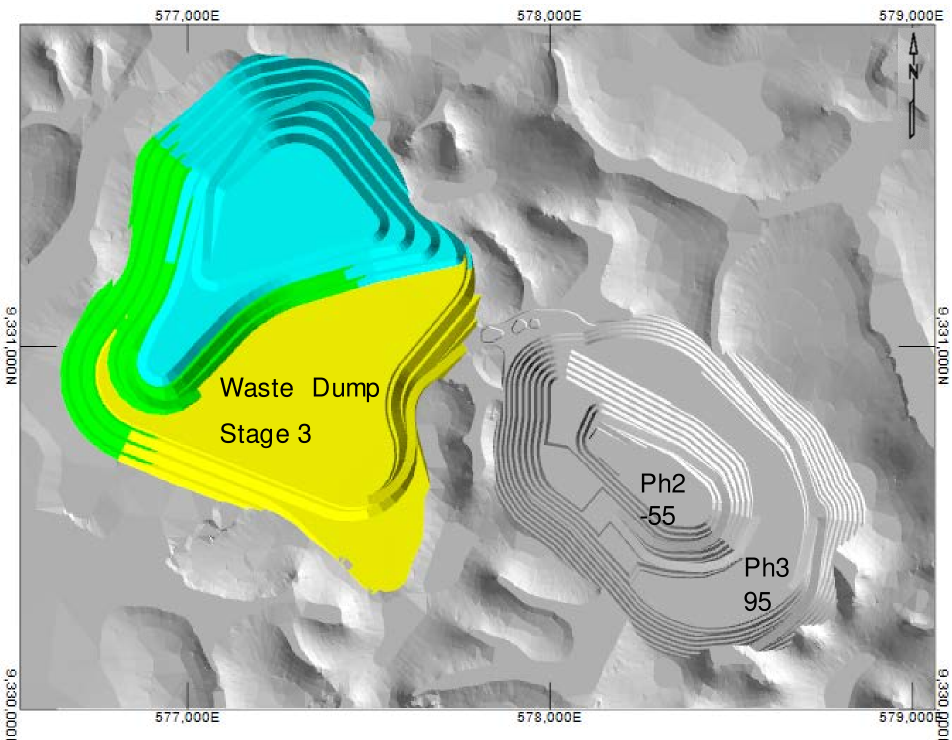
Source: GMS, 2021

Figure 16.18: Mine Development – End of Year 4



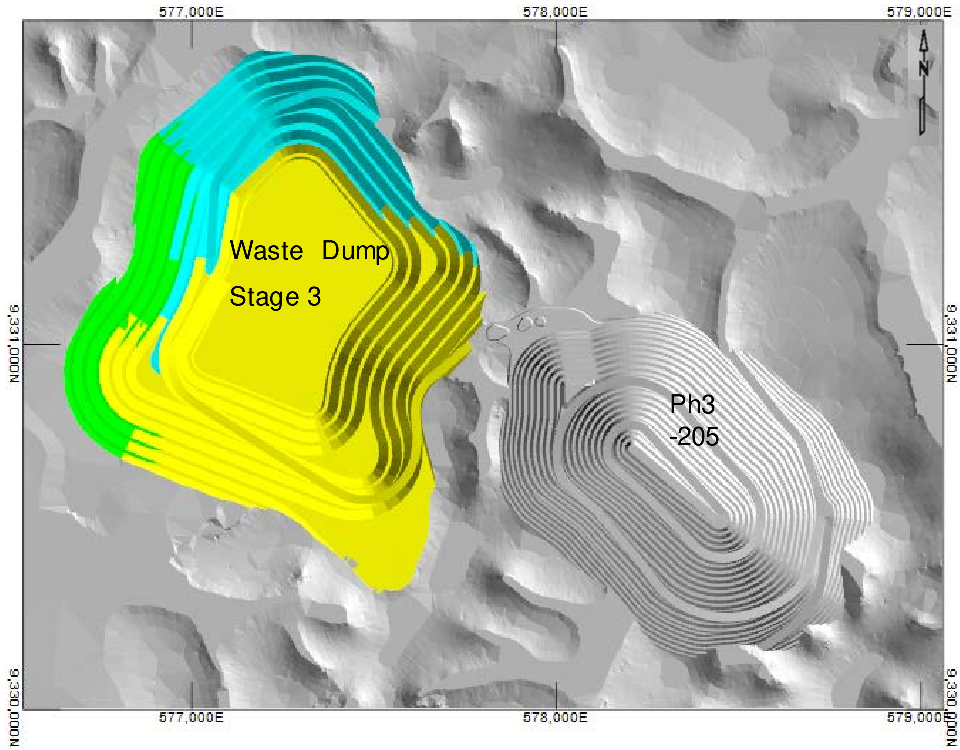
Source: GMS, 2021

Figure 16.19: Mine Development – End of Year 5



Source: GMS, 2021

Figure 16.20: Mine Development – End of Year 10



Source: GMS, 2021

Table 16.4: Mining Production Schedule Summary

Mine Schedule		Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Total Tonnage Moved	000 t	229,024	2,650	7,003	15,740	26,937	26,681	25,381	28,301	27,616	23,910	20,785	12,244	7,553	4,222
Total Tonnage Mined	000 t	212,067	2,650	7,003	14,950	25,918	26,000	25,000	27,500	24,782	21,387	19,625	12,000	5,253	-
Waste Tonnage	000 t	163,391	2,004	4,527	11,391	21,203	20,707	16,273	23,277	22,911	19,132	15,013	5,507	1,446	-
Saprolite	000 t	*29,715*	*1,301*	*2,811*	*6,743*	*7,381*	*4*	*4,445*	*7,030*	-	-	-	-	-	-
Tailings	000 t	*491*	*69*	*267*	*123*	*31*	-	-	*2*	-	-	-	-	-	-
Rock Granite	000 t	*127,869*	*133*	*546*	*3,142*	*12,818*	*19,331*	*11,646*	*16,246*	*22,911*	*19,132*	*15,013*	*5,507*	*1,446*	-
Rock Andesite	000 t	*5,316*	*501*	*904*	*1,384*	*973*	*1,372*	*181*	-	-	-	-	-	-	-
Ore Tonnage	000 t	48,676	646	2,476	3,559	4,715	5,293	8,727	4,222	1,872	2,255	4,612	6,493	3,807	-
Saprolite	000 t	*1,581*	*375*	*667*	*303*	*235*	-	-	-	-	-	-	-	-	-
Tailings	000 t	*1,308*	*5*	*762*	*255*	*149*	-	-	*137*	-	-	-	-	-	-
Rock	000 t	*45,787*	*266*	*1,047*	*3,001*	*4,330*	*5,293*	*8,727*	*4,085*	*1,872*	*2,255*	*4,612*	*6,493*	*3,807*	-
Ore Au Grade	g/t	1.31	1.19	0.99	1.25	1.37	1.07	1.27	1.60	1.69	1.44	1.32	1.34	1.24	-
Strip Ratio	W:O	3.36	3.10	1.83	3.20	4.50	3.91	1.86	5.51	12.24	8.48	3.26	0.85	0.38	-

16.6.2 Processing Schedule

The mill schedule includes two (2) months of commissioning with ore with the second month planned to achieve 60% of nameplate throughput after which commercial production is achieved with 10.5 years of operation. During commissioning and ramp-up the process recoveries are derated while the process plant is stabilized according to the profile presented in Table 16.5.

Table 16.5: Mill Ramp-Up Assumptions

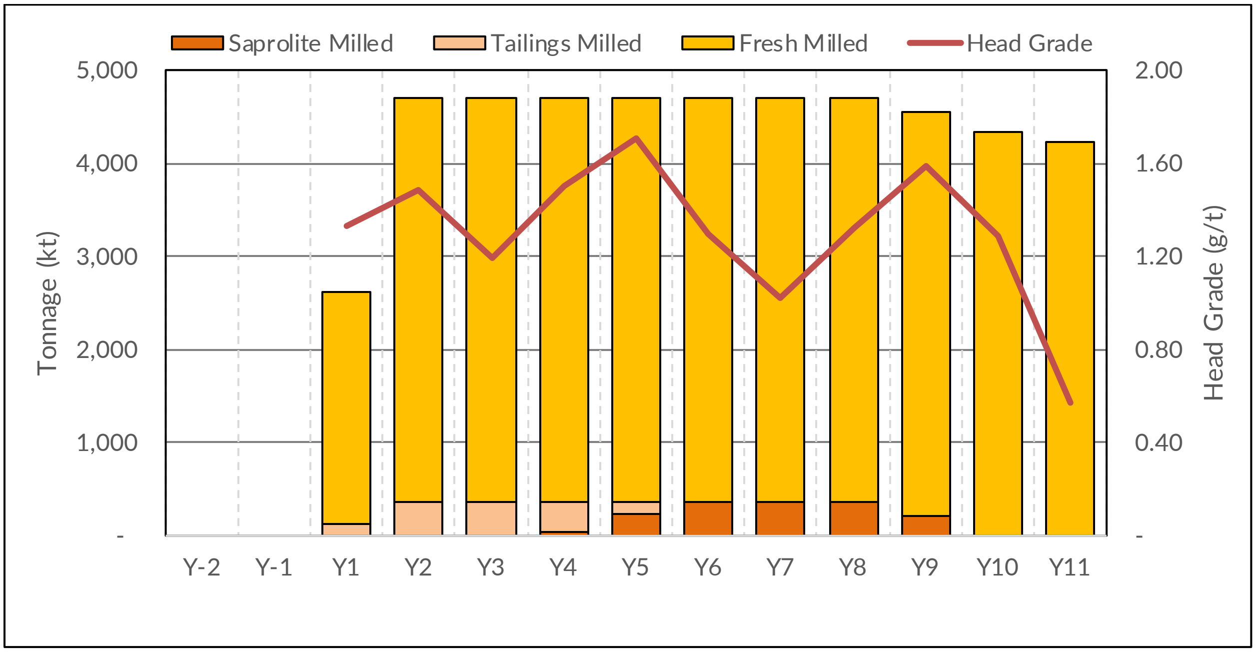
Month	Period	Material Fed Ore Type / Waste	Tonnage Throughput % of Designed	Gold Recovery %			Feed Type
				TAIL	SAP	ROCK	
1	Commissioning	Waste - Rock	20%				Fresh Rock
2	Commissioning	Low Grade - Rock	40%	N/A	N/A	50.0%	Fresh Rock
3	Commissioning	Low Grade - Rock	60%	N/A	N/A	60.0%	Fresh Rock
4	Commercial	All Grades	80%	45.0%	45.0%	80.0%	Fresh Rock
5	Commercial	All Grades	90%	70.0%	60.0%	85.0%	Fresh Rock
6	Commercial	All Grades	100%	85.4%	70.8%	90.9%	Any Type

The peak milling capacity is 4,705 kt/y or 12,890 t/d of nominal throughput and is maintained for the first 7.5 years while softer saprolite and tailings material is available as “supplemental” mill feed at a rate of 1,000 t/d in addition to the fresh rock. Fresh rock represents 94% of the total mill feed with saprolite and tailings representing only 6%. Mill feed is maximized with direct feed from the pit and rehandled stockpiled material. Figure 16.21 and Figure 16.22 present the mill feed by source and the resulting gold head grade by ore type and source respectively. depicts the stockpile inventories by period and grade bin. The long-term stockpiles fulfill two (2) main functions, the first is controlling soft rock (i.e., saprolite and tailings) feed to the mill and the second is improving the overall head grade profile by processing the low grade at the end of the mine life. Excluding the first and last periods, stockpile reclaim is kept at approximately 20% of total feed with the notable exception of years 2029 and 2030 where it reaches up to 60%, due to waste stripping of Phase 3.

At the start of commercial production, there is a stockpile of 4.1 Mt available, of which 1.6 Mt are high and medium grade ore, enough to feed the mill for a four-month period at full capacity. The maximum amount of material stored in the stockpiles is reached in Y4 at 8.7 Mt.

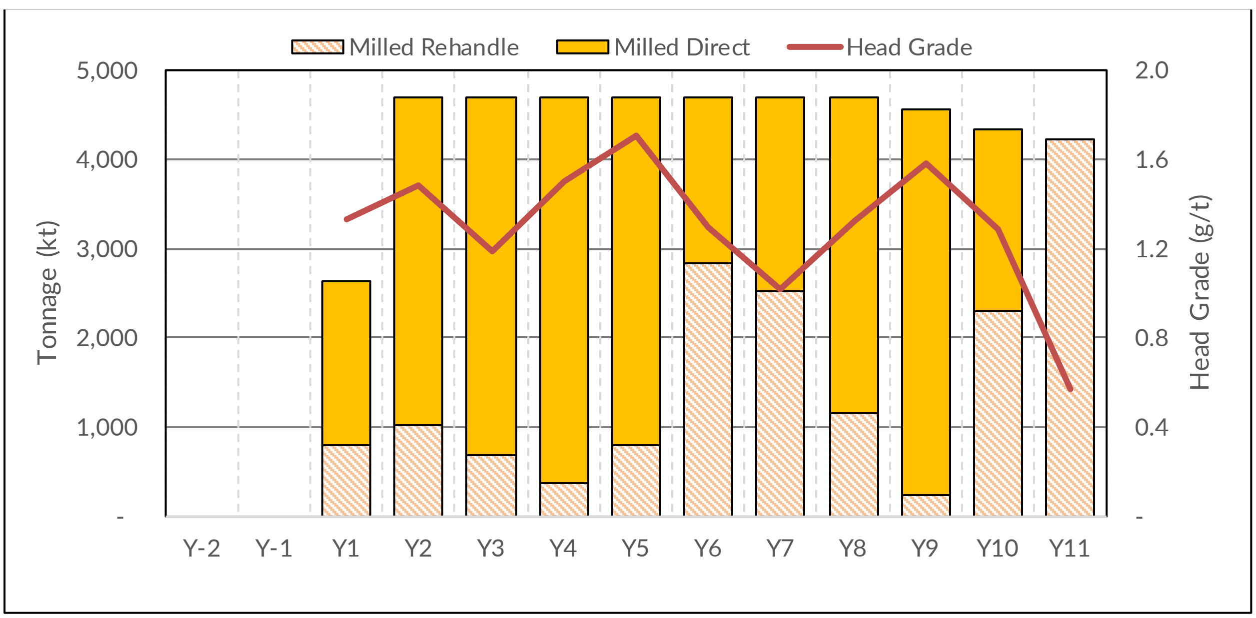
Total gold production is 1,838 koz (Figure 16.24) with an average metallurgical recovery of 90%. Included in this total is 4 koz of gold recovered during pre-production with the balance of 1,834 koz during commercial production. The average gold production during the 10.5 years of commercial production is 175 koz/y. The process production schedule summary is presented in Table 16.6.

Figure 16.21: Mill Feed by Rock Type



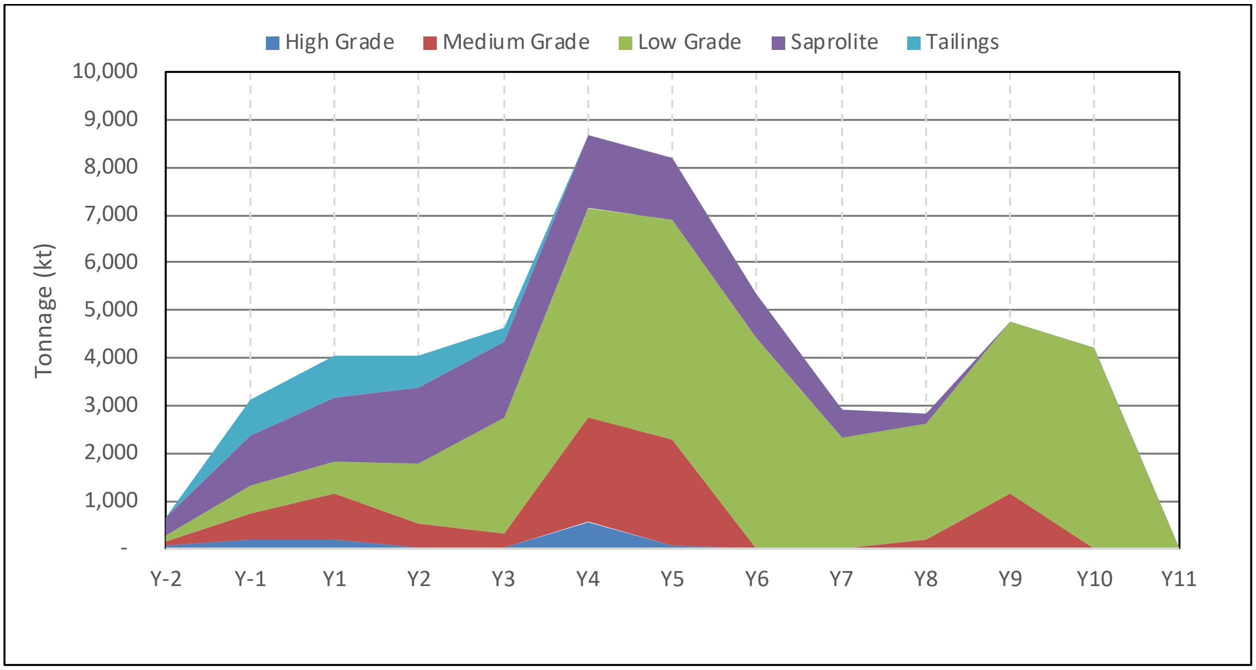
Source: GMS, 2021

Figure 16.22: Mill Feed by Source



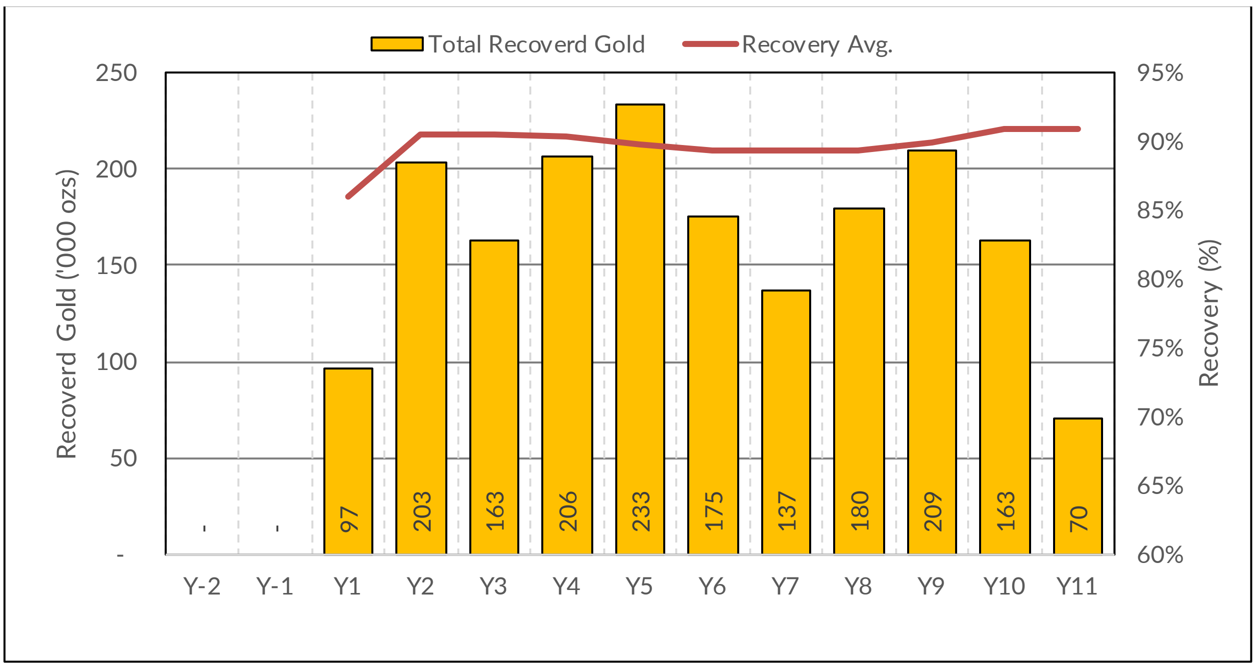
Source: GMS, 2021

Figure 16.23: Stockpile Inventories



Source: GMS, 2021

Figure 16.24: Gold Production



Source: GMS, 2021

Table 16.6: Milling Production Schedule Summary

Process Plant Schedule[1]		Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Total Ore															
Total Ore Milled	000 t	**48,676**	-	-	2,627	4,705	4,705	4,705	4,705	4,705	4,705	4,705	4,552	4,340	4,222
Head Grade	g/t	**1.31**	-	-	1.33	1.48	1.19	1.51	1.71	1.29	1.02	1.33	1.58	1.29	0.57
Recovered Gold	000 oz	**1,838**	-	-	97	203	163	206	233	175	137	180	209	163	70
Saprolite Ore															
Saprolite Milled	000 t	**1,581**	-	-	-	-	-	46	228	365	365	365	212	-	-
Head Grade	g/t	**1.03**	-	-	-	-	-	1.03	1.03	1.03	1.03	1.03	1.03	-	-
Recovered Gold	000 oz	**37**	-	-	-	-	-	1	5	9	9	9	5	-	-
Tailings Ore															
Tailings Milled	000 t	**1,308**	-	-	122	365	365	319	137	-	-	-	-	-	-
Head Grade	g/t	**1.11**	-	-	1.22	1.10	1.12	1.12	0.96	-	-	-	-	-	-
Recovered Gold	000 oz	**40**	-	-	4	11	11	10	4	-	-	-	-	-	-
Fresh Rock Ore															
Fresh Rock Milled	000 t	**45,787**	-	-	2,505	4,340	4,340	4,340	4,340	4,340	4,340	4,340	4,340	4,340	4,222
Head Grade	g/t	**1.32**	-	-	1.34	1.51	1.20	1.54	1.77	1.32	1.02	1.35	1.61	1.29	0.57
Recovered Gold	000 oz	**1,761**	-	-	93	192	152	195	224	167	129	171	204	163	70
EOP SP[2] Inventory	000 t		646	3,122	4,054	4,063	4,651	8,673	8,190	5,357	2,907	2,814	4,755	4,222	-

Notes: 1: Includes pre-production tonnage, 2: EOP SP = End of Period Stockpile Inventory

16.7 Mine Operations and Equipment Selection

Mine operations planning and equipment selection are a function of the materials encountered during mining, the pit geometry, bench height and the production requirements. Unit operations consist of drilling, blasting, loading, hauling and support activities. The size of the fleet was estimated to meet the production requirements based on productivity estimates and overall equipment usage assumptions that are described in the following subsections.

Three (3) main lithologies will be mined in the pit consisting of:

- 1.8 Mt of artisanal miner tailings (0.8%) located at surface on the pit footprint.
- 31.3 Mt of saprolite (14.8%), generally about 30 m in thickness at surface.
- 179.0 Mt of fresh rock (84.4%), the majority consisting of granite (97%) and the other being an andesite formation near surface (3%).

16.7.1 Drilling and Blasting

Production drilling is planned on 10m benches using 216 mm (or 8.5") diameter holes. Production drills capable of single pass drilling specifically designed for 10 m benches with a hole range of 152 mm to 270 mm are selected which are capable of rotary drilling or down the hole ("DTH") drilling. Rotary drilling will be efficient for tailings and saprolite and it is expected that DTH drilling will be more efficient for fresh rock.

The saprolite material will be primarily free digging but a provision has been made that 10% will require blasting albeit with a lower powder factor. The artisanal miner tailings have the consistency of sand and will not require blasting. Production blast holes cuttings will be sampled for grade control purposes and for this reason saprolite mineralized areas will be systematically drilled.

Drill and blast specifications are established according to material type and whether ore or waste. A uniform drill pattern is proposed with a 5.5 m burden and 6.5 m spacing with 1.7 m of sub-drill in fresh rock and no subdrill in saprolite. These drill parameters combined with a high energy bulk emulsion with a density of 1.2 kg/m^3 result in a powder factor of 0.35 kg/t for ore, 0.28 kg/t for waste and 0.20 kg/t in saprolite. Blast holes are planned to be initiated with non-electric detonators and primed with boosters.

Table 16.7 summarizes the drill parameters that are utilized in estimating drill requirements.

Table 16.7: Drill & Blast Parameters

Drill Pattern by Material Type		Saprolite Ore	Fresh Rock Ore	Saprolite Waste	Fresh Rock Waste
Drill Patterns					
Burden	*m*	5.50	5.50	5.50	5.50
Spacing	*m*	6.50	6.50	6.50	6.50
Subdrill	*m*		1.70		1.70
Stemming	*m*	7.10	4.25	7.10	5.40
Bench Height	*m*	10.0	10.0	10.0	10.0
Blasthole Length	*m*	10	11.7	10	11.7
Pattern Yield					
Rock Density	*t/bcm*	1.78	2.62	1.78	2.72
BCM / Hole	*bcm/hole*	358	358	358	358
Yield per Hole	*t/hole*	636	937	636	972
Yield per Meter Drilled	*t/m drilled*	64	80	64	83
Powder Factor	*kg/t*	0.20	0.35	0.20	0.28
Drill Productivity					
Re-Drills	*%*	5.0%	2.0%	5.0%	2.0%
Pure Penetration Rate	*m/h*	75.0	37.5	75.0	37.5
Overall Drilling Factor (%)	*%*	65%	65%	65%	65%
Overall Penetration Rate	*m/h*	48.8	24.4	48.8	24.4
Drilling Efficiency	*t/h*	3,102	1,951	3,102	2,026
Drilling Efficiency	*holes/h*	4.88	2.08	4.88	2.08
Drill OEE	*%*	55%	55%	55%	55%

Controlled blasting techniques will be used including buffer blasts and pre-splits. The pre-split consists of closely spaced holes along the design 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 crack that forms the excavation limit and helps to prevent wall rock damage by venting explosive gases and reflecting shock waves. As a best practice, it is recommended that operations restrict production blasts to within 50 m of an unblasted pre-shear line. Once the pre-split is shot, production blasts will be taken to within 10 m of the pre-shear and then a trim shot used to clean the face. Pre-split holes spaced 2.0 m apart will be 20 m in length and drilled with a smaller diameter of 127 mm (5 in.).

A pre-split drill rig (such as the Sandvik DI650i S5 or equivalent) was selected for this application, additionally, it will be used for pioneering work due to its mobility and drilling range (4.5 to 8 in).

Blasting activities will be outsourced to an explosives provider who will be responsible for supplying and delivering explosives in the hole through a shot service contract. The mine engineering department will be responsible for designing blast patterns, relaying hole information to the drilling team, and supervising all blasting activities, according to Brazilian legislation.

16.7.2 Loading

The loading fleet consists of three CAT 6030 diesel hydraulic diggers with 16.5-17 m^3 buckets, two (2) in a shovel configuration and one in a backhoe configuration, and a diesel front-end wheel loader ("FEL") with a 10.5 m^3 bucket. This primary fleet will be supported by two (2) CAT 395 excavators during the start of operation.

The excavators will be matched with a fleet of CAT 777E mine trucks. Although interchangeable, the hydraulic excavators will primarily be operating in ROM and waste rock. The wheel loader will primarily be taking care of the stockpile rehandling activities while complementing the excavators in waste rock. Table 16.8 shows the loading productivity assumptions per loading unit.

Table 16.8: Loading Productivity Assumptions

Loading Unit		6030 Shovel		6030 Backhoe		992 FEL		395 Excavator		
Material		Sap.	Rock	Sap.	Rock	Sap.	Rock	Sap.	Rock	Tails
Rated Truck Payload	*t*	92.5	92.5	92.5	92.5	92.5	92.5	92.5	92.5	92.5
Heaped Tray Volume	*m^3*	63.5	63.5	63.5	63.5	63.5	63.5	63.5	63.5	63.5
Bucket Capacity	*m^3*	16.5	16.5	17	17	10.5	10.5	6.5	6.5	6.5
Bucket Fill Factor	*%*	90%	80%	90%	80%	90%	80%	90%	80%	90%
Passes (whole)	*#*	4.5	3.5	4.5	3.5	7	6	11.5	9	12.5
Productivity										
Loading Time	*min*	3.15	2.45	3.15	2.45	5.6	4.8	8.15	6.4	8.85
Dry Tonnes / Op. h	*t/h*	1,227	1,588	1,265	1,636	683	884	478	616	425

16.7.3 Hauling

Haulage will be performed with 92 t mine trucks (type CAT 777E). The truck hours and cycle times have been calculated with the Deswik extension Landform & Haulage (LHS) where the cycle times have been estimated for each period and all possible destinations as there are several waste storage areas. The following assumptions and design criteria were used to guide the simulations:

- Max speed limit of 40 km/h
- Max speed loaded and max speed downhill of 30 km/h
- Average rolling resistance of 4%

Table 16.9: Fixed Cycle Time Components

Time (mins)	CAT 6030	CAT 992	CAT 395
Wait Time Loader	0.6	0.6	0.6
Spot Time at Loader	0.2	0.2	0.2
Loading Time (Sap-Rock)	3.14 - 2.45	5.6 - 4.8	8.15 - 6.40
Wait Time at Dump	0.5	0.5	0.5
Spot Time at Dump	0.5	0.5	0.5
Dumping Time	0.5	0.5	0.5
Total Fixed Time	4.74	6.9	10.2

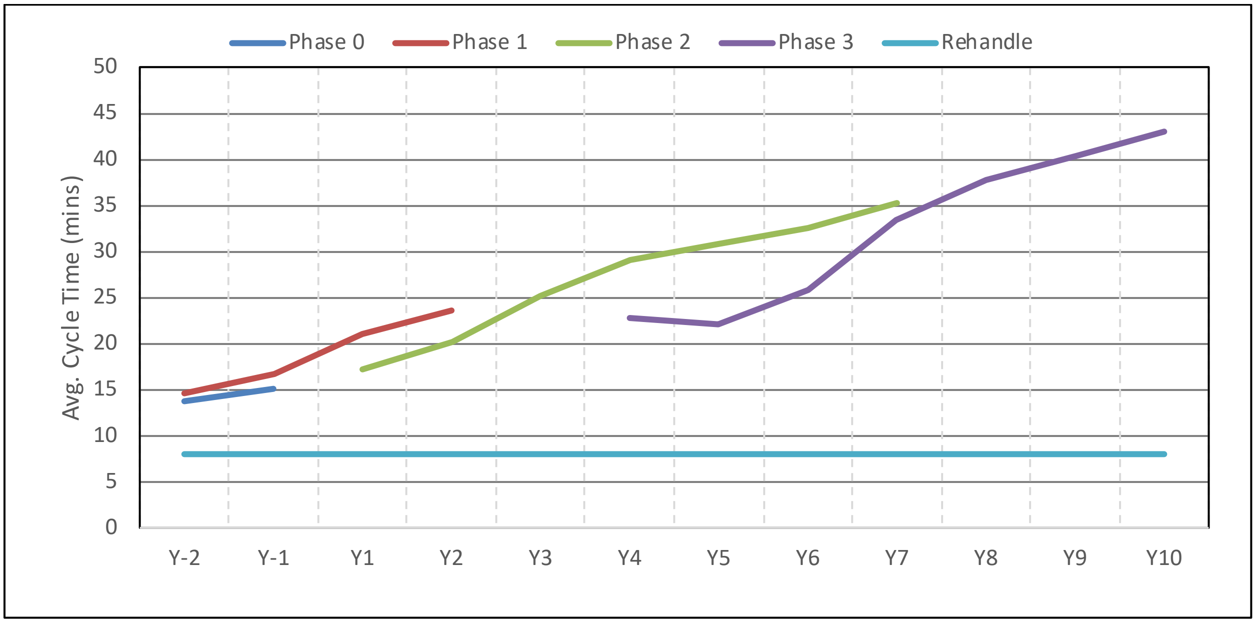
A multiple waste dump strategy is used to minimize the truck requirements for the Project. During the critical years of the Project, waste rock is sent to the closest waste dump to smooth truck requirements. Figure 16.25 shows the truck cycle times by destination, while Figure 16.26 summarizes the haulage hours by destination. Typically, cycle time increases with the increase of the pit depth over the mine life.

Cycle time is also dependant on the dumping schedule and the distance each dump is from the pits. The dump schedule is planned so that cycle time tends to plateau at a maximum limit to allow for a consistent fleet over most of the mine life. The variation in cycle time between years within the same phase represents material being diverted to a different destination with a new corresponding cycle time.

The total haulage hours required by period, coupled with the truck mechanical availability, were used to determine the number of trucks required throughout the LOM. The truck fleet was optimized to reach a maximum of 26 units in Y6 and it remains at this level until Y8 before it starts decreasing because of a decrease in mining rate. Figure 16.27 below summarizes the truck requirements.

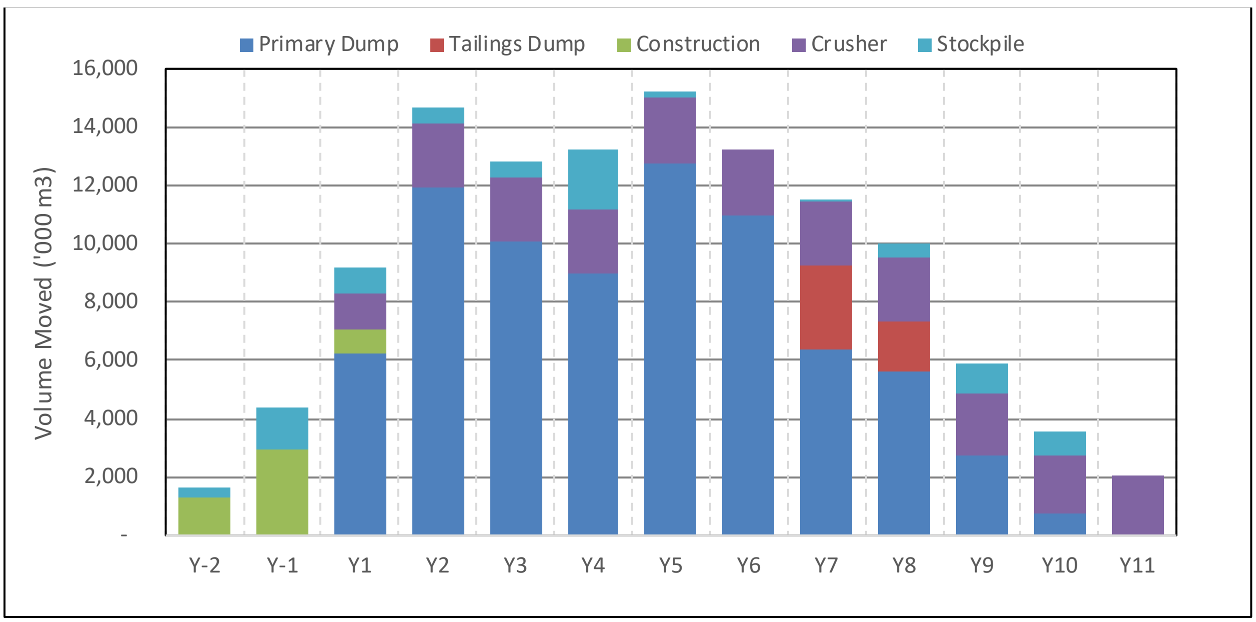
Over the LOM, the truck fleet will have an average fuel consumption rate of 60 liters per hour per truck.

Figure 16.25: Truck Cycle Times by Destination



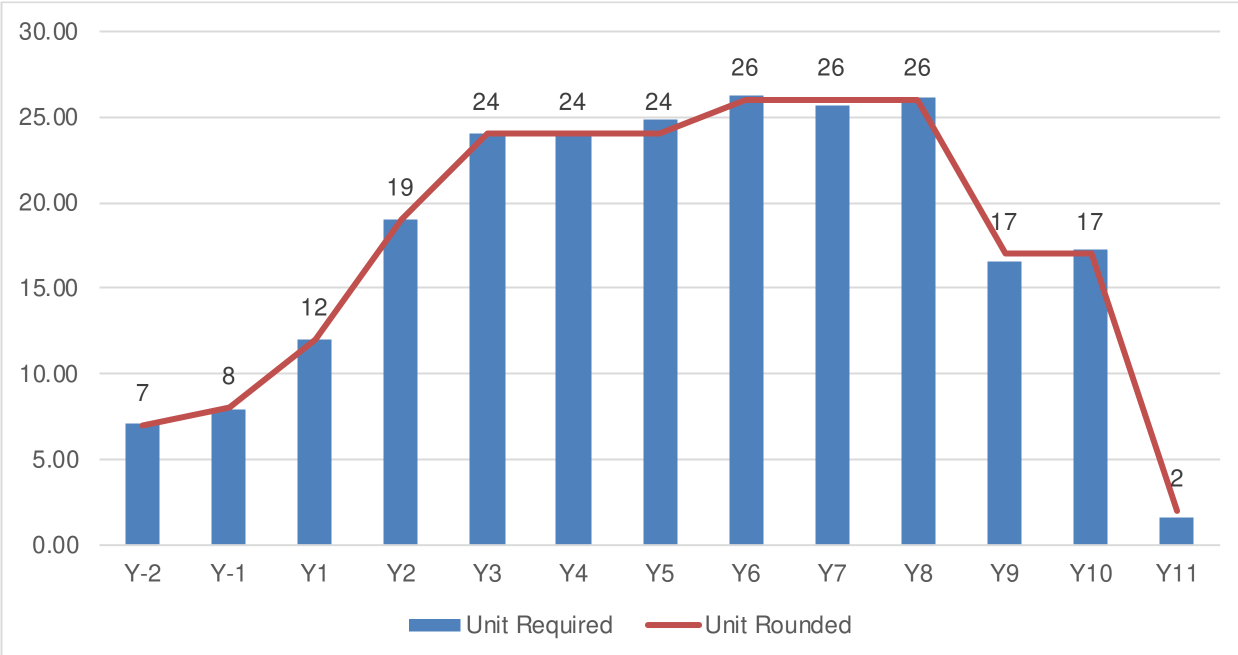
Source: GMS, 2021

Figure 16.26: Material Movement by Destination



Source: GMS, 2021

Figure 16.27: Truck Requirements



Source: GMS, 2021

16.7.4 <u>Support Operations</u>

Support equipment requirements are based on typical open pit mine operation and maintenance requirements to safely support the loading, hauling and drilling fleets.

Support equipment is planned for maintaining dump areas, stockpiles, pit floors and mine roads. The fleet of support equipment consist of the following:

- 3x 436 HP and 2x 215 HP track dozers.
- 2x 16 ft blade motor graders and 1x 14 ft motor grader for the access road.
- 2x water trucks for dust suppression.

A crushing plant will be required to provide mainly two (2) types of crushed rock material: stemming for the blastholes and crushed rock for road maintenance. The stemming rock will be 15 – 20 mm in size and the road maintenance rock will be 50 – 60 mm.

All construction related work, such as berm construction and water ditch cleaning will be done by two (2) 49 t excavators (equipped with an optional hydraulic hammer when required).

A fleet of five (5) troop carries will transport workers to their assigned workplace and a total of 12 pick-ups will be purchased for all the mining departments. Several other equipment purchases are included to support the mining and maintenance activities.

16.7.5 Mine Dewatering

16.7.5.1 Hydrogeological Analysis

The intrusive igneous bedrock complex in the Tocantinzinho Project area is extensive and shows no foliation or cleavage features. Below approximately 70 m depth, the rock mass is fresh, very competent, exhibits few fractures and has low permeability. Within the mineralized zone of the shear zone, however, the rock mass is more altered and fractured, and some evidence of water from staining alteration were observed in fractures during exploratory drilling.

The residual soil and saprolite layer vary in thickness from 10 to 35 m. The texture of the saprolite varies, depending on the original parent rock. Saprolites derived from granitic rocks typically have a sandy-clayey texture.

Below the residual soil / saprolitic layer is a horizon of highly fractured fresh bedrock, with thickness ranging from 20 to 70 m. The average thickness of this horizon is approximately 50 m. No hydraulic conductivity or permeability testing data is available, but for this analysis it was estimated to be “moderate”.

Groundwater recharge in the project area is not well known but was estimated to range from 220 to 660 mm per year, based on 10 to 30 percent of average annual rainfall. However, based on experience of working on projects in similar conditions (i.e., sub-tropical climates with low permeability saprolitic soil profiles), the average annual recharge rate is more likely to be in the range of 5 to 10 percent of annual rainfall, which results in a range of approximately 100 to 200 mm per year.

Field investigations carried out in September 2010 found 42 springs and seep locations. Measurable seepage from these springs ranged from 0.04 to 0.91 litres/second (L/s).

Figure 16.28 presents a groundwater contour map in the Tocantinzinho Project area in the vicinity of the proposed open pit location. The groundwater system is recharged via the infiltration of precipitation in topographically higher areas with discharge in the topographically lower areas (i.e., stream drainages and lakes). In general, localized groundwater flow is from hill tops to stream drainages or lakes. Groundwater flow in the soil / saprolite unit is dissected into about four different areas due to the undulating topography. The general groundwater flow directions are summarized below:

- Area 1 (Northwest area) – groundwater flow is ultimately to the west.
- Area 2 (Central area) – groundwater flow is ultimately to the south-southwest, following the mainstream channel.
- Area 3 (East-Northeast area) – groundwater flow is ultimately to the north-northwest.
- Area 4 (Southeast area) – groundwater flow is ultimately to the east-southeast.

Mine inflows will be from groundwater, surface runoff, and precipitation. For the groundwater inflow component, the primary sources include the saprolite (SR), the highly-fractured rock mass horizon (RMF), and the fractured fresh rock mass within the mineralized zone (RSF).

Dewatering or depressurization of the SR using wells will probably not be effective, because of the low permeability and high storage capacity characteristics of this unit. If dewatering and depressurization of the SR unit is required for geotechnical stability purposes, horizontal drains would be required. Water from the drains would be collected on the pit benches and pumped from the pit.

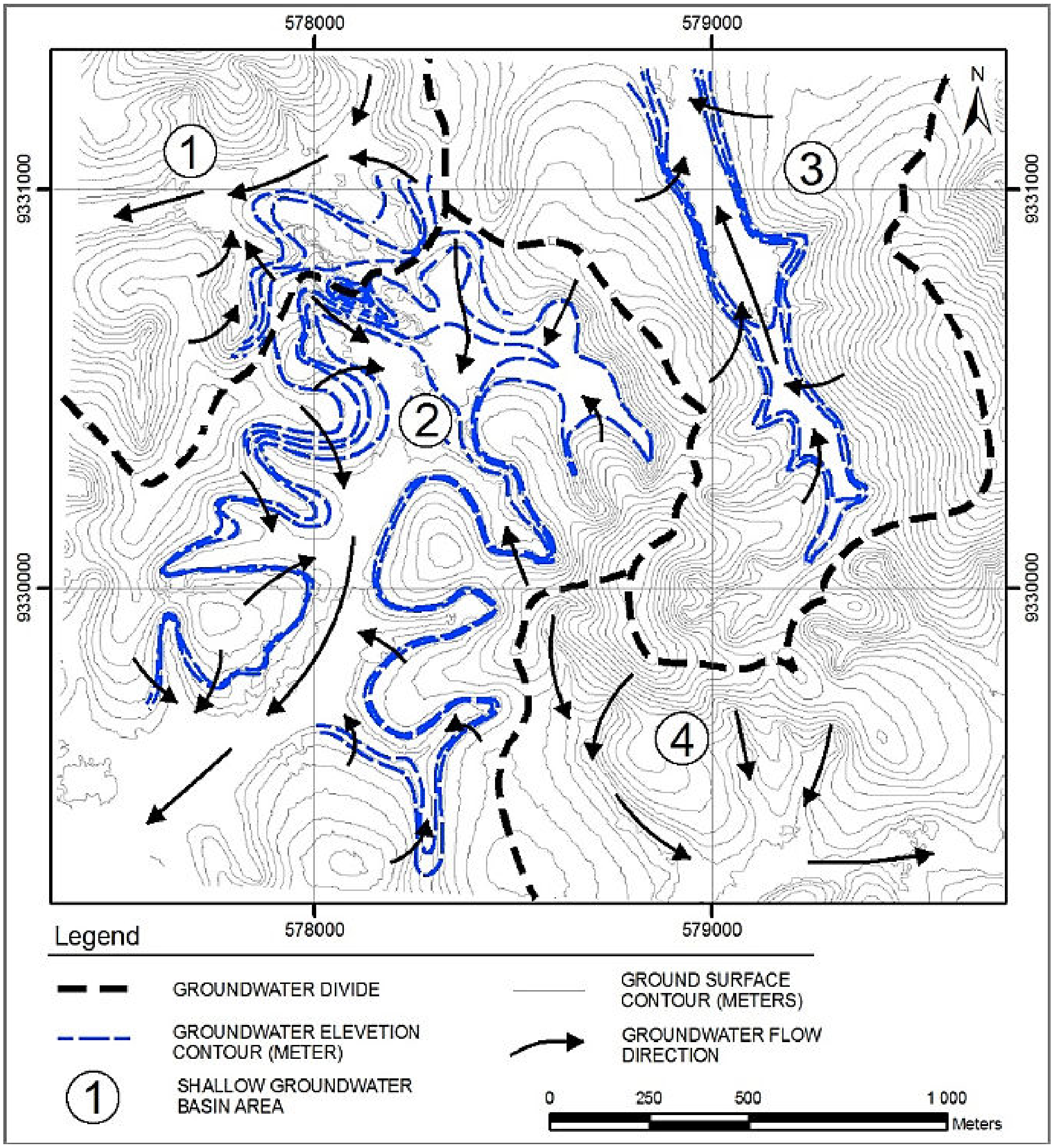
The RMF will likely have the best potential for dewatering from groundwater extraction wells, due to the degree of fracturing and absence of fine-grained materials.

During the later stages of mining, more of the fresh rock mass (RS) will be exposed in the pit walls. Groundwater contribution to the pit from the slightly fractured wall rock (RSPF) is expected to be minimal and can be controlled by sump pumps installed in the bottom of the pit. Groundwater inflow from the fracture rock mass (RSF) may be more of a concern, depending on the permeability of this unit and its hydraulic connection to the Tocantinzinho River.

Mine inflows from precipitation and surface water run-off from the pit slopes will occur primarily during the rainy season (i.e., November to June), with the greatest amount of inflow occurring during February and March (on average). In the dry season (i.e., July to October), evaporation exceeds monthly rainfall on average thus inflow from rainfall and surface run-off will be minimal.

The water balance was prepared for the pit to estimate the quantities of precipitation, surface water and groundwater inflow to the open pit based on the hydrogeological and hydrological investigations. The dewatering system was designed based on these estimates on an annualized basis.

Figure 16.28: Groundwater Contour Map



Source: VOGBR, 2010

16.7.5.2 <u>Initial Dewatering</u>

The current topography in and around the future open pit has some small ponds and areas that were previously mined by artisanal miners and where water has accumulated. Twelve (12) areas were identified and an approximate of 0.6 Mm³ will require to be pumped out before the start of the mining operations. To minimize cost and maximize efficiency, the pumping equipment selected for the operations will be used for the initial dewatering. This water will be pumped into one of the local creeks that leads to the Tocantins River. The maximum head pressure is estimated at 15 m with a maximum of 850 m of piping. Using the pump at a flow rate of 350 m^3/h, it will take an estimate of 72 days to finish the initial dewatering. This initial dewatering will be phased targeting areas with higher priority to start operations.

16.7.5.3 Dewatering During Operations

It is assumed that the average annual precipitation will be of 2,451 mm for rainwater and up to 1.2 Mm^3 for ground water influx, for a total of 20.6 Mm^3 of water over the mine life. The maximum pipe length is estimated to be 5,600 m at the end of the mine life with a total final static head of 370 m. There will be a permanent sump in the pit that will be available to use starting Year 7 of operations. The water will be pumped to the flotation tailings storage facility east of the pit. Pumps will be setup in a series configuration to achieve the total head required without overpressure in the pipes.

16.7.5.4 Pumping System

A total of five (5) pumps will be needed over the mine life. The pump model chosen is the Godwin HL160M as it provides the required maximum flow rate and the total dynamic head. The pump will have a 6” to 10” adaptor. Piping will consist of HDPE 10” for the full length and throughout the life of the mine. The plan will be to buy the pumps from the start of the project. The pumps are not planned to be replaced over the life of the operations. Major components will be changed, and regular preventive maintenance will allow the pumps to last.

16.7.6 Mine Technical Services

The engineering and geology departments will provide support to the operations team by providing short term and long-term planning, grade control, surveying, mine reserves estimation, pit slope monitoring and other technical functions. The Engineering department will consist of a Long Range, Short Range, Drill and Blast, Geotechnical and a Survey department.

Long Range planners will produce a strategic plan that will feed into a rolling three-month plan. Short term planners will use the three (3)-month rolling plan as guidance for their short-term plan. Surveyors will be responsible for pickup up the faces on a weekly basis as well as calculating volumes moved / mined. They will also be responsible for staking out the drill holes and picking up the actual versus planned. After the blast they will stake out the geology packets incorporating any blast movements of the ore / waste contacts.

The geotechnical team will be responsible for monitoring stability of pit slopes, dumps and tailings dams. Once pit slopes are developed, a pit slope monitoring system will be used to gather any information on micro and macro movements of the pit walls. It usually consists of strategically placed prisms that are surveyed under a controlled environment. At this stage, an allowance has been made for INSAR satellite imagery and GNSS receptors to monitor slope stability in the pit, the waste dumps and the TSF dam, but detailing the monitoring requirements should be an element of focus in the detail engineering stage.

16.7.7 Mine Overhead & Management

Despite having a single pit to manage it is planned to install a fleet management system once in commercial production to assist the operations team to maximize efficiency of the fleet, monitor machine health and track KPIs. The system will be managed by a dispatcher on each crew who will control the system by sending operators onscreen instructions to work at peak efficiency.

The Project has not included a maintenance and repair contract ("MARC") for its mobile equipment fleet. 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 OEM's dealer. Major critical parts will be purchased and kept at the mine site as spare components to accommodate the shipping and repair/rebuild delays. The final details on the supply agreements with the manufacturers are yet to be finalized.

Some other equipment will also be purchased to facilitate the maintenance activities and support the operation, such as a forklift, one (1) telehandler TL943, one (1) diesel forklift, one (1) fuel and lube truck, one (1) 100-t low-boy trailer and tractor for moving the tracked equipment. Other small equipment such as mechanic service trucks, generators, compressors, light towers, welding machines and water pumps will be added.

16.8 Mine Fleet Requirements

Table 16.10 summarizes the gross operating hours used for subsequent equipment fleet requirement calculations. The mine is expected to operate 22 hours per day, 355 days per year. This accounts for shift changes and weather delays from heavy rain events. Additional delays and applied factors are described in productivity calculations for each fleet.

Table 16.10: Equipment Usage Assumptions

Item	Unit	Shovels	Loaders	Trucks	Drills	Ancillary	Support
Days in Period	*days*	365	365	365	365	365	365
Weather, Schedule Outages	*days*	10.0	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	2.0
Hours per Shift	*h/shift*	11.0	11.0	11.0	11.0	11.0	11.0
Availability	*%*	82.0	80.0	85.0	80.0	85.0	85.0
Use of Availability	*%*	90.0	90.0	90.0	90.0	85.0	80.0
Utilization	*%*	73.8	72	76.5	72	72.25	68
Effectiveness	*%*	90.0	85.0	87.0	85.0	80.0	80.0
OEE*	*%*	66.4	61.2	66.6	61.2	57.8	54.4
Total Hours	*h*	8,030	8,030	8,030	8,030	8,030	8,030
Scheduled Hours	*h*	7,810	7,810	7,810	7,810	7,810	7,810
Down Hours	*h*	1,406	1,562	1,172	1,562	1,172	1,172
Delay Hours	*h*	576	843	777	843	1,129	1,062
Standby Hours	*h*	640	625	664	625	996	1,328
Operating Hours	*h*	5,764	5,623	5,975	5,623	5,643	5,311
Ready Hours	*h*	5,187	4,780	5,198	4,780	4,514	4,249

**Note: Overall Equipment Effectiveness*

With the equipment production rates and scheduled mine plan tonnage requirements determined, the total mining fleet requirements over the mine life are determined. The number of excavators, haul trucks and drills are based on the scheduled production values provided above while the secondary and support equipment fleet requirements are generally based on the number of excavators and trucks required.

Fleet requirements and unit purchases over the LOM are presented in the following tables. The unit purchases also reflect additions to the fleet as well as equipment replacements planned for support equipment.

Table 16.11: Major Equipment Requirement Schedule

Major Equipment	Max	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Production Drill (6 - 10")	3	-	1	1	3	3	3	3	3	3	3	2	2	-
Auxiliary Pre-Split Drill (4.5 - 8")	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Diesel Hydraulic Shovel (16 m³)	2	-	-	1	2	2	2	2	2	2	2	1	1	-
Diesel Hydraulic Excavator (16 m³)	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Excavator (90 t)	2	2	2	1	1	1	1	1	1	1	-	-	1	1
Wheel Loader (10.7 m³)	1	-	-	1	1	1	1	1	1	1	1	1	1	1
Mining Haul Truck (100 t)	26	7	8	12	20	25	25	25	26	26	26	17	17	2
Track Dozer (436 HP)	3	2	3	3	3	3	3	3	3	3	3	3	2	-
Track Dozer (215 HP)	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Motor Grader (16 ft)	2	1	1	2	2	2	2	2	2	2	2	2	1	-
Motor Grader (14 ft)	1	-	1	1	1	1	1	1	1	1	1	1	1	1
Water Truck (15 kl tank)	2	1	2	2	2	2	2	2	2	2	2	2	2	1

Table 16.12: Major Equipment Purchase Schedule

Major Equipment	Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Production Drill (6 - 10")	3	-	1	-	2	-	-	-	-	-	-	-	-	-
Auxiliary Pre-Split Drill (4.5 - 8")	1	1	-	-	-	-	-	-	-	-	-	-	-	-
Diesel Hydraulic Shovel (16 m^3)	2	-	1	1	-	-	-	-	-	-	-	-	-	-
Diesel Hydraulic Excavator (16 m^3)	1	1	-	-	-	-	-	-	-	-	-	-	-	-
Excavator (90 t)	2	2	-	-	-	-	-	-	-	-	-	-	-	-
Wheel Loader (10.7 m^3)	1	-	-	1	-	-	-	-	-	-	-	-	-	-
Mining Haul Truck (100 t)	26	12	-	7	3	3	-	-	1	-	-	-	-	-
Track Dozer (436 HP)	5	2	1	-	-	-	-	-	-	-	2	-	-	-
Track Dozer (215 HP)	4	2	-	-	-	-	-	2	-	-	-	-	-	-
Motor Grader (16 ft)	4	1	-	1	-	-	-	-	1	-	1	-	-	-
Motor Grader (14 ft)	1	-	1	-	-	-	-	-	-	-	-	-	-	-
Water Truck (15 kL tank)	6	2	-	-	-	-	2	-	-	-	1	1	-	-

Table 16.13: Support Equipment Requirement Schedule

Support Equipment	Max	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Articulated Dump Truck (45 t)	2	2	2	2	2	2	2	2	2	2	2	2	1	-
Excavator (49 t)	2	1	2	2	2	2	2	2	2	2	2	2	2	1
Excavator (90 t)	1	-	-	-	-	1	-	-	-	1	1	1	1	-
Hydraulic Hammers for Excavator 49 t	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Wheel Loader 425 HP	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Truck Flatbed 5.5 t	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Skid Steer Loader	1	-	1	1	1	1	1	1	1	1	1	1	1	1
Vibratory Compactor	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Knuckle Boom Truck (10 t)	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Crane Rough Terrain 67 t	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Telehandler	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Forklift Diesel 2.5 t	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mechanic Service Truck	3	1	2	2	3	3	3	3	3	3	3	3	3	2
Welding Truck	1	-	-	1	1	1	1	1	1	1	1	1	1	-
Tire Handler Loader	1	-	-	-	1	1	1	1	1	1	1	1	1	1
Tire Handler Attachment	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tire Handler Tooling & Equipment	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fuel Truck	1	-	-	-	1	1	1	1	1	1	1	1	1	-
Fuel & Lube Truck 10 Wheel	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Truck Tractor for Trailers	1	-	-	1	1	1	1	1	1	1	1	1	1	1
Trailer Lowboy	1	-	-	1	1	1	1	1	1	1	1	1	1	1
Multipurpose Truck - Platform/WT/etc.	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Pick-up Truck	12	12	12	12	12	12	12	12	12	12	12	12	12	6
Troop Carrier	5	5	5	5	5	5	5	5	5	5	5	5	5	1

Table 16.14: Support Equipment Purchase Schedule

Support Equipment	Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Articulated Dump Truck (45 t)	3	2	-	-	-	-	-	-	-	1	-	-	-	-
Excavator (49 t)	4	2	-	-	-	-	-	-	-	1	1	-	-	-
Excavator (90 t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Hydraulic Hammers for Excavator 49 t	5	1	-	1	-	1	-	-	1	-	-	1	-	-
Wheel Loader 425 HP	1	1	-	-	-	-	-	-	-	-	-	-	-	-
Truck Flatbed 5.5 t	3	1	-	-	-	1	-	-	-	1	-	-	-	-
Skid Steer Loader	2	-	1	-	-	-	-	-	-	-	1	-	-	-
Vibratory Compactor	2	1	-	-	-	-	-	-	-	1	-	-	-	-
Knuckle Boom Truck (10 t)	2	1	-	-	-	-	-	-	-	-	1	-	-	-
Crane Rough Terrain 67 t	1	1	-	-	-	-	-	-	-	-	-	-	-	-
Telehandler	2	1	-	-	-	-	-	1	-	-	-	-	-	-
Forklift Diesel 2.5 t	2	1	-	-	-	-	-	-	-	1	-	-	-	-
Mechanic Service Truck	5	2	-	-	1	-	-	-	-	-	2	-	-	-
Welding Truck	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tire Handler Loader	1	-	-	1	-	-	-	-	-	-	-	-	-	-
Tire Handler Attachment	1	1	-	-	-	-	-	-	-	-	-	-	-	-
Tire Handler Tooling & Equipment	1	1	-	-	-	-	-	-	-	-	-	-	-	-
Fuel Truck	2	-	-	1	-	-	-	-	-	1	-	-	-	-
Fuel & Lube Truck 10 Wheel	2	1	-	-	-	-	-	-	-	1	-	-	-	-
Truck Tractor for Trailers	1	-	-	1	-	-	-	-	-	-	-	-	-	-
Trailer Lowboy	1	-	-	1	-	-	-	-	-	-	-	-	-	-
Multipurpose Truck - Platform/WT/etc.	2	1	-	-	-	-	1	-	-	-	-	-	-	-
Pick-up Truck	24	12	-	-	-	-	-	-	12	-	-	-	-	-
Troop Carrier	10	3	-	2	-	-	-	-	-	-	5	-	-	-

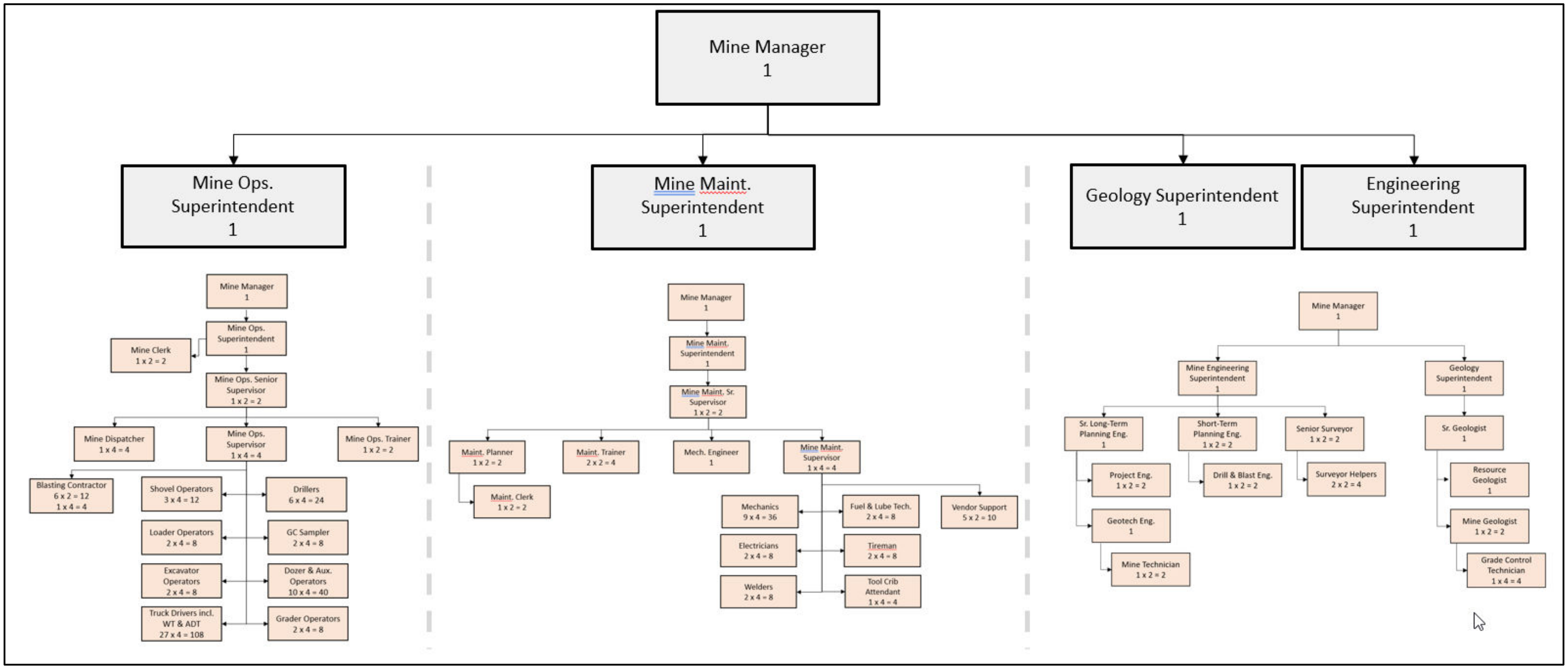
16.9 Mine Manpower Requirements

Mine personnel were divided into hourly and staff positions and were divided between mine operations, mine maintenance, mine engineering and geology. Hourly positions were all associated with a shift roster of 14 days on and 14 days off and as such each unit of equipment requires four (4) operators hired in hourly positions.

Staff positions in management, supervision or technical services roles will also be on roster schedule. In some case where 24-hour support in the staff role was necessary, the staff position was planned to be on the same 14 days on / 14 days off schedule as the hourly staff.

Table 16.15 shows the estimated mine workforce requirements over the life of mine. The mine workforce peaks at 352 individuals in Y7. Figure 16.29 shows the organizational chart of the mine departments.

Figure 16.29: Mining Department Organizational Chart



Source: GMS, 2021

Table 16.15: Workforce Forecast

Mine Operations	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Mine Manager	1	1	1	1	1	1	1	1	1	1	1	1	0	0
Mine Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Ops. General Foreman	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Supervisor	4	4	4	4	4	4	4	4	4	4	4	4	0	0
Clerk	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Trainer	2	2	2	2	2	2	2	2	2	2	2	0	0	0
Dispatcher	4	4	4	4	4	4	4	4	4	4	4	4	0	0
Shovel/Excavator Operator	6	12	12	16	20	16	16	16	20	16	12	12	4	0
Loader Operator	0	0	4	4	4	4	4	4	4	4	4	4	4	0
Small Excavator Operator	2	8	8	8	8	8	8	8	8	8	8	8	4	0
Haul Truck Operator	18	44	60	84	104	104	104	108	108	108	76	72	12	0
Drill Operator	4	6	8	16	16	20	16	16	16	16	12	12	0	0
Ancillary Equipment Operator	6	8	8	8	8	8	8	8	8	8	8	8	8	0
Dozer Operator	8	20	20	20	20	20	20	20	20	20	20	12	4	0
Grader Operator	2	4	8	8	8	8	8	8	8	8	8	4	0	0
Mine Maintenance	**Y-2**	**Y-1**	**Y1**	**Y2**	**Y3**	**Y4**	**Y5**	**Y6**	**Y7**	**Y8**	**Y9**	**Y10**	**Y11**	**Y12**
Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	0	0
General Foreman	1	2	2	2	2	2	2	2	2	2	2	2	1	0
Supervisor	0	4	4	4	4	4	4	4	4	4	4	4	1	0
Planner	1	2	2	2	2	2	2	2	2	2	2	2	1	0
Trainer	0	4	4	4	4	4	4	4	4	4	4	4	0	0
Mechanical Engineer	0	1	1	1	1	1	1	1	1	1	1	1	0	0

Mine Operations	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Clerk	1	2	2	2	2	2	2	2	2	2	2	2	1	0
Mobile Mechanic	8	16	24	36	36	36	38	38	38	36	28	28	10	0
Electrician	2	4	6	8	8	8	9	9	9	8	6	6	3	0
Mine Maintenance	**Y-2**	**Y-1**	**Y1**	**Y2**	**Y3**	**Y4**	**Y5**	**Y6**	**Y7**	**Y8**	**Y9**	**Y10**	**Y11**	**Y12**
Welder / Machinist	2	4	6	8	8	8	9	9	9	8	6	6	3	0
Fuel & Lube Technician	2	2	2	2	2	2	2	2	2	2	2	2	2	0
Tireman	2	4	8	8	8	8	8	8	8	8	8	2	2	0
Toolcrib Attendant	2	4	4	4	4	4	4	4	4	4	4	2	2	0
Helper	2	4	8	8	8	8	8	8	8	8	8	2	2	0
Engineering	**Y-2**	**Y-1**	**Y1**	**Y2**	**Y3**	**Y4**	**Y5**	**Y6**	**Y7**	**Y8**	**Y9**	**Y10**	**Y11**	**Y12**
Engineering Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	0	0
Long-Term Planning Engineer	1	1	1	1	1	1	1	1	1	1	1	0	0	0
Short-Term Planning Engineer	1	2	2	2	2	2	2	2	2	2	2	2	0	0
Drill & Blast Engineer	1	2	2	2	2	2	2	2	2	2	2	2	0	0
Mine Engineer	0	2	2	2	2	2	2	2	2	2	2	2	0	0
Geotechnical Engineer	0	1	1	1	1	1	1	1	1	1	1	0	0	0
Geotechnical Technician	2	2	2	2	2	2	2	2	2	2	2	2	0	0
Senior Surveyor	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Surveyor - Helper	2	4	4	4	4	4	4	4	4	4	4	4	1	1
Geology	**Y-2**	**Y-1**	**Y1**	**Y2**	**Y3**	**Y4**	**Y5**	**Y6**	**Y7**	**Y8**	**Y9**	**Y10**	**Y11**	**Y12**
Geology Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	0	0
Senior Geologist	0	0	0	0	1	1	1	1	1	1	1	1	0	0
Resource Geologist	0	1	1	1	1	1	1	1	1	1	1	1	0	0

Mine Operations	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Junior Geologist	1	2	2	2	2	2	2	2	2	2	2	2	0	0
Grade Control Laborers / Samplers	2	2	2	4	4	4	4	4	4	4	4	4	0	0
Contractors	**Y-2**	**Y-1**	**Y1**	**Y2**	**Y3**	**Y4**	**Y5**	**Y6**	**Y7**	**Y8**	**Y9**	**Y10**	**Y11**	**Y12**
Blasting Services	6	7	13	19	19	19	19	19	19	19	19	19	0	0
Vendor Mine Equipment	1	2	2	3	3	1	1	1	1	1	1	1	1	0
Vendor Drills	1	2	3	4	4	4	4	4	4	4	4	4	0	0

17 RECOVERY METHODS

17.1 Introduction

The process plant design for the Tocantinzinho Project is based on a robust metallurgical flowsheet to treat gold bearing ore 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 treat 4.34 Mtpy of granite ore and will consist of comminution, gravity concentration, gold flotation, cyanide leach and adsorption of the gold concentrate via carbon-in-leach (CIL), carbon elution and gold recovery circuits. CIL tailings will be treated in a cyanide destruction circuit and dewatered to produce a tailings slurry for storage onsite. The process plant feed will consist of run of mine (ROM) granite ore, along with minor amounts of saprolite and garimpeiros tailings. Figure 17.1 presents the overall flowsheet for the Tocantinzinho Project. Figure 17.2 and Figure 17.3 present the overall layout for the Tocantinzinho process plant.

The key project design criteria for the process plant are listed below:

- Nominal throughput of 4.34 Mtpy of granite ore and up to 4.70 Mtpy when saprolite and garimpeiros tailings are available in the blend, which is limited to 1,000 t/d.
- Crushing plant availability of 75%.
- Grinding, gravity, flotation, CIL, gold recovery and tailings handling circuit availability of 92% through the use of standby equipment in critical areas, inline crushed ore stockpile and reliable power supply.
- Comminution circuit to produce a particle size of 80% passing (P80) 125 µm.
- Gold flotation circuit with an average mass pull of 4.5%.
- CIL residence time of 36 hours to achieve optimal gold extraction.
- Cyanide destruction circuit to produce weak acid dissociable (WAD) cyanide levels of less than 1 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

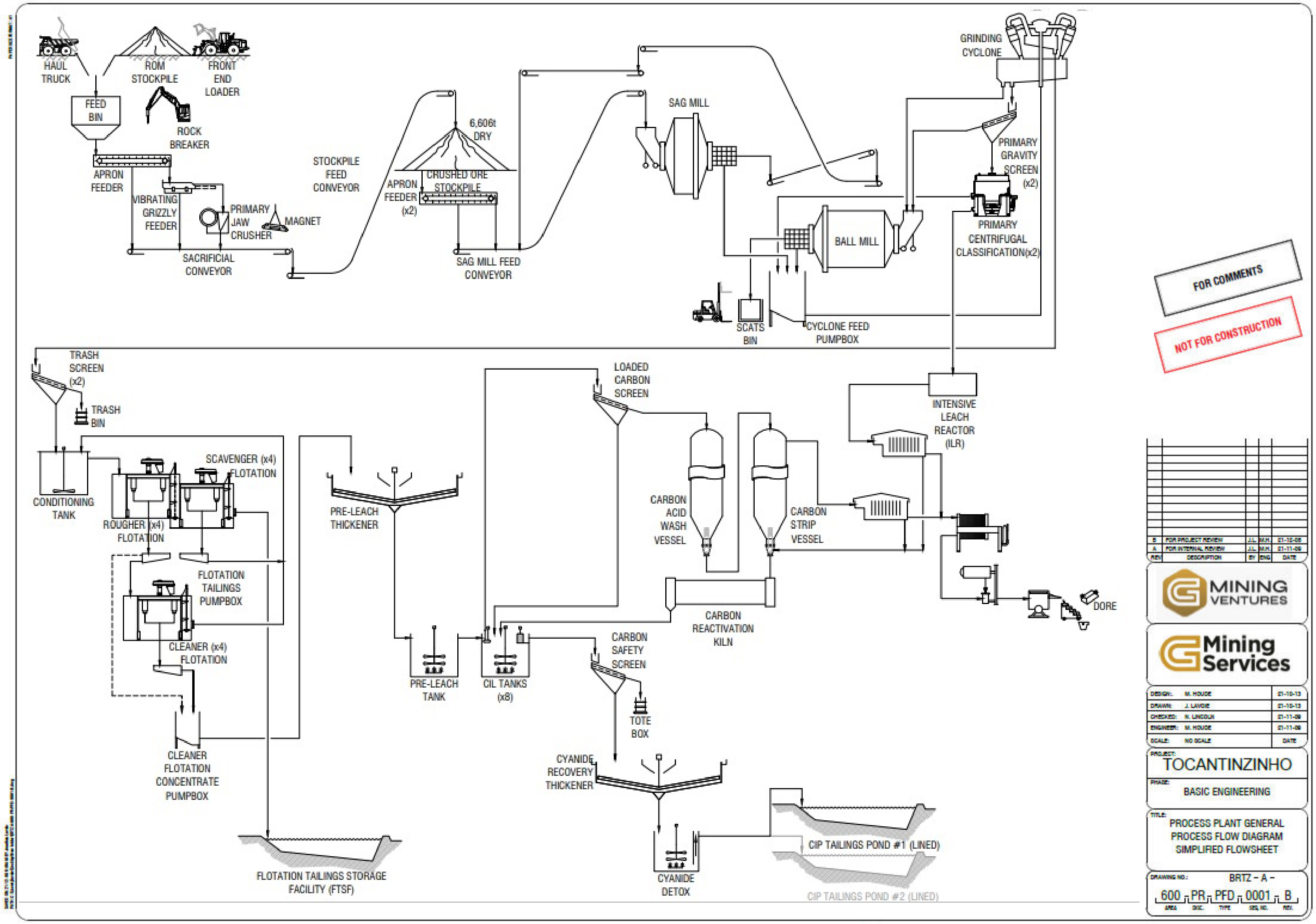


Figure 17.2: Processing Facility Overall Layout

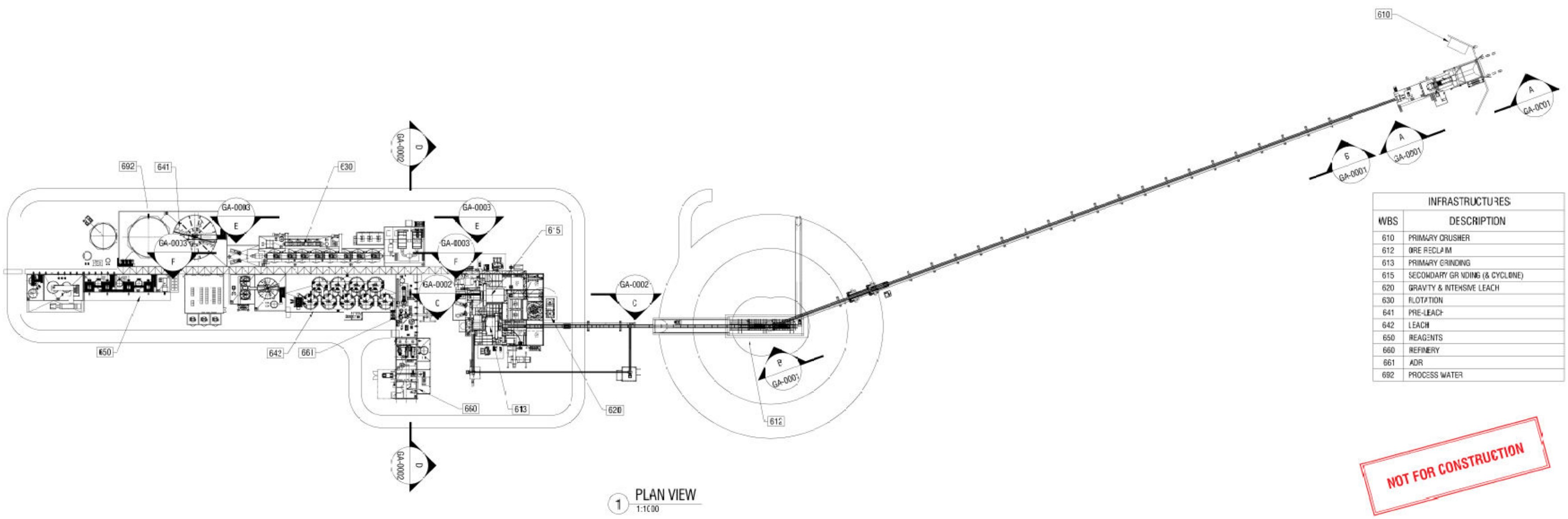
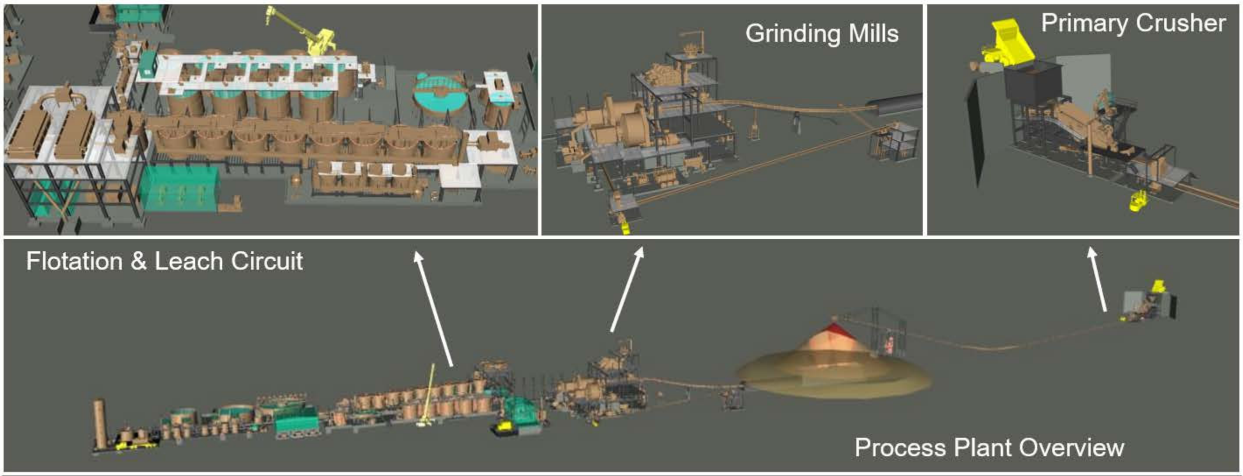


Figure 17.3: Process Plant Overall Layout



Source: GMS, 2021

17.2 Process Design Criteria

The proposed process plant will consist of the following unit operations:

- Primary crushing of ROM.
- Coarse ore stockpile and reclaim.
- Grinding consisting of semi-autogenous (SAG) mill and ball mill with hydrocyclones producing a final product P80 of 125 µm.
- Gravity concentration to produce a gold-rich concentrate for intensive leaching and subsequent gold recovery via electrowinning.
- Two-stage flotation circuit to produce sulphide concentrate for cyanide leaching.
- Pre-leach, cyanide leaching, and carbon adsorption via a Carbon-in-Leach (CIL) circuit.
- Carbon elution via a Pressure Zadra circuit.
- Carbon handling and regeneration.
- Electrowinning and smelting to produce doré.
- Cyanide destruction of CIL tailings using SO_2 / air process.
- Tailings – Flotation tailings and concentrate cyanidation tailings (i.e. CIL tailings) are stored in separate tailings storage facilities.
- Air and oxygen circuits.
- Water systems (potable water, raw 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 (granite ore)	t/y	4,340,000
	Nominal Daily Throughput	t/d	11,890
	Crusher Plant Availability	%	75
	Process Plant Availability	%	92
	Average Gold Head Grade	g/t	1.42
	Average Granite Gold Recovery	%	91%
	Average Saprolite Gold Recovery	%	71%
	Average Tailings Retreatment Recovery	%	85%
Crushing & Storage	Crusher Work Index	kWh/t	15.4
	Run of Mine (ROM), Maximum Size	mm	750
	Crusher Circuit Product Size (P80)	mm	123
	Stockpile Capacity (live)	h	12
Grinding	SMC A x b	-	51.5
	Bond Ball Mill Work Index (85^{th} percentile)	kWh/t	18.2
	Grinding Circuit Product Size (P80)	µm	125
Gravity Concentration	Type	-	2 x KC-QS48
	Intensive Leach Reactor	-	CS3000
Flotation	Rougher Flotation Design Retention Time	min	16
	Rougher-Scavenger Flotation Design Retention Time	min	16
	Cleaner Flotation Design Retention Time	min	21
	Overall Flotation Mass Pull (granite)	%	4.5
Pre-Leach Thickening	Thickener Underflow Density	%w/w	65
	Solids Loading	t/m^3h	0.12
CIL	Pre-Leach Residence Time	h	5
	CIL Tanks	-	8
	Overall CIL Residence Time	h	36
	Circuit Density	%w/w	45

Area	Criteria	Unit	Nominal Value
ADR	Elution Batch Size (Carbon)	t	3
Cyanide Destruction	Cyanide Destruction Technology	-	SO_2 / air
	Number of Stages	-	2
	Total Retention Time	min	186

17.3 Process Plant Description

17.3.1 Primary Crushing

Ore from the open pit, at an estimated maximum lump size of 750 mm, will be transported to the plant by 92-t capacity rear dump trucks. The trucks will tip directly to the ROM bin. However, if the trucks are not permitted to directly tip into the ROM bin, then the truck load will be dumped onto the ROM pad. The ROM pad will be primarily utilized for emergency storage and blending with saprolite and garimpeiros tailings when required by the mine plan. ROM material will be reclaimed to the ROM bin by a front-end loader.

Ore will be withdrawn from the ROM bin, by a variable-speed apron feeder, directly to a vibrating grizzly (~100 mm aperture size). Oversize from the grizzly will report directly to the jaw crusher, which will operate in open circuit. A rock breaker and a grapper will be installed to assist in breaking down oversize material retained above the jaw crusher. Crushed ore from the crusher discharges, together with undersize from the grizzly will be withdrawn by a sacrificial conveyor. A belt magnet at the sacrificial conveyor discharge will recover any trash metal. The sacrificial conveyor will feed crushed material to the stockpile feed conveyor to the crushed 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.

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 Crushed Ore Stockpile

The crushed material will be conveyed to the crushed material stockpile. The stockpile will have a live capacity of approximately 6,470 t (equivalent to 12 hours of mill feed). Two reclaim apron feeders located underneath the stockpile will be installed with variable speed drives (VSDs) to control the reclaim rate feeding the grinding circuit. A single apron feeder will be capable of providing the total plant throughput of 539 t/h and will feed the ore to the SAG mill feed conveyor.

17.3.3 Grinding

Reclaimed ore from the crushed ore stockpile will feed an 8.5 m diameter by 5.0 m effective grinding length (EGL) SAG mill via the SAG mill feed conveyor. The SAG mill will be installed with a 6,500-kW synchronous motor 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 70% slurry discharge density. SAG mill discharge will pass through a trommel screen to remove grinding media scats and a small amount of pebbles. The SAG trommel undersize will report to the cyclone feed pump box, combining with ball mill discharge. The SAG mill discharge will have an average transfer size (T80) of 600 µm. SAG trommel oversize will be conveyed to the SAG mill feed conveyor.

Slurry from the cyclone feed pump box will be pumped to a cluster of 15 (12 operating / 3 standby) 508 mm hydrocyclones for size classification. The cyclone overflow, at a final target product P80 of 125 µm, will flow via gravity to the rougher flotation conditioning tank prior to sulphide flotation. The hydrocyclones have been designed for a 275% circulating load.

Cyclone underflow will feed a 6.4 m diameter by 9.15 m EGL ball mill with an installed 6,500 kW fixed speed motor. 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 underflow will feed a gravity separation circuit for coarse gold recovery.

17.3.4 Gravity Gold Recovery

The gravity recovery will consist of two (2) centrifugal gravity concentrator units equipped with a feed screen and an intensive cyanidation unit. The circuit has been designed to treat 40% of cyclone underflow. The gravity gold recovery unit will be located in a secured area within the mill structure.

Cyclone underflow will be screened using a vibrating screen to remove +2 mm material. The oversized material will overflow directly to the ball mill feed. 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 pre-leach thickener.

17.3.5 Flotation

The cyclone overflow will flow by gravity into the rougher flotation conditioning tank. The rougher flotation conditioning tank will provide 5 minutes conditioning time for flotation chemicals including copper sulphate (activator) and sodium iso-butyl xanthate (SIBX) (collector). Lime will be added into the tank to adjust pH, if required.

The rougher and scavenger flotation circuit will consist of a single bank of eight (8) 160 m^3-mechanical tank-cells; four (4) cells for the rougher circuit and four (4) cells for the scavenger circuit. The conditioning tank overflow, scavenger concentrate and cleaner tailings will feed the rougher flotation circuit. The rougher concentrate will flow by gravity to the cleaner flotation circuit for further upgrading.

The scavenger concentrate will flow into the cleaner tailings tank and will be pumped back to the rougher feed box. The scavenger flotation tailings will be pumped to the flotation tailings storage facility.

The cleaner flotation will be a single bank of four (4) 30 m^3 tank-cells. Cleaner flotation concentrate will be pumped to the pre-leach thickener and cleaner flotation tailings will be pumped back to the rougher flotation feed box.

17.3.6 Pre-Leach Thickening and CIL

Cleaner flotation concentrate will be pumped to a 21 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.

The thickener underflow at 65% w/w solids will be pumped to the CIL circuit consisting of one (1) 270 m^3 pre-leach tank and eight (8) 270 m^3 CIL tanks. The CIL tanks will provide a total retention time of 36 hours and the tanks will be sparged with oxygen. The CIL tanks will be equipped with inter-stage screens and pumps to advance the loaded carbon upwards to the next CIL tank. Activated carbon will be added into the

CIL tanks 7 and 8 and loaded carbon will leave the CIL circuit from the first and second CIL tanks. Activated carbon concentrations will vary between 10 and 20 g/L slurry within the CIL tanks.

Sodium cyanide will be added to the CIL circuit to dissolve the gold and lime slurry will be added to maintain the slurry pH of approximately 11.0 - 11.6.

The loaded carbon will be transferred to the carbon stripping circuit, while the leach residue from the last tank will be sent to a carbon safety screen to recover any carbon fines. The screen undersize will be pumped to the cyanide recovery thickener.

17.3.7 Cyanide Recovery and Detoxification

The 10 m diameter cyanide recovery thickener will produce a thickened slurry of 50% solids which will be pumped to the cyanide destruction circuit. Thickener overflow water will be recycled back to the CIL circuit to recover any cyanide. Flocculant will be added to the thickener feed to promote the settling of solids.

The cyanide destruction circuit will consist of two (2) 104 m^3 mechanically agitated tanks, each with 120 minutes retention time; 240 minutes retention time total. The conventional SO_2 / air process will be used for cyanide destruction. Treated slurry will flow by gravity to the cyanide destruction tailings pump box for pumping to the CIP tailings pond.

The cyanide destruction circuit will treat thickened slurry from the thickener, process spills from various contained areas and process bleed streams: cold cyanide barren solution effluent, acid wash effluent, CIP tailings pond secondary treatment effluent and area sump pump discharge.

Oxygen will be sparged into the cyanide destruction tanks. Hydrated lime 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 solution as the source of SO_2. The process will reduce total cyanide in solution to 0.4 mg/L and WAD cyanide in solution to 0.2 mg/L. The total cyanide and WAD cyanide level in solutions will eventually drop below 0.16 and 0.15 mg/L after aging in the CIP pond.

17.3.8 Acid Wash and Elution

Loaded carbon from the CIL 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 the elution vessel using recycled carbon transfer water. Carbon slurry will discharge directly into the top of the elution vessel.

The carbon stripping (elution) cycle will utilize barren solution to strip gold rich carbon to create a pregnant solution. The strip column will hold 3.0 t of carbon and strip once a day. 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.

17.3.9 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. A 167 kg/h diesel fired kiln will be utilized to treat 3.0 t of carbon per day, equivalent to 100% regeneration of carbon. The regeneration kiln discharge 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 CIL 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.

17.3.10 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 pump box where it will be pumped to the barren solution tank.

The primary flow from the barren solution pump box 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 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 pump box 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.

A Jerome meter will be used by Operators to monitor any possible mercury generated from the off-gas systems that may originate from the garimpeiro tailings in the plant feed.

17.3.11 Tailings Storage Ponds

For the total life of mine (LOM), there will be three (3) tailings ponds: the flotation tailings pond, CIL tailings Pond #1 and CIL tailings Pond #2.

The flotation tailings pond will receive tailings from the flotation circuit as well as the mine dewatering flow. Tailings pond supernatant (reclaim water) will be pumped back to the process water tank using vertical pumps on a barge.

The CIL tailings ponds will receive tailings from the cyanide destruction circuit. The CIL tailings pond will be lined and monitored to prevent any effluent from entering the environment. The CIL tailings will provide natural cyanide degradation to further decrease total cyanide prior to the release of any effluent into the environment.

The CIL tailings ponds will be designed to hold settled CIL tailings for the LOM. Supernatant will be reclaimed for use in the cyanide destruction circuit to reduce the slurry density. Excess water from the CIL tailings ponds in subsequent operating years will flow into the environment after further treatment (refer to Section 18).

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, hydrated lime, hydrochloric acid, sulphuric acid, sodium hydroxide, copper sulphate, sodium iso-butyl xanthate, frother sodium metabisulphite, antiscalant, flocculant, 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, flotation, thickeners, CIL, 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 day of production. The reagents will be delivered in dry form, except for hydrochloric 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 CIL circuit.

17.4.2 Hydrated Lime

Hydrated lime will be used as a pH modifier and will be supplied in dry form by road in bulk tankers and off-loaded into the storage silo using a blower. The storage silo will hold up to four (4) days consumption. Hydrated lime will be added into a mix tank to prepare a milk of lime slurry before addition into the process.

17.4.3 Copper Sulphate

Copper sulphate ($CuSO_4$) will be an activator for sulphide flotation and will also be used as a catalyst for cyanide destruction. The copper sulphate will be supplied as a dry flake in one (1)-tonne bulk bags and stored in the reagents 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 (1)-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 Iso-Butyl Xanthate

Sodium iso-butyl xanthate (SIBX) will be used as a sulphide mineral collector in the flotation circuit and will be supplied in 850 kg bulk bags as a dry reagent. SIBX will be shipped by road to site, offloaded by forklift and stored in the reagents storage area adjacent to the SIBX & frother mixing facility. The SIBX will be mixed with fresh water to form a SIBX solution prior to addition to the processing facility.

17.4.6 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 (1)-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.

17.4.7 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 drums and stored in the reagent storage area adjacent to the reagent mixing facility.

17.4.8 Flocculant

Flocculant is a liquid polymer that will be used in both thickeners to settle solids. It will be supplied in 25 kg bulk bags as a dry reagent. Flocculant will be shipped by road to site, offloaded by forklift, and stored in the reagents 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.

17.4.9 Frother

DF250 will be used as a frother to mechanically sustain bubbles in the flotation cells. The frother will be supplied in 1,000 L IBC, offloaded by forklift, and stored in the reagents storage area adjacent to the SIBX & frother mixing facility. IBCs of frother will be unloaded into the frother storage tank by drum pump prior to being added into the flotation circuit.

17.5 Plant Services

17.5.1 Blower Air

Blowers will supply low pressure process air to the flotation cells. Three (3) blowers (2 duty, 1 standby) will be installed to meet flotation air requirements.

17.5.2 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.

The air compressors can bypass the oxygen generation circuit and feed the leaching circuit directly, if required.

17.5.3 Oxygen Generation

An oxygen generation plant will be used to provide industrial grade oxygen for the CIL 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. The oxygen generation system will produce up to 5 tonnes of 93% purity oxygen per day.

17.5.4 Fresh and Fire Water

Fresh water, also known as domestic water, will be sourced under permit from Veados Creek. Fresh water from the creek will be pumped to the plant fresh / fire water tank by vertical turbine pumps. The tank will be located on a hill 60 m above the plant site and will use gravity to supply fresh water to all required users.

The plant fresh / fire water tank will serve as a combined storage for both fresh and fire water supply. Fresh water will draw from part way up the tank while the lower section of the tank is held in reserve for a dedicated fire water supply.

The fire water portion of the tank will have minimum capacity of 108 m^3 and will feed the plant and permanent campfire suppression systems fire hydrants and hose reels via a fire water ring main. Fresh water in the tank will be used to supply the following services:

- Primary crushing circuit dust suppression water
- 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

Fresh water will be pumped to the fresh / fire water tank through multimedia filter to remove particulates.

17.5.5 Potable Water

Feed to the potable water system is supplied from wells using vertical well pumps. The water will be treated in a vendor-supplied potable water plant to produce potable water for the process plant and camp facilities distribution. The potable water will be used in the process plant for safety showers and washrooms.

17.5.6 Gland Seal Water

Water for the gland seal water system will be supplied by fresh water from the fresh / fire water tank and cooling water returning from the elution circuit cooling heat exchanger. 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 with no cyanide will be comprised of pre-leach thickener overflow, flotation tailings pond reclaim water, and fresh makeup water when required. Process water will be stored in the process water storage tank and distributed by the process water pumps, in a duty – standby configuration, to non-cyanide consumption points in grinding and flotation.

Process water with cyanide from the cyanide recovery thickener will be recycled back to the CIL circuits for density control and to recycle cyanide in solution.

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
- Flotation tailings
- CIL 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 plant, 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 Water

A preliminary water balance for the process plant was completed. Approximately 70 m^3/h of raw water is required for makeup water.

17.8.2 Energy

The power demand for the process plant, along with the rest of the project, will be provided by grid power. The average power demand for the process plant at 19.885 MW is summarized in Table 17.2 when processing granite ore only. The average power demand when processing softer ores decreases to 15.053 MW. The average power demand does not reflect the instantaneous power demand for equipment start-up and power plant capacity sizing.

Table 17.2: Power Requirements

Area	Average Power (kW)	Annual Power Consumption (MWh)
Comminution	11,946	104,644
Gravity Concentration	152	1,332
Flotation	2,999	26,271
Leach / CIL / Cyanide Destruction	355	3,110
Reagents	109	955
Elution / Gold Room	662	5,799
Tailings Handling	405	3,548
Water / Air / Oxygen Services	609	5,335
Balance of Plant	2,648	23,196
Total	**19,885**	**174,190**

Source: GMS, 2021

17.8.3 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 17.3 and Table 17.4.

Table 17.3: Reagents Consumption

Description	Delivered Form	Daily Usage
Sodium Cyanide	Briquette ISO tank	3.4 t/d
Lime (@90% CaO)	Bulk (dry)	0.73 t/d
Hydrochloric Acid (32% strength)	1,000 L IBC	345 l/d
Sodium Hydroxide	1 t bags (dry)	0.19 t/d
Copper Sulphate	1 t bags (dry)	1.3 t/d
SMBS	1 t bags (dry)	2.4 t/d
Frother	1,000 L IBC	0.71 L/d
Collector (SIBX)	850 kg bags (dry)	1.2 t/d
Flocculant	25 kg bags (dry)	45 kg/d
Activated Carbon	500 kg bags (dry)	12 g/d

Source: GMS, 2021

Table 17.4: Consumables Consumption

Description	Delivered Form	Usage
Primary Jaw Crusher - Fixed Jaw	lot	13.1 sets / year
Primary Jaw Crusher - Movable Jaw	lot	9.2 set / year
Primary Jaw Crusher - Upper Check Plate	lot	3.3 set / year
Primary Jaw Crusher - Lower Check Plate	lot	6.6 sets / year
SAG Mill Liners	lot	0.47 sets / year
Ball Mill Liners	lot	0.94 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
Loaded Carbon Screen panels	lot	1.3 sets / year
Barren Carbon Screen panels	lot	1.3 sets / year
Carbon Safety Screen panels	lot	2.0 sets / year
Interstage Screens panels	lot	2.0 sets / year
SAG Mill Grinding Media (5 in)	bulk	0.47 kg/t
Ball Mill Grinding Media (3 in)	bulk	0.70 kg/t

Source: GMS, 2021

17.9 Process Plant Personnel

The personnel for the process plant will consist of management, operations, maintenance, and laboratory. Operating staff will work 12-hour days and night shifts on a 2-week on-2-week off rotation cycle and management will work 12-hour days.

Annual process plant personnel requirements are provided in Table 17.5.

Table 17.5: Process Plant Personnel

Department	Position	Compliment
Plant Management	Plant Manager	1
	Senior Metallurgist	1
	Plant Metallurgist	1

Department	Position	Compliment
	Metallurgical Technician	4
	Mill Clerk	2
Plant Operations	Plant Superintendent	1
	General Foreman	1
	Shift Foreman	4
	Control Room Operators	4
	Crushing Operators	4
	Milling Operators	4
	Flotation Operators	4
	Leaching / CIL Operators	4
	Detox / Utilities Operators	4
	Reagent Operators	4
	Gold Room Technicians	4
	Mobile Equip. Operators	4
	General Helpers	8
Plant maintenance	Plant Maintenance Superintendent	1
	Maintenance Planner	1
	Maintenance Inspectors	2
	Maintenance Trainer	1
	Mechanical Supervisors	2
	Electrical Supervisor	2
	Millwrights	4
	Welders	4
	Pipe Fitters	2
	Trade Assistants	4
	Shift Mechanics	4
	Electricians	4
	Instrument Technicians	4
	Maintenance General Helpers	4
Laboratory	Chief Laboratory Chemists	2

Department	Position	Compliment
	Senior Chemists	4
	Laboratory Technicians	8
	Labourers	4
Total		**116**

Source: GMS, 2021

17.10 RECOMMENDATIONS

The following is recommended related to the process plant:

- Finalize SAG and ball mill sizing once further comminution test work has been completed.
- Finalize gravity, flotation, CIL and cyanide destruction circuit sizing once further metallurgical test work has been completed.
- Once further metallurgical test work has been completed, evaluate the need for a concentrate regrind circuit to potentially increase gold recoveries. Currently, layout provision has been allowed for the future installation of a regrind mill.
- Optimize process plant reagent consumption once further metallurgical test work has been completed.

18 PROJECT INFRASTRUCTURE

18.1 General

The Project infrastructure is designed to support the operation of a 25 Mtpy (peak of 27.5 Mtpy) open pit mine and a nominal 4.34 Mtpy processing facility, operating on a 24-hour per day, 7-day per week basis. It is designed in consideration of local conditions and topography.

18.2 Site Layout

The overall Project site plan with the onsite infrastructure is presented in Figure 18.1. It has been designed to minimize environmental impacts, provide security-controlled site access, minimize construction costs, and optimize operational efficiency. Primary buildings have been located to allow easy access for construction and to utilize existing topography to minimize bulk earthworks volumes.

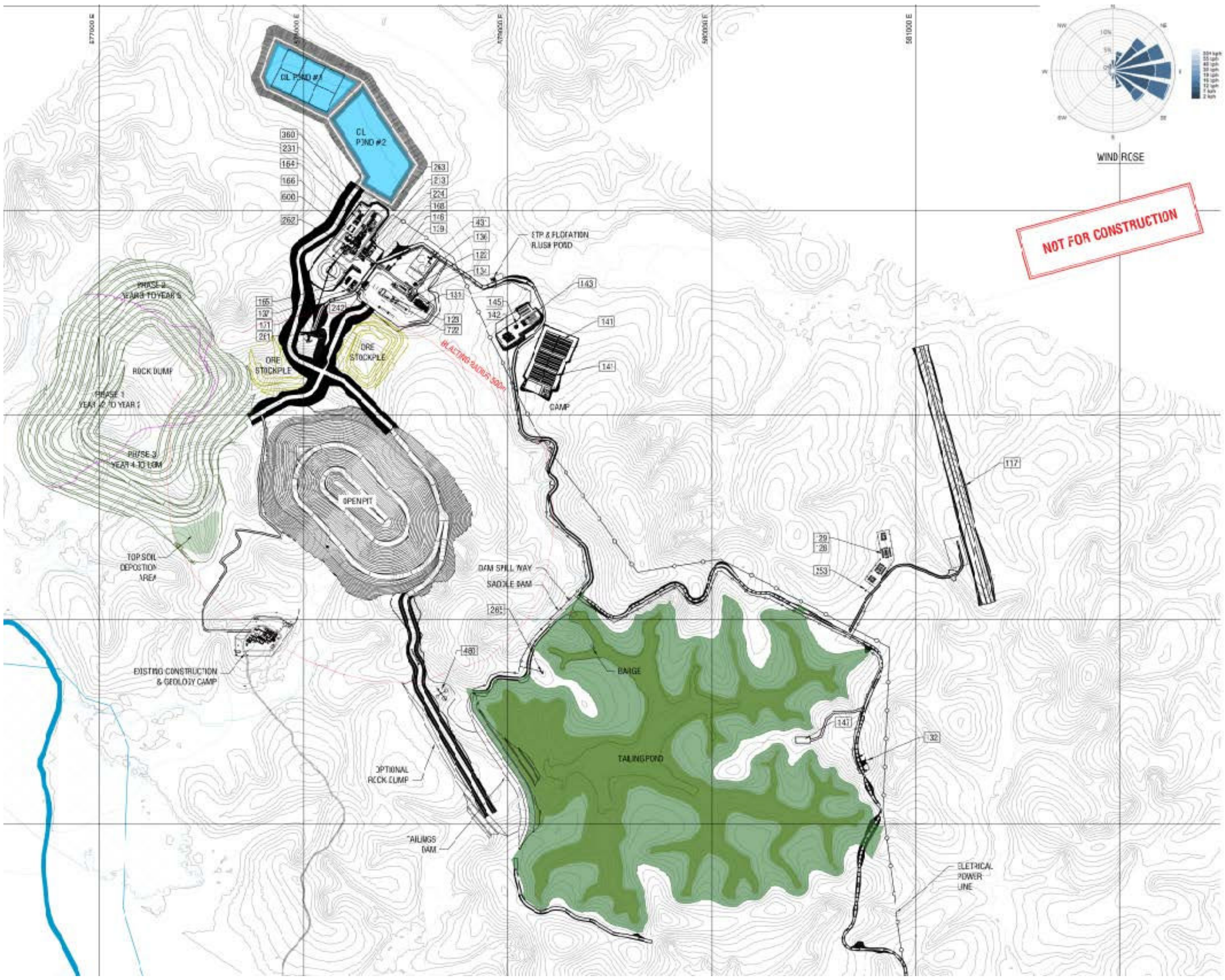
- Access
    - Road access is via an existing 108 km dirt road connecting the Project to the BR-163 Federal Highway in Moraes Almeida. Upgrades to certain segments are planned to improve gradients, surfacing and drainage. An existing radio communication system with four repeater towers allows for continuous communications along the access road.
    - A gate and guardhouse will be located at the property entrance along the access road.
    - Service roads of 14.8 km will connect the various infrastructures located on the property, notably the airstrip, explosives storage facility, tailings storage facilities, operations site and camp site.
    - Air access is via an existing 775 m long airstrip that will be used during construction. A new 1,300 m-long airstrip rated category 2, capable of landing larger planes is planned. This airstrip will be used for personnel, supplies, medical emergencies and exporting doré.
- Power supply and distribution
    - Primary power supply is via a 198 km long 138 kV transmission line from the switching station in Novo Progresso to the substation at the mine site near the process plant. Average power consumption is estimated at 20 MW with a peak requirement of 24 MW.
    - The main substation will consist of two 20 / 25MVA 138-13.8 kV transformers.
    - Site power distribution is planned with two 13.8 kV overhead lines.
    - Secondary power supply (i.e., emergency power) consists of 6.2 MW from four generators (3 x 1.8 MW and 1 x 0.8 MW).

- Process plant support buildings
  - Mill office (500 m^2) for plant staff and will be located within the fenced off area of the process plant. A process plant security building will be located at the mill entrance to access this secured area.
  - The reagent storage building (1,300 m^2) will have ample storage capacity sized according to supply chain considerations.
- Mine site
  - The mine includes a main open pit with two waste storage facilities to store a total of 163.4 Mt of material and an ore stockpile area to contain up to 8.9 Mt of ore.
  - Mine haul roads, 4.2 km in total, connect the open pit to the dumps, flotation tailings storage facility (FTSF), carbon-in-leach (CIL) ponds and mine support facilities. The open pit will be a source of waste fill material for various construction activities.
- Mine support infrastructures
  - Facilities are located south-east of process plant to allow for easy access for heavy equipment while a safe traffic is ensured by segregation of light and heavy traffic.
  - Permanent mine maintenance facility will have five heavy duty bays serviced by 30 / 5 t overhead crane with an additional six light duty bays for heavy and light vehicles. The maintenance facility will include office space for the maintenance staff, tool crib and lube storage.
  - Temporary maintenance facility (475 m^2) will be constructed at the onset will have four bays. This facility will be built with containers and a fabric top to be used during construction and subsequently repurposed as a welding bay.
  - Warehouse (815 m^2 area) close to the maintenance facility will be used to store parts and supplies.
  - Fuel storage will have a total capacity of 420 kL equivalent to 12 days of consumption.
  - 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.
  - Explosive storage facility is designed for a capacity of 160 t of emulsion using 40 t skid mounted tanks, 18 t of explosives products in a magazine with another magazine for accessories. Storage capacity is sufficient for 30 days at peak consumption.
  - Assay laboratory (784 m^2) is configured to process up to 350 samples per day for mine grade control, exploration and metallurgical samples.

- Administration and general infrastructure
  - Administration complex (1,700 m^2) for mine management, engineering, geology and G&A functions. Included in this complex is a clinic and security office to monitor all cameras on the property.
  - Communications will be provided by an existing network of interconnected telecom towers with the bandwidth to be increased with demand.
  - A greenhouse and nursery (200 m^2) will be established to cultivate plants for future revegetation activities.
  - A recycling and sorting facility will be set up to sort waste materials. Inert solid waste will be disposed in a landfill.
- Camp facilities
  - The temporary camp (i.e., for exploration and construction) will be expanded from the current 100 beds to 200 beds. This camp facility consists of soft-shell buildings and includes kitchen and dining hall with required services such as power, water and sewage. The temporary camp will serve during early works and peak requirements during construction.
  - The permanent camp facility will have a 1,200-person capacity during construction with three types of camp modules. The camp will easily service the approximate 800-person capacity during operations. The permanent camp facility includes kitchen and dining area for 320 people, recreation facilities, camp office and laundry facilities with associated water and sewage services.
- Tailings and water management
  - The FTSF for inert material will require one main dam. The starter dam has a height of 29 m and provides storage capacity for three years. Subsequent raises to the final dam height of 44 m will provide a storage capacity of 29.3 Mm^3. A total volume of 1.49 Mm^3 of fill material consisting of a saprolite core and compacted rockfill on the downstream slope is required to construct the main dam. A small saddle dyke will be required for the final facility.
  - A reclaim water barge will be installed in the FTSF to recycle water back to the process plant or for discharge to the environment at a rate of 401 m^3/h.
  - The CIL tailings will report to two ponds lined with geomembrane. The first pond will be built as part of the initial project and the second pond during the first year of operations.
  - Effluent treatment plant for excess water from the CIL ponds to be constructed during the first year of operations. This facility will have a capacity of 100 m^3/h principally to treat for dissolved copper by hydroxide precipitation via lime addition.

- A vertical pumping station in Veados Creek will provide up to 200 m^3/h of raw water. An industrial water tank will have a provision 108 m^3 capacity in the bottom for fire water storage. Water from Veados Creek will be treated for industrial consumption with further treatment for potable water.

Figure 18.1: Overall Site Layout



Mining Services	7900 W. Taschereau Blvd. Suite D-200 Brossard, Qc J4X 1C2	Title: Overall site General Arrangement	Author: Pascal Gauthier Date: 21/11/16

18.3 Mine

The open pit mine and waste rock dumps are outlined in Section 16. There will be two stockpiles for high grade saprolite and small-scale miners' (also know as garimpeiros) tailings. A dewatering system will be installed to pump pit water to the flotation tailings pond. The mine operations will be carried out by an owner-fleet of heavy equipment, for which several supporting facilities will be necessary. These are presented further in this section.

18.4 Process Plant

The recovery methods outlined in Section 17 govern the process plant installations. These will consist of a comminution system, gravity and intensive leach, flotation and concentrate leaching, carbon handling, refinery, cyanide destruction and tailings pumping. The process will be supported by systems for reagents and various services such as air, oxygen and water.

18.5 Water Infrastructure

18.5.1 Industrial / Fire Water

A vertical pumping station will be installed in Veados Creek which will supply raw water to be filtered and qualified as industrial water. Industrial water will be distributed to the site and process plant from a water tank beside the process plant. The lower portion of the tank will be dedicated to fire water. The fire water system will supply the building sprinkler systems and hydrants at the process plant and main camp complex.

18.5.2 Raw Water Treatment

Raw water from the Veados Creek will be treated at the industrial water treatment plant to remove suspended solids and perform pH adjustment, and disinfection. After treatment, this industrial water will be used at the camp, process plant and mining infrastructures.

18.5.3 Potable Water

Industrial water will be further treated into potable water standard by a packaged potable water treatment plant located at the kitchen. The potable water system will only supply the permanent camp kitchen and human consumption requirements. Potable water for facilities outside the main site will be provided by tanker truck or bottles.

18.5.4 Sewage Treatment and Oil Water Separation

A sewage treatment plant is planned to treat sewage from the plant and camp. Sewage water will be handled by standard septic tank collection systems and treated using natural breakdown bioreactors prior to discharge.

The sanitary waste from the process plant and the main camp site will drain directly to the sewage treatment plant located to the northwest of the camp area.

Sewage will be treated, separated, and the liquid discharged. Sludge from the sewage treatment plant will be disposed off-site.

Three oil water separation systems with be located adjacent to the location of contamination. Water will be collected from the truck shop and the diesel fueling station to be redirected to an oil water separation tank for processing and discharging clean water to the environment. At the mobile equipment wash bay, an oil skimmer will capture any oil before water is reused to clean equipment.

18.6 Support Infrastructure

18.6.1 Roads and Drainage

There are various existing exploration roads that are currently accessible by either light vehicle or ATV.

A network of gravel surfaced roads for light and heavy vehicles will be built. The heavy haul roads around the mine will provide large truck access to the maintenance facility, crusher, aggregate plant and tailing dam areas. All other areas of the site will be accessible by the light vehicle roads.

Ditching and culverts will divert water away from plant site infrastructures, process plant, mine services facilities and roads. Plant site pads are sloped away from buildings towards diversion ditching and directed to decant ponds.

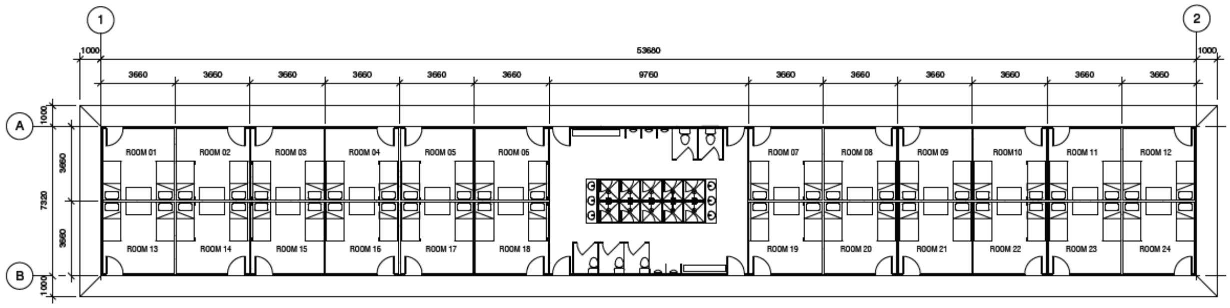
18.6.2 Camp Accommodations

There is an existing camp on site that will be upgraded to support the early works construction activities. The current camp consists of individual Weather Haven soft shell buildings that can comfortably accommodate about 100 people.

The existing camp will initially be expanded to 200 beds with additional infrastructure including new temporary kitchen, dining hall, laundry and dorms.

A 1,200-person permanent camp will be located within walking distance of the process plant. The occupancy is based on four people per room during construction. During operations, there will be a maximum of two people per room and the camp capacity will decrease to 800 people.

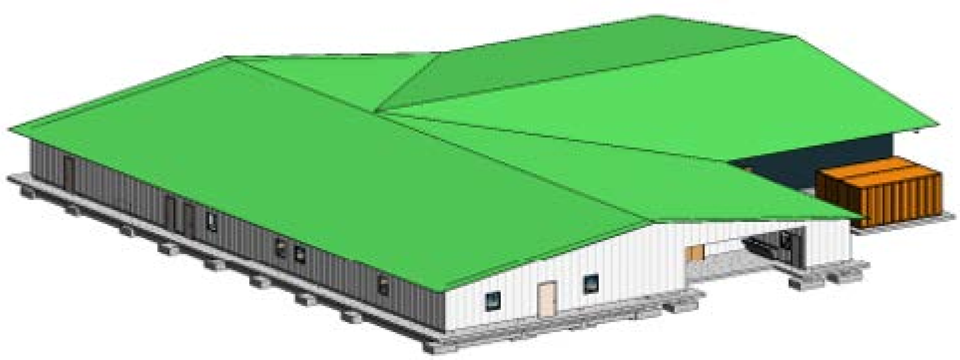
Figure 18.2: Typical Camp Dorm



Source: GMS (2021)

The main camp complex will include a canteen, kitchen (Figure 18.3), administration, recreation facilities, general store and potable water plant.

Figure 18.3: Kitchen



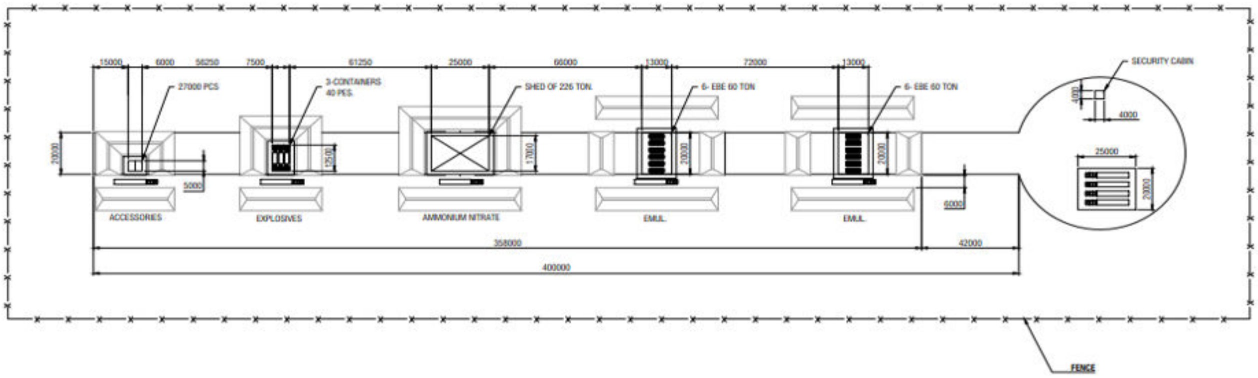
Source: GMS (2021)

18.6.3 Explosives Storage

The explosives facilities will be located near the new air strip. The area will be secured with fencing, electronic security and access will be controlled. The total area will be comprised of two separate locations each containing two 40-tonne skid-mounted emulsion tanks for a total capacity of 160 tonnes of emulsion, one location for an explosives' magazine with a capacity of 18 tonnes and one location with an accessory magazine. The four locations will be separated by distance and berms in accordance with explosives storage's local regulations.

The storage capacity is sufficient for approximately 15 days of emulsion and 30 days of accessories. Periodical deliveries will be organized based on operational requirements.

Figure 18.4: Explosives Storage



Source: GMS (2021)

18.6.4 Greenhouse and Nursery

A greenhouse and nursery will be located near the warehouse. Flora will be grown here for future reclamation activities.

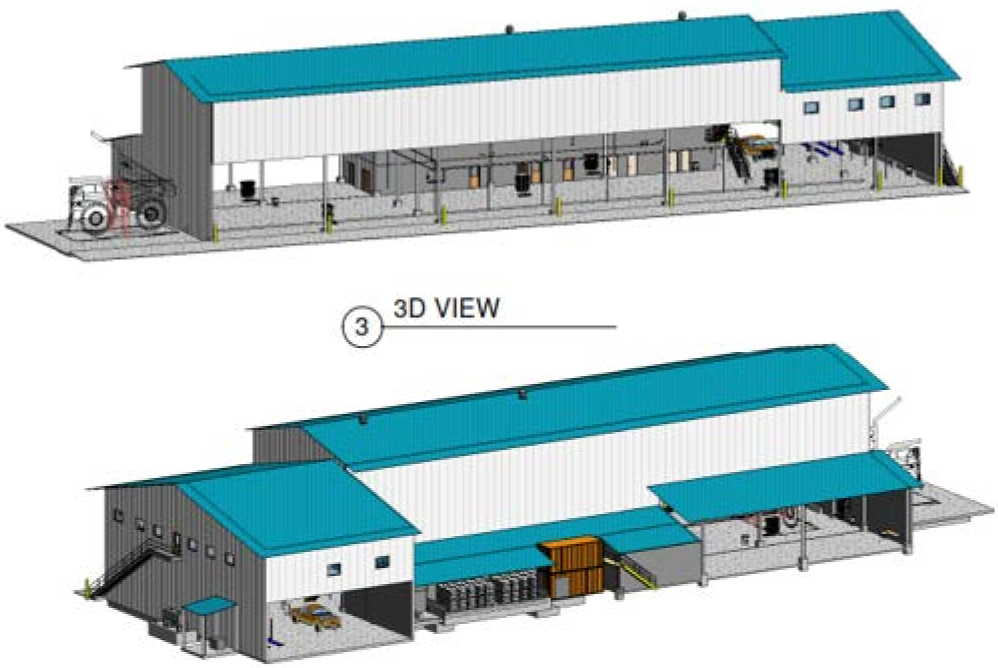
18.6.5 Site Buildings

Buildings will be equipped with smoke, carbon monoxide and heat detectors and appropriate chemical fire extinguishers and looped together to one main fire alarm panel. All infrastructure will comply with the national building codes (Associaçao Brasileira de Normas Técnicas ABNT).

18.6.5.1 Mine Maintenance Facility

The Mine maintenance facility will be comprised of two buildings, one a fabric building (also referred to as the temporary shop) and the main shop (Figure 18.5) which will include five heavy duty bays sized for Cat 777 trucks and six light duty bays. The construction of the main shop is a conventional steel structure with overhead crane. A masonry two story building is included to accommodate the tool crib and offices for 25 employees and all required facilities.

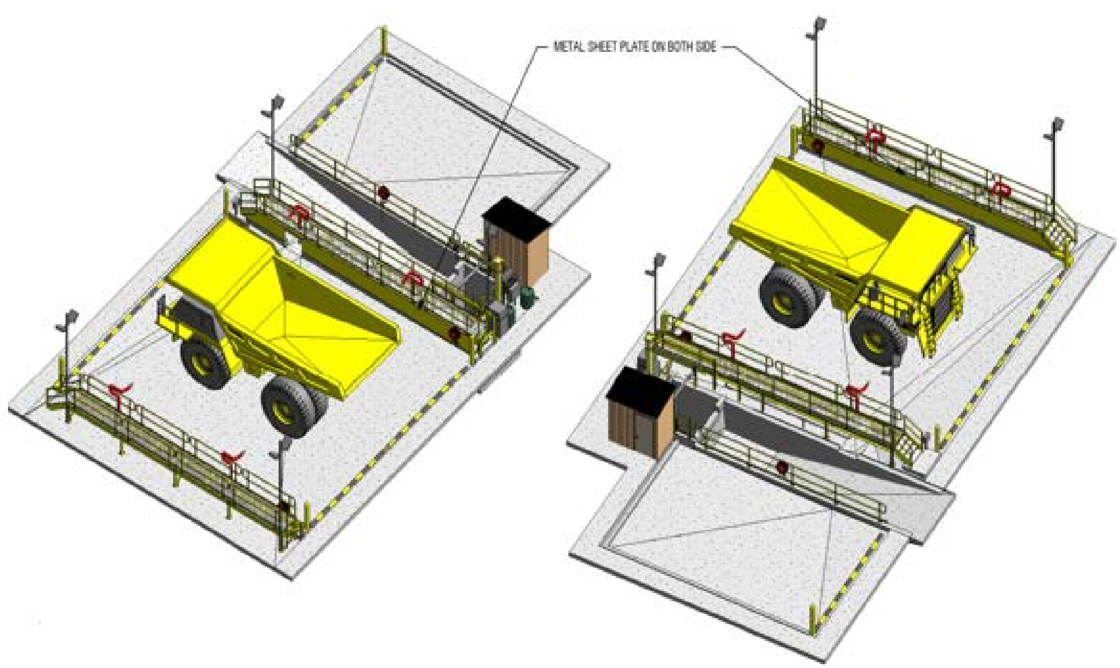
Figure 18.5: Maintenance Facility



Source: GMS (2021)

A wash bay (Figure 18.6), also sized to service Cat 777 trucks will be located near the maintenance facility. Wash water will be captured, settled, oil will be separated, and water will be recycled.

Figure 18.6: Wash Bay



Source: GMS (2021)

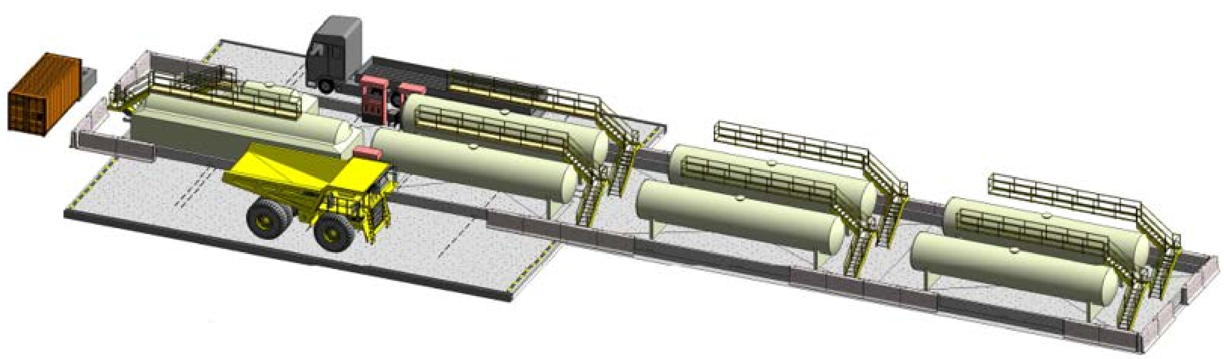
A separate container area will be dedicated to the storage of special lubricants and greases and a controlled area will be provided around flammable products such as solvents and paints.

18.6.5.2 Fuel Storage and Distribution

A fuel storage facility will service the mining and site fleet with seven tanks of 60,000 liters diesel fuel for approximately 12 days of mobile equipment operations and a 5,000 liters tank of gasoline for light vehicles. The main tanks will be storing S500 fuel whereas the last one will store S10 fuel. This infrastructure will be located west of the maintenance facility and will service both light and heavy vehicles.

During the first years of mine operations, two additional 60,000 liters tanks will be added to accommodate the increasing needs. One will be for S500 fuel and the other S10. The layout of the fuel storage and distribution is shown on Figure 18.7.

Figure 18.7: Fuel Storage



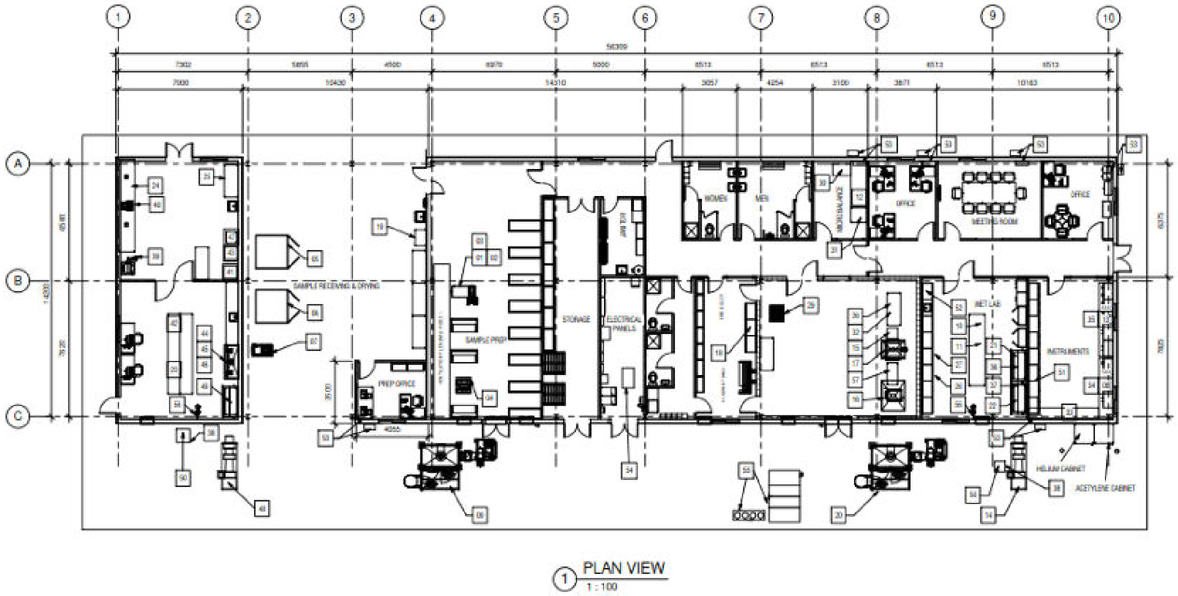
Source: GMS (2021)

18.6.5.3 Assay Laboratory

A complete sample preparation and assay laboratory facility will be built adjacent to the process plant. It will include a complete lab equipment package that will comply with international standards and best practices. It will serve as the site testing and analytical facility for the metallurgy, mine grade control and environmental departments. The floor plan is shown on Figure 18.8.

This facility will consist of a 14 m x 56 m structural steel building with offices for staff and is equipped with an air compressor and storage space for various lab consumables. It will be capable of processing 350 samples per day. The facility will be outfitted with all required lab equipment to perform Sample Preparation, Fire Assay, XRF, Atomic Absorption (AA), Gravimetry, Leach, Settling and wet chemical tests.

Figure 18.8: Assay Laboratory

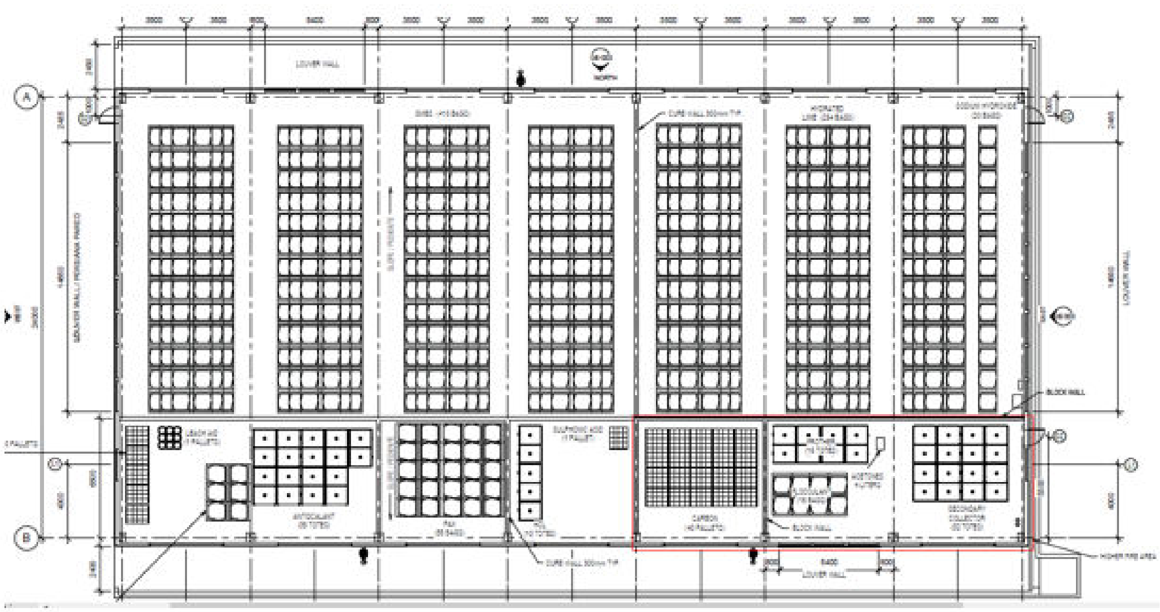


Source: GMS (2021)

18.6.5.4 Reagent Storage

All process plant reagents, except cyanide and lime, will be stored in a warehouse made of two fabric top building on concrete slab with approximately 1,300 m^2 of covered storage space (approximately 24 m x 54 m). 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 on Figure 18.9.

Figure 18.9: Reagent Storage Floor Plan



Source: GMS (2021)

A separate area will house the sodium cyanide storage area. Sodium cyanide will be shipped to site and stored in iso-containers. The delivery trucks will return with an empty iso-container for exchange. All contact water in this area will be collected and sent to the process plant.

These reagent storage areas will be within the fenced and gated area of the process plant.

18.6.6 Site Access and Security

An access gate and guard facility including a search and site access control building will be installed at the site entrance, at the south-east end of the property. Preliminary screening of all traffic entering 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. Employees and visitors will park their vehicles at this location before passing through the security control point.

18.6.7 Site Access Road

The site is accessed via 103 km of all-weather / unsealed roads, starting from the National Highway, BR 163 at Morais Almeida. The first 31 km is on the Transgarimpeiro state road. Approximately 20 km from

Morais Almeida on this state road, is the town of Jardim do Ouro, where there is a barge crossing over the Jamanxim River.

The remaining 71 km distance to site is through an access road constructed by Eldorado in 2015. The 71 km site access road is a municipal road and is accessible to the public.

18.6.8 Airstrip

There is an existing 775 m long air strip located south of the existing exploration camp. Minor upgrades will be made to the airstrip during construction, until the new airstrip is built north-east of the property.

18.6.9 Soil Stockpile Areas

Topsoil removed for the preparation of site facilities, will be stockpiled on site. The stockpiles will be reclaimed and used for site closure.

Deforestation pads will be located around the site to stockpile merchantable timber and made available for shipment off site.

18.6.10 Solid and Hazardous Waste Management

18.6.10.1 Non-Hazardous Solid Waste

The disposal of inert solid waste will be done in a landfill on site. The landfill waste facility will be located off the main access road south of the gate house.

All waste streams to be taken off site will be sorted into hazardous and recyclables and will be temporarily stored in this area.

18.6.10.2 Hazardous Waste Disposal

Anticipated hazardous wastes will consist primarily of waste oils, process reagents and laboratory chemicals. Waste oils will be incinerated or recycled by the supplier. Most reagents and chemicals that require disposal will be disposed within the process plant, and the remainder will be recycled to the supplier.

All cyanide containers and other reagent containers will be washed using industrial water. Washing will be done in contained areas. Washing of cyanide containers will comply with the International Cyanide Code standards for disposal of cyanide products.

Laboratory fire assay wastes may contain small amounts of lead. These wastes, along with any lead contaminated dust from the bag houses, will be disposed in accordance with local regulations.

Clean up of spills of hazardous materials on site will be given the highest operating priority and will generally involve the excavation of contaminated soils, neutralization of the affected site, and disposal of the affected soils on site or at a licensed facility off site.

18.7 Power Supply and Distribution

18.7.1 Power Supply

The site will be fed by a new 198 km / 138 kV power transmission line which connects to the National Grid at Novo Progresso and terminates at the site substation near the process plant.

The power line includes 476 towers and upgrades to the exit bay at Novo Progresso. The new line will be parallel to the state highway 163 to Morais Almeida, then will turn west along the site access road and eventually connecting to the site substation adjacent to the plant site.

18.7.2 Power Distribution

The substation will step the power from the new 138 kV power line down to 13.8 kV through two (2) 20 / 25 MVA, 138-13.8 kV transformers. Each transformer will power a 13.8 kV switchgear lineup and have capacity to power the entire site. A compensator, as required by the utility provider, will also be located in the substation.

From the 13.8 kV switchgear lineups, power will be distributed throughout the site. The voltage will be further stepped down to either 4.16 kV or 3.0 kV for larger loads while the remaining power will be transformed to 480 V. Single phase loads such as lighting and other low voltage loads will be powered at 220 / 380 V.

Two overhead lines will feed facilities outside of the main site. The north overhead line will provide power to the sewage plant, raw water intake pumps, CIL Pond #1 and the future CIL Pond #2.

The south overhead line will deliver power to the flotation tailings pond, exploration camp, explosives area, main gatehouse and landfill.

Emergency diesel generators located near the process plant will provide backup power to strategic loads as required in the event of a loss of utility power. The pit dewatering pump system will be diesel powered.

18.8 Communications

The site currently has two radio towers and one telecom tower which provide communications and internet. The systems will be upgraded to accommodate construction and operations. Currently, the area does not have cellular coverage. Mobile equipment and security will use handheld radios for communications.

18.9 Tailings Disposal System

The Project tailings disposal systems will consist of a flotation tailings facility (FTSF) and two CIL effluent ponds (CTFS).

These designs were elaborated according to the criteria established by Brazilian Standards NBR 13.028: 2006 and NBR 10.157: 1987, whenever applicable). The design also complies with the most recent version of the Brazilian Standards for tailings dam design (NBR 13.028 - ABNT, 2017). The tailings classification followed the criteria specified by Brazilian Standard NBR 10.004: 2004.

18.9.1 Flotation Tailings Storage Facility

The flotation tailings are classified II-B (Non-Hazardous, Inert) according to Brazilian standard. The FTSF will use a large natural valley in the southern part of the Project site. A tailings deposition basin will be created by building a main dam at the downstream end of the valley.

18.9.1.1 Design Description

The starter dam design (crest El. 151 m) is an earth dam formed by compacted clayey soil, with internal drainage composed of a vertical filter and a horizontal sand/gravel blanket. The final dam will have a mixed section, composed of an upstream clayey core and a downstream face of compacted rockfill (mine waste), with crest at El. 165.3 m. A proper filter of sand and crushed rock will be placed between the compacted clayey soil and rockfill.

The reservoir capacity for the starter dam is 7.68 Mm³, which will exceed the needs of the first three (3) years of mine production. The final dam can hold 29.8 Mm³, of flotation tailings equivalent to ten years of operations, additional capacity may be available from the mined-out pit pending environmental evaluation and approvals.

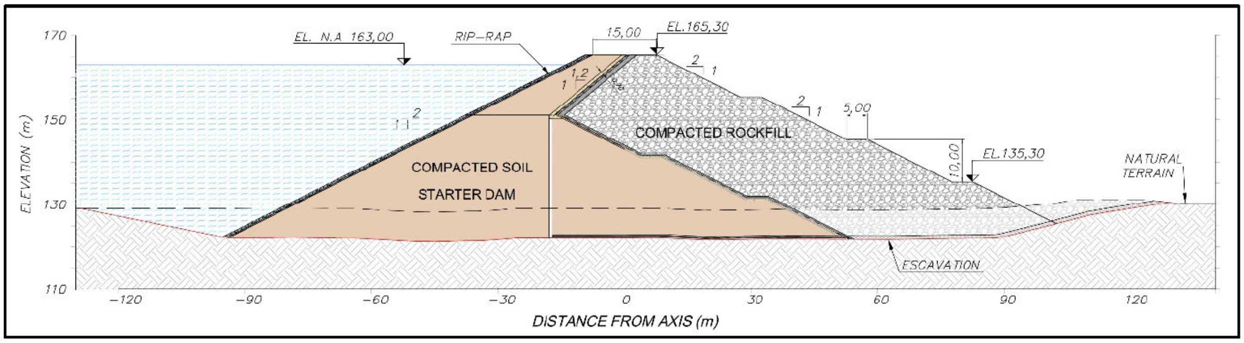
The starter dam will have a 20 m crest width, a maximum height of 29.5 m, and upstream and downstream slopes of 1V:2H and 1V:2.5H, respectively. The final dam, largely composed of compacted rockfill, will have a downstream slope of 1V:2H, the same as the upstream slope. The downstream slope in both phases will have 5 m wide berms at every 10 m in height.

Table 18.1 shows the main parameters for the starter and final dam design. A typical section of the main dam is presented in Figure 18.10.

Table 18.1: Flotation Tailings Dam Parameters

Features	Unit	Starter Dam	Final Dam
Crest Elevation	m	151.0	165.3
Crest Length	m	578.0	884.5
Crest Width	m	20.0	15.0
Dam Height	m	29.5	44.0
Reservoir Elevation (Maximum Normal Water Level)	m	148.0	163.0
Dam Volume – Compacted Soil	k m³	622	103
Dam Volume – Rockfill and Internal Drainage	k m³	49	716
Reservoir Volume	Mm³	7.68	29.8
Catchment Area	Mm²	2.84	2.84
Design Return Period	years	10,000	10,000
Maximum Inflow	m³/s	7.6	10.9
Maximum Outflow	m³/s	6.5	8.0

Figure 18.10: Typical Section – Flotation Tailings Dam



Source: Tec3, 2018

The main dam foundation is composed of residual soils from granite alteration. The existing "curimã" ("garimpo" sediments) will be totally removed. Foundation treatment for the final dam will take place during the construction of the starter dam.

The construction of a saddle dyke for the final pond will also be required; it will be constructed with compacted earth fill to a final crest elevation at El. 165.3 m over a length of 175 m and a maximum height of 11 m.

Positioned in the left abutment of the main dam, the spillway system is of free sill. For the starter phase, it will be composed of a rockfill inlet channel that drains into a stepped channel and into a stilling basin. For the final dam, both constructed with concrete. For the final dam, a new emissary channel will be built to accommodate the change in elevation, while both the stepped waterfall and the stilling basin will be kept.

18.9.1.2 Geological-Geotechnical Studies

The following outlines the studies done to determine the existing conditions:

- Geological-geotechnical mapping with identification and characterization of the main existing lithotypes at the foundation, the borrow areas for the compacted soil and for the internal and superficial drainage systems (aggregate).
- 22 SPT borings, eight (8) diamond core drillings with SPT, five (5) additional diamond core drilling and five (5) sampling wells, where undisturbed samples were collected, being four (4) at the main dam foundation and one (1) at the saddle dyke area.
- 34 auger borings for borrow areas characterization purpose.

- Electrical resistivity Geophysical investigations (3,591.20 m in eight (8) sections) and electromagnetic (ground penetrating radar - 3,591.20 m in eight (8) sections) to determine the unconsolidated material thicknesses, at the thalweg area.
- 60 soil infiltration tests and 11 rock water loss tests.
- Laboratory tests for physical and special characterization to determine the foundation and fill material properties. These lab tests consisted of grain size determination, Atterberg Limits, moisture content, one-dimensional consolidation, variable load permeability and triaxial compression tests. Besides those tests, some geotechnical tests were carried out for flotation tailings characterization, such as grain size determination, specific weight, one-dimensional consolidation and hydraulic consolidation test (HCT).

18.9.1.3 Stability Analysis

Stability analyses were performed to confirm the geometry for the starter Dam according to Brazilian standard NBR 13.028, by using the Slide 6.0 software (ROCSCIENCE, 2010). The circular failure analysis used the Spencer Limit Equilibrium method. The material strength parameters were obtained either from lab test results or estimated according to TEC3 expertise. The adopted parameters are shown in Table 18.2.

The stability analysis results are presented in Table 18.3; all evaluated sections have safety factors above 1.5 for normal operation and above 1.3 for critical water level; these results meet Brazilian Standard (NBR 13.028: 2006).

A monitoring system to record water level and pore pressure development is planned to verify dam performance. Visual inspections of the dam, at least twice a month, is also planned.

Tailings geotechnical properties were obtained from tests performed on two flotation tailings samples obtained from the pilot plant tests. These tests included granulometry, specific weight, one-dimensional consolidation and HCT (hydraulic consolidation test). The granulometry tests showed that the tailings have a fine sand predominance with more than 70 % passing through the #200 sieve.

Table 18.2: Geotechnical Parameters for the Starter Dam Stability Analysis

Material		Specific Weight kN/m³	Effective Cohesion kPa	Internal Friction Angle °	Source
Dam	Compacted Soil	18.5	17	28	Laboratory
	Filters and Transition	20	0	32	Estimated
Foundation	Mature Residual Soil	15	14	21	Laboratory
	Fresh Residual Soil	17	19	27	Estimated
	Rock Foundation	20	500	35	Estimated
	Rockfill	20	0	43	Estimated
Alluvium		18	0	30	Estimated

- Further consideration of undrained strength parameters in the foundation mature residual soil was carried out in compliance with standards and current engineering practices. Based on laboratory characterization and triaxial tests, peak undrained strength ratios from 0.38 to 0.44 were obtained. The minimum value of $Su/\sigma'd = 0.38$ was input in the analyses.
- The stability analysis results showed that all evaluated sections have safety factors above 1.3 for end of construction, 1.5 for normal operation and above 1.3 for critical water level, which meets Brazilian Standard (NBR 13.028/2017) - Table 18.3.
- The standard considers a minimum safety factor of 1.1 for seismic loading. A sensitivity analysis was performed, starting with Eletrobrás (Brazilian Energy company) recommendations, i.e., PGA (peak ground acceleration) of 0.05 g horizontal and 0.03 g vertical. It was found that the safety factor is below 1.1 only when applying 0.12 g for PGA, which is a value 240% greater than the Eletrobrás recommendation.
- For undrained strengths, the minimum required Factors of Safety were 1.3 for peak and 1.1 for residual parameters.
- A monitoring system to record water level and pore pressure development is planned to verify dam performance. Visual inspection of the dam, at least twice a month, is also suggested.
- Tailings geotechnical properties were obtained from tests performed in two flotation tailings samples obtained from the project pilot plant test. These tests were complete Granulometry (sieving and sedimentation); Specific weight; One-Dimensional Consolidation and HCT (hydraulic consolidation test).

- The GSD tests showed that the tailing has a fine sand predominance with a more than 70 % passing through the # 200 sieve.
- At the consolidation tailings tests, differences were observed from the results of the initial void ratios. It is highly recommended to perform new tests during operation, with samples collected in the reservoir thus enabling to accurately estimate the consolidation behavior during operations.

Table 18.3: Results of the Tailings Dam Stability Analyses

Local	Section	Loading Condition	Minimum Required FoS	Obtained FoS
Starter Dam (stretch 1)	Stake 12+0,0 (right shoulder)	Normal operation regime, seismic loading	1.1	1.27
		Normal operation regime, residual soil undrained	1.3	1.53
	Stake 19+15,6 (central)	Normal operation regime, seismic loading	1.1	1.36
		Normal operation regime, residual soil undrained	1.3	1.84
	Stake 24+0,0 (left shoulder)	Normal operation regime, seismic loading	1.1	1.12
		Normal operation regime, residual soil undrained	1.3	1.32
Starter Dam (stretch 2)	Stake 25+0,0 (right shoulder)	Normal operation regime, seismic loading	1.1	1.19
		Normal operation regime, residual soil undrained	1.3	1.39
	Stake 28+2,9 (central)	Normal operation regime, seismic loading	1.1	1.21
		Normal operation regime, residual soil undrained	1.3	1.54
	Stake 30+14,4 (left shoulder)	Normal operation regime, seismic loading	1.1	1.61
		Normal operation regime, residual soil undrained	1.3	1.80
Final Dam	Stake 16+0,0	End of construction (upstream slope)	1.3	1.65
		Normal operation regime	1.5	1.77
		Normal operation regime, seismic loading	1.1	1.24
		Normal operation regime, failure through contact compacted soil-rockfill	1.5	2.67

Local	Section	Loading Condition	Minimum Required FoS	Obtained FoS
		Normal operation regime, seismic loading, failure through contact compacted soil-rockfill	1.5	1.91
		Critical operation regime	1.3	1.59
		Critical operation regime, failure through contact compacted soil-rockfill	1.3	1.50
Saddle Dike	Stake 4+8,13	Endo f construction (upstream slope)	1.3	1.63
		End of construction (downstream slope)	1.3	2.11
		Normal operation regime	1.5	1.84
		Normal operation regime, seismic loading	1.1	1.29
		Normal operation regime, residual soil undrained	1.3	1.36
		Critical operation regime	1.1	1.44

18.9.1.4 Hydrological Studies

These studies were performed to dimension the hydraulic structures and to provide support for the dam reservoir water balance.

The considered contribution basin has a drainage area of 2,842 km², a perimeter of 7.1 km and a main river average slope of 0.032 m / m. It has a rounded shape, being characterized by a dense vegetation in most parts.

The climatological data from 1961 to 1990 was obtained from the Itaituba station, of the National Department of Meteorology. It shows that the rainy season occurs between November and April with an average annual precipitation of approximately 2,100 mm. Evaporation is more significant between the months of July and January with an average annual evaporation value of 900 mm. Characterization data for the area intense rainfall regime was obtained from HIDROWEB of the National Water Agency (ANA). The rainfall station Km 1326 BR-163 (555000) was adopted due to its proximity to the Project area, it has data since 1980.

The spillway system was designed for floods with a return period of 10,000 years. The results of the hydrological simulations to determine the spillway design floods and cofferdams indicate that the design peak flood flows are equal to 6.59 m^3/s, 6.50 m^3/s and 8.00 m^3/s for the first year of the diversion channel and the spillway and the end of the third year, respectively.

18.9.1.5 Tailings Deposition Plan

The tailings deposition plan was elaborated during the Basic Design phase when 22 disposal scenarios were performed considering the disposal beginning in Year 1 until the tailings dam final geometry is totally completed in Year 10. Tailing beaches were determined from the initially proposed geometry with the emerged slope equal to 1% and the submerged slope equal to 3% and a dry apparent density of 1.325 t/m^3. It will be important to evaluate the dry density during operation to verify the impacts of lower densities in the disposal plan.

18.9.1.6 Hypothetical Dam Failure

Dam break evaluation for both the flotation tailings dam and CIL ponds considered these structures in their final configurations, with reservoirs at full capacity. The simulation has also considered a simultaneous failure of the dam and the two ponds, to evaluate the most critical scenario and the maximum hazard. Failure modes considered overtopping at the dam and piping (internal erosion) at the CIL ponds. The likelihood of the occurrence of such failures is extremely low. Four scenarios were evaluated:

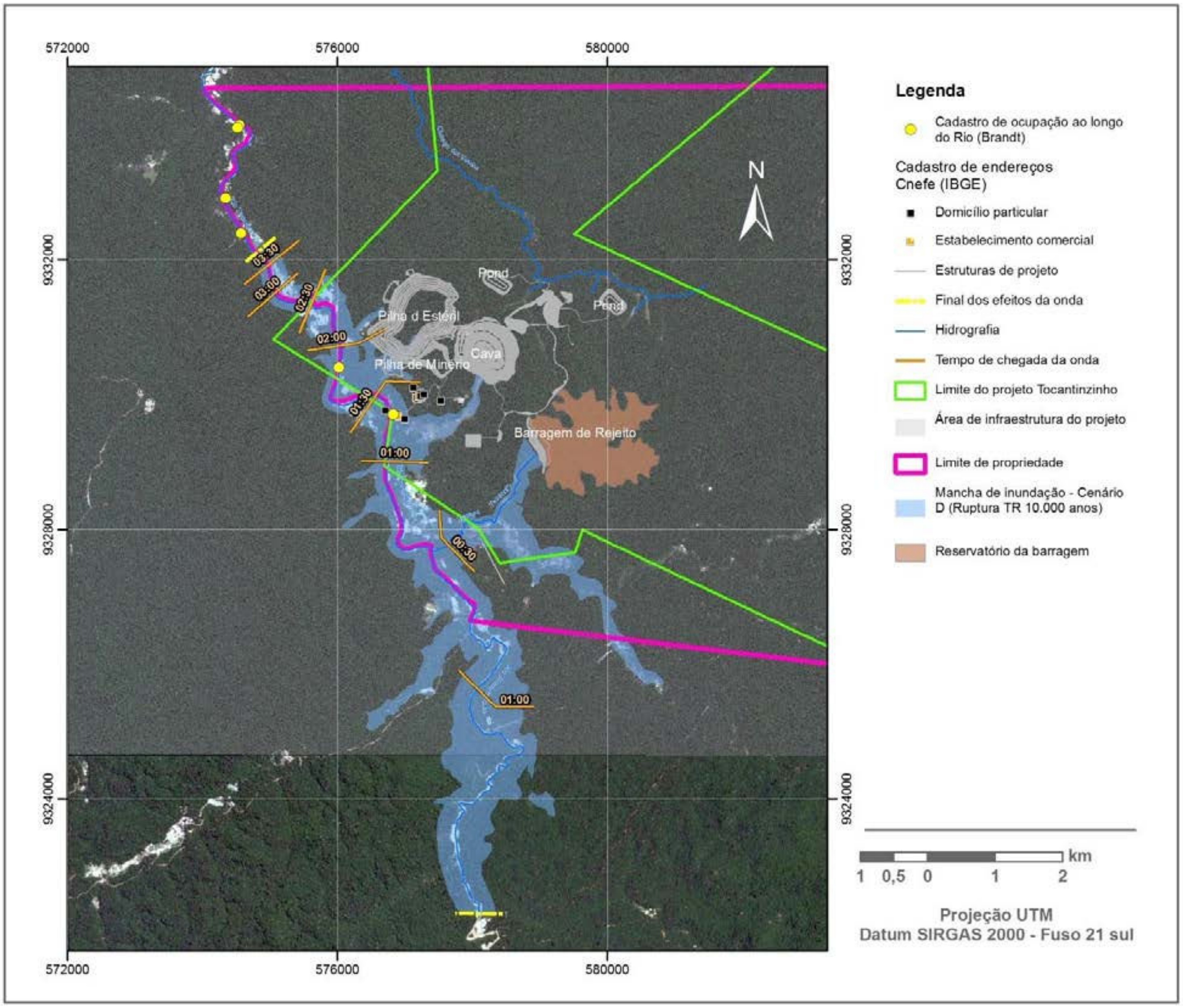
- A (base case): no failure, to simulate the impacts of a natural flood equivalent to two years return period on the downstream water courses.
- B: considers the impacts of a failure on the downstream water courses, during the dry season.
- C: no failure, to simulate the impacts of a natural flood equivalent to 10 years return period on the downstream water courses.
- D: rainy day, considering a failure during an extreme event on TSF and CIL, equivalent to 10,000 years return period.

The results of the simulations show the maximum flood levels downstream of the structures, the hydrodynamical risks and the maximum depth of the water level in several sections along the flooded area, in an event of failure.

As there is no permanent human population downstream and the area is composed of native forest, the main impacts are those associated with the environment. However, as there is eventual presence of artisanal miners (“garimpeiros”), they can also be impacted. In scenario D, the backwater effects could also reach the pit and the waste dump.

Figure 18.11 shows the flood plume for scenario D.

Figure 18.11: Scenario D Impacts



Tec3, 2018

18.9.2 <u>CIL Tailings Storage Facility</u>

The location of the two ponds for the CIL Tailings Storage Facility (CTSF) was modified recently as part of the Project site optimization. The selected new site is expected to have very similar topography and soils conditions. Testing will be repeated to confirm suitability of the site. Design is assumed to be identical.

18.9.2.1 Design Description

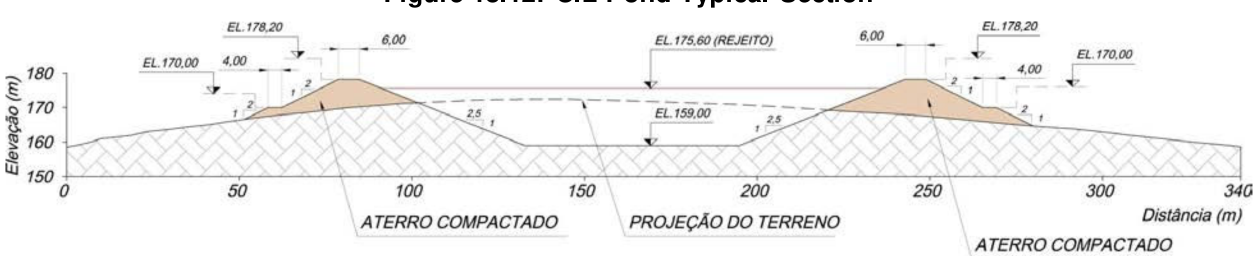
The design concept requires the ponds to be excavated in soil down to a defined level, and the excavated material to be used for the compacted perimeter dykes to minimize both the disposal volume and borrow materials. Each pond is expected to contain about five (5) years of mine operation, starting with Pond #1 (South Pond). Pond #2 (North Pond) will be constructed during Year 1 of operations and put in use in Year 2.

A storage volume of 882,000 m^3 is estimated for Pond #1 and 700,000 m^3 for Pond #2, for the ponds in the new location. Table 18.4 presents the main features for the ponds and Figure 18.12, the typical cross section.

Table 18.4: CIL Pond Parameters

Parameter	Unit	Pond #1	Pond #2
Crest Elevation	m	168	168
Reservoir Volume	000 m^3	882	700
Elevation Bottom of Pond	m	157	157
Max Height of Water	M	9	9

Figure 18.12: CIL Pond Typical Section



Source: Tec3, 2018

The CIL tailings are classified as Class 1 – Hazardous because of its acid generating potential caused by the high proportion of sulphides.

The design specifies that the whole internal face and the bottom of the ponds will be lined with a geomembrane liner to create impervious ponds and minimize groundwater contamination.

Underneath the geomembrane, each pond will have a leak detection system composed of a sand layer connected to a pipeline and a pumping system for percolation exhaustion. This layer will be confined by a double layer of geomembrane, the top layer being conductive, and the bottom layer being textured on both sides. In the contact between the natural terrain and the geomembrane, a geotextile layer will be placed to protect against tears/rips.

No overflow system is foreseen in the pond during operation; however, a pumping system is planned to remove the decant water for treatment and subsequent discharge in the environment starting in Year 3 of operations as the water treatment plant will be operational in Year 2.

18.9.2.2 Geological-Geotechnical Studies

For the initial site of the CTSF, 20 auger borings, four (4) investigation wells, six (6) SPT borings and 24 soil infiltration tests were performed for Pond #1 and two (2) SPT borings and eight (8) soil infiltration tests for Pond #2.

Physical characterization and special laboratory tests were performed to determine foundation and landfill material properties. The tests performed were complete granulometry, Atterberg limits, moisture content, one-dimensional densification, triaxial compression and variable load permeability test.

Stability analyzes were performed to verify the geometry (according to Brazilian Standard NBR 13.028), through Slide 6.0 software (ROCSCIENCE, 2010). Circular failures were evaluated by the Spencer limit equilibrium method.

Unfortunately, a change of site for the CIL storage pond (CTSF) will require that the studies be performed again. However, the new site has a similar topography and foundation and is expected to have similar geological and geotechnical properties.

18.9.2.3 Hydrological Studies

The hydrological studies were performed both for designing the hydraulic structures and to provide support for the pond water balance. Events associated with a 1,000-year return period to verify the capacity of the recirculated water pumping system with a 1 m minimum freeboard were considered as design events.

About 5% of the ore feed to the process plant will report as CIL tailings or about 2,434,000 tonnes over the LOM.

A dry bulk density of 1.705 t/m³ was estimated for the settled tailings: the higher density is explained by the higher density of the sulfide particles in the CIL tailings.

To evaluate the safety of the reservoir against the overtopping, since the pond will not have an emergency spillway, the reservoir hydric balance was elaborated in a simulation model with daily interval, where daily rainfall is obtained from stochastic hydrological modeling, simulating the structures life in 1,000 operating scenarios.

The results obtained at the end of those simulations considered a pumping system with a capacity of 100 m³/h (one pump operating and one backup pump, each pump with a 100 m³/h capacity) to assure the minimum freeboard in the most critical simulation which resulted in a freeboard of 1.53 m for 1,000 years of recurrence. Current design specifies 2 m of freeboard.

Complete hydrological studies were carried out for the original geometry and location of the ponds, associated with a hydric balance. These studies will be rerun for the current system layout.

Closure arrangement are to be finalized but could be either a one-meter water cover or an impervious material covering the tailings with a spillway to discharge the surplus water.

19 MARKET STUDIES AND CONTRACTS

19.1 Market Study

19.1.1 Gold Market

The 2021 World Bank Commodities Price Forecast indicates a decrease in the average price of gold to USD 1,750/oz in 2022 from an average of USD 1,799/oz in 2021. Leading up to 2030, the gold price is forecast to decrease to USD 1,653/oz, and eventually to USD 1,600/oz by 2035, as presented in Table 15.1.

Table 19.1: World Bank Gold Price Forecast (nominal US dollars)

Gold Price (USD/oz)	Calendar Year	Annual Average
Actual	2015	1,160
	2016	1,251
	2017	1,257
	2018	1,268
	2019	1,393
	2020	1,770
	2021	1,799
Forecast	2022	1,750
	2023	1,730
	2024	1,719
	2025	1,708
	2030	1,653
	2035	1,600

Source: World Bank – Commodities Market Outlook, October – 2021 – Commodities Prices Forecasts

Gold has emotional, cultural and financial value and different people across the globe buy gold for different reasons, often influenced by a range of national socio-cultural factors, local market conditions and wider macro-economic drivers. Gold's diverse uses, in jewellery, technology and by central banks and investors, mean different sectors of the gold market rise to prominence at different points in the global economic cycle.

This diversity of demand and self-balancing nature of the gold market underpin gold's robust qualities as an investment asset.

Analysis by the World Gold Council has shown that the value of gold during periods of low-interest rates is about twice as high as the historical average. Moreover, gold seems to be more effective in portfolio diversification, mitigation of risk, and long-term returns compared with government bonds.

Therefore, in the current conditions of low-to-negative interest rates, demand for gold from investors and Central Banks is forecast to remain relatively flat over the next five years.

19.1.2 Gold Price

The price of gold is the largest single factor in determining profitability and cash flow from operations. The financial performance of the project is closely linked to the price of gold. Historical gold prices are shown in Figure 19.1, with quarterly and yearly trailing averages shown in Table 19.2.

Mineral Reserves have been modelled at a gold price of USD 1,400/oz while Mineral Resources are at USD 1,600/oz. Project economics have also been assessed at a base case gold price of USD 1,600/oz, which is below the three-year historical average gold price of USD 1,654/oz as of December 31, 2021. Project economics at a range of gold prices are evaluated as part of project sensitivity analysis in Section 22.

Figure 19.1: Three Year Historical Gold Price

Sources: FastMarkets, ICE Benchmark Administration, Thomson Reuters, World Gold Council

Table 19.2: Historical Gold Prices – Quarterly and Trailing Average

<table>
<tr><th rowspan="2">Calendar Period</th><th rowspan="2">Quarterly Average</th><th colspan="3">Historical Trailing Avg.</th></tr>
<tr><th>3 Year</th><th>2 Year</th><th>1 Year</th></tr>
<tr><td>Q1 2019</td><td>1,304</td><td rowspan="12">1,654</td><td rowspan="4"></td><td rowspan="8"></td></tr>
<tr><td>Q2 2019</td><td>1,309</td></tr>
<tr><td>Q3 2019</td><td>1,472</td></tr>
<tr><td>Q4 2019</td><td>1,481</td></tr>
<tr><td>Q1 2020</td><td>1,583</td><td rowspan="8">1,784</td></tr>
<tr><td>Q2 2020</td><td>1,711</td></tr>
<tr><td>Q3 2020</td><td>1,909</td></tr>
<tr><td>Q4 2020</td><td>1,874</td></tr>
<tr><td>Q1 2021</td><td>1,794</td><td rowspan="4">1,799</td></tr>
<tr><td>Q2 2021</td><td>1,816</td></tr>
<tr><td>Q3 2021</td><td>1,790</td></tr>
<tr><td>Q4 2021</td><td>1,795</td></tr>
</table>

Sources: FastMarkets, ICE Benchmark Administration, Thomson Reuters, World Gold Council

19.2 Contracts

Gold bullion transportation and refining contracts will be negotiated and finalized during the construction phase of the Project.

GMIN has entered into a Master Service Agreement with GMS, a Canadian Mining-focused engineering and mine development firm, for the engineering and construction management of the TZ Project.

The other important contract which requires finalization is the power supply contract which is currently in discussion with the relevant Brazilian utilities.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies

In 2011, Brazauro completed an Environmental Impact Assessment ("EIA"), carried out by Brandt Amazônia, with the objective of obtaining its first Preliminary License for the Project, in accordance with the legal requirements in Brazil for permitting purposes.

Environmental baseline studies were completed on the Project including flora and fauna studies, hydrology and hydrogeology monitoring and studies, archeological surveys, geochemistry analysis and geotechnical studies which are summarized below.

20.1.1 Flora and Fauna

The Project is located in the Amazonian biome, which is comprised of extensive areas covered by forest formations that over the years have witnessed human occupation resulting in a high rate of deforestation caused mainly by agriculture, logging and artisanal mining activities. In general, the local forest typology is known as ombrophilous forest, prevalent throughout the Amazon Region. Qualitative and quantitative studies of flora within the Project site registered 196 botanical species, some of them being classified as threatened.

In the fauna studies, aquatic and terrestrial ecosystems were considered. Within the aquatic ecosystem, limnological organisms (algae and small aquatic organisms) and fish were investigated. The terrestrial ecosystem included groups of insects, amphibians, reptiles, birds, mammals and bats.

In the study areas, various limnological organisms and 91 species of fish were identified. Of this total, 20 species are important for human consumption and 25 for ornamental fish culture practice. Also identified were 126 species of insects (mainly mosquitoes and ants), 21 amphibian species, 33 reptile species, 222 bird species, 45 mammal species (10 threatened and one (1) endemic) and 48 species of bats (one (1) vulnerable).

The biotic environment study was completed to predict impacts caused by the installation of the Project and establish mitigation measures to reduce those impacts on the ecosystems and maintain biodiversity in the Project's area of influence.

20.1.2 Hydrology and Hydrogeology

Studies to evaluate water resources were completed in the Tapajós River basin, the sub-basin of the Jamanxim River, with emphasis on the micro-basin of the Tocantinzinho River, including Teodorão and Veados creeks, its tributaries. Qualitative and quantitative aspects of water resources were considered, taking into account the geographic, hydrological, physicochemical and bacteriological parameters of the study area.

The Tocantinzinho River has an extension of 197 km and its basin measures 5,566 km². River navigation is generally limited as the riverbed is rocky and has many rapids and sandbanks in the dry season. The micro-basins are characterized by having embedded valleys in sedimentary rocks, of average altitude of 160 m, with dendritic pattern, high angularity, medium density and medium to high asymmetry. In these regions, processes of deposition predominate, developing extensive fluvial plains composed of recent alluvial material, subject to flooding during the most recent period.

Considering that the hydrogeological conditions belong to the Crystalline Basement and sediments of the Amazon Basin, the exploration of underground water in the region is done through shallow or tube wells. The underground flow direction is mainly from NE to SW and is directed to the thalweg of the Tocantinzinho River.

Systematic monitoring of hydrological, and hydrogeological variables was executed within the Project site and the monitored data provided rich background information on several variables of interest for the Project. The quantity and quality of the data obtained made it possible to establish several correlations between rainfall, runoff flows and water levels (whether surface or underground).

The hydrological regime of the region is defined by the Tapajós River. The flood season extends from January to May, always peaking in the month of March. However, the analysis of the data collected showed mainly that the annual rainfall in the Project area is not uniform throughout the years, showing great variability in the monthly totals. The Tocantinzinho River and Teodorão Creek have a flow regime characterized by floods followed by a long-lasting recession period. Nevertheless, the minimum flow is considered more than sufficient to meet the demand of fresh water for the Project.

20.1.3 Air Quality

Air quality is dependent on atmospheric emissions from each Project stages and localized climatic and topographic conditions of the area. Currently, compliance with the legal standards is favorable since the area surrounding the Project is predominantly forest and there are no industrial activities that have fixed sources of atmospheric emission. Thermal inversion is not a problem due to the low wind velocity and high rainfall. The dry season is long but there is still rainfall, resulting in the relative humidity of the air to always remain high.

The analysis of the results obtained from the total suspended particles ("TSP") monitoring concludes that, in general, the air quality in the Project site is considered good when compared with standards established by the local regulatory bodies. To ensure the maintenance of emissions within permitted thresholds, potential sources of atmospheric emissions, which may be stationary or mobile, will be monitored during all stages of the Project and operation.

20.1.4 Geochemistry

The chemical species formed by the weathering of rocky materials are largely stored in sediments and soils. Due to the wide artisanal mining activity in the past, extensive sampling of soils and sediments were completed to determine the concentrations of chemical parameters and to establish geochemical background for the site. The information also provided data for the evaluation of possible environmental interferences resulting from future mining activity in the area. The analysis of the soil and sediment samples presented higher concentrations of aluminum in certain areas.

Mercury also requires attention, since the assay results have shown that this element is found in several areas due to the history of artisanal mining activity. Although the magnitude of the mercury found is not of concern, it is important that this element be investigated in greater detail in the next phases of the Project as more can be encountered during the pit pre-stripping.

Brazauro conducted two campaigns of static testing to evaluate the potential for acid rock drainage ("ARD") and metal leaching ("ML") of its ore and waste materials. The first campaign was carried out in 2010 by the company VOGBR and the second campaign by CLAM Engenharia Hidrocnese ("CLAM") in 2012, which report consolidated both campaigns.

The results of the metal leaching tests in both campaigns showed that there is no release of any excess elements to the limits established in Annex F to ABNT NBR 10004:2004, which allowed to conclude that the materials do not fall within the category of hazardous waste (class I A). The analysis of the results of static tests obtained from both campaigns showed that the waste materials of the Project, especially quartz-monzonite, have high PN/PA ratios due to the presence of carbonate in all materials and consequently have low to no potential for acid generation.

The process plant tailings were not available for ARD potential testing, only for ML. The ML tests performed on the flotation tailings by SGS Geosol Laboratories in 2016 according to the standards of ABNT NBR 10004, determined that the flotation tailings samples were classified as Class II B (Not hazardous - Inert), for not having any of its solubilized constituents with concentrations higher than the standards of water potability, excepting appearance, color, turbidity, hardness and taste.

It is recommended to complete ARD characterization for tailings and to perform additional ML tests for tailings and waste materials.

20.1.5 Geotechnical Analysis

The geotechnical studies were done to provide technical information to design earthworks for access, earthworks in cuts and fills and the buildings and structures required for the Project. These studies were developed with the main objective to characterize the quality of the soils of foundation (subgrade) and the quality/volume of the materials for construction of the structural layers such as rock and sand.

In a geotechnical-construction context, a geotechnical monitoring program will be installed to monitor the predicted estimates for soils behavior and structures, natural deformation, specifically in embankments and in the slopes and foundations.

20.1.6 Archeological Surveys

An archaeological survey was completed in the areas of the Project site and the power transmission line. Analysis of data through non-intrusive exploration was conducted and areas were identified that have potential for the existence of archaeological sites.

During a second phase of the study, such potential sites within the Project area were reviewed through soil subsurface verification and heritage education activities were carried out with local communities. During the survey, six archaeological sites were identified in the area of the Project site and seven in the area of the power transmission line. The archaeological sites identified have been through the rescue stage, curator activities and analysis of the material collected in the laboratory with the final report approved by the Instituto do Patrimônio Histórico e Artístico Nacional (“IPHAN”). These areas can now be used for the Project.

20.2 Environmental Management

20.2.1 Water Management

CLAM completed several hydrology and hydrogeology studies related to surface water availability, creek crossings, groundwater uptake for potable water and a site-wide water balance.

The Project is within an area with hydrological surplus, resulting from annual precipitation of 2,248 mm and average pan evaporation of 834 mm as measured at Itaituba from 2006 to 2020.

The project has an estimated raw water demand of 200 m³/h for the process plant and multiple industrial uses. CLAM's report suggests that the Veados Creek has enough water availability to provide the 200 m³/h. Veados Creek has a total available flow of 387 m³/h, almost double the project demand, with limited upstream or downstream potential usage given the remote location. Water from the creek will be filtered before being stored and distributed as domestic water throughout the plant and camp. Some of the domestic water will be further treated to qualify as potable water. To maximize water conversation, process water required for the process plant will be recirculated from the flotation tailings storage facility and process water for cyanide destruction will use decanted water from the CIL tailings ponds.

During construction, an initial dam will be built to create a reservoir to store flotation tailings discharged from the process plant. The reservoir (FTSF) will initially store sufficient rainwater to enable the start of process plant operations. Tailings will be initially spigotted on the upstream face of the main dam as per best practice to keep decant water away from the dam and improve the dry density of deposited tailings. A barge holding sufficient pumping capacity will be located in a bay northwest of the main dam to optimize tailings sedimentation; the pumping arrangement will be designed to recycle water to the process plant and discharge surplus water in the environment once operations have reached steady state.

Some low drainage basins will receive contact water from the mine operations. It is planned to line all slopes with waste rock from pit operations to minimize erosion and addition of solids in suspension to the existing drainage. If required, small dams with spillways may be required to allow sedimentation of solids before discharge in the environment.

In terms of sewage disposal, sewage treatment plants will be located at the plant and at both the initial construction camp and the permanent camp.

As part of the requirements for obtaining the water grants for the Project, Brazauro requested from CLAM Engenharia Hidrocnese the development of a hydrogeological model for the pit area to estimate groundwater inflows. As a result, CLAM has estimated maximum inflow of 40 m^3/h. Water from the pit will be pumped to the environment or to the FTSF to allow sedimentation if required. On average, the inflow used is 34 m^3/h for the life of the mine. The groundwater inflow varies between dry and wet seasons.

20.2.2 Tailings Management

The project considers two tailings streams from the process plant: tailings from the flotation circuit and tailings from the leach / CIL circuit.

The flotation tailings storage facility (FTSF) will receive approximately 95% of the tailings from the process plant and will require the construction of a main dam that will be phased over the life of the Project as described earlier. Given the tailings are classified as non-hazardous and inert, there is no requirement for a liner system. The effluent from the FTSF will be discharged without any treatment since the tailings are inert. Sufficient settling time will ensure respect of the solids in suspension criteria. The FTSF will have a total volume capacity of 29.8 Mm^3.

The CIL circuit tailings representing the remaining 5% of the process plant tailings will be stored in a separate storage facility ("CTSF") which will consist of containment provided by two ponds. The CIL ponds #1 and #2 are designed with storage capacity for the life-of-mine solid tailings from cyanide leaching and CIL gold recovery from the gold concentrates. The effluents from the CIL circuit will be treated in a cyanide detoxification circuit using the conventional SO_2 / air process before deposition in the ponds. The CIL Pond #2 is planned to be constructed in Year 1 of the mine life for use in Year 2. The two ponds will have a total capacity of 1.58 Mm^3. To avoid contamination of the groundwater the whole internal face and the bottom of the pond will be lined with a layer of geomembrane liner to guarantee impervious ponds.

Those capacities are consistent with the study prepared by Brandt Amazonia (2011) that presents an estimate tailings volume (Flotation + CIL) of 39.68 Mm^3 and are thus consistent with the information provided to the environmental agency as part of the licensing process.

The National Dams Safety Policy ("PNSB") provides for the main rules regarding the construction, operation and maintenance of dams for accumulation of water for any use, final or temporary disposal of tailings and the accumulation of industrial waste. In addition, mining companies must comply with the Brazilian Mining Code, ANM's regulations and other applicable technical standards regarding dams erected for mining activities.

Mining companies operating dams must submit to ANM and other relevant environmental authorities, updated technical information about the dams, and their Emergency Action Plan (Plano de Ação Emergencial).

Cyanide destruction testing by SGS Canada in 2017 of the Leach and CIL tailings confirmed that the tailings are classified as hazardous and potentially acid generating because of the high sulphide content. A minimum one meter of water will be maintained above the deposited tailings to control any oxidation of the tailings. Analysis of the quality of the water after the cyanide destruction process was obtained from laboratory testing. Based on two-stage cyanide destruction test work at 40% solids, the detox product achieved the following results for water contaminants:

- 0.41 mg/L total cyanide (below 1 mg/L effluent limit)
- 0.2 mg/L cyanide WAD (on effluent limit of 0.2 mg/L)
- 4.28 mg/L Cu (above 1 mg/L effluent limit)
- <0.05 mg/L Fe
- 23.9 mg/L N (above 20 mg/L effluent limit for total ammoniacal nitrogen as N)

It is currently assumed that dilution from rainwater, natural degradation and volatilization will bring the cyanide and ammonia concentrations in the CIL tailings ponds into compliance with discharge criteria. The only parameter for which treatment should clearly be anticipated is copper. No removal of dissolved copper is expected as a result of aging in the ponds. It is currently assumed that it will be necessary to treat and remove copper by hydroxide precipitation via lime addition. The resulting solids will be recovered by ballasted clarification (e.g., WesTech RapiSand). The reaction with lime should be staged upstream of the RapiSand unit in a tank with sufficient residence time (~40 minutes at peak flow) to allow the precipitation reactions to run to completion. This will avoid scaling issues in the RapiSand equipment package. The treated water will require neutralization at the outlet from the RapiSand unit. It is assumed that treatment sludge from the RapiSand unit can be returned to the CIL tailings pond.

All decant water will be contained in the two ponds for the first two years of operations. This provides enough time to assess water quality and adjust the design of the effluent treatment in the first year of operations, construct the facility and put it in use for the third year of operations.

20.2.3 Waste Rock Management

Waste rock from the open pit will be stored in a main waste rock storage facility ("WRSF") to the northwest of the open pit with a designed capacity of 78.7 Mm³, which is lower but consistent with EIA which presented an estimated waste rock volume of 94.57 Mm^3. Footprint of the main waste storage rock area was reduced to minimize deforestation requirements.

A second waste rock storage facility is proposed downstream of the main FTSF embankment to provide for additional long-term stability of the structure resulting in greater safety factors. This second waste rock storage facility is seen as a project improvement.

20.2.4 Waste Management

Management of solid waste generated in BRM's activities must comply with the National Policy on Waste Management ("PNRS") and with applicable state and municipal regulations. Among the PNRS obligations, companies generating mining waste (extraction or processing) must submit a Solid Waste Management Plan ("PGRS") under the environmental licensing procedure, providing information on generation, treatment, packing, transportation, and final disposal of solid waste.

While there is a PGRS for the Project (presented in the licensing procedure), it has yet to be implemented due to the low quantities of waste currently generated.

20.3 Permitting

20.3.1 Permitting Process

In parallel with securing mining concessions and undertaking development and mining construction activities, environmental licenses are required.

The Brazilian National Environmental Policy (Federal Law No. 6.938/1981) requires that all potentially or effectively polluting activities be subject to the environmental licensing process. Applicable rules regarding the licensing procedure were established by Resolution No. 237 of the National Council for Environment ("CONAMA") dated December 19, 1997. The Federal Complementary Law No. 140/2011, in turn, describes the criteria establishing jurisdiction for environmental licensing by the union, the states, the federal district and the municipalities.

By means of the licensing process (which licenses will be issued by the competent environmental authority), the issuing agency determines the conditions, limits, and measures for control and use of natural resources, and allows the installation and implementation of an activity. Usually, the environmental licensing process follows three steps:

4. a Preliminary License ("LP"), granted during the preliminary stage of planning the facility or activity, which approves the location and the project conception.
5. an Installation License ("LI"), authorizing the facility or activity setting up in accordance with approved plans, programs and designs. and
6. an Operation License ("LO"), authorizing the operation of the facility or activity, after actual compliance with the prior licenses.

As mentioned beforehand, in 2011, Brazauro completed its EIA, carried out by Brandt Amazônia, with the objective of obtaining its initial LP for the Project. That EIA was approved in September 2012, with the granting of the relevant LP, which covers two main structures: the site, including activities related to mining and ore processing and the access road to the Project.

A public hearing for the Project was held June 14, 2012, in Itaituba as part of the environmental permitting process. The meeting was attended by federal, state and municipal representatives along with trade unions and the general public. Brazauro outlined the Project and presented its employment policy regarding the local population. Discussions were focused on the opportunities for local businesses in Itaituba and Moraes Almeida, to service the Project demands during its eventual construction and operation stages. While the meeting was viewed as very positive, Brazauro acknowledges that public concerns will need to be continually monitored and addressed.

In January 2016, an Installation License ("LI") for the Project was requested, was subsequently granted in April 2017 with additional modifications granted in August 2017.

20.3.2 <u>Status of Permits</u>

The environmental licensing process for the Project is conducted by the State of Pará environmental agency Secretaria de Estado de Meio Ambiente e Sustentabilidade ("SEMAS"). Considering its geographical location, the Project involves several associated structures (*e.g.*, transmission line, tailings dam and CIL pond, fuel station, concrete batch plant, access road and landfill) to ensure the basic infrastructure for mining operations.

BRM has obtained seven LIs to begin the Project's installation. However, installation activities have not yet begun and, in 2019, BRM requested SEMAS to suspend 6 out of 7 LIs, which SEMAS consented to do for "a maximum period" of 730 days. It is important to note that, while the suspension of an LI is not equivalent to the extension of the license term and / or the renewal of a license, SEMAS has discretionary power to so decide and, as long as duly justified, it may suspend licenses / authorizations when there are temporary constraints (e.g., of economic or technical nature) to proceed with the licensed activity (situation in which the expiration date is postponed). In BRM's case, the environmental agency recognized that the infrastructure for energy transmission by the local energy concessionaire was delayed, which justified the suspension of the relevant LI.

The (LI) suspension requests were previously agreed with SEMAS; during the suspension period, the licenses are "frozen", and the 2-year period is not considered for the purposes of calculating the maximum 5-year term for extending an LI provided by the legislation.

In order to start the construction activities immediately, BRM must formally request SEMAS to revoke the LIs suspensions and formally reactivate them, according to what is foreseen in the suspension letters. After authorization by SEMAS to resume the activities (which will include the analysis of the renewal requests presented), construction works may immediately commence. Revocation requests were submitted by BRM and granted by SEMAS. SEMAS will also administer requests to extend the validity of the LIs by an additional two years which requires a technical analysis and site inspection.

The permits relating to vegetation suppression, wildlife monitoring, capture, collection, rescue, transport and release were previously obtained but have expired, as no activities being the subject matter of these permits are currently carried out. New permits must be requested from the relevant environmental authorities before construction begins.

Similarly, the preliminary water permits to drill water wells and permits for effluent release have expired and must be re-applied for.

A summary of all permits and environmental authorizations is outlined in Table 20.1. Brazauro has regularly filed reports with the environmental agency to comply with its legal obligations as per the conditions of each license.

Table 20.1: Actual Licenses, Authorization and Water Grants for the Tocantinzinho Project

Project Aspect	Related License	Permit #	Expiration Date	Description
Tocantinzinho Site	Installation License (LI)	2771/2017	Original date: 18/04/2020 Suspension expiry: 27/12/2021 Revocation: 17/12/2021	License to construct infrastructures for the Project, including process plant and support infrastructure.
	Deforestation Permit	3383/2017	18/04/2020	New authorizations are required when construction will begin, to carry out deforestation activities and wildlife management.
	Permit to Monitor Wildlife	3381/2017	18/04/2020	
	Permit to capture, collect, rescue, transport, and release wildlife	3384/2017	19/04/2018	
Drainage Crossings	Final Water Permit	2772/2017	02/03/2022	Installation of culverts in creek crossings (12) draining into the Tocantins River. Layout changes will require renewal of definitive permit.
Transmission Line	Installation License (LI)	2797/2017	Original date: 27/12/2020 Suspension expiry: 18/03/2022 Revocation: 05/01/2022	License to construct 138 kV transmission line between Novo Progresso and TZ substation.
	Vegetation Suppression	3642/2017	28/12/2018	New authorizations are required when construction will begin, to carry out deforestation activities and wildlife management.
	Permit to Monitor Wildlife	3967/2019	26/03/2020	
	Permit to capture, collect, rescue, transport, and release wildlife	3643/2017	28/12/2018	
Fuel Station	Installation License (LI)	2816/2018	Original date: 09/01/2021 Suspension expiry: 26/03/2022 Revocation: 17/12/2021	License to construct fuel station.

Project Aspect	Related License	Permit #	Expiration Date	Description
Concrete Batch Plant	Installation License (LI)	2830/2018	Original date: 11/04/2021 Suspension expiry: 26/03/2022 Revocation: 17/12/2021	License to construct concrete batch plant
Landfill	Installation License (LI)	2869/2018	Original date: 28/10/2021 Suspension expiry: 16/03/2022 Revocation: 05/01/2022	License to construct landfill
Tailings Dam & CIL Pond	Installation License (LI)	2796/2017	Original date: 20/11/2020 Suspension expiry: 18/03/2022 Revocation: 05/01/2022	License to construct TSF dam and CIL Pond #1
	Final Water Permit	3103/2018	01/02/2023	Stream flow regularization tailings storage facility. Total volume: 7.37 Mm^3. Minimum stream flow downstream: 184.9 m^3/day
Access Road	Installation License (LI)	2862/2018	Original date: 19/08/2020 Renewal date: 19/08/2022	Access road was constructed. The application for a permanent authorization will permit maintenance and improvements works. However, for deforestation a specific authorization is required.
	Final Water Permit	5824/2021	29/09/2026	Water catchment at 11 locations along access road.
Industrial Water Supply	Preliminary Water Permit	693/2016	08/09/2018	Water intake in Veados Creek – 4800 m^3/day
Supply Wells	Preliminary Water Permit	655/2016	29/04/2018	Drilling of 8 wells (potable water)
	Preliminary Water Permit	877/2018	29/01/2020	Drilling of 3 wells (potable water)

Project Aspect	Related License	Permit #	Expiration Date	Description
	Final Water Permit	4827/2020	24/09/2025	Groundwater uptake well #1 at Explo. Camp (9.6 m^3/d)
	Final Water Permit	4411/2020	16/06/2025	Groundwater uptake well #2 for Explo. Camp (22.2 m^3/d)
Effluent Release	Preliminary Water Permit	740/2017	09/01/2019	Tocantins river catchment for effluent dilution and discharge of sewage (74.21 m^3/h uptake, 13.35 m^3/h discharge)
	Final Water Permit	In Process	N/A	Authorization for effluent discharge (sewage, oil separators, laundry)
Pit Dewatering	Final Water Permit	2481/2016	03/05/2020	Pit dewatering 350,400 m^3/year

20.3.3 Environmental Compensation

Conservation Units ("UCs") are specially protected areas created and managed by federal, state, or municipal governments with relevant natural characteristics legally instituted by the relevant authority, with conservation purposes and defined limits, under a special administration regime.

Brazilian legislation provides that the licensing of activities with significant environmental impact that may affect a specific UC or its buffer zone ("ZA"), based on an Environmental Impact Study and respective Environmental Impact Report (*Estudo de Impacto Ambiental* (EIA) *e Relatório de Impacto Ambiental* – "EIA/RIMA"), may only be granted after authorization by the authority responsible for managing the UC. Such authorization must be requested by the licensing environmental agency, before the issuance of the first license (in this case, before the LP), to the authority responsible for the for managing the UC, which will manifest itself conclusively after evaluating the environmental studies required within the environmental licensing proceeding.

According to the Project's EIA/RIMA, the Project is: (i) located within APA Tapajós; (ii) some of the mineral rights are located within the buffer zone (ZA) of the Jamanxim National Park; (iii) some of the mineral rights are located close to, but outside of the buffer zones of Crepori and Jamanxim National Forests. BRM confirmed that such UCs are the only ones direct or indirectly affected by the Project.

Applicable law requires BRM to compensate unmitigated negative impacts identified during the environmental licensing proceeding. The relevant authority is responsible for calculating such compensation, which cannot exceed a maximum of 0.5% of the value of the Project's investment. After the percentage is fixed and its destination is defined by the authority during the licensing procedure, a Term of Commitment for Environmental Compensation must be signed.

BRM signed terms of commitment with IDEFLOR-Bio and with ICMBio (environmental authorities responsible for the management of UCs at the state and federal levels, respectively) for environmental compensation. The total compensation amount is BRL 9,720,456.06 (historic value) representing 0,9721% of the estimated costs to implement the Project. From this total amount, BRL 6,804,325.55 (historical value) was employed for direct improvement works in the Utinga State Park, for the benefit of IDEFLOR-Bio. The remaining BRL 2,916,139.52 (historical value) is to be paid in kind, directly to ICMBio, in three annual instalments of equal value, accrued with monetary correction. The amount is to be allocated to the APA Tapajós and the Jamanxim National Park. Two out of the three instalments were paid and there is one instalment due in October 2022.

20.4 Social and Community Impacts

The Project site is located within the Environmental Protection Area (APA) Tapajós. However, the mining activity for this specific APA is authorized with the proper environmental management plan that aim on guaranteeing sustainability. In letter #2012/2009//INCRA/SR(30)G dated July 31st, 2009, the National Institute for Colonization and Agrarian Reform (INCRA) states that the Tocantinzinho project is within a conservation unit (APA Tapajós – Area 2), and has no relation to indigenous areas, quilombolas and / or settlements.

BRM identified several occupants in the area who had requested, via administrative proceedings, the recognition of possession rights before the Federal Government. Most occupants indemnified between 2016 and 2019 have left the area. Five (5) of those occupants, although indemnified, did not leave the area. None of those occupants are located within the Project's footprint except one which must be relocated.

20.4.1 Communities of Interest

Considering that the site is remote from any residence other than those of *garimpeiros* (artisanal miners) in the vicinity of the site, the Project will have minimal impact on any existing communities. The nearest village is Jardim do Ouro which is approximately 85 km away from the site. With its small population of approximately 1,000 inhabitants and minimal infrastructure, the Project's impacts on such village and its surroundings, other than employment, are considered to be minimal. Activity in Jardim do Ouro is expected to increase as this will become the hub for transportation of materials to the site especially during construction.

The Project will employ people from Para State with travel hubs expected to be set up in Moraes Almeida, Jardim do Ouro, Mamoal and Itaituba. An objective of at least 30% local labour from these communities will be targeted and preferably higher. In addition to employment hubs, small businesses that will support the mine are expected to be centered in the region.

Brazauro has developed a series of social investment actions designed to improve the living conditions of individuals in the neighbouring communities. As part of these actions Brazauro provided the following:

- Construction material to build a police station in Moraes Almeida and Jardim do Ouro village.
- Educational material to equip a library and a multidisciplinary laboratory in a Municipal School in Moraes Almeida.
- Logistic support to the teams of the Ministry of Health and the Municipal Health Department of Itaituba for tropical diseases control services in the areas surrounding the Project.

- Training provided to forty-four (44) teachers in Jardim do Ouro and Moraes Almeida communities in order to better equip teachers to impart knowledge to their students (2019).
- Donation of ambulance (4x4 vehicle) to Moraes Almeida community (2020).

The Project will maintain a close relationship with governmental entities in the federal, state and municipal levels that regulate the mining activity.

20.4.2 Future Community Engagement

Several social programs relating to community development were created and all of them are scheduled to be implemented during the Project installation. They include:

- Social communication program and relationship with stakeholders in the Project that includes a plan for development and implementation of the Project's website. The main objective is to contribute to the strengthening of the social dialogue between the community and Brazauro, to give greater support to all activities that involve execution.
- Local development promotion program, which includes a rural economy promotion program.
- Training, qualification and improvement of the workforce, which includes an action plan for labor mobilization.
- Occupational health and safety program.
- Public management support program.
- Environmental education program.

20.5 Monitoring and Reporting

As part of its legal obligations and regardless of the Project's commencement of operations, Brazauro is regularly conducting a monitoring program to enhance its knowledge about the Project's area. Brazauro has hired CLAM in the past to perform the following monitoring:

- Water resources (quality and quantity)
- Air quality
- Noise
- Fauna (terrestrial and aquatic)
- Flora, and
- Erosion

On an annual basis, Brazauro will present an Environmental Information Report ("RIAA") for each license and additional information as required to the environmental agency SEMAS. The RIAA will include monitoring reports and results of tests performed.

20.6 Closure and Reclamation

The closure plan was established to identify environmental, social and economic risks after production will terminate and to determine measures to be implemented during construction, operation and closure. It will be continuously updated and implemented prior to the shutdown of the Project's operations.

The closure strategy was initially prepared by Brandt Amazonia (2011) and updated and complemented by a Plan for Reclamation of Degraded Areas prepared by Brandt in 2017, as part of the environmental control plan presented to the environmental agency.

During the Project construction, deforestation materials and topsoil will be stored in various locations on the site to be used for reclamation. A progressive rehabilitation approach will be used to reduce the long-term closure liability. Actively rehabilitating areas during the operational stage will provide the opportunity to develop and test the most effective methodologies. Area drainage will be modified as required prior to the reclamation process.

A seedling nursery will be built on site to grow native plants that will eventually be planted in reclamation areas.

The current closure strategy is as follows:

- Open Pit: equipment and infrastructure will be removed from the pit which will fill with water. All mine-influenced water that is not suitable for discharge to the environment will be treated. Testing and studies will be carried out to predict the future water quality in the pit.
- Waste Rock Pile: the pile was designed with gentle slope angles so no further sloping will be required to accommodate topsoil placement. Rock capping of the slopes will minimize erosion. Tests for acid rock drainage prediction (ARD) were completed and most of the tests of waste have shown elevated PN/PA ratio in excess of two and mostly in excess of four and are considered non-acid rock generating; in spite of the typical gold sulphide mineralization in the Project site, there is no evidence of pH alteration in the surface waters due to acid drainage generation processes. This can be explained by the significant carbonate content in the ore and waste rocks.

- Tailings Storage Facility: Depending on the results of further geochemical investigations, a permanent wet cover could be implemented in the CTSF to avoid ARD, otherwise an impermeable soil cover will be placed.
- For the FTSF, vegetation will be implemented and drainage through a permanent spillway will control water accumulation.
- Process Plant, Camp, Onsite Infrastructure, Onsite Roads, Onsite Power Line: concepts for closure will depend on future land use. Equipment will be evaluated for potential reuse. Non-reusable equipment and metallic structures will be segregated from other materials to be sold as scrap. Hazardous waste generated during demolition will be segregated and disposed of properly. The areas will be reclaimed by revegetation of native species.
- Monitoring and Maintenance: Monitoring and maintenance will be necessary in the post-closure period to ensure proper revegetation and repair any erosion that may occur, if applicable.

The cost estimate for mine closure is USD 23.5 M which is in line for the scale of the Project. This estimate includes direct and indirect costs as well as monitoring. It is recommended that Brazauro develops a detailed strategy for mine closure aiming at cost estimation and consistent with the Regulatory Norms for Mining (NRM), specifically NRM #20 and NRM #21, which refer to closure minimum aspects and rehabilitation.

21 CAPITAL AND OPERATING COSTS

Life-of-mine project capital costs are estimated to total USD 564.3 million consisting of the following three distinct phases:

- Initial Capital Expenditure – This phase includes all costs to develop the property with a process plant capacity of 4.34 Mtpy of granite ore and up to 4.70 Mtpy when saprolite and garimpeiros tailings are available. Initial capital costs total USD 457.8M net of recoverable taxes and tax credits of $17.9M (including $38.3 million for contingency and $5.5 million in pre-production revenue), which will be expended over a 29-month design, construction, pre-production and commissioning period.
- Sustaining Capital Costs – This phase includes all costs related to the acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations. Sustaining capital costs are estimated to be $82.9 million and does not include contingency.
- Closure Costs – This phase includes all costs related to the closure, reclamation, and ongoing monitoring of the mine after operations. Closure costs total $23.5 million and include a 30% contingency.

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 -10% / +15% range. The base date of the CAPEX estimate is Q1-2022. The initial capital expenditure ("CAPEX") duration is planned over a period of 29 months, assumed from February 2022 to end of June 2024. The CAPEX estimate is aligned with an owner-managed project delivery model.

Expenditures are planned in several currencies with the native currencies retained as part of the estimate. The initial CAPEX estimated is presented in US dollars using an exchanges rate of 5.20 BRL/USD.

The capital cost estimate is a detailed, bottoms-up, built-up effort by major facility and discipline. Each discipline performed material take offs from the basic and detailed engineering drawings and concepts. Each discipline executed a detailed cost build up by cost type consisting of labour, material, construction equipment, consumables, construction materials, and services costs.

Labour and construction equipment costs for the Project were built up in separate analyses, to be included in each individual estimate. Material take offs were also performed to generate the baseline quantities for the Project. Cost estimate by each discipline included complete cost type details and quantities and were consistent with the Project WBS and accumulated in a master estimate summary.

GMS managed the overall engineering effort, with inputs from various external engineering firms.

According to standards established at the outset of the Project, pricing of equipment, material and labour were estimated according to the following guidelines:

- Equipment proposals for major, high value and long lead items were specified and quoted specifically for the Project; equipment prices for minor items were derived from recent projects or from GMS's current database.
- Material prices were based on quotations received from suppliers. and,
- Labour rates were based on quotations received from contractors, labour suppliers and wage surveys completed in northern Brazil.

21.1.2 Tax Analysis

A tax analysis was performed by L&M Advisory, a tax specialist in Brazil, and consisted of the following:

- OPEX and CAPEX: Analysis and application of tax incidences on operating cost items; ICMS balance projection highlighting OPEX and CAPEX credits and tax balances by period over the life of the project; simulation of the Drawback tax benefit.
- CAPEX: Tax review of the main commercial proposals, calculation of factors and application of tax incidences on CAPEX items. This step consisted of a complete analysis of taxation including the basic incidence, tax calculations, application of benefits and tax compensation provided for in the law.
- Applicable taxes were based on the definition of the Tax Classification on all items presented and included in the CAPEX. The Tax Classification is contained in the General Rules of the Common External Tariff (TEC) and also in the Industrialized Products Tax Table (TIPI), defined by current law.
- One opportunity identified and included in the tax planning suggested by L&M for the Project is an Additional Credit on the Basic rate of PIS and COFINS from CAPEX.

21.1.3 Initial CAPEX Summary

The initial CAPEX is estimated at USD 457.8M net of recoverable taxes (USD 13.8M) and tax credits (USD 4.1M) for a total of of USD 17.9M with approximately 59% planned to be spent in local BRL currency. The capital expenditure is summarized in Table 21.1 according to the Level 1 work breakdown structure ("WBS"). WBS Areas 100 to 600 include the Project's direct costs, while WBS Areas 700 to 900 cover indirect costs, owner's costs, and pre-production costs. This amount includes pre-production revenues of approximately USD 5.5M for 3.79 koz of gold recovered during commissioning.

The CAPEX includes a contingency of USD 38.3M, which is 9.1% of the total before contingency or 10.3% of the total excluding major mining equipment. The total hours for construction for the initial CAPEX phase are 4.05M.

Table 21.1: Initial Capital Expenditures Summary

Capital Expenditures (k USD)	Total Pre-Tax	Total Taxes Payable	Total Post-Tax	Recoverable Taxes and Tax Credits	Total Net of RT and Credits
100 – Infrastructure	38,472	5,263	**43,735**	2,822	**40,913**
200 - Power and Electrical	57,666	8,575	**66,241**	4,382	**61,859**
300 – Water Management	12,216	1,750	**13,966**	902	**13,064**
400 – Surface Operations	10,695	1,739	**12,434**	407	**12,026**
500 - Mining	42,811	4,403	**47,214**	189	**47,025**
600 - Process Plant	78,530	10,067	**88,597**	2,561	**86,036**
700 - Construction Indirect	52,676	5,319	**57,995**	2,640	**55,355**
800 - General Services / Owner's Cost	54,611	3,568	**58,179**	1,193	**56,986**
900 - Pre-production, Start-up, Comm.	41,074	8,020	**49,094**	2,808	**46,286**
990 - Contingency	38,295	0	**38,295**	0	**38,295**
Total	**427,046**	**48,704**	**475,750**	**17,904**	**457,845**

21.1.4 Infrastructures

The CAPEX estimate for WBS 100 - Infrastructure is summarized in Table 21.2 net of recoverable taxes. The detailed description of infrastructures is presented in Section 18.

Table 21.2: Infrastructures Capital Expenditures

Capital Expenditures (k USD)	
110 - Roads, Bridges and Fencing	**12,263**
111 - General Earthwork	5,520
112 - Site Roads	3,647
113 - External Access Roads	1,329
114 – Bridges	51
115 – Site Drainage & Trenches	353
116 – Fencing	286
117 – Airstrip	1,077
120 – Mine Infrastructure	**8,297**
122 – Truck Shop (includes Oil & Tire Storage)	5,936
123 – Wash Bay	702
128 – Explosive Magazine	626
129 – Emulsion Building	40
120 – Mine Infrastructure Earthwork Platform	994
130 – Support Infrastructure	**5,324**
131 – Administrative Building	1,546
132 – Site Guard House	198
134 – Warehouse	687
135 – Laydown	287
137 – Assay Lab (Includes Met Lab)	2,535
139 – Greenhouse	70
140 – Camp Facilities	**11,450**
141 – Camp Dormitories	7,084
142 – Kitchen	3,164
143 – Camp Office / Welcome Center/ Laundry	670
145 – Recreational Room	269
146 – Recycling / Sorting Facility	177

Capital Expenditures (k USD)	
147 – Domestic Waste	32
148 – Sports Field	56
160 – Process Plant Infrastructure	**2,599**
161 – Cyanide Storage	36
162 – Leach CIL Satellite Lab	110
164 – Reagents Storage Building	839
165 – Mill Office (with Canteen)	497
166 – Workshop	956
168 – Process Plant Security Gate	140
169 – Control Room	22
170 – Fuel Systems Storage	**980**
171 – Heavy and Light Vehicle	887
172 – Fuel Systems Air Strip	93
Total	**40,913**

21.1.5 Power Supply and Electrical

The CAPEX estimate for WBS Area 200 – Power Supply and Communications is summarized in Table 21.3 net of recoverable taxes.

Table 21.3: Power Supply and Communications Capital Expenditures

Capital Expenditures (k USD)	
210 - Main Power Generation	**35,956**
212 - Power Transmission Line	30,211
213 - Site Main Substation	5,745
220 - Secondary Power Generation	**6,954**
221 - Process Main E-Room	2,787
223 - Camp Power Generation	184
224 - Emergency Power Generation	3,983
240 - Service Electrical Room	**109**
242 - Fuel Storage E-Room	109
250 - Mine Electrical Room	**228**
253 - Explosive Magazine E-room	228
260 - Process Plant Electrical Rooms	**12,977**
261 - Crushing Electrical Room	405
262 - Grinding Electrical Room	9,063
263 - CIL and Detox	2,061
265 - Pumping Station (Barge) E-room	1,244
266 - Refinery Electrical Room	205
270 - MV Distribution O/H Line	**1,394**
280 - Automation Network	**2,117**
281 - Automation Network	2,117
290 - IT Network & Fire Detection	**2,125**
291 - IT Network	514
292 - IT Hardware & Systems	157
293 - Fire Detection Network	1,076
294 - Security Network	188
295 - Server Room	63
296 - Mine Communication System	126
Total	**61,859**

21.1.6 Water Management

A capital expenditures summary for water is presented in Table 21.4 net of recoverable taxes.

Table 21.4: Water Capital Expenditures

Capital Expenditures (k USD)	
310 - Fresh water Intake / Wells	**69**
311 - Fresh Water	69
320 - Water Ponds and Water Management	**3,011**
321 - Water Management	2,101
322 - Process Plant Event Pond	171
323 - CIL Tailing Line Water Collection Ponds	129
324 - Mine Dewatering Transfer Pond	171
325 - Ore Storage Event Pond	171
328 - Flotation Tailing and Dump Sedimentation Pond	268
330 - Domestic Water	**464**
340 - Sewage Water	**639**
350 - Fire Protection	**2,405**
370 - Tailings Storage Facility (TSF)	**6,476**
371 - Flotation Tailings Storage Facility (FTSF)	2,358
372 - CIL Tailings Storage Facilities (CTSF)	3,779
374 - FTSF Spillway	339
Total	**13,064**

21.1.7 Surface Operation

A summary for the capital expenditures for mobile equipment, concrete batch plant and aggregate plant is presented in Table 21.5.

Construction mobile equipment includes purchasing costs for lifting equipment, utility vehicles and specialized construction equipment. Rental costs for equipment required for short periods are included in construction indirects.

Table 21.5: Surface Equipment Capital Expenditures

Capital Expenditures (k USD)	
410 - Surface Operations Equipment	**8,504**
411 - Construction Mobile Equipment	6,256
412 - Process Plant Mobile Equipment	58
414 - G&A Mobile Equipment	2,190
430 - Concrete Batch Plant	**294**
480 - Aggregate Plant	**3,228**
Grand Total	**12,026**

21.1.8 Mining

The capital costs estimate for the mining areas are presented in Table 21.6 net of recoverable taxes. The costs are based on an owner-operated mine fleet.

Table 21.6: Mining Capital Expenditures

Capital Expenditures (k USD)	
540 - Mine Infrastructure	**942**
550 - Mine Equipment	**46,083**
551 - Primary Mining Equipment	28,400
552 - Secondary Mining Equipment	8,013
553 - Ancillary Mining Equipment	4,062
554 - Other Equipment	4,876
555 - FMS/Dispatch/Equipment Communication Systems	365
556 - Mining Software/Survey Equipment	367
Grand Total	**47,025**

21.1.9 Process Plant

The capital costs estimate for the processing areas are presented in Table 21.7 net of recoverable taxes. The detailed description of process plant is presented in Section 17.

Table 21.7: Processing Capital Expenditures

Capital Expenditures (k USD)	
600 - Process Plant	**3,098**
601 - Site Preparation / Road / Berms	1,046
602 - Pipe Rack	1,416
603 - Buried Services	576
604 - ROM pad	61
610 - Comminution	**39,407**
611 - Primary Crusher	6,969
612 - Ore Reclaim	4,409
613 - Primary Grinding	18,046
614 - Pebble Conveyors	948
615 - Secondary Grinding (& Cyclone)	9,035
620 - Gravity & Intensive Leach	**2,483**
621 - Gravity	1,600
622 - Intensive Leach	883
630 - Flotation, Regrind & Concentrate	**11,626**
631 - Rougher & Scavenger Flotation	7,859
632 - Cleaner Flotation	1,797
638 - Flotation Reagent	966
639 - Flotation Services	1,005
640 – CIL & Detox	**8,648**
641 - Pre-Leach	1,584
642 - CIL (Carbon in Leach)	5,044
644 - Cyanide Recovery	996
645 - Detox	1,023
650 - Reagents	**4,487**
651 - Flocculant System	783

Capital Expenditures (k USD)	
652 - Lime System	996
653 - Cyanide System	991
654 - Caustic System	564
655 - SMBS	537
656 - Sulphuric Acid	168
657 - Copper Sulfate	448
660 - Refinery	**6,330**
661 - ADR	1,022
662 - Carbon Regeneration	3,726
663 - Gold Room	1,475
664 - HCL	107
680 - Tailings Management	**5,227**
680 - Tailings Management	352
682 - Tailings Pumping	98
683 - Tailings Pipelines	1,701
684 - Reclaim Water FTSF	2,474
685 - Reclaim Water CTSF	603
690 - Process Plant Services	**4,731**
691 - Plant Air (c/w instrument air)	889
692 - Process Water	1,082
693 - Industrial Water	1,116
694 - Gland Water	406
695 - Emergency Domestic Water	223
696 - Filtered Cooling Treated Water	226
697 - Oxygen Plant	789
Total	**86,036**

21.1.10 Construction Indirects

Construction Indirect costs are presented in Table 21.8 net of recoverable taxes.

Table 21.8: Construction Indirect Capitals

Capital Expenditures (k USD)	
710 - Engineering, CM, PM	**15,784**
711 - Site CM Staff and Consultants	12,049
713 - Surveying	328
714 - QA/QC	1,266
715 - Induction	106
716 - Project Control	2,035
720 - Construction Offices, Facilities & Services	**3,988**
721 - Construction Offices / Trailers	230
722 - Temporary Truck Shop	469
724 - Temporary Laydown Facilities	1,221
725 - Camp Construction Temporary Facilities	829
727 - Site Toilets / Ablution Units	228
728 - Construction Temp Power Distribution	961
729 - Construction Temp water and piping network	50
730 - Shops	**482**
733 - Carpentry, Rebar and Civil	100
737 - Lifting Equipment	383
740 - Construction Equipment & Tools	**13,616**
742 - Rentals	3,586
743 - Operation and Maintenance	2,825
744 - Major Construction Tools	3,246
745 - Small Tools & Consumables	1,623
746 - LOTOTO Team	100
747 - EPP for Construction	1,785
748 - Scaffolding	450
760 - Energy	**11,395**
790 - External Engineering	**10,090**
Total	**55,355**

21.1.11 General Services

Cost estimates for General Services are presented in Table 21.9. General Services include all the support departments that will be staffed and organized to assist during the development stage of the Project and will continue their functions during the operating phase; it includes the following:

- General Management
- Finance and Accounting
- Information Technology
- Environment & Permitting
- Health & Safety
- Supply Chain
- Human Resources & Training
- Business Sustainability
- Security
- Site Services including camp costs and personnel transportation
- Logistics and Insurance

Pricing for freight for the project was obtained from shipping and logistics companies and was applied as a percentage per each WBS. Overall, it represents an average of 11% of the total value of materials and equipment.

Insurance costs of USD 1.4M (included in Finance & Accounting) cover all liabilities and loss coverage during the construction period.

Camp operating expenditures include food, lodging and inland transportation costs for all personnel during pre-production for an estimated total of 425,000 man-days.

Temporary power costs include fuel and maintenance for power consumption at the camp sites and construction sites.

Table 21.9: General Services Expenditures

Capital Expenditures (k USD)	
810 - G&A Departments	**11,627**
811 - General Management	2,283
818 - Accounting and Finances	2,400
817 - IT & Telecommunications Service	1,742
814 - Supply Chain	2,037
813 - HR & Training	2,018
816 - Community Social Responsibility (CSR)	1,146
815 - Security	**1,602**
820 - Logistics / Taxes / Insurance	23,451
822 - Sea Freight (International) and Offshore Freight	6,537
823 - Air Freight	496
824 - In-Country Land Freight	14,077
825 - Customs, Taxes and Duties	2,341
830 - Camps and Site Services (Operating Expenses)	**16,534**
840 - Environmental and Permitting	**2,134**
850 - Health and Safety	**1,638**
Total	**56,986**

21.1.12 Pre-production and Commissioning Expenditures

Pre-production and commissioning expenditures are presented in Table 21.10. The costs cover mine management and technical services to supervise all earthworks and pre-production activities. Mine access roads, pit and dump preparation are included as well as the stripping and ore stockpiling. Some 9,653,000 tonnes of ore and waste will be mined during pre-production.

The process plant and pre-production includes initial fills as well as salaries, reagents and fuel during the commissioning and ramp-up period to commercial production. Staffing and training of mill personnel is planned progressively in the three-month period before commissioning. Some 362,000 of ore will be processed during commissioning. Pre-production revenue of USD 5.5M net of royalties and selling costs was included from selling 3.8 koz of gold at a price of USD 1,600/oz of gold.

Table 21.10: Pre-production and Commissioning Expenditures

Capital Expenditures (k USD)	
910 - Mining Pre-Prod	41,160
911 - Mine Administration Labour	7,598
912 - Mine Engineering Labour	3,566
913 - Mine Geology	252
914 - Mine Operations Pre-prod	16,568
916 - Mine Maintenance	5,442
919 - Mine Support Operations	3,499
Total Non-Recoverable Taxes	4,235
950 - Process Plant Pre-Production	**2,914**
952 - Process Plant Pre-Production	5,142
954 - Vendor Reps	3,250
955 - Pre-Prod Revenue	- 5,478
960 - First Fill, Spares & Consumables	**2,212**
961 - Spare Parts Capital (3.5% of Mech cost)	1,588
965 - First Fill	624
990 - Contingency	**38,295**
Total	**84,581**

21.1.13 Contingency

Contingency has been assigned to the cost estimate per area and discipline basis as a deterministic allowance by assessing the level of confidence of the scope definition, supply cost and installation cost, and then applying Monte Carlo iteration analysis. The overall recommended contingency resulted in 9% of direct and indirect expenditures.

21.2 Sustaining Capital

Sustaining capital is presented in Table 21.11. Sustaining capital for the mine includes additional equipment purchases for a total of USD 50.0M. Major equipment repairs were kept in the operating costs.

Additional work is required for raising the main embankment of the flotation tailings storage facility ("FTSF") and the construction of the second pond as part of the CIL tailings storage facility ("CTSF"). The continued raising of the FTSF will be completed by the mine operations team with fill material from the open pit mine. An effluent water treatment plant will be constructed in Year 2 to treat water from the CTSF prior to discharge. Tailings and water management sustaining capital is estimated at USD 16.7M.

Table 21.11: Sustaining Capital Costs

Cost by Area (M USD)	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Mobile Equipment	**50.04**	**19.53**	**10.99**	**5.35**	**1.13**	**1.12**	**3.85**	**2.11**	**5.15**	**0.72**	**0.07**	**0.01**
Primary	**29.22**	12.47	10.02	5.05	-	-	1.68	-	-	-	-	-
Secondary	**7.60**	1.04	-	-	0.77	0.59	1.04	-	3.77	0.38	-	-
Ancillary	**3.52**	1.05	-	-	-	-	-	2.00	0.47	-	-	-
Others	**5.51**	1.94	0.22	0.19	0.37	0.28	1.09	0.11	0.90	0.34	0.07	0.01
Fleet Mgmt. System	**4.18**	3.02	0.76	0.11	-	0.25	0.04	-	-	-	-	-
Truckshop Tooling	**-**	-	-	-	-	-	-	-	-	-	-	-
Tailings & Water Mgmt.	**16.66**	**3.55**	**7.65**	**2.35**	**-**	**3.11**	**-**	**-**	**-**	**-**	**-**	**-**
Deforestation	**2.59**	-	2.59	-	-	-	-	-	-	-	-	-
Tailings FTSF	**6.25**	-	1.88	1.88	-	2.48	-	-	-	-	-	-
Tailings CTSF	**2.14**	2.14	-	-	-	-	-	-	-	-	-	-
Effluent Treatment Plant	**2.36**	0.71	1.65	-	-	-	-	-	-	-	-	-
Construction Indirects	**3.33**	0.71	1.53	0.47	-	0.62	-	-	-	-	-	-
Process Plant	-	-	-	-	-	-	-	-	-	-	-	-
Process Plant Allowance	**4.68**	-	0.84	0.48	0.48	0.48	0.48	0.48	0.48	0.48	0.48	-
Taxes	-	-	-	-	-	-	-	-	-	-	-	-
Non-recoverable Taxes	**11.54**	4.50	2.53	1.23	0.26	0.26	0.89	0.49	1.19	0.17	0.01	0.00
Total incl. NRT	**82.92**	**27.59**	**22.01**	**9.42**	**1.87**	**4.97**	**5.22**	**3.08**	**6.81**	**1.37**	**0.56**	**0.01**

21.3 Closure Costs

The closure costs are estimated to be USD 23.5M as summarized in Table 21.12. Closure costs would cover the following activities:

- Removal/demolition of facilities and shaping, re-vegetating of the pads
- Shaping of service roads, mine haul roads and airstrip
- Fencing, shaping and re-vegetating of pits and borrow areas
- Shaping and re-vegetating of waste dumps, stockpiles and landfills
- Final water treatment of the CIL tailings storage pond decant water and construct permanent spillway
- Shaping and re-vegetating the flotation and CIL tailings storage facilities
- Monitoring (cost for analyzing samples and labour)

Table 21.12: Closure Cost Summary

Closure Cost	Cost (k USD)
Owner Management Costs	589
Dismantling and Demolition	6,261
Pit Closure	194
Waste Piles Recovery	2,360
Dams Recovery	5,997
Accesses Recovery	892
Industrial and Civil Areas Recovery	553
Old Mining Site and Ore Pile Recovery	784
Environmental Monitoring	472
Contingency	5,431
Sub-Total (Pre-Tax)	**23,533**
Non-Recoverable Taxes	0
Total (Incl. NRT)	**23,533**

The final waste dump closure will occur after operations; however progressive rehabilitation of completed lifts will occur continuously during operations. Mining of the pit will be completed approximately 1.5 years before the end of processing of low-grade stockpiles. The waste dumps will be rehabilitated during this period with mine equipment will be used for shaping, scarifying and material placement.

Growth media will be rehandled from an area in the waste dump and placed on areas with exposed rock. Areas without vegetation will be hydroseeded after which local plant species grown in a nursery will be planted to revegetate the area.

Removal of facilities consists of taking steel structures apart and demolishing the concrete foundations. Process plant and some mine major equipment will have some salvage value after operations estimated at USD 12.6M which is not included in the closure costs but taken into account in the cash flow model in Section 22.

21.4 Operating Costs

Operating Costs are summarized in Table 21.13 and presented by year in Table 21.14. The operating costs include mining, processing, general services and administration ("G&A"), gold transportation and refining and royalties. The average LOM operating cost is USD 623/oz of gold or USD 23.68/t milled. The average LOM all-in sustaining cost ("AISC") is USD 681/oz of gold or USD 25.88/t milled. Operating costs include non-recoverable taxes.

Table 21.13: Operating Costs Summary

Cost Summary	Total LOM Cost (M USD)	Unit Cost (USD/t milled)	Cost per oz (USD/oz)
Mining	459	9.51	250
Processing	427	8.83	233
G&A	151	3.13	82
Total Site Costs	**1,037**	**21.48**	**565**
Transport & Refining	18	0.38	10
Private Royalty (1.5% NSR)	44	0.91	24
Govt. Royalty (1.5% GOR)	44	0.91	24
Total Operating Costs	**1,143**	**23.68**	**623**
Sustaining Capital	83	1.72	45
Closure & Reclamation	24	0.49	13
AISC	**1,250**	**25.88**	**681**

Table 21.14: Total Operating Costs Summary by Year

Operating Cost Summary	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13+
Production Highlights														
Tonnage milled (kt)	**48,284**	2,235	4,705	4,705	4,705	4,705	4,705	4,705	4,705	4,552	4,340	4,222		
Tonnage mined (kt)	**194,939**	7,475	25,918	26,000	25,000	27,500	24,782	21,387	19,625	12,000	5,253	-		
Recovered gold (koz)	**1,834**	93	203	163	206	233	175	137	180	209	163	70		
Operating Costs (M USD)														
Mining	**459.37**	17.05	47.41	57.56	50.55	56.68	56.23	57.61	53.32	37.47	19.88	5.61	-	-
Processing	**426.55**	19.65	41.16	41.16	41.16	41.16	41.16	41.16	41.16	40.49	39.55	38.75	-	-
G&A	**151.18**	7.89	15.74	15.71	15.73	15.74	15.74	15.76	15.49	14.32	10.26	8.80	-	-
Transport & Refining	**18.34**	0.93	2.03	1.63	2.06	2.33	1.75	1.37	1.80	2.09	1.63	0.70	-	-
Private Royalty	**43.75**	2.22	4.85	3.89	4.91	5.56	4.19	3.28	4.29	5.00	3.89	1.68	-	-
Govt. Royalty	**44.02**	2.24	4.88	3.92	4.94	5.59	4.21	3.30	4.32	5.03	3.92	1.69	-	-
Total Operating Cost	**1,143.21**	49.98	116.05	123.87	119.35	127.05	123.28	122.48	120.37	104.40	79.13	57.24	-	-
Sustaining Capital	**82.92**	27.59	22.01	9.42	1.87	4.97	5.22	3.08	6.81	1.37	0.56	0.01	-	-
Closure Cost	**23.53**	-	-	-	-	-	-	-	-	-	1.25	2.53	9.08	10.67
AISC	**1,249.66**	77.57	138.07	133.29	121.22	132.02	128.50	125.57	127.19	105.77	80.94	59.78	9.08	10.67
Unit Operating Costs														
Mining Cost / t mined	**2.36**	2.28	1.83	2.21	2.02	2.06	2.27	2.69	2.72	3.12	3.78	-		
Process Cost / t milled	**8.83**	8.79	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.89	9.11	9.18		
Total OPEX / t milled	**23.68**	22.36	24.67	26.33	25.37	27.00	26.20	26.03	25.58	22.93	18.23	13.56		
Total OPEX / oz	**623**	537	571	759	579	545	703	891	669	498	485	812		
AISC / oz	**681**	833	679	817	588	567	732	914	707	505	496	848		

21.4.1 Mining Costs

A detailed mine cost build up was constructed from basic cost elements such as salary 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 BRL 4.25/L which is inclusive of transport costs to site and non-recoverable taxes (Table 21.15).

Table 21.15: Diesel Fuel Price

Diesel Fuel Price (S500)	BRL/L	USD/L
Net Price FOB	3.2692	0.6287
Freight	0.1865	0.0359
PIS	0.0773	0.0149
COFINS	0.3561	0.0685
ICMS	0.7966	0.1532
Total Price incl. Taxes	4.6857	0.9011
Total + NRT	**4.2523**	**0.8177**

The average LOM operating cost is USD 2.36/t mined including rehandling cost. Fuel, labour, and maintenance parts are the dominant cost centers representing respectively 23%, 23% and 18% of pre-tax operating costs. Table 21.16 presents the breakdown of mining costs by department while Table 21.17 presents the major cost elements for the mine.

Table 21.16: Mining Cost Summary

Mining Costs (M USD)	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Mine Management	84.27	4.19	9.55	9.75	9.36	9.42	9.39	9.50	9.41	7.77	4.15	1.77
Technical Services	18.95	1.86	2.56	2.21	1.89	1.88	1.91	1.89	1.89	1.78	0.80	0.29
Grade Control	2.12	0.06	0.23	0.25	0.28	0.23	0.21	0.20	0.22	0.22	0.16	0.07
Production Drilling	21.23	0.53	2.20	3.07	2.53	2.85	2.84	2.63	2.38	1.63	0.57	-
Blasting	36.89	1.14	4.07	5.13	4.51	4.37	4.83	4.31	4.12	3.00	1.40	-
Pre-Split D&B	3.98	0.12	0.37	0.40	0.53	0.49	0.58	0.56	0.35	0.37	0.20	-
Loading	39.15	1.32	5.03	4.83	4.66	6.43	5.20	3.57	3.68	2.47	1.35	0.62
Hauling	128.55	2.96	10.31	17.35	13.81	16.33	15.88	19.45	17.32	10.18	4.96	-
Dump Maintenance	15.11	0.91	1.96	1.72	1.89	1.73	1.83	1.85	1.56	1.18	0.43	0.05
Road Maintenance	10.19	0.43	1.13	1.16	1.17	1.17	1.23	1.23	1.03	0.98	0.56	0.10
Dewatering	1.30	0.04	0.11	0.11	0.13	0.14	0.14	0.14	0.14	0.14	0.14	0.05
Support Equipment	29.48	1.36	3.43	3.45	2.90	3.42	3.29	3.19	3.46	2.88	1.65	0.43
Rehandling	6.62	0.22	0.42	0.28	0.15	0.33	1.16	1.02	0.46	0.10	0.93	1.56
Sub-Total Pre-Tax	**397.86**	**15.16**	**41.36**	**49.72**	**43.81**	**48.79**	**48.48**	**49.55**	**46.03**	**32.69**	**17.31**	**4.96**
Non-Rec. Tax	**61.51**	**1.90**	**6.05**	**7.85**	**6.74**	**7.88**	**7.75**	**8.06**	**7.29**	**4.78**	**2.57**	**0.65**
Total Mining Cost	**459.37**	**17.05**	**47.41**	**57.56**	**50.55**	**56.68**	**56.23**	**57.61**	**53.32**	**37.47**	**19.88**	**5.61**
Unit Cost ($/t Mined)	**2.36**	**2.28**	**1.83**	**2.21**	**2.02**	**2.06**	**2.27**	**2.69**	**2.72**	**3.12**	**3.78**	**-**

Table 21.17: Mining Costs by Cost Element

Mining Costs (M USD)	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Fuel	**90.90**	3.23	9.90	10.69	10.58	11.23	11.17	11.02	10.63	7.41	3.91	1.15
Labour	**87.56**	4.41	9.81	10.11	9.79	9.84	9.84	9.93	9.84	8.25	4.11	1.66
Maintenance Parts	**74.67**	2.87	8.44	8.66	8.52	9.48	9.20	8.72	8.41	5.99	3.18	1.20
Major Components	**41.94**	0.30	1.68	7.23	3.05	6.38	5.71	7.79	5.45	2.59	1.71	0.06
Explosives Products	**29.34**	0.83	3.22	4.27	3.66	3.53	3.99	3.48	3.27	2.12	0.98	-
Tires	**25.23**	0.78	2.49	2.91	2.96	2.98	3.08	3.29	3.23	2.14	1.09	0.29
External Services	**12.77**	0.60	1.67	1.56	1.33	1.43	1.41	1.39	1.38	1.18	0.64	0.18
Drilling Tools	**10.80**	0.32	1.23	1.54	1.32	1.34	1.49	1.30	1.19	0.73	0.34	-
Oils & Lubes	**6.28**	0.22	0.68	0.73	0.74	0.77	0.77	0.77	0.75	0.52	0.27	0.07
Other	**18.37**	1.60	2.26	2.01	1.88	1.83	1.83	1.87	1.90	1.76	1.09	0.34
Sub-Total Pre-Tax	**397.86**	**15.16**	**41.36**	**49.72**	**43.81**	**48.79**	**48.48**	**49.55**	**46.03**	**32.69**	**17.31**	**4.96**

Mining Costs (%)	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Fuel	**22.8%**	21.3%	23.9%	21.5%	24.2%	23.0%	23.0%	22.2%	23.1%	22.7%	22.6%	23.2%
Labour	**22.0%**	29.1%	23.7%	20.3%	22.3%	20.2%	20.3%	20.0%	21.4%	25.2%	23.7%	33.4%
Maintenance Parts	**18.8%**	18.9%	20.4%	17.4%	19.4%	19.4%	19.0%	17.6%	18.3%	18.3%	18.4%	24.3%
Major Components	**10.5%**	2.0%	4.1%	14.5%	7.0%	13.1%	11.8%	15.7%	11.8%	7.9%	9.9%	1.2%
Explosives Products	**7.4%**	5.5%	7.8%	8.6%	8.3%	7.2%	8.2%	7.0%	7.1%	6.5%	5.7%	-
Tires	**6.3%**	5.2%	6.0%	5.9%	6.7%	6.1%	6.4%	6.6%	7.0%	6.5%	6.3%	5.9%
External Services	**3.2%**	4.0%	4.0%	3.1%	3.0%	2.9%	2.9%	2.8%	3.0%	3.6%	3.7%	3.7%
Drilling Tools	**2.7%**	2.1%	3.0%	3.1%	3.0%	2.7%	3.1%	2.6%	2.6%	2.2%	2.0%	-
Oils & Lubes	**1.6%**	1.5%	1.6%	1.5%	1.7%	1.6%	1.6%	1.6%	1.6%	1.6%	1.5%	1.5%
Other	**4.6%**	10.5%	5.5%	4.0%	4.3%	3.8%	3.8%	3.8%	4.1%	5.4%	6.3%	6.8%
Sub-Total Pre-Tax	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**

21.4.2 Processing Costs

The processing flowsheet is explained in detail in Section 17 – Recovery Methods. The processing operating cost estimate is for a plant capable of treating 4,705 ktpy (or 12,890 t/d) when saprolite and tailings are available for blending and 4,340 ktpy (or 11,890 t/d) when only fresh rock ore is fed to the mill. The grinding media and reagents consumptions and costs have been determined for the various ore types (Table 21.18). These costs were then applied to the mix of ore types detailed in the mill feed schedule as dictated by the mine plan.

The components to the process plant operating cost consist of labour, power, wear parts, maintenance parts, grinding media, reagents and general plant costs. Total annual processing costs, including power for the plant and site, are presented in Table 21.19.

Labour costs were developed based on the process plant manpower summarized in Section 17 and current market rates for personnel salaries and wages.

Power consumption estimates have been adopted from the electrical load analysis. Power consumption is estimated at 34.8 kWh/t for fresh rock and 25.0 kWh/t for saprolite and tailings which requires minimal crushing and grinding effort. In addition, there is a base load of 2.6 MW for the balance of plant. The total estimated power consumption per annum is 174,190 MWh when processing fresh rock. The LOM power consumptions including the balance of plant (BOP) is 39.3 kWh/t as presented in Table 21.20

Power supply costs were based on the ANEEL ("National Agency of Electric Energy") and the power market for the free consumer, inclusive of non-recoverable taxes, of BRL 405.18/MWh or USD 0.078/kWh. Detailed pricing is presented in Table 21.21.

Table 21.18: Process Plant Consumables

Plant Consumables	Net Unit Pricing	Units	Saprolite	Tailings	Fresh Rock
Reagents		**Consumption**			
Sodium Cyanide (100% NaCN basis)	$2,300/t	g/t	476	176	282
Copper Sulphate (100% $CuSO_4$ basis)	$3,480/t	g/t	181	67	107
Hydrochloric acid (100% HCl basis) - 32%	BRL 7,072/t	g/t	49	18	29
Lime (100% CaO basis) - 92%	BRL 748/t	g/t	103	38	61
Sodium Hydroxide (100% NaOH basis)	BRL 15,812/t	g/t	27	10	16
Sodium Isobutyl Xanthate (100% SIBX basis)	$1,900/t	g/t	171	63	101
Flocculant	$4,100/t	g/t	6.5	2.4	3.8
Frother DF250 (100% basis)	$2,400/t	g/t	103	38	61
Activated Carbon (granular form)	$3,100/t	g/t	0.0169	0.0062	0.0100
Sodium Metabisulphite (100% SMBS basis)	$610/t	g/t	342.8	126.3	203.0
Grinding Media					
SAG Mill grinding media (5”)	$1,395/t	g/t	24	24	474
Ball Mill grinding media (3”)	$1,298/t	g/t	175	175	701
Gold Room Reagents					
Anhydrous Borax	$1,400/t	g/t	1.7	1.7	1.7
Silica	$600/t	g/t	0.8	0.8	0.8
Sodium Nitrate	$2,000/t	g/t	0.4	0.4	0.4
Soda Ash	$660/t	g/t	0.4	0.4	0.4
Water Treatment Plant Reagents					
Hydrated Lime	BRL 749/t	g/t	135	135	135
Flocculant	$4,100/t	g/t	0.3	0.3	0.3
Sulphuric Acid	BRL 734/t	g/t	0.2	0.2	0.2

Table 21.19: Processing Cost Summary

Processing Costs (M USD)	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Labour	**41.24**	1.96	3.93	3.93	3.93	3.93	3.93	3.93	3.93	3.93	3.93	3.93
Power	**107.99**	5.02	10.45	10.45	10.45	10.45	10.45	10.45	10.45	10.23	9.93	9.69
Wear Parts	**10.57**	0.50	0.99	0.99	0.99	1.00	1.00	1.00	1.00	1.02	1.04	1.04
Maintenance Parts	**18.50**	0.88	1.76	1.76	1.76	1.76	1.76	1.76	1.76	1.76	1.76	1.76
Grinding Media	**76.04**	3.54	7.29	7.29	7.29	7.29	7.29	7.29	7.29	7.25	7.19	7.00
Reagents	**108.00**	5.00	10.52	10.52	10.52	10.52	10.52	10.52	10.52	10.18	9.71	9.44
General Plant	**8.42**	0.14	0.83	0.83	0.83	0.83	0.83	0.83	0.83	0.83	0.83	0.83
Sub-Total Pre-Tax	**370.77**	**17.05**	**35.77**	**35.77**	**35.77**	**35.78**	**35.78**	**35.78**	**35.78**	**35.20**	**34.39**	**33.70**
Non-Recoverable Taxes	**55.78**	**2.60**	**5.38**	**5.38**	**5.38**	**5.38**	**5.38**	**5.38**	**5.38**	**5.29**	**5.16**	**5.05**
Total Plant Cost (incl. NRT)	**426.55**	**19.65**	**41.16**	**41.16**	**41.16**	**41.16**	**41.16**	**41.16**	**41.16**	**40.49**	**39.55**	**38.75**
Unit Plant Cost ($/t milled)	**8.83**	**8.79**	**8.75**	**8.75**	**8.75**	**8.75**	**8.75**	**8.75**	**8.75**	**8.89**	**9.11**	**9.18**

Table 21.20: Power Consumption

Power	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Annual Consumption (MWh)	**1,895,249**	88,165	183,329	183,329	183,329	183,329	183,329	183,329	183,329	179,508	174,190	170,084
Average Power Consumption (MW)	**20.6**	20.1	20.9	20.9	20.9	20.9	20.9	20.9	20.9	20.5	19.9	19.4
Annual Power Cost (k USD)	**107,993**	5,024	10,446	10,446	10,446	10,446	10,446	10,446	10,446	10,229	9,925	9,692
Unit Consumption (kWh/t)	**39.3**	39.4	39.0	39.0	39.0	39.0	39.0	39.0	39.0	39.4	40.1	40.3

Table 21.21: Power Electricity Price

Grid Electricity Price		Free Market
Net Price	R$/MWh	296.34
PIS/COFINS Rate	%	9.25%
PIS/COFINS	R$/MWh	30.20
PIS/COFINS Included	R$/MWh	326.54
ICMS Rate	%	25.00%
ICMS	R$/MWh	108.85
Net + Non-Recoverable Tax	R$/MWh	405.18
Gross Price (all taxes included)	R$/MWh	435.39
Exchange Rate	R$/USD	5.20
Net + Non-Recoverable Tax	USD/MWh	77.9
Net + Non-Recoverable Tax	USD/kWh	0.078

Costs were for maintenance parts based on a percentage of initial capital costs for mechanical plant equipment per unit operations. An overall percentage of 3.2% was calculated.

Reagents, wear parts and grinding media costs were developed for all reagents and consumables required for the operation of the process plant including the effluent treatment plant with updated pricing which includes transportation costs to site. Reagent consumption was based on historical metallurgical test work results and available information from similar operations. The reagent consumptions for chemicals used in the acid wash and elution/electrowinning process have been determined from industry averages. Wear parts and grinding media consumption were based on comminution modeling and benchmark information. Table 21.18 present the consumption and unit costs of consumables and reagents by ore type that are applied proportionally to the mine plan to develop annual costs.

21.4.3 General and Administration

General Services include general management, accounting and finance, IT, environmental and social management, human resources, supply chain, health and safety, security and site services which includes camp and personnel transportation costs. In most cases, these services represent fixed costs for the site. The General Services costs exclude certain costs such as transport and refining of final products and environmental rehabilitation costs.

A summary of G&A costs is presented in Table 21.22. As the open pit is depleted 1.5 years before milling operations are completed, there are fewer people on site resulting in lower G&A costs.

Table 21.22: General Services & Administration Cost Summary

G&A Costs (M USD)	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
General Management	**6.66**	0.38	0.75	0.63	0.63	0.63	0.63	0.63	0.63	0.60	0.57	0.57
Finance	**12.93**	0.66	1.32	1.32	1.32	1.32	1.32	1.32	1.31	1.28	0.89	0.88
Information Technology	**7.29**	0.31	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.50	0.48
Environment & Permitting	**6.86**	0.35	0.71	0.71	0.71	0.71	0.71	0.71	0.71	0.68	0.51	0.39
Health & Safety	**8.81**	0.48	0.97	0.97	0.97	0.97	0.97	0.97	0.80	0.69	0.52	0.48
Supply Chain	**22.88**	1.15	2.31	2.31	2.31	2.31	2.31	2.31	2.27	2.16	1.81	1.64
HR & Training	**6.34**	0.36	0.70	0.70	0.70	0.70	0.70	0.70	0.70	0.54	0.26	0.30
Business Sustainability	**3.57**	0.17	0.34	0.34	0.34	0.34	0.34	0.34	0.34	0.34	0.33	0.31
Security	**7.92**	0.40	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.78	0.64	0.50
Site Services	**66.25**	3.54	6.93	7.03	7.04	7.05	7.05	7.07	7.02	6.35	4.07	3.11
Sub-Total Pre-Tax	**149.50**	**7.81**	**15.57**	**15.55**	**15.56**	**15.58**	**15.58**	**15.60**	**15.33**	**14.16**	**10.10**	**8.66**
Non-Recoverable Taxes	**1.67**	**0.08**	**0.16**	**0.16**	**0.16**	**0.16**	**0.16**	**0.16**	**0.16**	**0.16**	**0.16**	**0.15**
Total G&A Costs (incl. NRT)	**151.18**	**7.89**	**15.74**	**15.71**	**15.73**	**15.74**	**15.74**	**15.76**	**15.49**	**14.32**	**10.26**	**8.80**
G&A Unit Cost ($/t milled)	**3.13**	**3.53**	**3.34**	**3.34**	**3.34**	**3.35**	**3.35**	**3.35**	**3.29**	**3.15**	**2.36**	**2.08**

22 ECONOMIC ANALYSES

This section presents all elements of the economic model which principally consist of metal production and revenues, royalty agreements, operating costs, capital costs, sustaining capital, salvage value, closure and reclamation costs, taxation and net Project cash flow.

The economic analysis is carried out in real terms (i.e., without inflation factors) in Q1 2022 United-States dollars. The economic results are calculated as of the beginning of Year -2, which corresponds to the start of the 29 months of the initial CAPEX phase, including engineering and procurement, with all prior costs treated as sunk costs but considered for the purposes of taxation calculations. The economic results such as the net present value ("NPV") and internal rate of return ("IRR") are calculated on an annual basis.

22.1 Assumptions

The key assumptions materially influencing the economics of the Project include:

- Gold price in USD/oz
- Brazilian Real (BRL) to United States dollar (USD) exchange rate ("BRL/USD")

22.1.1 Gold Price

The price of gold is the largest single factor in determining profitability and cash flow from operations. The financial performance of the project is closely linked to the price of gold. Historical gold prices are shown in Table 22.1, with quarterly and yearly trailing averages shown in Figure 22.1.

Reserves and resources have been modelled at a gold price of USD 1,400/oz. Project economics have been assessed at a Base Case gold price of USD 1,600/oz, which is below the three-year historical average gold price of USD 1,664/oz as of December 31, 2021. Project economics at a range of gold prices are assessed as part of project sensitivity analysis.

Figure 22.1: Three Year Historical Gold Prices

Sources: FastMarkets, ICE Benchmark Administration, Thomson Reuters, World Gold Council

Table 22.1: Historical Gold Prices - Quarterly and Yearly Trailing Averages

Calendar Period	Quarterly Average	Historical Trailing Avg. 3 Year	Historical Trailing Avg. 2 Year	Historical Trailing Avg. 1 Year
Q1 2019	1,304	1,654		
Q2 2019	1,309			
Q3 2019	1,472			
Q4 2019	1,481			
Q1 2020	1,583		1,784	
Q2 2020	1,711			
Q3 2020	1,909			
Q4 2020	1,874			
Q1 2021	1,794			1,799
Q2 2021	1,816			
Q3 2021	1,790			
Q4 2021	1,795			

Sources: FastMarkets, ICE Benchmark Administration, Thomson Reuters, World Gold Council

22.1.2 Exchange Rates

The local Brazilian Real exchange rate is the key exchange rate given that a majority of expenditures will be incurred in the local currency. This exchange rate is key to converting costs into USD currency. The BRL has weakened in recent years as shown in Table 22.2 and Figure 22.2. Consensus forecasts, as summarized in Table 22.3, indicate that the BRL/USD exchange rate will stabilize at current levels. The Project economic analysis assumes an average rate of 5.2 BRL/USD. Other exchange rates used in the CAPEX include the CAD/USD of 1.30.

Table 22.2: Historical Brazilian Real Exchange Rate Statistics

Historical Statistics	Brazilian Real (BRL/USD)		
	Average	High	Low
Q4 2021 (3 months)	5.53	5.75	5.25
H2 2021 (6 months)	5.35	5.75	4.93
2021 (12 months)	5.37	5.81	4.93
2020-2021 (2 years)	5.24	5.92	4.02
2019-2021 (3 years)	4.81	5.92	3.65

Table 22.3: Consensus Brazilian Real Exchange Rate Forecasts

Consensus Forecast	Brazilian Real (BRL/USD)		
	Median	High	Low
2022	5.70	6.10	5.25
2023	5.50	6.15	5.27
2024	5.45	6.30	4.95
2025	5.30	6.45	4.95
2026	5.60	6.60	5.00

Figure 22.2: Brazilian Real History

Source: S&P Capital IQ.

22.2 Metal Production and Revenues

Gold production over the Project life is 1,838 koz based on an average recovery of 90%. Recovered gold during pre-production is 3.8 koz, generating estimated revenue of USD 5.9M (net of transportation, refining and royalty costs) which offsets pre-production CAPEX. A total of 1,834 koz will be recovered during operations and will generate gross revenue is USD 2,935M. A total of 1.5 koz of inventory will be built up in circuit and is accounted for as working capital.

Table 22.4: Milling Production Schedule Summary

Process Plant Schedule[1]		Total	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Total Ore															
Total Ore Milled	000 t	**48,676**	-	-	2,627	4,705	4,705	4,705	4,705	4,705	4,705	4,705	4,552	4,340	4,222
Head Grade	g/t	**1.31**	-	-	1.33	1.48	1.19	1.51	1.71	1.29	1.02	1.33	1.58	1.29	0.57
Recovered Gold	000 oz	**1,838**	-	-	97	203	163	206	233	175	137	180	209	163	70
Saprolite Ore															
Saprolite Milled	000 t	**1,581**	-	-	-	-	-	46	228	365	365	365	212	-	-
Head Grade	g/t	**1.03**	-	-	-	-	-	1.03	1.03	1.03	1.03	1.03	1.03	-	-
Recovered Gold	000 oz	**37**	-	-	-	-	-	1	5	9	9	9	5	-	-
Tailings Ore															
Tailings Milled	000 t	**1,308**	-	-	122	365	365	319	137	-	-	-	-	-	-
Head Grade	g/t	**1.11**	-	-	1.22	1.10	1.12	1.12	0.96	-	-	-	-	-	-
Recovered Gold	000 oz	**40**	-	-	4	11	11	10	4	-	-	-	-	-	-
Fresh Rock Ore															
Fresh Rock Milled	000 t	**45,787**	-	-	2,505	4,340	4,340	4,340	4,340	4,340	4,340	4,340	4,340	4,340	4,222
Head Grade	g/t	**1.32**	-	-	1.34	1.51	1.20	1.54	1.77	1.32	1.02	1.35	1.61	1.29	0.57
Recovered Gold	000 oz	**1,761**	-	-	93	192	152	195	224	167	129	171	204	163	70
EOP SP[2] Inventory	000 t		646	3,122	4,054	4,063	4,651	8,673	8,190	5,357	2,907	2,814	4,755	4,222	-

Notes: 1: Includes pre-production tonnage, 2: EOP SP = End of Period Stockpile Inventory

22.3 Royalties

There are private royalties and a Federal Government royalty on the Project.

22.3.1 Government Royalty

The Federal Constitution of Brazil has established that the states, municipalities, Federal districts and certain agencies of the federal administration are entitled to receive royalties for the exploitation of mineral resources by holders of mining concessions. The royalty rate for gold, referred to as CFEM ("*Compensação Financeira pela Exploração de Recursos Minerais*"), is 1.5% of gross sales of the mineral product resulting in USD 44M in payments for the life-of-mine.

22.3.2 Private Royalties

A Net Smelter Royalty of 2.75% and 0.75% was due to Osisko Gold Royalties Ltd. ("Osisko") and Metalla Royalty & Streaming Ltd. ("Metalla") respectively, on future revenues derived from the sale of gold mined from the Project.

GMIN exercised the first of its two Buy-Down Rights (as defined in the relevant agreement) to reduce Osisko's royalty by 1.0%, to 1.75%, and paid a cash consideration of USD 2.0M. GMIN's second Buy-Down Right to further reduce Osisko's royalty by an additional 1.0% to 0.75%, can be exercised for a cash consideration of USD 3.5M within 30 days of a construction decision in respect of the Project.

Once the second Buy-Down Right is exercised the private royalties will be reduced to 1.5% NSR. The Buy-Down Right is not included in the CAPEX presented in Section 21; however, it is included in the economic analysis calculations. Over the LOM, private royalty payments total USD 43.7M.

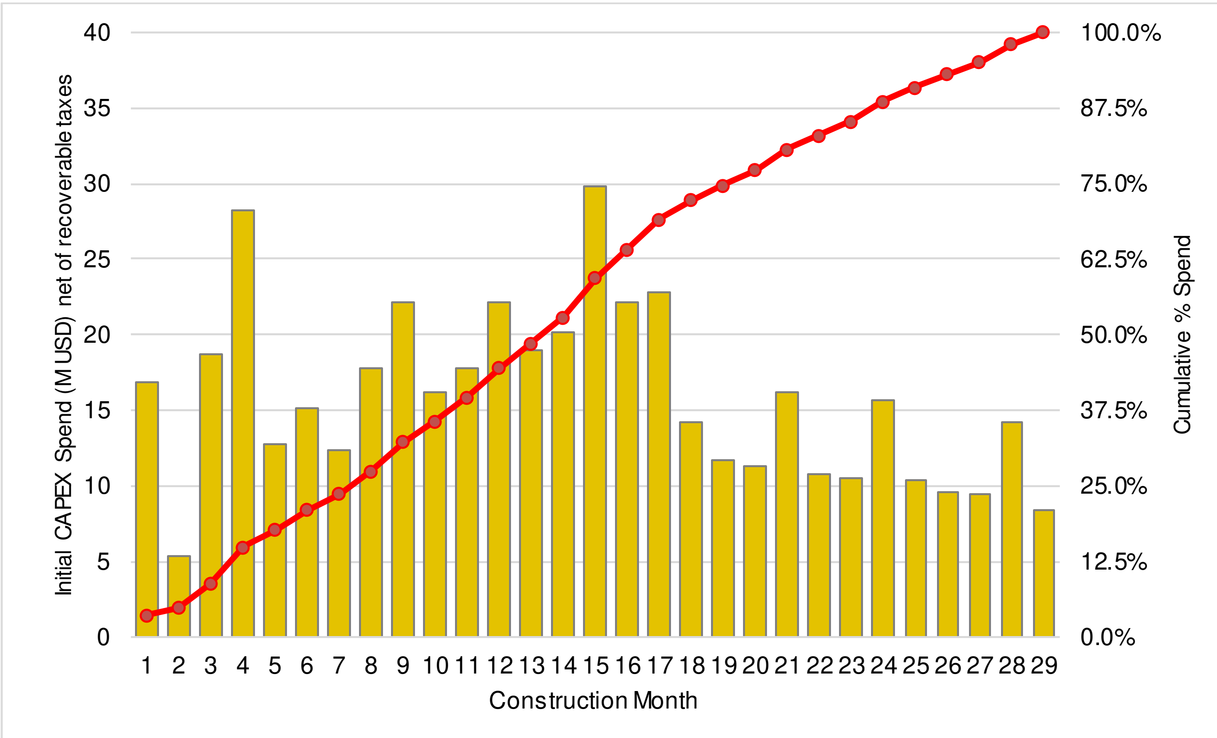
22.4 Capital Expenditures

The capital expenditures include initial capital expenditures ("CAPEX") as well as sustaining CAPEX to be spent after commencement of commercial operations.

22.4.1 Initial Capital

The initial CAPEX for Project construction, including processing, mine equipment purchases, pre-production activities, infrastructures and other direct and indirect costs is estimated to be USD 457.8M net of recoverable taxes and tax credits of 17.9M. The total initial Project capital includes a contingency of USD 38.3M which is 9.1% of the total CAPEX. The monthly CAPEX estimate is presented in Figure 22.3.

Figure 22.3: Initial CAPEX by Month



Source: GMS, 2022

22.4.2 <u>Sustaining Capital</u>

Sustaining capital is required during operations for additional equipment purchases for the mine. Additional work is required for raising the main embankment of the flotation tailings storage facility ("FTSF") and the construction of the second pond as part of the CIL tailings storage facility ("CTSF"). The continued raising of the FTSF will be completed by the mine operations team with fill material from the open pit mine. An effluent water treatment plant will be constructed in Year 2 to treat water from the CTSF prior to discharge. The sustaining capital is estimated at USD 82.9M net of recoverable taxes as summarized in Table 22.5. Additional details on sustaining CAPEX are presented in Section 21.

Table 22.5: Sustaining CAPEX Summary

Sustaining Capital	M USD
Mine Equipment	50.0
Tailings & Water Management	16.7
Process Plant	4.7
Non-Recoverable Taxes	11.5
Total Net of Recoverable Taxes	**82.9**

22.5 Working Capital

Working capital is required to finance supplies in inventory. Inventories for locally sourced consumables are kept at lower levels of around 30 days whereas consumables sourced internationally are kept at higher levels of around 60 days. Fuel will be delivered on a regular basis with approximately seven days of storage capacity.

Gold in-circuit constitutes a working capital item which is estimated at 1.5 koz. In addition, approximately one week of gold production is assumed in transit towards the refinery.

22.6 Closure Costs and Salvage Value

22.6.1 Reclamation & Closure Cost

Reclamation and closure costs include infrastructure decommissioning, site shaping and revegetation, maintenance and post closure monitoring. The reclamation activities for the mine will be initiated during operations and activities will continue once the low-grade stockpiles will have been processed. The total reclamation and closure cost is estimated at USD 23.5M and is discussed in Section 21.

22.6.2 Salvage Value

A salvage value is estimated for some mining equipment purchased during operations that will not have been utilized to its useful life. A residual value is estimated for some of the major process plant equipment such as grinding mills, crushers and tank agitators. The salvage value is estimated at USD 12.6M.

22.7 Operating Cost Summary

Operating Costs are presented by year in Table 22.6. The operating costs include mining, processing, general services and administration ("G&A"), gold transportation and refining and royalties. The average LOM operating cost is USD 623/oz of gold or USD 23.68/t milled. The average LOM all-in sustaining cost ("AISC") is USD 681/oz of gold or USD 25.88/t milled. Operating costs include non-recoverable taxes.

Table 22.6: Operating Cost Summary

Operating Cost Summary	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13+
Production Highlights														
Tonnage Milled (kt)	**48,284**	2,235	4,705	4,705	4,705	4,705	4,705	4,705	4,705	4,552	4,340	4,222		
Tonnage Mined (kt)	**194,939**	7,475	25,918	26,000	25,000	27,500	24,782	21,387	19,625	12,000	5,253	-		
Recovered Gold (koz)	**1,834**	93	203	163	206	233	175	137	180	209	163	70		
Operating Costs (M USD)														
Mining	**459.37**	17.05	47.41	57.56	50.55	56.68	56.23	57.61	53.32	37.47	19.88	5.61	-	-
Processing	**426.55**	19.65	41.16	41.16	41.16	41.16	41.16	41.16	41.16	40.49	39.55	38.75	-	-
G&A	**151.18**	7.89	15.74	15.71	15.73	15.74	15.74	15.76	15.49	14.32	10.26	8.80	-	-
Transport & Refining	**18.34**	0.93	2.03	1.63	2.06	2.33	1.75	1.37	1.80	2.09	1.63	0.70	-	-
Private Royalty	**43.75**	2.22	4.85	3.89	4.91	5.56	4.19	3.28	4.29	5.00	3.89	1.68	-	-
Govt. Royalty	**44.02**	2.24	4.88	3.92	4.94	5.59	4.21	3.30	4.32	5.03	3.92	1.69	-	-
Total Operating Cost	**1,143.21**	**49.98**	**116.05**	**123.87**	**119.35**	**127.05**	**123.28**	**122.48**	**120.37**	**104.40**	**79.13**	**57.24**	**-**	**-**
Sustaining Capital	**82.92**	27.59	22.01	9.42	1.87	4.97	5.22	3.08	6.81	1.37	0.56	0.01	-	-
Closure Cost	**23.53**	-	-	-	-	-	-	-	-	-	1.25	2.53	9.08	10.67
AISC	**1,249.66**	**77.57**	**138.07**	**133.29**	**121.22**	**132.02**	**128.50**	**125.57**	**127.19**	**105.77**	**80.94**	**59.78**	**9.08**	**10.67**
Unit Operating Costs														
Mining Cost / t mined	**2.36**	2.28	1.83	2.21	2.02	2.06	2.27	2.69	2.72	3.12	3.78	-		
Process Cost / t milled	**8.83**	8.79	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.89	9.11	9.18		
Total OPEX / t milled	**23.68**	22.36	24.67	26.33	25.37	27.00	26.20	26.03	25.58	22.93	18.23	13.56		
Total OPEX / oz	**623**	537	571	759	579	545	703	891	669	498	485	812		
AISC / oz	**681**	833	679	817	588	567	732	914	707	505	496	848		

22.8 Taxation

The applicable taxes are included in the economic analysis. Tax analysis was performed by L&M Advisory, a Brazilian specialized tax advisor. A number of fiscal benefits are available and are also included in the analysis.

22.8.1 Federal Taxes

The Federal taxes include:

- II: Imposto de Importação → Import tax
- IPI: Imposto sobre Produtos Industrializados → Tax on industrialized products
- IRPJ: Imposto de Renda da Pessoa Jurídica → Corporate income tax
- CSLL: Contribuição Social sobre o Lucro Líquido → Social contribution tax
- COFINS: Contribuição para o Financiamento da Seguridade Social → Contribution to social security financing
- PIS: Programa de Integração Social → Social integration program tax
- CFEM: Compensação Financeira pela Exploração de Recursos Minerais → Government royalty
- AFRMM: Adicional ao Frete para Renovação da Marinha Mercante → Additional freight charged in the port operation
- IRRF: Imposto de Renda Retido na Fonte → Withholding taxes
- CIDE: Contribuição de Intervenção no Domínio Econômico – Remessas para o Exterior → Remittances abroad

22.8.2 State Taxes

State taxes include:

- ICMS: Imposto sobre Operações Relativas à Circulação de Mercadorias e sobre Prestação de Serviços de Transporte Interestadual e Intermunicipal e de Comunicação → Tax on Transactions Related to the Circulation of Goods and on Rendering of Interstate and Intermunicipal Transport and Communication Services.
- DIFAL: Complemento relativo ao Diferencial de Alíquotas do ICMS → complement related to the ICMS rate differential.

22.8.3 Municipal Taxes

Municipal taxes include:

- ISSQN: Imposto sobre Serviços de Qualquer Natureza → Tax on services

22.8.4 Fiscal Benefits

Fiscal benefits at Federal level include:

- RECAP - Suspension of PIS and COFINS on the acquisitions of new machinery, instrumentation and equipment in the construction phase. The rules and the granting of the benefit are determined by the Receita Federal do Brasil (“RFB”). The legal basis of RECAP is in effect and provided for in Articles 12 to 16 of Law Nº 11,196, of November 21, 2005, and the list of items considered as “BK” is contained in the Federal Decree Nº 6581 of September 26, 2008. For the period of three years, such suspension is converted into an exemption if the obligations described below are met.
- SUDAM - INCOME TAX - The Company is subject to corporate income tax in Brazil at a rate of 25% and to social contribution tax at a rate of 9%. The Company is entitled to a special Brazilian tax incentive granted by the Superintendence for the Development of the Amazon (“SUDAM”) that provides a 75% reduction to the corporate income taxes payable on eligible profits earned for the year in relation to the Tocantinzinho operations. The Company is entitled to the SUDAM tax incentive for a 10-year period commencing in the year of receipt of the Appraisal Certificate from SUDAM. To receive the full benefits of the exemption, the Company is required to make an application the SUDAM tax incentive for the implementation of the new operations. Such applications are subject to approval by SUDAM. Legal basis: Federal Law Nº 13,799, of January 3rd, 2019. Overall burden can reduce from a combined rate of 34.0% to 15.25%. The SUDAM’s tax incentive may be renewed for successive periods of 10 years.
- INCENTIVIZED ACCELERATED DEPRECIATION - SUDAM: This benefit allows for acceleration of the depreciation and amortization expenses for the purposes of income tax calculation. Legal basis: art. 31 of Law Nº 11196 of November 21, 2005; Decree Nº 5988, of October 19, 2006; Decree Nº 4212, of April 26, 2002; and Decree Nº 4213, of April 26, 2002.
- PIS and COFINS CREDITS ANTICIPATION - SUDAM: Granting period of 12 months from the purchase of credits of the contribution for the PIS and COFINS. Legal basis: art. 31 of Law Nº 11196 of November 21, 2005; item III of §1 of art. 3 of Law Nº 10637, of December 30, 2002; item III of §1 of art. 3 of Law Nº 10833, of December 29, 2003; paragraph 4 of art. 15 of Law Nº 10865, of April 30, 2004; Decree Nº 5988, of December 19, 2006; Decree Nº 5789, of May 25,

2006; Decree Nº 4212, of April 26, 2002; and Decree Nº 4213, of April 26, 2002. This benefit ensures that the PIS and COFINS paid on purchases are credited.

- The DRAWBACK regime is considered an export incentive and may be applied in the following modalities: I - suspension - allows the suspension of the payment of import tax, tax on industrialized products, contribution to PIS/PASEP, COFINS, contribution to PIS/PASEP-Import and COFINS-Import, on import, in combination or not with the acquisition in the domestic market, of goods for use or consumption in the industrialization of the product to be exported. As of April 24, 2010, the acquisition in the domestic market of goods for employment or consumption in the industrialization of the product to be exported began to be carried out with suspension of PIS and COFINS, incidents based on billing, pursuant to art. 12 of Law 11,945, of June 4, 2009, disciplined by Joint Ordinance RFB/SECEX 467, of March 25, 2010, in the special regime called Integrated Suspension DRAWBACK.

22.9 Economic Results

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 22.7 and the annual Project cash flows are presented in Table 22.8. The total after-tax cash flow over the Project life is USD 1,043M and NPV 5% is USD 752M before tax and USD 622M after tax. The after-tax Project cash flow results in a 3.2-year payback period from the commencement of commercial operations with an IRR of 27.3% before tax and 24.2% after tax.

Table 22.7: Project Economic Results Summary

Project Economics Results		**Base Case**
Production Summary[1]		
Tonnage Mined	Mt	194.94
Ore Milled	Mt	48.28
Head Grade	g Au/t	1.31
Gold Processed	k oz	2,036
Recovery	%	90%
Gold Production	k oz	1,834
Cash Flow Summary		
Gross Revenue	M USD	2,935
Mining Costs (incl. rehandle)	M USD	(459)
Processing Costs	M USD	(427)
G&A Costs	M USD	(151)
Transport & Refining Costs	M USD	(18)
Royalty Costs	M USD	(88)
Total Operating Costs	M USD	(1,143)
Operating Cash Flow Before Taxes	M USD	1,792
Initial CAPEX[2]	M USD	(458)
Sustaining CAPEX	M USD	(83)
Closure Costs	M USD	(24)
Salvage Value	M USD	13
Total CAPEX	M USD	(556)
Royalty Buy-Back	M USD	(4)
Before-Tax Results		
Before-Tax Undiscounted Cash Flow	M USD	1,232
NPV 5% Before-Tax	M USD	752
Project Before-Tax Payback Period	years	3.1
Project Before-Tax IRR	%	27.3
After-Tax Results		
After-Tax Undiscounted Cash Flow	M USD	1,043
NPV 5% After-Tax	M USD	622
Project After-Tax Payback Period	years	3.2
Project After-Tax IRR	%	24.2

Note 1: Production period only, 2: USD 4.1M in tax credits netted from Iniital CAPEX.

Table 22.8: Project Cash Flow Summary

Cash Flow (M USD)	Total	(Y2)	(Y1)	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13+
Gold Price (USD/oz)		1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600		
Rec. Gold (koz)	1,834			93	203	163	206	233	175	137	180	209	163	70		
Gross Revenue	2,935			149	325	261	330	373	281	220	288	335	261	113		
Mining Costs	459			17	47	58	51	57	56	58	53	37	20	6		
Processing Costs	427			20	41	41	41	41	41	41	41	40	40	39		
G&A Costs	151			8	16	16	16	16	16	16	15	14	10	9		
Transp. & Refining	18			1	2	2	2	2	2	1	2	2	2	1		
Royalty Costs	88			4	10	8	10	11	8	7	9	10	8	3		
Total Operating Costs	1,143			50	116	124	119	127	123	122	120	104	79	57		
EBITDA	1,792			99	209	137	210	246	157	97	167	231	182	56		
Initial CAPEX (1)	462	183	211	68												
Sustaining CAPEX	83			28	22	9	2	5	5	3	7	1	1	0		
Closure Costs	24												1	3	9	11
Salvage Value	(13)											(2)	(4)	(0)	(6)	
Total Capital costs	556	183	211	96	22	9	2	5	5	3	7	(1)	(2)	2	3	11
Royalty Buyback	4	4														
Working Capital	0			7	1	1	(1)	1	(1)	0	1	(3)	(1)	(2)	(4)	
Total Other	4	4	0	7	1	1	(1)	1	(1)	0	1	(3)	(1)	(2)	(4)	
Pre-Tax Cash Flow	1,232	(187)	(211)	(3)	186	127	209	240	153	94	159	234	185	55	1	(11)
Taxes	189	5	6	3	10	1	22	26	17	11	24	31	22	9	(0)	0
After-Tax Cash Flow	1,043	(192)	(217)	(7)	176	126	187	213	135	83	136	203	163	46	1	(11)

Note1: Additional $4.1 million of direct tax credits reflected in Taxes.

22.10 Sensitivity Analysis

The Project financial performance is most sensitive to the gold price and exchange rate as well as the capital and operating costs. The results of the sensitivity analysis of the project in terms of NPV and IRR to the gold price are summarized in Table 22.9. The Project financial performance is also sensitive to fluctuations in the exchange rate of the Brazilian real. In the past year the real has ranged between 5.4 to 5.8 BRL/USD, sensitivities were run between 4.5 and 6.0 BRL/USD and are shown in Table 22.10.

Table 22.9: Gold Price Sensitivity

Gold Price	1,300	1,400	1,500	1,600	1,700	1,800	1,900	2,000
After Tax NPV 5% (M USD)	304	410	516	622	727	833	939	1,044
After Tax IRR (%)	15%	19%	21%	24%	27%	29%	32%	34%
EBITDA (M USD)	1,259	1,437	1,614	1,792	1,969	2,147	2,324	2,502
Free Cash Flow (M USD)	594	744	893	1,043	1,193	1,343	1,492	1,642
Payback (years)	4.1	3.7	3.4	3.2	3.0	2.7	2.5	2.3

Table 22.10: Brazilian Real Sensitivity

BRL/USD FX	4.00	4.50	5.00	5.20	5.50	6.00	6.50	7.00
After Tax NPV 5% (M USD)	451	533	599	622	653	698	736	769
After Tax IRR (%)	18%	21%	23%	24%	25%	27%	29%	30%
EBITDA (M USD)	1,615	1,700	1,768	1,792	1,824	1,870	1,910	1,943
Free Cash Flow (M USD)	826	930	1,014	1,043	1,083	1,140	1,189	1,231
Payback (years)	3.7	3.5	3.3	3.2	3.1	2.9	2.8	2.6

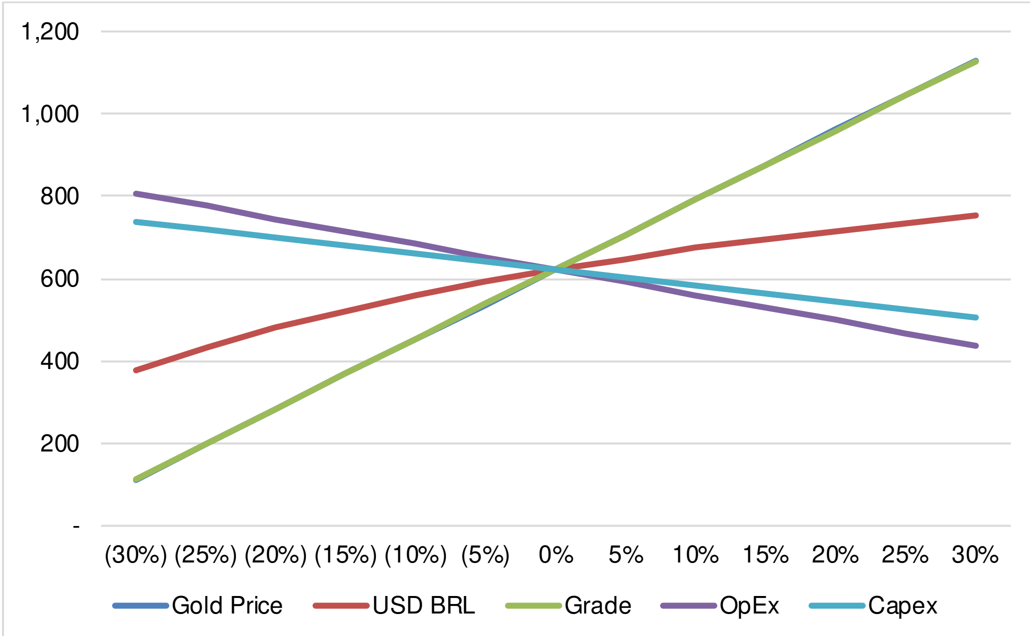
Table 22.11 presents the impact on the after-tax NPV 5% and the IRR for different combinations of gold price and BRL exchange rate assumptions.

Table 22.11: Gold Price & Brazilian Real Sensitivity

After-Tax NPV5%								
BRL/USD	Gold Price (USD/oz)							
FX	1,300	1,400	1,500	1,600	1,700	1,800	1,900	2,000
4.00	130	238	345	451	557	662	768	874
4.50	214	321	427	533	639	744	850	956
5.00	281	387	493	599	705	810	916	1,022
5.20	304	410	516	622	727	833	939	1,044
5.50	335	441	547	653	758	864	970	1,075
6.00	381	486	592	698	803	909	1,015	1,120
6.50	419	525	630	736	842	947	1,053	1,159
7.00	359	465	571	676	782	888	993	1,099
After-Tax IRR								
BRL/USD	Gold Price (USD/oz)							
FX	1,300	1,400	1,500	1,600	1,700	1,800	1,900	2,000
4.00	9%	12%	15%	18%	21%	23%	25%	28%
4.50	12%	15%	18%	21%	23%	26%	28%	30%
5.00	15%	18%	21%	23%	26%	28%	31%	33%
5.20	15%	19%	21%	24%	27%	29%	32%	34%
5.50	17%	20%	23%	25%	28%	31%	33%	35%
6.00	18%	22%	25%	27%	30%	33%	35%	37%
6.50	20%	23%	26%	29%	32%	34%	37%	39%
7.00	22%	25%	28%	30%	33%	36%	38%	41%

Figure 1.6 shows a comparison of the sensitivities on the after-tax NPV 5% to changes to the gold price, grade, CAPEX, OPEX and the Brazilian real exchange rate. Figure 22.5 shows a comparison of the sensitivities on the after-tax IRR to changes to the gold price, grade, CAPEX, OPEX and the Brazilian real exchange rate.

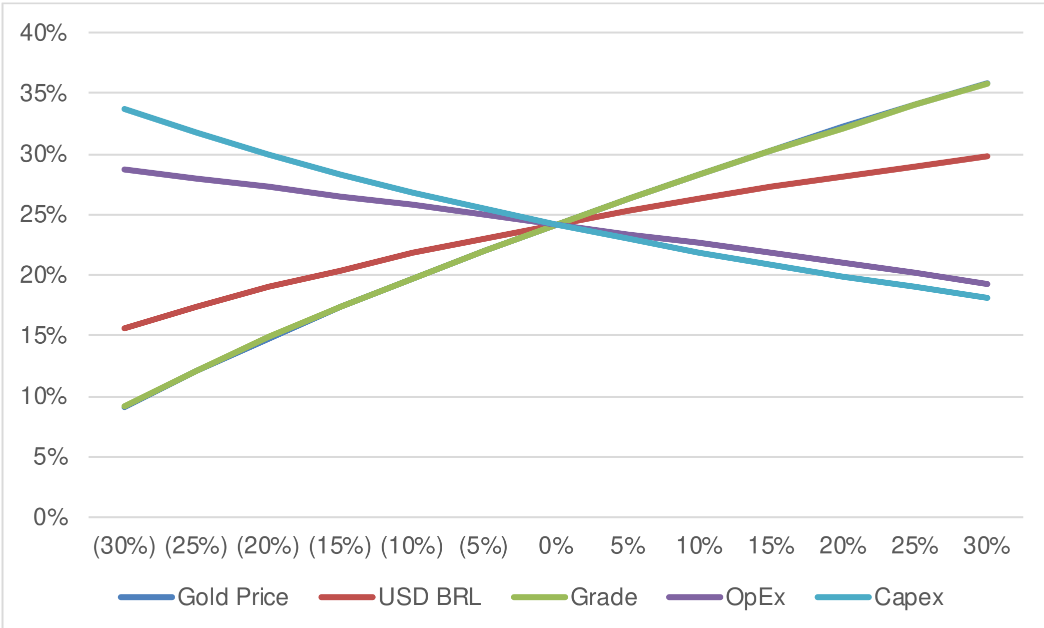
Figure 22.4: Sensitivity Analysis – After-Tax NPV 5%



Source: GMS (2022)

After-Tax NPV5%					
% Change	**Gold Price**	**BRL/USD**	**Grade**	**OPEX**	**CAPEX**
(30%)	111	378	113	806	738
(25%)	198	432	199	775	718
(20%)	283	479	284	745	699
(15%)	368	521	369	714	680
(10%)	453	558	453	683	660
(5%)	537	592	537	652	641
0%	622	622	622	622	622
5%	706	649	706	591	602
10%	791	674	790	560	583
15%	875	696	874	529	564
20%	960	717	959	499	544
25%	1,044	736	1,043	468	525
30%	1,129	753	1,127	437	506

Figure 22.5: Sensitivity Analysis – After-Tax IRR



Source: GMS (2022)

After-Tax IRR					
% Change	**Gold Price**	**BRL/USD**	**Grade**	**OPEX**	**CAPEX**
(30%)	9%	16%	9%	29%	34%
(25%)	12%	17%	12%	28%	32%
(20%)	15%	19%	15%	27%	30%
(15%)	17%	20%	17%	26%	28%
(10%)	20%	22%	20%	26%	27%
(5%)	22%	23%	22%	25%	25%
0%	24%	24%	24%	24%	24%
5%	26%	25%	26%	23%	23%
10%	28%	26%	28%	23%	22%
15%	30%	27%	30%	22%	21%
20%	32%	28%	32%	21%	20%
25%	34%	29%	34%	20%	19%
30%	36%	30%	36%	19%	18%

23 ADJACENT PROPERTIES

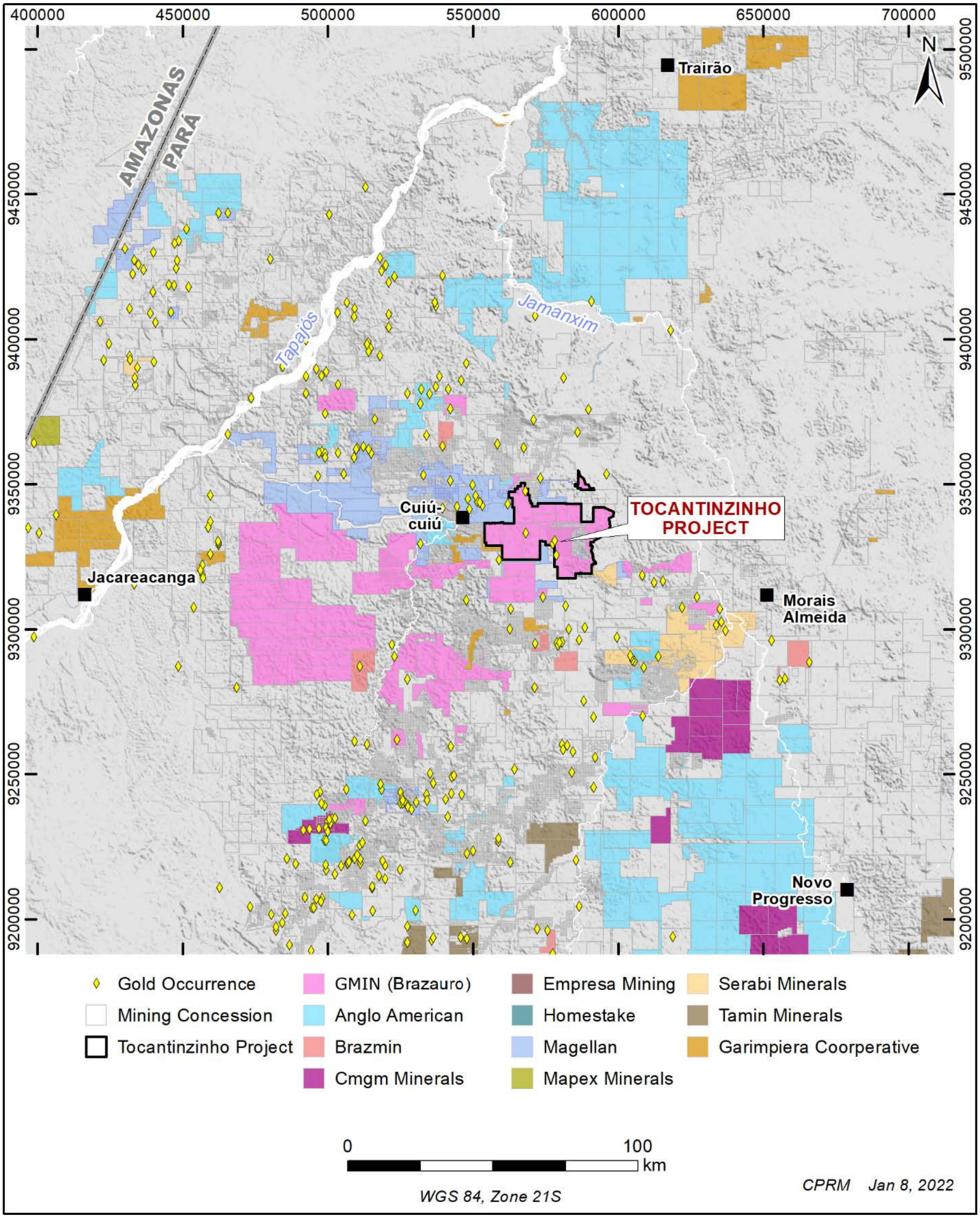
The Tocantinzinho Project is situated within the Tapajós Gold Province. The Tapajo's River Basin is well known for being the focal point for Brazil's modern gold rush. Since the 1950s, the Tapajo's River and its tributaries have been mined for alluvial gold. Through the 1960s, gold mining activity increased with the discovery of the rich ores in the tributary rivers. A combination of events in the 1970s then led to the modern gold rush with Artisanal and Small-scale Gold Mining (ASGM) activity, which continues to this day.

In Brazil alone, (ASGM) gold production is responsible for 30 tonnes of gold per year, of which about 26% is produced in the Tapajós River Basin by approximately 50,000 miners (or garimpeiros) who are scattered in more than 300 mining sites.

There are four (4) major gold deposits being developed by public companies within 100 km of the Tocantinzinho Project, including those owned by Cabral Gold Inc., Serabi Gold Plc, and Gold Mining Inc. (See Figure 23.1). There are numerous exploration permits in the area focused on copper exploration.

All information for adjacent properties was obtained through company websites and technical reports on www.sedar.com.

Figure 23.1: Adjacent Properties to the Tocantinzinho Gold Project



23.1 Cabral Gold Inc.

Cabral Gold Inc. (“Cabral Gold”) owns two (2) nearby projects, namely Cuiú Cuiú and Bom Jardim.

The Cuiú Cuiú Project is wholly owned by Cabral Gold and is located approximately 32 km west-northwest of the Tocantinzinho Project. Gold mineralization is hosted within several shear zones in granitic rocks of the Cuiú Cuiú Complex. The Cuiú Cuiú claim group consists of 41,576 ha of exploration licences and exploration and exploitation licence applications.

The project holds a NI 43-101 compliant mineral resource estimate of 5,886 kt of indicated material with an average grade of 0.90 g/t for a total of 171 koz of contained gold in this category, and 19,520 kt of inferred material at 1.24 g/t for 776 koz of gold in this category, effective December 31, 2017. The mineral resources are divided between open pit resources reported at a cut-off grade of 0.35 g/t gold and underground resources reported at a cut-off grade of 1.3 g/t gold (Micon, 2017).

Cabral Gold’s Bom Jardim Project consists of two (2) large exploration permits located about 25 km NW of the Cuiú Cuiú property and approximately 45 km NW of the Tocantinzinho Project. Exploration work on this property has identified significant NNE and E-W trending mineralized structures with strike lengths of up to 7 km.

23.2 Serabi Gold Plc

Serabi Gold Plc (“Serabi”) holds interest in the Palito Mining Complex involving 56,631 ha in exploration permits and applications including the Palito and São Chico underground mines, located approximately 65 km SE from the Tocantinzinho Project.

Mineralization at the Palito and São Chico deposits is hosted in granite and granodiorite of the Paráuari suite. Mineralization at the Palito Mine is hosted within three (3) granitoids and is intimately associated with northwest-southeast vertical to sub-vertical mesothermal quartz-chalcopyrite-pyrite veins and pyrite disseminations filling the brittle-ductile fault sets.

The Palito Mine is a high-grade, narrow vein underground mining operation that uses the shrinkage stoping method to extract gold and copper bearing ore. The São Chico Mine is a 140 t/d high-grade, narrow-vein long-hole stoping operation.

Serabi operates a 500 t/d plant to process ore from both the Palito and São Chico mines. Palito ore is processed through a flowsheet that includes crushing, grinding, copper flotation and carbon-in-pulp (CIP) cyanidation of gold and silver values from the copper flotation tailing. The São Chico ore is processed in a separate grinding circuit that includes gravity concentration and intensive cyanide leaching of the gravity concentrate.

The Palito Mine was operated by Serabi from 2003 until 2008 when it was placed on care and maintenance. A Preliminary Economic Assessment was completed in June 2012, based on a small selective high-grade mining initiative with 24,000 ounces produced per annum. The mine went into commercial production again as of July 1, 2014.

Based on the NI 43-101 Technical Report completed by SRK Consulting (U.S.), Inc., 2018, the São Chico mine held 90 kt of proven and probable mineral reserves with an average grade of 8.43 g/t, amounting to 24 koz of contained gold. The same report documented the Palito Mine of holding 613 kt of proven and probably mineral reserves with an average grade of 7.99 g/t gold and 0.37% of copper, for a total of 157 koz gold and 2.3 kt copper. Both of these statements were effective June 30, 2017.

23.3 Gold Mining Inc.

Gold Mining Inc. ("Gold Mining") holds interest in the São Jorge Project consisting of exploration permits located approximately 93 km from the Tocantinzinho Project. Gold mineralization occurs within the 1.2 kilometres-wide São Jorge granitic pluton which trends at 290 degrees, sub-parallel to the Cuiú Cuiú – Tocantinzinho shear zone.

The most recent NI 43-101 complaint mineral resource estimate is of 14,275 kt of indicated material grading 1.55 g Au/t amounting to 712 koz gold, and 17,582 kt of inferred material with an average grade of 1.27 g Au/t and 717 koz of contained gold. The mineral resource estimate is supported by an NI 43-101 technical report by GE21 Consultoria Mineral with an effective date of May 31, 2021. Mineral resources are reported at a cut-off grade of 0.30 g Au/t.

23.4 Anglo American Plc

Based on Brazil's National Mining Agency (ANM) public records, Anglo American Plc ("Anglo American") owns a total of 1,722,000 ha of exploration permits and applications in Para State and in the area around the Tocantinzinho Project. The majority of these exploration permits are focused on copper mineralization.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution Plan

24.1.1 Introduction

The purpose of the Project Execution Plan (PEP) is to describe how the Project will be developed. The information and procedures in the PEP guide the project team in the execution of the design and construction of all Project facilities.

24.1.2 Preparation, Revision, & Control of the PEP

The project management team is responsible for preparing the PEP in alignment with the project execution strategy. The company's senior management is responsible for approving the PEP, and the project director is responsible for communicating its required use to project members. The project managers are responsible for implementing the PEP within their groups.

The PEP is comprised of several sections and provides detailed information about the following:

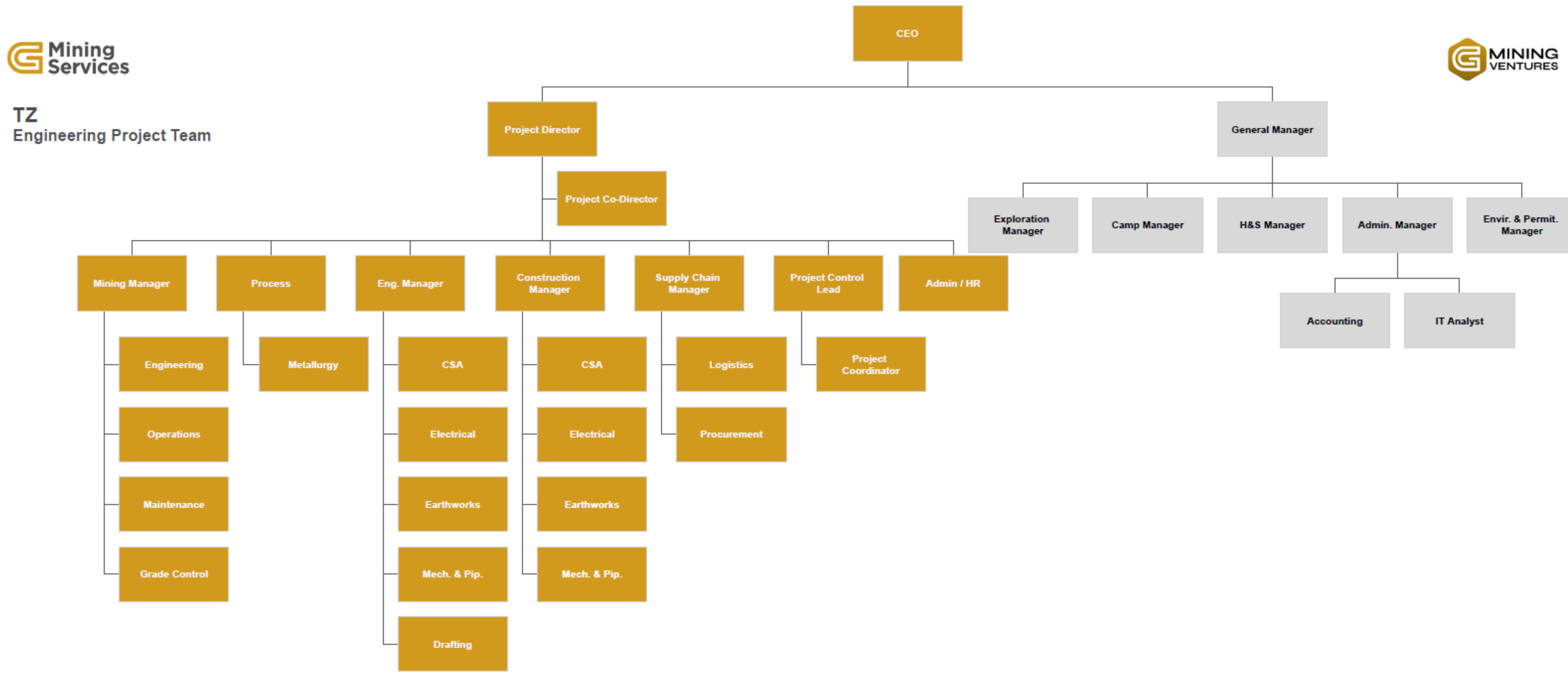
- Project organization
- Roles and responsibilities
- Practices and procedures
- Reference systems

24.1.3 Summary

An integrated project management team (IPMT) will be created to lead the execution of the Tocantinzinho project using a self-perform approach. The project team will be supplemented by contractors working within the IPMT for both specialized needs and peak manpower 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 Tocantinzinho mine team. Off-site infrastructure, including the access road and powerline, will be built or upgraded by a contractor under the supervision of the IPMT.

As established in Figure 24.1, the project team reports to the project director and works 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.

Figure 24.1: Tocantinzinho Organizational Chart



Source: GMS, (2022)

The IPMT shares the responsibility for the planning and execution of the project. Schedule development begins at the management level schedule (highest) 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. It is a statement of project objectives, as detailed in Figure 24.2. This Level 1 schedule contains the least amount of detail but combined with the major milestones presented in Table 24.1, provides a high-level tool for management to evaluate and track the main project milestones captured in the plan.

The Project schedule defines established tasks, the duration of each task, the relationship to one another, the responsibility assignments, as well as the resources required for each task.

The project team will use a quality assurance / quality control (QA/QC) system in all phases of the project (engineering, manufacturing of equipment, 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.

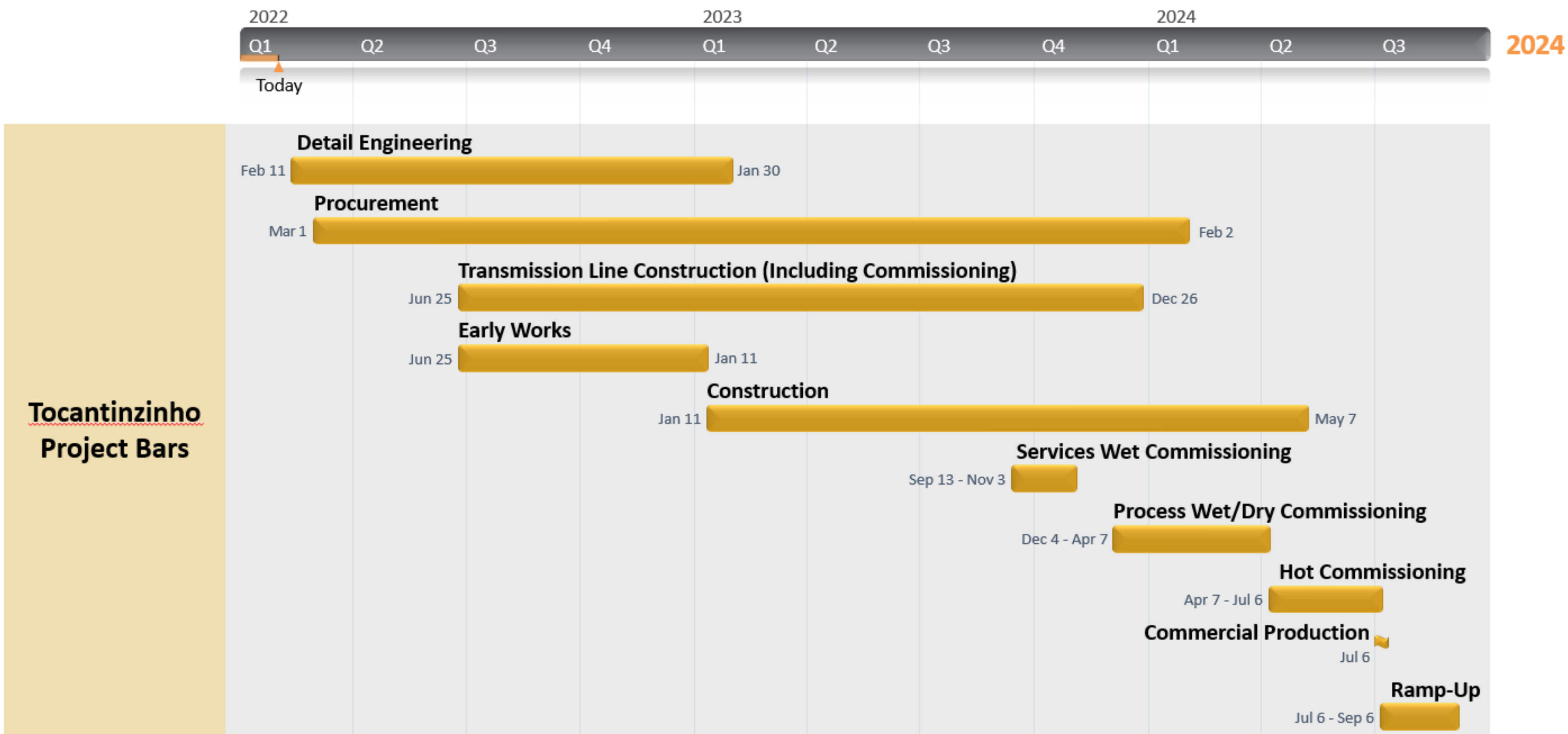
All general service departments will be staffed with Owner employees to service the construction efforts. These departments will become fully operational early in the construction period, with a trained workforce and established programs and service providers. As construction is demobilized, the service departments will seamlessly continue into operations.

The general service departments are as follows:

- Human Resources (HR)
- Information Technology (IT)
- Site Services
- Accounting
- General Administration
- Supply Chain Management (SCM)
- Community & Social Responsibility (CSR)
- Environment & Permitting
- Security
- Health and Safety (HS)
- Finance

A preliminary project execution schedule is provided in Figure 24.2 and milestones provided in Table 24.1.

Figure 24.2: Tocantinzinho Project Schedule – Level 1



Source: GMS (2022)

Table 24.1: Project Milestones

Description	Start	Finish
Start Detailed Engineering	11-Feb-22	
Awards (Conditional to Budget Approval) for Mine Major Equipment Purchases		24-Feb-22
Awards for Long Lead Items (Grinding Mills)		01-Apr-22
Start Power Line Construction	25-Jun-22	
Start Early Works	25-Jun-22	
Start Open Pit Pre-Production Stripping	25-Jul-22	
Start Earthworks with Mining Fleet	07-Aug-22	
Start Construction	11-Jan-23	
Start Services Wet commissioning	13-Sep-23	
Start Process Dry/Wet Commissioning	4-Dec-23	
Power Line Construction Completion (Including Commissioning)		26-Dec-23
Mechanical Completion Infrastructure (Area 100)		16-Feb-24
Mechanical Completion Process (Area 600)		24-Mar-24
Start Hot Commissioning	7-Apr-24	
Start Commercial Production	6-Jul-24	

Source: GMS (2022)

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 headed by the project director and general manager will provide overall site 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 last day of a period of 30 days when the ore tonnage processed averaged at least 60% of nameplate.

25 INTERPRETATION AND CONCLUSIONS

25.1 Summary

The Project has been studied and optimized and the results and findings provided in this Technical Report. This Feasibility Study actualizes the Project for the current long-term gold price, exchange rate, final pit design and reserves, life-of-mine production schedule, process plant design, infrastructure designs and execution approach.

This NI 43-101 Technical Report reconfirms the technical feasibility and economic viability of the Project based on an open pit mining operation with average gold production at 175 koz per year over a 10.5-year life-of-mine ("LOM"). It is recommended to advance the Project to the execution phase.

Table 25.1: Feasibility Study Update – LOM Results

Technical Report Feasibility Study Update Life-of-Mine Results	
Gold Price – Base Case (USD/oz)	1,600
Exchange Rate (BRL/USD)	5.20
Mine Life (operation years)	10.5
Strip Ratio (W:O)	3.36
Avg. Process Rate (kt/d)	12.6
Average Grade (g Au/t)	1.31
Average Gold Recovery (%)	90%
Average Annual Gold Production (koz)	175
Total Recovered Gold (koz) Incl. Pre-prod.	1,838
Initial Capital Expenditures (USD M) [1]	457.8
Sustaining Capital (USD M)	82.9
All-in Sustaining Cost (USD/oz)	681
Project After-tax NPV5 (USD M)	622
Project After-tax IRR (%)	24.2

Note: 1. Including pre-production revenue credit. Excludes working capital

The principal interpretations and conclusions by area are detailed below.

25.2 Mining Titles, Surface Rights, Water Rights and Royalty Agreements

The project is designed within the footprint of the two mining concessions covering an area of 12,889 ha. The mining concessions are in good standing and have an expiry date corresponding to the depletion of the mineral deposit. Even though some exploration licences have already expired, the mineral rights attaching thereto remain fully valid and in force during the analysis period (of the relevant extension applications by the National Mining Agency ("ANM").

Brazauro has concluded negotiations with occupants between 2015 and 2019 and paid relevant indemnifications. These acquired hectares are sufficient for the Project, including all areas required for the pit, waste dump, process plant, tailings dam and ponds, camp and administrative buildings. However, one occupant was granted the right to stay in the project footprint area by an interim judicial order and will require eventual relocation.

To construct the transmission line Brazauro has concluded easement agreements with 117 of the 148 land occupants. Negotiations are to be resumed.

There are no other underlying agreements of significance currently affecting the Project, save for royalty agreements whereby GMIN has a second Buy-Down Right for a cash consideration of USD 3.5M to be exercised within 30 days of a construction decision to further reduce the private royalty to an effective rate of 1.5% NSR.

25.3 Geology and Mineral Resources

- SRK carried out a detailed quality control review of the geological database (core and tailing drilling). Overall, SRK considers analytical results from core conducted between 2004 and 2015 at the Tocantinzinho Project are globally sufficiently reliable for the purpose of resource estimation. The data examined by SRK do not show any obvious evidence of analytical bias.
- The qualified person audited the resource estimation process and confirmed that the mineral resource model reasonably reflects the data that is available. Additionally, the qualified person tested the sensitivity of mineral resource modeling outcomes to alternative estimation methodologies and found the reported mineral resource statement to reasonably represent the mineral deposit at the current level of sampling and geological interpretation.
- The mineral resource estimates for the Tocantinzinho Project were made from a 3D block model constrained by geological domains and a grade shell. Gold was estimated in a single estimation domain using ordinary kriging and commercial software. The gold domain shows the desirable

attributes of a low nugget (20%) and an adequate coefficient of variation (1.45), which is a reflection of the disseminated nature of mineralization observed at the drill sample scale.

- The Mineral Resource estimate is based on combined drilling of 296 core boreholes for approximately 82,805 m. Several auger and reverse circulation holes were also drilled but not used for the resource estimation.
- Sampling methods are consistent with industry practices and adequate to support Mineral Resource and Mineral Reserve estimation and mine planning.
- Sample preparation and analytical procedures since 2009 are consistent with typical industry practices at the time the samples were prepared and are adequate to support Mineral Resource and Mineral Reserve estimation and mine planning.
- Density determinations are acceptable to support Mineral Resource and Mineral Reserve estimation and mine planning.
- Sample security procedures met industry standards at the time the samples were collected. Current sample storage procedures and storage areas are consistent with industry standards.
- Mineral resources have been estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" (2019) guidelines and are reported in accordance with the Canadian Securities Administrators' National Instrument 43-101.
- The Mineral Resources estimated at a cut-off grade of 0.30 g Au/t by SRK are as follows:
    - Saprolite
        - Measured & Indicated Resources: 1,460 kt at 1.11 g Au/t for 52 koz of gold
        - Inferred Resources: 59 kt at 0.66 g Au/t for 1 koz gold
    - Fresh rock:
        - Measured & Indicated Resources: 45,222 kt at 1.38 g Au/t for 2,000 koz of gold
        - Inferred Resources: 732 kt at 0.92 g Au/t for 22 koz of gold
    - Garimpeiros tailings:
        - Measured & Indicated Resources: 1,432 kt at 1.10 g Au/t for 50 koz of gold
        - Inferred Resources: 789 kt at 1.07 g Au/t for 27 koz of gold

25.4 Mining and Mineral Reserves

- Mineral Reserve estimation was performed by GMS. Optimization runs were carried out only using Measured and Indicated Mineral Resources to define the optimal mining limits. Inferred Mineral Resources were considered as waste.

- The mine design and Mineral Reserve estimate have been completed to a Feasibility Level with the pit designs following criteria recommended from pit slope geotechnical studies carried out by Golder Associates.
- The Proven and Probable Ore Reserve estimate at a cut-off grade of 0.36 g Au/t is as follows:
  - Proven Mineral Reserve: 17,973 kt at 1.46 g Au/t for 842 k in-situ ounces of gold.
  - Probable Mineral Reserve: 30,703 kt at 1.22 g Au/t for 1,200 k in-situ ounces of gold.
  - Total P&P Reserve: 48,676 kt at 1.31 g Au/t for 2,042 thousand in-situ ounces of gold.
- The Mineral Reserves incorporate appropriate mining dilution and mining recovery estimations. The average mining dilution is 5.5% with minimal isolated ore blocks resulting in negligeable ore losses.
- The mining activities will occur over a period of 11 years which includes two years of pre-production activities. The mining rate reaches a peak of 27.5 Mtpy in Year 5 of commercial production. The LOM plan mine's a total tonnage of 212.1 Mt which includes 163.4 Mt of waste for a strip ratio of 3.36 (W:O). The mine consists of a single open pit that will be developed in four phases.
- Mining of the Tocantinzinho deposit is planned as a conventional open pit operation using 17 m^3 hydraulic excavators and 92 t class haul trucks. A bulk mining approach is well suited for the massive ore body with mining to take place on 10 m high benches. The mine is planned as an owner mining operation with blasting activities to be outsourced.

25.5 Metallurgical Testing and Mineral Processing

- The process design criteria were established based on metallurgical test work results, industry best practices and vendor recommendations
- Circuit selection and processing options for the Project were selected based on the results of this test work, trade off studies and conventional unit operations used in the mineral processing industry.
- Ore hardness and grindability remains to be confirmed during detail engineering. An opportunity exists to optimize SAG and ball mill sizing once further comminution variability testing has been completed.
- Silver has not been quantified in the resource; however metallurgical test work has indicated that there will be a significant amount of silver extracted with the gold. The silver represents a substantial opportunity to increase the revenues from the Tocantinzinho mine.

25.6 Infrastructure

- Infrastructure required to support operations will include waste rock storage areas, ore stockpiles, flotation tailings storage facility and CIL tailings storage ponds, access and internal road, powerlines and power distribution networks, process plant, accommodation facilities and mine support facilities including offices, fuel storage, explosives storage, workshops, and warehouse.
- The access road has been constructed and is currently used by others. Additional improvements are planned for the access road to correct gradients and to improve surfacing and drainage.
- The design of the flotation tailings impoundment and the two leach effluent ponds meet recent Brazilian code and are sufficient for the life of mine. The dams are designed with a downstream construction method and the two CIL effluent ponds will be lined with a geomembrane. Extensive investigation on the Flotation Tailings foundation and construction materials was completed and the stability and seepage analyses show adequate factors of safety in the design while some geotechnical investigation is outstanding for the CIL ponds.
- Power supply is planned via a 198 km, 138 kV transmission line. A formal agreement with the electric utility is yet to be completed.

25.7 Environmental Considerations

- Baseline environmental and social studies have been completed including flora and fauna, hydrology and hydrogeology, air quality, geochemistry, geotechnical and archeology. The archaeological sites identified have been through the rescue stage, curator activities and analysis of the material collected in the laboratory with the final report approved by the Instituto do Patrimônio Histórico e Artístico Nacional ("IPHAN"). These areas can now be used for the Project.
- Brazauro conducted two campaigns of static testing to evaluate the potential for acid rock drainage ("ARD") and metal leaching ("ML") of its ore and waste materials. The results of the metal leaching tests in both campaigns showed that there is no release of any excess elements to the limits established in Annex F to ABNT NBR 10004:2004, which allowed to conclude that the materials do not fall within the category of hazardous waste (class I A). Of major importance, the results of static tests showed that the waste rocks mined have high PN/PA ratios due to the presence of carbonate and consequently have low to no potential for acid generation.
- The flotation tailings storage facility (FTSF) will receive approximately 95% of the tailings from the process plant and will require the construction of a main dam that will be phased over the life of the Project. Given the tailings are classified as non-hazardous and inert, there is no requirement for a liner system. The CIL tailings representing the remaining 5% of the process plant tailings will be

stored in a separate storage facility which will consist of containment provided by two ponds. The effluents from the CIL circuit will be treated in a cyanide detoxification circuit using the conventional SO_2/air process before deposition in the ponds.

- The Project has an estimated industrial water demand of 200 m^3/h for the process plant and multiple industrial uses. The Veados creek has sufficient water to meet this demand. To maximize water conservation, process water will be recirculated from the flotation tailings storage facility and process water for cyanide destruction will use decanted water from the CIL tailings ponds.
- BRM had obtained seven LIs to begin the Project's installation. However, installation activities have not yet begun and, in 2019, BRM requested SEMAS to suspend six out of seven LIs, which SEMAS consented to do for "a maximum period" of 730 days.
- BRM requested recently a revocation of the suspensions which have been granted by SEMAS. SEMAS must conduct a technical evaluation and site visit as part of the request to extend the validity of the installation licenses by an additional two years.
- Permits relating to vegetation suppression, wildlife monitoring, capture, collection, rescue, transport and release were previously obtained but have expired. New permits must be requested from the relevant environmental authorities before construction begins. Similarly, the preliminary water permits to drill water wells and permits for effluent release have expired and must be re-applied for.

25.8 Capital and Operating Costs

- The initial capital cost is estimated to be USD 457.8M net of recoverable taxes and credits; it includes a contingency of USD 38.3M which represents 9.1% of total expenditures or 10.2% of expenditures when excluding the mining equipment. Included in this estimate are capital expenditures for the mining fleet, which has an initial capital cost of USD 46M; a major portion of the mine fleet capex can be financed using capital lease agreements with vendors.
- Sustaining costs over the LOM are estimated to total USD 82.9M mostly related to additional mining equipment and tailings management.
- Reclamation and closure costs are estimated at USD 23.5M to be offset by salvage values from the sale of major processing and mining equipment at USD 12.6M.
- The average LOM operating cost is USD 623/oz of gold or USD 23.68/t milled. The average LOM all-in sustaining cost ("AISC") is USD 681/oz of gold or USD 25.88/t milled.

25.9 Economic Analysis

- The economic analysis is carried out in real terms (i.e., without inflation factors) in Q1 2022 United-States dollars. A base case BRL exchange rate of 5.2 BRL/USD was used.
- The base case economic analysis using a gold price of USD 1,600/oz has an after-tax NPV 5% of USD 622M, IRR of 24.2% and payback of 3.2 years after the start of production.

25.10 Other Relevant Data and Information

The Project construction timeline is developed over a period of 29 months which includes three months of commissioning of the process plant. The process plant continues to ramp-up to nameplate capacity over two months after the start of commercial production.

25.11 Risks and Opportunities

25.11.1 Risks

The main Project risks are:

- Completion of permit updates for the revised footprint
- Finalization of a power agreement contract
- Manpower availability in remote location
- Impact of foreign exchange on capital cost estimate
- Continued inflationary pressure
- Availability of goods and services in a remote location
- Lengthening of lead times for equipment and materials in general and specifically due to supply chain issues related to COVID
- Longer construction period due to COVID outbreaks affecting employees and contractors on site.

25.11.2 Opportunities

The main Project opportunities are:

- Increased Resources and Reserves at depth
- Exploration success in large surrounding exploration land package

- Optimization of comminution circuit following additional metallurgical test work
- Improved gold recovery of about 2% with fine grinding of sulphide concentrate
- Silver revenues expected as evidenced by metallurgical test work but not quantified in the resource model

26 RECOMMENDATIONS

Based on the robust economic results of this Feasibility Study it is recommended that GMIN progress the Tocantinzinho Gold Project to detailed engineering and construction. The initial capital expenditures detailed in Section 21 are estimated at USD 457.8M; net of recoverable taxes and credits.

During the course of the Feasibility Study, items were identified as requiring additional information to further improve precision and information as part of the detailed engineering. Certain risks were also identified that require significant initiatives and continuous monitoring.

26.1 Geology and Mineral Resources

- A total of 678 and 56 specific gravity measurements were taken for unweathered and weathered material, respectively. Thus, a larger data set is required to provide an accurate assessment of the saprolite specific gravity.
- Additional specific gravity measurements in the tailings material should be generated to increase the confidence in the assessment of tailings specific gravity.
- A drilling program is currently underway to test extensions to mineralization at depth, and infill areas of the resource close to surface that represent mill feed scheduled in the early years of operations. This data should be used to update the mineral resource block model once results are available. An estimated cost of USD 50,000 is planned to complete this work and is included in the cost of the Project.

26.2 Mining and Mineral Reserves

- Grade control should be performed prior to the start of pit operations. The in-pit quarry should be targeted for this early grade control as it is mined prior to the start of on-site assaying. The approximate value of the program is USD 0.8M and is included in the project as a standard blast hole grade control and an RC in-fill drilling campaign.
- Detailed planning for earthwork and deforestation should detail the tasks to be performed by the owner with its mining fleet and by local contractors.

26.3 Metallurgical Testing and Mineral Processing

It is recommended to complete additional metallurgical test work which has been initiated at SGS Geosol:

- To assess the variability of ore hardness to adequately size the SAG and Ball mills. This includes completing SMC and BWI tests on 20 representative granite samples and SPI tests on five of those samples.
- To confirm gravity recovery results. This includes completing gravity separating tests via a laboratory gravity centrifuge on 20 representative granite samples.
- To confirm flotation recovery results at P80 of 125 µm. This includes completing rougher and cleaner flotation tests on gravity tailings samples generated from the gravity tests above.
- To confirm leach gold extraction results. This includes completing bottle roll tests encompassing intensive leaching, cyanidation kinetics, CIL, gold-to-carbon adsorption curves and Freundlich isotherms.
- To investigate the potential of increasing gold extraction by regrinding the flotation concentrate. This includes completing regrind/leach tests and establishing a regrind curve for mill sizing.
- To confirm and optimize cyanide destruction parameters. This includes completing cyanide destruction tests using SO_2/air and Caro's acid methods.
- To complete additional geochemical testing of the CIL tailings.

Following the completion of the additional metallurgical test work, it is recommended to confirm and/or update the process design criteria, gold recoveries and confirm equipment sizing. An estimated cost of USD 132K is planned to complete this work and included in the cost of the Project.

26.4 Recovery Methods

The following is recommended related to the process plant:

- Finalize SAG and ball mill sizing once further comminution test work has been completed.
- Finalize gravity, flotation, CIL and cyanide destruction circuit sizing once further metallurgical test work has been completed.
- Once further metallurgical test work has been completed, evaluate the need for a concentrate regrind circuit to potentially increase gold recoveries. Currently layout provision has been allowed for the future installation of a regrind mill.

- Optimize process plant reagent consumption once further metallurgical test work has been completed.

An estimated cost of USD 250K is planned to complete this work and included in the cost of the Project.

26.5 Infrastructure

The following is recommended to be completed during the detailed engineering phase of the project infrastructure.

- Conduct geotechnical investigation of the new locations of the CTSF ponds and the process plant.
- Complete test work required to confirm adequate source of on-site sand for concrete quality requirements.
- Review water balance and tailings breach analysis with revised CTSF locations.

An estimated cost of USD 720K in planned to complete this work and included in the cost of the Project.

26.6 Environmental, Permitting and Social Considerations

It is planned to continue ARD testing for both flotation and CIL tailings and refine closure plan that will ensure that long term effluents meet discharge standards. During the first year of operations, assaying and testing will be performed on the decant water at the CIL tailings pond to enable a satisfactory design for the effluent treatment plant.

BRM and SEMAS must complete the process to revoke the LIs permits and formally reactivate them to enable the Project to proceed with construction. Other permits will be requested from relevant environmental authorities before construction can begin. Also permit modifications will be required to reflect the site optimization and arrangement for some of the facilities.

Previous meetings with the communities of interest have confirmed the strong desire for employment and supply of goods and services from the immediate region. BRM will have to formalize its relations with the surrounding communities to identify joint opportunities and meet expectations.

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