Feasibility Study Technical Report (NI 43-101) for the Matupá Gold Project, Matupá Municipality, Mato Grosso, Brazil

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1. EXECUTIVE SUMMARY

1.1 PROPERTY DESCRIPTION AND OWNERSHIP

The Matupá Gold Project area is located in the Alta Floresta Gold Province, which lies in the extreme north-central part of Mato Grosso State, Brazil. The Project area encompasses an area surrounding the towns of Matupá and Guarantã do Norte, approximately 700 km north of Cuiabá, the Mato Grosso State Capitol and 200 km north of Sinop, an important commercial centre and fourth city population. The Matupá Gold Project refers to Aura's, and previously Rio Novo's and Aura's, on-going exploration, economic evaluation and planned development by surface mining of gold deposits in the province. This report focuses on the X1 and Serrinhas gold deposits.

The X1 Deposit is located near to Matupá city, approximately 11 km north of its urban area and approximately 11 km south of the town of Guarantã do Norte, both municipalities are located along Highway BR-163.

Aura holds the mineral rights for 9 properties, of which 3 cover an area of 15,333.81 hectares ("ha") located within an existing Mining Concession (X1 Deposit, Serrinhas and Guarantã Ridge Targets), another 6 properties totaling 47,172.65 ha are under an Exploration Permit. The Property totals 62,506.46 hectares in the Alta Floresta Gold Province.

The Matupá Gold Project includes the properties covered by the Mining Concessions ANM number 866.428/2002 that includes the X1 Deposit, the property under ANM number 866.324/1991 including the Serrinhas Target, and the property ANM number 866.072/2001 covering the Guarantã Ridge Target.

1.2 GEOLOGY AND EXPLORATION

The Alta Floresta Gold Province ("AFGP") is located in the south-central portion of the Amazon Craton (Almeida, 1978; Almeida et al., 1981), a crustal segment north of South America that would have stabilized at 1.0 Ga, which is surrounded by the mobile Neoproterozoic mobile belts of Tucavaca (in Bolivia), Araguaia-Cuiabá (Central Brazil) and Tocantins (northern Brazil) (Almeidaet al., 1976; Cordani et al., 1988; Tassinari & Macambira, 1999). As the AFGP covers an area ofapproximately 430,000 km^2, it represents one of the largest cratonic regions on the planet, comprising two Precambrian shields: the Central Brazil (or Guaporé) and Guiana shields, that are separated by the Paleozoic Solimões-Amazonas (Tassinari) basin (Macambira, 1999; Dardene &Schobbenhaus, 2000; Tassinari et al., 2000) (Figure 7.1).

The Alta Floresta Gold Province (AFGP) is mostly comprised of plutono-volcanic sequences generated in paleo- and mesoproterozoic continental arcs, in addition to deformed and metamorphic units in restricted greenschist facies to its central and northwestern portions. The units that comprise the province, especially its eastern segment, are essentially represented by oxidized calcium-alkaline plutonic and volcanic rocks, of medium to high potassium (K), meta- to peraluminous, belonging to the magnetite series (type I granites). of volcanic, sub-volcanic and alkaline granitoids (type A granites).

As a whole, these units are arranged on a west-northwest oriented belt. These units exhibit ages ranging from 1.75 to 2.03 Ga, with model ages (TDM) between 2.76 and 2.15 Ga and εNd(t) values between -7.62 and 3.09 (Santos et al., 1997; Moura, 1998; JICA/MMAJ, 2000; Santos etal., 2000; Pimentel, 2001; Pinho et al., 2003; Souza et al., 2005; Paes de Barros, 2007; Silva & Abram, 2008; Miguel-Jr, 2011; Assis et al., 2014), suggestive of magmatism with an Archean topredominantly Paleoproterozoic source, in a juvenile arch environment, but with a small continental contribution.

According to Assis (2015), in general, these units can be grouped into four main sets:

I. Deformed and metamorphized granitic basement (2.81 to 1.99 Ga);

II. Felsic plutono-volcanic and volcano-sedimentary sequences belonging to the magnetite series (type I granites; 1.97 to 178 Ga);

III. Post-collisional and anorogenic plutono-vucanic units (1.78-1.77 Ga); and

IV. Clastic sedimentary sequences (~1.37 Ga).

The basement of this portion of the province corresponds to heavily razed areas and lacks outcrops. The basement unit is currently divided into two main complexes: (i) Bacueri-Mogno 2.24 Ga (Pimentel, 2001), not exposed in the eastern segment of the AFGP; and (ii) Cuiú-Cuiú 1992 ±7 Ma (Souza et al., 2005). The first main complex comprises pyroxene-rich orthoamphibolites, orthogneisses, paragneisses (garnet-silimanite-cordierite-biotite gneiss, illimanite-biotite gneiss and illimanite gneiss), enderbitic plutonics, banded iron formations, calc-silicate-quartzite-granite, quartzite-granitic rocks, metagabbro-norite and metapyroxene that exhibit mylonitic foliation and/or medium- to high-dip gneiss banding that are oriented east-west to east-southeast-west-northwest (Souza et al., 2005; Silva & Abram, 2008). Pimentel (2011) obtained isochronic Sm-Nd ages of 2.25 Ga and εNd(t) of 2.4 for amphibolite in this complex, corresponding, therefore, to the oldest age in the region. The Cuiú-Cuiú Complex, the second main complex, however, outcrops near the cities of Peixoto de Azevedo and Novo Mundo, and consists essentially of granitic to tonalitic gneisses, migmatites intruded by calcium-alkaline foliated granitoids of tonalitic to monzogranitic composition (Paes deBarros, 2007), in addition to shales, mafic and ultramafic rocks and banded iron formations (Dardenne & Schobbenhaus, 2001).

The Matupá Gold Project area is part of the granitic bodies of the Matupá Intrusive Suite, which has an intrusive geological relationship to the gneiss basement of the Cuiú-Cuiú complex and to the Diorite/Gabbro bodies, the oldest regional event. The Matupá Intrusive Suite are mostly identified from soil due to few available outcrop exposures or by diamond drilling. These rocks were intruded by quartz feldspar porphyries and late fine-grained mafic to intermediate dykes. These sequences are in contact with volcanic and pyroclastic rocks of the Colíder Group located north of Guarantã do Norte city.

The lithology of the basement rocks in the Project area includes biotite-tonalitic gneisses representative of the Cuiú-Cuiú complex. Among the most significant lithologies in the properties, particularly surrounding the X1 Deposit and parts of the Alto Alegre block, are medium, inequigranular and porphyritic biotite-granodiorites (potassium feldspar porphyries up to 3 cm in size) of light gray color, essentially isotropic, and may locally present incipient to little penetrative foliation when close to zones of regional magnitude shear or smaller shear zones reflecting them. The porphyritic biotite-granodiorites are composed of quartz, plagioclase, potassium feldspar phenocrysts (pink microcline), biotite and magnetite.

The Diorite/Gabbro bodies, the oldest regional event, present greater spatial expression, are phaneritic, magnetic, fine to medium grained, isotropic, and are mainly composed of plagioclase and hornblende, but locally phenocrystals of local plagioclase are also observed.

The initial exploration work was first carried out in 1996 by Mineração Bom Futuro in partnership with Western Mining Corporation ("WMC") later followed by Rio Tinto ("RTZ") in 2000, resulting in the discovery of the Serrinhas of Matupá target, currently known as the Serrinhas Target. Among the historical exploration activities performed was geological mapping, geochemical sampling of rock and soil, ground geophysical surveys (Gamma spectrometry and Gradient IP), followed by auger drilling. Reverse circulation and diamond drilling campaigns, sample results ranged from 0.2 g/t to 24.09g/t Au, followed by detailed geological mapping at a 1:1,000 scale were performed. Later exploration work performed by Vale involved ground and airborne geophysical surveys and initial diamond drilling campaigns, resulting in the discovery of the Guarantã Ridge and X1 Deposits in 2002 and 2003, respectively.

1.3 DRILLING, SAMPLING AND ASSAYING

Drilling on the Matupá Gold Project has been completed in various campaigns since 1996 by WMC, Rio Tinto (RTZ), Crescent Resources ("CRESCENT"), Vale, Mineração Santa Elina ("MSE"), and Rio Novo. The implemented drilling methods were diamond drilling, reverse circulation (RC), and auger drilling. For the purposes of previous studies, Rio Novo decided not to use the reverse circulation drill hole information for the geological models and Mineral Resource Estimates, now historical estimates, for any deposits. This was done to assure that the quality of assay results and other drill hole information met Rio Novo's quality control standards. The current study also follows the same logic regarding drilling and has not used RC drilling in modelling, estimation, and classification.

In total, there have been 148 diamond drill holes drilled in the X1 area, totaling 30,184.66 m. Table 1-1 summarizes the X1 drilling and drill core results.

Table 1-1 – Total Drilling in the X1 Area.

TYPE	COMPANY	PERIOD	NUMBER OF HOLES	TOTAL LENGTH (m)	AVERAGE HOLE DEPTH (m)	NUMBER OF SAMPLES	DRILL HOLE SERIES
Diamond Drilling	VALE	1999-2004	18	3,190.05	177.23	3,139	FD-029 to FD-046
	MSE	2006 - 2010	63	14,106.34	223.91	8,158	SEX1-01 to SEX1-063
	RNM	2010 - 2018	60	11,469.66	191.16	10,318	FX1D-0001 to FX1D-0061
	Aura	2019 to date	7	1,418.61	202.66	697	
	Subtotal		**148**	**30,184.66**	**198.74**	**22,312**	
RC Drilling	Aura	2019 to date	43	2,242	52.14	2,242	FX1R-0001 to FX1R-0043
	Subtotal		**43**	**2,242**	**52.14**	**2,242**	

At X1, the known extents of mineralization were first drilled on nominal 50 m-center spacing which was later followed by Rio Novo's infill drilling program, reducing the spacing in some parts of the deposit to approximately 25 m centers. Drilling covers an area of about 500 m along strike and 350 m across strike. Additional scout holes have been drilled around the perimeter. The X1 Deposit is primarily drilled out to a vertical depth of 250 m to 300 m, although individual drill holes are as deep as 500 m (vertical depth).

For the 2010-2012 drilling campaigns at X1, Rio Novo hired Geosol, a Brazilian drilling company to perform drilling on the X1 Deposit. Geosol drilling crews extracted the core, placed it in wooden core boxes, which were sealed with tape or straps prior to transport. The core was then transported by truck to Rio Novo's core-shack processing facility at Guarantã do Norte city where the core was laid out, washed, and photographed. The core was then logged, and sample intervals were marked by Rio Novo geologists. Sample intervals were generally 1 m; however, variations were allowed for special samples or special interval breaks. The maximum sample interval was 1.5 m, and the minimum was 0.5 m. Core logging included lithology, alteration, mineral zone, structural and geotechnical logging. Structural and geotechnical details that were noted included foliation, fractures, vein orientation, and faults. Percent core recovery and rock quality determination ("RQD") measurements were taken and calculated for all drill intervals. Samples were tagged with electronic bar codes, with one tag inside the bag and one tag outside. Sample bags were also marked by hand in permanent ink and the sample numbers were electronically entered into the database, according to the proper sample intervals. This system then provided an electronic sample submittal form.

All the Rio Novo sample preparation took place at SGS' laboratory. To ensure that the correct particle size and sample reduction procedures are achieved during sample preparation, the SGS Geosol Laboratory used established protocols for the preparation of samples of rock/core and soil/stream sediments. Before starting sample preparation, proper equipment must be set up, calibrated, and monitored to ensure quality specifications are met. Quality control measures conducted during sample preparation by SGS Geosol were as follows:

- Equipment is designed and set up to produce representative sample fractions during splitting;
- Equipment was cleaned with barren rock followed by compressed air between each sample run;
- Screen tests for coarse gold were conducted on crushed and pulverized sample fractions at the rate of one test per 20 sample batch.

The primary analytical laboratory used by Rio Novo for the Matupá Project was the SGS Geosol laboratories, located in: Vespasiano, Minas Gerais State; Goiânia, Goiás State; and finally, Várzea Grande, Mato Grosso State, Brazil. The Várzea Grande lab was used by Rio Novo from 2011 onwards. The laboratory has ISO 9001 certification and ISO 14001:2004, ISO 17025:2009 certification for environmental chemical analysis. SGS Geosol employs modern, industry standard techniques and analytical methods. For routine gold analysis at the Matupá Gold Project, fire assay Atomic Absorption with fusion 50 g aliquots finish was used most frequently. Multielement analysis (34 elements) were determined by ICP after the digestion of samples either in aqua regia or in four acids. The analytical detection limit for gold by fire assay-AA finish is 5 ppb. For samples with gold grades higher than 10,000 ppb, the analysis MET 150 (Metallic Screen) were applied. The Barium, Copper, Lead, Molybdenum and Zinc assays over-limits of 10,000 ppb and the manganese assays over-limits of 15% are re-assayed by ICP90Q, which makes the determination of six elements by fusion with sodium peroxide. The second laboratory used by Rio Novo for check assays (5 to 10% of external check assays of mineralized samples with a cut-off of 0.3 g/t Au) was ALS Chemex in Vespasiano, Minas Gerais State and Goiânia, Goiás State, Brazil. The analysis was made in Lima, Peru. All analytical results and certificates from both laboratories were provided separately and digital copies of the files were stored in the Rio Novo digital database.

The Rio Novo quality assurance/quality control ("QA/QC") program includes both blind (introduction of their own control samples – internal control) and non-blind (laboratory check assays – external control) submittals of standards and blanks; intra and inter-laboratory check assays; core duplicate assays, and pulp and coarse reject duplicate assays. The QA/QC program of Rio Novo included the insertion of the following control samples: high-, medium- or low-grade standards in each batch of 20 samples; blanks in each batch (mainly after mineralized zones); 1/20 core duplicate (5%); 5% to 10% external check assays of the mineralized samples (cut-off of 0.3 g/ton Au) tested in a second laboratory. The internal QA/QC program monitored the control sample results including internal standards, coarse and pulp duplicates, and size checks during preparation. Additionally, systematic checks were processed in a digital database against original signed certificates.

A total of 287 purchased reference standards, 281 blank samples, and 279 duplicates were inserted into the sample stream during the 2010 to 2012 drill program starting with hole FX1D-0001 in the X1 Deposit. The initial set of standards was purchased from Geostats Pty Ltd. of Western Australia. When these were exhausted, a second set was purchased from ITAK located in João Monlevade, Minas Gerais State, Brazil. A total of 20 blank samples failed in the QA/QC with value above 3 detection limits (15 ppb Au) and 10 samples surrounding the blank sample position in the batch were re-assayed. A total of 279 field duplicate samples were included in the sample stream sent to the SGS laboratory.

1.4 DATA VERIFICATION

The underlying data from previous drilling campaigns were verified by Aura's geologist and database manager under the supervision of Qualified Person ("QP") Farshid Ghazanfari, P.Geo. The data verification by Aura included the following activities:

- Database and data entry;
- Reviewing drilling and geological logs;
- Re-surveying drill holes collar's location;
- Reviewing and validation assay certificates versus assay data in the database;
- Preparing new topography surface both in SAD69 and SIRGAS and validation against collar table of drill holes in the database; and
- Re-sampling of selected drill core intervals from previous drilling campaigns.

The QP is satisfied that the exploration, sampling, security, and QA/QC procedures employed at the X1 Deposit from the Matupá Gold Project, and their results, are sufficient to produce data adequate for the purposes used in this technical report.

1.5 MINERAL PROCESSING AND METALLURGICAL TESTING

The metallurgical work conducted on the Matupá Gold Project mineralized materials included three test campaigns. The first metallurgical campaign was carried out in 2018-2019, including metallurgical assessment and characterizations conducted by SGS-Geosol Laboratories, in Belo Horizonte, Brazil, together with mineralogical characterizations conducted at SGS Lakefield Laboratory at Lakefield City, Canada. The X1 deposit was classified into three ore types, which were used throughout the first metallurgical campaign.

Further studies resulted in the reclassification of ore types i.e., Fresh Rock and Oxide, which were used in the second metallurgical campaign, the latter including various tests and assessments carried out by different companies, described as follows:

- Mineral Processing Solutions Ltda. (MinPro Solutions);
- Testwork Desenvolvimento de Processo Ltda. (Testwork Lab.);
- SGS Geosol Laboratorios Ltda. (SGS Geosol Lab.);
- FLSmidth Ltd. (Knelson Division);
- FLSmidth Brasil; and
- COTEPROM (Consultoria e Assessoria em Processos Ltda.).

The main objectives of the second metallurgical campaign were:

- Complete the characterization of the ore typologies to be dealt with in the Project, determining the comminution parameters for industrial circuit sizing;
- Select the grinding size based on metallurgical recovery results, as well as operational technical aspects;
- Assess the contribution of the gravity circuit;
- Set process and design parameters for the leaching stage (cyanidation);
- Assess the efficiency of the SO_2/air method for destructing the residual cyanide contained in the tailings; and
- Generation of tailing samples for characterization of chemical and environmental aspects.

The two selected ore types for the Matupá Project represented by Fresh Rock and Oxide samples were considered amenable to direct leaching (cyanidation), as well as gravity tailing leaching, resulting in very high gold recovery figures for a 24-hour leaching period of samples ground to an 80% percent passing rate (“P_{80}”) at 0.125 mm.

Averaged gold recovery figures obtained in the second metallurgical campaign were adopted in the Preliminary Economic Assessment(“PEA”) stage for the two selected ore types of the Matupá Project as listed in Table 1-2.

Table 1-2 – Gold Recovery Results- Second Metallurgical Campaign.

Ore Types	Recovery - Au (%)		
	Gravity Recovery (%)	Gravity Taiings Leaching	Leaching Recovery - P80 125 microns
Fresh Rock	36,5%	92,12%	97,81%
Oxide Ore	12,6%	93,68%	93,55%
ILR Effiiciency*	98,0%	* Intensive Leaching Reactor.	

A third metallurgical campaign was carried out in 2021-2022 based upon the progression of the Life of Mine (LOM) plan as well as the engineering aspects of the Project. The objectives of the third metallurgical campaigns were as follows:

- Consolidate the gold metallurgical recovery as a function of gold grade in order to estimate gold production according to the blending predicted by the LOM plan;
- Confirm the design and project criteria for the selected metallurgical flow sheet based on blending, predicted by the LOM plan;
- Enhance the assessments related to filtering and dry-stacking of tailings for obtaining consolidated design parameters; and
- Assess the effects of gold grade variability on metallurgical performance for both Fresh Rock and Oxide ore types.

The testing campaigns and assessments were carried out by different companies, described as follows:

- SGS Geosol Laboratorios Ltda. (SGS Geosol Lab.);
- SGS Mineral Services / JKTech;
- FLSmidth Brasil;
- Pattrol - Investigações Geotécnicas Ltda;
- Jenike & Johanson; and
- COTEPROM (Consultoria e Assessoria em Processos Ltda.).

The variability assessments were carried out on four samples i.e., high- and low-grade blends for each one of the two ore types (Fresh Rock and Oxide). Figure 1-1 summarizes results obtained for both gravity concentration tests and tailing leach tests.

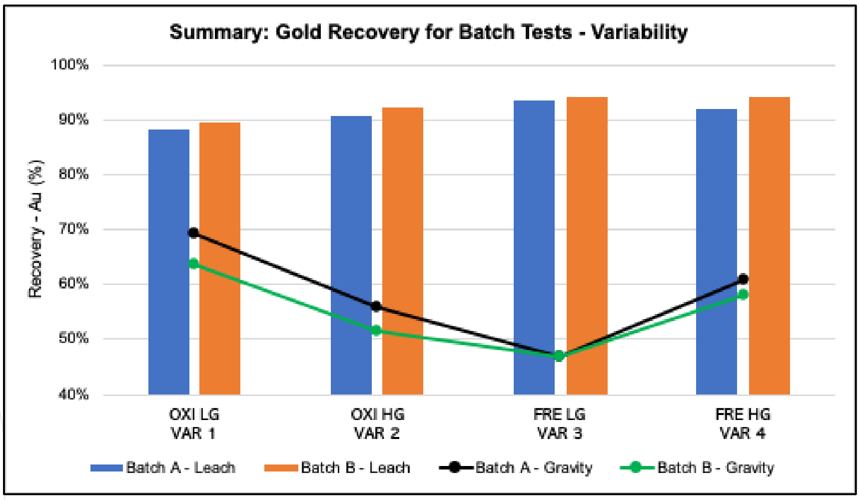


Figure 1-1 - Summary of Testing Results – Gravity/Leaching - Variability.

Data obtained from both consolidation and variability testing campaigns were used for estimating the gold recovery as a function of the processing plant, gold head-grade, as shown in Figure 1-2, for both Direct Leaching (w/o Gravity) and Gravity-Leaching (w/Gravity) routes.

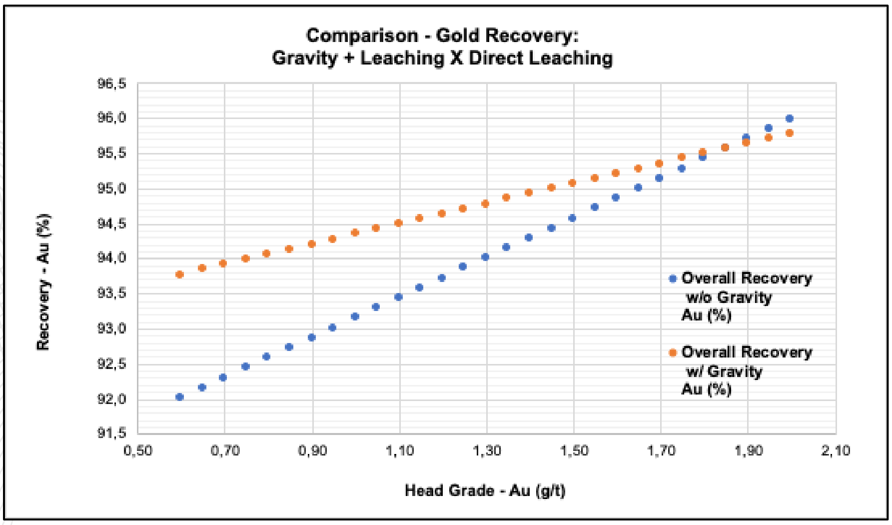


Figure 1-2 - Gold Recovery for Direct Leaching and Gravity/Leaching Routes - Consolidation.

The graph shown in Figure 1-2 indicates higher gold recoveries for the Gravity/Leaching route compared with the Direct Leaching route for gold head-grades smaller than 1.85 g/t Au. These results determined the Gravity/Leaching route to be used for the Matupá Project.

1.6 MINERAL RESOURCES

The Matupá Gold Project Mineral Resource Estimate is limited to the X1 Deposit. The Mineral Resource Estimate updates were performed for all current information using a validated database. 3-D updated models were constructed in the GEOVIA GEMS™ and Surpac™ software platform (version 6.3). Mineral Resources were estimated using the same software platform by Farshid Ghazanfari, P.Geo. and QP for Aura Minerals. In opinion of QP for this section, the Mineral Resource Estimates meet industry standards and the general guidelines for NI 43-101 compliant Mineral Resources for Measured, Indicated, and Inferred confidence levels as discussed herein.

A new, ground-survey topography was carried out in 2021 over the X1 property, covering all areas surrounding the X1 Deposit that would be subject to any future mining operation. A drone based topographic survey was performed in 2020, covering a larger area including potential areas where infrastructure and processing plants could possibly be located.

The X1 Deposit database includes different drilling campaigns conducted by various companies, Vale, Santa Elina, Rio Novo and Aura, carried out between 2003 to 2021. The older data was received as part of the acquisition of Rio Novo by Aura Minerals.

The X1 Deposit occupies a topographic high point (hill) in the area of the Project and is hosted by the Matupá Intrusive Suite near the contact with the mafic/ultramafic Flor da Serra Suite. The X1 Deposit extends 400 metres along strike from east to west and 250 metres from north to south.

Two alteration models were developed based on lithological and alteration logging information of all drill holes that intersected mineralization on the X1 property. These two models, with some minor adjustments, were used for the Mineral Resource Estimate for the X1 Deposit for this feasibility study (Figure 1-3). Three 3-D models were created for saprolite, weathered, and fresh rocks after grade interpolation had been performed.These models coded appropriately within the X1 block model for the Oxide attribute. The alteration model consists of oxide and sulfide materials with separate tonnes and grades calculated for each material type. Figure 1-4 shows digital terrain models ("DTM") and weathering profiles in the X1 Deposit.

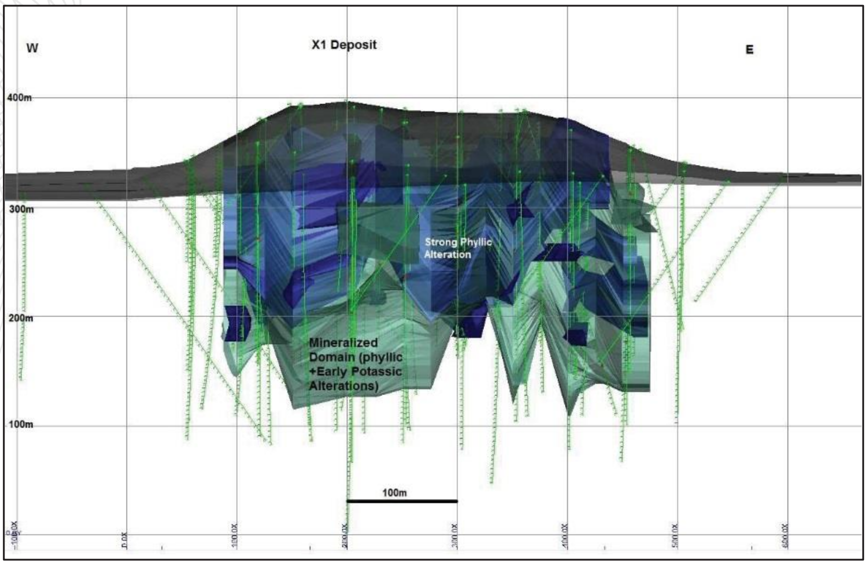


Figure 1-3 - X1 Deposit 3-D Alteration Models and Trace of Drill Holes, Vertical Cross Section.

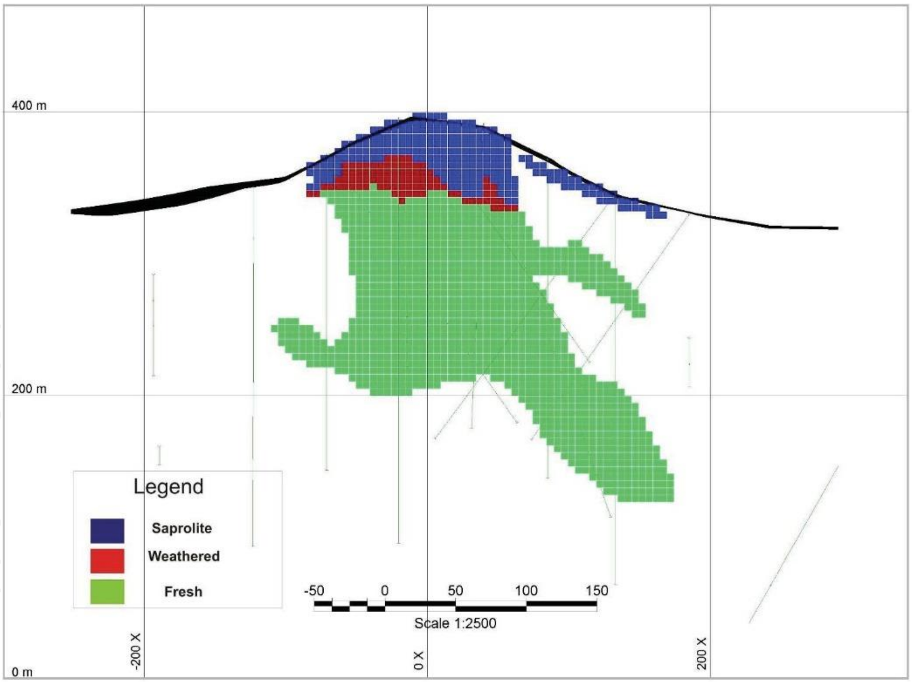


Figure 1-4 - X1 Deposit 3-D Weathering Profiles and Trace of Drill Holes in a Typical Vertical Cross Section, Looking SE.

The X1 density database represents different lithologies, mineralization types, and degree of alteration and weathering. The database has 1,261 density measurements for fresh rock on drill core and 68 samples for weathered, soil and saprolite samples. The water immersion method was used by Rio Novo geologists for weathered, saprolite, and soil samples, and the porous samples were sealed in plastic bags.

Table 1-3 summarizes the average density values that were recorded and stored in in the X1 database.

Table 1-3 – Average Density Values for Different Rock Types at the X1 Deposit.

Lithology	*Density t/m³*
Saprolite & Soil	*1.51*
Weathered	*2.44*
Fresh Rock	*2.71*

The X1 database contains sufficient data to determine a Mineral Resource Estimate.The X1 database contains 21,663 samples with Au and Ag values equal to or greater than zero. Sample lengths are variable, from 0.34 m to 6.45 m, with an average length of 1.33 m.

The point data set for statistical analysis used drill holes that intersected the mineralized wireframe domains and all assays were extracted with their corresponding lengths. The total number of samples residing inside of the mineralized domains and used for grade interpolation are 7,567 samples for Au and 7,420 samples for Ag, with an average length of 1.37 m.

Samples within the mineralized envelopes were processed into 2.0 m composites and capped, after compositing, at 20.0 g/t Au and 170 g/t Ag.

A statistical summary for the generated 2.0 m composites, by lithotype, for Au is shown in Table 1-4.

Table 1-4 – Summary Statistics – 2.0 m Composites (Au).

LITHOLOGY	NUMBER	MINIMUM	MAXIMUM	MEAN	VARIANCE	STANDDEV	LOGESTMN	CV
Saprolite	453	0.00	5.14	0.23	0.22	0.47	0.24	2.04
Weathered	563	0.00	14.99	0.34	0.76	0.87	0.36	2.55
Fresh	4298	0.00	68.10	0.54	4.29	2.07	0.52	3.87
All	5314	0.00	68.10	0.49	3.58	1.89	0.48	3.87

Variography for composited samples was completed using Snowden's Supervisor software. A variography model was fitted to the composited data within the X1 alteration ore models. For continuity modelling, a normal scores transform was used.

The anisotropy directions are coincident with the deposit shape (geological models). The strike of the deposit used was the azimuth of the major axis, selected to be 0° with a plunge of -50°.

The block model limits were defined using UTM coordinates, and the block size selected for the model was 5 m x 5 m x 5 m. The model was not rotated. The block model definition is given in Table 1-5.

Table 1-5 – Block Model Definition (X1).

Direction	Origin	Block Size (m)	No. Blocks
Easting (X)	727,750	5	120
Northing (Y)	8,886,350	5	110
Elevation	425	5	65

The grade interpolation used Ordinary Kriging ("OK"). The updated, 3-D alteration models, coded in the block model, were interpolated using only the data points from inside that specific zone as the data source. The strong phyllic alteration was coded as rock type 4, weaker phyllic alteration coded as rock type 6, and a separate composite data set for each domain was used in grade interpolation.

The search parameters for each interpolation run are listed in Table 1-6 for Au. A typical cross- section with Au values through the estimated block model isshown in Figure 1-5.

Table 1-6 – Au Grade Interpolation Parameters (X1).

Search Reference	Search Distances (m)			Minimum No. of Composites	Maximum No. of Composites	Maximum Composites per Hole
	X	Y	Z			
1	25	20	22	4	24	3
2	70	60	35	4	24	3
3	140	120	70	4	24	1

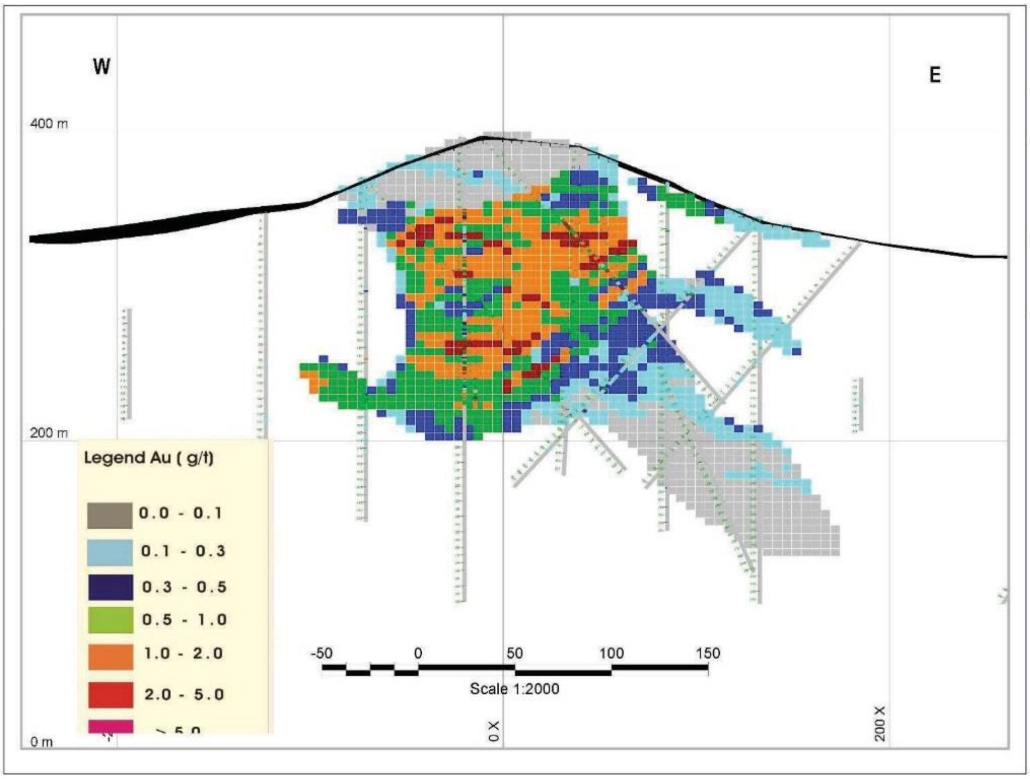


Figure 1-5 - Typical Vertical Cross-Section Through the X1 Block Model Estimated Grades (Au) (Looking N).

The Mineral Resources for the X1 Deposit have been classified in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") definitions and standards for Mineral Resources and Mineral Reserves (CIM, 2014). The classification parameters consider the proximity and number of composite data. The block model is then coded accordingly for Measured (1), Indicated (2) and Inferred (3) classification for all three deposits.

The Mineral Resource classification criteria applied in the current study are shown in Table1-7.

Table 1-7 – Mineral Resource Classification Criteria (X1).

Classification	*Pass No.*	*Approximate*	*Actual Distance (m)*	*Min No. Samples*	*Max No. Samples*	*Max. Samples per Hole*
Measured	*1*	*≤ 25 m*	*≤ 40*	*4*	*24*	*3*
Indicated	*2*	*>25 and ≤ 70*	*≤ 40*	*4*	*24*	*3*
Inferred	*3*	*No limit*	*>40*	*4*	*24*	*1*

The updated Mineral Resource Estimate is based on the alteration models which encompassed all economic gold mineralization in the X1 Deposit. These mineralized domains were analyzed for grade capping values and variography and were interpolated using the ordinary kriging method. Once the block model was completed it was classified into Measured, Indicated, and Inferred Mineral Resources. A Lerchs-Grossman open pit optimization process was performed, resulting in the updated Mineral Resource Estimate presented in Table 1-8, showing Mineral Resources within an optimized pit at $1,800/oz gold price in cross-sectional view.

*Table 1-8 – X1 Deposit Measured and Indicated Mineral Resource Estimate *.*

Resources Classification	Tonnage (t)	Grade Au (g/t)	Total Au (ounces)	Grade Ag (g/t)	Total Ag (ounces)
Measured	4,692,520	1.14	172,00	3.85	580,810
Indicated	4,653,150	0.96	143,600	4.39	656,430
Measured +Indicated	9,345,670	1.05	315,600	4.12	1,238,240

**Mineral Resource Notes and Assumptions*

1. *The Mineral Resource Estimate has an effective date of August 31, 2022.*
2. *Mineral Resources are inclusive of Mineral Reserve*
3. *Mineral Resources do not have demonstrated economic viability and are not Mineral Reserves*
4. *The Mineral Resources in this estimate were calculated using the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions.*
5. *The base case cut-off grade for the estimate of Mineral Resources is 0.35 g/t Au*
6. *The Measured and Indicated Mineral Resources are contained within a limiting pit shell (using a gold price of US$ 1,800 per ounce Au) and comprise a coherent body.*
7. *A density model based on alteration and rock type was established for volume to tonnes conversion averaging 2.76 tonnes/m^3.*
8. *Contained metal figures may not add due to rounding.*
9. *Surface topography used in the models was surveyed July 31, 2021.*
10. *The Mineral Resource Estimate for the X1 Deposit was prepared by Farshid Ghazanfari, P.Geo., a Qualified Person as defined in NI 43-101 regulations.*

1.7 MINERAL RESERVE

The Mineral Reserves estimation was prepared using industry standard methods and provide an acceptable representation of the deposit. Engenharia de Minas ME ("EDEM") reviewed the reported Mineral Resources, production schedules, and factors for conversion from Mineral Resources to Mineral Reserves.

Based on this review, it is the author's opinion that the Measured and Indicated Mineral Resource within the final pit designs at Matupá Project can be classified as Proven and Probable Mineral Reserves.

The Mineral Reserve Estimates have been prepared in accordance with, and the classification of Proven and Probable Reserves conform to CH 20.235 definitions. Economic analysis of the Life of Mine (LOM) plan generates a positive cash flow and, in EDEM's opinion, meets the requirements for the classification of Mineral Reserves.

The designed open-pit's Proven and Probable Mineral Reserves of gold, are estimated to be about 8.5 Mt, with a grade of 1,14 g/t Au, totaling around 293,000 ounces of gold metal contained. The Mineral Reserves' input parameters and estimated results for the Proven and Probable classification are presented in the Table 1-9.

Table 1-9 – Mineral Reserve by Classification, Tonnage, and Related Grade g/t Au.

MINERAL RESERVE ESTIMATE						
CLASSIFICATION	**PROVEN**		**PROBABLE**		**TOTAL**	
Ore Type	**Tonnes (kt)**	**AU (g/t)**	**Tonnes (kt)**	**AU (g/t)**	**Tonnes (kt)**	**AU (g/t)**
LOW-GRADE ORE	**203.2**	**0.40**	**245.3**	**0.40**	**448.5**	**0.40**
HIGH-GRADE ORE	**3,596**	**1.36**	**4,440.3**	**1.03**	**8,036.4**	**1.18**
TOTAL MINERAL RESERVE	**3,799**	**1.31**	**4,685.6**	**0.99**	**8,485**	**1.13**

**Notes:*

1. The Mineral Reserve estimates were prepared in accordance with the CIM Standards on Mineral Resources and Reserves.
2. The Mineral Reserve Estimate has an effective date of August 31, 2022.
3. The Mineral Reserve Estimate is based on an updated optimized shell using US$1,500/oz gold price, average dilution of 3%, mining recovery of 100% and break-even cut off grades of 0.35 g/t Au for X1 pit.

4. Contained metal figures may not be added due to rounding.
5. Surface topography as of July 31, 2021.
6. Mineral Reserve estimate for Matupá Project was prepared under the supervision of Luiz Pignatari, P.Eng.,a “qualified person”, as that term is defined by NI 43-101.
7. The concentration plant recovery was established by Consolidations Tests Recovery model presented in the “technical report”.
8. The silver grades and metal contents were not considered in the reserve calculation as still there are doubts about the metallurgical recovery during the gold production process.

1.8 MINING METHOD

Mining costs are based on the rock types, lithologies to be mined, including drilling, blasting, loading, transport to crushing patio and specific stockpiles, and all the necessary infrastructure required for mining production.

The mining operation proposed for the X1 deposit, Matupá’ s Gold Project, will feed lithologies, high-grade ore first, to the process plant; the estimated tonnage by lithology is presented in the Table 1-10.

Table 1-10 – Planned Ore from X1’s Pit, by Lithology, to be Concentration Plant Feed by Year.

	TOTAL ROCK TO BE FED TO THE PLANT				
	All Lithologies		**High-Grade Ore**		**Low-Grade Ore**
Year ▼	Total	Average Grade	Fresh Rock	Soil + Saprolite	All Lithologies
	(kt)	(g/t)	(kt)	(kt)	(kt)
1	1,170	1.39	1,076	94.3	-
2	1,300	1.49	1,200	100.1	-
3	1,300	1.43	1,200	99.5	-
4	1,300	1.32	1,200	99.8	-
5	1,300	0.76	1,198	102.4	-
6	1,300	0.79	1,200	100.1	-
7	915	0.39	403	56.4	456

The planned tonnages to be mined and basis for all costs calculations are presented in Table 1-11, which shows the planned ore yearly tonnage that is to go to the concentration plant or stockpiles, and the total waste to be removed and dumped in a specific stockpile.

The mined Run Of Mine (“ROM”) destinations are:

- High-Grade ore stockpile to crushing plant area: ore will be taken up by a Front End Loader to feed the crushing plant. Alternatively, a high-grade ore volume is stored in the same area to be fed later in case a higher grade is needed to improve the Au production in the concentration plant.

- Low-grade ore stockpile: following a strategy to maximize the net present value (“NPV”), the low-grade value rock will be destined to the low-grade ore stockpile located close to the concentration plant. The low-grade ore will be taken to the processing plant at the end of the LOM.

- Waste dump piles: will be located close to it the pit the waste material was mined from and be part of the environment reclamation at the end of LOM, when appropriate treatment will be applied.

Table 1-11 – Yearly X1's Pit Volumes: Ore Ronnage to be Feed in the Concentration Plant, Tonnage Moved and Dump Piles.

Origin ▶	Mine						Ore Patio		Oxidized Ore Pile		Low-Grade Ore	
Destin ▶	Ore Patio/Piles						Crushing Plant		Crushing Plant		Crushing Plant	
Ore/Waste ▶	High-Grade Ore		Low-Grade Ore		Waste		High-Grade Ore		Low-Grade Ore		Low-Grade Ore	
Year▼	Tonnes (kt)	ATD (km)	Tonnes (kt)	ATD (m)	Tonnes (kt)	ATD (m)	Tonnes (kt)	ATD (m)	Tonnes (kt)	ATD (m)	Tonnes (kt)	ATD (m)
Pre-Oper,	400	1,373	20	1,505	2834	1,335		-	-	-	-	-
1	1,400	1,283	52	1,649	2744	1,431	1,170	30	94	800		-
2	1,200	1,392	42	1,780	2233	1,491	1,300	30	100	800		-
3	1,187	1,731	39	2,084	2080	1,890	1,300	30	100	800		-
4	1,200	1,301	126	1,476	2584	1,291	1,300	30	100	800		-
5	1,200	1,335	110	1,658	1901	1,472	1,300	30	102	800		-
6	1,200	1,339	57	1,732	284	1,475	1,300	30	100		-	-
7	229	1,890	10	2,356	31	2,556	915	30			455.7	900

1.8.1 Operational Production Mining

The mining operation for the Matupá' Gold Project uses conventional open pit mining. The mine development plan allows access to grade levels to maximize gold production and provides operational flexibility by mining several benches simultaneously.

The waste rock comprises soil, saprolite, altered rock mass, and fresh rock. The excavation plan for these deposits is to drill and blast, with explosives, all fresh rock and 30% of the saprolite. Load and haulage will be performed mainly by hydraulic excavators, backhoes, and front-end loaders, and material transported by trucks (vocational).

Benches will be configured as follows:

- A minimum mining width of 30 m on a 10 m-high bench is used, including a final bench access incorporating an operational mining width of 15 m to maximize access to the mineralized zone.
- The waste and ore benches will be mined as 5 m thick layers, leaving a designed 10 m maximum bench height.
- The ore and waste zones have been analyzed and it is possible to operate with a proper berm width and in-pit dumping operational space.
- The benches will have a slight decline from crest to the toe of the upper bench face slope, in the direction of the open side to drain rainfall and to maintain designed slope angles. A good drainage design inside the pit and for rainwater collection contribution areas around the pit, allow for the minimization of operational disturbances during heavy rain.

The processing plant is located about 1.0 km from the X1 pit.

The mining faces will be accessed by 15-m wide double lane roads with 10% gradient. All roads will have 2.0 cm/m transversal gradient, from the center to the lateral edge of the road, with drainage ditches along the roads. Road conditions must be compatible with good practices and safety for the operation of mining equipment.

The Matupá Project's gold mining concept is based on the application of conventional techniques for surface rock mass excavation with a maximum level of mechanization:

1. Grade control with dedicated drilling: sample collecting to provide good support to the grade control engineering and short-term mining plan. The technology being considered is Down the Hole hammer with reverse circulation.

2. Blastholes: the holes are going to be drilled, most probably by a hydraulic Top Hammer drilling rig.

3. Primary rock blasting: most of the rock, ore, and waste, will be fragmented by using explosives. The ore fragmentation has special requirements, specifically for the ore we are considering the use of electronic caps.

The present review considers that the mining operation will be carried out by a contractor using 70-t class operating weight hydraulic excavators in backhoe configuration, which will load 8 x 4 trucks with 22 m^3 dump box size (Struck), that means about 10% more for heaped capacity, for 58 t PBT (Total Gross Weight) but for a bigger truck capacity, a bigger dump box can be considered: market has already 66 t PBT capacity.

Mining is planned to be carried out in 10-m high benches. However, along the ore / waste contacts mining will be undertaken using 5 m high benches to improve mining selectivity.

1.9 RECOVERY METHODS

The stipulated capacity for the Matupá industrial circuit is 1.3 Mtpa for processing blends of Fresh Rock and Oxide ore types. The selected treatment flow sheet for Matupá includes crushing, grinding, gravity concentration, and intensive leaching, followed by leaching (leaching - carbon in leach), carbon adsorption, cyanide neutralization (Detox), tailing thickening, and filtering for final disposal in piles, as shown in Figure 1-6. Based on an extensive testing campaign, gold recovery was modelled as a function of the Life of Mine (LOM) gold grades.

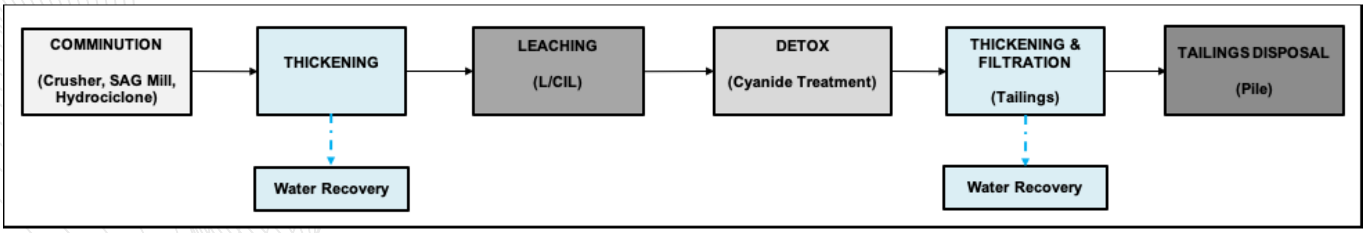


Figure 1-6 - Summary of Process Flow Sheet - Matupá Project.

The crushing circuit is designed for a nominal capacity of 3,562 tpd and 70% availability. The run of mine (ROM) will be hauled and dumped in stockpiles, reclaimed with front-end loaders into the crushing feed hopper that is equipped with a static grizzly for retaining the oversize material, while a mobile rock breaker is used to break oversize rocks. From the hopper a vibrating grizzly feeder modulates the feeding flow rate, and separates material into coarse (oversize) and relatively fine (undersize) fractions. The former size flows by gravity to the primary jaw crusher chamber, while the fine material, together with the primary crusher discharge, is conveyed to a surge bin. Given that the crushing and milling circuits are designed according to different availabilities, an excess of crushed material will result when the crushing plant is fully operational. This excess material will be piled in a dedicated stockpile and reclaimed by a front-end-loader to a reclaim bin equipped with a vibrating feeder that also feeds the milling circuit. Based on selected ROM size distribution, equipment design, and circuit simulations the predicted crushing circuit P_{80} is 90 mm.

The single stage griding circuit will include a high-aspect semi-autogenous ("SAG") mill operating in a closed configuration with hydrocyclones. The grinding circuit was designed on the basis of feed and product P_{80} of 90 mm and 0.125 mm respectively. The fresh feed reclaimed from the crushing plant surge bin is conveyed to the SAG mill, whose discharge pulp flows to a dedicated trommel screen. The material retained in the trommel screen (pebbles) is conveyed back to the SAG mill feed, whereas the trommel undersize gravitates to an underneath sump, from which it is pumped to a single hydrocyclones nest. The relatively coarse fraction (underflow) will be split in two fractions. The first will flow through the gravity concentration stage, whose tailings will flow to the SAG mill feed. The second fraction will flow straight back to the SAG mill feed. The gravity concentration circuit will include a scalp screen, a centrifugal concentrator, and an intensive leaching reactor. The hydrocyclones nest overflow is the grinding circuit product. The hydrocyclones overflow will be directed to a trash screen, where undersize material will flow to a thickener to increasing the concentration of solids prior to processing in a leaching-carbon-in-leach ("L-CIL") circuit.

The leach-adsorption circuit will consist of two leach tanks and six carbon-in-leach ("CIL") tanks. Mechanical agitation installed in all tanks will maintain the suspension of solids, as well as an adequate reagent homogenization. Fresh and regenerated carbon from the carbon regeneration circuit will be added to the CIL circuit for gold and silver adsorption. Carbon will flow counter-current to the slurry flow by pumping slurry and carbon. Slurry from the last CIL tank will gravitate to the cyanide detoxification tanks. Once a day, the pulp from the first carbon tank will be pumped into a dedicated screen to separate the loaded carbon from the pulp; the carbon will be processed through to the acid washing and a Zadra elution circuit. After regeneration, the carbon will return to the circuit passing through a dewatering screen.

Both the elution and intensive leaching solutions will be pumped to the pregnant solution tank for feeding the electrowinning cell. The sludge gold-rich cathodes will be washed, filtered and dried. The dry material obtained will be mixed with smelting fluxes and smelted in a furnace to produce gold doré (bullion).

The pulp from the leaching and adsorption circuit will flow by gravity to the cyanide neutralization circuit by using the SO_2/air method (Detox or Inco). The pulp from the neutralization circuit will flow to a safety screen to retaining any loaded carbon, which will be stored for recirculation in the CIL circuit. The screen undersize material will be pumped to the tailing thickener.

Tailings resulting from the Detox circuit will be transferred to a high-rate thickener, whose underflow, at 60% w/w (weight per weight) solids, will be transferred to the filtration circuit where a horizontal vacuum filter will reduce the cake moisture to 21-23%. The filtering water and the thickening water will be recirculated within the processing plant. The filtered product will be transferred to disposal piles. Water runoff from these piles will also be recirculated in the processing plant. The filtered tailings will be transferred to the disposal area (Dry Stacking).

The majority of water consumed in the processing plant is designed to derive from recirculation within the industrial installation. Make-up water will be pumped from the Porcão River, which is located close to the future industrial installations. Water from the Porcão River will also be used for reagent preparation, elution, pump sealing water, as well as for the potable water treatment unit.

The main reagents to be used in the Matupá industrial plant are: sodium cyanide, hydrated lime, sodium hydroxide, sodium metabisulfite, hydrochloric acid, and copper sulfate pentahydrate.

1.10 PROJECT INFRASTRUCTURE

The overall site plan (see Figure 1-7) shows the major project facilities, including the open pit mines, tailings management facility ("TMF"), waste rock facilities, mine services, and access roads. Access to the facility is from the east side of the property from the existing access road. Main access will be via the security gate near the process plant.

The site will be fenced to deter access by unauthorised people. The process plant is located east of the X1 Deposit.

Site selection took into consideration the following factors:

- upgrade and utilise the existing access road to reach the site;
- locate mining, administration and processing plant staff offices close together to limit walking distances between them; and
- locate the ready line close to the mining administration/office area and changehouse.

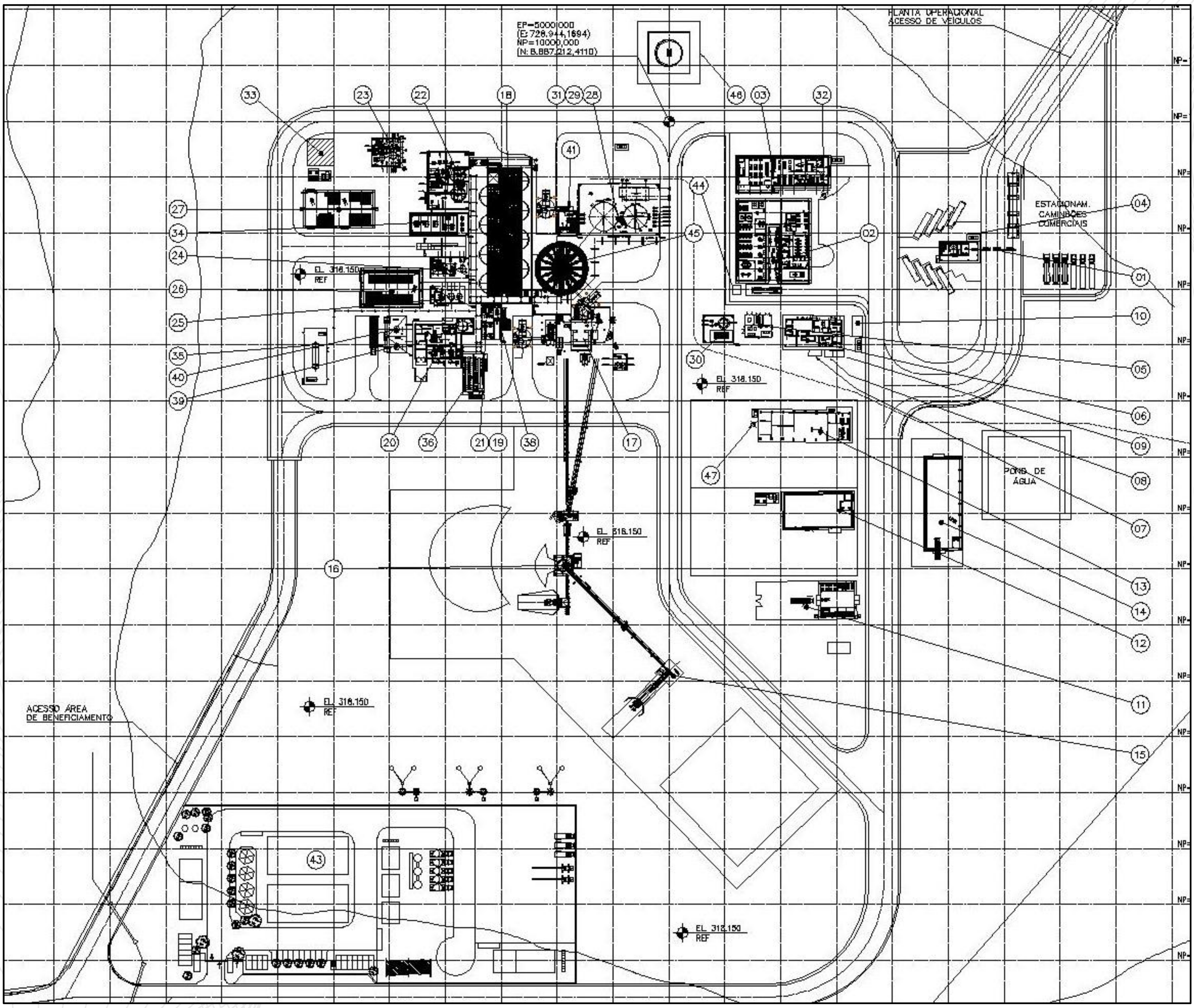


Figure 1-7 - Overall Site Plan (MTP-B-DS-0000-P-0002-RB).

1.11 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The Environmental Impact Assessment and respective Environmental Impact Assessment Report (“EIA/RIMA”), were filed with SEMA-MT on November 30, 2021 in compliance with the Term of Reference. Aura presented the Matupá Project to the technical team of the environmental agency on March 15, 2022 and the Public Hearing took place on May 10, 2022 with good reception by the mayors of Matupá and Guarantã do Norte, and by the local community. The next steps will be the visit of the SEMA-MT technical team to the Project area, analysis of the EIA/RIMA, and the issues raised at the public hearing, and finally, the issuance of the Technical Opinion and Preliminary License.

It is estimated that the Preliminary License (“LP”) will be issued by SEMA-MT and endorsed by the Mato Grosso State Environment Council (CONSEMA), around the third and fourth quarter (Q3, Q4) of 2022. The Installation License will be required between the third and fourth quarter of 2022 and its issuance is scheduled for the end of the first semester of 2023. The Operating License will be required four months prior to completion of construction.

During this process, accessory licenses will be required, such as the Water Use Grant, Vegetation Suppression Authorization, and others that may be required to complete the licensing process. The studies and projects necessary to support the application for these licenses are in progress for proceeding in the second half of 2022.

The potential for acid drainage generation was investigated in representative waste rock and tailings samples. Static prediction tests (MABA and NAG) and kinetic tests (Free Draining Kinetic Column Leach) were conducted. The static tests showed potential for generating acid drainage, but the kinetic tests did not confirm the potential, presenting a more positive scenario than previously suggested by the static tests. However, the engineering project contains structures to contain any future contamination that may occur, such as drainage systems (internal and superficial) and lining the base of the waste rock piles, tailings and low-grade ore to avoid any contact with effluents, with the watercourses, and Sumps (Bains), for sedimentation of solids and eventual treatment of effluents that may be necessary.

Based on the evaluation of the engineering project and integrated analysis of the environmental diagnosis, identification and assessment of environmental impacts, proposal of environmental measures and programs and their prognosis, the Environmental Impact Study concluded that the Matupá Project is environmentally viable, provided that the identified negative effects are resolved and are not impediments to the project.

1.12 CAPITAL AND OPERATING COSTS

The CapEx study presented has a variation of +15% and -10%. The CapEx estimate presented includes the cost for project execution, acquisition, construction and commissioning of all facilities. The estimate was based on basic engineering of the disciplines of mechanics, electrical, civil, instrumentation and pipes. In addition to the quantitative and definitions coming from the consolidated basic project, other definitions of scope were considered together with Aura Minerals, such as the values of pile construction, mine and other costs, including indirect. The values shown in the table are already with the application of the tax benefit, according to a study by EY Consulting.

Item	FEASIBILITY		
	Cost (USD)		%
Services	$	32,177,944	30%
Supplies	$	40,562,051	38%
Mine, pile, and Transmission Line	$	14,109,618	13%
Indirect costs	$	13,008,310	12%
Subtotal	$	99,821,923	93%
Contingency	$	7,300,848	7%
Total Investment	$	107,122,771	100%
Limit Inferior (-10%)	$	96,410,494	
Limit Superior (+15%)	$	123,191,187	

Operating costs are in the next table, in which the unit costs in tonnes of ROM/year are presented for labor, g&a, laboratory, access maintenance, equipment rental, water and sewage treatment plant, pile and mine.

Item	Cost (USD)		%
	USD / t (metric) ROM		
	$	22.71	100%
Labor (Fixed Costs)	$	3.53	16%
G&A (Fixed Cost)	$	1.69	7%
Laboratory (Fixed Cost)	$	1.26	6%

Access Maintenance (Fixed Cost)	$	-	0%
Equipment rental (Fixed Cost)	$	0.02	0%
Energy (Variable Costs)	$	2.18	10%
Reagents and Consumables (Variable Costs)	$	7.74	34%
Maintenance	$	1.12	5%
Water and sewage treatment plant	$	0.01	0%
Pile	$	1.30	6%
Mine	$	3.84	17%

1.13 ECONOMIC ANALYSIS

The financial model adopts the concept of project free cash flow, in which all the Project's cash generation capacity is evaluated by countering this flow with a weighted discount rate ("WACC") which reflects the average cost of sources of funds (cost of equity and third parties). The amounts in the cash flow were expressed in thousand US Dollars (US$ x 1,000) and on a real basis (without inflation).

Based on the assumptions adopted, the post-tax net present value ("NPV") of Aura Minerals Gold Almas Project base case amounts to US$ 96,128 million, at a Discount Rate of 5.0%.

The internal rate of return ("IRR") is 27.5% and the annual average EBITDA (from year 1 to year 7, full run rate production period) is US$ 280,318 million. Payback after the start-up of operations is 2.04 years.

The leveraged IRR calculation was performed considering a debt of 50% leverage and the calculated value was 49.9%.

The results are summarized in Table 1-12 and the operating income statement, and the Project cash flow are respectively presented in Table 1-13 and Table 1-14.

Table 1-12 – Financial Results Summary.

VALUATION - BASIC PROJECT		
NPV	**96,128**	**US$ x 1.000**
IRR	**27.5%**	
Leveraged IRR	**49.9%**	
Profitability Index	**1.95**	
Discounted Payback	**4.56**	**years**
Simple Payback (Including Start-Up)	**4.29**	**years**
Simple Payback (After Start-Up)	**2.04**	**years**
Discont Rate	**5.0%**	
Invest in the Project?	**Yes**	

Table 1-13 – Project Cash Flow.

EBITDA	**R$ x 1.000**			**252,617**	**318,339**	**303,817**	**268,217**	**95,636**	**128,553**	**54,220**
	US$ x 1.000			**51,652**	**63,386**	**59,726**	**52,013**	**18,391**	**24,722**	**10,427**
CAPEX	R$ x 1.000	(220,673)	(331,009)	-	-	-	-	-	-	-
	US$ x 1.000	(42,849)	(64,274)	-	-	-	-	-	-	-
CAPEX Sustaining	R$ x 1.000			(8,530)	(14,545)	(37,644)	(15,678)	(650)	-	(627)
	US$ x 1.000			(1,656)	(2,797)	(7,239)	(3,015)	(125)	-	(121)
Working Capital Variation	R$ x 1.000			(30,129)	306	596	(81)	4,129	1,759	23,713
	US$ x 1.000			(5,850)	59	115	(16)	794	338	4,560
Mine Closure Cost (Present Value form Aura info - 9 years projection)	R$ x 1.000			-	-	-	-	-	-	(36,196)
	US$ x 1.000			-	-	-	-	-		(6,961)
Salvage Value	R$ x 1.000			-	-	-	-	-	-	85,232
	US$ x 1.000			-	-	-	-	-	-	16,391
Income Tax / Social Contribution	R$ x 1.000			(23,242)	(33,351)	(33,686)	(27,437)	(777)	(5,783)	-
	US$ x 1.000			(4,513)	(6,414)	(6,478)	(5,276)	(149)	(1,112)	-
Capex Tax Recovery	R$ x 1.000			-	-	-	-	-	-	-
	US$ x 1.000			-	-	-	-	-	-	-
FCFF Nominal	**R$ x 1.000**	**(220,673)**	**(331,009)**	**204,106**	**282,017**	**239,843**	**227,274**	**98,337**	**124,530**	**126,341**
	US$ x 1.000	**(42,849)**	**(64,274)**	**39,632**	**54,234**	**46,124**	**43,707**	**18,911**	**23,948**	**24,296**
WACC	5.00%	0.98	0.93	0.89	0.84	0.80	0.76	0.73	0.69	0.66
FCFF Discounted	**R$ x 1.000**	**(215,354)**	**(307,649)**	**180,668**	**237,745**	**192,564**	**173,784**	**71,612**	**86,368**	**83,452**
	US$ x 1.000	**(41,816)**	**(59,738)**	**35,081**	**45,720**	**37,032**	**33,420**	**13,772**	**16,609**	**16,048**
FCFF Discounted Acumulated	**R$ x 1.000**	**(215,354)**	**(523,004)**	**(342,335)**	**(104,590)**	**87,975**	**261,758**	**333,371**	**419,739**	**503,190**
	US$ x 1.000	**(41,816)**	**(101,554)**	**(66,473)**	**(20,753)**	**16,279**	**49,699**	**63,471**	**80,080**	**96,128**

Table 1-14 – Operating Income Statement.

Gross Revenue	**R$ x 1.000**	**454,517**	**508,555**	**487,799**	**450,659**	**256,449**	**265,583**	**113,786**
	US$ x 1.000	**88,256**	**97,799**	**93,808**	**86,665**	**49,317**	**51,074**	**21,882**
Cash Cost	**R$ x 1.000**	**(156,138)**	**(144,322)**	**(143,424)**	**(150,022)**	**(141,266)**	**(118,549)**	**(47,148)**
	US$ x 1.000	**(30,318)**	**(27,754)**	**(27,582)**	**(28,850)**	**(27,167)**	**(22,798)**	**(9,067)**
Mining Costs	R$ / ton							
	R$ x 1.000	(32,272)	(32,166)	(31,532)	(35,633)	(30,159)	(15,577)	(4,585)
	US$ x 1.000	(6,266)	(6,186)	(6,064)	(6,853)	(5,800)	(2,996)	(882)
Processing Costs	R$ x 1.000	(123,865)	(112,156)	(111,893)	(114,389)	(111,107)	(102,972)	(42,563)
	US$ x 1.000	(24,052)	(21,568)	(21,518)	(21,998)	(21,367)	(19,802)	(8,185)
Contingencies	0%							
	R$ x 1.000	-	-	-	-	-	-	-
	US$ x 1.000	-	-	-	-	-	-	-
Freight / Refining	**R$ x 1.000**	**(1,648)**	**(1,954)**	**(1,874)**	**(1,732)**	**(985)**	**(1,021)**	**(437)**
	US$ x 1.000	**(320)**	**(376)**	**(360)**	**(333)**	**(190)**	**(196)**	**(84)**
Freight to Refinery	US$ / oz	5.74	5.74	5.74	5.74	5.74	5.74	5.74
Refining	US$ / oz	0.6	0.6	0.6	0.6	0.6	0.6	0.6
Gross Profit (considering Depreciation)	**R$ x 1.000**	**217,920**	**281,472**	**259,616**	**210,643**	**23,695**	**55,418**	**(52,937)**
	US$ x 1.000	**42,315**	**54,129**	**49,926**	**40,508**	**4,557**	**10,657**	**(10,180)**
SG&A | Despesas Gerais e Administrativas	R$ x 1.000	(14,317)	(14,317)	(14,317)	(12,169)	(9,306)	(7,874)	(7,874)
	US$ x 1.000	(2,780)	(2,753)	(2,753)	(2,340)	(1,790)	(1,514)	(1,514)
CFEM	R$ x 1.000	(6,818)	(7,628)	(7,317)	(6,760)	(3,847)	(3,984)	(1,707)
	US$ x 1.000	(1,324)	(1,467)	(1,407)	(1,300)	(740)	(766)	(328)
Royalties	R$ x 1.000	(9,590)	(10,728)	(10,290)	(9,507)	(5,410)	(5,602)	(2,400)
	US$ x 1.000	(1,862)	(2,063)	(1,979)	(1,828)	(1,040)	(1,077)	(462)
Basis Calculation	US$ x 1.000	86,612	95,956	92,040	85,032	48,388	50,111	21,470
Gross Revenue	US$ x 1.000	88,256	97,799	93,808	86,665	49,317	51,074	21,882
Freight	US$ x 1.000	(290)	(340)	(326)	(301)	(172)	(178)	(76)
Refining	US$ x 1.000	(30)	(36)	(34)	(32)	(18)	(19)	(8)
Taxes (CFEM)	US$ x 1.000	(1,324)	(1,467)	(1,407)	(1,300)	(740)	(766)	(328)
Interest expenses	R$ x 1.000	(13,390)	(11,267)	(6,760)	(2,253)	-	-	-
	US$ x 1.000	(2,600)	(2,167)	(1,300)	(433)	-	-	-
EBIT	**R$ x 1.000**	**173,806**	**237,532**	**220,932**	**179,954**	**5,133**	**37,958**	**(64,918)**
	US$ x 1.000	**33,749**	**45,679**	**42,487**	**34,607**	**987**	**7,300**	**(12,484)**
Depreciação & Amortização	R$ x 1.000	78,812	80,807	82,885	88,263	90,502	90,595	119,138
	US$ x 1.000	15,303	15,540	15,939	16,974	17,404	17,422	22,911
EBITDA	**R$ x 1.000**	**252,617**	**318,339**	**303,817**	**268,217**	**95,636**	**128,553**	**54,220**
	US$ x 1.000	**51,652**	**63,386**	**59,726**	**52,013**	**18,391**	**24,722**	**10,427**

The sensitivity analysis shows the impact of the variation of the gold price, exchange rates, operating costs (OpEx), Recovery Ratio, weighted average cost of capital (WACC), and capital costs (CapEx) upon the Project net present value (NPV) and internal rate of return (IRR). The analysis encompasses the following range of variation in the key inputs:

- Gold price: ±20%.
- Exchange Rate: ±20%.
- OpEx (Cost): ±20%.
- WACC (Discount Rate): ±20%
- CapEx: ±20%.

In assessing the sensitivity of the Project returns, each of these parameters is varied independently of the others. Scenarios combining beneficial or adverse variations simultaneously in two or more variables will have a more marked effect on the economics of the Project than will the individual variations considered. The sensitivity analysis has been conducted assuming no change to the mine plan or schedule.

Figure 1-8 illustrates the results of the sensitivity analysis for Project NPV (after tax) and these effects for each of the critical variables. Figure 1-9 presents the same scenario for the IRR. NPV results are reported at a discount rate of 5.0%.

NPV (US$ MM)	80%	90%	95%	100%	105%	110%	120%
Gold Price	31.21	63.88	80.08	96.13	112.11	128.04	159.92
Exchange Rate	63.70	81.80	89.42	96.13	102.22	107.73	117.36
OpEx (Costs)	121.60	108.88	102.53	96.13	89.76	83.25	70.21
Recovery Ratio	31.46	64.00	80.14	96.13	112.05	127.92	159.67
WACC	104.43	100.23	98.18	96.13	94.19	92.24	88.45
CapEx	111.77	103.97	100.07	96.13	92.27	88.31	80.34

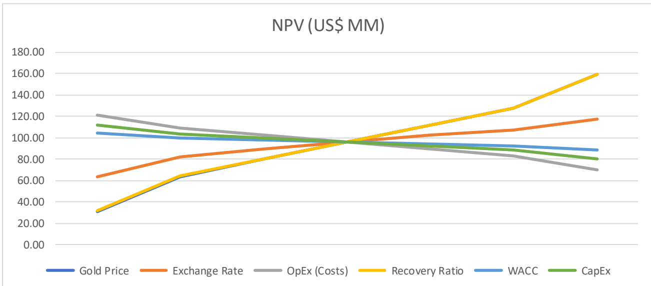


Figure 1-8 - Sensitivity Analysis Graph – NPV.

IRR (%)	80%	90%	95%	100%	105%	110%	120%
Gold Price	13.06%	20.63%	24.14%	27.48%	30.72%	33.84%	39.82%
Exchange Rate	20.72%	24.57%	26.13%	27.48%	28.70%	29.79%	31.65%
OpEx (Costs)	32.46%	30.01%	28.76%	27.48%	26.20%	24.87%	22.13%
Recovery Ratio	13.13%	20.65%	24.15%	27.48%	30.70%	33.81%	39.77%
WACC	27.49%	27.49%	27.49%	27.48%	27.49%	27.49%	27.49%
CapEx	36.25%	31.47%	29.40%	27.48%	25.74%	24.11%	21.20%

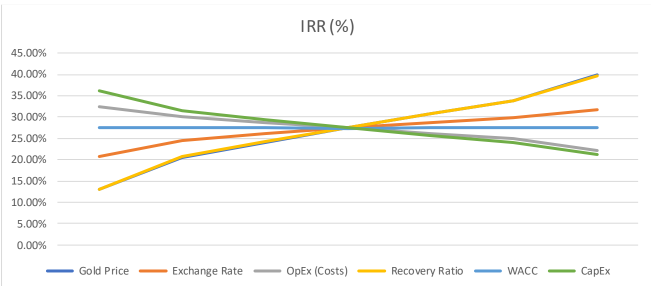


Figure 1-9 - Sensitivity Analysis Graph – IRR.

Considering the WACC of 5% and the feasibility analysis of the Project through the applied methodology, the rate of return and the NPV are good enough to consider the viability of the project. Further analysis may improve the perceived Project viability.

1.14 CONCLUSION

The Matupá Gold Project Feasibility Study has demonstrated that at a gold average price of $1,664/oz, an investment of US$107M would be required to build a processing plant and associated facilities, which over a mine life of 7 years would yield a return on investment of 27.5%. At a discounted rate of 5% the “all equity” Net Present Value after taxation is $96.128M. Average annual gold production is expected to be 41,889 oz.

1.15 RECOMMENDATIONS

This Feasibility Study Report and the results herein have been verified and approved by the Qualified Persons: Mr. Luiz Pignatari, P.Eng. (QP, Mining); Dr. Homero Delboni Jr. (QP, Metallurgy); and Mr. Farshid Ghazanfari. P.Geo. (QP, Geology and Resources).

Specific recommendations can be found in section 26.

2. INTRODUCTION AND TERMS OF REFERENCE

The Matupá Gold Project is located in northern part of the state of Mato Grosso, Brazil. The previous technical report (NI 43-101) was prepared by GE 21 Consultoria Mineral Ltda in December 2021 (Independent Technical Report – Preliminary Economic Assessment, Matupá Project, Mato Grosso, Brazil). The Matupá Gold Project was acquired by Aura from its previous owner, Rio Novo Mineracao Ltda., in 2018. Rio Novo performed additional infill drilling after the completion of the 2010 technical report, this additional data was not included in the previous report. Since Acquisition in 2018, Aura reviewed all information received from Rio Novo Mineracao, in detail. Aura then preformed database validation, relogged of some existing drill holes in the X1 Deposit area, resurveyed a set of selected drill hole collars, and updated the Mineral Resource Estimate for the X1 Deposit with additional infill drill holes drilled during 2011 and 2012. Aura also performed additional metallurgical work and a preliminary mining study. The results of these studies are incorporated into the preparation of this current report.

Aura, in collaboration with Promon Engenharia Ltda ("Promon"), EDEM, HDA Serviços S/S Ltda.("HDA") and a few independent consultants prepared an NI 43-101-compliant Feasibility Study Report. This report is a new report for the Matupá Gold Project considering all required changes in technical information and reflecting the current financial conditions. This report was prepared to meet the requirements of Canadian National Instrument 43-101 ("NI 43-101") and conforms to Form 43-101 F1 for Qualifying Reports. This new Technical Report meets the requirements of NI 43-101.

2.1 PROJECT BACKGROUND

The Matupá Gold Project is located in the municipality of Matupá, in Mato Grosso State, Brazil. The X1 gold deposit is the main focus of this Feasibility Study and will be the primary source of potential ore. The Matupá Gold Project, besides the X1 Deposit, includes some satellite deposits and targets, such as Serrinhas and Guaranta Ridge which are briefly discussed in the exploration section of this report but are not incorporated into current study.

The following activities and project developments were completed by Aura between 2018 and 2021:

- Database validation and QA/QC review.
- Collar resurveying of selected drill hole collars.
- New ground survey of topography and topographic surfaces (large and deposit scales).
- Mineral Resource estimation updates for the X1 Deposit.
- Preliminary mining studies and pit optimization.
- Recovery of metallurgical samples by diamond core drilling and the completion of metallurgical test work programs to determine crushing and grinding characteristics of the deposits, as well as to develop a process for the recovery of gold.
- Preliminary estimates of capital and operating expenditures for the Project, a discounted cash flow for the life of the Project, a project implementation plan, and a site rehabilitation plan for the decommissioning of the Project.
- Acquiring permits for a wildlife survey, specifically a fish and insect field survey as part as the EIA/RIMA (Environmental Impact Study and Environmental Impact Report) general field survey.

2.2 QUALIFIED PERSONS

The following individuals, by virtue of their education, experience, and professional association, are considered Qualified Persons (QP) as defined in NI 43-101 and are members in good standing of appropriate professional institutions.

The QPs are responsible for the specific sections as follows:

- Farshid Ghazanfari, M.Sc., (P.Geo), Member of the Association of Professional Geologists of Ontario, Canada (PGO), Aura Mineral Geology and Resource Director (Geology), is the QP responsible for Sections 6, 7, 8 , 9, 10, 11, 12, 14, 23, and a co-author for Sections 4 and 5, as well as providing summaries for Sections 1, 3, 24, 25, 26, 27 and 28.

- Luis Pignatari, (P.Eng), Professional Engineer, EDEM Mining Consultants (Engenharia de Minas ME), is the QP responsible for Sections 15, 16, 19 and 20, and a co-author for Sections 4, 5 and 18, as well as providing summaries for Sections 1, 3, 24, 25, 26, 27 and 28.

- Dr. Homero Delboni, Jr. Ph.D. (MAusIMM – CP Mettallurgy), Independent Senior Consultant (Metallurgy), is the QP responsible for Sections 2, 13, 17, 21 and 22, and a co-author for Section 18, as well as providing summaries for Sections 1, 3, 24, 25, 26, 27 and 28.

Table 2-1 - Responsibilities Matrix.

RESPONSIBILITIES MATRIX			
Chapter	**Contents**	**Responsible QP**	**Site Visit**
1	Summary	ALL QPs	
2	Introduction	Homero Delboni	
3	Reliance on other Experts	ALL QPs	
4	Property Description and Location	Farshid G. and L Pignatari	
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Farshid G. and L Pignatari	
6	History	Farshid G.	June 24 to June 29 of 2019 Oct16 to Oct 18 of 2021
7	Geological Setting and Mineralization	Farshid G.	
8	Deposit Types	Farshid G.	
9	Exploration	Farshid G.	
10	Drilling	Farshid G.	
11	Sample Preparation, Analysis and Security	Farshid G.	
12	Data Verification	Farshid G.	
13	Mineral Processing and Metallurgical Testing	Homero Delboni	
14	Mineral Resource Estimates	Farshid G.	
15	Mineral Reserve Estimates	L Pignatari	July 13th to 15th 2022
16	Mining Methods	L Pignatari	
17	Recovery Methods	Homero Delboni	
18	Project Infrastructure	L. Pignatari and Homero Delboni	
19	Market Studies and Contracts	L Pignatari	
20	Environmental Studies, Permitting and Social or Community Impact	L Pignatari	
21	Capital and Operating Costs	Homero Delboni	
22	Economic Analysis	Homero Delboni	
23	Adjacent Properties	Farshid G.	
24	Other Relevant Data and Information	ALL QPs	
25	Interpretation and Conclusions	ALL QPs	
26	Recommendations	ALL QPs	
27	References	ALL QPs	
28	Certificates of Qualified Persons	ALL QPs	

2.3 QUALIFIED PERSONS SITE VISITS

Mr. Farshid Ghazanfari, QP (Aura, Geology and Mineral Resources) has been involved with the Matupá Gold Project since 2018, as well as during the due diligence assessment by Aura prior to Aura's acquisition. Mr. Ghazanfari visited the Matupá Gold Project on a few occasions during the past three years. His most recent site visit was from October 16 to 18, 2021.

Mr. Luiz Eduardo Campos Pignatari, QP (Mineral Reserves), to ensure the procedures used meet industry accepted standards, visited the Matupá Gold Project from July 13th to 15th 2022.

2.4 TERMS AND DEFINITIONS

The following terms and definitions are used in this report.

- Aura refers to Aura Minerals 360 Mining.
- ANM refers to the (Agência Nacional de Mineração de Brazil) (see Section 4.3).
- Ausenco refers to Ausenco Mining Consultants (Toronto Office, Canada).
- Rio Novo refers to Rio Novo Mineracao Ltda.
- EDEM refers to Engenharia de Minas ME (Sao Paulo, Brazil).
- HDA refers to HDA Serviços S/S Ltda. (São Paulo, Brazil)
- GE21 refers to GE21 Consultoria Mineral Ltda. (Belo Horizonte, Brazil).
- Promon refers to Promon Engenharia Ltda (São Paulo, Brazil).

2.5 UNITS, SYMBOLS AND ABBREVIATIONS

Aura has based all measurements in the metric system, exceptions to this primarily list both English and Metric standards. Currencies are generally based on the October 22, 2020, US Dollar, with the conversion exchange rate of 5.143 Brazilian Reals per 1 US Dollar for the long-term exchange rate unless otherwise stated. Dollars are United States Dollars, and weights are in metric tonnes of 1,000 kilograms (2,204.62 pounds). Coordinates in this report are in the South American Datum (1969), UTM Zone 21 South – a map projection used throughout this report.

The abbreviations used in this report are described in Table 2-2.

Table 2-2 - Units, Symbols and Abbreviations.

UNITS, SYMBOLS AND ABBREVIATIONS	
Unit	**Meaning**
%	Percent(age)
$ / USD / US$	United States Dollars
AA/AAS	Atomic Adsorption Spectroscopy
AARL	Anglo American Research Laboratories
Ai	Abrasion index

UNITS, SYMBOLS AND ABBREVIATIONS	
Unit	**Meaning**
AISC	All-In-Sustaining Costs
AIG	Australian Institute of Geoscientists
AusIMM	Australasian Institute of Mining and Metallurgy
AFRIMM	Additional of Freight
ANM	National Mining Agency (*Agência Nacional de Mineração*)
AT	Assay tonne
Au	Gold
R$	Brazilian Reais
BWI	Bond Work Index
$Ca(OH)_2$	Calcium hydroxide
CapEx	Capital Expenditure
CFEM	Financial Compensation for the Exploration of Mineral Resources (*Compensação Financeira pela Exploração de Recursos Minerais*)
CIL	Carbon-in-Leach
CIP	Carbon-in-Pulp
CN	Cyanide
CNwad	Weak acid dissociable cyanide
CP	Chartered Professional
CPG	Certified professional geologist
CRM	Certified reference material
CSLL	Social Contribution (*Contribuição Social Sobre o Lucro Líquido*)
DCF	Discounted Cash Flow
DDH	Diamond Drill Hole
DWI	Drop-Weight Index
EBIT	Earnings Before Interest and Taxes
EBITDA	Earnings Before Interest, Taxes, Depreciation, and Amortization
FAIG	Fellow of the Australian Institute of Geoscientists
FCFF	Free Cash Flow to Firm
F_{80}	Feed- 80% passing particle size
FEL	Front End Loaded Project Evaluation Study
ft	feet
ft^3	cubic feet
g	gram
Ga	Gigaannum, a unit of time equal to one billion years
g/cc	gram per cubic centimeter
g/cm^3	gram per cubic centimeter
g/L	gram per liter
g/t	gram per metric tonne
G&A	General and Administrative
GPS	Global Positioning System

UNITS, SYMBOLS AND ABBREVIATIONS	
Unit	**Meaning**
GRG	Gravity Recoverable Gold
xH:xV	horizontal to vertical ratio where x = numeric values
HTS Code	Harmonized Tariff Schedule Code
Hz	Hertz
IBC	Intermediate Bulk Container
ICP	Inductively Coupled Plasma
ID^2	Inverse Distance Squared
ILR	Intensive Leach Reactor
in	Inch
IRPJ	Income Tax (*Imposto de Renda de Pessoa Jurídica*)
IRR	Internal Rate of Return
IPI	Taxes over industrialized products (*Imposto sobre Produtos Industrializados*)
ISO	International Standards Organization
ISU	International System of Units
ITR	Independent Technical Report
JORC	Australasian Code for Reporting of Exploration Results, Mineral Resources, and Ore Reserves
k	thousands
kg	kilogram
kg/t	kilogram per metric tonne
km	kilometer
kPag	kilopascals, gauge
kV	kilovolts
kW	kilowatts
kWh/t	kilowatt-hour per metric tonne
LI	Installation License (*Licença de Instalação*)
LMC	Linear co-regionalization model
LO	Operating License (*Licença de Operação*)
LOM	Life of Mine
LP	Preliminary License (*Licença Preliminar*)
M	millions
m	meter
m/h	meter per hour
m^2/tpd	square meter per tonnes per day
m^3	cubic meter
Ma	Megaannum, a unit of time equal to one million years
MCW	Metres of Column of Water
mg/L	milligram per liter

UNITS, SYMBOLS AND ABBREVIATIONS	
Unit	**Meaning**
mm	millimeters
Mt or mt	Million metric tonnes
Mt/a	Million metric tonnes per annum (year)
mtpy	Million metric tonnes per year
mV	millivolt
MW	Megawatts
MVA	megavolt amperes
NI 43-101	Canadian National Instrument 43-101
NN	Nearest Neighbor
NPI	Net Profitability Index
NPV	Net Present Value
OK	Ordinary Kriging
ONAN/ONAF	Oil Natural Air Natural/Oil Natural Air Forced
OpEx	Operational Expenditure
oz or toz	Troy ounces
P_{80}	Product- 80% passing particle size
PIS and COFINS	Recoverable taxes (*Programa de Integração Social – Contribuição para o Financiamento da Seguridade Social*)
ppb	parts per billion
ppm	parts per million
QA/QC	Quality assurance/Quality control
QP	Qualified person
Q95	the flow that is present in the river during at least 95% of the time
R$ / BRL$	Brazilian Real
RC	Reverse circulation drilling
ROM	Run of Mine
SAG mill	semi-autogenous grinding mill
SG	Specific Gravity
SI	International System of Units
SMBS, $Na_2S_2O_5$	Sodium Meta-bisulfite
SMC test	SAG mill comminution test
SO_2	Sulfur dioxide
SUDAM	Amazon Development Superintendent Agency (*Superintendência de Desenvolvimento da Amazônia*)
t	metric tonne, tonne (1,000 kg or 2,204.6 lbs)
t/a or tpa	metric tonnes per annum
t/d or tpd	metric tonnes per day
t/h or tph	metric tonnes per hour
t/m³	tonnes per cubic meter
TDA	Total De-clustered Average
TDS	Total Dissolved Solids

UNITS, SYMBOLS AND ABBREVIATIONS	
Unit	**Meaning**
TMF	Tailings Management Facility
toz	Troy ounce
Tpa or tpy	Metric tonnes per annum/year
tph/m^2	Metric tonne per hour per square meter
TSF	Tailings Storage Facility
TSS	Total Suspended Solids
UTM	Universal Transverse Mercator coordinate system
xV:xH	vertical to horizontal ratio where x = numeric values
VAT	Value-added tax
WACC	Weighted Average Cost of Capital
w/v	Weight by volume ratio
w/w	Weight by weight ratio
XRF	X-Ray Fluorescence
y	Year
yd^3	cubic yards
µm	micron or micrometer

Table 2-3 - Common Chemical Symbols.

COMMON CHEMICAL SYMBOLS	
Al	Aluminum
Ca	Calcium
Cl	Chlorine
Co	Cobalt
Cu	Copper
Au	Gold
Fe	Iron
Pb	Lead
Mg	Magnesium
Mn	Manganese
Mo	Molybdenum
Ni	Nickel
O_2	Oxygen
K	Potassium
Ag	Silver
S	Sulfur
SiO_2	Silicon dioxide
Ti	Titanium

Table 2-4 - Units Conversion.

UNITS CONVERSION			
1 troy ounce (oz)		31.1034768 grams (g)	
1 metric tonne	1,000 kilograms	2,204.62 pounds	
1 gram per metric tonne		0.0292 troy ounces per short ton	
1 foot (ft)		0.3048 meters (m)	
1 mile (mi)	1.6093 kilometers (km)	5,280 feet	
1 meter	39.370 inches (in)	3.28083 feet	
1 kilometer	0.627371 miles	3,280 feet	
1 acre (ac)		0.4047 hectares	
1 square kilometer (sq. km)	247.1 acres	100 hectares	0.3861 square miles
Degrees Fahrenheit (°F)	32 X 5/9	Degrees Celsius (°C)	
1 ppm	1 g/t	0.0001%	1,000 ppb

3. RELIANCE ON OTHER EXPERTS

This report was prepared by Aura and is based in part on information presented in the 2021 PEA report under title of "Independent Technical Report, Preliminary Economic Assessment. Matupá Project, Mato Grosso State, Brazil, GE21 Ltda, September 30, 2021", on geological, geochemical, engineering, metallurgical, legal, environmental, and other reports and documents completed by others, as well as opinions from other persons. Some of these persons are not Qualified Persons under the definitions of NI 43-101.

Aura conducted surface land status evaluations and applied for environmental permits for the Project. Much of this work was conducted by persons who are not QPs. Mr. Luiz Pignatari, P.Eng. (QP, Mining), Dr. Homero Delboni Jr. (QP, Metallurgy) and Mr. Farshid Ghazanfari. P.Geo. (QP, Geology and Resources) have relied on this data, as necessary, to complete this report.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Matupá Gold Project area is located in the Alta Floresta Gold Province (AFGA), which lies in the extreme north-central part of Mato Grosso State, Brazil (Figure 4-1). The Project area encompasses an area surrounding the towns of Matupá and Guarantã do Norte, approximately 700 km north of Cuiabá, the Mato Grosso State Capitol and 200 km north of Sinop, an important commercial center and fourth largest city by population. The Matupá Gold Project refers to Aura's, and previously Rio Novo's and Aura's, on-going exploration, economic evaluation, and planned development by surface mining of gold deposits in the AFGA. This report focuses on the X1 gold deposit.

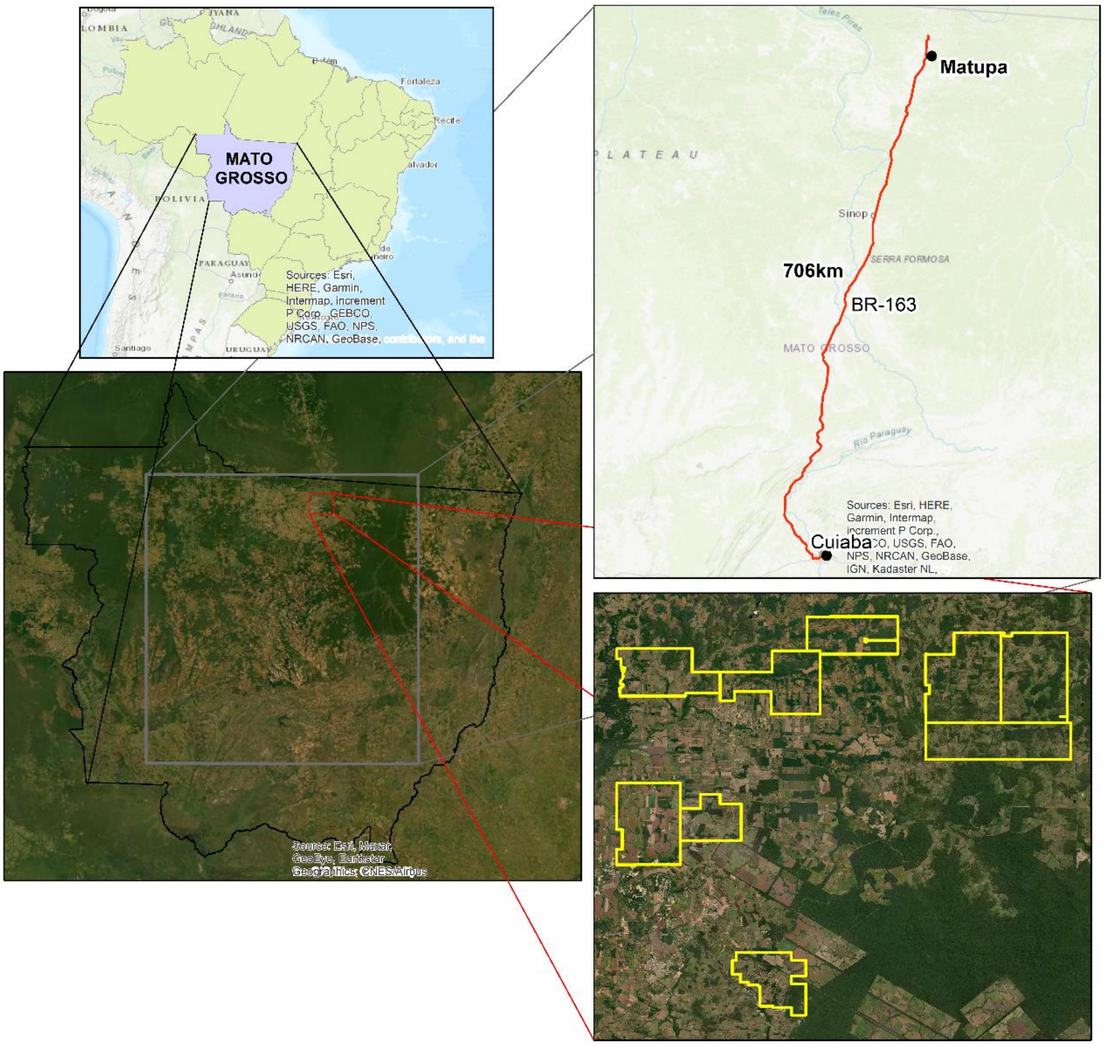


Figure 4-1 - The Matupá Gold Project Location (Source: Aura Minerals Inc).

The X1 Deposit is located near to Matupá city, approximately 11 km north of its urban area and approximately 11 km south of the town of Guarantã do Norte, both municipalities are located along Highway BR-163. All other known mineralization and exploration areas are highlighted on Figure 4-2.

This region contains numerous small scales, artisanal gold mining sites, locally termed garimpos, whose development preceded Rio Novo's exploration activities. The preponderance of gold deposits of the historical garimpos are associated with alluvium or lode quartz gold, associated with shear zones in the geological contact area of the granitoids and basic to intermediate rocks.

There is also a pluton-related disseminated style of mineralization, such as the X1 Deposit which is hosted in porphyritic granodiorites intruded by quartz feldspar porphyry, and the Serrinhas Target hosted in granodiorites. In addition, there are epithermal systems represented by high-grade veining and hydrothermal breccias hosted in acidic volcanic rocks. The approximate centers of both principal deposits in the Project area are given below in coordinates referenced to the South American Datum (1969), UTM Zone 21 South – a map projection used throughout this report.

- X1 Deposit 728117.80 m East, 8885898.28 m North.
- Serrinhas Target 733427.48 m East, 8865243.02 m North.

4.2 MINERAL RIGHTS

Aura holds the mineral rights for 9 properties, of which 3 cover an area of 15,333.81 ha located within an existing Mining Concession (X1 Deposit, Serrinhas and Guarantã Ridge Targets), another 6 properties totaling 47,172.65 ha are under an Exploration Permit. The Property totals 62,506.46 hectares in the Alta Floresta Gold Province.

The status of Aura's Exploration Permits ("Autorizações de Pesquisa"), Application Mining Concession ("Requerimento de Concessão de Lavra"), and Mining Concessions ("Portarias de Lavra") as of June 6, 2022, are summarized in Table 4-1 and shown in Figure 4-2.

Table 4-1 - List of Mineral Rights Under Control of Aura.

ANM ID	Area (Ha)	Status	Comments	Expiration Date
866428/2002	3,696.08	Mining Concession	Start of mining extension requested	Indeterminate
866072/2001	6,639.42	Mining Concession	Start of mining extension requested	Indeterminate
866324/1991	4,998.31	Mining Concession	Start of mining extension requested	Indeterminate
866261/2017	8,032.41	Exploration Permit	Renewal application on 04/04/2023	06/03/2023
866149/2019	5,308.21	Exploration Permit	Renewal application on 12/14/2023	02/12/2024
866864/2021	8,458.99	Exploration Permit	Renewal application on 09/16/2024	11/18/2024
867202/2021	6,296.92	Exploration Permit	Renewal application on 01/24/2025	03/25/2025
866863/2021	9,667.24	Exploration Permit	Renewal application on 09/16/2024	11/18/2024
866865/2021	9,408.88	Exploration Permit	Renewal application on 09/16/2024	11/18/2024
Total	**62,506.46**			

Note: ANM = Agência Nacional de Mineração de Brazil, the government agency who approve mining, concessions exploration permits, etc.
Note: Coordinates are South American Datum (1969), UTM Zone 21 South.

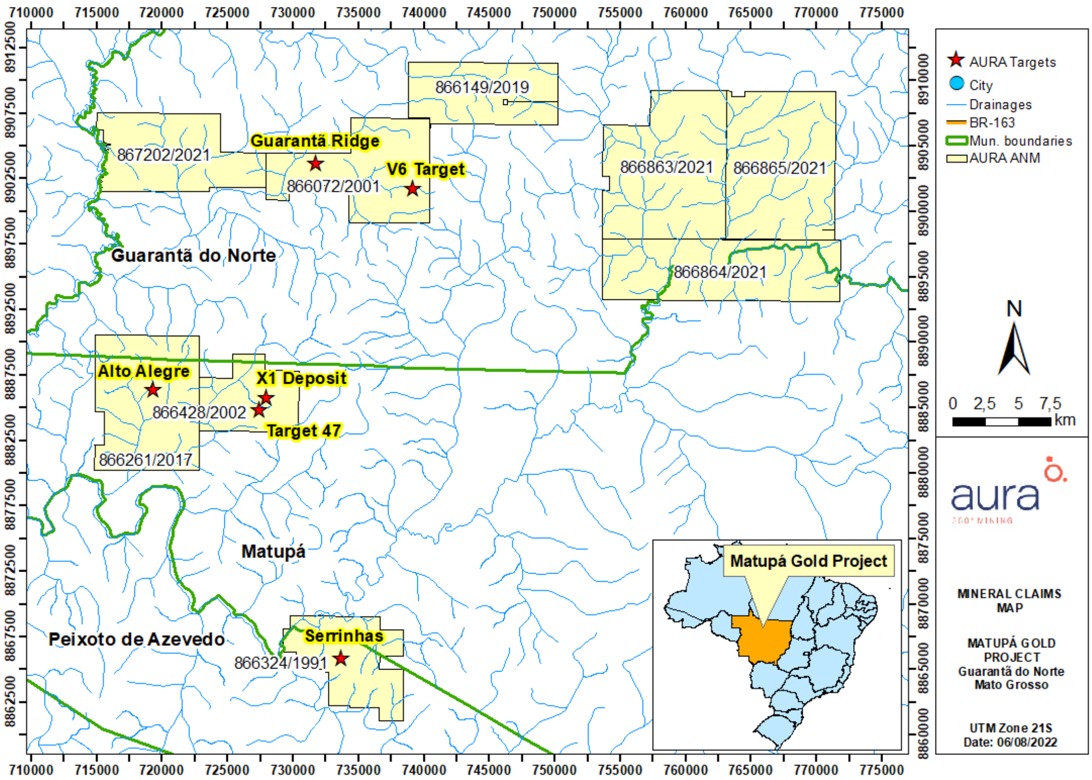


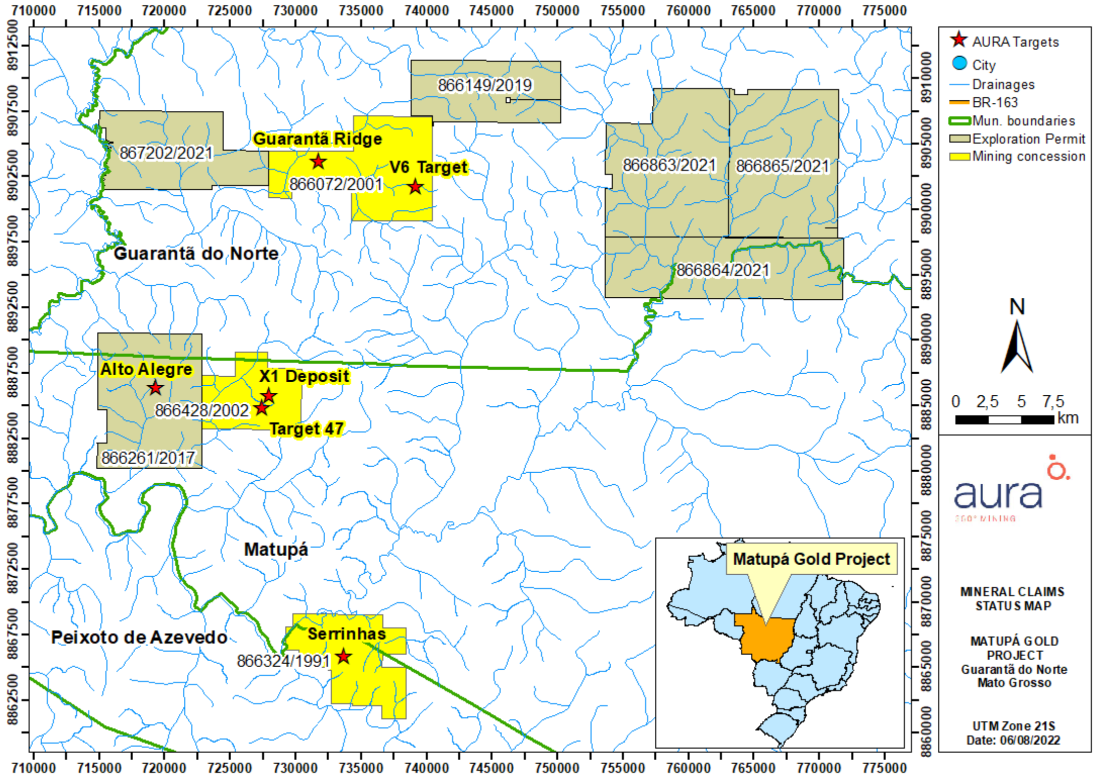
Figure 4-2 - Mineral Rights Map Showing Current Known Mineralization.

4.3 MINING CONCESSION AND APPLICATION OF MINING CONCESSION

Within the 1988 Brazilian Federal Constitution, mineral resources are defined as assets of the Federal Government. The legal right to mine is assigned to the mining company by the Federal Government of Brazil in the form of a Mining Decree in accordance with the Mining Code that was originally established under Decree Law No. 227, dated February 28, 1967. Under Brazilian law there is a separation of the surface rights from the mineral rights; therefore, a business entity may hold valid mining rights from the Federal Government but must still negotiate legal access with the surface rights holder.

The Mining Code, which has been amended several times since its approval on February 28th, 1967, addresses both issuance of prospecting permits as well as Mining Concession permits (Mining Decree), which are issued after the project proponent has demonstrated the technical and economic viability of the project. The Mining Decree, along with the appropriate environmental permitting, forms the basis of the right to mine a mineral deposit. The Mining Decree is granted for a specific area and for the exploitation of a specific mineral. The federal department responsible for issuing the mining rights is the National Mining Agency (Agência Nacional de Mineração, ANM).

Figure 4-3 is a map showing the status of the mineral licenses, current as of June 2022.



Note: Coordinates are South American Datum (1969), UTM Zone 21 South.

Figure 4-3 - Mineral Rights and License Status, November 11, 2021.

The Matupá Gold Project includes the properties covered by the Mining Concessions ANM number 866.428/2002 – X1 Deposit, the property under ANM number 866.324/1991 – Serrinhas Target, and the property ANM number 866.072/2001 – Guarantã Ridge Target, shown in yellow on Figure 4-3.

An application for an extension to the start of mining activities was made for the X1 Deposit Mining Concession on June 13, 2013. An easement area claim occurrences was started on August 19, 2021, which is necessary for the environmental licensing process.

The EIA/RIMA protocol at SEMA-MT (Environmental Regulatory Agency of Mato Grosso State) occurred on November 30th, 2021 and a public hearing for presenting the RIMA (Environmental Impact Report) to the local community was realized on May 10th, 2022. Now it is estimated that the Preliminary License (LP) will be issued and endorsed by the CONSEMA (Environment Counsel of Mato Grosso State) around the third quarter (Q3) of 2022. The Installation License will be required between the third and fourth quarters of 2022 and its issuance is scheduled for the end of the first quarter of 2023. The operating license will be required 4 months prior to completion of construction.

Throughout this process, additional licenses will be required, such as the Water Use Grant, Effluents Release Grant, Vegetation Clearing Permit, and others that may be required to complete the licensing process.

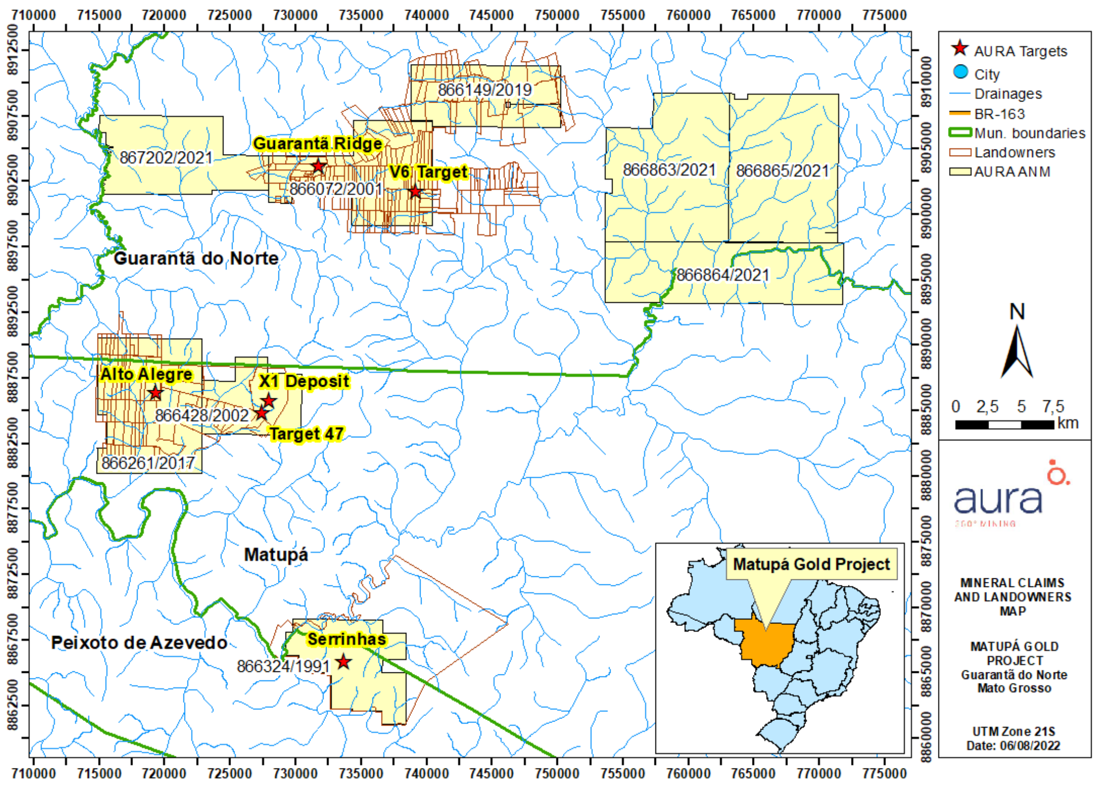
4.4 SURFACE RIGHTS AND ACCESS TO LAND

4.4.1 FOREIGN OWNERSHIP OF RURAL LANDS

The Mato Grosso State is known to be the greatest grain producer in Brazil, including soy, corn, and cotton. In the Matupá Gold Project region there is significant soybean and corn farming. In the areas of X1 Deposit and Guarantã Ridge Target there is only minor cattle ranching, at Serrinhas there is a combined activity of cattle ranching and farming of soy and corn.

Prior to initiating negotiations for the land acquisition, Aura developed procedures to assess the value of land surface rights based on the Brazilian Mining Code and the Brazilian Technical Norms for acquisition and compensation of public assets. These norms are in line with the International Finance Corporation's Performance Standard 5 (IFC PS5) on Land Acquisition and Involuntary Resettlement. The Project's land area and costs has been optimized by reducing the amount of land to be negotiated.

Rio Novo maintained good relationships with the local community and developed access agreements with the landowners to carry out exploration activities. The project is currently on a maintenance basis and exploration activities are being conducted to address further mineralization potential for the properties, especially at Serrinhas. Figure 4-4 shows the Project landowners control and the target locations inside the properties.



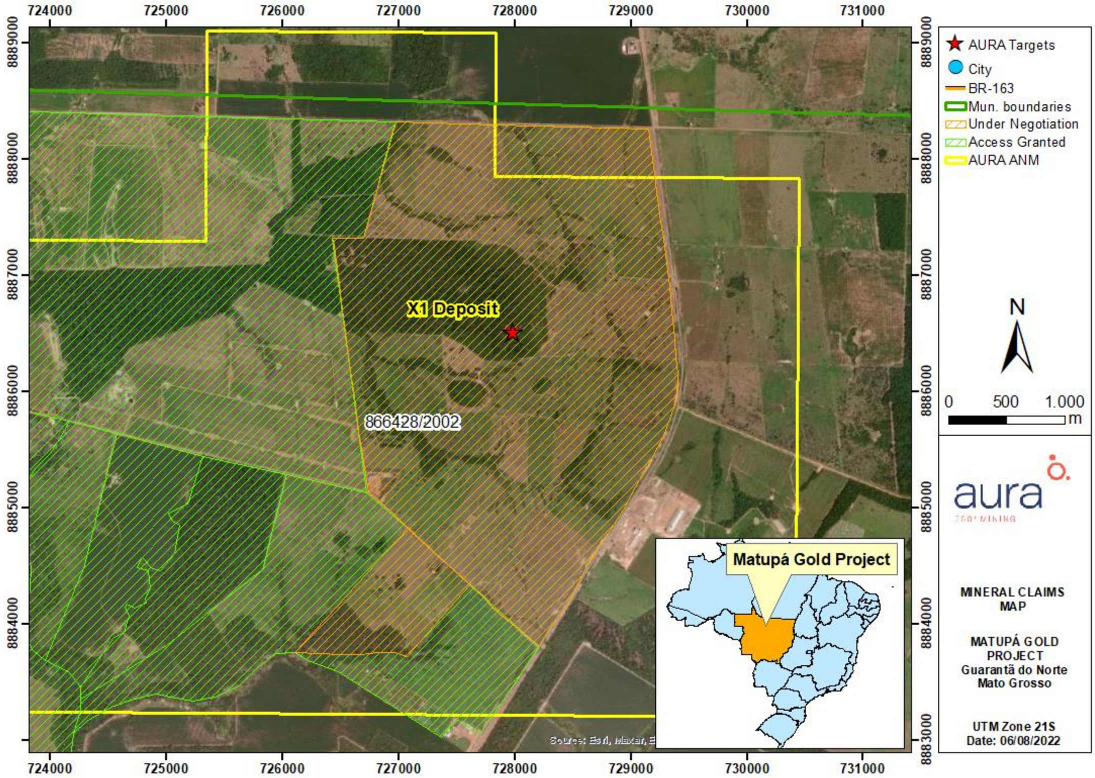
Note: Coordinates are South American Datum (1969), UTM Zone 21 South.

Figure 4-4 - Matupá Gold Project Landowners Control Map.

4.4.2 X1 DEPOSIT AREA

Aura controls 100% ownership of the mineral rights and private landowners own the surface rights. Brazilian law assures the use of surface rights to mining companies who can demonstrate that they have economic deposits in the sub-surface. Regarding the X1 Deposit, Aura will have to pay the Brazilian Government (ANM) a tax on Financial Compensation for Mineral Exploration (CFEM) of gold of 1.5% on net sales. Additional royalties of 1.95% on net sales will have to be paid for landowner and private agreements.

The status of Aura's surface and mineral rights in the vicinity of the X1 Deposit, as of June 2022, is shown in Figure 4-5.



Note: Coordinates are South American Datum (1969), UTM Zone 21 South.

Figure 4-5 - Mineral and Surface Status.

4.5 ROYALTIES AND EXPLOITATION TAXES

The ANM imposes a 1.5% royalty on any proposed gold production, which is referred to as the Financial Compensation for the Exploitation of Mineral Resources (CFEM). This royalty is divided between the municipality, the state and the Federal government with the municipality receiving the majority. Out of the CFEM amount collected, 65% is earmarked for the municipalities where the production takes place, 23% goes to the States or the Federal District, and 12% goes to ANM. ANM, in turn, must allocate 2% to environmental protection, through IBAMA (Instituto Brasileiro do Meio Ambiente e dos Recursos Naturais Renováveis), the Federal Ministry of Environment's enforcement agency.

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

5.1 ACCESSIBILITY

The Project surrounds the township of Guarantã do Norte and Matupá and all target areas are readily accessible by a network of secondary roads off the Cuiabá – Santarém highway (BR 163). This highway is entirely paved from Cuiabá to Guarantã do Norte, approximately 730 km, and is in good condition considering the heavy traffic due to grain production within the State.

The nearby cities around the X1 Deposit, Guarantã do Norte (population of 36,000) and Matupá (population of 17,000), are cities with good infrastructure, service companies, commerce, and a few industries.

The X1 Deposit is marked by a prominent hill located in Matupá approximately about 11 km north of its urban area and approximately 11 km south of Guarantã do Norte, both along Highway BR 163.

The Guarantã Ridge Target lies 4 km north from the urban area of Guarantã do Norte and is accessed and cut by Highway BR 163.

The Serrinhas Target is located around 10 km southeast from the urban area of Matupá and is accessed by Highway MT-322.

Regular commercial flights are available from Cuiabá (capital of the State) to the nearby cities of Sinop (population of 149,000), located 240 km south of Guarantã do Norte, and Alta Floresta (population of 80,000) located 130 km west. Matupá and Guarantã do Norte have airports for charter flights.

5.2 CLIMATE

The territorial extension of the State of Mato Grosso, located in the central portion of South America between latitudes 8º to 19º S and 51º to 62º W, has the Continental Climate characteristics of the intertropical latitudes of South America. One of the main climatological properties of this region is that it is in a transition area between the Continental Tropical Climates, covered with Cerrado, and the Continental Equatorial Climates covered with Amazon Forest.

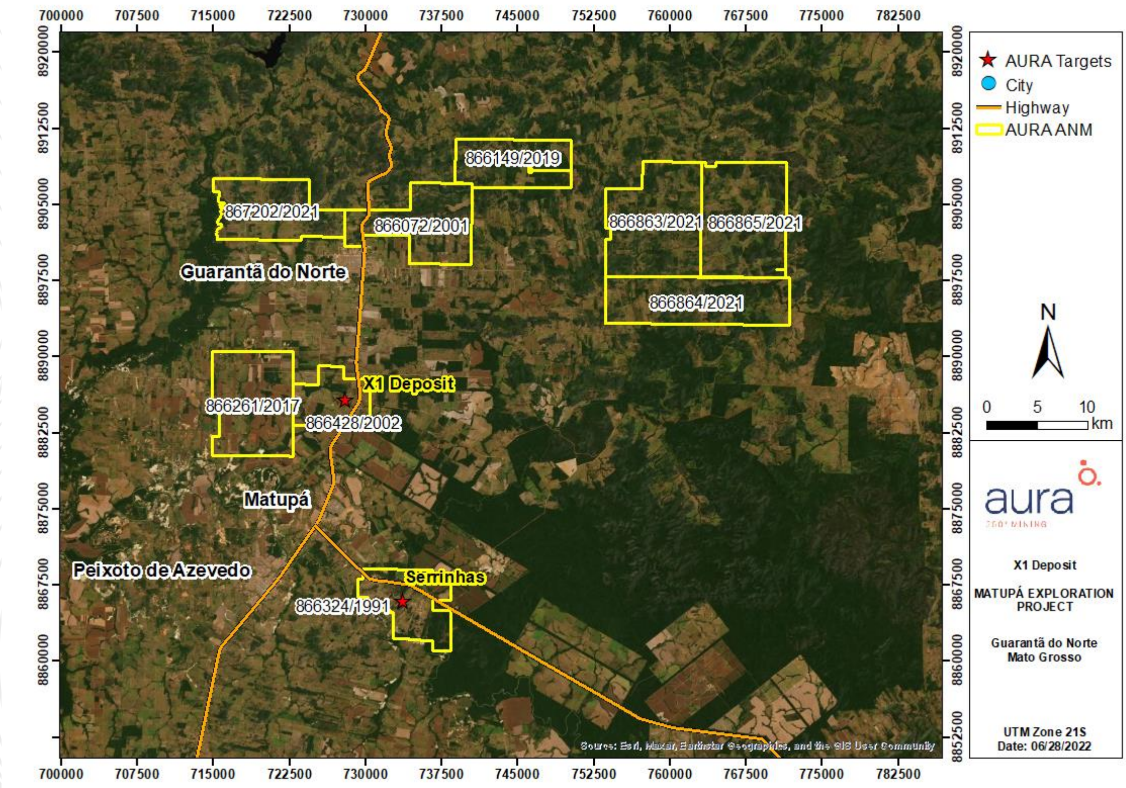
Likewise, the continental location, distant between 1,400 to 2,000 km from the Atlantic Ocean, gives it seasonal weather patterns with a wet season from November to April and a dry season from May to September. The large latitudinal extension (8º to 9º S) alters this seasonal distribution, causing the rainy season in the extreme south to generally start 1 to 2 months in advance, September-October, while in the extreme north there is a delay, starting in November or December. On the other hand, the beginning of the dry season is similarly anticipated in the south, starting in March to April, while in the extreme north, the Amazon summer only starts in May to June.

The variation of the average annual minimum temperature in the State of Mato Grosso ranges from 16 to 22°C. The extreme northwest of the State of Mato Grosso is the region that presents the lowest degree of nocturnal cooling, with the average annual minimum temperature being between 21 and 22°C. There are low areas, altitudes between 100 to 200 m, where low cloud ceilings are concentrated almost all year round. The high humidity and cloudiness should reduce nighttime cooling.

5.3 PHYSIOGRAPHY

According to the Municipality of Matupá (2022), the local climate is equatorial, hot, and humid with an average annual temperature of 24°C, maximum of 40°C and minimum of 4°C. Annual rainfall is 2,500 mm, with a rainy season from January to March and 3 months of dry seasons from June to August. The Project property lies in the Amazon Basin in an area of flat to subdued topography with small hills related to granitic intrusions. To the north of the city of Guarantã do Norte the topography becomes more rugged with relief in the range of 350 m.

Most of the land is used for soybean and corn farming and cultivation for livestock. There are only very small areas of native vegetation preserved on top of some hills and along drainages. This can be seen on the satellite image in Figure 5-1 of Guarantã do Norte showing the locations of the X1 Deposit, and Guarantã Ridge and Serrinhas Targets.



Note: Coordinates are South American Datum (1969), UTM Zone 21 South.

Figure 5-1 - Satellite Imagery of Guarantã do Norte Showing X1 Deposit and Serrinhas and Guarantã Ridge Targets.

The original vegetation forms two layers: an upper layer, with trees more than 20 m high and a lower one, with low-to medium-size plants not exceeding 10 m in height. Guarantã do Norte lies at an elevation of 345 m above sea level.

Two main rivers draining the area are Rio Braço Norte and Rio Braço Sul which, along with several springs, provide plenty of water for the agriculture industry. Tributaries of Rio Braço Norte are used for hydroelectric power generation to the north of the city.

5.4 LOCAL RESOURCES AND INFRASTRUCTURE

The main economic activity in the Project region is soybean and corn farming and livestock activity. Forestry and dairy agriculture have decreased overtime during the last 10 years. This is one of the fastest growing economic regions in the country mainly because of the export-oriented agribusiness and an aggressive occupation and use of recently deforested areas.

According to the Mato Grosso Institute of Agricultural Economics (IMEA) the state's 2021/2022 soybean crop is expected to reach 38.14 million tonnes – an increase of 5.5% compared to the previous cycle. Regarding corn production the estimate for the cereal is to reach 39.65 million tonnes, 17.9% more than in the previous harvest.

Mato Grosso State also leads in the Brazil's economic growth projection for 2021, with an increase of 4.97%. According to announcement of MB Associados, a macroeconomic analysis consultancy, agribusiness is the main driver of the state's GDP growth. The state is the main grain producer in the country and should be responsible for more than 30% of the national harvest this year.

Universities and technical centers involved in teaching the agribusiness are well established and expanding in Sinop, located 230 km south from Guarantã do Norte.

Guarantã do Norte has private, public primary and high schools to serve the almost 5,000 students of the municipality (urban and rural areas). The city also has two hospitals, communication systems, banks, police and local justice offices. Guarantã do Norte is well situated to support many of the necessities of the Project (personnel, accommodation, basic services, transport, health, and others).

A power supply is available at the Project site. A power line of 138/34.5 kV owned by the regional electricity company ENERGISA (Energisa Mato Grosso Distribuidora de Energia) crosses the property area.

There is an ongoing Federal plan to build a rail service in the region. The project is called Ferrogrão and is programmed to be operational by 2025. Ferrogrão, technically called EF-170, is the project for a longitudinal Brazilian railroad that will form Brazil's railroad corridor for export through the Amazon Basin, in the northern region of the country. The railway will have a length of 933 km, connecting the grain-producing region of the Midwest to the Port of Miritituba on the right bank of the Tapajós River, in Itaituba, in the state of Pará. This railroad will follow the BR-163 highway.

6. HISTORY

The region of Guarantã do Norte, Matupá and Peixoto de Azevedo has a recorded history of gold production that started in 1978 with the arrival of garimpeiros (artisanal miners) during the opening of the Cuiabá-Santarém highway (BR-163).

In the 1980s, the increase in the international price of gold, combined with the deterioration of economic and social conditions in Brazil, caused a real rush to the region in search of gold. At the end of that decade, the northern region of Mato Grosso became a region with intense mining activity. Initially, only the alluvium material of the rivers was explored, and over time, gold was discovered in primary rocks associated with granitic rocks and also in lode quartz.

Gold production reached its peak in 1989 and started to decline in 1999. From 1980 to 1999 the production of gold within the limits of the gold province of Alta Floresta was estimated to have been on the order of 160 tonnes. Despite the relative exhaustion of the alluvial and the supergene enrichment deposits, this region still holds significant exploration potential for primary gold deposits.

With the depletion of easy reserves of alluvium and residual soils, the fall in the price of gold and the increase in operating costs from 1999 onwards, garimpeiro production practically ceased, opening up the possibility of evaluating several areas with occurrences of primary gold in the region through surveys by mining companies.

6.1 PROJECT OWNERSHIP

Recent project ownership commenced in 1996 with a joint venture between Mineração Bom Futuro Ltda. (first owner of Serrinhas Target) with the company WMC Mineração Ltda (Brazilian subsidiary of former WMC Resources Limited). This episode established the beginning of mining companies carrying out exploration programs in the region. This work was followed by reconnaissance mapping and prospecting by Companhia Vale do Rio Doce (Vale formerly CVRD) in 1999 which resulted in the discovery of a gold occurrence in a road cut inside East-West ridge, 3 km north of Guarantã do Norte, now known as Guarantã Ridge Target.

Three years after discovering X1, Vale sold the mining rights to Mineração Santa Elina (MSE), who continued the exploration work until 2010, when the Project was sold to Rio Novo Mineração (RNM).

On December 2017, Aura Minerals and Rio Novo announced a merger transaction under the BVI Business Companies Act of 2004. The merger was completed in March 2018, under which Aura was merged with Rio Novo and the separate corporate existence of Rio Novo ceased. Aura continued as the surviving company in the merger combining a strong portfolio of mining properties. Table 6-1 summarizes the chronology of ownership.

Table 6-1 - Summary of Ownership of Matupá Gold Project.

Ownership	Properties	Period
WMC	Serrinhas	1996 - 2000
RTZ	Serrinhas	2000 - 2002
CRESCENT	Serrinhas	2002 - 2006
Vale	X1 + Guarantã Ridge	1999 - 2006
MSE	X1 + Guarantã Ridge	2006 - 2010
RNM	X1 + Guarantã Ridge + Serrinhas	2010 - 2018
AURA	X1 + Guarantã Ridge + Serrinhas	2018 - Present

6.2 EXPLORATION HISTORY

Gold has been the primary focus for exploration in the district, however, pollymetalic deposit discoveries have been made on the Matupá Property from prior exploration. Discoveries thus far, have been made by a combination of geophysical surveys, mapping, soil and rock sampling, followed by drilling. To date, exploration has primarily targeted near-surface gold anomalies and is therefore still in the early stages of exploration. The major exploration milestones are highlighted below:

- 1996: joint venture between Mineração Bom Futuro Ltda. (first owner of Serrinhas Target) with the company WMC Mineração Ltda (Brazilian subsidiary of former WMC Resources Limited);
- 1996 to 2000: Primary mineralization was confirmed in known area of intense garimpo activity called Serrinhas from Matupá, Serrinhas Target;
- 1999: Vale (formerly CVRD) starts regional gold project with reconnaissance program including geological mapping, soil and rock sampling. Discovery of Guarantã Ridge polymetallic target after first positive results for Gold and Silver;
- 2000 – 2002: Mineração Bom Futuro Ltda. created a joint venture, the work focused mainly on a greater understanding of the primary mineralization associated with hydrothermalized granites;
- 2002 – 2006: Exploration work in Serrinhas continued under Crescent Resources (CRESCENT), delineating first mineral resources (non-compliant);
- 2003: Continuing exploration activities Vale discovers X1 anomalous area and confirmed with soil and rock sampling. Initial diamond core drilling confirming mineralization;
- 2006: Vale sold the mineral rights to Mineração Santa Elina (MSE);
- 2006 – 2010: MSE begins regional exploration activities, delineating nearby surface gold occurrences and first Mineral Resource delineations for the X1 Deposit;
- 2010: MSE sold the mineral rights to Rio Novo Mineração (RNM);
- 2010: February 2010, RNM issues a Technical Report and Audit of The Preliminary Resource Estimate on The Guarantã Gold Project (former name of Matupá Gold Project);
- 2010 – 2012: RNM conducts additional exploration works on Guarantã Ridge and infill drilling at the X1 Deposit;
- 2013: RNM adds claims for X1 and Guarantã Ridge Mining Concessions after positive reports to Agência Nacional de Mineração de Brazil (ANM) (see Section 4.3);
- 2017: December 2017 Aura Minerals and Rio Novo announced a merger transaction under the BVI Business Companies Act of 2004. The merger was completed in March 2018, under which Aura was merged with Rio Novo and the separate corporate existence of Rio Novo ceased. Aura continued as the surviving company in the merger combining a strong portfolio of mining properties;
- 2018: Aura reviews the Project historical exploration data and identifies several drill hole intersections, and chip and soil samples with gold, copper and zinc values that merit further analysis within the Project area;
- 2019: Aura restarts the exploration activities for the Matupá Gold Project.

6.3 HISTORICAL MINERAL RESOURCE ESTIMATES

The first mineral resource, an historical mineral resource, for the X1 Deposit was estimated by MSE in 2008. Rio Novo also did some internal mineral resource estimates for the Serrinhas and Guaranta Ridge targets in 2010. Only Rio Novo performed a Mineral Resource Estimate in 2010 for the X1 Deposit, which was audited by GeoSim Services from Canada, and is considered to be NI 43-101 compliant. This X1 Mineral Resource Estimate is historical in nature and is superseded by the Mineral Resource Estimate in this technical report. Table 6-2 outlines these historic mineral resource estimates.

Table 6-2 - Summary of Historic Mineral Resource Estimates.

Deposit or Area	Classification	Tonnes (kt)	Grade (g/t Au)	Grade (g/t Ag)	Contained (koz Au)	Contained (koz Ag)	Note
X1	M+I	9,100	1.65	4.14	478	1,300	MSE 2008
X1	M+I	8,000	1.35	4.6	347.4	1,180	RNM 2010
Guarantã Ridge	N/A	3,000 to 4,000	1.6 to 2.2	57 to 77			RNM 2010
Serrinhas	N/A	500 to 650	2.2 to 3.0	10 to 14			RNM 2010

Marston & Marston, Inc. (Marston) and Geosim Services Inc. (Geosim) were retained by Rio Novo Gold Inc. (Rio Novo) to prepare a Canadian National Instrument 43-101 (NI 43-101) Technical Report for the formerly Guarantã Gold Project (the current Matupá Gold Project) located in the northern part of the State of Mato Grosso, Brazil. Marston's and Geosim's Scope of work included a site visit to the property, a data review and an audit of the geological model and Mineral Resource Estimate. The Marston and Geosim technical report was intended to report on the status of the Project in 2010 including the exploration to date and a Mineral Resource Estimate for the X1 Deposit. A compliant Mineral Resource model for the X1 Deposit was estimated based on mineralized domains and grade information up to date as of 2010. Grade was interpolated into blocks using the Ordinary Kriging ("OK") method. Rio Novo reported 8.0 Mt averaging 1.35 g/t Au and 4.6 g/t Ag (364,000 oz Au equivalent). An additional 471,100 tonnes averaging 1.43 g/t Au and 4.5 g/t Ag are classified as Inferred Mineral Resources. This historical Mineral Resource Estimate was determined for the X1 Deposit only.

After the Marston and Geosim technical report, Rio Novo performed an infill and exploration drilling campaign on the X1 Deposit throughout 2011. The results of this drilling have not been incorporated into any Mineral Resource Estimate until the effective date of this report.

7. GEOLOGICAL SETTING AND MINERALIZATION

7.1 TECTONIC SETTING

The Alta Floresta Gold Province (AFGP) is located in the south-central portion of the Amazon Craton (Almeida, 1978; Almeida et al., 1981), a crustal segment north of South America that would have stabilized 1.0 Ga, and which is surrounded by the Neoproterozoic mobile belts of Tucavaca (in Bolivia), Araguaia-Cuiabá (Central Brazil) and Tocantins (northern Brazil) (Almeida et al., 1976; Cordani et al., 1988; Tassinari & Macambira, 1999). As the AFGP covers an area of approximately 430,000 km2, it represents one of the largest cratonic regions on the planet, being individualized into two main Precambrian shields: the Central Brazil (or Guaporé) and Guiana shields, separated by the Paleozoic Solimões-Amazonas (Tassinari) basin (Macambira, 1999; Dardene & Schobbenhaus, 2000; Tassinari et al., 2000) (Figure 7-1).

Two main concepts regarding the Amazonian Craton's geodynamic evolution have been postulated. The first proposes that the Amazonian Craton's evolution has been characterized by reactivation of the platform and formation of continental blocks (or paleoplates) through continental crust reworking during the Archean and Paleoproterozoic, with subsequent reactivation and/or reworking during the Mesoproterozoic (Amaral, 1974; Almeida, 1978; Issler, 1977; Hasui et al., 1984).

The second concept, currently more accepted and proven, is based on the mobile hypothesis of modern orogenies. During the Archean, Paleo- and Mesoproterozoic there would have been successive continental magmatic arcs, with the formation of juvenile material derived from the upper mantle, resulting in crustal reworking (Cordani et al., 1979; Tassinari, 1981; Teixeira et al., 1989; Tassinari et al., 1996; Tassinari & Macambira, 1999; Santos et al., 2000; Santos et al., 2006; Cordani & Teixeira, 2007).

In this context, based on isotopic determinations, proportions between lithotypes, structural and geochemical data, together with geophysical evidence, the Amazon Craton is subdivided, depending on the model adopted, into different geochronological or tectonic provinces with a preferential northeast-southwest orientation (Tassinari & Macambira, 1999; Santos et al., 2000; Cordani & Teixeira, 2007).

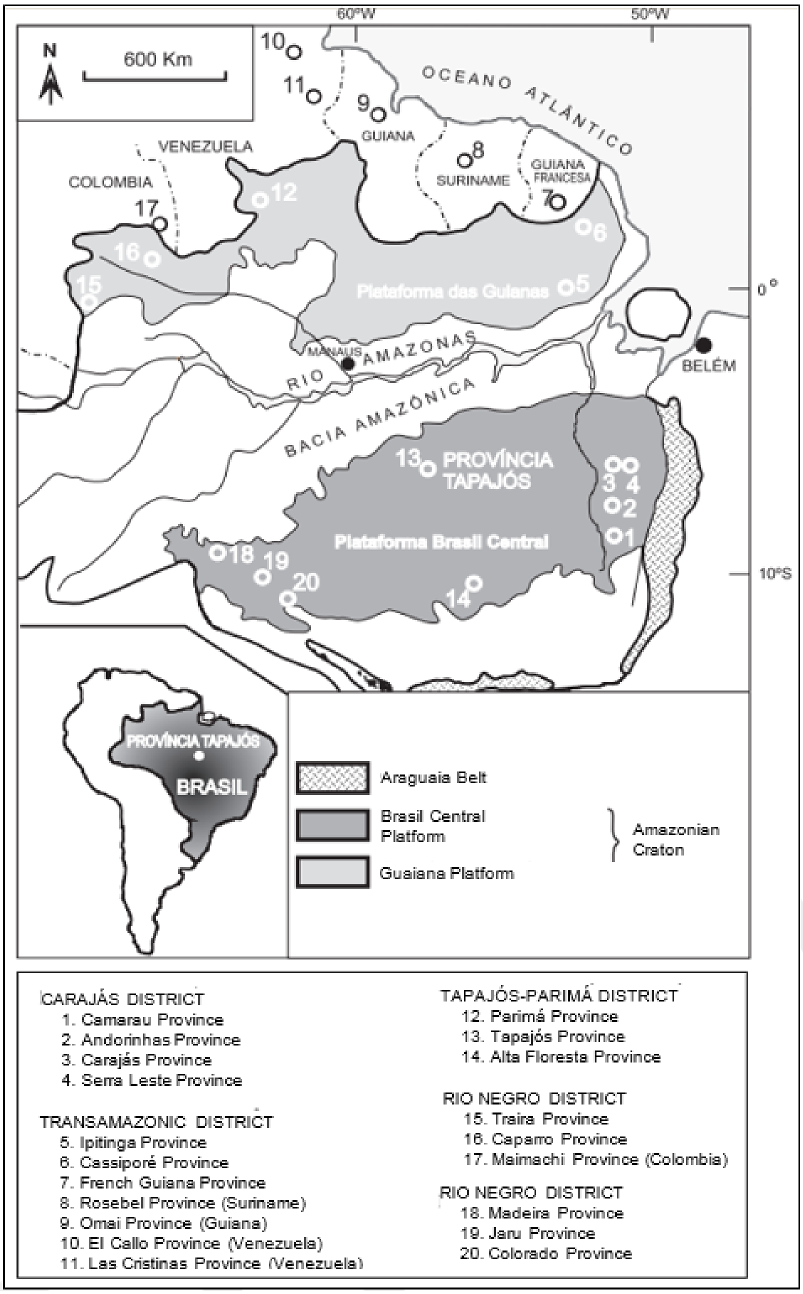


Figure 7-1 - Distribution Map of the Main Gold Districts and Provinces in the Amazon Craton (Modified from Coutinho, 2008).

Depending on the geotectonic model adopted, the AFGP can be considered as part of different geochronological provinces, whose temporal and spatial limits differ in the geotectonic compartmentation of the Amazon Craton. In this sense, the AFGP falls between the geochronological provinces Ventuari – Tapajós (1.95-1.8 Ga) and Rio Negro – Juruena (1.8-1.55 Ga) in the conception of Tassinari and Macambira (1999), or between the tectonic provinces Tapajós – Parima (2.1-1.87 Ga) and Rondônia – Juruena (1.82-1.54 Ga), in the model of Santos (2003) and Santos et al. (2000; 2006) as presented in Figure 7-2.

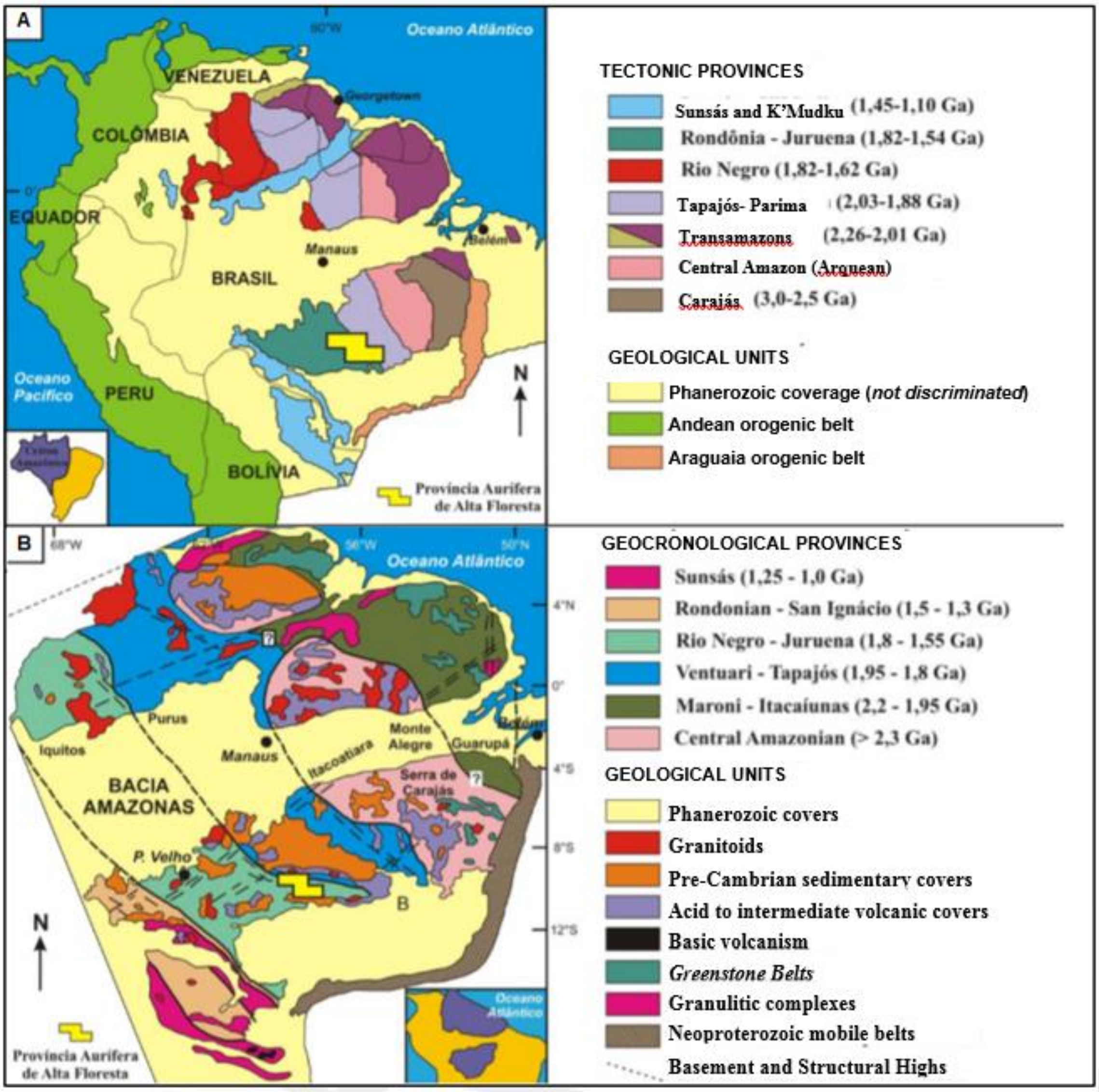


Figure 7-2 - Location of the Alta Floresta Gold Province According to the Models by (A) Santos et al. (2006); and (B) Tassinari & Macambira (1999). The PAAF grid corresponds to the regional mapping area of the project "Geology and Mineral Resources of the Mineral Province of Alta Floresta", carried out by CPRM (Geological Service of Brazil) and integrated by Souza et al. (2005).

7.2 REGIONAL GEOLOGY

The Alta Floresta Gold Province (AFGP), inserted in the south-central portion of the Amazonian Craton, mostly constituted by plutono-volcanic sequences generated in Paleo- and Mesoproterozoic continental arcs, in addition to deformed and metamorphic units in restricted greenschist facies located in the central and northwestern portions. The units that compose the province, especially its eastern segment, are essentially represented by oxidized calcium-alkaline plutonic and volcanic rocks, of medium to high potassium (K), meta- to peraluminous, belonging to the magnetite series (type I granites). Of volcanic, sub-volcanic and alkaline granitoids (type A granites). (Assis, 2015).

As a whole, these units are arranged on a west-northwest steering belt, and exhibit ages ranging from 1.75 to 2.03 Ga, with model ages (TDM) between 2.76 and 2.15 Ga, and εNd(t) values between -7.62 and 3.09 (Santos et al., 1997; Moura, 1998; JICA/MMAJ, 2000; Santos et al., 2000; Pimentel, 2001; Pinho et al., 2003; Souza et al., 2005; Paes de Barros, 2007; Silva & Abram , 2008; Miguel-Jr, 2011; Assis et al., 2014). This is suggestive of magmatism with an Archean to predominantly Paleoproterozoic source, in a juvenile arch environment, but with a small continental contribution.

According to Assis (2015), in general, these units can be grouped into four main sets:

i. Deformed and metamorphized granitic basement (2.81 to 1.99 Ga),

ii. Felsic plutono-volcanic and volcano-sedimentary sequences belonging to the magnetite series (type I granites; 1.97 to 1.78 Ga);

iii. Post-collisional and anorogenic plutono-vucanic units (1.78-1.77 Ga); and

iv. Clastic sedimentary sequences (~1.37 Ga).

The basement portion of the province corresponds to heavily razed areas and lack of outcrops. It is currently divided into two main complexes: (i) Bacueri-Mogno (2.24 Ga; Pimentel, 2001), not exposed in the eastern segment of the AFGP; and (ii) Cuiú-Cuiú (1992 ±7 Ma; Souza et al., 2005). The first comprises pyroxene-rich orthoamphibolites, orthogneisses, paragneisses (garnet-silimanite-cordierite-biotite gneiss, silimanite -biotite gneiss and silimanite gneiss), enderbitic plutonics, banded iron formations, calciosilicate-quartzite-granite, quartzite granitic rocks metagabbro-norite and metapyroxene that exhibit mylonitic foliation and/or medium-to-high dip, gneiss banding and oriented according to eastwest and east-southeast–west-northwest (Souza et al., 2005; Silva & Abram, 2008). Pimentel (2011) obtained isochronic Sm-Nd ages of 2.25 Ga and εNd(t) of 2.4 for amphibolite of this complex, corresponding, therefore, to the oldest age in the region. The Cuiú-Cuiú Complex, however, outcrops near the cities of Peixoto de Azevedo and Novo Mundo, and consists essentially of granitic to tonalitic gneisses, migmatites intruded by calcium-alkaline foliated granitoids of tonalitic to monzogranitic composition (Paes de Barros, 2007), in addition to shales, mafic and ultramafic rocks, and banded iron formations (Dardenne & Schobbenhaus, 2001).

The Cuiú-Cuiú Complex is intruded by a series of oxidized calcium-alkaline plutons (magnetite series; type I granites), exemplified by the Pé Quente Suite (Assis, 2011), Novo Mundo granites (Paes de Barros, 2007), Aragão (Vitório, 2010) and Flor da Mata Granite (Ramos, 2011). These units are not deformed and/or metamorphized, and because they exhibit crystallization ages in the range of 1.97 to 1.93 Ga, they represent the oldest granitic systems in this segment of the province. With the exception of Flor da Mata, all the other plutons host lode quartz or disseminated gold mineralization. These suites are still truncated by Nhandu granite (Souza et al., 1979; Paes de Barros, 2007), Matupá Intrusive Suite (Moura, 1998; Souza et al., 2005), and Peixoto granite (Paes de Barros, 2007), aged between 1.88 and 1.79 Ga.

The Pé Quente Intrusive Suite (Assis, 2011) corresponds to an expanded magmatic series poor in quartz and composed of two cogenetic suites: (1) Suite Pé Quente, which hosts the homonym deposit and includes medium leukomonzonite, medium quartz monzodiorite, monzodiorite, fine albitite, aplitic granodiorite dykes and medium tonalite biotite (Assis, 2011; Assis et al., 2014); and (2) Monzonite Suite, represented by coarse monzonite in addition to subordinate monzonite quartz and monzodiorite quartz. Basalt and diabase dykes, in addition to several tonalitic to calcium-alkaline and oxidized monzogranitic plutons, tentatively correlated to the Matupá Suite (1,872 ±12 Ma), still occur intrusively in these suites (Assis, 2011). U-Pb dating by LA-ICP-MS on zircon from monzonite of the Pé Quente Suite indicates an age of crystallization in 1979 ±31 Ma (Miguel-Jr, 2011). The age of the Monzonitic Suite, however, remains unknown.

The Novo Mundo Granite (Abreu, 2004; Paes de Barros, 2007), in turn, corresponds to an elongated pluton in the northwest-southeast regional direction, in which recrystallization and stretching of the quartz crystals is observed (Lx=N15W/10º), possibly due to housing under structural control late in the development of the shear zones that delimit its northeastern and southwestern edges. This deformation is characterized by ribbon-quartz, quartz recrystallized and polygonized in sub-grains, plagioclase with faulted, arched and/or kink-banded twinning lamellae, green chlorite with arcuate cleavage, plagioclase relicts that resemble porphyroclasts, in addition to fractured plagioclase and potassic feldspar with fragmented edges (Paes de Barros, 2007). This granite exhibits faciological variation from syenogranite to monzogranite, in addition to subordinate granodiorite, quartz monzonite and monzonite, later cut by multiple dykes of gabbro and diorite. Geochronological studies indicate that host monzogranite and syenogranite have Pb-Pb (zircon evaporation) ages of 1,970 ±3 Ma and 1,964 ±1 Ma, respectively (Paes de

Barros, 2007). The syenogranite exhibits age TDM = 2.76 Ga and εNd(1,964) = -7.62; while the monzogranite has an age TDM = 2.55 Ga and εNd(1,964) = -4.48. According to Assis (2015), taken together, these data indicate both the participation of continental crust and the presence of an Archean source for the generation of this magma.

Still intrusive in the Cuiú-Cuiú basement, the Aragão Granite corresponds to an elongated pluton in the northeast-southwest direction, which outcrops southwest of the city of Novo Mundo and hosts dozens of lode quartz-gold deposits structurally controlled by transcurrent shear zones of sinister kinematics, partially exploited by mining activities (Miguel Jr., 2011; Assis et al., 2014). It exhibits a composition ranging from syenogranite to fine- to medium-grained isotropic monzogranite, equigranular textures with porphyritic, medium phaneritic and microgranular facies (Vitório, 2010). U-Pb geochronological data by LA-ICP-MS on zircon indicate that it would have crystallized at 1,931 ±12 Ma (Miguel Jr., 2011).

Of still unknown age, the Flor da Mata Granite corresponds to an isolated intrusive body northeast of the city of Novo Mundo, previously defined as belonging to the Teles Pires Intrusive Suite (TP1 by Paes de Barros, 2007). This intrusion consists essentially of slightly evolved alkali-feldspar granite to type I monzogranite, with light to strongly oriented quartz crystals (Ramos, 2011). The author also proposes that Granite Flor da Mata may be temporally equivalent to Granite Novo Mundo (1,970 ±3 Ma to 1,964 ±1 Ma), due to its petrographic and geochemical similarities.

Magnetite-biotite monzogranite and calcium-alkaline syenogranite, with enclaves of diorite to quartz monzodiorite, in addition to subordinate sub-volcanic granites, fine-grain quartz syenite and granophyres, composes the Nhandu granite (Souza et al., 1979; Moreton & Martins, 2005; Souza et al., 2005; Paes de Barros, 2007), which frequently hosts sulfide gold mineralization (e.g., Natal and Trairão deposits). The age of the Nhandu granite was established between 1,889 ±17 Ma to 1,879 ±5.5 Ma (U-Pb in zircon), with model ages between 2.14 and 2.17 Ga, and εNd(t) of -0.91 (Silva & Abram, 2008). In addition, JICA/MMAJ (2000) obtained crystallization ages of 1,848 ±17 Ma for this suite.

The Matupá Suite corresponds to one of the units with the greatest regional extension in the eastern sector of the AFGP. The Matupá Suite consists of four lithofacies that include biotite granodiorite and biotite monzogranite porphyritic (facies 1); hornblende monzogranite, biotite-hornblende monzonite and hornblende monzodiorite (facies 2); clinopyroxene-hornblende monzogranite and clinopyroxene-hornblende monzodiorite (facies 3); and granite, biotite granite and monzogranite with microgranite and subordinate granophores (facies 4) (Moura, 1998; Moreton & Martins, 2005). Facies 1 and 2 host several suffocated gold mineralizations, with the Serrinha Deposit being the one with the best geological investigation (Moura et al., 2006).

Pb-Pb geochronological data on zircon from facies 1 rocks indicate a crystallization age of 1,872 ±12 Ma, in addition to TDM model ages ranging from 2.34 to 2.47 Ga, with εNd(t) between -2.7 and -4.3 (Moura, 1998). Similar TDM model ages (2.15 – 2.34 Ga) were obtained by Souza et al. (2005), however, with slightly higher εNd(t) values, between -0.98 and +3.04, and is indicative of juvenile magmas from a Paleoproterozoic source and with a small contribution of crustal material. Additionally, several calcium-alkaline and oxidized plutons, of syenogranitic to tonalitic composition, intrusive in the Pé Quente Intrusive Suite are, although the lack of geochronological data to date, tentatively correlated to the Matupá Suite (Assis, 2011; Assis et al., 2012). In the vicinity of Agrovila de União do Norte (city of Peixoto de Azevedo), there is a granitic body of granodioritic to tonalitic composition, called União Granodioritic Suite, with U-Pb crystallization age in zircon by LAICP-MS of 1,853 ±23 Ma (Assis, 2011; Miguel-Jr, 2011) and again, possibly correlated with the Matupá Suite.

The Peixoto Granite (Paes de Barros, 2007), outcropping near the city of Peixoto de Azevedo, is intrusive in basement rocks and sometimes called Juruena Granite (Paes de Barros, 1994) or is considered to belong to the Matupá Intrusive Suite (Lacerda Filho et al., 2004). According to Paes de Barros (2007), this unit comprises biotite monzogranite, biotite granodiorite with hornblende and biotite tonalite, leucrocratic, isotropic, equigranular to porphyritic, with centimetric crystals of zoned plagioclase. Elongated diorite enclaves are common. Zircon crystals from biotite monzogranite provided Pb-Pb crystallization ages of 1,792±2 Ma (Paes de Barros, 2007).

Intrusive on the anterior domain, or else in the form of extensive felsic outflows, are the post-collisional to anorogenic plutono-volcanic units (1.78-1.77 Ga), represented by the Colíder and Teles Pires suites (Moreton & Martins, 2005; Souza et al., 2005; Silva & Abram, 2008).

The Colíder Suite is represented by a great diversity of sub-volcanic, volcanic, pyroclastic and epiclastic rocks with a dominantly intermediate composition, and to a lesser extent acidic (Souza et al., 2005; Silva & Abram, 2008). The Colíder Suite is a unit related to the Juruena Magmatic Arc, aged between 1.85 and 1.75 Ga (Souza et al., 2005), and includes volcanic rocks of a calcium-alkaline nature and evolution linked to the Paranaíta granitic suites, Juruena and Nhandu (Souza et al., 2005).

The sub-volcanic terms are represented by microgranite, micro-quartz monzonite, micromonzonite, micromonzogranite and granophyre, all associated with rhyolite spills, porphyritic dacites and locally microporphyritic andesites (Moreton & Martins, 2005). In general, they correspond to peraluminous to meta-aluminous high-potassium calcium-alkaline rocks, which exhibit geochemical affinity with syn- to post-collisional orogenic granitic series (Moreton & Martins, 2005; Souza et al., 2005). Volcaniclastics are represented by sandy-conglomeratic sediments, sometimes interspersed with conglomeratic lenses and sandy sediments (Souza et al., 2005). This unit exhibits tectonic contacts with the Matupá Intrusive Suite and the Nhandu Granite. Dating on porphyritic rhyolite by the U-Pb zircon method reveals crystallization ages of 1,786 ±17 Ma (JICA/MMAJ, 2000) and 1,781 ±8 Ma (Pimentel, 2001). Silva & Abram (2008) obtained a crystallization age of 1,785 ± 6.3 (LA-ICP-MS on zircon). Souza et al., (2005), in turn, obtained model ages (TDM) of 2.34 Ga, with εNd(t) of -3.75. The set of lithological, geochemical and geochronological data obtained for this unit indicates a calc-alkaline magmatism with crustal contamination and correlated to the Juruena Magmatic Arc (Pimentel, 2001; Souza et al., 2005; Silva & Abram, 2008).

Intrusive bodies in all of the previous units occur as batholiths and stocks from the Teles Pires Intrusive Suite, mostly constituted by biotite granites to alkali-feldspar granites, reddish colored, medium to coarse grained, locally with porphyritic, granophyric, rapakivi and anti-rapakivi textures (Souza et al., 1979; Silva et al., 1980; Souza et al., 2005). The geochemical data point to type A granites, of medium to high K calcium-alkaline nature, met aluminous to peraluminous, which correspond to post-collisional intrusions, with U-Pb ages in zircon from 1,757 ±16 Ma to 1,782 ±17 Ma , in addition to TDM ages ranging from 1.94 to 2.28 Ga and εNd(t) values between -3.4 and +3.0; indicative of magmas of mantle origin with strong involvement of crustal material (Santos, 2000; Pinho et al., 2003; Moreton & Martins, 2005; Silva & Abram, 2008). Alkaline and anorogenic porphyry quartz-feldspar lacolites, consisting of porphyritic alkali-feldspar granite to porphyritic monzogranite, designated as União do Norte Porphyry (Assis, 2011; Assis et al., 2012), occur near the Agrovila União do Norte and are petrographically and geochemically correlated to the Teles Pires Suite. U-Pb dating by LA-ICP-MS on zircon indicates that its crystallization would have occurred at 1,774 ±7.5 Ma (Miguel-Jr, 2011). In addition, this unit would be spatially and genetically related to the genesis of epithermal systems of intermediate sulfidation mineralized to Au + base metals, such as the Bigode (Assis, 2008) and Francisco (Assis, 2011) deposits.

Basic rocks, grouped under the name Basic Dykes (Moreton & Martins, 2005; Souza et al., 2005), comprise mainly diabase dykes and gabbro stocks that occur in the southwestern portion of the area, and thus complete the Paleoproterozoic stratigraphic picture, which culminated in the formation of an extensive magmatic arc called the Juruena Magmatic Arc. (Assis, 2015).

Finally, all these units are covered by sequences of sandstone and Arcosean sandstone, both of medium granulometry and with frequent conglomeratic layers, belonging to the Dardanelos Formation (Caiabis Group). These layers have planar parallel and channeled cross stratifications interpreted as a system of alluvial fans of intertwined rivers (Moreton and Martins, 2005). U-Pb ages in detrital zircon crystals range from 1,987 ±4 Ma to 1,377 ±13 Ma (Saes & Leite, 2003), suggesting the maximum age of 1.44 Ga as representative for the beginning of the sedimentation of the Dardanelos Formation (Souza et al., 2005). The Caiabis Group is still interpreted as a pull-apart basin, or possibly a strike-slip type where the main accidental occurrence zones, northwest-southeast, were responsible for its generation (Souza et al., 2005).

According to Souza et al. (2005), all these plutonic-volcanic units would have been generated in distinct orogenic cycles, temporally organized as follows: (1) Cuiú-Cuiú Magmatic Arc (2.1-1.95 Ga), represented by migmatites, gneisses and granites sin- to post orogenic; (2) Juruena Magmatic Arc (1.95-1.75 Ga), juxtaposed to the previous unit and composed of low-grade metamorphic calcium-alkaline plutonic-volcanic sequences and, to a lesser extent, ductile deformed medium- to high-grade metamorphic rocks, in addition to (3) metavolcanics-sedimentary ones originated in an extensional environment and intruded by deformed calcium-alkaline granitoids (1,743 ±4 Ma; Souza et al., 2005; Silva & Abram, 2008). Collectively, the plutonic-volcanic units, when arranged in tectonic ambience diagrams combined with available geochronological data, indicate a representative trend of the province's magmatic evolution as a function of its distinct orogenic cycles. This evolution would have started with the generation of the most primitive and oldest rocks in a volcanic arc environment (e.g., Suite Pé Quente), until the end of orogenetic events at a later stage, with the generation of more fractional and evolved rocks (e.g., Suite Teles Saucer), possibly due to the end of the last orogenetic cycle, on the continental platform. (Assis, 2015)

7.3 REGIONAL ALTERATION AND MINERAL DEPOSITS

The intimate spatial association of the hydrothermal and mineralized zones of the deposits inside the eastern part of the AFGP, with oxidized type I calcium-alkaline granitic plutons (magnetite series) suggests that the district gold deposits represent Paleoproterozoic-age magmatic-hydrothermal systems.

The processes of hydrothermal alteration in the eastern part of AFGP are understood as arising from a magmatic-hydrothermal system with a strong relationship to the regional shear zone in the northwest direction, with great extension, which are possibly correlated to the generation of important deposits in the region.

Disseminated gold mineralization are commonly associated with strong phyllic alteration envelopes (sericite+quartz+pyrite) spatially and genetically related to acid porphyritic dykes or related to potassic alteration represented by biotite enrichment or microcline and replacement of original K-feldspar, with or without silicification, and associated with disseminated sulfides, mostly pyrite.

Additionally, shear zones and high-grade lode quartz veins represents the most frequent mineralization in the district, commonly associated with potassic and/or chloritic alteration with silica enrichment and sulfides gangue such as pyrite, chalcopyrite and sphalerite as the most frequents.

7.4 DISTRICT GEOLOGY

The subordinate copper auriferous deposits of the eastern segment of the Alta Floresta Gold Province (AFGP) are eminently hosted in Paleoproterozoic granitic systems that vary in composition from tonalite-granodiorite to syeno-monzogranite, derived from oxidized magmas, type I, from calcium-alkaline to sub-alkaline, metaluminous to peraluminous, medium to high potassium and magnesian with a slightly ferrous affinity. (Assis et al., 2014).

However, the gold deposits associated with base metals (Zn + Pb ± Cu) are hosted by epiclastic, plutonic and Paleoproterozoic sub-volcanic sediments with a composition ranging from granite alkali-feldspar to granodioritic (Assis, 2011; Assis et al., 2014; Trevisan, 2015), which exhibit affinities with type A, meta- to peraluminous and ferrous alkaline magmas (Assis, 2011).

7.5 DISTRICT STRATIGRAPHY

The Matupá Gold Project area is part of the granitic bodies of the Matupá Intrusive Suite, which has an intrusive geological relationship to the gneiss basement of the Cuiú-Cuiú complex and to the Diorite/Gabbro bodies, the oldest regional event, are mostly identified from soil due to few exposures or by diamond drilling. The Matupá Intrusive Suite were intruded by quartz feldspar porphyries and late fine-grained mafic to intermediate dykes. These sequences are in contact with volcanic and pyroclastic rocks of the Colíder Group located north of Guarantã do Norte city.

7.5.1 Cuiú-cuiú Gneiss Complex

The lithology, part of the basement rocks in the Project area with ages ranging from 2.1-1.95 Ga (Souza et. al., 2005), includes biotite-tonalitic gneisses representative of the Cuiú-Cuiú complex, sometimes with migmatitic aspects, which typically occur in meter sized blocks and slabs in the northwest and southern regions of Alto Alegre Block (Figure 7-3), and has also identified in diamond drilling.

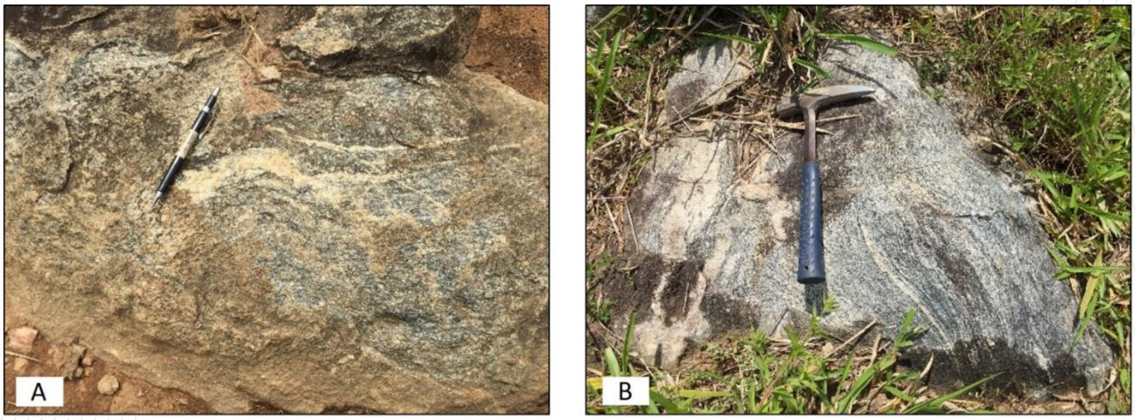


Figure 7-3 - A) Slab Outcrop of Orthogneiss of Cuiú-Cuiú Complex inside the Northwestern Part of Alto Alegre Block; B) In-situ blocks of Orthogneiss of Cuiú-Cuiú complex inside the southern part of Alto Alegre block.

7.5.2 BIOTITE PORPHYRITC GRANODIORITES

Among the most significant lithologies in the properties, specially surrounding the X1 Deposit and parts of the Alto Alegre block, are medium, inequigranular and porphyritic biotite-granodiorites (potassium feldspar porphyries up to 3 cm in size) of light gray color, essentially isotropic, and may locally present incipient to little penetrative foliation when close to zones of regional magnitude shear or smaller shear zones reflecting them. The porphyritic biotite granodiorite is composed of quartz, plagioclase, potassium feldspar phenocrysts (pink microcline), biotite and magnetite. According to Pineschi (2022), the age of this lithology is 1.873 +- 7.3 Ma. Figure 7-4 shows the primary granodiorite occurrence type, which when preserved, outcrops in meter sized blocks or semi-weathered slabs close to drainages, sometimes with characteristic spheroidal exfoliation.

Figure 7-4 - A) In-situ Blocks of Porphyritic Biotite Granodiorite of Matupá Suite inside the Northern Part of Alto Alegre Block; B) In-situ blocks of porphyritic biotite granodiorite of Matupá Suite inside X1 property.

The weathering profile of this lithology shows great development, reaching depths up to 60 m, and may present a large soil profile. which tend to be weakly magnetic, light brown to orange in color, sandy-clay texture, poorly selected with subangular quartz grains.

On the exploration historical data thin section report of this lithotype presents characteristics such as coarse granulation exhibiting hypidiomorphic granular texture, where the main minerals are quartz, plagioclase, biotite and muscovite, followed by alanite, apatite, zircon and rutile as traces and chlorite and epidote are altering minerals.

- Quartz is anhedric and interstitial, with a strong wave extinction (35%);
- Plagioclase forms medium to coarse-grained euhedral to sub-euhedral crystals exhibiting intense saussuritization and kaolinization (35%);
- The microcline forms medium to coarse-grained anhedral crystals, exhibiting pertitic growths and weak kaolinization (25%);
- Biotite forms medium-grained sub-euhedral crystals showing pleochroism in shades of yellow and light brown and is partially changed to chlorite (2%);
- Muscovite occurs as thin to medium slat-shaped crystals, usually included in plagioclase or interspersed with biotite-chlorite (2%);
- Chlorite occurs by replacing biotite as an alteration product. It is rich in Fe, showing pleochroism in shades of bright yellow and light green (1%);
- Epidote is included in plagioclase as a product of saussuritization (traits);
- Opaque, apatite, alanite and zircon are accessories that form euhedral, subeuhedral and anhedric crystals dispersed in the rock.

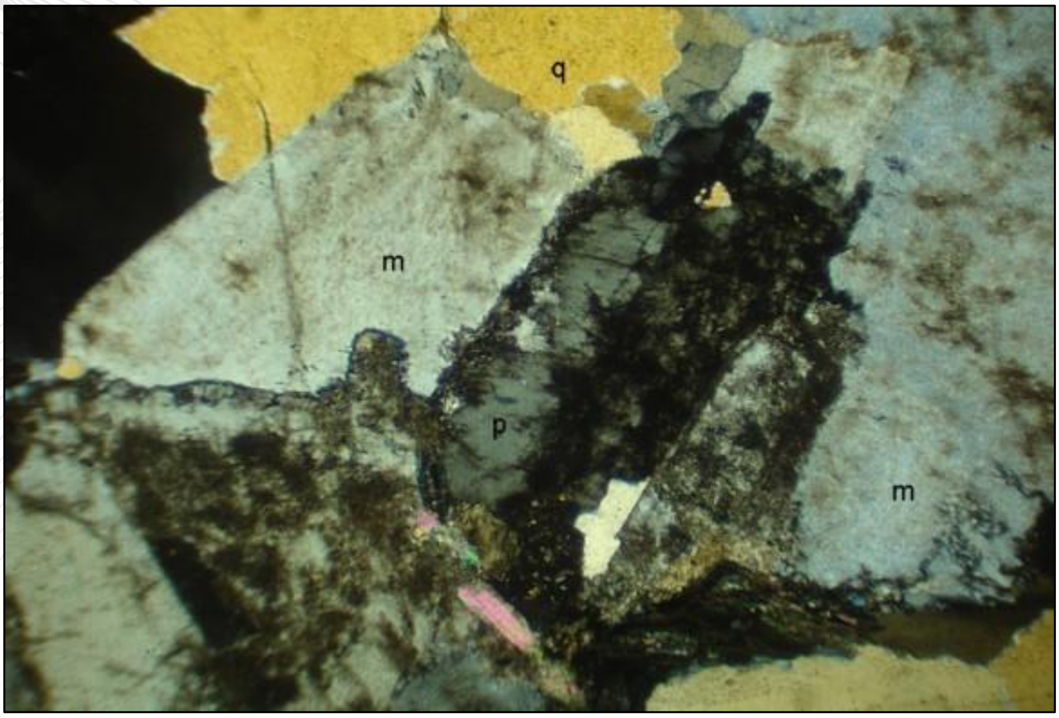


Figure 7-5 - Biotite Granodiorite with Microcline (m) Surrounding Plagioclase (p) and Apparently Replacing It. Interstitial Quartz (q) and Laths of Muscovite. 25X, crossed nicols.

7.5.3 DIORITE/GABBRO INTRUSIVE STOCK

The Diorite/Gabbro bodies, which age dates from 1.763 +- 9.1 Ma (Pineschi, 2022), cover a large area, are phaneritic, magnetic, fine to medium grained, isotropic, and are mainly composed of plagioclase and hornblende, but locally phenocrystals of local plagioclase are also observed. When found in outcrop, these Diorite/Gabbro units form blocks and slabs. Using information from the field and from diamond drilling (Figure 7-6 – A and B), it is noted that sometimes these units may display potassic alteration associated with structures (millimeter to sub-centimeter thick quartz veins and fractures), which in turn may present associated epidote and carbonate alteration (calcite) and sulfide occurrence.

A thin section was generated from a sample of the diamond drill hole (DDH) FX1D-0058 (Figure 7-6 – B and Figure 7-7) core and the petrographic description resulted in the following:

- Plagioclase occurs in subhedral, saussuritized crystals, including total pseudomorphosis in some portions of the plot (32%);
- Dark green amphibole (hornblende-edenite series) changes to greenish to colorless amphibole (tremolite-actinolite series), reddish brown biotite, carbonate, epidote (including metamitic allanite), goethite or directly to chlorite and clay mineral. (18%);
- Irregular quartz (6%) grains and K-feldspar (3%) crystals occupy the intergranular spaces between the tabular plagioclase crystals (6%);
- Magnetite occurs in crystals preserved or partially oxidized to hematite (6%);
- White clay/Sericite (6%);
- Clinopyroxene (diopside-augite series) is mainly replaced by dark green amphibole (4%);
- Chlorite (4%);
- Rutile/Anatasius (2%);
- Biotite, in turn, changes to chlorite (Trace);
- Clay mineral (<1%);
- Epidote (12%);
- Carbonate <1%.

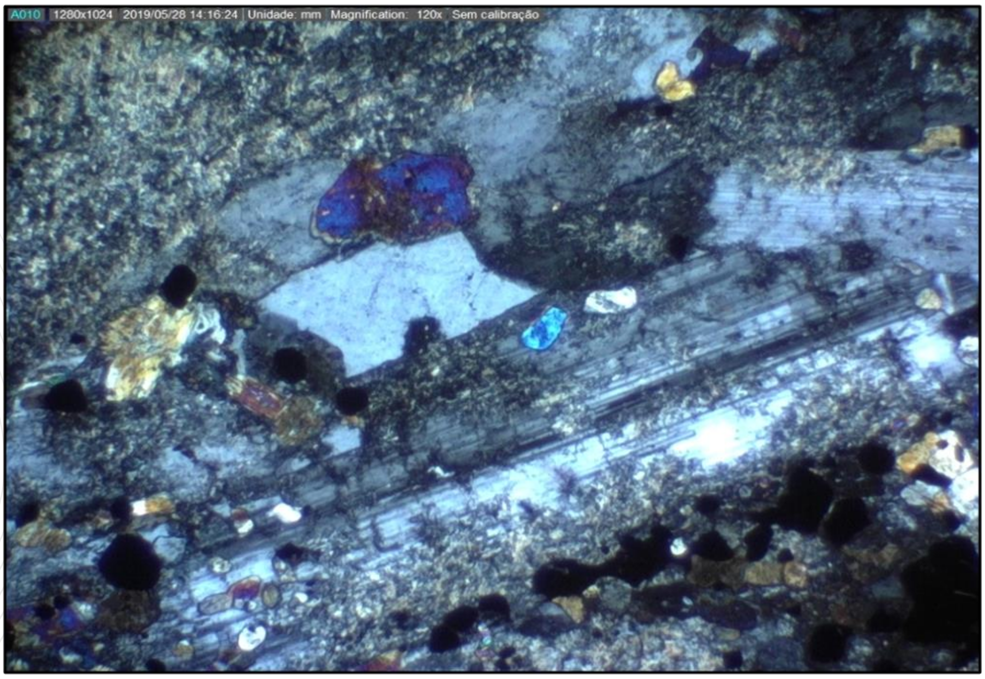


Figure 7-6 - A) In-situ Blocks of Magnetic Porphyritic Diorite in X1 Property; B) Magnetic porphyritic diorite presenting potassic and epidote alteration seen in DDH FX1D-0058 (130.40 m) in X1 property; C) In-situ blocks of magnetic diabase with disseminated pyrite in Alto Alegre blocks; D) In-situ block of Magnetic diabase with disseminated pyrite seen from Alto Alegre blocks.

Figure 7-7 - Photomicrograph of sample MTP-PET-018 - Hole: FX1D-0058 - Depth: 130.40 m. Texture aspect of the rock. Sausuritized plagioclase (Pl) crystals associated with clinopyroxene (Cpx) and amphibole (Amp) crystals. Observe quartz (Qz) in intergranular space. Transmitted light, crossed nicols, 2.5x objective, 10x eyepieces.

In addition to diorite/gabbro stocks, other basic instructive bodies occur in the area intruding both gabbro/diorite as well as biotite granodiorite. These bodies are represented by diabase dykes, with dimensions that vary in the order of a few tens of meters long and a few meters thick, up to small centimeter injections (field work and drilling information). These diabase dykes are generally very fine-grained, green in color and composed mainly of plagioclase, hornblende and chlorites, which can occur in an aphanitic form, possibly when fragments from the edges of the dykes are observed (Figure 7-5 – C and D). They are magnetic (weak to moderate magnetism) and usually present sulfidation represented by disseminated pyrites (up to 3%).

A thin section was generated from a hand sample (Figure 7-6 – D and Figure 7-8) of this lithotype collected from outcrops in Alto Alegre blocks resulting in the following:

- Granular rock, homogeneous and of fine- to medium-grained locally, hypidiomorphic still preserved despite the rock showing intense processes of mineralogical substitutions;
- Plagioclase occurs in subhedral (lath) maculated crystals according to albite/Carlsbad and saussuritized;
- K-feldspar is represented by the microcline, in anhedral (irregular) to subhedral crystals, incipiently argilized and impregnated with goethite;
- Clinopyroxene (diopside-augite series) changes to dark green to brown amphibole (hornblende-edenite series), brown to greenish biotite, epidote, clay mineral, goethite, carbonate or directly to chlorite;
- Amphibole crystals are also replaced by chlorite. Relict biotite crystals are replaced by chlorite and are associated with neoformed epidote crystals, sometimes with metamitic allanite nuclei;
- Randomly arranged, fine apatite crystals are common in this web. Titanite/leucoxene fine crystals/aggregates are dispersed throughout the rock and mainly associated with amphibole, chlorite and opaque minerals;
- Opaque minerals constitute euhedral to anhedric crystals/blasts, disseminated through the rock or as exsolution products of amphibole and phyllosilicate crystals. Some opaque minerals are associated with the newly formed epidote crystals. They are represented by oxides and sulfides, isolated or associated with each other;
- Pyrite appears in the form of blasts or fine crystals dispersed throughout the rock. Some pyrite crystals are laterally associated with chalcopyrite;

The rock is cut by random fractures (submillimeter width) filled with epidote ± carbonate ± chlorite ± quartz ± amphibole.

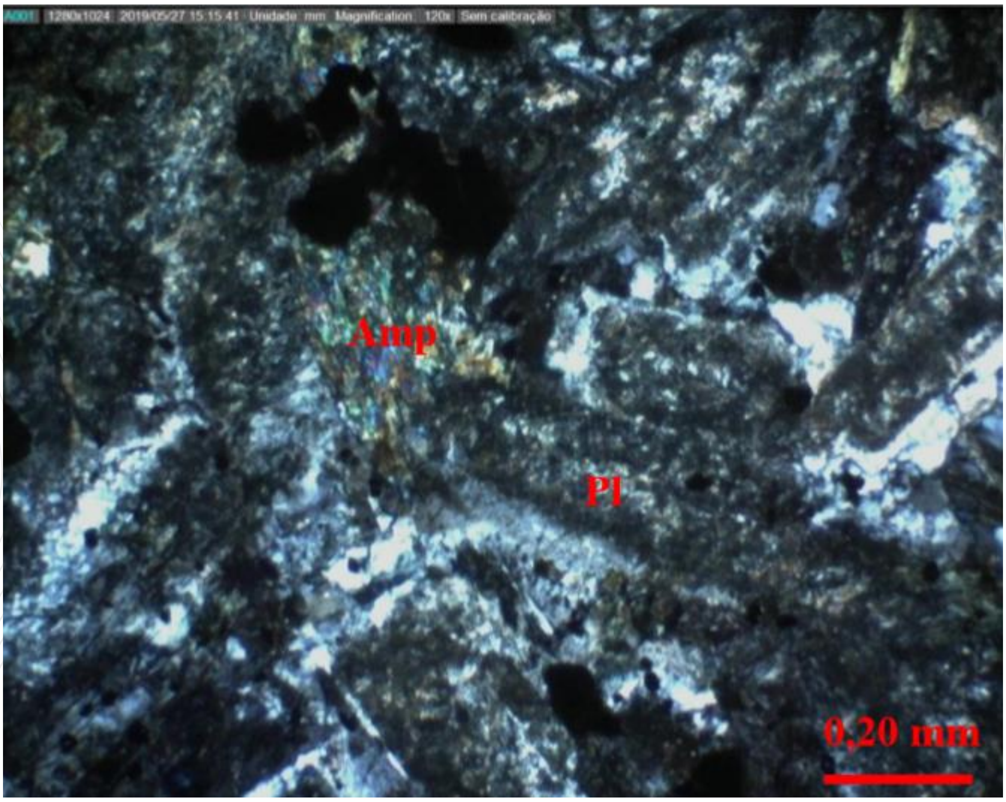


Figure 7-8 - Textural Aspect of Diabase. Sausuritized plagioclase (Pl) crystals associated with amphibole crystals (Amp). Transmitted light, crossed nicols, 5x objective, 10x eyepieces.

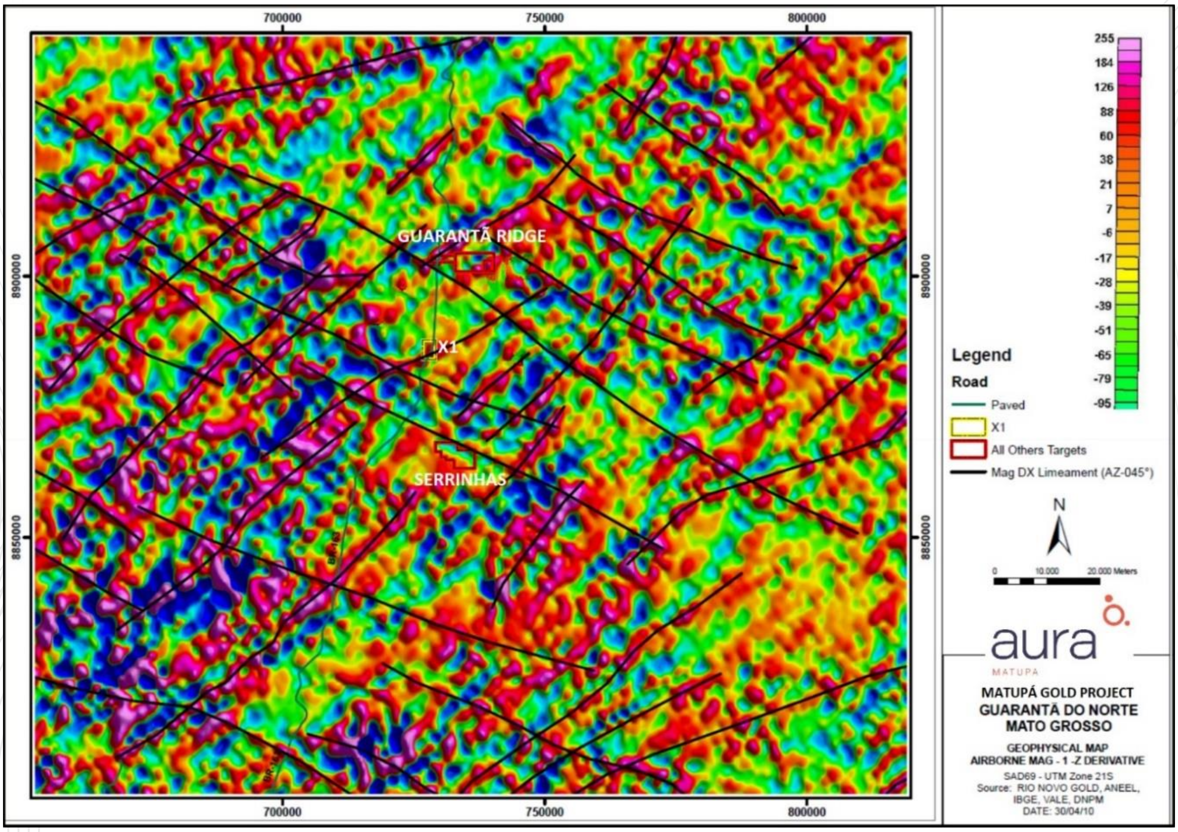
7.6 DISTRICT STRUCTURAL GEOLOGY

According to Lacerda Filho et al. (2000 and 2001), and Souza et al. (2005) two tectono-structural domains have been identified for the Juruena-Teles Pires Mineral Province. The first one (2.2 Ga and 1.99 Ga) is characterized by progressive ductile compressional tectonics to the northeast which resulted in transcurrent zones orientated northwest–southeast and east-west, that were later reactivated.

The second one had a ductile-brittle regime with N55°E as the main direction, that also correlates to its maximum compression. This second structural domain was characterized as the last deformational event that affected this geological province, when mega shear zones were formed, which followed a northwest–southeast sinistral and dextral north–south ductile zones, which correlate to major events of gold mineralization in this area.

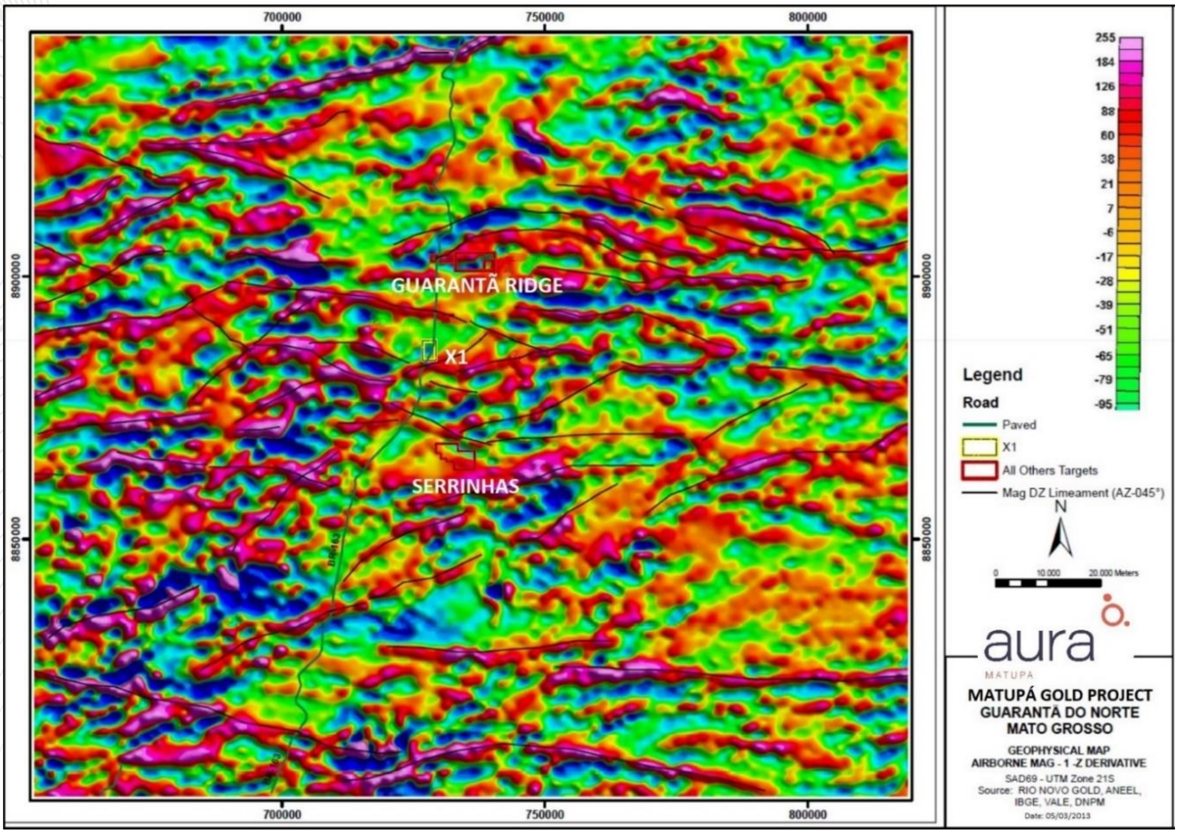
At the X1 Deposit and extensions two main fault directions (NW-SE and NE-SW) and two secondary (E-W and N-S) were identified, both were observed in outcrop on a regional scale in satellite images and magnetometric geophysical images in first horizontal and vertical derivates (Figures 7-9 and 7-10).

The phyllic alteration zones are usually cut by fractures filled with veins, or sometimes with oxide films, and quartz boudins, which have many structural directions, forming an incipient foliation only in the contact between them, also sigmoid indicating sinistral and dextral movement is displayed. Outcrops where the veins and/or quartz boudins, with or without sulfides, form structures with Y and/or T shapes, these were interpreted as the hydrothermal, syn to late tectonic breccia matrix, filling the spaces between the rounded clasts of phyllic alteration.



Note: Coordinates are South American Datum (1969), UTM Zone 21 South.

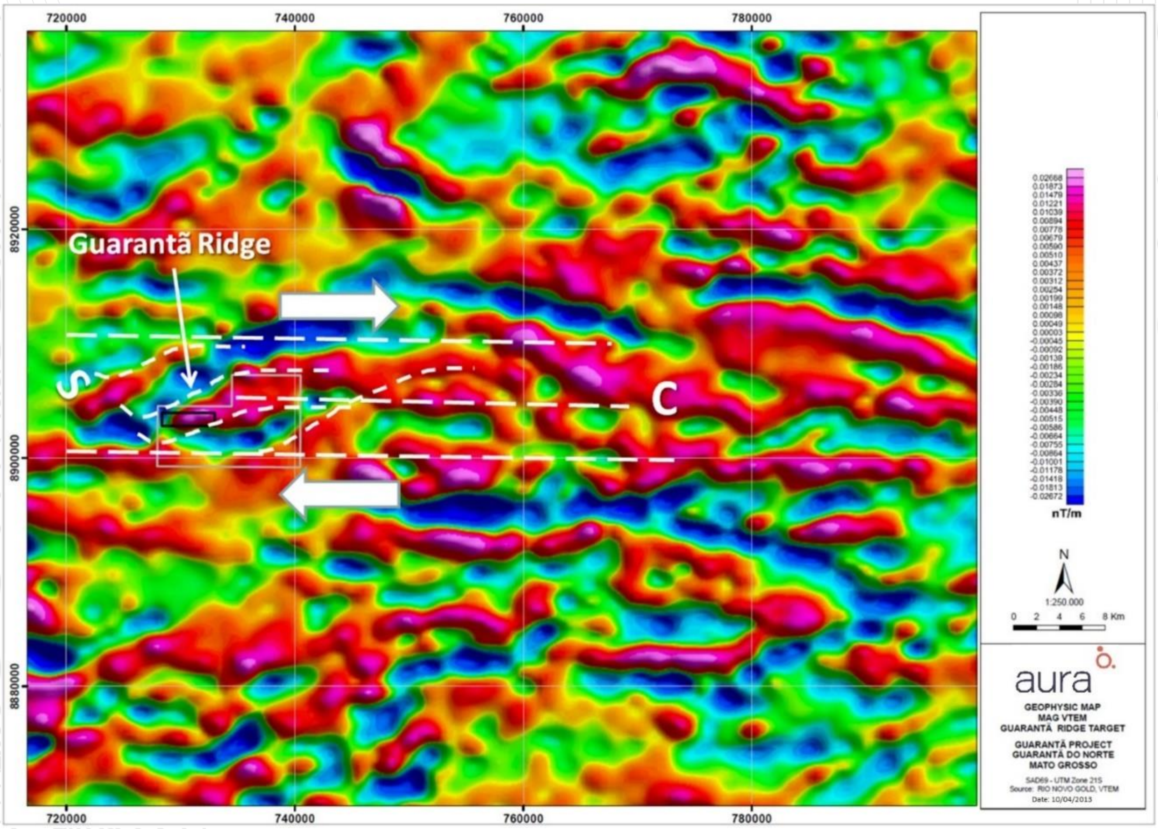
Figure 7-9 - Magnetometric Image of First Horizontal Derivate (CPRM) Showing Strong NW-SE Trend Cutting a NE-SW Early Lineament.



Note: Coordinates are South American Datum (1969), UTM Zone 21 South.

Figure 7-10 - Magnetometric Image of First Vertical Derivate (CPRM) Showing Strong E-W/WNW-ESE Trend Interpreted as the C Component of a Dextral Fault, and a Weak NE-SW Trend as its S Component.

Regionally, the Guarantã Ridge Target is within a strong tectonic structure striking east-west. Structurally, all the rocks of this target are oriented east-west and cut by regional faults (Figure 7-11) whose main trend is northwest and secondarily, north-south and northeast. Some northwest regional lineaments are interpreted as dextral or sinistral strike slip faults based on field data and geophysical surveys.



Note: Coordinates are South American Datum (1969), UTM Zone 21 South.

Figure 7-11 - Airborne MAG (First Derivate) Showing a Structural Interpretation for Guarantã Ridge as Part of a Dextral Regional Strike Slip Structure.

The volcanic bedding has an east-west strike dipping between 30° and 60 ° north, in spite of some rare occurrences were the bedding dips to the south. The main structural features found during detailed geological mapping in the Guarantã Ridge Target were sets of dilatational fractures, strike slip and normal faults and volcanic bedding. Hydrothermal breccias occur filling these main structures east-west varying from N60°W to N65°E, as a Riedell system of fracturing so that it can be seen in outcrop.

7.7 DISTRICT ALTERATION AND MINERALIZATION

The processes of hydrothermal alteration, observed in the garimpo mining pits or confirmed in diamond drilling, are believed to result from a magmatic-hydrothermal system with a strong relationship to the regional shear zone in the northwest direction, tens of kilometers in length, which are thought to correlate to the genesis of mineral deposits in the southeast of the AFGP.

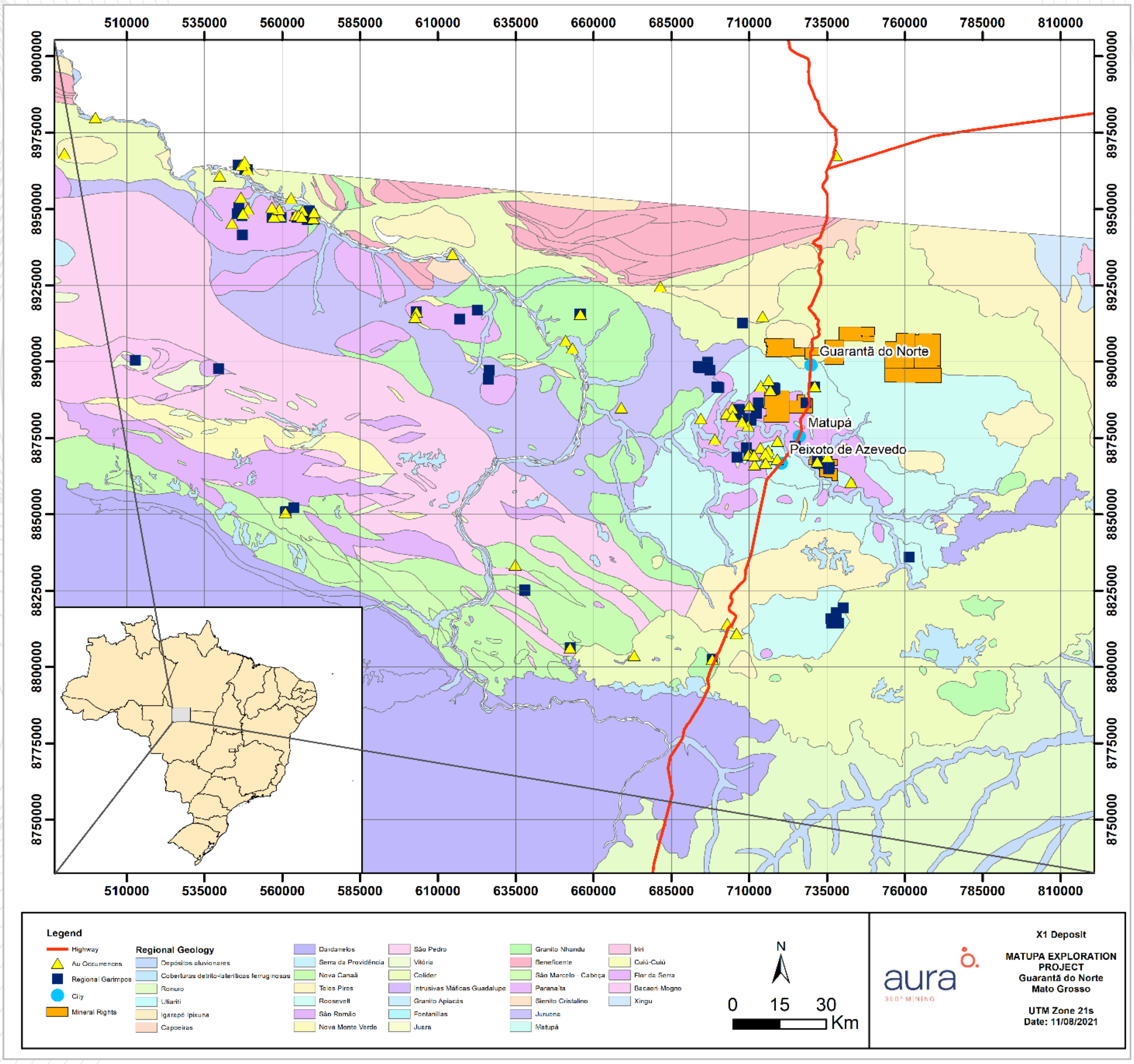
Disseminated gold mineralization from AFGP displays a strong phyllic alteration envelope (sericite+quartz+pyrite) spatially and is genetically related to acid porphyritic dykes or to potassic alteration, represented by biotite enrichment or microcline, and replacement of original K-feldspar, with or without silicification, and associated to disseminated sulfides, mostly pyrite.

Gold deposits at AFGP are divided into two major groups: (1) Epithermal and pluton-related high-grade quartz vein deposits ±(Zn-Pb-Cu-Fe) with restricted sericite, chorite and carbonate alteration; and (2) Low-grade disseminated deposits ±(Ag-Bi-Te-Pb-Zn) with “phyllic” fault-controlled alteration (muscovite-quartz-pyrite) and strong to moderate silicification.

The X1 Deposit, Serrinhas and V6 targets represent disseminated gold mineralization hosted by the Matupá Granite.

Guarantã Ridge Target represents epithermal stockwork mineralization where the ore occurs in hydrothermal breccias zones and epithermal veins hosted by interbedded sediments and volcanic rocks of the Colíder Group. In addition, lode quartz veins, sometimes associated with massive sulfide mineralization, are found in shear zones at the Alto Alegre Block, such as the Valdemar and Eron veins, usually presenting high grades of up to 261.1 g/t Au and 0.91% Cu.

Many of the rocks that host mining occurrences in the region occur essentially in contact zones of intrusive basic rocks in the Matupá Suite granodiorites and may also be similarly related to gneisses. These typical weakness zones formed in the lithological contacts were reactivated, hydrothermalized and mineralized, possibly as a reflection of the intrusion of younger bodies during the regional shear, which would be responsible for the source of heat and fluids mineralizing shallower systems (Figure 7-12).



Note: Coordinates are South American Datum (1969), UTM Zone 21 South.

Figure 7-12 - District Geology Showing Known Garimpos.

The remobilization of silica and the percolation of these fluids in a shallower crust environment provided, in the region, several features of epithermal to mesothermal deposits, with typical filling textures (open space filling) such as quartz in comb and also cock-comb textures, crustiform banding – colloform, pseudo acicular, saccharoidal texture, zoned crystals and even bladed silica (replacement of calcite by silica). These textures are mainly seen on the surface, but also in drill cores.

7.8 X1 DEPOSIT TYPES

The X1 Deposit occupies a topographic high point (hill) and is hosted by the Matupá Intrusive Suite near the contact with the mafic/ultramafic Flor da Serra Suite. The extents of the deposit have dimensions of 400 m east-west and 250 m north-south.

Porphyritic granitic and others acidic intrusive rocks are the main lithotypes at the X1 area. Basic intrusive to intermediate rocks were also mapped and described from DDH. Therefore, the main rock types identified during the geological mapping (Figure 7-13) and core logging are:

- Porphyritic Biotite Monzogranite (BMGR) and porphyritic granodiorite to coarse-grain fabric, light gray color, composed by quartz, plagioclase, potassium feldspar phenocrysts (pink microcline), biotite and magnetite (Figures 7-14 – A and B). Usually isotropic but might presents incipient foliation when close to shear zones;

- Quartz Feldspar Porphyry (QFP) is pinkish, with fine-grain matrix composed by quartz, feldspar and traces of biotite, including rounded quartz phenocrysts, confirming its subvolcanic origin (Figures 7-14 – C and D);

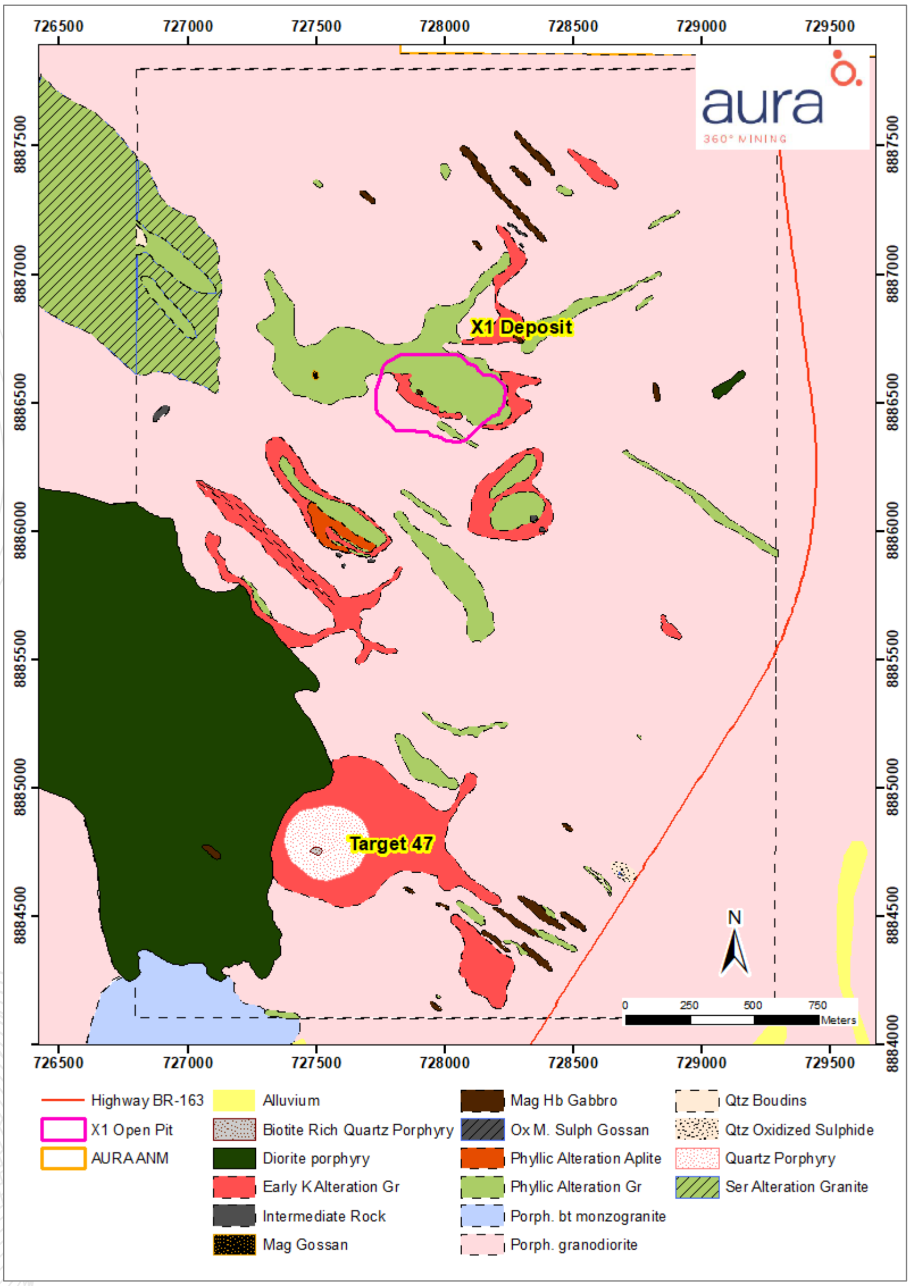
- Early Potassic Alteration (EKGR) occurs in all rocks described above as a weak to medium alteration intensity. The early K alteration prints on the rocks a greenish-red color due the microclinization, sericitization and chloritization of the feldspars and biotite (Figures 7-15 – A and B);

- Phyllic Alteration (PHGR) occurs in granitic rocks and in quartz feldspar porphyry. PHGR is oxidized in surface, composed of quartz, sericite/muscovite and oxidized sulfide (disseminated or venules), which also can usually be found as millimeter to centimeter boxworks sulfides (pyrite and chalcopyrite) disseminate. In the DDH the phyllic alteration has the same composition, but the sulfides are clearly observed, with pyrite is the primary sulfide, and chalcopyrite and bornite are subordinates. Sulfides usually occur as disseminated, but also occurs as semi to massive sulfides and with millimeter to centimeter clusters (Figures 7-15 – C and D and Figure 7-16 – A and B);

- Diorite (DR) is phaneritic, with fine- to medium-grain, isotropic and comprised mainly of plagioclase and hornblende, with locally plagioclase phenocrysts, occurring as boulder outcrops (Figure 7-6 – A and B);

- Diabase (DB) with fine-grain subvolcanic texture, is magnetic and composed of hornblende, plagioclase and traces of sulfides as pyrite and chalcopyrite (Figure 7-6 – C and D);

- Intermediate Volcanic Rock (IVR) with very fine-grain, green color, is composed mainly of plagioclase, hornblende and chlorite.



Note: Coordinates are South American Datum (1969), UTM Zone 21 South.

Figure 7-13 – X1 Deposit Geological Map (1:10,000 scale).

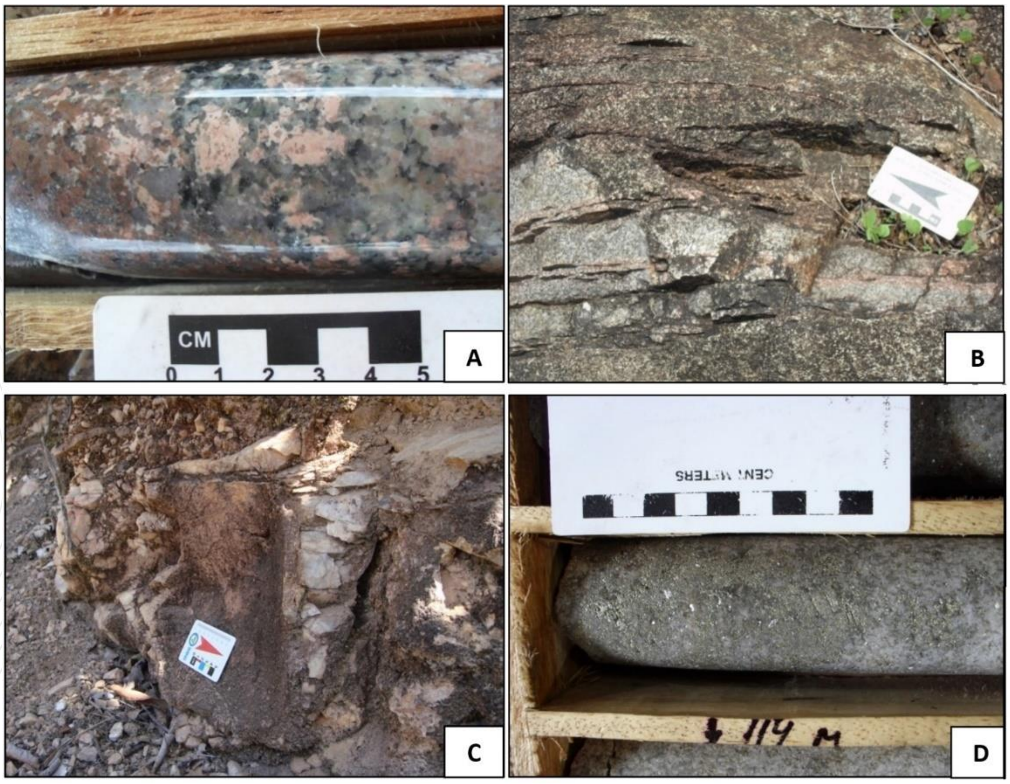


Figure 7-14 - A) Early K Alteration with pinkish and green color (FX1D-0002 - 56.0 m). B) Outcrop of biotite granodiorite with fractures associated to early K alteration (pinkish to reddish color). C) Phyllic alteration outcrop at X1 main body showing Quartz + Sericite + boxwork of pyrite. D) Phyllic alteration in granite (FX1D-0017 – 115.83 m).

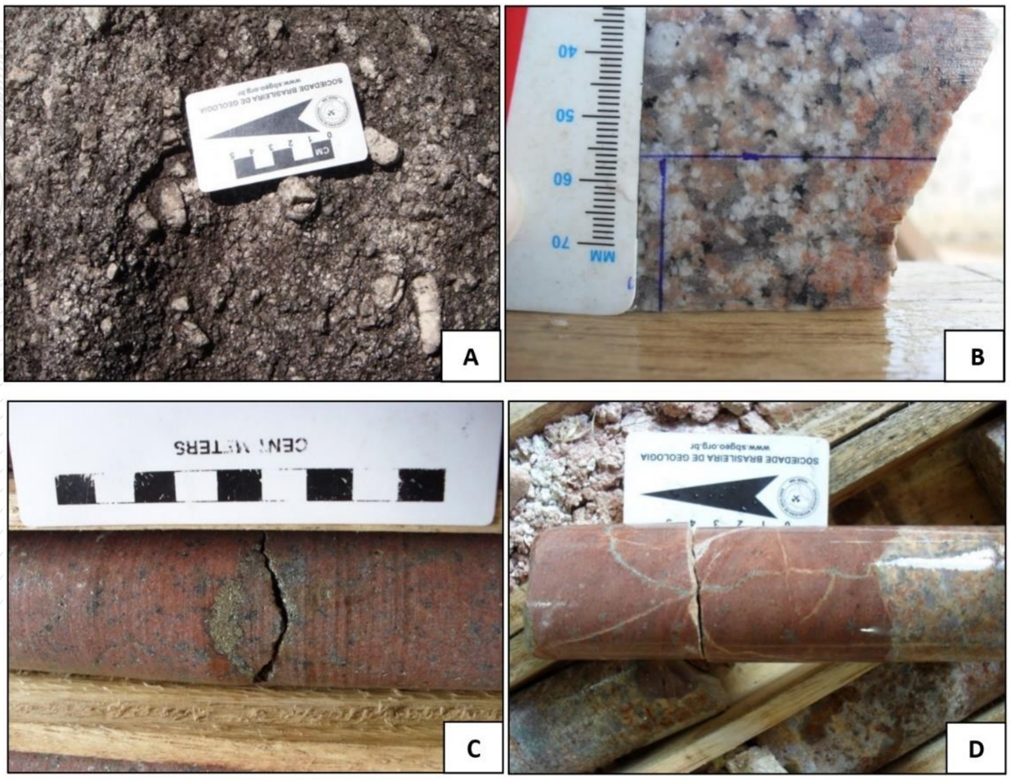


Figure 7-15 - A) – Outcrop of Porphyritic Biotite Granodiorite. B) Porphyritic biotite monzogranite in the DDH FX1D-0002 (221.22 m). C) – Quartz Feldspar Porphyry with pinkish color and pyritic vein (FX1D-0017 – 178.47 m). D) Pinkish Quartz Feldspar Porphyry with disseminated clusters of pyrite and intrusive contact with EKGR (FX1D-0003 – 52.70 m).

Figure 7-16 - A) DDH SEX1-01 (74.00 m) Where Massive Pyrite was Observed. B) DDH SEX1-01 (120 and 121 m) with Pyrite clusters texture named "Pele de Onça".

7.8.1 EARLY POTASSIC ALTERATION GRANODIORITE (EKGR)

Three thin sections were created from the Early Potassic Alteration rocks from the granodiorite samples in order to classify this specific alteration that was thought to be sodic-calcic alteration. The petrographic resulted on the following:

- Quartz occurs as coarse-grained anhedral and interstitial crystals displaying strong undulatory (30-35%);
- Plagioclase is andesine forming medium to coarse-grained subhedral crystals displaying intense sericitization (30-40%);
- Microcline forms medium to coarse-grained anhedral crystals, also occurs surrounding plagioclase and was locally observed replacing it, suggesting a potassium metasomatism. The microcline shows perthitic intergrowths, intense kaolinization and locally tartan twining. When the rock is weathered it displays a pinkish red color due to kaolinization and fine hematite dissemination (20-25%);
- Muscovite occurs as fine-grained lath-shaped non-oriented crystals disseminated in the rock and usually included in plagioclase, locally replacing plagioclase crystals forming pseudomorphs (1-06%);
- Chlorite forms medium-grained tabular crystal pseudomorphs after biotite, displaying lots of rutile inclusions and showing pleochroism in pale yellow and light green tints suggesting high Fe grade (3-06%);
- Carbonate has local occurrence and was observed in fractures (1%);
- Epidote is an accessory mineral forming sparse fine-grained crystals disseminated in the rock, occurring in one thin section only (Tr);

- Rutile forms fine-grained crystals included in chlorite and some medium-grained crystals associated with carbonate and displaying intense alteration to leucoxene. Rutile also, occurs locally as medium-grained prismatic crystals associated with pyrite (tr-02%);
- Pyrite occurs as fine- to medium-grained anhedral to subhedral crystals, in fractures and disseminated in the rock, forming locally large aggregates (Tr-04%);
- Chalcopyrite, bornite and chalcocite are trace minerals and were locally observed as fine-grained anhedral, disseminated crystals usually intergrown with each other. Chalcopyrite is being replaced by bornite and the latter is being replaced by chalcocite;
- Sphalerite occurs as a trace mineral associated with chalcopyrite;
- Magnetite was locally observed as rare fine-grained euhedral, disseminated crystals, partially altered to hematite;
- Apatite and zircon are accessories forming sparse fine-grained disseminated crystals.

The Early Potassic Alteration rocks are an acid intrusive igneous rock, weakly (sericitization, kaolinization, early microclinitization and chloritization) to intensely hydrothermally altered (silicification, sericitization and sulfide enrichment), having the original igneous idiomorphic granular texture well preserved (Figure 7-17).

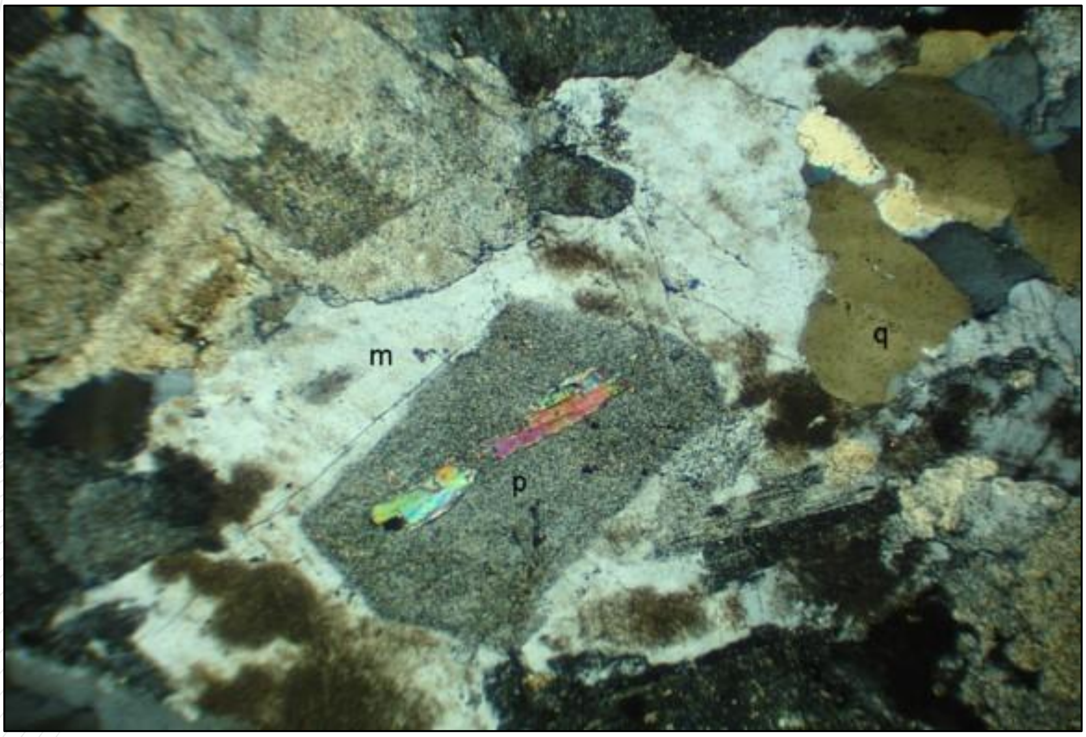


Figure 7-17 - Microcline (m) Surrounding Plagioclase (p) and Apparently Replacing It Suggesting a Potassium Metasomatism. Interstitial quartz (q) and laths of muscovite (bright colors) included in plagioclase. 25X, crossed nicols.

7.8.2 PHYLLIC ALTERATION WITH AU ASSAY RESULTS

From five thin sections with good assay results, only two of them showed gold, but all of them had occurrences of Cu monosulfides as resulting from chalcopyrite oxidation. Those rocks were classified as light gray coarse-grained massive rock, non-magnetic, displaying colourless quartz, light gray laths of muscovite and disseminated sulfides, locally displaying milky white crystals of quartz.

The microscopic texture was described as coarse-grained massive rock displaying granoblastic decussate texture. Sometimes this rock displays relicts from the original volcanic porphyritic texture.

- Quartz occurs as coarse-grained anhedral crystals forming a massive aggregate associated with muscovite, or quartz is present as relict euhedral to subhedral phenocrysts and a second generation of fine-grained polygonal crystals the product of metamorphic recrystallization (crystalloblastesis) forming massive aggregates in a granoblastic textural pattern (3-65%);

- Muscovite occurs as medium-grained lath-shaped non-oriented crystals, disseminated and forming irregular aggregates. Muscovite was observed forming euhedral prismatic pseudomorphs possibly after feldspars (27-40%);

- Plagioclase was described and forms sparse subhedral medium-grained crystals intensely altered to sericite and limonite and included in muscovite aggregates (Tr);

- Chlorite was observed as small flakes included in muscovite and intergrown with sulfides (Tr);

- Pyrite forms medium to coarse-grained euhedral to subhedral crystals concentrated in some zones of the rock and sometimes included in chalcopyrite (3-63%);

- Chalcopyrite forms medium-grained anhedral crystals partially replaced by covellite and associated mostly with muscovite aggregates and included in pyrite, replacing it. Sometimes chalcopyrite is alternated to other Cu sulfides (2-15%);

- Covellite is seen replacing chalcopyrite, locally forming pseudomorphs, and also infilling fractures in pyrite. Covellite is bright blue in color, displaying strong anisotropism and pleochroism (Tr-03%);

- Chalcocite usually occurs in chalcopyrite as alteration products of it. However, some thin sections show chalcocite as medium-grained euhedral, subhedral and anhedral crystals, associated with muscovite aggregates, and is partially replaced by covellite. Chalcocite replaces pyrite and chalcopyrite and is replaced by covellite. The crystals display a pinkish gray color, which probably represents a solid solution chalcocite-bornite (Tr-01%);

- Bornite was locally observed filling fractures in chalcopyrite as alteration products of it (Tr);

- Galena occurs intergrown with pyrite, on the borders and along fractures, apparently replacing it, or as fine-grained anhedral crystals usually surrounding chalcopyrite and usually displaying typical triangular pits. Disseminated galena occurs in the transparent gangue (Tr);

- Rutile, zircon and apatite are accessories forming sparse fine-grained euhedral to subhedral crystals disseminated in the rock;

- Ilmenite is an accessory mineral forming medium-grained subhedral crystals partially altered to rutile;

- Native Gold (Sample 19084 – SEX1-01) occurs in fractures in pyrite associated with covellite and chalcocite and also occurs as includes in pyrite. Twenty-six grains of native gold were detected, 10 μm – 350 μm in size, 9 in covellite in fractures in pyrite, 14 in pyrite, 2 in chalcocite including in pyrite and 1 in chalcocite in a pyrite fracture (Tr) (Figures 7-18, 7-19 and 7-20);

- Native Gold (Sample 19125 – SEX1-01) occurs mostly as inclusions in chalcopyrite in fractures in pyrite. Seventy-nine particles of native gold were detected, 2 μm – 150 μm in size, 54 in chalcopyrite in pyrite filled fractures, 11 in pyrite, 13 in chalcocite in fractures in pyrite, and 1 at the junction of chalcocite-silicates. Some of the gold grains are bright yellow suggesting low-grade silver is present and some are pale yellow suggesting high-grade silver (electrum). Apparently, the bright yellow gold particles predominate but the amounts of each type are similar. Chalcocite including gold grains is replacing chalcopyrite (Figure 7-19).

The rock is a product of intense hydrothermal alteration (silicification, muscovitization, sulfide enrichment and gold enrichment), preserving no textural relicts from the original rock, making it difficult to suggest about its protolith origin. After the review of historical information carried out by Aura, a relog of historical drilling was performed in order to identify the mineralization distribution on the X1 Deposit, which made clear that the ore is related to the quartz feldspar porphyry intrusion affected by an intense retrograde phyllic alteration (quartz+pyrite+muscovite).

In terms of temporal occurrence, the potassic alteration took place first, with microcline ± hematite replacing plagioclase and original K-feldspar (ortoclase) in the matrix and rock's phenocrystals in the X1 rocks, such as granodiorite and quartz feldspar porphyry, which was later overprinted by the phyllic alteration (muscovite + quartz + pyrite) on a retrograde phase. Figure 7-22.

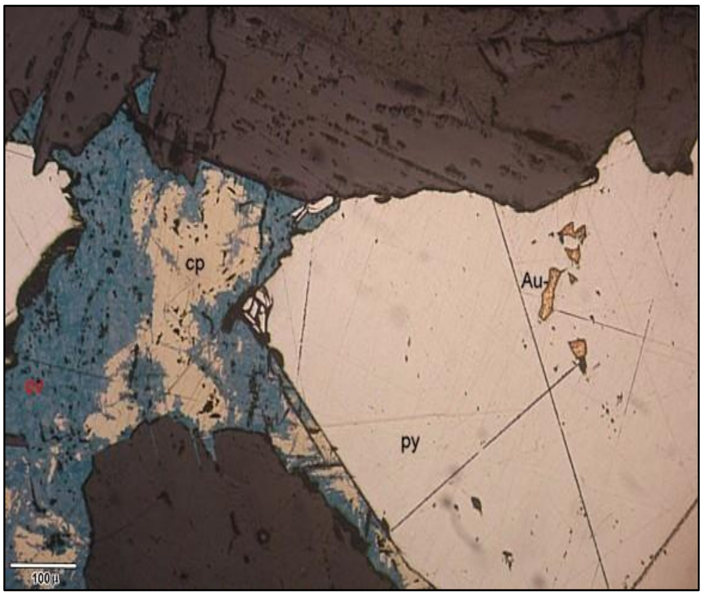


Figure 7-18 - Phyllic Alteration with Native Gold (Au) Included in Pyrite (py). Covellite (cv) replacing chalcopyrite (cp). Sample 19084, 100X, plane reflected light.

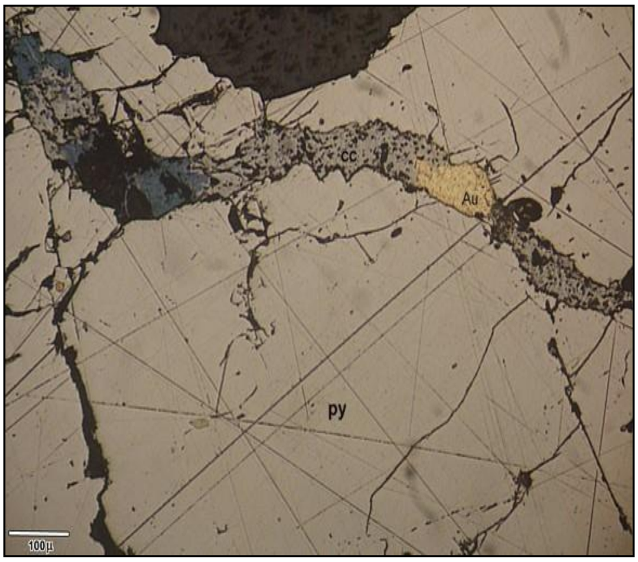


Figure 7-19 - Phyllic Alteration with Native Gold (Au) in Chalcocite (cc) filling fracture in pyrite (py). Covellite (blue) replacing chalcocite. Sample 19084, 100X, plane reflected light.

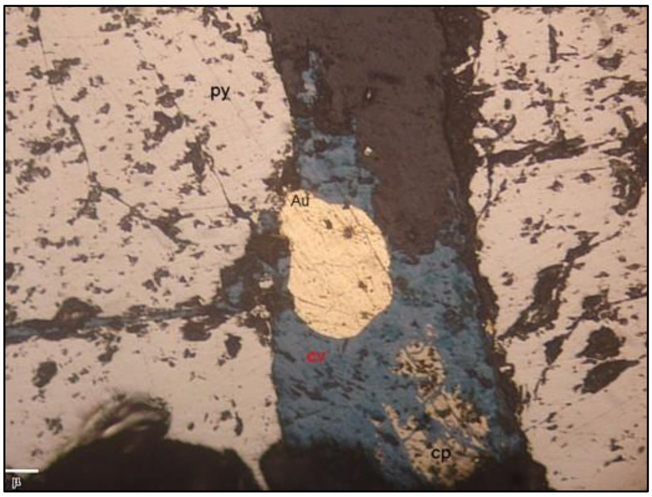


Figure 7-20 - Phyllic Alteration with Native gold (Au) in Covellite (cv) in Fracture in Pyrite and Replacing Chalcopyrite (cp). Sample 19084, 200X, plane reflected light.

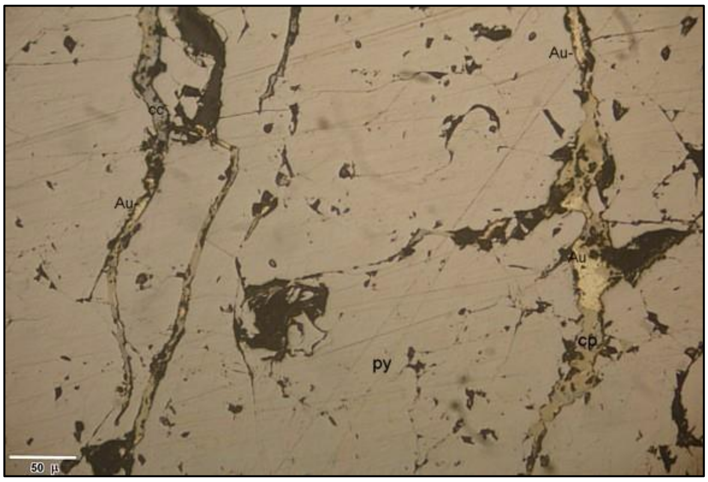


Figure 7-21 - Phyllic Alteration with Native Gold (Au) in Chalcopyrite (cp) in Fracture in Pyrite (py). Sample 19125, 200X, plane reflected light.

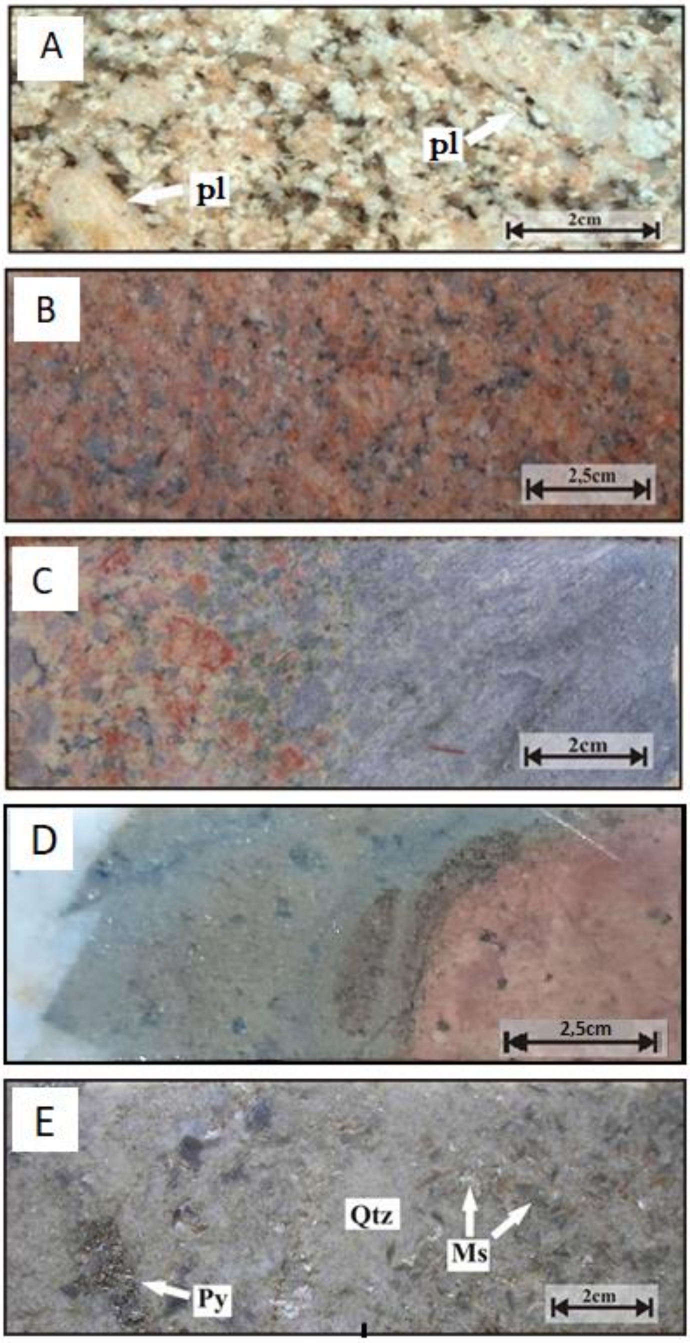
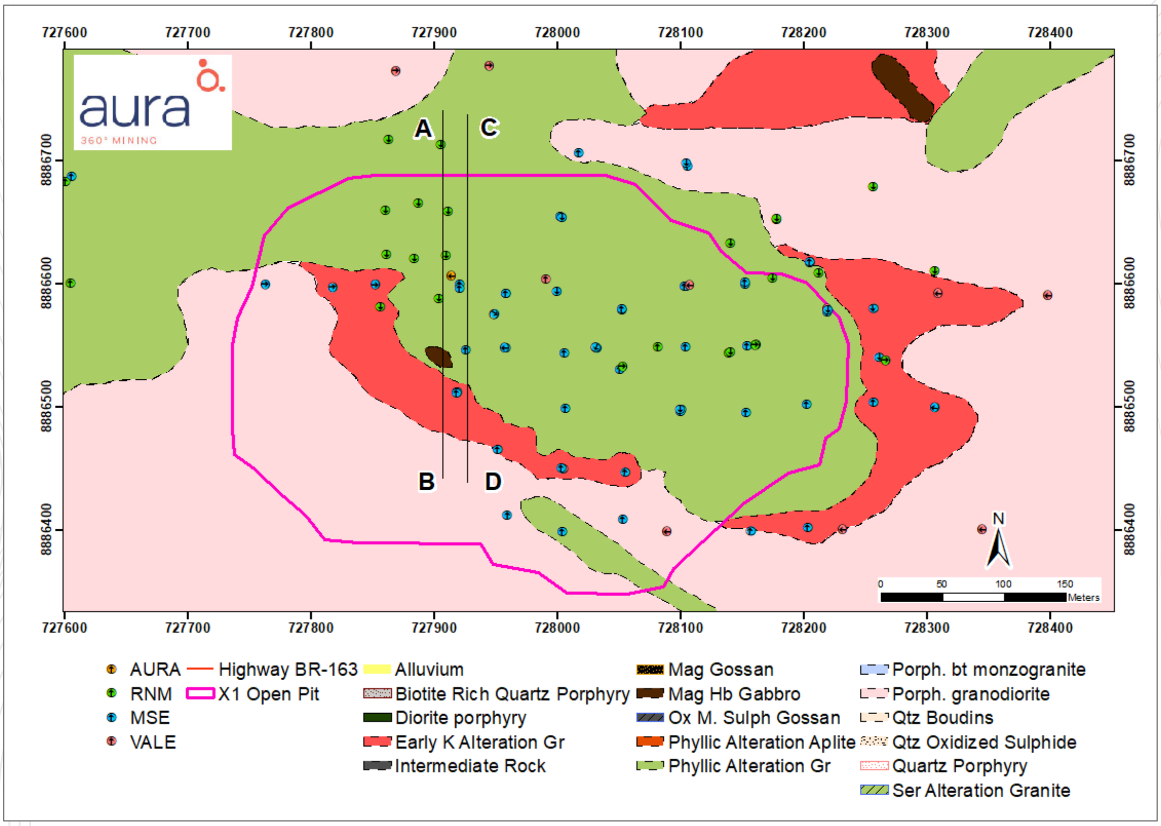


Figure 7-22 - A) Unaltered Porphyritic Biotite Granodiorite. B) Early potassic alteration. C) Phyllic overprinting in granodiorite. D) Phyllic overprinting in Quartz Feldspar Porphyry. E) Phyllic alteration (muscovite + quartz + Pyrite alteration).

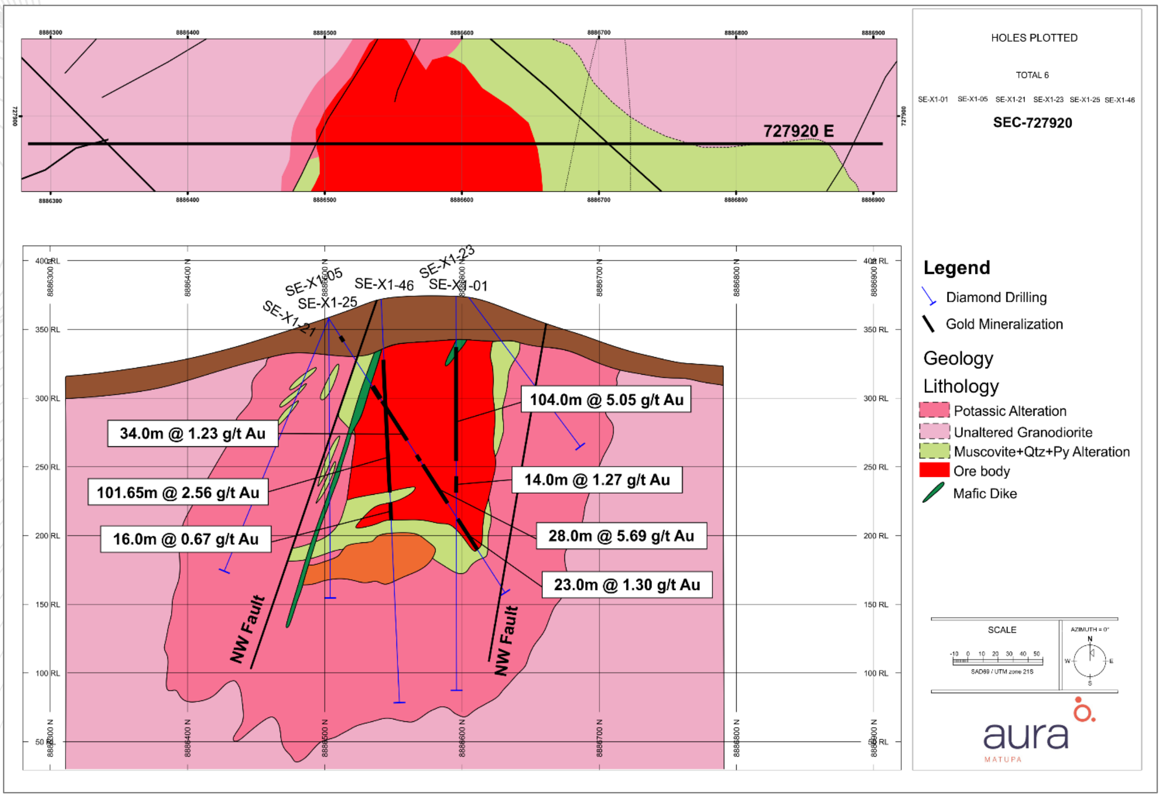
As detailed in section 6, Rio Novo carried out an infill drilling program at the X1 Deposit oriented by a ground geophysical survey. This detailed drilling program allowed the reinterpretation of the hydrothermal alteration and improvement of geological understanding of the Deposit.

After completing the infill program, Rio Novo geologists carried out a cross section interpretation (Figure 7-23, 7-24, and 7-25) based on the core logging, which in turn was based on hydrothermal alteration, since the ore zone are directly related to the phyllic alteration (quartz + muscovite + pyrite). The purpose of this interpretation was to guide a new 3-D Mineral Resource model and estimation.



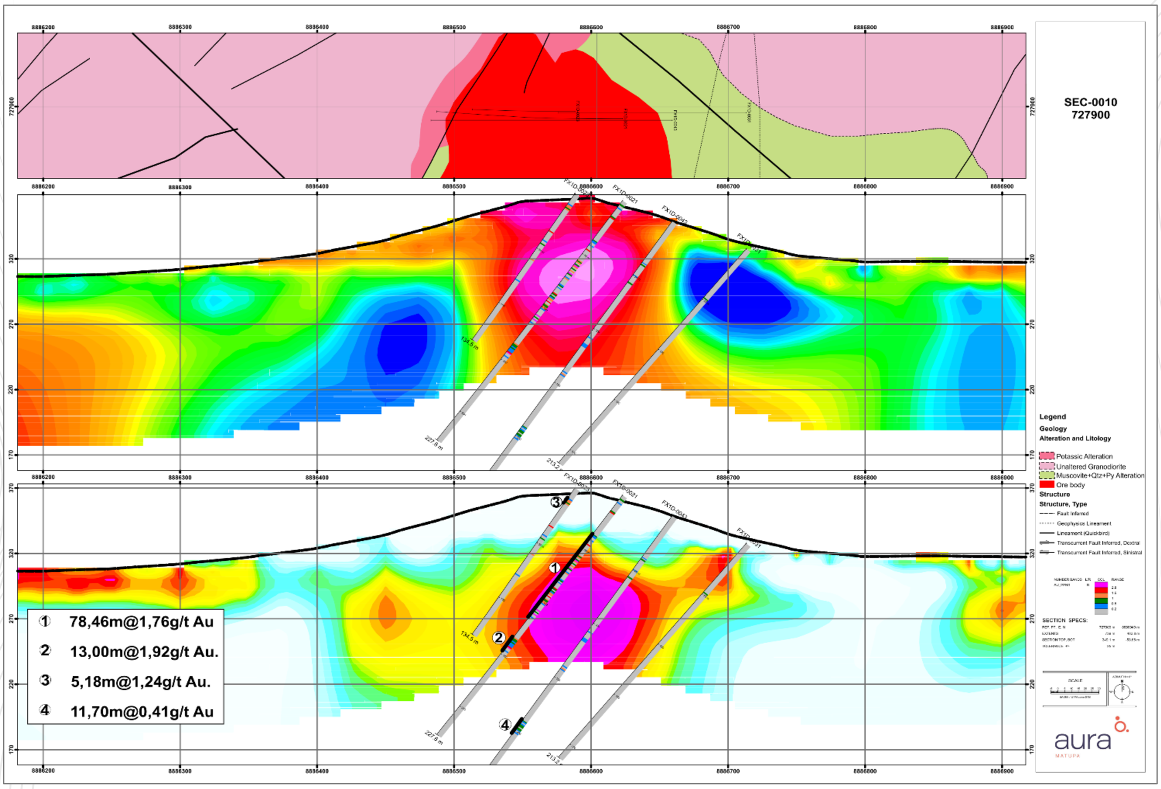
Note: Coordinates are South American Datum (1969), UTM Zone 21 South.

Figure 7-23 - Diamond Core Drilling at X1 Deposit.



Note: Coordinates are South American Datum (1969), UTM Zone 21 South.

Figure 7-24 - Geophysical Cross Section A-B of X1 Deposit.



Note: Coordinates are South American Datum (1969), UTM Zone 21 South.

Figure 7-25 - Geological Cross Section C-D of X1 Deposit (Looking South).

Knowledge about the geology and localized alteration were improved during Rio Novo exploration activities. The lithologies of X1 Deposit and its rock codes are summarized in Table 7-1.

Table 7-1 - X1 Deposit Lithologies and Rock Codes.

Rock Code	Lithologies
AP	Aplite
BMGR	Biotite Monzogranite
BMGREK	Biotite Monzogranite Early Potassic Altered
BR	Breccia
EKGR	Potassic Alteration Granite
EKGRPH	Potassic Alteration Granite + Phyllic overprint
QFP	Potassic alteration Quartz Feldspar Porphyry
FTZ	Fault zone
GR	Granite
DR	Quartz Diorite
IBR	Igneous Breccia
IVR	Intermediate Volcanic Rock
LHBGB	Leuc Hornblend Gabbro
PHGR	Phyllic Alteration Granite
PHMG	Phyllic Alteration Quartz Feldspar Porphyry

7.8.3 SERRINHAS GEOLOGY

The Serrinhas Target is comprised of a sequence of small positive topographic hills distributed along a northwest anomalous trend and a primary garimpo pit which is also oriented by this northwest regional structure. Historically, each hill received a sequential name, from MP1 to MP7 and during Rio Novo exploration activies another anomalous hill was discovered, named MP8. During Aura's exploration activities another anomalous target was discovered, named Serrinhas 9. Additionally, there is a famous garimpo pit called Parazinho's pit, thus completing 10 prospective targets in the prospect. Currently the targets are named Serrinhas 1 to Serrinhas 9 and Parazinho garimpo.

Since most of the historical drilling was focused on the Serrinhas 2 target, Rio Novo relogged the drill core after recovering the historical core boxes from the target farm area. According to the report on Serrinhas 2 historical drill core relogging, the hydrothermal alteration was not considered as a relevant aspect in the old core logging by previous companies. The rock types were defined based on the descriptive petrographic classification (Streickeisen's diagram). Those companies considered the different alterations as different facies of igneous rocks in multiple intrusions. This core relogging by Rio Novo was based on hydrothermal alterations, aiming to identify the potential zones and corridors inside the target.

The rocks of Serrinhas 2 target are part of the Matupá Suite (1,872 Ma Pb-Pb) and includes biotite monzogranite (dominant), clinopyroxene-hornblende monzogranite, clinopyroxene-hornblende monzodiorite and subordinated microgranites. According to the report the granitoids show great heterogeneity with variable facies, which normally presents as porphyritic varying from biotite monzogranite, granite and sienogranite. This variation can be observed on the drill core in sub-metric to metric scale within the granitoid body, presenting gradational contacts. Occasionally mafic and acid dykes cut these main host rocks, possibly related to Flor da Serra Suite and Iriri Group, respectively.

Although the hills are distributed along the northwest structure, all of them are oriented to northeast, which is also the main direction of later mafic dyke events in the area.

During the core relogging based on the hydrothermal alteration, 7 main lithotypes were identified, not including the regolith or mafic and acid dykes. The lithologies and the respective rock codes used in this relogging, including the regolith, which are shown in Table 7-3. These lithologies were divided in to two groups (with or without gold) and four types of hydrothermal rocks were defined. The hydrothermal rocks are not magnetic.

Figure 7-26 shows some of the barren lithologies of Serrinhas 2 and Figure 7-27 presents mineralized lithologies of the target.

Figure 7-26 - Pictures from Rio Novo Historical Relog Report. A) Medium porphyritic granite with moderate silicification, epidotization, weak chloritization and potassification with no visible sulfidation. Barren interval. (Drill hole FSR-016 – 19.10 m); B) Medium porphyritic monzogranite, magnetic and with incipient chloritization and potassification. Barren interval. (Drill hole MAT-020 – 91.25 m); C) Porphyritic sienogranite with moderate potassic alteration and sericitization, presenting up to 3% of Py. Barren interval. (Drill hole MAT-014 – 75.50 m); D) Porphyritic sienogranite with intense chloritic alteration, weak potassic alteration and up to 10% of Py. Barren interval. (FSR-023 – 155.00 m).

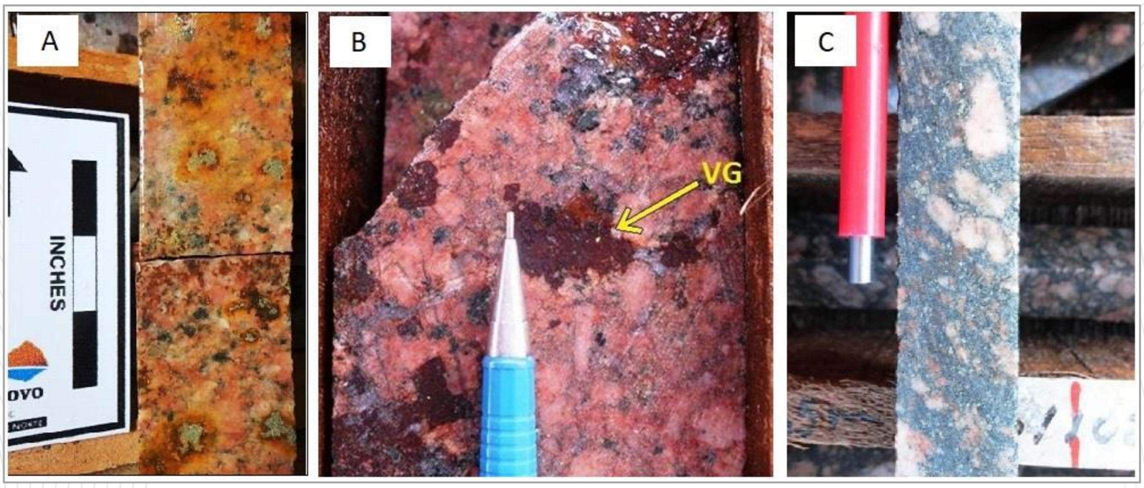


Figure 7-27 - Pictures from Rio Novo Historical Relog Report. A) Granite showing weak to moderate potassic alteration and chloritization and up to 20% of Py. 1.0 m @ 1.0 g/t Au. (Drill hole FSR-023 – 168.40 m); B) Visible gold occurrence inside oxidized Py cluster associated to mm thick quartz veinlet. 0.50 m @ 10.95 g/t Au. (Drill hole MAT-009 – 19.40 m); C) Porphyritic chlorite granite presenting up to 1.5% of disseminated Py. 1.50 m @ 0.92 g/t Au (MAT-007 – 251.50 m).

The hydrothermal alterations observed during this drill core were described as potassification, chloritization, epidotization and minor sericitization, silicification and carbonation. As well as the granitic body the hydrothermalism seen in Serrinhas 2 also presents heterogeneity and at the current moment there was no confirmation of any structural control of the mineralization, thus understood as disseminated style. Figure 7-28 shows the schematic representation of the hydrothermal process in Serrinhas 2 (former MP2).

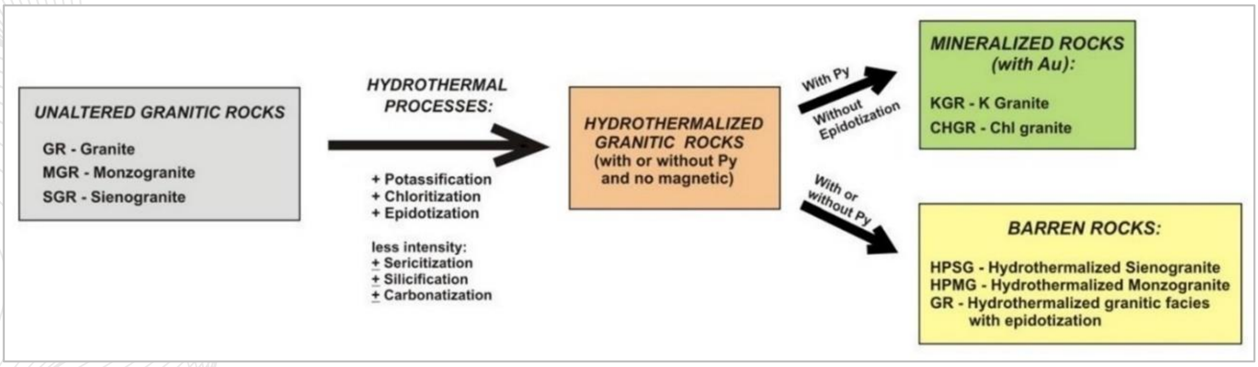


Figure 7-28 - Hydrothermal Alteration Schematic Model for Serrinhas 2 Target.

After completing the historical drilling relog activity at Serrinhas 2, Rio Novo geologists interpreted geological cross sections, as the example presented by Figure 7-28.

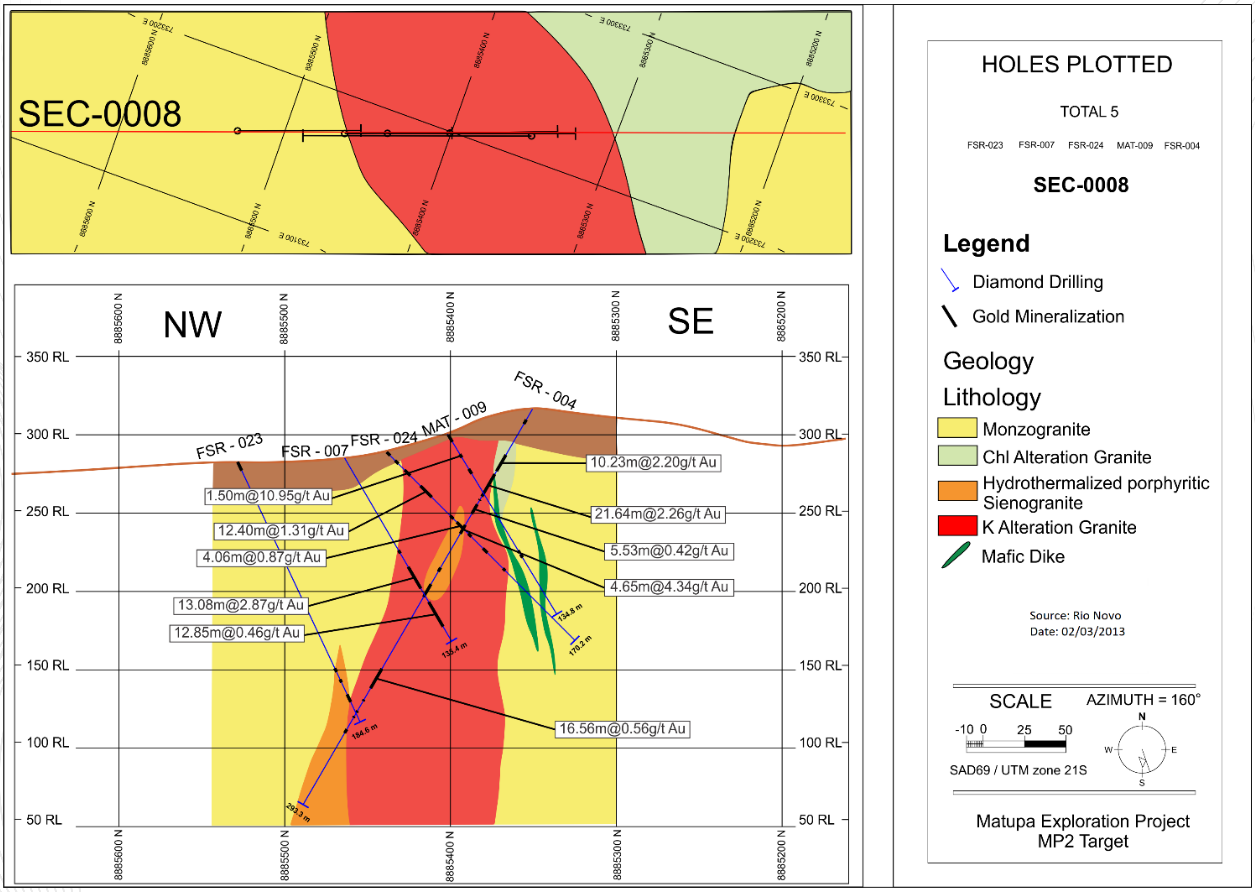


Figure 7-29 - Geological Cross Section of Serrinhas 2 Target (Former MP2).

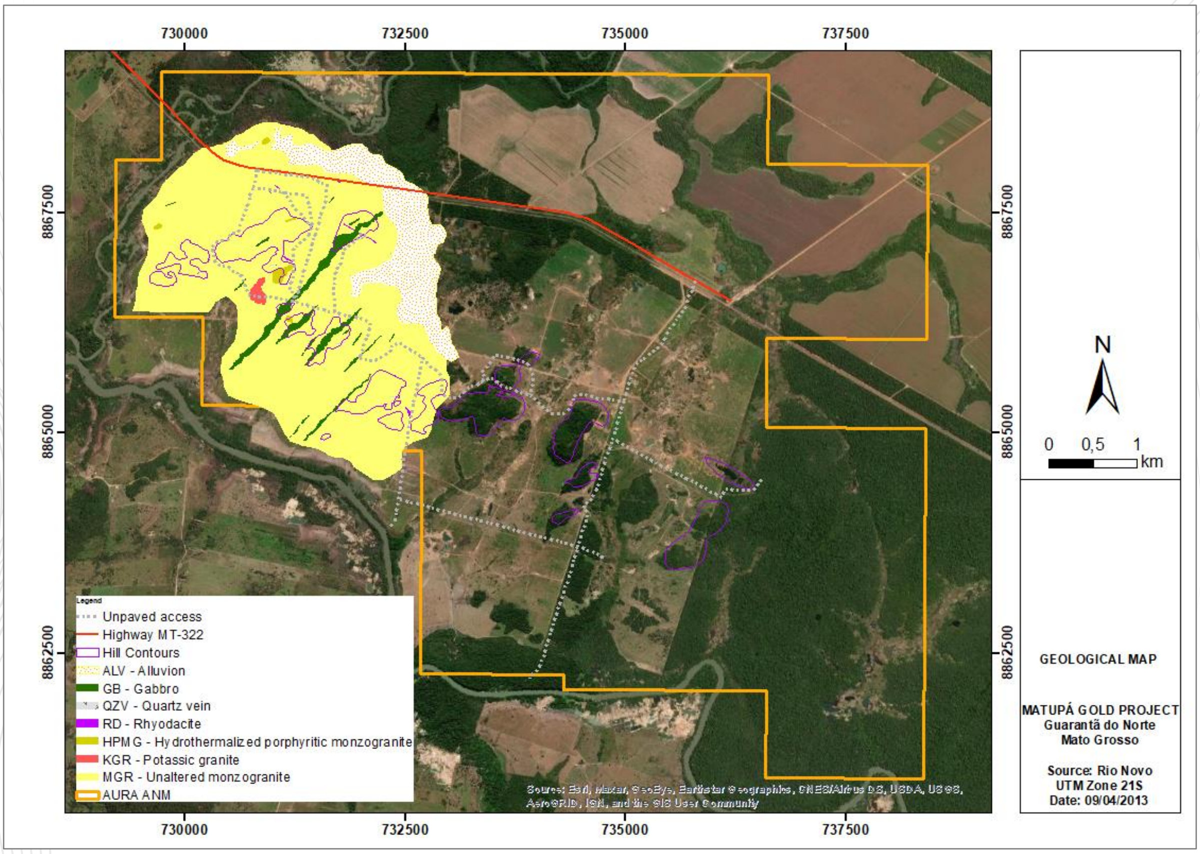
The alterations mentioned and schematically shown in Figure 7-28 are some examples of the hydrothermalism variation, which may represent different stages of the alteration at the Serrinhas 2 target. These alteration zones could have a good relationship with the low magnetic zones seen in the airborne magnetic survey. This low magnetic signature was observed in the core relogging report and a good correlation with the analytical signal from the MAG data (RNM Airborne MAG) was observed. The hydrothermal zones seen at drill core, at the Serrinhas 2 target, are associated with low magnetic zones. Considering this magnetic signature as potential for zones of hydrothermalism and mineralization within the property opens great potential for further exploration on the other anomalous hills, especially due to the gold soil anomalies found in these areas covering more than 10 km along the northwest strike.

Serrinhas lithologies and rock codes from Rio Novo historical drilling relogging are presented in Table 7-2.

After increasing the geological and alteration understanding of Serrinhas 2 area, Rio Novo started its own geological mapping. This was not completed as presented in Figure 7-30.

Table 7-2 - Serrinhas 2 Lithologies and Rock Codes from Rio Novo Relog.

Regolith	Unaltered Rock	Hydrothermalized Rock	
SO - (Soil)	**GR** - (Granite)	**With gold**	**KGR - K alteration granite**
SAP - (Saprolite)	**MGR** - (Monzogranite)		**CHGR - Chl alteration granite**
WR - (Weathered rock)	**SGRP** - Sienogranite (Porphyritic sienogranite)	**Without gold**	**HPSG** - Hydrothermalized porphyritic sienogranite
			HPMG - Hydrothermalized porphyritic monzogranite



Note: Coordinates are South American Datum (1969), UTM Zone 21 South.

Figure 7-30 - Serrinhas Preliminary Geologycal Map.

8. DEPOSIT TYPES

The first study of gold deposits in the Peixoto de Azevedo Region was carried out by Paes de Barros (1994), who directed his research to determine the typology, morphology and origin of the deposits, characterizing them into three main types: Type I: veins filling fractures linked to structures generated in the basement; Type II: veins in faults and fractures of northeast preferential direction, in the contact zones between granitoids and the basement; and Type III: stockworks in granitic dome zones. The methods used in the work were basically field surveys, with detailed mapping of the mining fronts in the mines, which at the time were in full production and had good exposures. In this work, the author associates type I and II deposits with mesothermal deposits (lode gold) and suggests the hypothesis that type III deposits may be of the porphyry type.

The subordinate copper-auriferous deposits of the eastern segment of the Província Aurífera de Alta Floresta (PAAF) are eminently hosted in Paleoproterozoic granitic systems that vary in composition from tonalite-granodiorite to syeno-monzogranite, derived from oxidized magmas, type I, from calcium-alkaline to sub-alkaline, meta-aluminous to peraluminous, medium to high potassium and magnesian to slightly ferrous affinity (Assis et al., 2014).

The gold deposits associated with base metals (Zn + Pb ± Cu), however, are hosted by epiclastic, plutonic and Paleoproterozoic sub-volcanic sediments with a composition ranging from granite alkali-feldspar to granodioritic (Assis, 2011; Assis et al., 2014; Trevisan, 2015), which exhibit affinities with the oxidized alkaline type.

According to the regional knowledge of this portion of the province (Moreton & Martins, 2005; Souza et al., 2005; Silva & Abram, 2008), these deposits would be eminently hosted in the Paleoproterozoic Matupá suites (1,872 ±12 Ma; Moura, 1998) and Colíder (1,786 ±17 Ma to 1,781 ±8 Ma; JICA/MMAJ, 2000; Pimentel, 2001; Souza et al., 2005; Silva & Abram, 2008)

Collectively, Au ± Cu deposits are genetically linked to hydrothermal magmatic systems similar to Au-porphyry type systems, analogous to those found in the Maricunga (Chile) and La Colosa (Colombia) belt, while Au + base metals deposits would be equivalent to intermediate to low sulfidation epithermal systems (Assis et al., 2014).

For purpose of this Matupá feasibility study, only deposit types related to X1 are discussed and explained further.

8.1 X1 METALLOGENETIC MODEL

The intimate spatial and genetic association between granitoids and hydrothermal and mineralized alteration zones of the X1 Gold Deposit suggest that X1 is linked to a magmatic-hydrothermal system promoted by the percolation, in different intensities, of hydrothermal fluids of magmatic origin.

The continental margin arc environment, the types and distribution of the hydrothermal alteration, as well as the paragenetic association of the ore, collectively suggest that the processes involved in the formation of disseminated Au ± Cu (± Bi ± Te ± Mo ± Ag) deposits are similar to Au-(Cu) porphyry-type systems (Corbett & Leach 1998; Seedorff et al. 2005; Sillitoe, 2010). However, the X1 Deposit exhibits some characteristics that differ from those considered classic for the porphyry type Au-Mo-Cu type system, such as: (1) absence of pre-, sin- and post- veining/venules mineralization; (2) low concentration of sulfides in mineralized zones; and (3) aqueous fluids coexisting with fluids rich in CO_2; and (4) deeper crustal positioning. (Assis, 2015).

Experimental studies demonstrate that the solubility of CO_2 in felsic magmas decreases with decreasing pressure and/or increasing temperature, or with increasing magma alkalinity (Lowenstern, 2001).

Magmas that crystallize at greater depths tend to generate fluids more enriched in CO_2, since the solubility of this component increases with the pressure and alkalinity of the magma. Since the granitic hosts of Deposit X1 exhibit geochemical affinity with the calcium-alkaline series of medium to high K (Assis, 2015), it is proposed that crustal level oscillations must have been determinant in the CO_2 degassing process of the magmatic chamber, much more than the chemical affinity of magma, as fluids exsolved deep in the crust exhibit high CO_2/H_2O ratios when compared to those exsolved from systems close to the surface (Lowenstern, 2001).

Therefore, the presence of this volatile would imply that the magmatic-hydrothermal system must have developed at deeper depths than those of the typical Andean porphyries, in this case, greater than 3-4 km, characterizing the formation environment of the X1 Deposit as mesothermal.

This degassing process usually corresponds to an important ore concentration and precipitation mechanism, since Au, as well as a range of metals of a chalcophilic nature (e.g., Cu, Mo) tend to partition into the rich vapor phase, while the Cl and several siderophilic elements (e.g., Fe, Pb and Zn) exhibit a strong tendency to fractionate to the salt phase and therefore are rich in water (Heinrich et al., 1999). In this sense, the degassing process proposed for the X1 Deposit would have been postponed by a progressive mixing of aquo-carbonic fluids (magmatic) with external aqueous fluids (meteoric?) (Figure 8-1). Pulses later exsolved from magmatic crystallization and, therefore, enriched in a phase of high salinity, would have ascended to shallower crustal levels and interacted with a larger external (meteoric?) colder, oxidizing and low salinity component. However, the mixture between magmatic fluids and external fluids would have caused the progressive dilution and lowering of the temperature of the system, which would have culminated in the reduction of the solubility of Au and other metals, with their consequent precipitation.

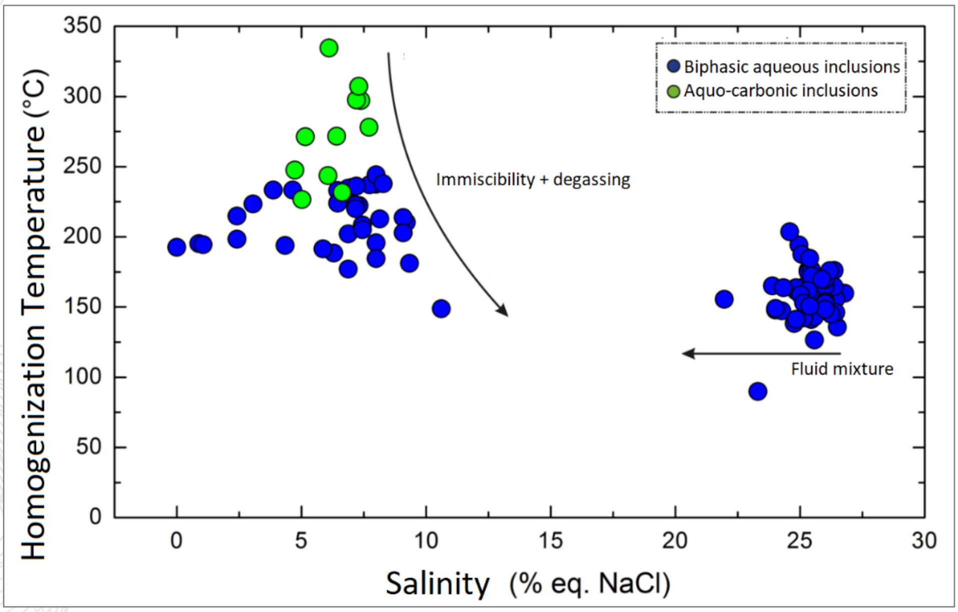


Figure 8-1 - Salinity Graph E. Total Homogenization Temperature for the Groups of Fluid Inclusions Described for Deposit X1 (Modified from Rodrigues, 2012; Trevisan, 2012b).

9. EXPLORATION

The gold exploration in Alta Floresta Gold Province (AFGP) started in the 1980s with the discovery of alluvium gold occurrences during the opening of Cuiabá-Santarém federal Highway BR-163, followed by an intense gold rush of garimpo activity. This gold rush reached its peak around 1989 and started to decline in 1999.

From 1980 to 1999 the production of gold within the limits of the gold province of Alta Floresta was estimated to have been on the order of 160 tonnes. Despite the relative exhaustion of the alluvial and the supergene enrichment deposits, this region still holds significant exploration potential for primary gold deposits.

The initial exploration works were first carried out in 1996 by Mineração Bom Futuro in partnership with WMC (Western Mining), later followed by RTZ (Rio Tinto) in 2000, resulting in the discovery of the Serrinhas de Matupá target, currently known as the Serrinhas Target. Among the historical exploration activities was geological mapping, geochemical sampling of rock and soil, ground geophysical surveys (Gamma spectrometry and Gradient IP) followed by auger drilling, and reverse circulation and diamond drilling campaigns.

During the opening of Cuiabá-Santarém federal highway BR-163, mineralized road-cut outcrops caught the attention of Vale geologists' who identified some manganese-rich structures associated with east-west normal faults. Samples taken assay values ranged from 0.2 g/t to 24.09 g/t Au. These results were then followed by detailed geological mapping at 1: 5,000 scale, improved to 1:1,000. Later ground and airborne geophysical surveys were carried out, followed by initial diamond drilling campaigns, resulting in the discovery of the Guarantã Ridge Target and X1 Deposits in 2002 and 2003, respectively.

The geological, geochemical, and geophysical surveys conducted by Vale have been passed on to Mineração Santa Elina (MSE) in 2006 and then to Rio Novo (RNM) in 2010. The data were collected in a professional and meticulous manner such that the quality is valid for continued use. Rio Novo typically conducted verification surveys on the geochemical data, and often completed infill geochemical surveys to improve on the data.

Rio Novo continued to conduct geological, geochemical, and geophysical surveys during exploration of areas adjacent to the known targets. These surveys led to the extension of the Guarantã Ridge Target and to an infill drilling program at the X1 Deposit.

In December 2017, Aura Minerals and Rio Novo announced a merger transaction under the BVI Business Companies Act, 2004. The merger was completed in March 2018, under which Aura has merged with Rio Novo and the separate corporate existence of Rio Novo ceased. Therefore, the exploration activities for the Matupá Project were resumed by Aura in 2019, starting with surface geochemistry sampling of rock and soil, generating early-stage targets, and reprocessing geophysical data from the Rio Novo survey (magnetic data from airborne VTEM) and from the Brazilian geological service (CPRM).

Among the exploration activities carried out by Aura, a geological review of the historical drilling of the X1 Deposit was realized to improve the lithological and structural control aiming to test the limits of the ore body to the West, which Rio Novo had interpreted to be caused by a fault system.

In 2020, after reviewing the geophysical magnetic signature, shape of the ore body and the grade distribution inside the X1 Deposit model, it was observed that, the general east-west orientation of the ore body, the high grades present, a preferential plunge slightly oriented to the northeast/southwest and dipping to the southwest with low angle before an abrupt limit.

Two drill holes were drilled by Aura, FX1D-0062 and FX1D-0063 (naming the drill holes after the Rio Novo numerical sequence), the first hole was aiming to check the same magnetic signature as the historical positive drill holes at X1 and the second hole was aiming to test the western limit of the ore body.

FX1D-0062 has confirmed the same halo of molybdenite, typically seen in the upper parts of the main mineralized zone in X1, but the hole was not drilled with the original programmed azimuth and dip due to vegetation limitations. Drill hole FX1D-0063 has confirmed the fault zone causing the west limit of the X1 ore body and confirmed the continuity of the quartz feldspar porphyry below the fault, which is known to be the host of the mineralization. More exploration activities will be done to test the continuity of X1 mineralization and also check untested magnetic and chargeability anomalies in the property.

Regarding the regional exploration conducted by Aura at the Matupá Project, surface activities such as soil and rock sampling with geological reconnaissance has been carried out since 2019. As result, a significant amount of gold occurrences and anomalies were identified within a 50 km radius from the X1 Deposit inside Aura's mineral rights. Part of these anomalous areas were then tested with initial scout diamond core drilling or shallow RC drilling campaigns, which confirmed many positive results generating early-stage drilling targets.

In the Serrinhas Target, the exploration activities began by reviewing the drilling assay database and historical drilling relog activities carried out by Rio Novo geologist' team and programing initial extension drilling at MP2 Central Zone and MP2 East Zone ore bodies.

10. DRILLING

Drilling on the Matupá Gold Project has been completed in various campaigns since 1996 by WMC, Rio Tinto (RTZ), Crescent Resources (CRESCENT), Vale, Mineração Santa Elina (MSE), and Rio Novo. The implemented drilling methods were diamond core, reverse circulation (RC), and auger drilling. For the purposes of previous studies, Rio Novo decided not to use the reverse circulation drill hole information for the geological models and Mineral Resource Estimates, now historical estimates, for any deposits. This was done to assure that the quality of assay results and other drill hole information met Rio Novo's quality standards. The current study also follows the same logic regarding drilling and has not used RC drilling in modelling, estimation and classification. Reverse circulation and auger drilling were historically used only on Serrinhas Target, mainly to test soil anomalies.

Since 2019, Aura has started exploration activities with the use of RC drilling and diamond core drilling.

10.1 DRILLING METHODS

Diamond drill core used in this study was a combination of HQ size (63.5 mm diameter) and NQ size (47.6 mm diameter). Drilling employed standard wireline methods, and generally used split core tubes. Drilling angles were in the range of 45 to 90 degrees to intersect the structure and ore zones as near perpendicular as possible. Rio Novo completed down hole surveys on all the core holes using Maxibor instrumentation, a standard international tool. Down hole surveys completed by Vale and MSE were also available for their core drilling programs. WMC carried out down hole surveys only in the first 9 of 25 drill holes, with poor results, no survey was carried out on the other 16 drill holes, and RTZ did not perform any down hole survey.

Diamond core drilling for Rio Novo was performed entirely by Geosol Drilling S.A., a world-wide drilling company with a large base of operations in Brazil. Rio Novo provided drilling plans to Geosol, surveyed drill sites, confirmed drill set-up (location, bearing, and angle), and assured overall quality of the drilling process. Geosol provided drilling crews and equipment, performed the actual drilling operations, and delivered the core samples to Rio Novo. There is no relation between Rio Novo and Geosol, other than a business relationship.

Historical diamond core drilling for Vale was done by Geosol Drilling S.A. and oriented by geologists' team of GEOEXPLORE – Consultoria e Serviços Ltda. Historical diamond core drilling for MSE was done by Mineração Mariana Ltda. and oriented by the MSE geologists' team.

Historical diamond core drilling for RTZ was done by Boart Longyear and oriented by the RTZ geologists' team, but there is no record for which company executed the drilling program. The same is for CRESCENT drilling, which was also done by Boart Longyear and oriented by CRESCENT geologist's team.

The auger drilling sampling done in Serrinhas Target was done meter by meter; each advanced meter is homogenized and collected. Half of each sample is sent to the laboratory; the other half was stored at the Serrinhas farm but is no longer available.

The RC drilling sampling is done meter by meter; each advanced meter is homogenized and collected. Half of each sample is sent to the laboratory, the other half was stored at the Serrinhas farm but is no longer available.

There is no registration of the companies who did the reversed circulation and auger drilling for WMC and RTZ in the Serrinhas Target.

The first diamond drilling campaign performed by Aura in 2019 and early 2020 was drilled by Mata Nativa, oriented by Aura geologists' team. From late 2020 and current diamond core drilling was done by Willemita Sondagens. RC Drilling campaigns in 2020 and 2021 was performed by Raio X Serviços Minerais. All diamond core drill holes have down hole surveys.

10.2 DRILLING EXTENT AND SPACING

Drilling discussed in this study covers X1 Deposit and Guarantã Ridge and Serrinhas Targets. The total drilling database includes 305 core holes, 87 reverse circulation holes, and 304 auger holes drilled between 1996 to the present, the effective date of this technical report. A total of 54,473.02 m comprises the current core drilling database.

At X1, the known extents of mineralization have been first drilled on nominal 50-m center spacing which was later followed by Rio Novo's infill drilling program reducing the spacing in some parts of the deposit to approximately 25 m centers. Drilling covers an area of about 500 m along strike and 350 m across strike. Additional scout holes have been drilled around the perimeter. The X1 Deposit is primarily drilled out to a vertical depth of 250 m to 300 m, although individual drill holes are as deep as 500 m (vertical depth). Most holes were oriented north to south and vice versa, or east to west and vice versa. Angle orientations ranged from 45 to 90 degrees, thus cutting the main structural and mineralization trends as near perpendicular as possible.

To this date, Aura has drilled 7 diamond core holes in the X1 Deposit, from which 5 holes were drilled for geotechnical purposes and 2 holes drilled as part of an extension program. The geotechnical holes were sampled specifically for geotechnical studies, thus without assaying, and the extension drill holes drilled so far have not returned positive results outside of the existing model. Therefore these 2 extension holes was not used for the geological model and Mineral Resource Estimate of X1 Deposit. Additionally, Aura has drilled 43 RC shallow drill holes in the X1 property, of which 22 were condemnation holes. The other 21 RC drill holes had exploration purposes and aimed to test either soil anomalies or geophysical magnetic anomalies, which were considered not mineralized. All RC drill holes were drilled outside the main area of the X1 Deposit and did not affect the current geological modeling and Mineral Resource estimation presented in this report. Figure 10-1 presents all drilling done so far by Aura in the X1 property.

In total, there have been 148 diamond core holes drilled in the X1 area, totaling 30,184.66 m. Table 10-1 summarizes the X1 drilling.

Table 10-1 - Total Historical Drilling in the X1 Area.

Type of Drilling	Company	Period/Year	No. of Holes	Amount (m)	Average Depth (m)	No. of Samples	Drill Hole Series
Diamond Drilling	Vale	1999-2004	18	3,190.05	177.23	3,139	FD-029 to FD-046
	MSE	2006 – 2010	63	14,106.34	223.91	8,158	SEX1-01 to SEX1-063
	RNM	2010 – 2018	60	11,469.66	191.16	10,318	FX1D-0001 to FX1D-0061
	Aura	2019 to date	7	1,418.61	202.66	697	
		Subtotal		**148**	**30,184.66**	**198.74**	**22,312**
RC Drilling	Aura	2019 to date	43	2,242	52.14	2,242	FX1R-0001 to FX1R-0043
	Subtotal			**43**	**2,242**	**52.14**	**2,242**

At Guarantã Ridge, the first drilling was done by Vale with 200 m spacing between drill holes. After Mineração Santa Elina's (MSE) drilling and especially Rio Novo's drilling, most of the target had been drilled out at on nominal 100 m x 50 m centers. The drilling covers an area of about 3,500 m along strike and 350 m across strike. Holes are oriented along a north to south and vice versa, with drilling angles of 45 to 65 degrees. The target is drilled to a vertical depth of about 150 m, with an average down hole drilling length of 140 m, the deepest holes reaching vertical depths of 150 m to 180 m.

Aura has carried a short exploration drilling program at Guarantã Ridge comprising 4 drill holes. A total of 88 drill core holes, for 13,504.69 m, were drilled between 2002 to the effective date of this report. Table 10-2 summaries the drilling on Guarantã Ridge.

Table 10-2 -Total Historical Drilling in the Guarantã Ridge Area.

Type	Company	Period/Year	No. of Holes	Amount (m)	Average Depth (m)	No. of Samples	Drill Hole Series
Diamond Drilling	VALE	1999-2004	28	3,036.37	108.44	2,903	FD-001 to FD-028
	MSE	2006 - 2010	6	708,1	118.01	677	GD-01 to GD-06
	RNM	2010 - 2018	50	8,062.4	161.25	8,151	FGRD-0001 to FGRD-0049
	Aura	2019 to date	04	1,697.82	424.46	1,440	FGRD-0050 – FGRD-0053
	Subtotal		**88**	**13,504.69**	**203.04**	**13,171**	

During 1996 and 2006, three drilling campaigns were performed in the Serrinhas area. In total, 62 diamond core holes were completed, totaling 8,930.84 m. The main drilling was oriented at 160 or 340 degrees, perpendicular to the overall strike of the targets. Holes were angled at -45 to -90 degrees dip. The target has been drilled to a vertical depth of 200 m to 250 m. Drill hole spacing in the Mineral Resource area is nominally 50 m x 25 m in the case of the main drilled region. Table 10-3 details the Serrinhas historical drilling summary.

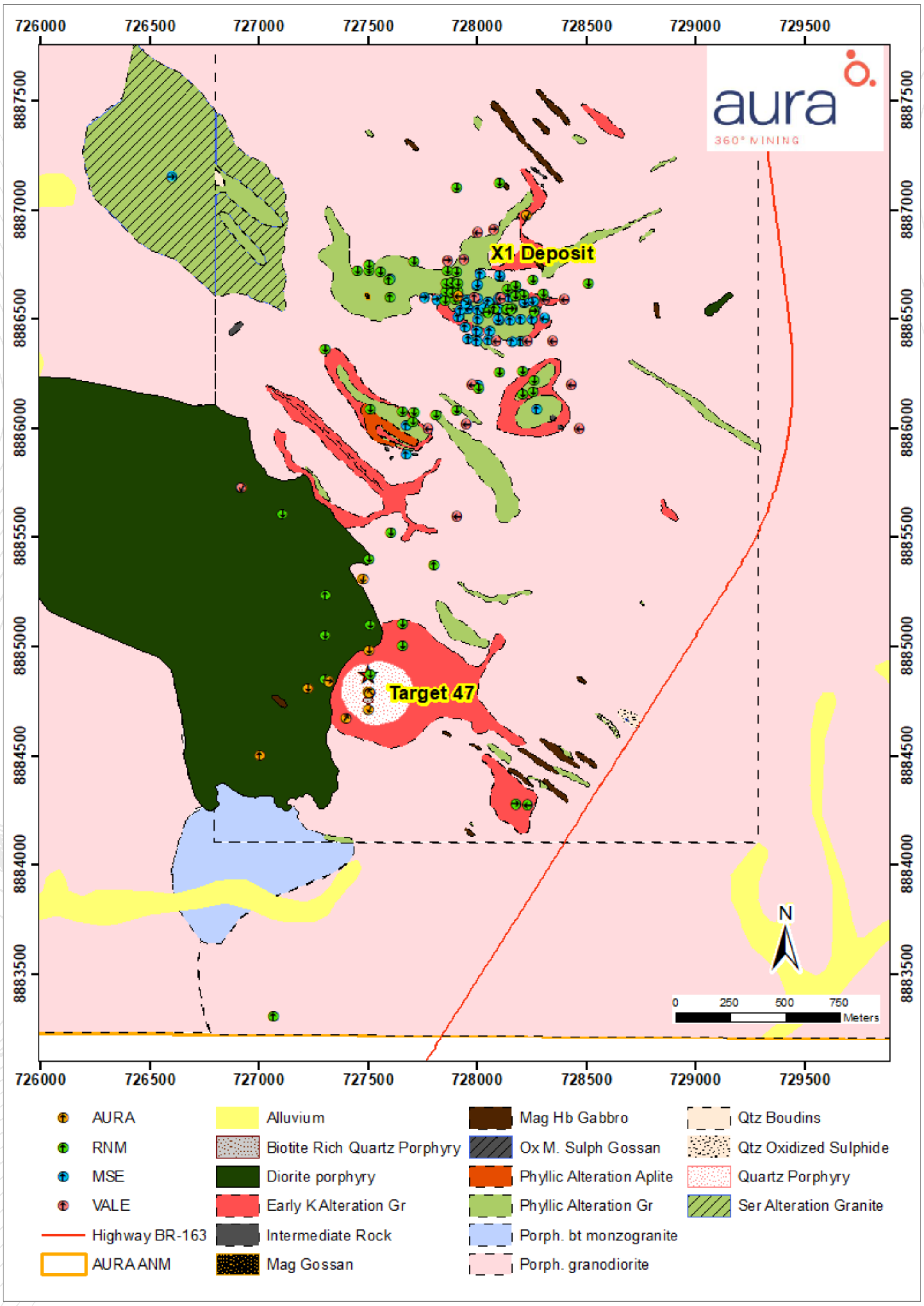
Aura has started diamond drilling in the Serrinhas area in August 2021. To the effective date of this report, 54 diamond drill holes have been completed, and sampling is ongoing.

Table 10-3 -Total Historical Drilling in the Serrinhas Area.

Type	Company	Period/Year	No. of Holes	Amount (m)	Average Depth (m)	No. of Samples	Drill Hole Series
Diamond Drilling	WMC	1996-2000	7	395,28	56,47	366	SRD-01 to SRD-07
Diamond Drilling RC Drilling	RTZ	2000-2001	30	4330,56	144,35	3618	FSR-001 to FSR-030
	CRESCENT	2002-2006	25	4205	168,2	2087	MAT-001 to MAT-025
	Subtotal		**62**	**8930,84**	**123,01**	**6071**	
	WMC	1999-2000	3	180	60	146	SRC-01 to SRC-03
RC Drilling Auger Drill	**Subtotal**		**3**	**180**	**60**	**146**	
	RTZ	2000-2001	127	762	6	759	FTS-001 to FTS-127
Auger Drill	WMC	1999-2000	177	1135,2	6,41	1232	SRA-001 to SRA-177
	Subtotal		**304**	**1897,2**	**6,21**	**1991**	

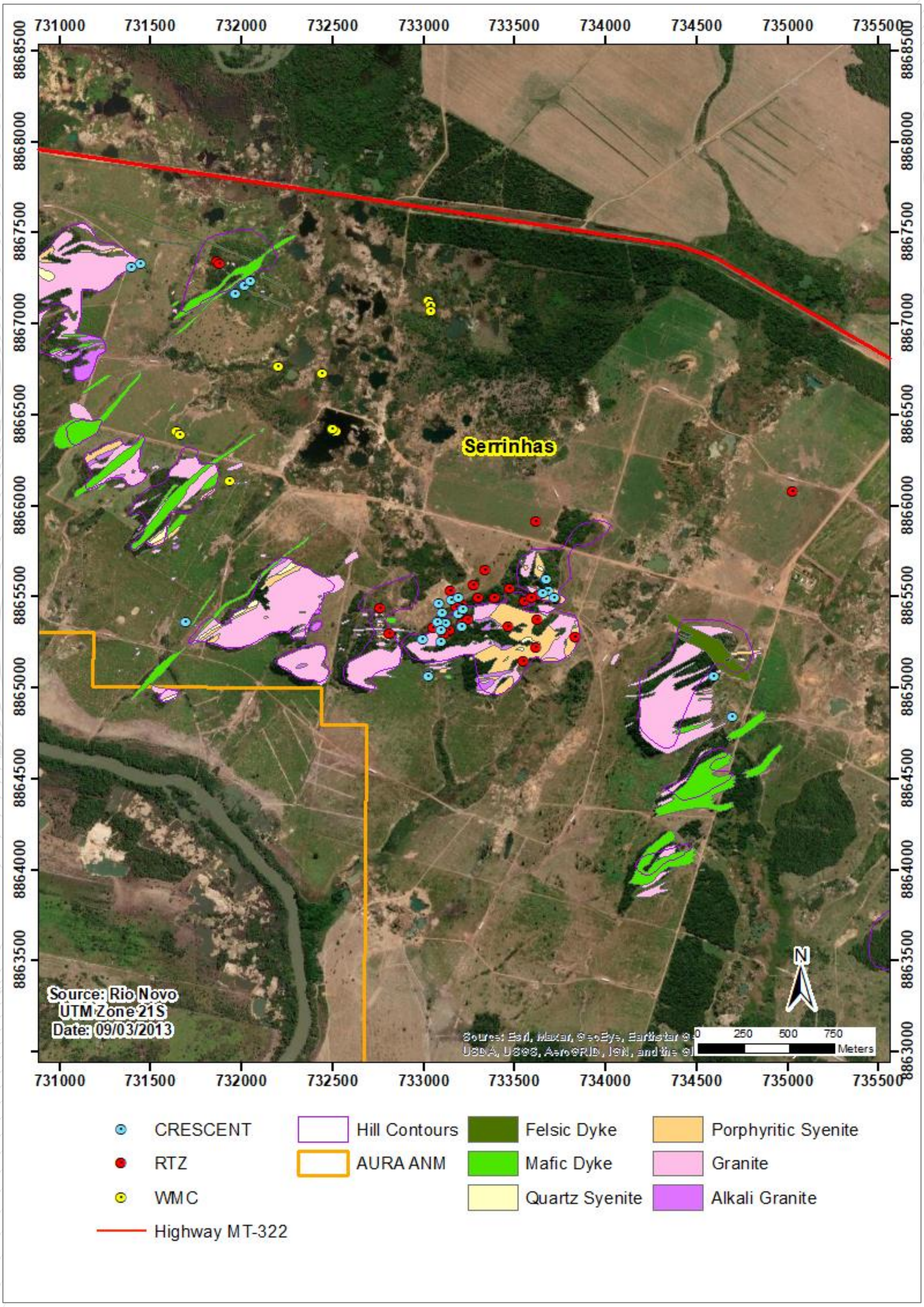
10.3 DRILL HOLE LOCATIONS

The location of the historical drill holes and Aura drill hole locations for X1 is shown in Figure 10-1. Figure 10-2 shows the historical drill hole locations for Serrinhas and Aura's current drilling campaign on this property is shown by Figure 10-3.



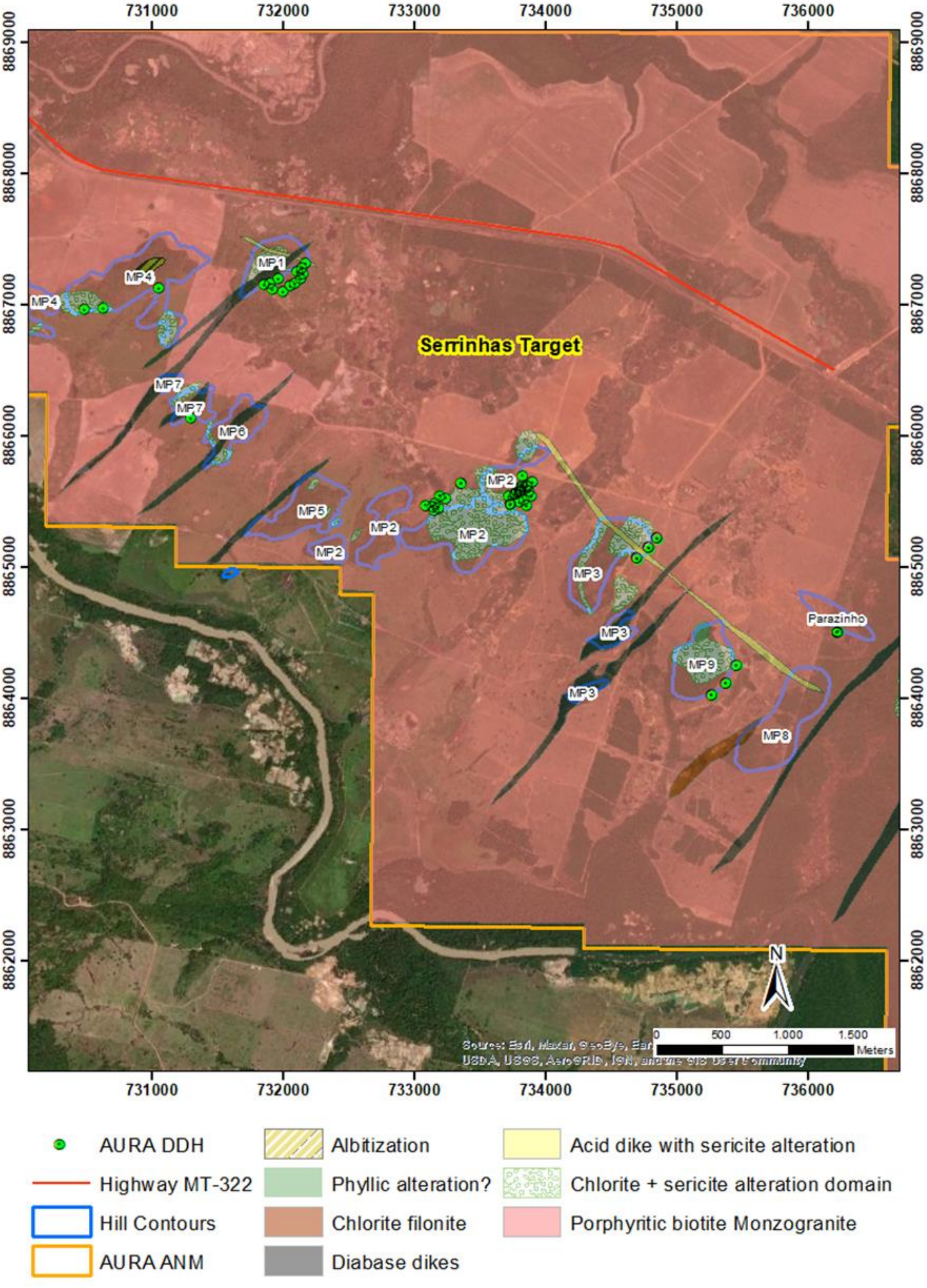
Note: Coordinates are South American Datum (1969), UTM Zone 21 South.

Figure 10-1 - X1 Deposit Diamond Drilling Location.



Note: Coordinates are South American Datum (1969), UTM Zone 21 South.

Figure 10-2 - Serrinhas Historical Diamond Drilling Location.



Note: Coordinates are South American Datum (1969), UTM Zone 21 South.

Figure 10-3 - Serrinhas Target Aura Diamond Drilling Location.

10.4 DRILLING QUALITY

The X1 Deposit has been drilled entirely by diamond drill core for the purposes of the Mineral Resource models used herein. The core drilling was completed by professional operators, using state of the art techniques and equipment. Core quality in the mineralized zones has been excellent, due both to the drilling procedures and the rock characteristics. In general, core recoveries have exceeded 95 percent in the mineralized areas.

Approximately 40 percent of the core drilled at the X1 Deposit and 68 percent of the core drilled at Guarantã Ridge were drilled by Rio Novo. Rio Novo maintained a good quality of procedure and met industry standards, standard operating procedures (SOP), for core drilling. The remainder of the core drilling by previous operators in Guarantã Ridge and X1 Deposit was reviewed in detail by Rio Novo and found to be of high quality.

The core drilling by WMC, RTZ and CRESCENT in Serrinhas has poor to no down hole survey information. Rio Novo has re-assayed 9.77% of the historical drilling at Serrinhas. Aura has now increased the re-assaying to 19.32%, is resurveying the available historical collars and will complete a twin drilling campaign to increase the QA/QC studies and data verification to match industry standards.

Aura has resurveyed historical drill hole collars on the X1 Deposit (before RNM) with RTK GPS and conducted a re-sampling of selected intervals from previous drilling campaigns in December 2021-January 2022. For this purpose, ¼ core were sampled and sent to SGS laboratory (SGS lab in Vespasiano-MG, Brazil) to further validate underlying data and investigate any bias in the ALS and SGS laboratories results during the Mineração Santa Elina and Rio Novo Mineração campaigns.

11. SAMPLE PREPARATION, ANALYSIS AND SECURITY

11.1 INTRODUCTION

The main part of this section discusses Rio Novo's sample preparation and analysis during the 2010-2012 drill campaigns. The sample preparation, analysis and security practices of previous operators prior to Rio Novo is also briefly discussed for historical reference of the Project. The QP has also reviewed Rio Novo's monthly quality assurance/quality control (QA/QC) reports. In the opinion of the QP, the Rio Novo QA/QC drill campaign met required best practice guidelines and industry standards.

11.2 CORE HANDLING, LOGGING, AND SAMPLING PROTOCOLS

The first diamond core drilling performed at the Serrinhas Target was conducted in 1996 by WMC. The core sampling took place at regular intervals of 1 meter along the hole. The sampling had ½ of their linear volume sampled and the rest of the sample remained in the core box. At the end of this survey, 366 samples distributed along 395.28 m were collected from this first 7 drill holes. There is no record of where the samples were sawed and no information on the diamond core average recovery, but cores were logged by WMC geologists. The core boxes were stored in the farm area and recovered by Rio Novo in 2012 to their core shack. WMC also carried a small campaign of reverse circulation drilling, completing 3 RC drill holes of 60 m each. The sampling took place at regular intervals of 1 m, but the first meter of each drill hole were not sampled, probably due to garimpo material. After completing this RC drilling program 146 samples distributed along 180 m were collected from these 3 drill holes. In addition, WMC also carried out an auger drilling campaign, which comprised 177 shallow holes of 6.4 m depth on average, and the deepest holes were 10 m deep. The sampling was carried at regular intervals of 1 m when possible. After completing the survey 1,991 samples distributed along 1,132.20 m were collected from the 177 auger drill holes. There is no record of which contracted company performed any of the drilling campaigns or on their sampling procedures. Logging information is only available for diamond core drilling.

The second drilling campaign in the Matupá Project was also on the Serrinhas Target, this time by RTZ under contract with Boart Longyear Ltd. in 2000. The core sampling ranged from regular intervals of 1 meter along the hole but some of them were partially sampled at regular intervals of 1.5 meters and in specific cases the samples reached 2.15 m. Both HQ and NQ diameters had ½ of their linear volume per meter sampled, but at some point, the core sampling was carried out in a selective manner, sampling only the most likely potential mineralized zones. At the end of the survey, 3,613 samples distributed along the 4,330.18 m were collected from the 30 drill holes. RTZ left the core boxes stored on the farm in poor conditions. Rio Novo has carried out a recovery activity of these historical drilling core boxes in 2012, exchanging the old core boxes to new ones and organizing them in their core-shack to be logged and some of them reanalyzed. Despite all efforts, some intervals were lost over time due to the mixing of the core. RTZ also carried out an auger drilling campaign, which comprised 127 shallow holes of 6 m depth in an average and a maximum depth of 12 m. The sampling was carried at regular intervals of 1 m. At the end of the survey, 760 samples distributed along 762 m were collected from the 127 auger drill holes. There is no record of which contracted company performed this auger drilling campaign or on the sampling procedures, but diamond core samples and auger samples were logged by RTZ geologists.

During the period 2003 to 2004, the first diamond core drilling at the X1 Deposit was conducted by Vale under contract with SGS Geosol Drilling Ltda. (Geosol). The core sampling system took place at regular intervals of 1 meter along the hole. The HQ diameter ranges had ¼ of their linear volume per meter sampled, while for the NQ diameter ranges ½ of the linear volume per meter was sampled. At the end of each hole, measurements greater than 0.5 meters were counted as a new sample. At the end of the survey, 3,188 samples distributed along the 3,190.05 m were collected from the first 18 drill holes. The recovery of the survey was always above 90%. There is no record on where the samples were sawed, but the core were logged by GEOEXPLORE – Consultoria e Serviços Ltda. Geologists.

The next drilling campaign at the Project was performed on the Serrinhas Target in 2006 by CRESCENT under contract with Boart Longyear Ltd., representing the last drilling at the area until the present. The core sampling system took place at regular intervals of 1.5 m or 2 m. Both HQ and NQ diameters had ½ of their linear volume per meter sampled, but at some point, the core sampling was carried in a selective manner, sampling only the most likely potentially mineralized zones. After the end of the survey 1,990 samples distributed along 4,294.85 m were collected from the 25 diamond drill holes.

MSE continued the diamond drilling program at the Project in 2008 under contract with Mineração Mariana Ltda. On arrival at the core processing facility, the core was laid out, washed, and photographed. The core was then logged, and sample intervals were marked by MSE geologists. Sample intervals started at regular intervals of 1 m and became 2 m or bigger at some drill holes or intervals. Both HQ and NQ diameters had ½ of their linear volume per meter sampled. After the end of the survey, 10,246 samples distributed along 17,197.04 m were collected from the 85 diamond drill holes, from which only 63 were drilled in X1 Deposit, the other ones were drilled on different regional targets, some of them are not part of the current Property of the Project.

During the period 2010 to 2012, diamond core drilling was conducted for Rio Novo under contract with SGS Geosol Drilling Ltda. (Geosol). Geosol drilling crews extracted the core, placed it in wooden core boxes, then sealed the boxes with tape or straps prior to transport. The core was then transported by truck to Rio Novo's core-shack processing facility at Guarantã do Norte city.

On arrival at the core processing facility, the core was laid out, washed, and photographed. The core was then logged, and sample intervals were marked by Rio Novo geologists. Sample intervals were generally one meter; however, variations were allowed for special samples or special interval breaks. The maximum sample interval was 1.5 m and the minimum was 0.5 m. Core logging included lithology, alteration, mineral zone, structural and geotechnical logging. Structural and geotechnical details that were noted included foliation, fractures, vein orientation, and faults. Percent core recovery and RQD measurements were taken and calculated for all drill intervals.

The core was then cut under Rio Novo supervision by Geosol personnel. There was no relationship between SGS Geosol and Rio Novo Mineração Ltda. except for a strictly contractual one for the provision of drilling and analytical services for the Company's exploration programs. Core was cut using diamond impregnated cutting saws, standard to the industry. To the extent possible, the core was cut perpendicular to a major vein or structure's orientations. Rio Novo's technical team then bagged the samples in plastic bags.

Samples were tagged with electronic bar codes, with one tag inside the bag and one tag outside. Sample bags were also marked by hand in permanent ink. The sample numbers were electronically entered into the database, according to the proper sample intervals. This system then provided an electronic sample submittal form. Core handling, logging, and sampling procedures practiced by Rio Novo and its contractors in the Matupá Gold Project are summarized in the flowsheet in Figure 11-1 and its practice presented in Figure 11-2.

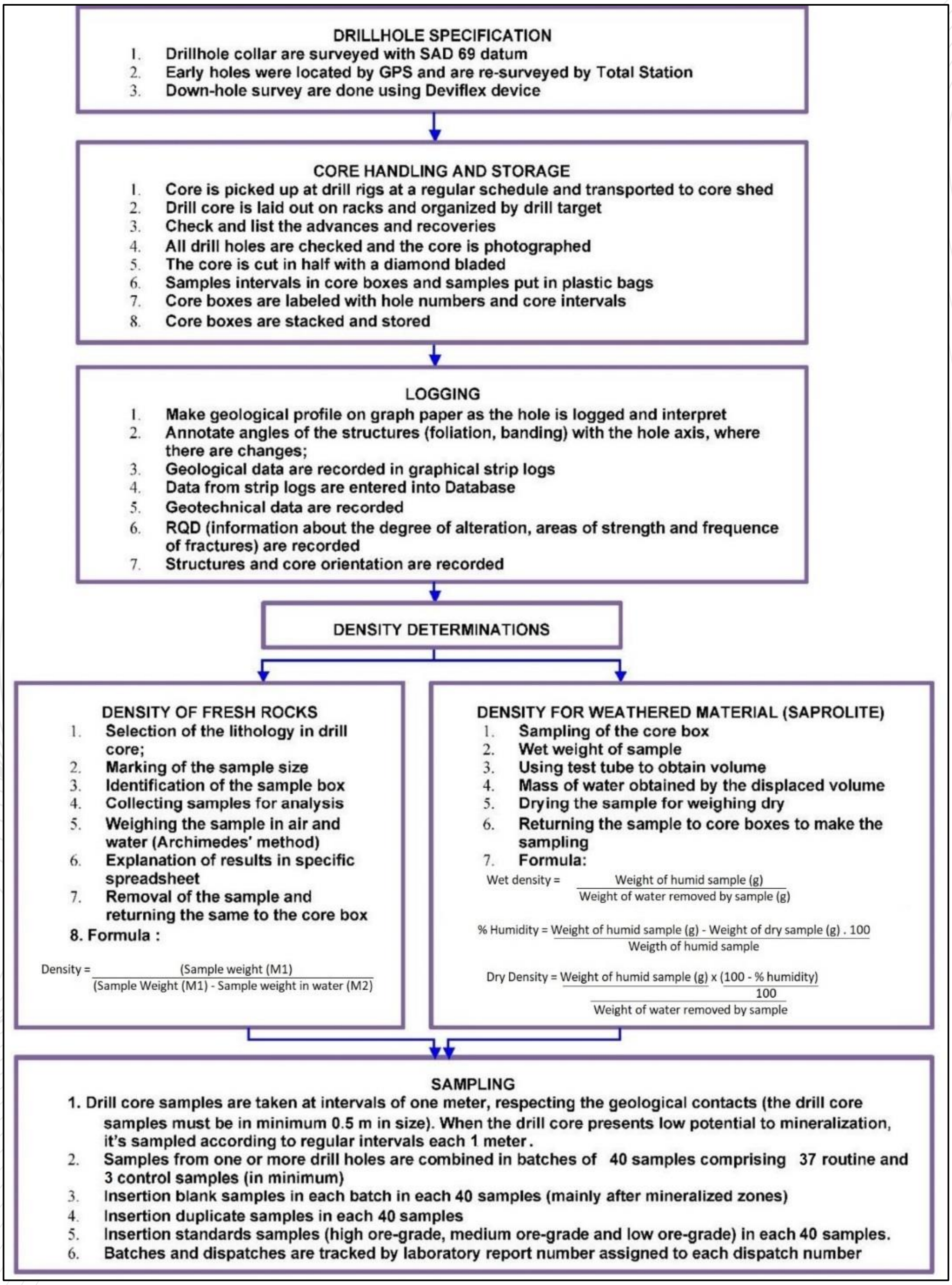


Figure 11-1 - Core Handling and Sampling Protocols (Rio Novo 2010-2012).

A) Core cutting facility in Rio Novo core-shack; B) Diamond drill core sampling at core-shack; C) Insertion of control samples; D) Samples inserted in tagged plastic bags.

Figure 11-2 - Rio Novo Core Handlinga and Sampling Procedures.

11.3 DENSITY MEASUREMENTS

Bulk densities of geological materials encountered in drill core are required to determine the mass for Mineral Resource estimation. Density data must be representative of the lithologies found in the deposit and determined on replicate samples.

Rio Novo used different methodologies to determine bulk densities depending on if the sample was fresh rock, weathered rock or saprolite. For fresh rock samples, the classic Archimedean method was used. The method is based on weighing the sample in air and water. Based on the wet sample weight and sample weight in water, the density of the material is calculated through a mathematical equation. Quality control on density determinations is maintained by insertion of a standard sample of known density for every 20 samples measured. This assures the accuracy of the operational procedures and equipment. Table 11-1 gives the average bulk density of un-weathered lithologies encountered in core holes from the two targets in the Matupá Gold Project area. Table 11-2 summarizes the average bulk densities of saprolite and weathered rocks from the same targets.

Saprolite is a soft, clay-rich material formed by the deep weathering of bedrock in tropical zones. In the calculation of bulk density for saprolite and for oxidized, weathered rocks, the sample is collected and preserved in a plastic envelope. The sample was weighed to determine the wet weight on an analytical balance. Then a plastic bucket was filled with water, a sample was put in the bucket until the water flows out. When the water stops flowing from the bucket, the sample was carefully removed. The water displaced by the mass of the sample was collected and weighed, thus obtaining the volume displaced. The sample was then dried in an oven and weighed again. The mass of the moist sample, the volume (given by the mass of water removed from the sample), and the dry weight was recorded and then used to calculate the bulk density by formula.

In 2012, Rio Novo carried a density sampling on the fresh rocks of historical diamond drill cores from RTZ and WMC drilling campaigns at the Serrinhas Target, using the same fresh rock density measurement procedure as used for its own drill cores.

Historical drilling density information is available only from MSE drilling at the X1 Deposit. According to the available data, the MSE density sampling procedure also considered weathered and fresh rocks, but sampling was not carried out at regular intervals and does not consider all the drill holes. There is no record of their historical density measurements procedure.

Table 11-1 - Bulk Density of Fresh Lithologies from Core Samples, Matupá Project.

Target	ROCK CODE	LEGEND	COUNT	MINIMUM G/Cc	MAXIMUM G/Cc	MEAN DENSITY G/Cc
Guarantã Ridge	BR	Breccia	1	2.84	2.84	2.84
Guarantã Ridge	BSR	Basic Rock	2	2.90	2.92	2.91
Guarantã Ridge	CBTF	Carbonate Tuff	4	2.76	2.87	2.82
Guarantã Ridge	FTZ	Fault zone	6	2.40	2.80	2.58
Guarantã Ridge	HBR	Hydrothermal Breccia	93	2.04	3.75	2.86
Guarantã Ridge	IVR	Intermediate Volcanic Rock	110	2.16	3.31	2.81
Guarantã Ridge	PYTF	Disseminated Pyrite Tuff	244	1.85	3.26	2.67
Guarantã Ridge	RH	Riolite	40	2.61	2.80	2.68
Guarantã Ridge	TBR	Tectonic Breccia	6	2.19	3.12	2.70
Guarantã Ridge	TF	Volcanic Acid Tuff	9	2.61	2.88	2.72
X1	AP	Aplite	2	2.62	2.69	2.66
X1	BMGR	Biotite Monzogranite	149	2.26	2.80	2.64
X1	BMGREK	Biotite Monzogranite Early Potassic Altered	6	2.64	2.65	2.64
X1	BR	Breccia	1	3.02	3.02	3.02
X1	EKGR	Potassic Alteration Granite	454	1.70	9.58	2.67
X1	EKGRPH	Potassic Alteration Granite + Phyllic overprint	13	2.57	4.90	2.89
X1	QFP	Potassic alteration Quartz Feldspar Porphyry	9	2.60	2.73	2.65
X1	FTZ	Fault zone	1	2.68	2.68	2.68
X1	GR	Granite	2	2.73	2.73	2.73
X1	DR	Quartz Diorite	60	2.66	2.99	2.82
X1	IBR	Igneous Breccia	2	2.70	2.74	2.72
X1	IVR	Intermediate Volcanic Rock	36	2.44	2.96	2.78
X1	LHBGB	Leuc Hornblend Gabbro	6	2.71	2.79	2.76
X1	PHGR	Phyllic Alteration Granite	272	1.82	4.27	2.77
X1	PHMG	Phyllic Alteration Quartz Feldspar Porphyry	2	2.73	2.75	2.74

Table 11-2 - Bulk Density of Saprolite and Weathered Rock from Core Samples, Matupá Project.

Target	ROCK CODE	LEGEND	COUNT	MINIMUM SG_Wet (G/Cc)	MAXIMUM SG_Wet (G/Cc)	AVERAGE DENSITY G/Cc	MINIMUM SG_DRY G/Cc	MAXIMUM SG_DRY G/Cc	AVERAGE SG_DRY G/Cc	AVERAGE Moisture (%)
Guarantã Ridge	SO	Soil	4	1.69	1.96	1.78	1.35	1.55	1.44	19.07
Guarantã Ridge	SAP	Saprolite	63	1.53	2.32	1.86	1.06	2.08	1.50	19.38
Guarantã Ridge	WR	Weathered Rock	4	1.91	2.14	2.03	1.80	1.91	1.84	8.85
X1	SAP	Saprolite	63	1.45	2.45	1.83	1.08	2.32	1.50	18.18
X1	WR	Weathered Rock	4	1.65	2.08	1.87	1.30	1.79	1.53	18.23

11.4 SAMPLE PREPARATION

There is no record of the historical sample preparation before the Rio Novo company, however, due to the available data and certificates, it is possible to assume that the samples preparation was carried out in laboratories, not in any local core-shack facility.

All the Rio Novo sample preparation took place at SGS' laboratory. To ensure that the correct particle size and sample reduction procedures are achieved during sample preparation, the SGS Geosol Laboratory used established protocols for the preparation of samples of rock/core and soil/stream sediments as summarized in Figure 11-3 and Figure 11-4, respectively. Before starting sample preparation, proper equipment must be set up, calibrated, and monitored to ensure quality specifications are met. Quality control measures conducted during sample preparation by SGS Geosol were as follows:

- Equipment is designed and set up to produce representative sample fractions during splitting;
- Equipment was cleaned with barren rock followed by compressed air between each sample run;
- Screen tests for coarse gold were conducted on crushed and pulverized sample fractions at the rate of one test per 20 sample batch.

11.5 SAMPLE ASSAYING

According to the historical certificates in the database, samples from Vale were assayed in their internal laboratory and verified in SGS Geosol as a secondary laboratory. There is no record on which SGS Geosol laboratory the samples were sent to. Vale's internal laboratory lower detection limit was 0.05 ppm using the VS method.

There is no record on which laboratory was used by RTZ and WMC regarding the Serrinhas historical samples.

Samples from MSE were sent to ALS Chemex, but there is no record on which ALS lab. Their lower detection limit was 0.01 ppm using the AA26 Method.

The primary analytical laboratory used by Rio Novo for the Matupá Project was the SGS Geosol laboratories, located in: Vespasiano, Minas Gerais State; Goiânia, Goiás State; and finally, Várzea Grande, Mato Grosso State, Brazil. The Várzea Grande lab was used by Rio Novo from 2011 onwards. The laboratory has ISO 9001 certification and ISO 14001:2004, ISO 17025:2009 certification for environmental chemical analysis. SGS Geosol employs modern, industry-standard techniques and analytical methods. For the purpose of routine gold analysis at the Matupá Gold Project, fire assay with atomic absorption (AA) finish was used most frequently. Multielement analysis (34 elements) were determined by ICP subsequent to the digestion of samples either in aqua regia or in four acids (Table 11-3).

Table 11-3 - SGS Geosol Laboratory: Analytical Methods with Detection Limits.

Geochemical - ICP	Detection Limits	
ICP 34 Elements	**Aqua Regia Digestion**	**Digestion Multi-Acid**
Ag, Ba, Be, Cd, Cr, Cu, Hg, Li, Mo, Ni, Sc, Sr, Zn, Zr, Y	1 ppm	3 ppm
Co, Pb, V	3 ppm	8 ppm
As, Sb	5 ppm	10 ppm
Bi, Sn, W	10 ppm	20 ppm
La	10 ppm	10 ppm
Al, Ca, Fe, K, Mg, Mn, Na, P, S, Ti	0,01%	0,01%
Determination of 34 elements	AR (ICP 34)	DT (ICP34)

Fire Assay with AA finish is a quantitative analysis through which precious metals are separated by melting a powdered mineral sample in a reducing environment. The precious metals are collected in the molten lead which separates from the slag by virtue of density differences. The lead button is then dissolved in aqua regia and the resulting acid solution containing the precious metals is analyzed by atomic absorption spectroscopy to determine the gold grade. The analytical detection limit for gold by fire assay-AA finish is 5 ppb. For gold assays in excess of 10,000 ppb, the samples were re-assayed using the metallic screen test (MET-150). The metallic screen test optimizes the accuracy and precision of higher gold concentrations associated with coarse gold grains. In this method, a larger sample is pulverized and sieved with a 150-mesh screen. The coarse sample fraction captures any coarse-grained gold in the sample. The +150 mesh and the -150 mesh sample fractions are assayed separately. The entire coarse fraction is analyzed while a 50 g (2 AT) sample is used on the fine fraction. The gold concentration of the total sample is reported as the weighted average of the two fractions.

The coarse fraction, generally containing most of the native metal in coarse gold deposits, is assayed in total and a 50 g aliquot of the fine fraction is assayed. Results are reported as the weighted average of the fractions. A flow chart showing the analytical procedures employed by SGS Geosol at the Matupá Gold Project is presented in Figure 11-5.

The second laboratory used by Rio Novo for check assays was ALS Chemex in Vespasiano, Minas Gerais State and Goiânia, Goiás State, Brazil. The analysis were made in Lima, Peru. The analytical codes and a brief description of the analysis of gold in the ALS laboratory is given in Table 11-4.

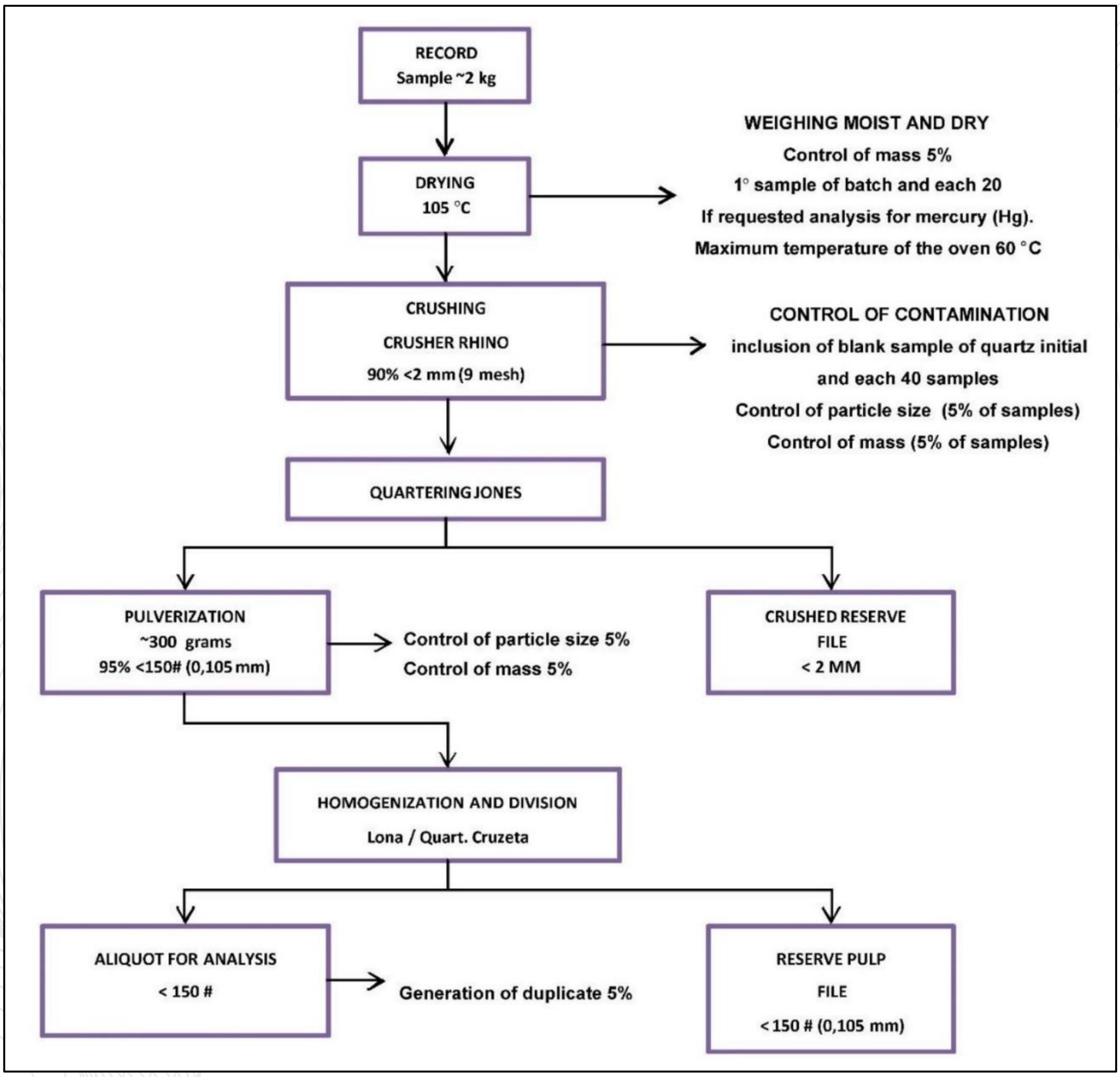


Figure 11-3 - Sample Preparation Protocol – Rock and Core (Rio Novo 2010-2012).

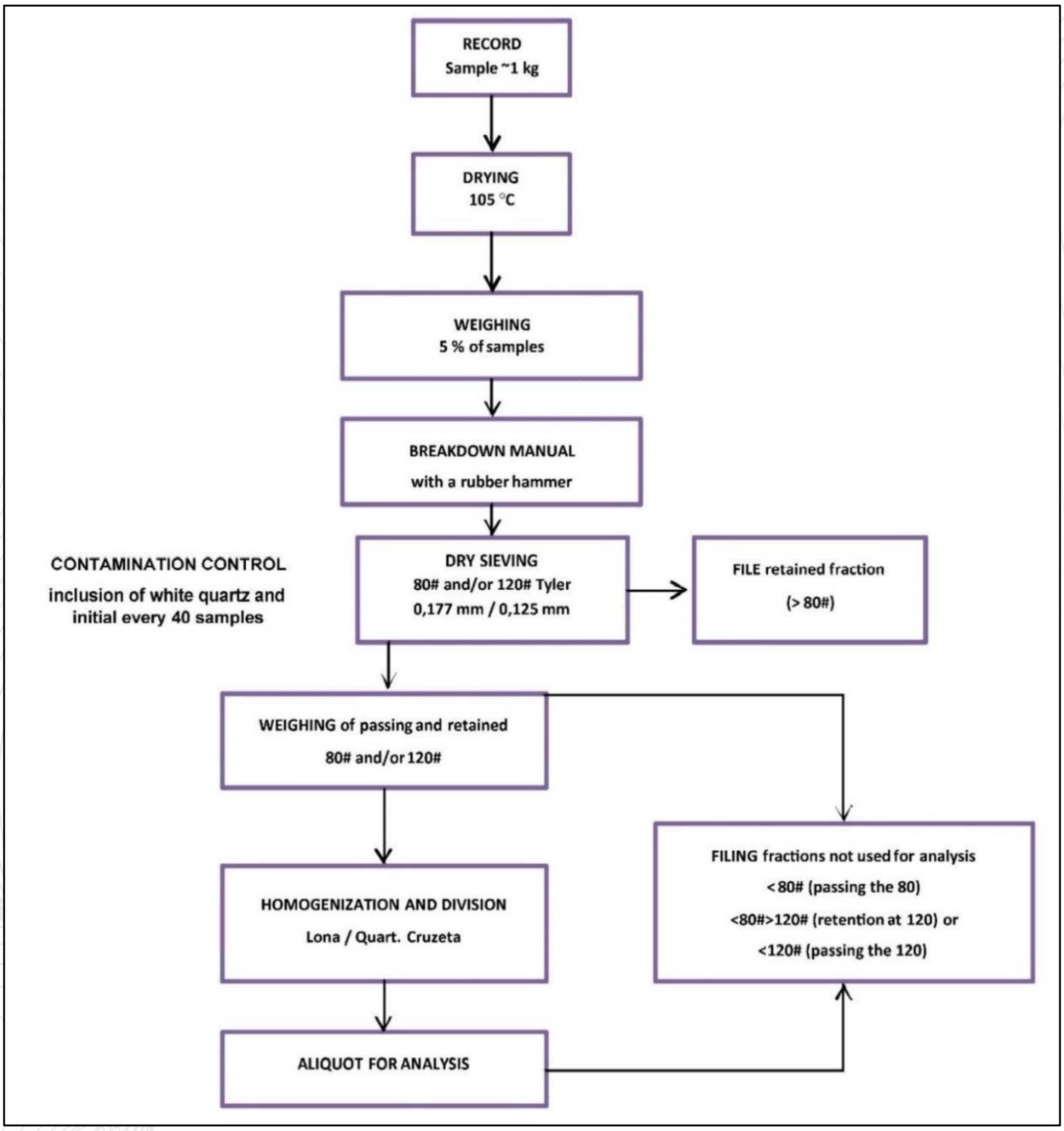


Figure 11-4 - Sample Preparation Protocol (Rio Novo 2010-2012).

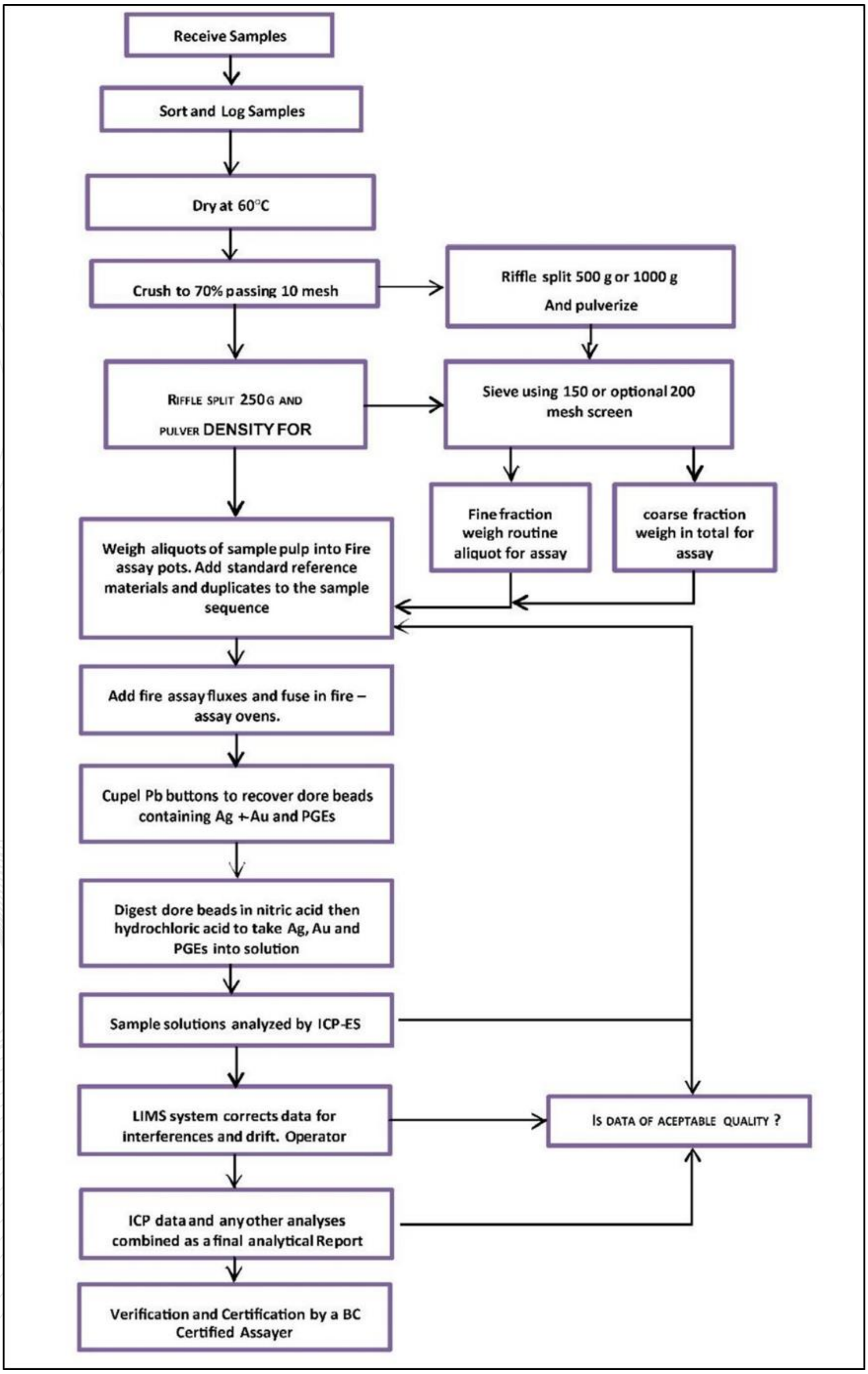


Figure 11-5 - Analytical Protocol (Rio Novo 2010-2012).

11.6 QA/QC PROGRAM

Analytical work was carried out by SGS Geosol Lab, in Vespasiano, Minas Gerais, Brazil. Drill core were sawed in the Project core-shed and shipped to SGS, where the samples were crushed, pulverized and homogenized, then pulp samples were analyzed for Gold using Fire Assay (Atomic Absorption – fusion 50 g aliquots) and analysis ICP12B for determination of up to 34 elements by ICP/Digestion with aqua regia. For samples with gold grades higher than 10,000 ppb, the analysis MET 150 (Metallic Screen) were applied. The remaining coarse and pulverized reject portion of the samples were returned to the Project facility for storage. Rio Novo had routine quality control procedures which ensured that every batch of 20 samples included one commercial standard (high, medium or low grade), one blank and one sample repeat. In addition, 5 to 10% of external check assays of mineralized samples (cut-off 0.3 g/t Au) were carried out by ALS Chemex Laboratories, Vespasiano, Minas Gerais, Brazil. SGS Geosol and ALS Chemex have their own routine quality control procedures which ensured the insertion of blanks, commercial standards and duplicates into each batch of samples to be analyzed. SGS replicates were also used. All analytical results and certificates from both laboratories were provided separately and digital copies of the files were stored in the Rio Novo digital database.

11.6.1 2007-2008 RIO NOVO QA/QC PROGRAM

A total of 217 purchased reference standards were inserted into the sample stream during the 2008 drill program by Santa Elina (MSE) starting with hole FDSEX1-004. The initial set of standards was purchased from Geostats Pty Ltd. of Western Australia. When these were exhausted, a second set was purchased from Ore Research & Exploration Pty Ltd. of Victoria, Australia. These standards were also depleted before the end of the program. An attempt was made to substitute copper standards for the last 10 holes, but the pulp size was inadequate for fire assays and these standards were not certified for gold. A total of 371 blank pulps were inserted into the sample stream during the 2008 drill program. No duplicate samples were sent for cross checks and none were checked at a different lab to check for analytical bias.

Table 11-4 - List of Rio Novo Standards and their Statistics from 2008 Drill Campaign.

Standard	Element	Certified Value ppm	+ 2Std	- 2Std	+ 3Std	- 3Std	Source
901-11	Au	1.340	1.220	1.460	1.160	1.520	Geostats Pty Ltd
905-5	Au	0.520	0.460	0.580	0.430	0.610	Geostats Pty Ltd
997-9	Au	5.160	4.520	5.800	4.200	6.120	Geostats Pty Ltd
OREAS 10Pb	Au	7.150	6.770	7.530	6.580	7.720	Oreas
OREAS 15Pa	Au	1.020	0.968	1.072	0.942	1.098	Oreas
OREAS 18Pb	Au	3.630	3.490	3.770	3.420	3.840	Oreas
OREAS 50Pb	Au	0.841	0.777	0.905	0.745	0.937	Oreas

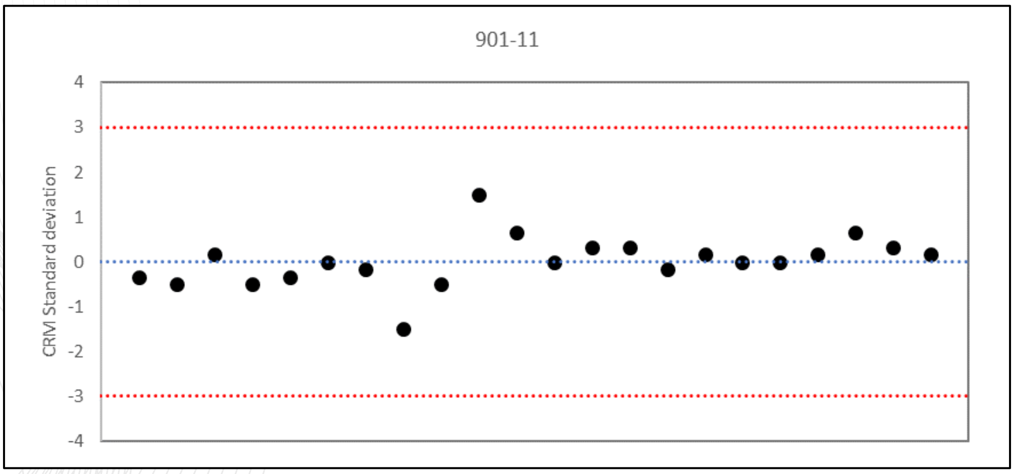
Note: Std = standard deviation.

Standard Samples

Standard performance was acceptable with only two instances of failure beyond the three standard deviation thresholds. A minor positive bias is apparent in high standards 10Pb and 18Pb but is not considered significant.

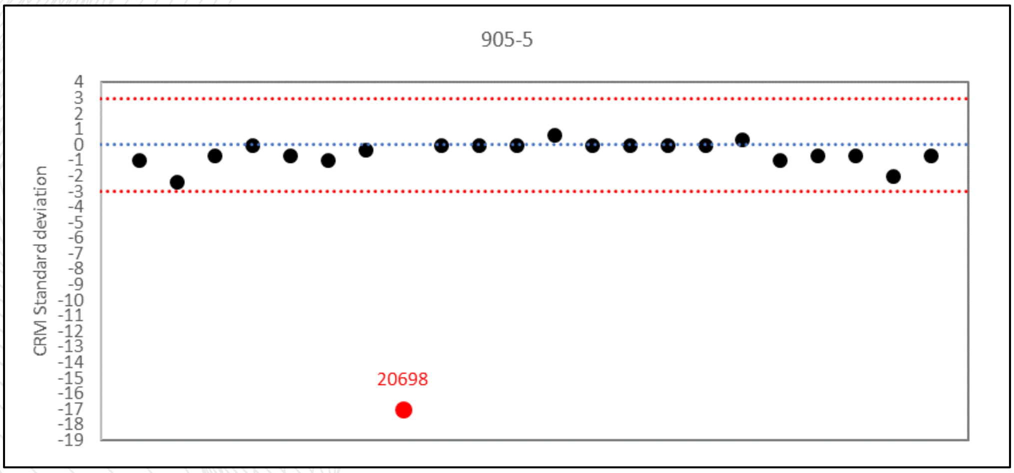
The standard sample 20698 (905-5) returned with results below the detection limit. Probably it is a blank instead of a standard. The standard failed but in the same batch there are other samples of the same standard that validate the batch. The standard sample 21887 (997-9) failed with values in the range of standard 905-5, a possible exchange of values is not ruled out. The certificate was not reanalysed. The standard sample 46256 (OREAS 15Pa) failed about 0.2 percent below the acceptable limit. It was not reassayed. Sample 46081 (OREAS 18Pb) failed below 3 standards deviation and the certificate was not reassayed. The sample 46297 (OREAS 50Pb) failed below 3 standards deviation and the certificate was not reassayed.

Sample sequence charts for the second set of standards are shown in Figure 11-6 to Figure 11-12.



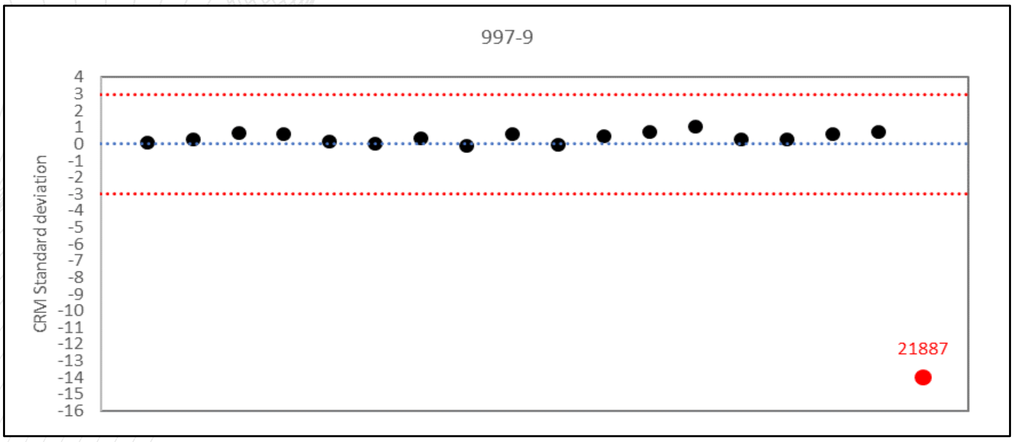
CRM name	901-11
CRM Au ppm	1.340
CRM Standard deviation	0.060
Samples	22
# Outside Error Limit	0
Mean	1.3414
Min	1.25
Max	1.43
Standard Deviation	0.0343
% Rel. Std. Dev.	2.5547
Coeff. of Var.	0.0255
Standard Error	0.0073
% Rel. Std. Err.	0.5447
Total Bias	0.0010
% Mean Bias	0.1018

Figure 11-6 - Standard (901-11) Sample Sequence Chart.



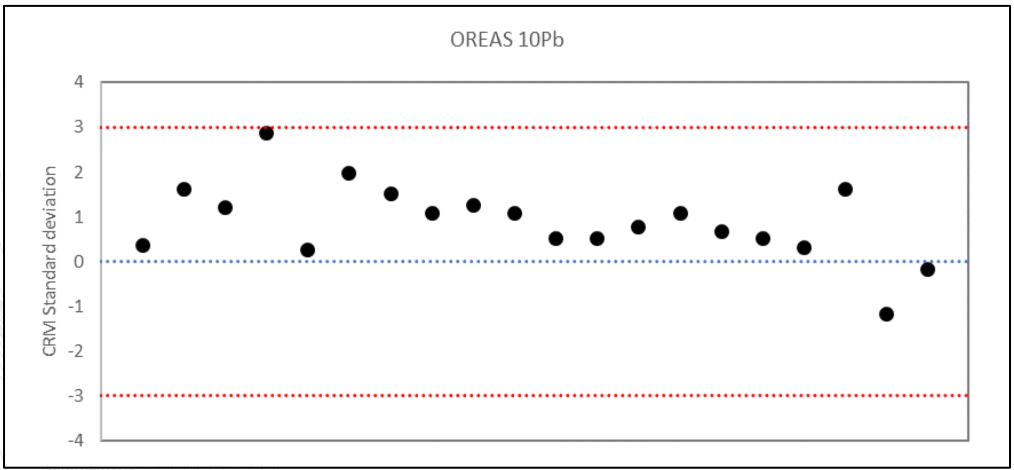
CRM name	905-5
CRM Au ppm	0.520
CRM Standard deviation	0.030
Samples	22
# Outside Error Limit	1
Mean	0.4832
Min	0.01
Max	0.54
Standard Deviation	0.1078
% Rel. Std. Dev.	22.3123
Coeff. of Var.	0.2231
Standard Error	0.0230
% Rel. Std. Err.	4.7570
Total Bias	-0.0708
% Mean Bias	-7.0804

Figure 11-7 - Standard (905-5) Sample Sequence Chart.



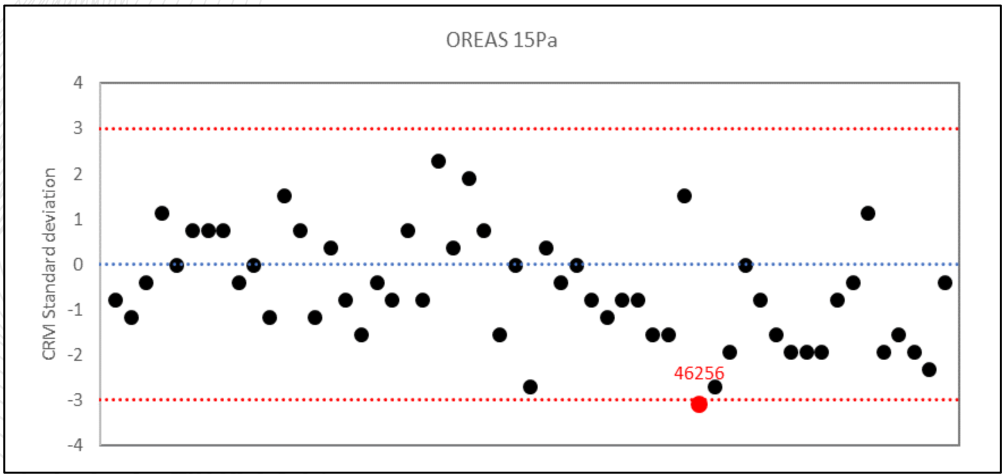
CRM name	997-9
CRM Au ppm	5.160
CRM Standard deviation	0.320
Samples	18
# Outside Error Limit	1
Mean	5.0433
Min	0.69
Max	5.51
Standard Deviation	1.0910
% Rel. Std. Dev.	21.6325
Coeff. of Var.	0.2163
Standard Error	0.2572
% Rel. Std. Err.	5.0988
Total Bias	-0.0226
% Mean Bias	-2.2610

Figure 11-8 - Standard (997-9) Sample Sequence Chart.



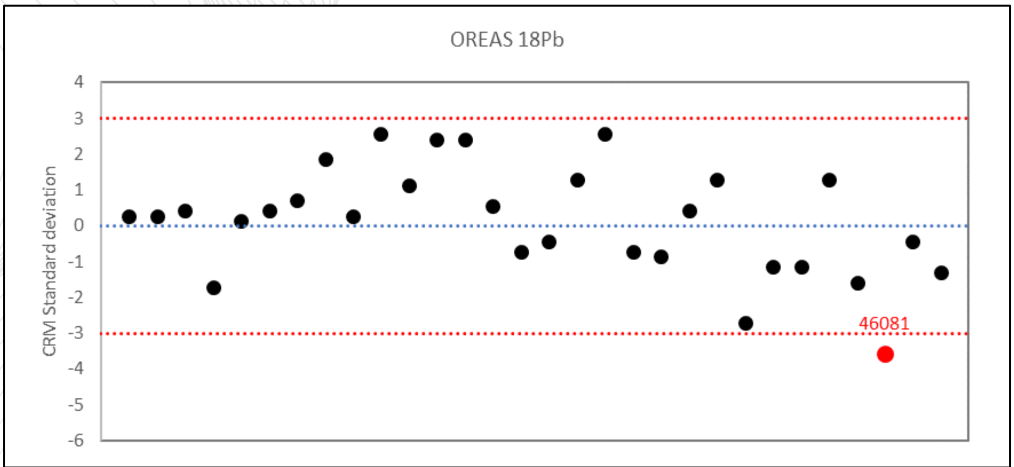
CRM name	OREAS 10Pb
CRM Au ppm	7.150
CRM Standard deviation	0.190
Samples	20
# Outside Error Limit	0
Mean	7.3225
Min	6.93
Max	7.7
Standard Deviation	0.1623
% Rel. Std. Dev.	2.2167
Coeff. of Var.	0.0222
Standard Error	0.0363
% Rel. Std. Err.	0.4957
Total Bias	0.0241
% Mean Bias	2.4126

Figure 11-9 - Standard (10Pb) Sample Sequence Chart.



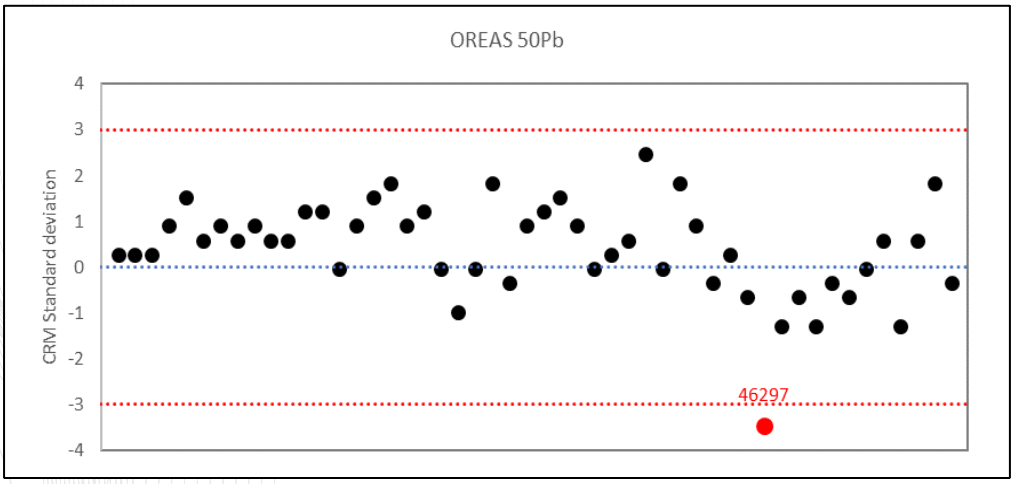
CRM name	OREAS 15Pa
CRM Au ppm	1.020
CRM Standard deviation	0.026
Samples	55
# Outside Error Limit	1
Mean	1.0058
Min	0.94
Max	1.08
Standard Deviation	0.0319
% Rel. Std. Dev.	3.1712
Coeff. of Var.	0.0317
Standard Error	0.0043
% Rel. Std. Err.	0.4276
Total Bias	-0.0139
% Mean Bias	-1.3904

Figure 11-10 - Standard (15Pb) Sample Sequence Chart.



CRM name	OREAS 18Pb
CRM Au ppm	3.630
CRM Standard deviation	0.070
Samples	30
# Outside Error Limit	1
Mean	3.6397
Min	3.38
Max	3.81
Standard Deviation	0.1070
% Rel. Std. Dev.	2.9393
Coeff. of Var.	0.0294
Standard Error	0.0195
% Rel. Std. Err.	0.5366
Total Bias	0.0027
% Mean Bias	0.2663

Figure 11-11 - Standard (18Pb) Sample Sequence Chart.

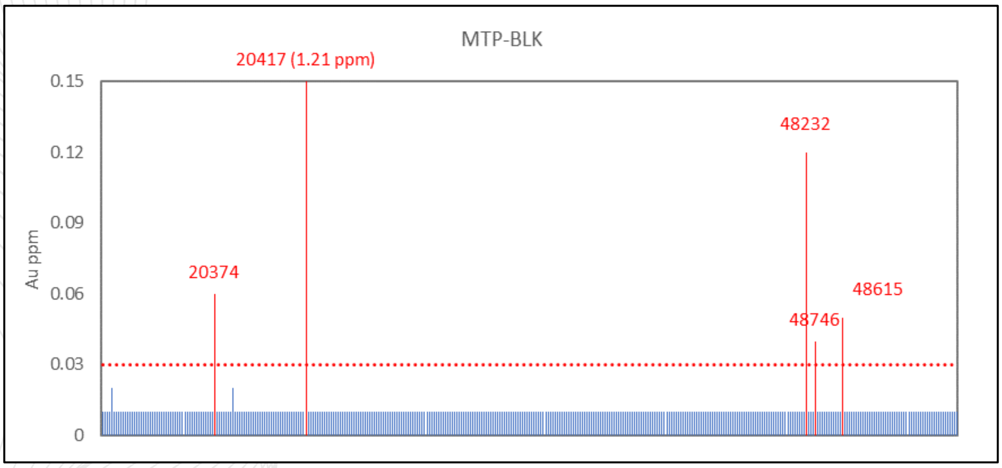


CRM name	OREAS 50Pb
CRM Au ppm	0.841
CRM Standard deviation	0.032
Samples	50
# Outside Error Limit	1
Mean	0.8540
Min	0.73
Max	0.92
Standard Deviation	0.0334
% Rel. Std. Dev.	3.9159
Coeff. of Var.	0.0392
Standard Error	0.0047
% Rel. Std. Err.	0.5538
Total Bias	0.0155
% Mean Bias	1.5458

Figure 11-12 - Standard (50Pb) Sample Sequence Chart.

<u>Blank Samples</u>

A total of 371 blank pulps were inserted into the sample stream during the 2008 drill program. Six of these failed the 0.025 g/t Au threshold as shown in Figure 11-13, Blank Performance. One of these is deemed significant assaying 3.08 g/t and coming in sequence close to a very high-grade sample (71.9 g/t). This likely indicates contamination in the preparation laboratory and is being investigated.



Blank name	MTP-BLK
Au ppm expected	0.01
Au ppm error limit	0.03
Samples	375
# Outside Error Limit	5
Mean	0.0139
Min	0.01
Max	1.21
Standard Deviation	0.0623
% Rel. Std. Dev.	449.2929
Coeff. of Var.	4.4929
Standard Error	0.0032
% Rel. Std. Err.	23.2014

Figure 11-13 - Blank Samples Chart.

11.6.2 2010-2012 RIO NOVO QA/QC PROGRAM

<u>Standard Samples</u>

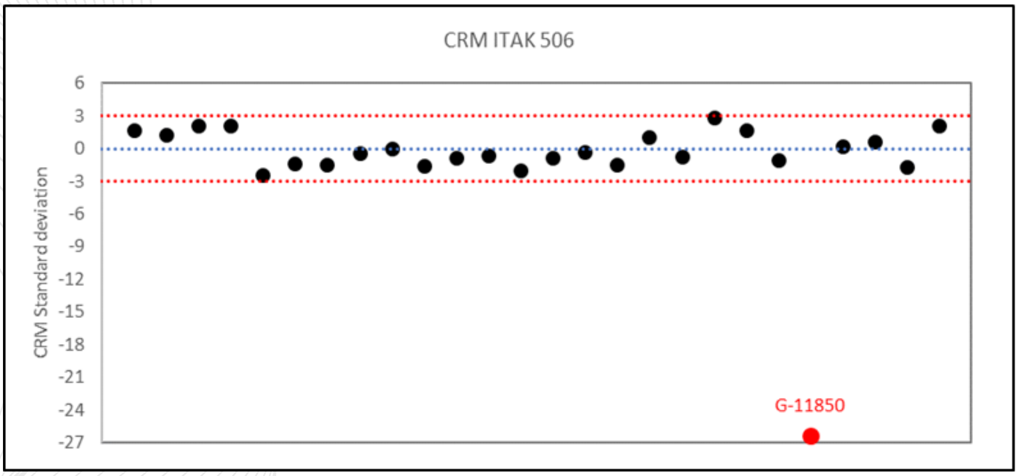
A total of 287 purchased reference standards were inserted into the sample stream during the 2010 to 2012 drill program starting with hole FX1D-0001 in the X1 Deposit. The initial set of standards was purchased from Geostats Pty Ltd. of Western Australia. When these were exhausted, a second set was purchased from ITAK located in João Monlevade, Minas Gerais State, Brazil.

Table 11-5 - List of Rio Novo Standards and their Statistics from 2010-2012 Drill Campaigns.

Standard	Element	Type	Certified Value ppb	+ 2Std	- 2Std	+ 3Std	- 3Std	Source
CRM ITAK 506	Au	High grade	8870	8870	8870	8870	8870	ITAK
CRM ITAK 518	Au	Low grade	547	547	547	547	547	ITAK
CRM ITAK 528	Au	High grade	2760	2760	2760	2760	2760	ITAK
CRM ITAK 529	Au	Low grade	315	315	315	315	315	ITAK
G305-2	Au	Low grade	320	320	320	320	320	Geostats Pty Ltd
G307-4	Au	Middle grade	1400	1400	1400	1400	1400	Geostats Pty Ltd
G900-5	Au	High grade	3210	3210	3210	3210	3210	Geostats Pty Ltd

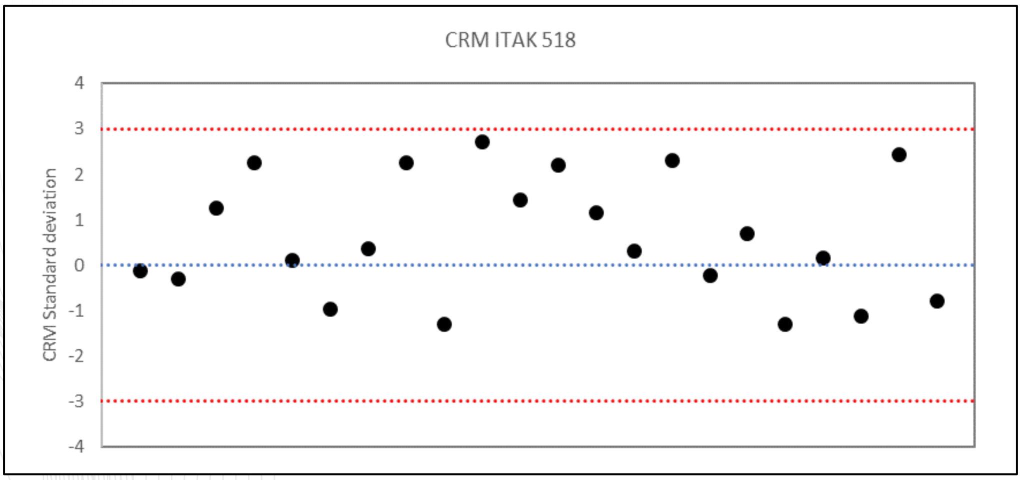
Note: Std = standard deviation

Standard performance was acceptable with six instances of failure beyond the three standard deviation thresholds. A minor positive bias is apparent in high standards ITAK 518 but is not considered to be significant. Sample sequence charts for the 2010-2012 drill campaigns set of field standards are shown in Figure 11-14 to Figure 11-20. Figure of 11-21 shows combined field standards performance for 2010-2012 drill campaigns and Table of 11-6 shows statistics of the same samples. Ten standard samples returned in 10 certificates failed outside 3 standards deviations. From these certificates only the two samples were re-assayed.



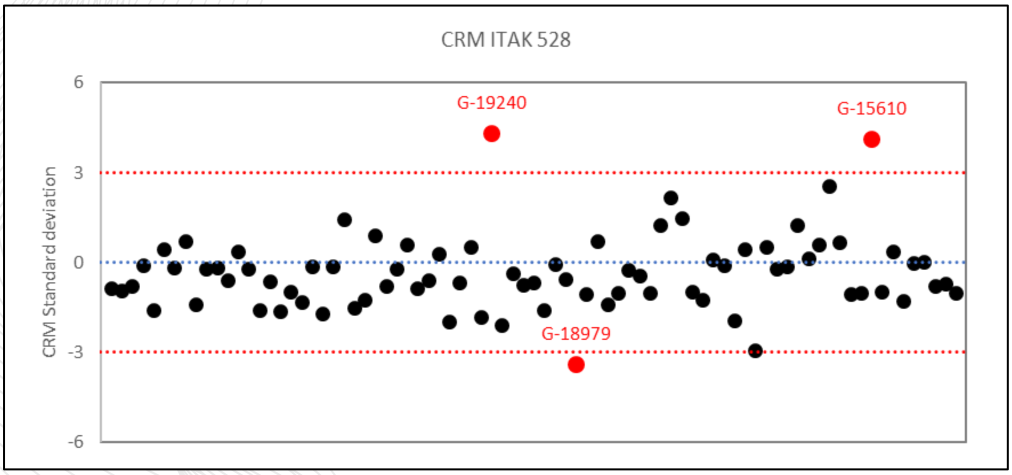
CRM name	CRM ITAK 506
CRM Au ppb	8870
CRM Standard deviation	270
Samples	26
#Outside Error Limit	1
Mean	8590.9231
Min	1760
Max	9638
Standard Deviation	1450.7108
%Rel. Std. Dev.	16.8866
Coeff. of Var.	0.1689
Standard Error	284.5078
%Rel. Std. Err.	3.3117
Total Bias	-0.0315
%Mean Bias	-3.1463

Figure 11-14 - Standard (ITAK 506) Sample Sequence Chart.



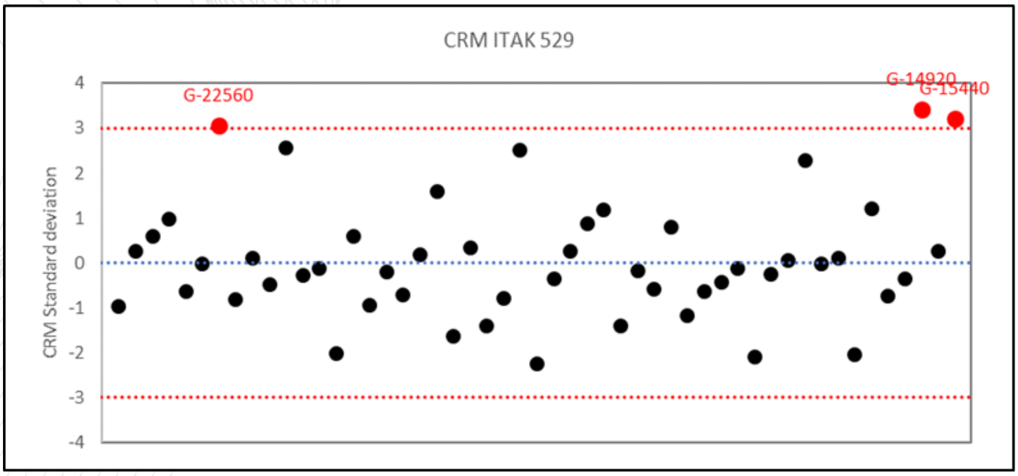
CRM name	CRM ITAK 518
CRM Au ppb	**547**
CRM Standard deviation	**18**
Samples	22
# Outside Error Limit	0
Mean	558.3636
Min	524
Max	596
Standard Deviation	24.0269
% Rel. Std. Dev.	4.3031
Coeff. of Var.	0.0430
Standard Error	5.1225
% Rel. Std. Err.	0.9174
Total Bias	0.0208
% Mean Bias	2.0774

Figure 11-15 - Standard (ITAK 518) Sample Sequence Chart.



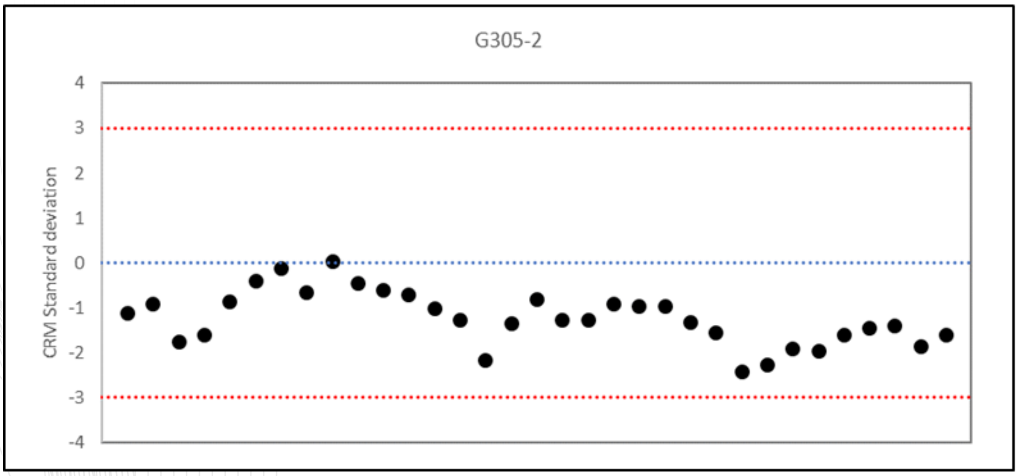
CRM name	CRM ITAK 528
CRM Au ppb	**2760**
CRM Standard deviation	**150**
Samples	81
# Outside Error Limit	3
Mean	2709.3704
Min	2250
Max	3412
Standard Deviation	190.0099
% Rel. Std. Dev.	7.0131
Coeff. of Var.	0.0701
Standard Error	21.1122
% Rel. Std. Err.	0.7792
Total Bias	-0.0183
% Mean Bias	-1.8344

Figure 11-16 - Standard (ITAK 528) Sample Sequence Chart.



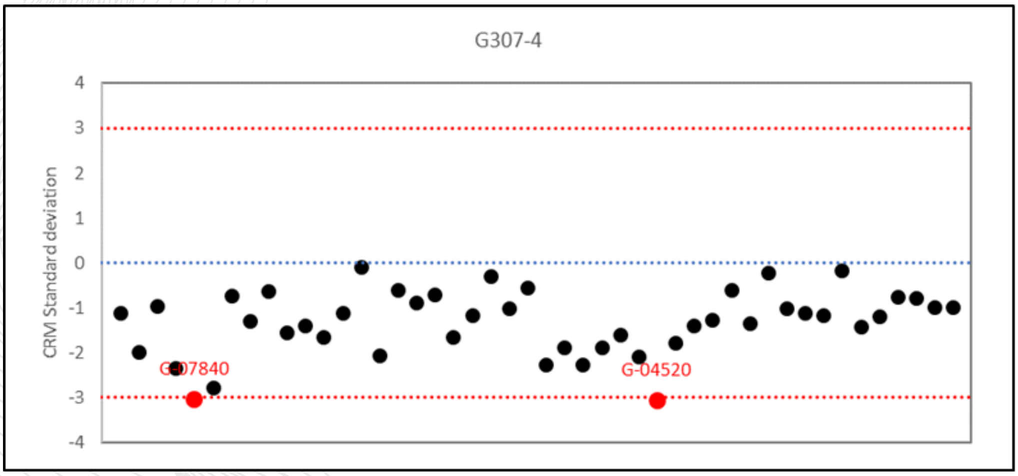
CRM name	CRM ITAK 529
CRM Au ppb	**315**
CRM Standard deviation	**26**
Samples	51
# Outside Error Limit	3
Mean	316.8039
Min	257
Max	404
Standard Deviation	34.9062
% Rel. Std. Dev.	11.0182
Coeff. of Var.	0.1102
Standard Error	4.8878
% Rel. Std. Err.	1.5429
Total Bias	0.0057
% Mean Bias	0.5727

Figure 11-17 - Standard (ITAK-529) Sample Sequence Chart.



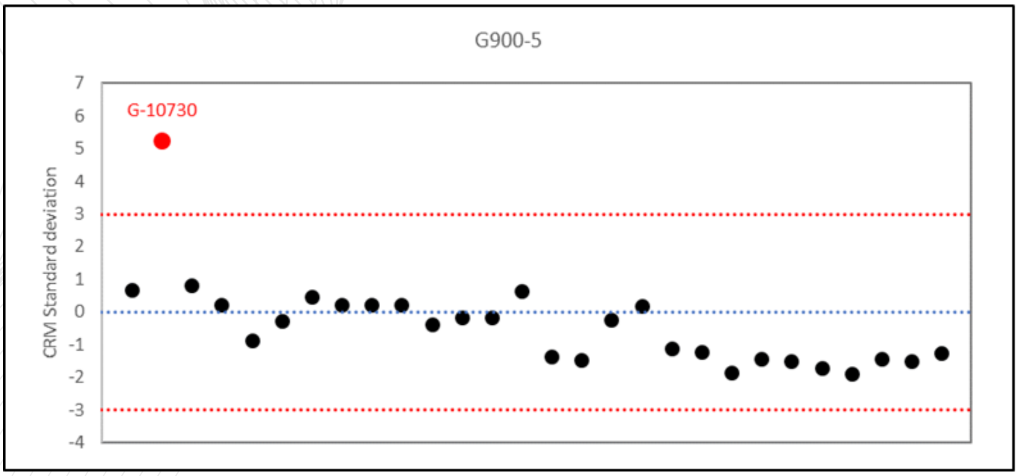
CRM name	G305-2
CRM Au ppb	320
CRM Standard deviation	20
Samples	33
# Outside Error Limit	0
Mean	295.6667
Min	272
Max	321
Standard Deviation	11.9809
% Rel. Std. Dev.	4.0522
Coeff. of Var.	0.0405
Standard Error	2.0856
% Rel. Std. Err.	0.7054
Total Bias	-0.0760
% Mean Bias	-7.6042

Figure 11-18 - Standard (G305-2) Sample Sequence Chart.



CRM name	G307-4
CRM Au ppb	1400
CRM Standard deviation	60
Samples	46
# Outside Error Limit	2
Mean	1321.0435
Min	1217
Max	1395
Standard Deviation	43.1596
% Rel. Std. Dev.	3.2671
Coeff. of Var.	0.0327
Standard Error	6.3635
% Rel. Std. Err.	0.4817
Total Bias	-0.0564
% Mean Bias	-5.6398

Figure 11-19 - Standard (G307-4) Sample Sequence Chart.



CRM name	G900-5
CRM Au ppb	3210
CRM Standard deviation	130
Samples	28
# Outside Error Limit	1
Mean	3159.6071
Min	2966
Max	3893
Standard Deviation	182.7071
% Rel. Std. Dev.	5.7826
Coeff. of Var.	0.0578
Standard Error	34.5284
% Rel. Std. Err.	1.0928
Total Bias	-0.0157
% Mean Bias	-1.5699

Figure 11-20 - Standard (G900-5) Sample Sequence Chart.

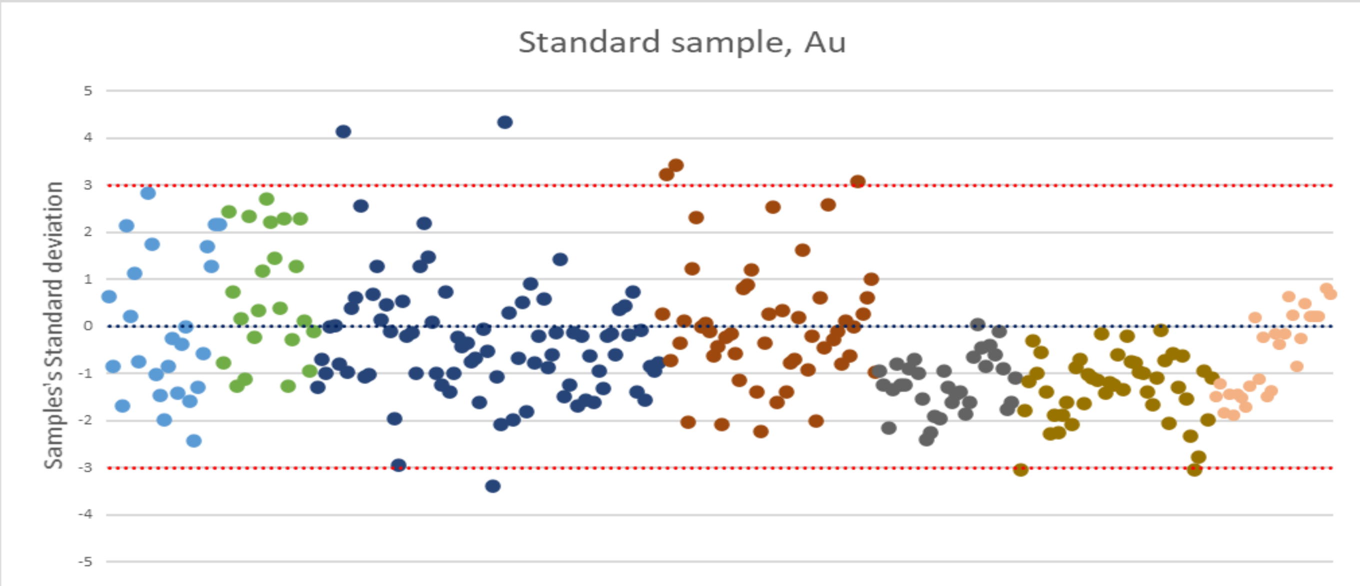


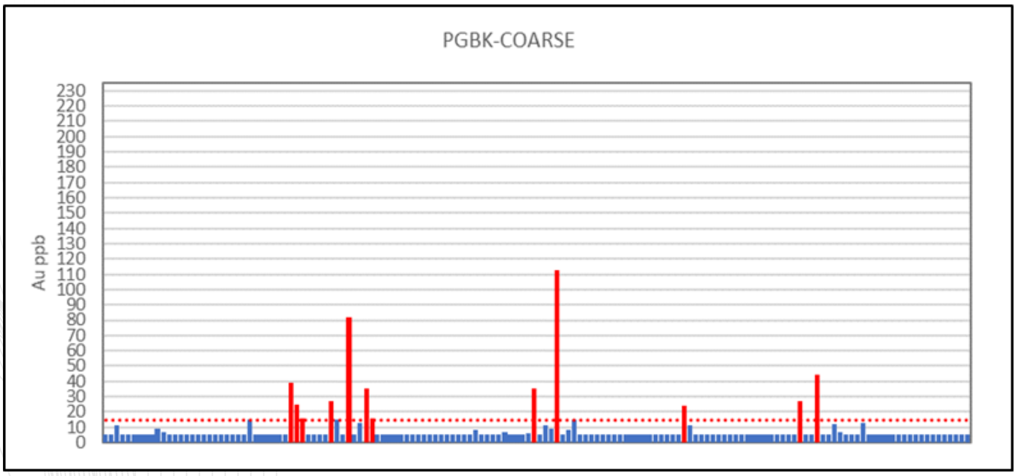
Figure 11-21 - Rio Novo Field Standards Performance (2010-2012).

Table 11-6 - Standards Samples Statistics (2010-2012).

Au_FAA505_ppb	ITAK 506	ITAK 518	ITAK 528	ITAK 529	G305-2	G307-4	G900-5
Legend color							
# of Analysis above Threshold	26	22	81	51	33	46	28
# Outside Warning Limit	6	6	7	9	3	8	1
# Outside Error Limit	1	0	3	3	0	2	1
# of Analysis below Threshold	0	0	0	0	0	0	0
% Outside Error Limit	3.846	0.000	3.704	5.882	0.000	4.348	3.571
% Positive	100	100	100	100	100	100	100
% Negative	0	0	0	0	0	0	0
% at Zero	0	0	0	0	0	0	0
Mean	8590.923	558.364	2709.370	316.804	295.667	1321.044	3159.607
Median	8742	553.5	2680	312	295	1329.5	3168.5
Min	1760	524	2250	257	272	1217	2966
Max	9638	596	3412	404	321	1395	3893
Standard Deviation	1450.711	24.027	190.010	34.906	11.981	43.160	182.707
% Rel. Std. Dev.	16.887	4.303	7.013	11.018	4.052	3.267	5.783
Coeff. of Var.	0.169	0.043	0.070	0.110	0.041	0.033	0.058
Standard Error	284.508	5.123	21.112	4.888	2.086	6.364	34.528
% Rel. Std. Err.	3.312	0.917	0.779	1.543	0.705	0.482	1.093
Total Bias	-0.032	0.021	-0.018	0.006	-0.076	-0.056	-0.016
% Mean Bias	-3.146	2.077	-1.834	0.573	-7.604	-5.640	-1.570

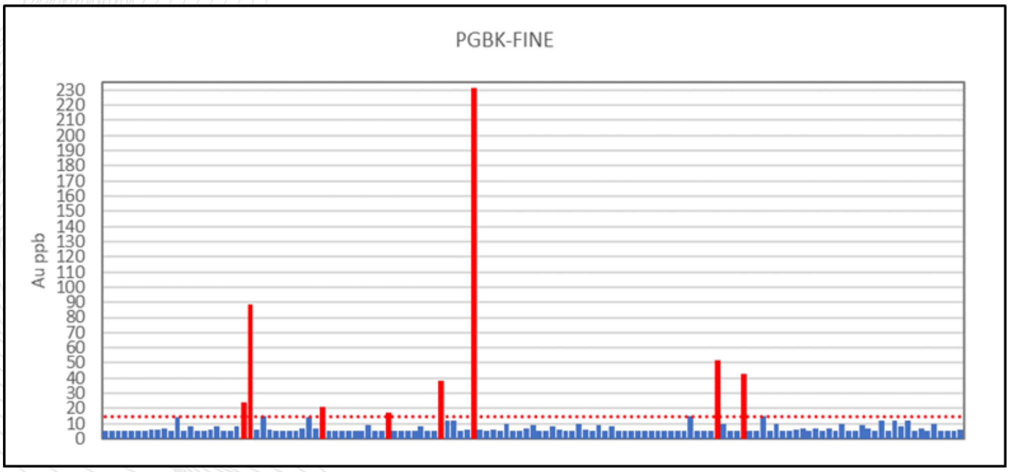
Blank Samples

A total of 20 blank samples failed in the QA/QC with value above 3 detection limits (15 ppb Au) and 10 samples surrounding the blank sample position in the batch were re-assayed. Only highlighted samples in the Table 11-7 were reanalyzed since the other failed blank samples were not part of the mineralized zone.



CRM name	PGBK-COARSE
CRM Au ppb	0
CRM Standard deviation	15
Samples	150
# Outside Error Limit	12
Mean	8.4333
Min	5
Max	113
Standard Deviation	12.4029
% Rel. Std. Dev.	147.0696
Coeff. of Var.	1.4707
Standard Error	1.0127
% Rel. Std. Err.	12.0082

Figure 11-22 - Blank Samples Chart (Coarse).



CRM name	PGBK-FINE
CRM Au ppb	0
CRM Standard deviation	15
Samples	131
# Outside Error Limit	8
Mean	10.0000
Min	5
Max	231
Standard Deviation	21.7312
% Rel. Std. Dev.	217.3123
Coeff. of Var.	2.1731
Standard Error	1.8987
% Rel. Std. Err.	18.9867

Figure 11-23 - Blank Samples Chart (Fine).

A total of 20 blank samples failed in the QA/QC with value above 3 detection limit (15 ppb Au) and 10 samples surrounding the blank sample position in the batch were re-assayed. Only highlighted samples in the Table 11-7 were reanalyzed since the other failed blank samples were not part of mineralized zone.

Table 11-7 - List of Reanalysed Blank Samples (Highlighted).

HOLEID	CHECKID	STANDARDID	STANDARDVALUE	STANDARDDEVIATION	ASSAYVALUE	ZSCORE	DESPATCHNO	LABJOBNO
FX1D-0017	G-08890	PGBK-COARSE	0	5	39		PG-0023	VG1100078
FX1D-0017	G-08970	PGBK-COARSE	0	5	25		PG-0023	VG1100079
FX1D-0017	G-09050	PGBK-COARSE	0	5	16		PG-0023	VG1100081
FX1D-0018	G-09490	PGBK-COARSE	0	5	27		PG-0025	VG1100096
FX1D-0021	G-10650	PGBK-FINE	0	5	24		PG-0027	VG1100109
FX1D-0021	G-10690	PGBK-COARSE	0	5	82		PG-0027	VG1100109
FX1D-0021	G-10720	PGBK-FINE	0	5	89		PG-0027	VG1100109
FX1D-0020	G-10890	PGBK-COARSE	0	5	35		PG-0028	VG1100112
FX1D-0020	G-10970	PGBK-COARSE	0	5	16		PG-0028	VG1100112
FX1D-0013	G-12010	PGBK-FINE	0	5	21		PG-0031	VG1100132
FX1D-0029	G-13410	PGBK-FINE	0	5	17		PG-0034	VG1100172
FX1D-0028	G-14890	PGBK-FINE	0	5	38		PG-0037	VG1100208
FX1D-0028	G-14930	PGBK-COARSE	0	5	35		PG-0037	VG1100208
FX1D-0028	G-15170	PGBK-COARSE	0	5	113		PG-0037	VG1100210
FX1D-0032	G-15570	PGBK-FINE	0	5	231		PG-0039	VG1100231
FX1D-0038	G-17140	PGBK-COARSE	0	5	24		PG-0045	VG1100265
FX1D-0048	G-19250	PGBK-FINE	0	5	52		PG-0049	VG1100301
FX1D-0043	G-19810	PGBK-FINE	0	5	43		PG-0050	VG1100310
FX1D-0045	G-20380	PGBK-COARSE	0	5	27		PG-0052	VG1100323
FX1D-0049	G-19290	PGBK-COARSE	0	5	44		PG-0056	VG1100339

Figure 11-24 shows that most of blank samples (combined fine and coarse) were below the warning threshold limit with only three samples above the failure limit in November 2012. In total 3,455 external blank assays were returned by the laboratories for which no contamination was observed by Rio Novo personnel. Table 11-8 Shows statistics of blank samples inserted into field QA/QC program for the period of 2010 to 2012 for Rio Novo.

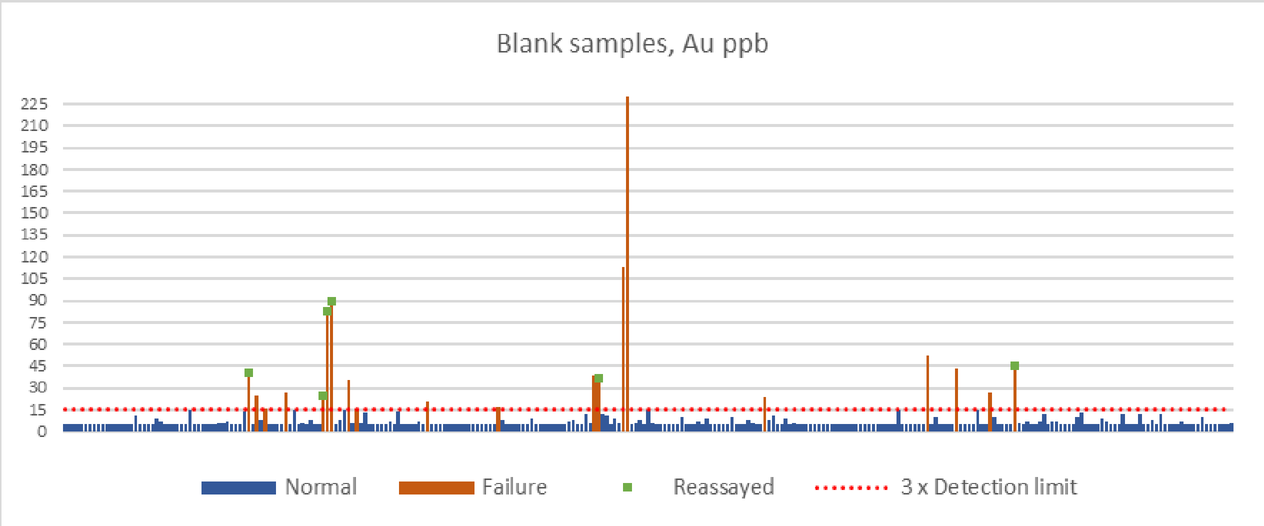


Figure 11-24 - Blank Samples Chart (Rio Novo Drilling 2010- 2012).

Table 11-8 - Blank Samples Statistics.

Au_FAA505_ppb	PGBK-COARSE	PGBK-FINE
# of Analyses above Threshold	150	131
# Outside Warning Limit	150	131
# Outside Error Limit	12	8
# of Analyses below Threshold	0	0
% Outside Error Limit	8.000	6.107
% Positive	100	100
% Negative	0	0
% at Zero	0	0
Mean	8.433	10.000
Median	5	5
Min	5	5
Max	113	231
Standard Deviation	12.403	21.731
% Rel. Std. Dev.	147.070	217.312
Coeff. of Var.	1.471	2.173
Standard Error	1.013	1.899
% Rel. Std. Err.	12.008	18.987
Total Bias	n/a	n/a
% Mean Bias	n/a	n/a

Duplicate Samples

A total of 279 field duplicate samples were included in the sample stream sent to the SGS laboratory. Figure 11-25 shows the performance of field duplicates samples and Table 11-9 shows basic statistics for the field duplicates. The assay results were proportionally reduced to fill in the chart.

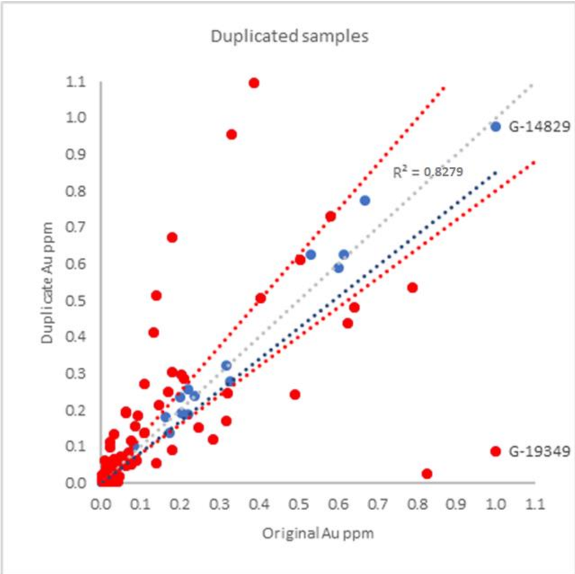


Figure 11-25 - Duplicate Performance (Rio Novo Drilling - 2010-2012).

Table 11-9 - Duplicate Statistics (Rio Novo Drilling 2010-2012).

Au_FAA505_ppb	Original	Check
# of Analysis above Threshold	279	
# of Analysis below Threshold	0	
# Outside Warning Limit	142	
# Outside Error Limit	107	
Mean	0.078	0.078
Median	0.006	0.005
Min	0.005	0.005
Max	3.181	3.112
Range	3.176	3.107
Variance	0.068	0.058
Coeff. of Var.	3.358	3.069
Sample Std. Dev.	0.261	0.240
Bias	0.008	
Corr. Coeff.	0.828	
RMA Error on Y Intercept	0.009	
RMA Error on Slope	0.031	
RMA Error	0.208	
Mean AMPRD	28.561	
% of Population with AMPRD > 5%	52.330	
% of Population with AMPRD > 10%	50.896	
% of Population with AMPRD > 15%	46.953	
% of Population with AMPRD > 20%	39.068	

Re-assayed Samples in SGS Lab

A total of 54 samples were reanalyzed in SGS GEOSOL Laboratórios Ltda. Vespasiano, Mato Grosso, Brazil in 2011, due to failure of the standards or blank samples. The R square value of 0.76 observed in the scatter plot for these samples indicates that the intensity of the correlation between samples is regular. (Figure 11-26).

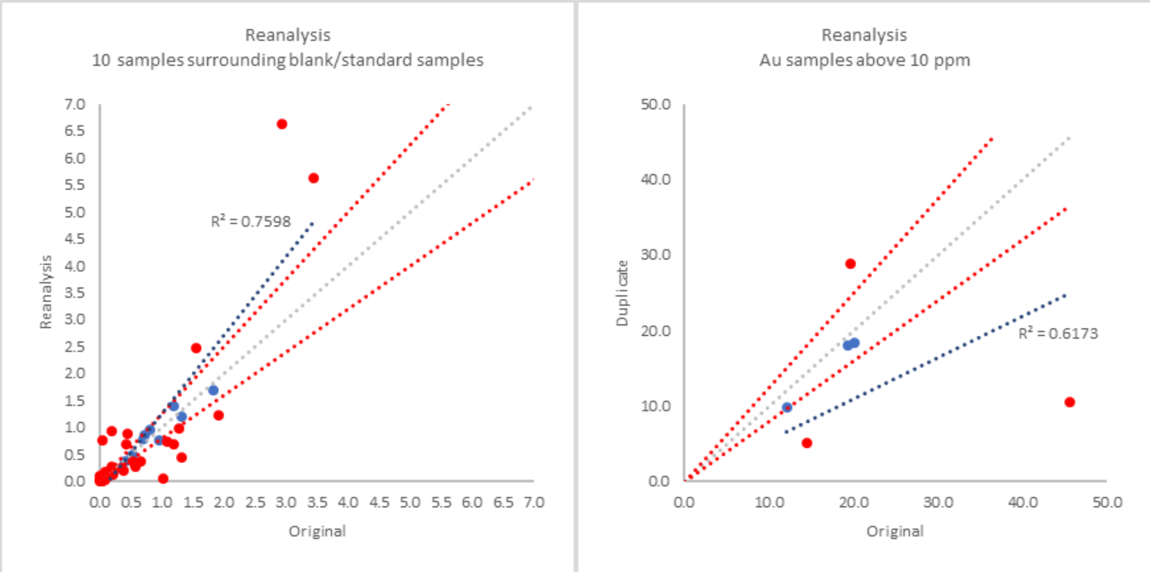


Figure 11-26 - Re-Assayed Samples Scatter Plots (SGS-2011).

A total of 5 samples were also reanalyzed using the Metallic Screen method, a procedure automatically applied to original samples above 10,000 ppb. The low correlation of those samples indicates the presence of nugget effect.

A total of 132 samples from hole FX1D-0022 were re-assayed (certificate VG1100131) despite of the fact that the blanks and standards samples did not fail in the QA/QC analysis. The comparison of reanalyzed samples showed good correlation (Figure 11-27).

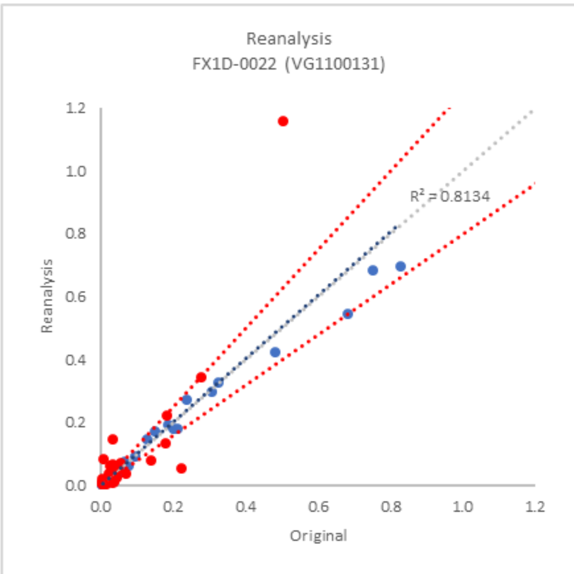


Figure 11-27 - Re-Assayed Samples Scatter Plots from Hole FX1D-0022 (SGS-2011).

Re-assayed Samples in ALS Lab

A total of 65 samples were reanalysed at the ALS Lab in Lima, Peru during 2011. These samples were selected from 15 drill holes and sent to the ALS laboratory to check the original assay results from (SGS LabGS GEOSOL Laboratórios Ltda. Vespasiano, Mato Grosso, Brazil). A total of 19 samples returned with values outside of 20% absolute difference between two sets of data (Figure 11-28).

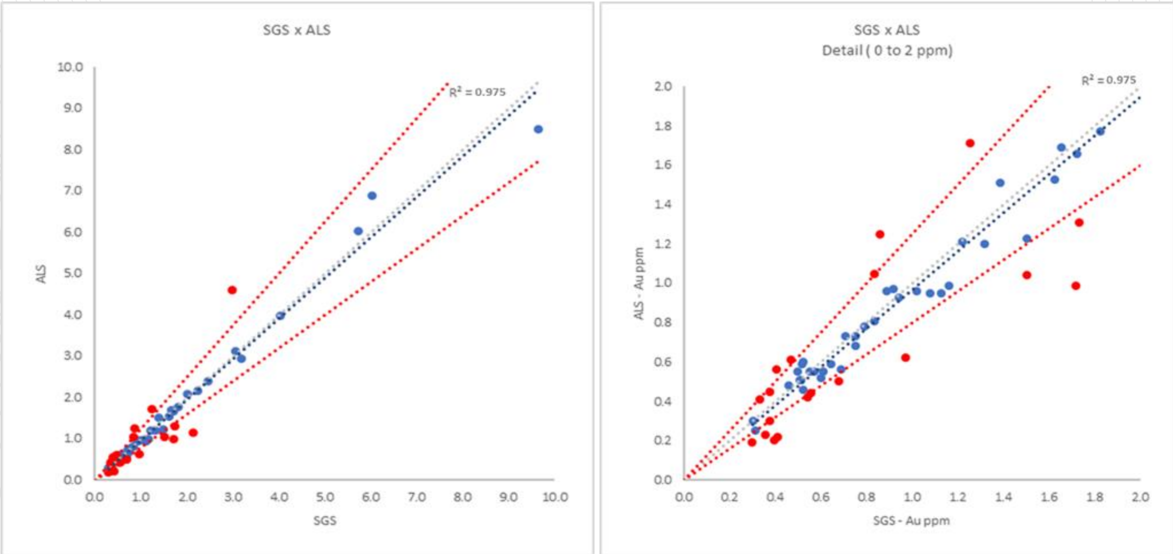


Figure 11-28 - ALS Lab Check Assays Scatter Plots (2011).

12. DATA VERIFICATION

Drilling on the Matupá Gold Project has been completed in various campaigns since 1996 by WMC, Rio Tinto (RTZ), Crescent Resources (CRESCENT), Vale, Mineração Santa Elina (MSE), Rio Novo. The implemented drilling methods were diamond core, reverse circulation, and auger drilling. Only diamond drill holes were used in previous Mineral Resource estimation and for the current Mineral Resource estimation for the X1 Deposit. Therefore, data verification since 2008 is only related to information from diamond drill holes.

In 2019, Aura began exploration activities with the use of RC drilling and diamond core drilling. None of these holes were drilled inside the current X1 Deposit boundary.

Data verification for the X1 Deposit has been performed at three different time periods since 2010 when Rio Novo became the Property owner. For the purpose of this report, only data verification for Au was summarized. The data verification measures used are summarized below.

12.1 DATA VERIFICATION BY GEOSIM INC. (2010)

An independent consultant (Mr. Ron Simpson) from Geosim Inc, Canada reviewed the procedures of Rio Novo and SGS Laboratory who performed the core sampling, preparing and assaying samples.

A site inspection was performed by R. Simpson on March 18, 2009. The inspections included examination of drill sites, drill core and surface outcrops as well as a review of sample preparation and QA/QC procedures. The author has also reviewed the geological information and reports from previous programs and other relevant data available at the company offices. Assay certificates have been examined and compared to the database. The author did not personally collect samples from the site.

In opinion of author of the 2010 technical report, Mr. Ron Simpson (Geosim Inc,), the programs and the data have been conducted and gathered in a professional and ethical manner and conform to standards acceptable within the industry.

12.2 DATA VERIFICATION BY RIO NOVO (2011)

12.2.1 RE-ASSAYING OF HISTORICAL DRILL HOLES

The drill hole SEX1-01 which was drilled by MSE and originally assayed by ALS Lima, Peru was sent to SGS Vespasiano laboratory in MG-Brazil for re-assaying by Rio Novo. A total of 293 samples from this hole were re-assayed. The result of this re-assaying is shown in Figure 12-1. Table 12-1 summarizes statistics of the Rio Novo re-assaying for the SEX1-01 drill hole.

Rio Novo included 15 blank samples in the sample stream to validate the SGS re-assaying. Blank samples with results above three times the detection limit is re-assayed as well (Figure 12-2).

The re-assaying results showed that 173 out of 293 re-assayed samples (Table 12-1) are located outside of the 20% error limit. This is considered to be a significant deviation that may cause concern especially since the re-assaying was performed on the pulp material, not from ½ of the drill core as the first assaying was.

However, the mean of re-assayed samples (2.37 g/t) is near to the mean of the original assay (2.42 g/t) which shows only a 2% difference and slight bias to a higher grade in ALS assay results for this hole. The correlation coefficient for re-assaying of this hole is about 0.94 and shows above 90% assurance.

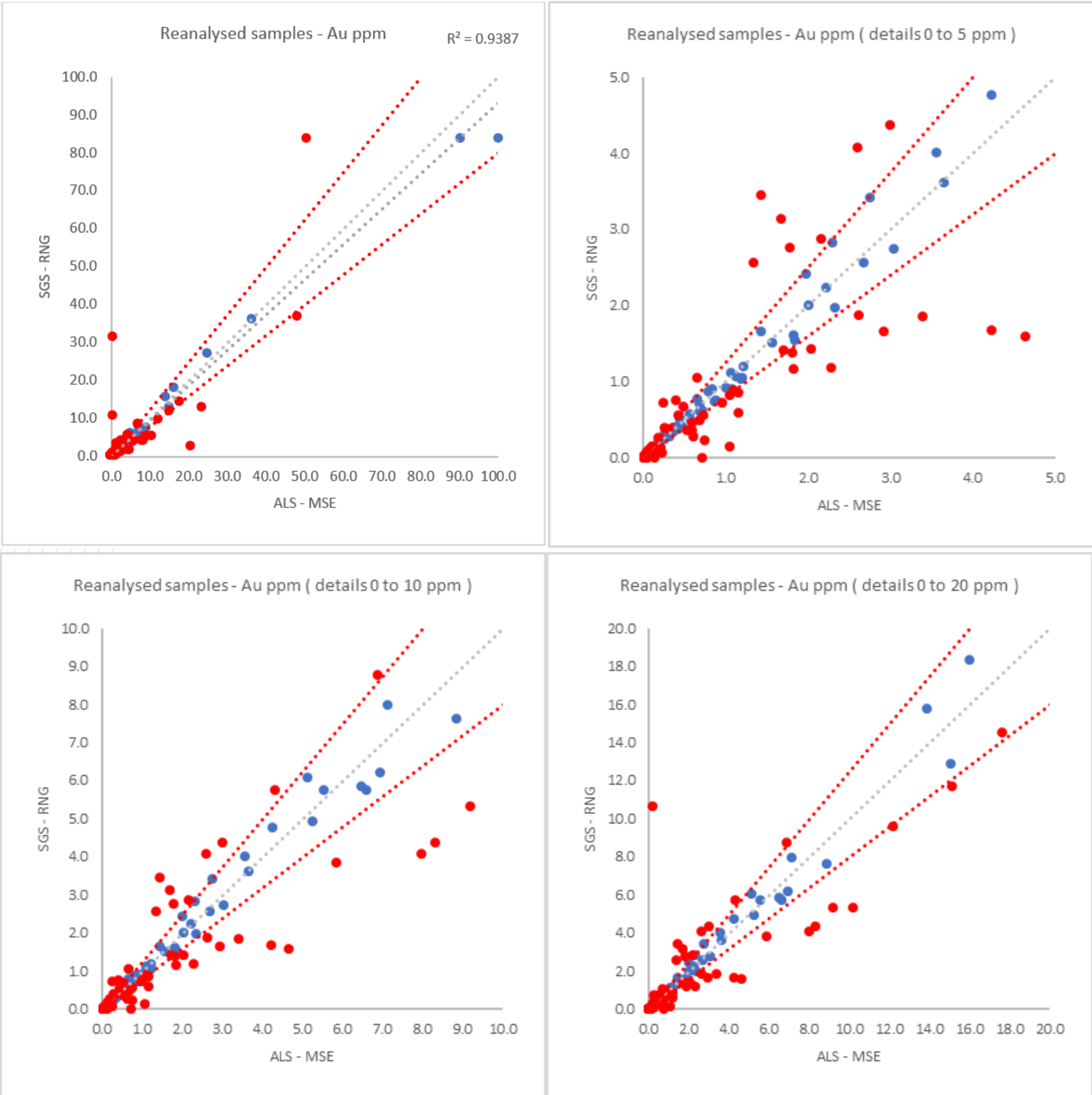


Figure 12-1 - ALS Versus SGS Lab Re-Assaying (Rio Novo 2011) Showing for Different Grade Thresholds.

Table 12-1 - Summary Statistics of Re-assaying for SEX1-01 (Rio Nov 2011).

Au_FAA505_ppb	Original -ALS	Reassay - SGS
# of Analysis above Threshold	293	
# of Analysis below Threshold	0	
# Outside Warning Limit	240	
# Outside Error Limit	173	
Mean	2.4206	2.3689
Median	0.07	0.055
Min	0.01	0.005
Max	100	83.593
Range	99.99	83.588
Variance	90.6848	88.6556
Coeff. of Var.	3.9407	3.9816
Sample Std. Dev.	9.5391	9.4318
Bias	-0.0214	
Corr. Coeff.	0.9387	
RMA Error on Y Intercept	0.1927	

Au_FAA505_ppb	Original -ALS	Reassay - SGS
RMA Error on Slope	0.0196	
RMA Error	4.6879	
Mean AMPRD	40.6879	
% of Population with AMPRD > 5%	90.4437	
% of Population with AMPRD > 10%	83.959	
% of Population with AMPRD > 15%	70.6485	
% of Population with AMPRD > 20%	62.4573	

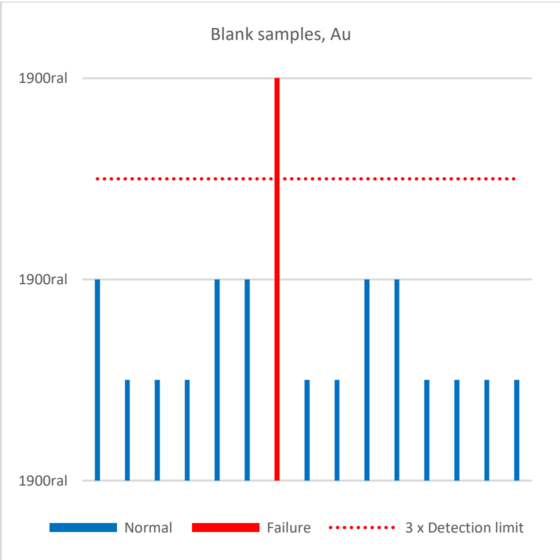


Figure 12-2 - ALS Versus SGS Lab Re-assaying (Rio Novo 2011) - Blank Performance.

12.2.2 RIO NOVO QUALITY CONTROL AND VERIFICATION PROCEDURES

Rio Novo had in place standard operating procedures and quality controls on the complete process of sampling, assaying, and data management including:

- Sampling procedures from source to final bagging;
- Replicate sampling;
- Check sampling;
- On-site sample preparation procedures;
- Sample coding
- On-site packing;
- Sample transport and delivery to laboratory;
- Sample reception at laboratory;
- Sample preparation at laboratory;
- Assaying protocols;

- Standards, blanks, and duplicate procedures;
- Internal (Rio Novo) and external (SGS) checking of assay results;
- Assay result reporting;
- Acceptance procedures of assay results;
- Sample decoding;
- Data archiving;
- Data processing and management; and,
- Storage of remaining coarse and pulverized rejects.

Many of these items are discussed in more detail in Section 11 of this report.

Rio Novo has implemented a monthly QA/QC report which provides clear, real time monitoring of both internal and external QA/QC procedures in exploration. It means that sample collection, sample preparation or laboratory errors can be identified and resolved in the quickest possible time. These reports were reviewed by the QP.

Rio Novo personnel conducted regular quality control inspections of company facilities on site. In addition, Rio Novo personnel have visited the laboratories on a regular basis to inspect for the quality of assay results. It was the opinion of Rio Novo that the company's standard operating procedures and quality control mechanisms meet the state-of-the-art or current best practices within the industry. In the opinion of Rio Novo, the data collected is of high quality and secure, and forms a solid basis for the feasibility work. The QP agrees with this assessment.

Rio Novo has implemented a monthly QA/QC report which provides clear, real-time monitoring of both internal and external QA/QC procedures in exploration. It means that sample collection, sample preparation or laboratory errors can be identified and resolved in the quickest possible time. These reports were reviewed by the QP.

12.3 DATA VERIFICATION BY AURA (2020 – 2021)

The underlaying data from previous drilling campaigns was verified by Aura's geologist and database manager under supervision of Aura's Director Mineral Resources, Farshid Ghazanfari, P.Geo., QP of this report.

12.3.1 DATABASE AND DATA ENTRY

Table 12-2 summarizes all information related to different drilling campaigns, number of samples and detection limit for the X1 Deposit.

Table 12-2 - X1 Assay Database Status.

Deposit	Company	Number of Holes	METERS DRILLED	METERS SAMPLED	Samples	Detection Limit
X1	AMI	2	686.39	686.39	697	476
X1	MSE	63	14106.34	14083.94	8158	2863
X1	RNG	60	11469.66	11436.32	10318	4860
X1	Vale	18	3190.05	3189.8	**3190**	**2163**

Aura's Database Manager, under supervision of the QP, arranged for checks of the data entry of a portion of the gold assay data in the database against all of the laboratory certificates provided by previous operators. Table 12-3 summarizes the result of the checks.

The following corrections were also made in the database before use for modelling and Mineral Resource estimation:

- The sample PGO-GUAR-FD041-0012 was not included in the Vale's database and the sample PGO-GUAR-FD041-0013 had been entered with wrong the interval data. Both were fixed in Aura's database.
- A total of 6 samples which were greater than 10 g/t Au sent to same laboratory an analyzed by FAASCR (Screen Fire Assays) method. The results (Table 12-4) were confirmatory of original assays (Au_FAA50) therefore original assays were kept in the final data the RNG database,
- A total of 27 samples, analyzed in the ALS laboratory returned with 2 results for the same sample (Au_AA26_ppm - Au2_AA26_ppm) and the final assay result used in the database was the average for the two values (Table 12-5).

Samples with no assay values marked with special character in the database (Table 12-6).

Table 12-3 - X1 Deposit Assay Entry Checks.

Status	Sample Count
Samples with different entries	35
Samples without assay results	50
Samples truncated at 2 decimal places	0
Detection limit samples	10362
Samples not found in certificates	119
No sample ID *	0
Okay	22159
Grand Total	22363

Table 12-4 - Samples with Different Assay Methods (X1).

HOLEID	SAMPLEID	FROM	TO	Au_FAASCR_ppm	LABJOBNO	Au_FAA505_ppb	LABJOBNO
FX1D-0021	G-10649	60	61	18.11	VG1100127	19.283	VG1100109
FX1D-0021	G-10657	67	68	18.49	VG1100127	20.067	VG1100109
FX1D-0021	G-10702	107	108	9.92	VG1100127	12.099	VG1100109
FX1D-0021	G-10706	111	112	10.62	VG1100127	45.546	VG1100109
FX1D-0021	G-10742	143	143.7	28.96	VG1100127	19.686	VG1100109
FX1D-0033	G-15786	104	105	5.06	VG1100269	14.464	VG1100232

Table 12-5 - Samples with Two Different Assay Values (X1).

HOLEID	SAMPLEID	FROM	TO	Au_AA26_ppm	Au_AA26_ppm	Au_AA26_AVG_ppm	LABJOBNO
SEX1-05	20330	55	56	12.3	13.1	12.7	BH08035257
SEX1-05	20344	68	69	1.06	2.53	1.795	BH08035257
SEX1-05	20348	72	73	0.53	0.15	0.34	BH08035257
SEX1-05	20370	92	93	1.94	2.71	2.325	BH08035257
SEX1-05	20371	93	94	9.88	9.47	9.675	BH08035257
SEX1-05	20372	94	95	1.95	1.51	1.73	BH08035257
SEX1-05	20376	97	98	0.15	0.18	0.165	BH08035257
SEX1-05	20390	111	112	42.4	26.4	34.4	BH08035257
SEX1-05	20393	114	115	0.57	0.64	0.605	BH08035257
SEX1-05	20414	133	134	71.9	82.7	77.3	BH08035257

HOLEID	SAMPLEID	FROM	TO	Au_AA26_ppm	Au_AA26_ppm	Au_AA26_AVG_ppm	LABJOBNO
SEX1-12	20033	9	10	0.09	0.21	0.15	BH08036127
SEX1-12	20070	44	45	14.7	14.8	14.75	BH08036127
SEX1-12	20072	46	47	5.09	5.32	5.205	BH08036127
SEX1-12	20109	80	81	8.66	9.43	9.045	BH08036127
SEX1-12	20168	135	136	6.95	6.05	6.5	BH08036127
SEX1-12	20186	151	152	27	22	24.5	BH08036127
SEX1-12	20195	160	161	0.64	0.6	0.62	BH08036127
SEX1-12	20196	161	162	1.08	1.66	1.37	BH08036127
SEX1-32	24249	68	70	9.07	9.05	9.06	BH08095666
SEX1-32	24262	90	92	7.77	8.02	7.895	BH08095666
SEX1-32	24272	108	110	2.79	2.45	2.62	BH08095666
SEX1-48	47550	51.6	54	2.29	2.18	2.235	BH08127894
SEX1-59	49933	144	146	2.48	1.71	2.095	BH08161169
SEX1-59	49934	146	148	6.07	9.37	7.72	BH08161169
SEX1-59	49939	154	156	2.91	2.08	2.495	BH08161169
SEX1-59	49940	156	158	5.35	3.75	4.55	BH08161169
SEX1-59	49946	168	170	61.8	94.8	78.3	BH08161169

Table 12-6 - Samples with Assay Values (X1).

HOLEID	SAMPLEID	FROM	TO
FD039	PGO-GUAR-FD039-0151	150	151
FD039	PGO-GUAR-FD039-0152	151	152
FD039	PGO-GUAR-FD039-0153	152	153
FD039	PGO-GUAR-FD039-0154	153	154
FD039	PGO-GUAR-FD039-0155	154	155
FD039	PGO-GUAR-FD039-0156	155	156
FD039	PGO-GUAR-FD039-0157	156	157
FD039	PGO-GUAR-FD039-0158	157	158
FD039	PGO-GUAR-FD039-0159	158	159
FD039	PGO-GUAR-FD039-0160	159	160
FD039	PGO-GUAR-FD039-0161	160	161
FD039	PGO-GUAR-FD039-0162	161	162
FD039	PGO-GUAR-FD039-0163	162	163
FD039	PGO-GUAR-FD039-0164	163	164
FD039	PGO-GUAR-FD039-0165	164	165
FD039	PGO-GUAR-FD039-0166	165	166
FD039	PGO-GUAR-FD039-0167	166	167
FD039	PGO-GUAR-FD039-0168	167	168
FD039	PGO-GUAR-FD039-0169	168	169
FD039	PGO-GUAR-FD039-0170	169	170
FD039	PGO-GUAR-FD039-0171	170	171
FD039	PGO-GUAR-FD039-0172	171	172
FD039	PGO-GUAR-FD039-0173	172	173
FD039	PGO-GUAR-FD039-0174	173	174
FD039	PGO-GUAR-FD039-0175	174	175
FD039	PGO-GUAR-FD039-0176	175	176
FD039	PGO-GUAR-FD039-0177	176	177
FD039	PGO-GUAR-FD039-0178	177	178
FD039	PGO-GUAR-FD039-0179	178	179
FD039	PGO-GUAR-FD039-0180	179	180
FD039	PGO-GUAR-FD039-0181	180	181
FD039	PGO-GUAR-FD039-0182	181	182
FD039	PGO-GUAR-FD039-0183	182	183
FD039	PGO-GUAR-FD039-0184	183	184

HOLEID	SAMPLEID	FROM	TO
FD039	PGO-GUAR-FD039-0185	184	185
FD039	PGO-GUAR-FD039-0186	185	186
FD039	PGO-GUAR-FD039-0187	186	187
FD039	PGO-GUAR-FD039-0188	187	188
FD039	PGO-GUAR-FD039-0189	188	189
FD039	PGO-GUAR-FD039-0190	189	190
FD039	PGO-GUAR-FD039-0191	190	191
FD039	PGO-GUAR-FD039-0192	191	192
FD039	PGO-GUAR-FD039-0193	192	193
FD039	PGO-GUAR-FD039-0194	193	194
FD039	PGO-GUAR-FD039-0195	194	195
FD039	PGO-GUAR-FD039-0196	195	196
FD039	PGO-GUAR-FD039-0197	196	197
FD039	PGO-GUAR-FD039-0198	197	198
FD039	PGO-GUAR-FD039-0199	198	199
FD039	PGO-GUAR-FD039-0200	199	200.4

12.3.2 REVIEWING OF DRILLING AND GEOLOGICAL LOGS

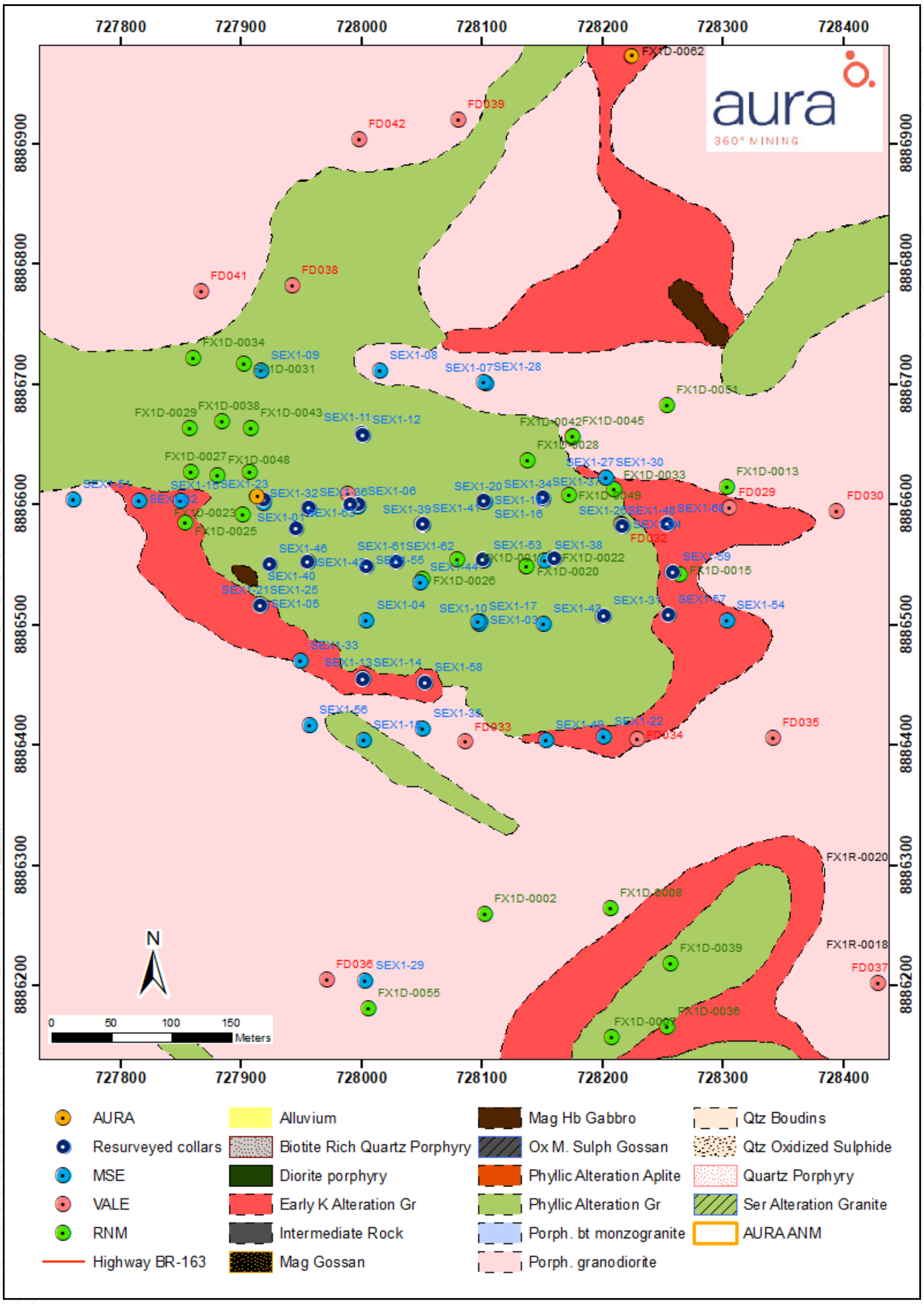
Digital log files with geological descriptions for drill holes from Vale were not located for verification. The data used in the Aura database are the same as those present in the database provided by Vale. All drill holes which were drilled by MSE have the original logs with geological description and information regarding sampling. This data is compatible with what is recorded in the database. All dill hole logs referring to the RNG (Rio Nov) holes are also available and the information is compatible with that used in the database. No overlapping or undescribed intervals were observed for these drill holes.

12.3.3 DRILL HOLE COLLAR LOCATION SURVEYS

In 2021, Aura selected 33 drill holes in the X1 Deposit area which were drilled inside of the estimated model for validation of collar surveys (Figure 12-3). Drill collars for these drilled holes were spotted in the field and photographed (Figure 12-4). A geodetic survey instrument, with centimetric scale precision, was used to resurvey drill hole collar locations found in the field.

Table 12-7 shows original and re-survey data and any difference in distance between two the points, original versus resurveyed, for X, Y coordinates and for elevation (Z) value separately. Most of the resurveyed drill holes came back with small differences (typically below 1 m) compared to the original values of the validated database. Two holes SEX1-06 and SEX1-38 show higher differences (6.68 m and 7.88 m respectively) in the Easting (X) compared to original values. This is probably due to errors in original surveys as the second survey was considered to be more accurate and supervised by Aura's geologists and reviewed in detail by the QP.

In the opinion of the QP, the difference in location of a few holes as described above is not material to the Mineral Resource estimation. Therefore, the validated database can be used for purpose of this study.



Note: Coordinates are South American Datum (1969), UTM Zone 21 South.

Figure 12-3 - Geology Map Showing Location of Drill Hole with Re-surveyed Collars.

Figure 12-4 - Collar Locations of Selected Drill Holes for Validation (MSE Drilling 2008) in X1 Deposit (SEX1-42 and SEX1-44).

Table 12-7 - Results of Drill Hole Collar Re-surveying (Aura 2021).

Hole	Original Collar Survey (UTM SAD69 S21)			Re-Survey (UTM SAD69 S21)			Difference (Meters)		
	X	Y	Z	X	Y	Z	X	Y	Z
FD031	728103.508	8886598.82	384.447	728104.023	8886599.03	384.257	0.52	0.21	0.19
FD032	728218.158	8886580.19	370.791	728219.119	8886578.40	370.704	0.96	1.79	0.09
SEX1-05	727918.473	8886512.58	359.403	727918.250	8886512.43	359.160	0.22	0.16	0.24
SEX1-06	727999.464	8886594.15	398.093	727992.829	8886596.65	398.237	6.64	2.50	0.14
SEX1-11	728002.155	8886655.52	370.056	728002.069	8886655.76	370.630	0.09	0.24	0.57
SEX1-12	728002.857	8886653.62	370.385	728003.072	8886653.70	370.494	0.22	0.08	0.11
SEX1-13	728003.451	8886451.60	342.437	728003.529	8886450.91	341.960	0.08	0.69	0.48
SEX1-16	728104.429	8886598.01	384.415	728104.023	8886599.03	384.257	0.41	1.02	0.16
SEX1-19	728104.487	8886598.69	384.446	728104.023	8886599.03	384.257	0.46	0.34	0.19
SEX1-20	728103.633	8886598.43	384.436	728104.023	8886599.03	384.257	0.39	0.60	0.18
SEX1-23	727920.919	8886600.03	381.372	727920.852	8886600.07	381.322	0.07	0.04	0.05
SEX1-24	728218.923	8886578.63	370.727	728219.119	8886578.40	370.704	0.20	0.24	0.02
SEX1-26	728218.942	8886577.93	370.764	728219.119	8886578.40	370.704	0.18	0.47	0.06
SEX1-31	728202.717	8886503.29	370.972	728203.213	8886503.02	371.713	0.50	0.28	0.74
SEX1-32	727958.395	8886592.72	394.952	727958.583	8886593.07	394.655	0.19	0.35	0.30
SEX1-34	728152.561	8886600.46	383.119	728152.706	8886600.52	383.009	0.15	0.07	0.11
SEX1-36	727958.407	8886593.33	394.769	727958.583	8886593.07	394.655	0.18	0.25	0.11
SEX1-37	728152.580	8886601.67	382.957	728152.666	8886601.87	382.892	0.09	0.19	0.06
SEX1-38	728154.534	8886549.87	389.618	728162.416	8886551.06	389.864	7.88	1.20	0.25
SEX1-39	728052.590	8886579.20	390.691	728052.750	8886579.08	390.459	0.16	0.12	0.23
SEX1-40	727958.039	8886548.79	392.672	727957.552	8886548.78	392.549	0.49	0.01	0.12
SEX1-41	728052.501	8886580.15	390.748	728052.591	8886580.38	390.559	0.09	0.23	0.19
SEX1-46	727925.703	8886546.77	380.259	727925.594	8886546.72	380.384	0.11	0.06	0.13
SEX1-48	728218.936	8886578.30	370.803	728219.119	8886578.40	370.704	0.18	0.10	0.10
SEX1-53	728103.838	8886549.90	387.910	728102.960	8886550.63	388.227	0.88	0.72	0.32
SEX1-55	728005.797	8886544.60	391.852	728005.865	8886544.62	391.841	0.07	0.01	0.01
SEX1-57	728256.784	8886504.68	352.571	728256.776	8886504.87	352.502	0.01	0.19	0.07
SEX1-58	728054.360	8886448.17	338.853	728054.541	8886448.13	338.935	0.18	0.04	0.08
SEX1-59	728260.248	8886540.05	356.713	728260.375	8886540.32	356.555	0.13	0.27	0.16
SEX1-60	728255.824	8886580.22	354.892	728255.943	8886580.40	354.929	0.12	0.18	0.04
SEX1-61	728031.013	8886548.92	389.746	728031.100	8886548.55	389.429	0.09	0.37	0.32
SEX1-62	728031.012	8886548.57	389.439	728031.100	8886548.55	389.429	0.09	0.02	0.01
SEX1-63	727948.144	8886576.30	394.679	727948.006	8886576.31	394.469	0.14	0.01	0.21

12.3.4 DRILL HOLE DOWN HOLE SURVEYS

The diamond core drilling by Vale does not have downhole survey data, most of them were drilled at -55° dip, 270° azimuth. Santa Elina and Rio Novo drill holes were surveyed downhole using a Maxibor survey instrument. However, the original survey raw data for MSE holes were not found. The Rio Novo (RNG) raw downhole survey data was compared to the database, and all were confirmed against the original data.

12.3.5 REVIEWING OF ASSAY DATA IN X1 DATABASE

Different laboratories were responsible for the samples' preparation and analysis during the management of each company. The assay results were provided by these laboratories and all the samples in the database have the final and validated information. The certificates were all validated and compared to the assay data to ensure that the authenticity of the information included in the database is the same as provided by the analysis laboratories. Table 12-8 shows the laboratories and analysis methods used.

Table 12-8 - X1 Database Assay Data Source and Detection Limits (Aura 2021).

Project	Company	Laboratory	Method	Lower detection limit (ppm)
X1	Vale	CVRD	VS	0.05
X1	Vale	SGS	FA30	0.01
X1	MSE	ALS	AA26	0.01
X1	RNG	SGS	FAA505	0.005
X1	AMI	SGS	FAA505	0.005

Aura selected 18 drill holes from the Vale data; 63 drill holes from the MSE data; 60 drill holes from RNG for validation of assays. This represented 100% drill hole data in the X1 project. A first step validation to detect gaps or numerical errors in the sampling intervals found no errors. Next, the assay information in the database was compared to the original certified assay reports from the lab. All assay data was correct.

Santa Lina (MSE) assay data with results below the detection limit were included in the database with the same value reported, however, Rio Novo (RNG) replaced all below detection data with the lower detection limit values as is shown in Table 12-9. For example, <0.01 ppm was included in the MSE drilling results but was replaced by 0.01 but following the RNG rule the value should be 0.0099.

Table 12-9 - X1 Database- Detection Limit Adjustments (Rio Novo 2012).

DSC	<0.01	<0.05	<10	<5	>100
VALUE	0.01	0.05	10	5	100
RULE	0.0099	0.0499	9.9	4.99	100.01

All analysis results were validated and imported into the Aura database, as well as all information regarding the analysis methods used, the detection limit information is also available, resulting in the best-accepted rule for the database. In Aura, the rule used for results below the detection limit is to use the return value divided by two.

A total of 6 samples has Au results for FAA505 and FAASCR methods. The second results should have a greater priority in the assay rank but in the RNG the assay results considered are for the method FAA505. FAASCR is used for samples with Au ppm greater than 10 ppm in the FAA505 (Table 12-10).

Table 12-10 - Examples of Assay Methods Differences in X1 Database (Rio Novo 2012).

Hole	Sampling_ID	From	To	Au_FAA505_ppm	Au_FAASCR_ppm
FX1D-0021	G-10649	60	61	19.283	18.11
FX1D-0021	G-10657	67	68	20.067	18.49
FX1D-0021	G-10702	107	108	12.099	9.92
FX1D-0021	G-10706	111	112	45.546	10.62
FX1D-0021	G-10742	143	143.7	19.686	28.96
FX1D-0033	G-15786	104	105	14.464	5.06

12.3.6 REVIEWING OF MSE AND RIO NOVO IMPLEMENTED QA/QC DATA

The quality controls and quality assurance measures applied to the drilling data generated by the MSE and RNG companies were reviewed to make sure that they meet the standard of the best practice in the industry and to guarantee data reliability. MSE used standard and blank samples in the sampling flow to validate the analytical results received from the contracted laboratories. RNG has a QA/QC protocol that is more complete using standards, blanks, core duplicates, and check assay to validate all the drilling sample results. The best drill holes in X1 drilled by MSE had the pulp samples reanalyzed for gold to validate the original data received. The correlation from original and re-assay is greater than 90% and considered good.

12.3.7 NEW TOPOGRAPHICAL SURVEY

In December 2020, Aura has carried out a topographical survey with the use of an unmanned Aerial Vehicle (UAV) covering the X1 Deposit region and its surroundings for project engineering purposes. Due to dense vegetation on the hill, it was necessary to improve the data by carrying a detailed ground topographical survey with the use of RTK GPS forming a grid of 50 m by 50 m resolution, completed on October 2021. In order to unite the products, UAV and ground RTK GPS, an integration was performed on October 2021 by GEO LINE Serviços Minerais (consulting company) aligning the 88 photographs from the UAV survey (cloud of points), construction of the dense cloud followed by generation of a 3-D model and Digital Elevation Model (DEM) using Agisoft Photoscan software (Agisoft Metashape Professional). With the use of GEOVIA software the consulting company integrated the DEM file from the UAV survey with the ground RTK GPS survey data, creating a Digital Terrain Model (DTM) which is more accurate. A DTM provides the elevation of the bare earth, without vegetation. The final procedure in GEOVIA software was to generate integrated level curves and the generation of a new DTM. Both topographical surveys used the South American Datum 1969 – UTM zone 21S to match the historical exploration database of the Matupá Project. The final integrated topographical base was also converted to SIRGAS 2000, which was defined as the official Brazilian Datum since 2015 and will be used for basic engineering, layouts and environmental purposes. The integration processing parameters included 87,420 points resulting in an RMS reprojection error of 0.154072.

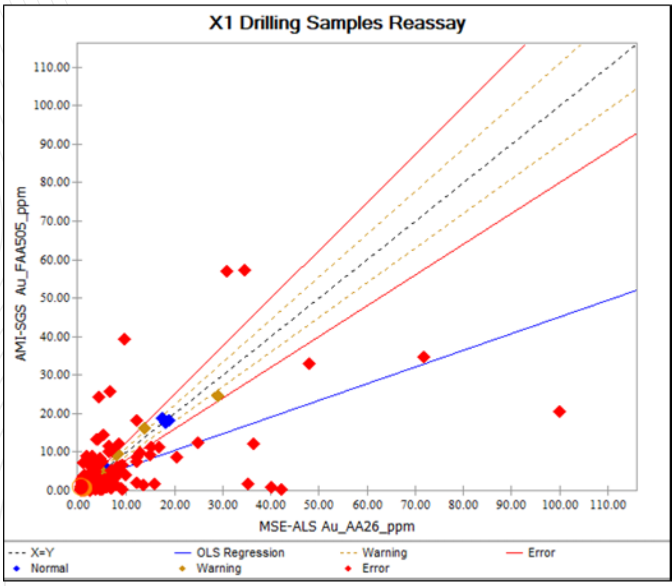
The QP reviewed the new topographical survey versus the topography dated back to 2010 which was used for other DTM surfaces and did not find any significant differences especially close to the X1 area and deposit boundaries. The re-surveyed collars matching the new topographical survey.

12.3.8 RE-SAMPLING OF SELECTED CORE INTERVALS FROM MSE AND RNG (2022)

Aura Minerals conducted a re-sampling of selected intervals from previous drilling campaigns in December 2021-January 2022. For this purpose, ¼ core and sampled and sent to SGS laboratory (SGS lab in Vespasiano-MG, Brazil) to further validate underlying data and investigate any bias in the ALS and SGS laboratories results during the Mineração Santa Elina and Rio Novo Mineração campaigns.

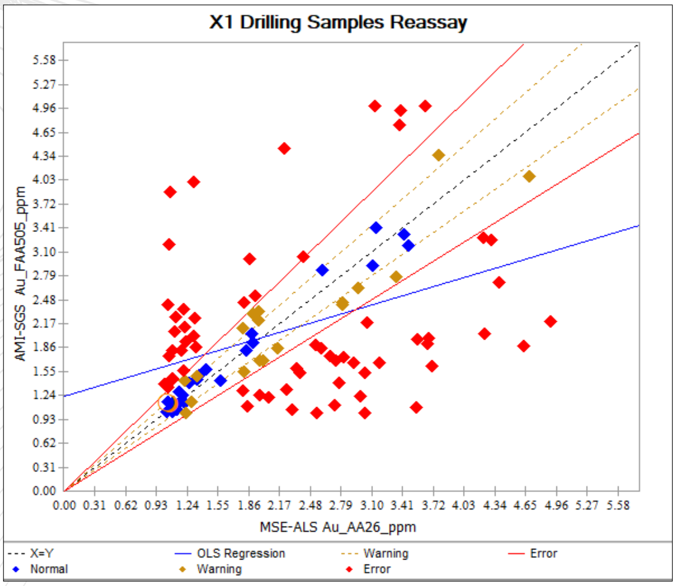
The activity comprised of the selection of 304 samples from Mineração Santa Elina (MSE) drilling, representing 3.73% of their historical drill hole data (8,158 samples) to be re-assayed on SGS Laboratories for comparison and validation of the previous analysis done in the ALS and also SGS laboratories. The selection of samples considered 20 of the 63 drill holes done by MSE at X1 Deposit, being all of them inside the Mineral Resource model. Besides their spatial distribution, the grade of the samples was also considered when conducting sample selection for re-assay, with the focus on higher grades of gold inside the model.

Figures 12-5, 12-6, and 12-7 are showing the results of this re-sampling for Santa Elina (MSE) core samples which are analyzed in ALS laboratory versus analyzed in SGS laboratory for Aura in 2022.



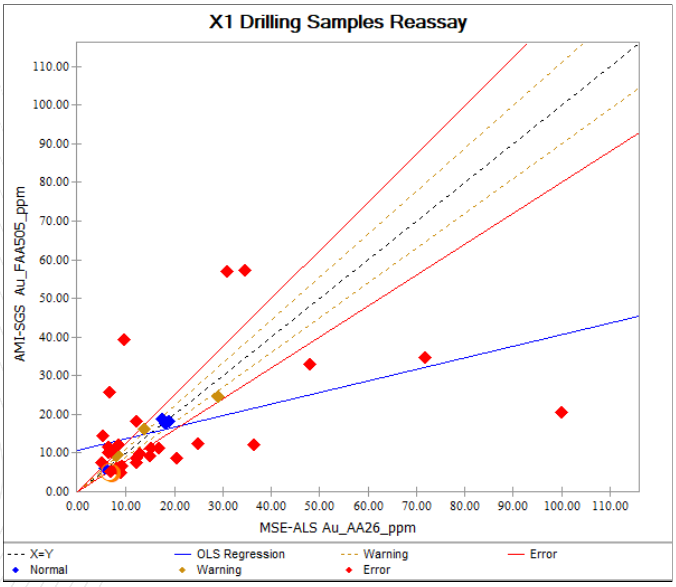
	Original	Check
# of Analyses above Threshold	304	
# of Analyses below Threshold	0	
# Outside Warning Limit	257	
# Outside Error Limit	207	
Mean	4.5027	3.5824
Median	1.335	1.317
Min	0.74	0.098
Max	100	57.252
Range	99.26	57.154
Variance	89.4144	46.7119
Coeff. of Var.	2.1035	1.911
Sample Std. Dev.	9.4715	6.8459
Bias	-0.2044	
Corr. Coeff.	0.601	
RMA Error on Y Intercept	0.3581	
RMA Error on Slope	0.0381	
RMA Error	10.4158	
Mean AMPRD	52.6577	
% of Population with AMPRD > 5%	93.0921	
% of Population with AMPRD > 10%	86.1842	
% of Population with AMPRD > 15%	78.2895	
% of Population with AMPRD > 20%	70.3947	

Figure 12-5 – ALS (MSE) Versus SGS Lab Re-Assaying (Aura 2022).



	Original	Check
# of Analyses above Threshold	103	
# of Analyses below Threshold	0	
# Outside Warning Limit	80	
# Outside Error Limit	62	
Mean	2.2291	2.0797
Median	1.96	1.849
Min	1.02	1.006
Max	4.91	4.995
Range	3.89	3.989
Variance	1.076	0.9346
Coeff. of Var.	0.4676	0.4671
Sample Std. Dev.	1.0424	0.9715
Bias	-0.067	
Corr. Coeff.	0.411	
RMA Error on Y Intercept	0.2045	
RMA Error on Slope	0.0866	
RMA Error	1.5421	
Mean AMPRD	36.6415	
% of Population with AMPRD > 5%	92.233	
% of Population with AMPRD > 10%	81.5534	
% of Population with AMPRD > 15%	70.8738	
% of Population with AMPRD > 20%	60.1942	

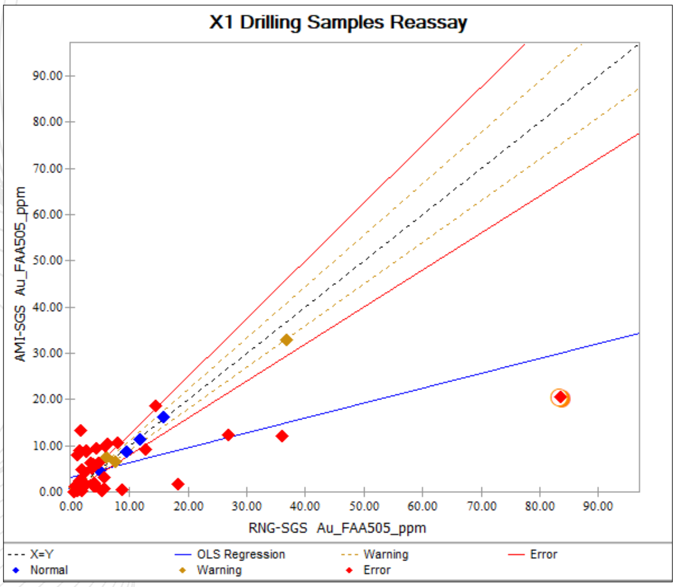
Figure 12-6 – ALS (MSE) Versus SGS Lab Re-Assaying (Aura, 2022) Showing for between 1 g/t and 5 g/t Thresholds.



	Original	Check
# of Analyses above Threshold	37	
# of Analyses below Threshold	0	
# Outside Warning Limit	33	
# Outside Error Limit	29	
Mean	18.6203	16.2881
Median	12.3	11.329
Min	5.11	5.037
Max	100	57.252
Range	94.89	52.215
Variance	366.528	162.282
Coeff. of Var.	1.0424	0.7929
Sample Std. Dev.	19.409	12.9147
Bias	-0.1252	
Corr. Coeff.	0.4493	
RMA Error on Y Intercept	2.5099	
RMA Error on Slope	0.1165	
RMA Error	24.1751	
Mean AMPRD	45.8541	
% of Population with AMPRD > 5%	91.8919	
% of Population with AMPRD > 10%	89.1892	
% of Population with AMPRD > 15%	83.7838	
% of Population with AMPRD > 20%	78.3784	

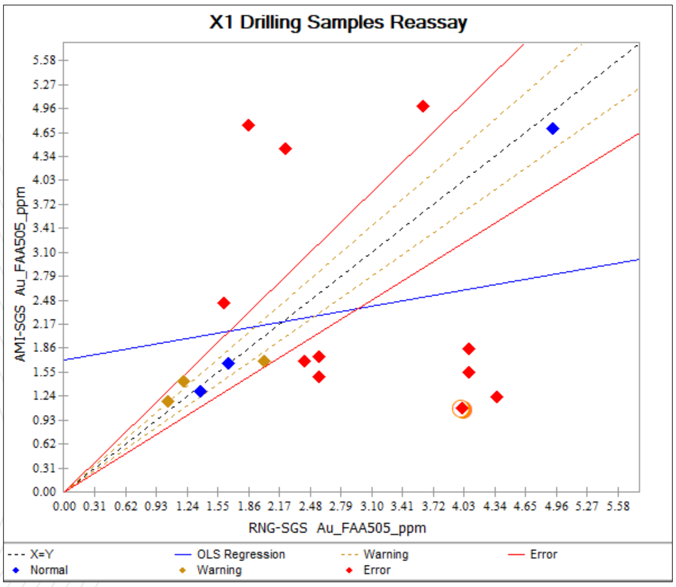
Figure 12-7 – ALS (MSE) Versus SGS Lab Re-Assaying (Aura, 2022) Showing above 5 g/t Threshold.

Figures 12-8, 12-9 and 12-10 are showing the results of this re-sampling for Rio Novo (RNG) core samples which are analyzed in SGS laboratory in 2011 versus analyzed in SGS laboratory for Aura in 2022.



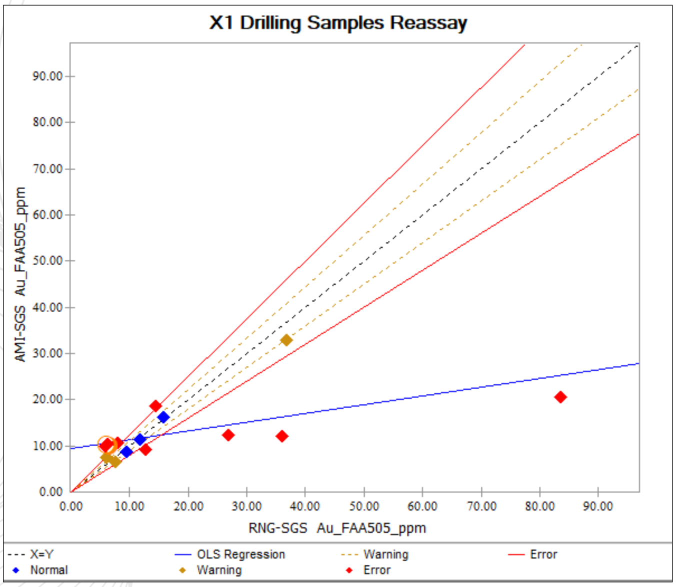
	Original	Check
# of Analyses above Threshold	61	
# of Analyses below Threshold	0	
# Outside Warning Limit	53	
# Outside Error Limit	43	
Mean	6.7991	5.3634
Median	2.635	1.849
Min	0.6	0.115
Max	83.593	32.849
Range	82.993	32.734
Variance	154.3427	37.0081
Coeff. of Var.	1.8424	1.1437
Sample Std. Dev.	12.5266	6.1339
Bias	-0.2112	
Corr. Coeff.	0.6584	
RMA Error on Y Intercept	0.6589	
RMA Error on Slope	0.0656	
RMA Error	11.5809	
Mean AMPRD	64.2582	
% of Population with AMPRD > 5%	93.4426	
% of Population with AMPRD > 10%	90.1639	
% of Population with AMPRD > 15%	81.9672	
% of Population with AMPRD > 20%	73.7705	

Figure 12-8 – SGS Lab (Rio Novo, 2011) Versus SGS Lab Re-Assaying (Aura 2022) .



	Original	Check
# of Analyses above Threshold	17	
# of Analyses below Threshold	0	
# Outside Warning Limit	14	
# Outside Error Limit	11	
Mean	2.6875	2.3078
Median	2.425	1.692
Min	1.044	1.09
Max	4.928	4.995
Range	3.884	3.905
Variance	1.4483	1.8896
Coeff. of Var.	0.4616	0.614
Sample Std. Dev.	1.2405	1.4169
Bias	-0.1413	
Corr. Coeff.	0.196	
RMA Error on Y Intercept	0.897	
RMA Error on Slope	0.2623	
RMA Error	2.4125	
Mean AMPRD	47.127	
% of Population with AMPRD > 5%	88.2353	
% of Population with AMPRD > 10%	82.3529	
% of Population with AMPRD > 15%	76.4706	
% of Population with AMPRD > 20%	64.7059	

Figure 12-9 – SGS Lab (RNG 2011) VersusSGS Lab Re-Assaying (Aura, 2022) Showing for between 1 g/t and 5 g/t Thresholds.



	Original	Check
# of Analyses above Threshold	14	
# of Analyses below Threshold	0	
# Outside Warning Limit	11	
# Outside Error Limit	8	
Mean	20.1326	13.349
Median	12.311	10.978
Min	5.843	6.433
Max	83.593	32.849
Range	77.75	26.416
Variance	413.848	44.4765
Coeff. of Var.	1.0486	0.5185
Sample Std. Dev.	21.1112	6.9208
Bias	-0.3369	
Corr. Coeff.	0.5759	
RMA Error on Y Intercept	1.9419	
RMA Error on Slope	0.1223	
RMA Error	19.8503	
Mean AMPRD	39.3214	
% of Population with AMPRD > 5%	85.7143	
% of Population with AMPRD > 10%	85.7143	
% of Population with AMPRD > 15%	71.4286	
% of Population with AMPRD > 20%	64.2857	

Figure 12-10 – SGS Lab (RNG 2011) VersusSGS Lab Re-Assaying (Aura, 2022) Showing above 5 g/t Threshold.

The re-assaying results (Figures 12-5 to 12-10) showed a 20% bias on the lower side for re-assayed intervals by SGS (Aura, 2022) compare to original assays. Considering that re-assays are based on ¼ core samples, this lower bias was expected. The nuggety nature of gold in the X1 especially in the strong phyllic zone is the main reason for such differences. However, the bias in analytical results also can not completely be ruled out.

The correlation coefficient for re-assaying is about 0.60 and 0.68 for re-sampling of Santa Elina (MSE) and Rio Novo (RNG) drill cores respectively.

In general, resampling of selected intervals showed a reasonable comparison between original assay data and resampled assays considering the nature of duplicate samples (1/4 core) and the nuggety nature of gold mineralization.

12.3.9 SILVER DETECTION LIMIT

Table 12-11 shows lower detection limits for analytical methods that were used for different drilling campaigns in the X1 Deposit. The ICP method mainly was used with variable detection limits to analyse the silver values. The percentage of data in database for each assay method is also shown in the table. The lower detection limit for all these methods is relatively high (< 2 ppm) and is half of the average grade of silver in the X1 Deposit. Analysis of silver values equal or lower than the detection limit of the analytical method may have more than 50% error.

In addition, there was no silver standard was inserted in the stream of QA/QC samples in any of the X1 drilling campaigns.

In the opinion of the QP, the silver values are informative at the best and can be considered as credit for X1 mineralization and Mineral Resource Estimate, but they do not have the accuracy required for a feasibility-level study to be the basis of an economic evaluation.

Table 12-11 – Silver Assay Analytical Methods in X1 Database.

COMPANY	METHOD	LDL (ppm)	LAB	PERCENTAGE IN THE DB
AMI	Pb_ICP40B_ppm	<8	SGS	3%
RNG	Pb_ICP12B_ppm	<3	SGS	48%
MSE	Pb_ICP40B_ppm	<8	SGS	1%
MSE	Pb_ICP61_ppm	<2	ALS	1%
MSE	Pb_ME-ICP61_ppm	<2	ALS	33%
VALE	Pb_AR_ppm	<3	VALE	12%
VALE	Pb_PL_ppm	<3	VALE	3%

12.4 CONCLUSION

The presence of gold at the X1 Deposit is supported by relatively close space drilling and sampling by MSE and Rio Novo. The underlying data are reliable enough and have been sufficiently verified by Rio Novo (2011-2012) and Aura (2020-2021) and the data verification level is adequate for purpose of this feasibility study.

The QP is satisfied that the exploration, sampling, security, and QA/QC procedures employed at the X1 Deposit from the Matupá Gold Project, and their results, are sufficient to produce data adequate for the purposes used in this technical report.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

This chapter outlines the results of metallurgical tests carried out for the Matupá Gold Project. Studies of mineralized material from X1 deposit and its properties were conducted through three test campaigns.

The first metallurgical campaign was carried out in 2018-2019, including metallurgical assessment and characterizations conducted by SGS-Geosol Laboratories, in Belo Horizonte, Brazil. The samples studied in the first campaign were also used for mineralogical characterization conducted at SGS Lakefield Laboratory - Lakefield City, Canada. At that time, the X1 Deposit was classified into three (3) ore types, which were used throughout these studies.

The first metallurgical campaign focused on assessing preliminary metallurgical flow sheets, together with the respective gold recovery for the tested samples. The main flow sheets assessed were as follows:

- Gravity separation followed by flotation and concentrate leaching.
- Gravity separation followed by leaching.

The reports addressing the above described assessments were as follows:

- SGS Mineral Service (Lakefield) - Mineralogical Study Report - Project 17013-01. Report: An Investigation by High-Definition Mineralogy into the Mineralogical Characteristics of Nine Composite Samples from the Almas and Matupá Gold Projects, February 7, 2019 (Reissued: August 10, 2021).
- SGS Geosol Laboratories Ltd. - Metallurgical Study Report - Project 3965 - 1801- Final Report: Gravity Separation, Flotation and Leaching Testwork on Gold Ore Samples from the Almas and Matupá Deposits, September 20, 2019.

Further studies resulted in the reclassification of ore types i.e., Fresh Rock and Oxide, which were used in the second metallurgical campaign, the latter including various tests and assessments carried out by different companies, described as follows:

- Mineral Processing Solutions Ltda. (MinPro Solutions) - Characterization of the comminution parameters. Simulations and sizing for crushing and grinding circuits. São Paulo, Brazil;
- Testwork Desenvolvimento de Processo Ltda. (Testwork Lab.) - Sample preparation, metallurgical testing (gravity, flotation, leaching) and cyanide destruction. Nova Lima, Brazil;
- SGS Geosol Laboratorios Ltda. (SGS Geosol Lab.) - Geochemical and environmental analysis and characterization. Vespasiano, Brazil;
- FLSmidth Ltd. (Knelson Division) - Modeling and sizing for gravity and intensive leaching circuit options. British Columbia, Canada;
- FLSmidth Brasil - Tests to characterize sedimentation, rheology, and filtration parameters. Sizing for the related equipment. Testing: Votorantim, Brazil. Sizing: Salt Lake City, USA; and
- COTEPROM (Consultoria e Assessoria em Processos Ltda.) - Test work design; follow-up and validation of studies; data processing and information reporting. São Paulo, Brazil.

The main objectives of the second metallurgical campaign were:

- Complete the characterization of the ore properties to be dealt with in the Project, determining the comminution parameters for industrial circuit sizing;
- Select the grinding size based on metallurgical recovery results, as well as operational technical aspects;
- Assess the contribution of the gravity circuit;
- Set process and design parameters for the leaching stage (cyanidation);
- Assess the efficiency of the SO_2/air method for destructing the residual cyanide contained in the tailings; and
- Generation of tailing samples for characterization of chemical and environmental aspects.

The third metallurgical campaign was carried out during 2021 and 2022, whose main objectives were:

- Consolidation:
    - Further develop a gold metallurgical recovery as a function of gold grade in blends associated to the Life of Mine (LOM) plan;
    - Confirm the metallurgical flow sheet and project design criteria based on tests carried out on blended samples. Consolidate the parameters associated with tailing filtering/dry stacking.
- Variability:
    - preliminary assessment of gold grade variability in the metallurgical performance for both Fresh Rock and Oxide ore types.

Here too, various tests and assessments were carried out by different companies described as follows:

- SGS Geosol Laboratorios Ltda. (SGS Geosol Lab.) – Sample preparation, comminution and metallurgical tests. Sample preparation and characterization for geochemistry and environmental. Vespasiano, Brazil;
- SGS Mineral Services / JKTech – SMC testing. Santiago, Chile;
- FLSmidth Brasil - Tests to characterize sedimentation, rheology, and filtration parameters. Sizing for the related equipment. Testing: Votorantim, Brazil. Sizing: Salt Lake City, USA;
- Pattrol - Investigações Geotécnicas Ltda. – Physical and geotechnical characterization for dry-staking. Belo Horizonte, Brazil;
- Jenike & Johanson – Tailing flow parameters. Vinhedo, Brazil; and
- COTEPROM (Consultoria e Assessoria em Processos Ltda.) - Test work program design; follow-up and validation of studies; data processing and information reporting. São Paulo, Brazil.

The results obtained in the three metallurgical campaigns carried out for the Matupá Project are detailed in chronological order in the following sections.

13.2 FIRST METALLURGICAL CAMPAIGN

13.2.1 SAMPLES AND PREPARATION

Representative composite samples from the three selected ore types(only from X1 deposit) for the first campaign were received by SGS-Geosol laboratory for conducting metallurgical tests and characterization. The composite samples were as follows:

- High Sulfide;
- Low Sulfide; and
- Oxide.

The list of the drill core fragments that were selected to make up each one of the above described composites is detailed in the report issued by SGS-Geosol as highlighted in the previous section. Figure 13-1 shows photographs of fragments associated with each composite.

Figure 13-1 – Photographs of Fragments Used to Prepare the Composites.

13.2.2 MINERALOGICAL CHARACTERIZATION

The mineralogical characterization studies carried out on all three samples at the SGS Lakefield Laboratory aimed to identify: (a) the mineral in each sample and (b) the liberation of sulfide minerals, by using the following methods:

- X-Ray Diffraction Analysis (XRD);
- Modal Mineralogical Analysis (QEM/SCAN) - Focused on sulfides; and Chemical Analysis and Optical Microscopy.

Table 13-1 shows the modal (quantitative) mineralogy results obtained by QEM/SCAN for the three Matupá project composite samples. The concentration of pyrite (Py) was reported as 14.2% in the High Sulfide sample, as well as 0.04% in the Oxide sample. The same analysis indicated pyrite liberation higher than 95% for both High Sulfide and Low Sulfide composites.

Table 13-1 - Modal Mineralogy Results Obtained by QEMSCAN.

MODAL MINERALOGY VIA QEMSCAN - MATUPA COMPOSITES			
source	High Sulfide	Low Sulfide	Oxide
Pyrite	14,2	1,13	0,04
Chalcopyrite	0,01	0,04	0,00
Sphalerite	0,01	0,06	0,00
Other Sulphides	0,01	0,04	0,00
Quartz	53,2	65,3	63,2
K-Feldspar	0,79	5,67	0,04
Plagioclase	0,43	5,40	0,01
Biotite	0,08	0,11	0,03
Sericite/Muscovite	30,4	19,5	31,3
Clays	0,46	1,82	1,01
Chlorites	0,11	0,13	0,02
Titanite	0,00	0,00	0,00
Other Silicates	0,00	0,03	0,05
Fe Oxides	0,12	0,05	2,83
Goethite	0,06	0,02	1,26
Rutile	0,04	0,07	0,10
Ilmenite	0,00	0,00	0,00
Calcite	0,03	0,36	0,01
Dolomite	0,00	0,22	0,00
Siderite	0,00	0,00	0,11
Ankerite	0,00	0,00	0,00
Apatite	0,07	0,04	0,00
Other	0,01	0,02	0,00
Total	100,0	100,0	100,0

13.2.3 CHEMICAL CHARACTERIZATION

The chemical analyzes of the composites for gold grade was carried out using two methodologies: Fire Assay of nine aliquots and by Metallic Screen. Individual and averaged gold grade results are shown in Table 13-2, together with the results of mass balancing.

Table 13-2 - Gold Grades Resulting from Chemical Analysis.

HEAD ASSAY BY DIFFERENT METHODS - MATUPA COMPOSITES			
source	High Sulphide	Low Sulphide	Oxide
nine aliquots	1,21	0,58	1,96
metallic screen	1,10	0,56	1,76
original samples	1,08	0,74	1,84
mass balancing	1,15	0,62	1,85
CERTIFICATES OF CHEMICAL ANALYSIS: BM1800662 and BM1800663			

Results of additional chemical analysis on the composites carried out by SAG Geosol Laboratory are listed in Table 13-3.

Table 13-3 - Additional Chemical Analysis Results.

composite	SG g/cm³	S %	S^{2-} %	C %	C org %	C graf %	Ag ppm	Al %	As ppm	Ba ppm	Be ppm	Bi ppm	Ca %	Cd ppm
HIGH SULPHIDE	2,89	7,35	6,30	0,01	< 0,05	< 0,05	4	6,4	< 10	354	< 3	< 20	0,08	< 3
LOW SULPHIDE	2,69	0,62	0,56	0,13	< 0,05	< 0,05	< 3	6,0	< 10	684	< 3	< 20	0,32	< 3
OXIDE	2,70	0,04	< 0,05	0,01	< 0,05	< 0,05	< 3	6,0	< 10	824	< 3	< 20	< 0,01	< 3
composite	Co ppm	Cr ppm	Cu ppm	Fe %	Hg ppm	K %	La ppm	Li ppm	Mg %	Mn %	Mo ppm	Na %	Ni ppm	P %
HIGH SULPHIDE	23	< 3	190	7,8	0,08	3,41	< 20	34	0,30	0,02	51	0,19	< 3	0,02
LOW SULPHIDE	<8	4	463	1,8	0,14	3,39	21	17	0,31	0,05	8	0,68	< 3	< 0,01
OXIDE	<8	4	33	3,3	0,10	3,03	< 20	20	0,16	0,02	56	0,1	< 3	< 0,01
composite	Pb ppm	Sb ppm	Sc ppm	Se ppm	Sn ppm	Sr ppm	Th ppm	Ti %	Tl ppm	U ppm	V ppm	W ppm	Zn ppm	Zr ppm
HIGH SULPHIDE	29	< 10	< 5	< 20	< 20	15	< 20	0,07	< 20	< 20	19	< 20	131	75
LOW SULPHIDE	330	< 10	< 5	< 20	< 20	32	< 20	0,05	< 20	< 20	< 8	< 20	781	66
OXIDE	26	< 10	< 5	< 20	< 20	24	< 20	0,05	< 20	< 20	12	< 20	23	70
CERTIFICATES OF CHEMICAL ANALYSIS: BM1800626 BM1800686 and BM1800878														

13.2.4 GRINDING TESTS

Grinding times were determined for all three composite samples for obtaining the following P_{80} (80% passing particle size): 0.212 mm, 0.150 mm, 0.106 mm, and 0.075 mm. These same samples were also used in Bond Work Index (BWI) tests for ball milling using a 0.250 closing screen. The results are listed in Table 13-4.

Table 13-4 - BWI Results.

Sample	BWI (kWh/t)*
High Sulphide	16.1
Low Sulphide	15.3
Oxide	13.7

** 0.250 mm closing screen.*

13.2.5 CONCENTRATION AND LEACHING TESTS

Figure 13-2 shows the flow sheet for sample preparation and testing as adopted for the first metallurgical campaign carried out with the three selected Matupá composite samples.

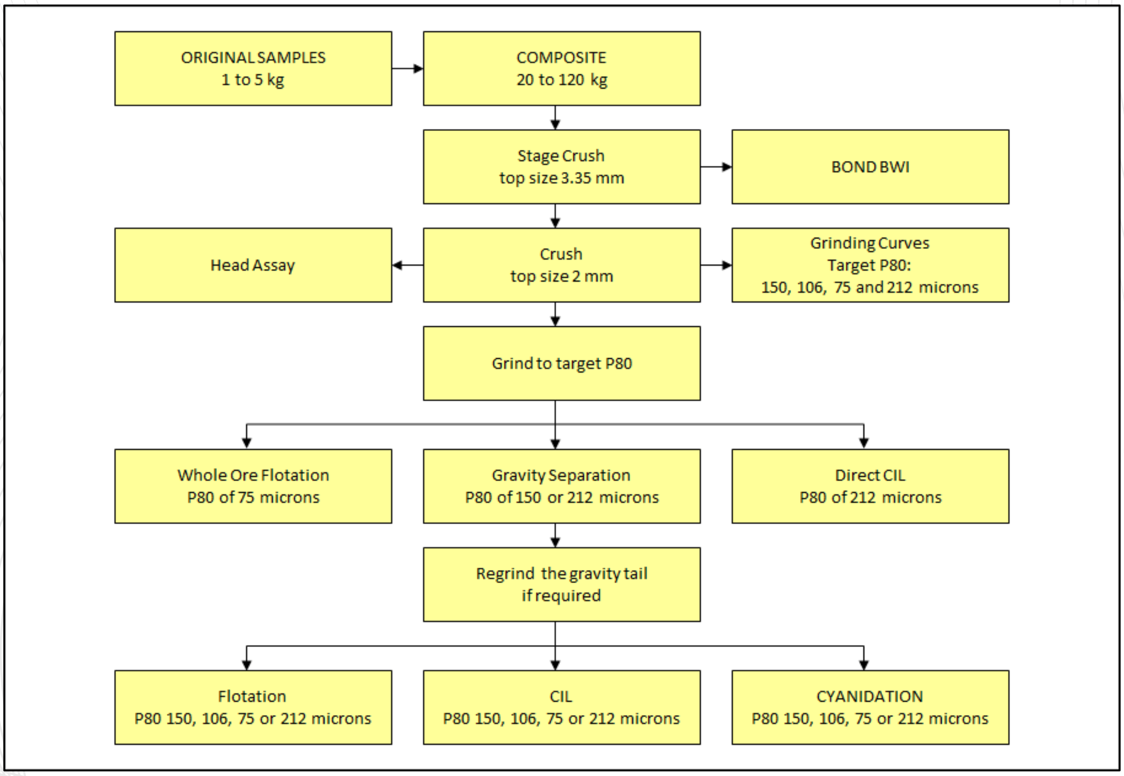


Figure 13-2 - Sample and Test Preparation Flow Sheet Adopted for the First Metallurgical Campaign.

The tests carried out were as follows:

- Rougher flotation tests for samples ground at a P_{80} of 0.075 mm using different types of collectors (PAX, MX980, A3418, A412, and OX-100), copper sulfate ($CuSO_4$) as an activator, and DF-250 as a frother. The highest obtained metallurgical gold recoveries were 96.7% for the High Sulfide sample and 77.4 - 77.9% for the Oxide sample, as well as mass recoveries of 15.2 and 3.5-6.8% respectively;

- Gravity concentration tests were carried out using a Knelson concentrator model MD-3 for samples previously ground at P_{80} of 0.150 mm, while the High Sulfide sample was also tested with a P_{80} of 0.212 mm. Resulting gold metallurgical recoveries for the 0.150 mm P_{80} ranged from 36.3% (Oxide) to 60.6% (Low Sulfide);

- Rougher flotation tests were carried out with High Sulfide sample on tailings from the gravity separation at four different grinding sizes i.e., P_{80} of 0.075 mm, 0.106 mm, 0.150 mm, and 0.212 mm. The same procedure was adopted with the Low Sulfide gravity separation tailing sample, excluding the P_{80} of 0.212 mm;

- Leaching tests were carried out on all three composite samples using the gravity separation tailings. Four different grinding sizes were used i.e., P_{80} of 0.075 mm, 0.106 mm, 0.150 mm, and 0.212 mm. The 48-hour leaching period results indicated gold recoveries higher than 90%; and

- Carbon in leach (CIL) was tested on three samples previously ground at P_{80} of 0.075 mm, 0.106 mm, 0.150 mm and 0.212 mm. The gravity tailing samples were also ground to the same sizes and CIL tested as well. All results indicated gold recoveries higher than 90%.

Direct leaching on High Sulfide sample previously ground at a P_{80} of 0.212 mm resulted in a 95.1% gold recovery, as shown in Table 13-5.

Table 13-5 - Direct Leaching Results of High Sulfide Sample.

DIRECT CIL - WHOLE ORE										
P80 212 MICRONS - 48 H LEACHING										
sample	Head Assay Au ppm Back Calc.	Mass of Residue grams	Residue Assay Au ppm	Mass of Liquor grams	Liquor Assay Au ppm	Mass of Carbon grams	Carbon Assay Au ppm	NaCN consumed kgpt	CaO consumed kgpt	Gold Extraction %
HIGH SULPHIDE	1,42	485	0,07	698	< 0,01	7,7	84,9	0,6	0,8	95,1

Table 13-6 summarizes the results obtained for the first metallurgical campaign.

Table 13-6 - Summary of Results Obtained in the First Metallurgical Testing Campaign.

OVERALL SUMMARY - MATUPA COMPOSITES								
sample	Head Assay Au ppm Mass Bal.	Bond BWI kWh/ton P80 212 µm	Rec % W. Ore Flot. P80 212 µm	Ext % Direct CIL P80 212 µm	Ext % Grav. Sep. P80 212 µm	Global Ext % Grav.Sep plus Flotation P80 212 µm	Global Ext % Grav.Sep plus Cyanidation P80 212 µm	Global Ext % Grav.Sep plus CIL P80 212 µm
HIGH SULFIDE	1,15	16,1	-	95,1	33,1	99,9	94,7	93,6
LOW SULFIDE	0,62	15,3	-	-	-	-	-	-
OXIDE	1,85	13,7	-	-	-	-	-	-
sample	Head Assay Au ppm Mass Bal.	Bond BWI kWh/ton P80 75 µm	Rec % W. Ore Flot. P80 75 µm	Ext % Direct CIL P80 75 µm	Ext % Grav. Sep. P80 150 µm	Global Ext % Grav.Sep plus Flotation P80 75 µm	Global Ext % Grav.Sep plus Cyanidation P80 75 µm	Global Ext % Grav.Sep plus CIL P80 75 µm
HIGH SULFIDE	1,15	-	96,7	-	38,6	95,9	95,0	97,1
LOW SULFIDE	0,62	-	-	-	60,6	99,0	98,3	99,3
OXIDE	1,85	-	77,4	-	36,3	-	96,2	96,5

13.3 SECOND METALLURGICAL CAMPAIGN

Further studies resulted in the reclassification of ore types i.e., Fresh Rock and Oxide, which were adopted in the second metallurgical campaign. Various tests were carried out in order to select a suitable ore processing flow sheet, as well as to assess the respective metallurgical performance for the selected samples that represented the two ore types.

13.3.1 PLANNING

The metallurgical tests carried out in the second campaign were based on the following samples: (a) Fresh Rock (samples identified as FRE or MATUF) and (b) Oxide Ore (samples identified as OXI or MATUO).

The sample preparation and metallurgical testing campaign was planned according to the flow sheet shown in Figure 13-3.

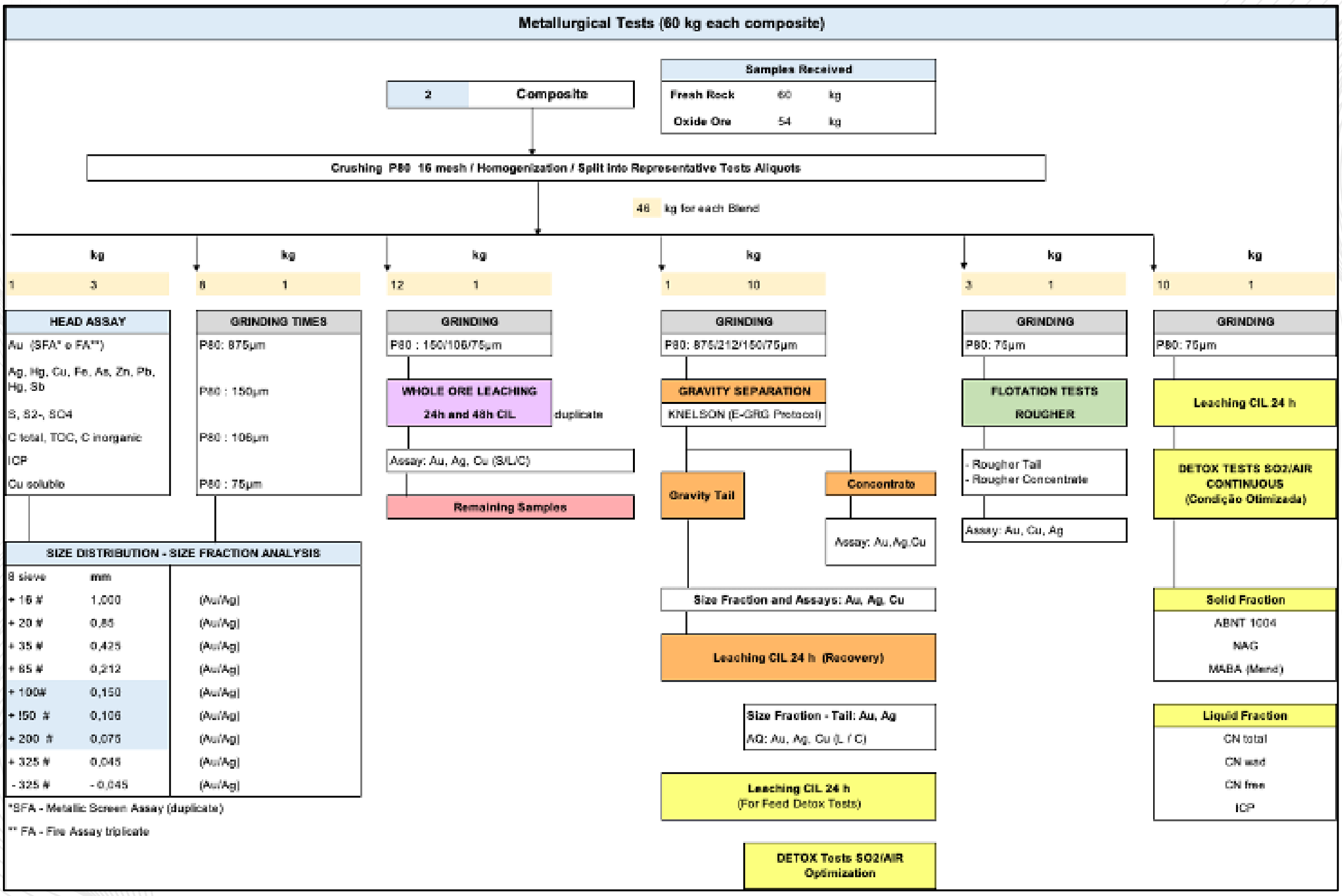


Figure 13-3 – Sample Preparation and Testing for the Second Campaign.

13.3.2 SAMPLES AND PREPARATION

Selected criteria and protocols were adopted for obtaining representative samples of each ore types. The sample description, masses and testing are shown in Tables 13-7 and 13-8.

Table 13-7 - Samples Used in Comminution Tests.

Sample	ID	Objective	Kg
FRESH ROCK	SAMPLE A	Comminution Tests - Part 1	107,38
	SAMPLE B	Comminution Tests - Part 2	13,8
	SAMPLE C	Comminution Tests - Part 2	14,95
OXIDE ORE	SAMPLE A	Comminution Tests - Part 1	99,06
	SAMPLE B	Comminution Tests - Part 2	18,9
	SAMPLE C	Comminution Tests - Part 2	18,9
TOTAL			273,0

Table 13-8 - Samples Used in Metallurgical Tests and Characterizations.

Sample	ID	Objective	Kg	Au (g/t)*	S (%)*
FRESH ROCK	FRE or MATUF	Metallurgical Tests and Characterization	60,5	1,55	2,22
OXIDE ORE	OXI or MATUO	Metallurgical Tests and Characterization	54,5	0,78	0,01
TOTAL			**115,0**	_	_

* *Weighted Average.*

Figure 13-4 shows photographs of some core fragments used to prepare the two composites representing the two ore types.

(a) (b)

Figure 13-4 - Examples of Drill Core Fragments Used in the Second Campaign: (a) Fresh Rock; (b) Oxide Ore.

The samples were prepared and sent to the following laboratories for comminution and metallurgical testing:

- Mineral Processing Solutions Ltda. (MinPro Solutions) - São Paulo: samples prepared for comminution testing; and
- Testwork Desenvolvimento de Processo Ltda. (Testwork) - Nova Lima: samples prepared for metallurgical characterization and testing.

The above described samples were prepared, handled and split according to low-grade gold ore QA/QC guidelines and protocols.

Each sample selected for the metallurgical testing was initially crushed to 2.0 mm (10 # Tyler) top size.

13.3.3 COMMINUTION TESTS

The samples identified as A in Table 13-7 were used to perform the following comminution tests referred as Batch 1:

- Bond Abrasion Index (Ai);

- BWI using a 0.106 mm closing screen; and
- Drop Weight Test - Simplified DWT.

The samples identified as B and C in Table 13-7 were used to perform the following comminution tests referred to as Batch 2:

- BWI using a 0.150 mm closing screen;
- BWI using a 0.180 mm closing screen; and
- Pycnometer for determining the specific gravity.

Standard methods and procedures were adopted for conducting comminution tests at Mineral Processing Solutions Ltda. (MinPro Solutions). These tests are listed in Table 13-9.

Table 13-9 - Comminution Testing.

Sample	Mass	Tests (Commintion Characterization)			
	Received (kg)	Part	BWI - Closing sieve (mm)	Ai Bond	DTW simplified
Fresh Rock (FRE - Sample A)	107.4	1	0.106	Yes	Yes
Fresh Rock (FRE - Sample B)	13.8	2	0.150 and 0.180	No	No
Fresh Rock (FRE - Sample C)	15	2	0.150 and 0.180	No	No
Oxide Ore (FRE - Sample A)	99.1	1	0.106	Yes	Yes
Oxide Ore (FRE - Sample B)	18.9	2	0.150 and 0.180	No	No
Oxide Ore (FRE - Sample C)	18.9	2	0.150 and 0.180	No	No

A summary of comminution test results is listed in Table 13-10.

Table 13-10 - Summary of Comminution Test Results.

Sample	Bond Wi for Ball Grinding			Picnometry	Bond Ai		DWT			
	Test Screen (mm)	Wi (kWh/t)	Tenacity	Specific Mass (g/cm3)	Ai	Abrasion	A	b	IQ	Classification
Oxide	**0.106**	**17.0**	**High**	-	**0.102**	**Low**	**75.2**	**5.43**	**392**	**Extremely Low**
Santa Elina Drill hole – SEX1-06	0.150	15.3	High	2,57						
Santa Elina Drill hole – SEX1-06	0.180	14.5	Moderately High							
Sanrta Elina Hole – SEX1-34	0.150	16.2	High	2,66						
Sanrta Elina Hole – SEX1-34	0.180	14.5	Moderately High							
Fresh Rock	**0.106**	**20.5**	**Very High**	-	**0.362**	**Medium**	**65.0**	**0.76**	**49.4**	**Moderately High**
FX 1D 0022	0.150	18.8	Very High	2,76						
FX 1D 0022	0.180	18.6	Very High							
FX 1D 0024	0.150	17.2	High	2,58						
FX 1D 0024	0.180	16.8	High							

The BWI tests carried out with 0.106 mm closing screen resulted in 20.5 kWh/t for the Fresh Rock sample and 17.0 kWh/t for the Oxide sample, therefore classifying them respectively as Very High tenacity and High tenacity to grinding in ball mills.

The DWT tests performed in the samples indicated an A*b parameter (Breakage Index - IQ) of 49.9 for the Fresh Rock sample and 392 for the Oxide sample. These results were classified respectively as Moderately High and Extremely Low to high energy breakage.

13.3.4 CHEMICAL ANALYSIS

The composites samples listed in Table 13-8 were prepared at Testwork laboratories resulting in aliquots which were sent to SGS-Geosol laboratories for chemical analysis listed below:

- Gold by Fire Assay, in duplicate;
- Multi-acid digestion/ICP-OES: 37 elements;
- LECO: Determination of the Carbon (C) phases by Infrared;
- Determination of sulfur in high-grade samples by firing in a resistive furnace;
- Determination of mercury by cold vapor generation - Atomic Absorption (AAS); and
- Determination of low-grade silver by partial digestion with aqua regia - Atomic Absorption (AAS).

The obtained results are listed in Table 13-11, 13-12, 13-13 and 13-14.

Table 13-11 - Gold Grade Results - Fresh Rock Sample.

FRACTION (Mesh)	Fresh Rock - Aliquot 1			Fresh Rock - Aliquot 2		
	WEIGHT, %	Au, g/t	% Au DISTR'N	WEIGHT, %	Au, g/t	% Au DISTR'N
+150 mesh	4,93	7,93	21,59	4,57	5,76	16,45
-150 mesh	95,07	1,50	78,41	95,43	1,40	83,55
Head - Total	100,00	1,81	100,00	100,00	1,60	100,00
Head - Average	1,71					

Table 13-12 - Gold Grade Results - Oxide Ore Sample.

FRACTION (Mesh)	Oxide Ore - Aliquot 1			Oxide Ore - Aliquot 2		
	WEIGHT, %	Au, g/t	% Au DISTR'N	WEIGHT, %	Au, g/t	% Au DISTR'N
+150 mesh	4,75	3,47	12,87	4,92	1,19	6,33
-150 mesh	95,26	1,17	87,13	95,08	0,91	93,67
Head - Total	100,00	1,28	100,00	100,00	0,92	100,00
Head - Average	1,10					

Table 13-13 - Complete Chemical Analysis Results.

Methd	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B
Element	Ag	Al	As	Ba	Be	Bi	Ca	Cd	Co	Cr	Cu	Fe	K	La
Sample	ppm	%	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%	%	ppm
Matupa - **Fresh Rock**	<3	5,42	<10	604	<3	<20	0,14	<3	19	4	376	3,04	2,8	<20
Matupa - **Oxide Ore**	<3	6,33	14	802	3	<20	0,06	<3	<8	20	59	4,62	3,14	20

Methd	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B
Element	Li	Mg	Mn	Mo	Na	Ni	P	Pb	S	Sb	Sc	Se	Sn	Sr
Sample	ppm	%	%	ppm	%	ppm	%	ppm	%	ppm	ppm	ppm	ppm	ppm
Matupa - **Fresh Rock**	15	0,18	0,04	28	0,38	<3	<0,01	223	1,82	<10	<5	<20	<20	14
Matupa - **Oxide Ore**	31	0,27	0,03	69	0,1	4	0,04	28	0,05	<10	6	<20	<20	60

Methd	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	CVA02B	CSA17V	CSA17V	CSA20V	CSA20V	CSA20V
Element	Th	Ti	Tl	U	V	W	Y	Zn	Zr	Hg	C	S	C_ORG	C_ELE	C_CAR
Sample	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	%	%	%	%	%
Matupa - **Fresh Rock**	<20	0,05	<20	<20	<8	<20	5	796	66	0,6	0,08	1,74	<0,05	<0,05	0,07
Matupa - **Oxide Ore**	<20	0,14	<20	<20	28	<20	4	34	92	0,53	0,07	0,04	0,06	<0,05	<0,05

Table 13-14 - Silver Grade Results.

Sample (ID)	FRE	OXI
AAS12E - Ag (g/t)	4,6	0,6
	4,0	0,6
	5,3	0,5
	4,4	0,6
Average - Ag (g/t)	**4,6**	**0,6**

Fresh Rock and Oxide Ore samples averaged gold grades of 1.71 g/t and 1.10 g/t Au, respectively. Accordingly, the sulfur grades were 1.74% and 0.04% S for Fresh Rock and Oxide samples respectively. The silver analysis was initially performed by ICP-OES and subsequently analyzed by partial digestion with aqua regia/AAS, which is an adequate method for relatively low Ag grades, resulting in 4.6 g/t Ag for the Fresh Rock sample and 0.6 g/t Ag for the Oxide sample. The silver results represent a potential gain to the Project, especially for the Fresh Rock ore type.

The analysis of copper grades - one possible cyanide-consuming element - resulted in 376 ppm Cu for the Fresh Rock sample and 59 ppm Cu for the Oxide sample. The analysis of zinc, another cyanide-consuming element, also indicated higher values for the Fresh Rock sample compared to Oxide sample - 796 ppm and 34 ppm zinc, respectively.

13.3.5 GRAVITY RECOVERABLE GOLD – GRG TESTS

Standard Gravity Recoverable Gold (“GRG”) tests were carried out on both Fresh Rock and Oxide samples. Figure 13-5 shows the adopted flow sheet for testing the two samples, including three grinding/concentration stages. The tests were carried out at Testwork laboratories, while the resulting samples were assayed at SGS Geosol laboratories.

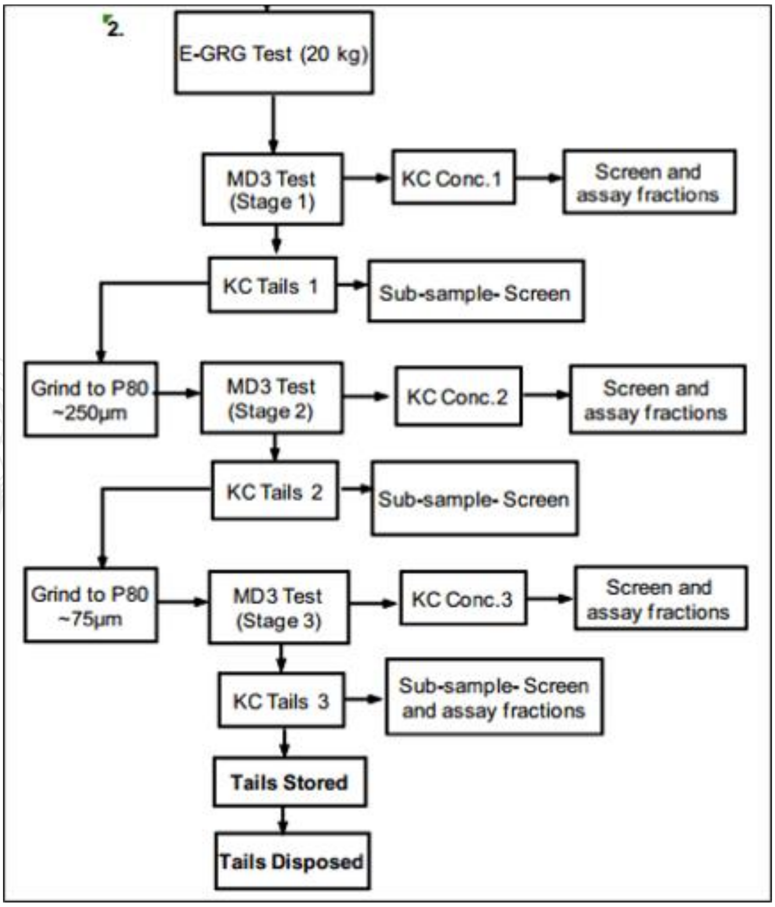


Figure 13-5 - Gravity Recoverable Gold Test Flow Sheet.

The results obtained for GRG tests in terms of mass and gold recoveries are shown in Table 13-15, which indicates an accumulated gravity gold recovery of 70.5% for the Fresh Rock sample and 35.8% for the Oxide sample.

Table 13-15 - Modeling Results for Extended Gravity Recoverable Gold (E-GRG).

Sample	GRG Test Results		Modelling
	Mass Recovery (%)	Gold Recovery (%)	Gold Recovery (%)
Fresh Rock - FRE	2.95	70.5	**36.5**
Oxide Ore - OXI	2.27	35.8	**12.6**

All data obtained in the GRG tests were sent to FLSmidth Ltd. (Knelson Division) for modelling and estimation of gravity gold recovery. The results were 36.5% and 12.6% gold recovery, respectively for Fresh Rock and Oxide samples. According to the Amira method, gravity gold was classified as coarse for Fresh Rock sample and fine to moderate for Oxide sample, as shown in Figure 13-6.

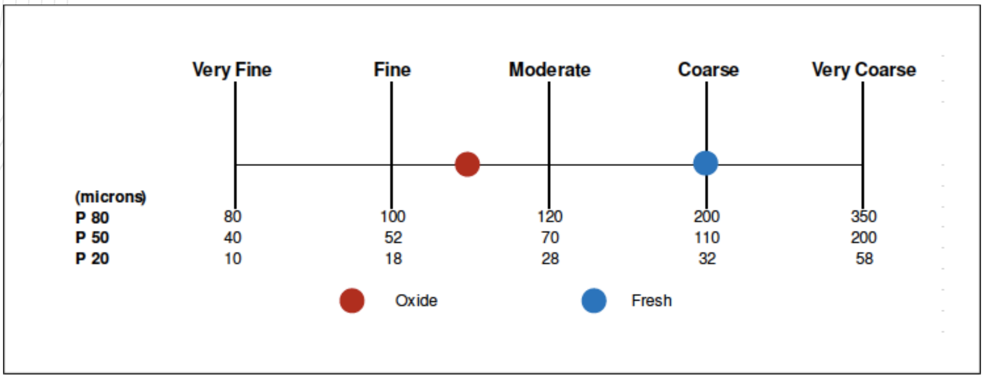


Figure 13-6 - Amira Scale for Gravity Tests (E-GRG).

FLSmidth also simulated and designed the respective equipment for the industrial gravity recovery circuit. The results were as follows:

- A centrifugal concentrator model QS40 installed in the cyclone underflow processing approximately 37% of the circulating load.
- An intensive leaching reactor model CS2000 for treating the gravity concentrate from the centrifugal concentrator.

13.3.6 EXPLORATORY FLOTATION TESTS

Exploratory flotation tests were carried out to assess the metallurgical performance associated with Fresh Rock and Oxide samples. Accordingly, triplicated tests were conducted in a single rougher stage at Testwork laboratories.

Samples were ground to a P_{80} of 0.075 mm and diluted to 35% w/w solids for testing at an Engedrar flotation cell - FB 1000 model. Lime was added to maintain a pH of 9.5 during all tests. Conditioning time and reagent dosages are listed in Table 13-16. Potassium Amyl Xanthate (PAX) was used as a collector, DF-250 as a frother, and copper sulfate as a promoter. Collecting periods were 14.5 minutes divided into five stages. Samples collected in each stage, together with the final tailings, were individually prepared and assayed at the SGS Geosol laboratories for determining gold and sulfur grades.

Table 13-16 - Conditions of the Flotation Tests.

Reagents, Dosage and Time:						
Step	**Rougher 1**	**Rougher 2**	**Rougher 3**	**Rougher 4**	**Rougher 5**	**Total**
Conditioning (min)	2 (w/ lime pH 9,5)	2	1	1	1	7
Flotation (min)	0,5	1	3	5	5	14,5
PAX (g/t)	50	20	10	10	10	100
DF250 (g/t)	20	5	5	10	10	50
CuSO4 (g/t)	10	_	_	_	_	10

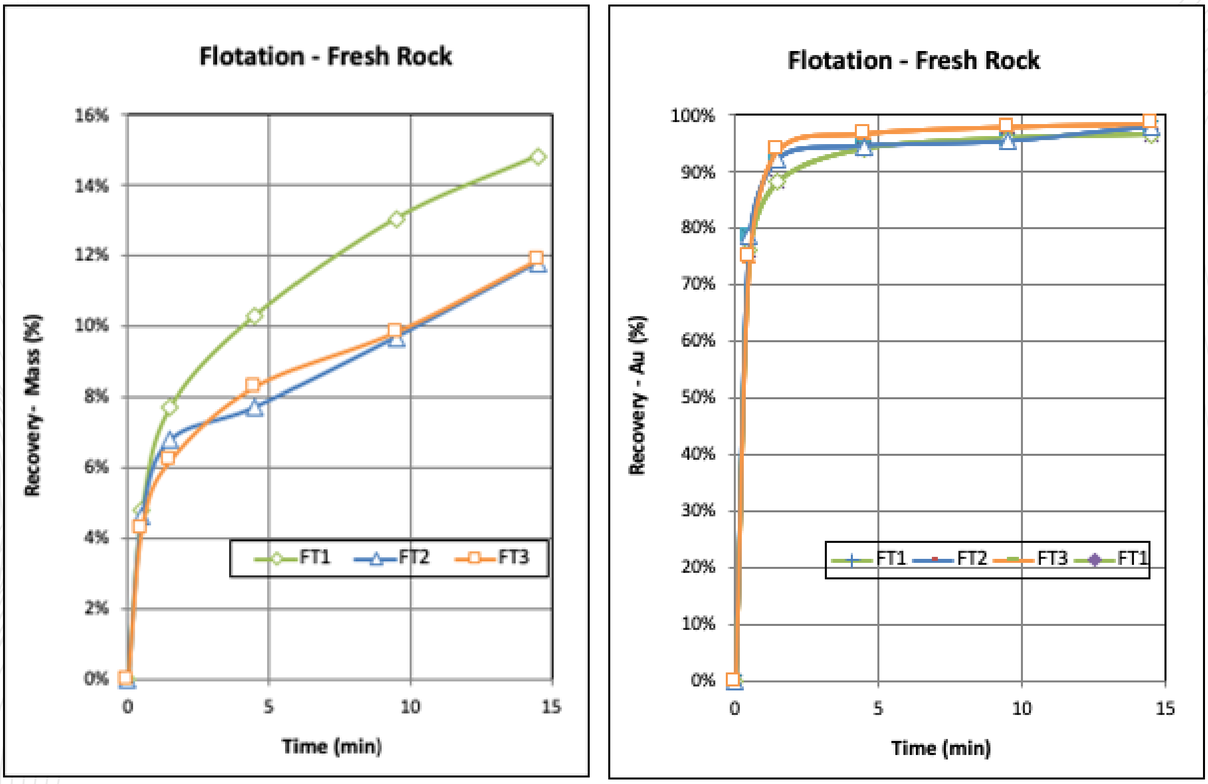
The flotation test results obtained in terms of mass, gold, and sulfur are shown in Table 13-17 and Table 13-18, respectively for Fresh Rock and Oxide samples. Detailed mass and gold recovery versus time curves are shown in Figure 13-7 and Figure 13-18.

Table 13-17 - Rougher Flotation Results for Fresh Rock Sample.

Flotation Tests (Rougher)									**FRESH ROCK**					
Test	**P_{80}**	**Test Condition (Reagent Dosage) (g/t)**				**Time (min)**	**Back Calculated w/o Adjusts Au (g/t)**	**Head Assay Au (g/t)**	**Mass Pull (%)**	**Back Calculated w/o Adjusts Au (g/t)**	**Flotation Concentrate Au (g/t)**	**Flotation Tail Au (g/t)**	**Flotation Recovery Au (%)**	**Flotation Recovery S (%)**
		$CuSO_4$	**SIBX**	**PAX**	**INT 102**									
FT1	75 µm	10	-	100	50	14,5	1,47	1,71	14,82%	1,22	7,96	0,05	96,52%	99,01%
FT2		10	-	100	50	14,5			11,78%	1,70	14,15	0,04	97,93%	98,88%
FT3		10	-	100	50	14,5			11,86%	1,50	12,41	0,03	98,43%	97,96%
Average							**1,47**	**1,71**	**12,8%**	**1,47**	**11,51**	**0,04**	**97,6%**	**98,6%**
** Rougher Flotation Tests: pH 9.5 with lime.*														

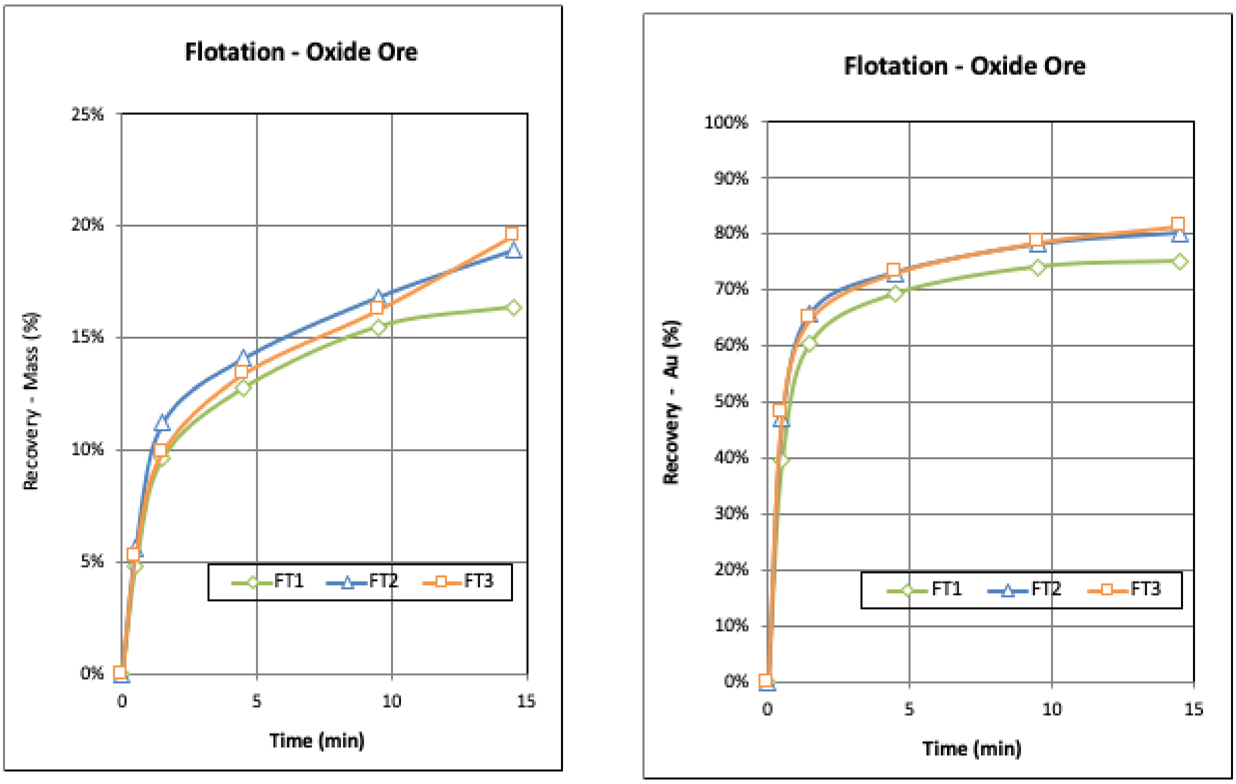
Table 13-18 - Rougher Flotation Results for Oxide Sample.

Flotation Tests (Rougher)									**OXIDE ORE**					
Test	**P_{80}**	**Test Condition (Reagent Dosage) (g/t)**				**Time (min)**	**Back Calculated w/o Adjusts Au (g/t)**	**Head Assay Au (g/t)**	**Mass Pull (%)**	**Back Calculated w/o Adjusts Au (g/t)**	**Flotation Concentrate Au (g/t)**	**Flotation Tail Au (g/t)**	**Flotation Recovery Au (%)**	**Flotation Recovery S (%)**
		$CuSO_4$	**SIBX**	**PAX**	**INT 102**									
FT1	75 µm	10	-	100	50	14,5	0,92	1,10	16,35%	0,96	4,41	0,29	75,03%	67,01%
FT2		10	-	100	50	14,5			18,92%	0,92	3,89	0,23	80,04%	66,64%
FT3		10	-	100	50	14,5			19,50%	0,88	3,67	0,20	81,39%	74,43%
Average							**0,92**	**1,10**	**18,3%**	**0,92**	**3,99**	**0,24**	**78,8%**	**69,4%**
** Rougher Flotation Tests: pH 9.5 with lime.*														



(a) (b)

Figure 13-7 - Rougher Flotation - Fresh Rock: (a) Mass Recovery; (b) Gold Recovery.



(a) (b)

Figure 13-8 - Rougher Flotation - Oxide Ore: (a) Mass Recovery; (b) Gold Recovery.

Rougher flotation results indicated final gold recovery of 97.6% for Fresh Rock sample and 78.8% for Oxide sample. The mass recoveries were 12.8% and 18.3% respectively for Fresh Rock and Oxide samples, together with averaged gold grades in the concentrate of 11.51 g/t and 3.99 g/t Au.

Based on relatively high Au grade as obtained in Oxide sample rougher tailings, together with potentially high losses associated with cleaner stages, it was decided to exclude flotation as a concentration method for the Matupá Project.

13.3.7 LEACHING TESTS

Direct leaching was further assessed in second metallurgical testing campaign, together with leaching the gravity concentration tailings, as described in this section. The tests were carried out in two phases, as follows:

Phase 1 - Initial Tests

Direct leaching tests on Fresh Rock and Oxide samples. Samples were initially ground to three grinding sizes – P_{80} of 0.075 mm, 0.106 mm and 0.150 mm. The required grinding times were estimated based on grinding curves, as previously obtained in the sample preparation stage.

Phase 2 - Additional Tests

Direct leaching tests based on results obtained in initial tests. In this case the adopted grinding P_{80} was 0.125 mm in triplicated tests. One repeated test was also carried out on a 0.106 mm P_{80} sample.

Direct leaching test conditions adopted were as follows:

- Grinding Size (P_{80}):

Step 1: 75, 106, and 0.150 mm.

Step 2: 106 and 0.125 mm.

- Pre-lime: 2 hours.

- Residence time – CIL.

Step 1: 24 and 48 hours.

Step 2: 24 hours.

- Concentration of solids: 45% w/w.

- Concentration of activated carbon: 25 g/L.

- pH: 10 to 11 modulated with lime (milk of lime – calcium hydroxide).

- Initial cyanide concentration (NaCN): 1,000 mg/L.

- Free cyanide concentration (CNFree) maintained above 250 mg/L.

- Concentration of dissolved oxygen (DO): > 4 mg/L.

- Replicates:

Step 1: Duplicate tests (2x).

Step 2: Triplicate tests (3x).

Leaching tests were carried out in rolling bottles. Aliquots were collected during the periods described below for assessing the following parameters: free cyanide (CNFree), Eh (oxy-reduction potential), DO (dissolved oxygen) and pH. When necessary, sodium cyanide and milk of lime were added to the pulp.

- Tests with 24-hour leaching time - monitoring: 2, 4, 6, 24 hours.

- Tests with 48-hour leaching time - monitoring: 2, 4, 6, 24, 28, 30, and 48 hours.

At the end of each test, solid, solution and carbon were accurately separated to avoid losses and/or contaminations. Each sample was weighed and/or volume measured, as well as further prepared to assaying at SGS-Geosol laboratories. Parameters measured were as follows:

- Solids: Mass, gold (duplicate), copper, and silver assays.

- Solution: Volume (mL), gold (duplicate), copper, and silver assays.

- Carbon: Mass and gold assay.

Estimates of recovery and specific reagent consumption were obtained from test results and chemical analysis.

Tables 13-19 and 13-20 show the results obtained in the Phase 1 direct leaching tests, respectively for Fresh Rock and Oxide samples. Accordingly, direct leaching tests LT1 to LT6 refer to 24-hour leaching time, while LT7 to LT12 tests are related to the 48-hour leaching time. The LT13 and LT14 tests were conducted with gravity concentration tailing samples according to a 24-hour leaching time.

Table 13-19 - Gold Recovery Results from Direct Leaching Tests - Fresh Rock – Initial Tests.

Tests Conditions (FRESH ROCK)							Total Gold Recovery (%)	
Test	Sample	P80 (microns)	Gravity Circuit	Pre Lime (2 h)	Circuit	Total Residence Time (h)	Calculated w/o Adjusts	Average
LT1	Fresh Rock - Ore	150	No	Yes	CIL 24 h	24	97,40%	96,76%
LT2	Fresh Rock - Ore	150	No	Yes	CIL 24 h	24	96,12%	
LT3	Fresh Rock - Ore	106	No	Yes	CIL 24 h	24	97,90%	98,28%
LT4	Fresh Rock - Ore	106	No	Yes	CIL 24 h	24	98,66%	
LT5	Fresh Rock - Ore	75	No	Yes	CIL 24 h	24	96,85%	97,70%
LT6	Fresh Rock - Ore	75	No	Yes	CIL 24 h	24	98,54%	
LT7	Fresh Rock - Ore	150	No	Yes	CIL 48 h	48	98,13%	97,42%
LT8	Fresh Rock - Ore	150	No	Yes	CIL 48 h	48	96,71%	
LT9	Fresh Rock - Ore	106	No	Yes	CIL 48 h	48	98,50%	98,17%
LT10	Fresh Rock - Ore	106	No	Yes	CIL 48 h	48	97,84%	
LT11	Fresh Rock - Ore	75	No	Yes	CIL 48 h	48	98,17%	98,37%
LT12	Fresh Rock - Ore	75	No	Yes	CIL 48 h	48	98,58%	
LT13	Fresh Rock - GRG Tailing	75	Yes	Yes	CIL 24 h	24	92,72%	92,12%
LT14	Fresh Rock - GRG Tailing	75	Yes	Yes	CIL 24 h	24	91,52%	

Table 13-20 - Gold Recovery Results from Direct Leaching Tests - Oxide Sample – Initial Tests.

Tests Conditions (OXIDE ORE)							Total Gold Recovery (%)	
Test	Sample	P80 (microns)	Gravity Circuit	Pre Lime (2 h)	Circuit	Total Residence Time (h)	Calculated w/o Adjusts	Average
LT1	Oxide Ore	150	No	Yes	CIL 24 h	24	91,12%	91,61%
LT2	Oxide Ore	150	No	Yes	CIL 24 h	24	92,10%	
LT3	Oxide Ore	106	No	Yes	CIL 24 h	24	94,51%	92,54%
LT4	Oxide Ore	106	No	Yes	CIL 24 h	24	90,57%	
LT5	Oxide Ore	75	No	Yes	CIL 24 h	24	96,03%	96,12%
LT6	Oxide Ore	75	No	Yes	CIL 24 h	24	96,22%	
LT7	Oxide Ore	150	No	Yes	CIL 48 h	48	92,99%	92,75%
LT8	Oxide Ore	150	No	Yes	CIL 48 h	48	92,51%	
LT9	Oxide Ore	106	No	Yes	CIL 48 h	48	95,86%	95,56%
LT10	Oxide Ore	106	No	Yes	CIL 48 h	48	95,27%	
LT11	Oxide Ore	75	No	Yes	CIL 48 h	48	95,69%	95,87%
LT12	Oxide Ore	75	No	Yes	CIL 48 h	48	96,06%	
LT13	Oxide Ore - GRG Tailing	75	Yes	Yes	CIL 24 h	24	93,77%	93,68%
LT14	Oxide Ore- GRG Tailing	75	Yes	Yes	CIL 24 h	24	93,58%	

According to Table 13-19 the Fresh Rock sample averaged 98.3% gold recovery for a P_{80} of 0.106 mm for a 24-hour leaching period. For the same grinding size and leaching time, the Oxide sample averaged 92.5% gold recovery. Averaged gold recovery values for Fresh Rock showed virtually no variation associated with tested grinding sizes. Conversely, Oxide sample results indicated a tendency of increased gold recovery for finer grinding sizes.

The same Table 13-19 shows that gold recovery in gravity concentration tailing leaching averaged 92.12% and 93.68%, respectively for Fresh Rock and Oxide samples.

The results obtained from Phase 2 direct leaching tests are listed in Table 13-21 and Table 13-22. Averaged gold recovery results indicated no significant differences for the two grinding sizes i.e., 0.106 mm and 0.125 mm.

Table 13-21 - Gold Recovery Results from Leaching Tests - Fresh Rock Ore – Additional Tests.

Tests Conditions (FRESH ROCK)							Total Gold Recovery (%)	
Test	Sample	P80 (microns)	Gravity Circuit	Pre Lime (2 h)	Circuit	Total Residence Time (h)	Calculated w/o Adjusts	Average
LT15	Fresh Rock - Ore	106	No	Yes	CIL 24h	24	97,25%	97,16%
LT16	Fresh Rock - Ore	106	No	Yes	CIL 24h	24	95,33%	
LT17	Fresh Rock - Ore	106	No	Yes	CIL 24h	24	98,89%	
LT18	Fresh Rock - Ore	**125**	No	Yes	CIL 24h	24	98,96%	**97,81%**
LT19	Fresh Rock - Ore	**125**	No	Yes	CIL 24h	24	97,28%	
LT20	Fresh Rock - Ore	**125**	No	Yes	CIL 24h	24	97,21%	

Table 13-22 - Gold Recovery Results from Leaching Tests - Oxide Ore – Additional Tests.

Tests Conditions (OXIDE ORE)							Total Gold Recovery (%)	
Test	Sample	P80 (microns)	Gravity Circuit	Pre Lime (2 h)	Circuit	Total Residence Time (h)	Calculated w/o Adjusts	Average
LT15	Oxide Ore	106	No	Yes	CIL 24h	24	91,30%	93,54%
LT16	Oxide Ore	106	No	Yes	CIL 24h	24	94,71%	
LT17	Oxide Ore	106	No	Yes	CIL 24h	24	94,59%	
LT18	Oxide Ore	**125**	No	Yes	CIL 24h	24	93,37%	**93,55%**
LT19	Oxide Ore	**125**	No	Yes	CIL 24h	24	93,59%	
LT20	Oxide Ore	**125**	No	Yes	CIL 24h	24	93,68%	

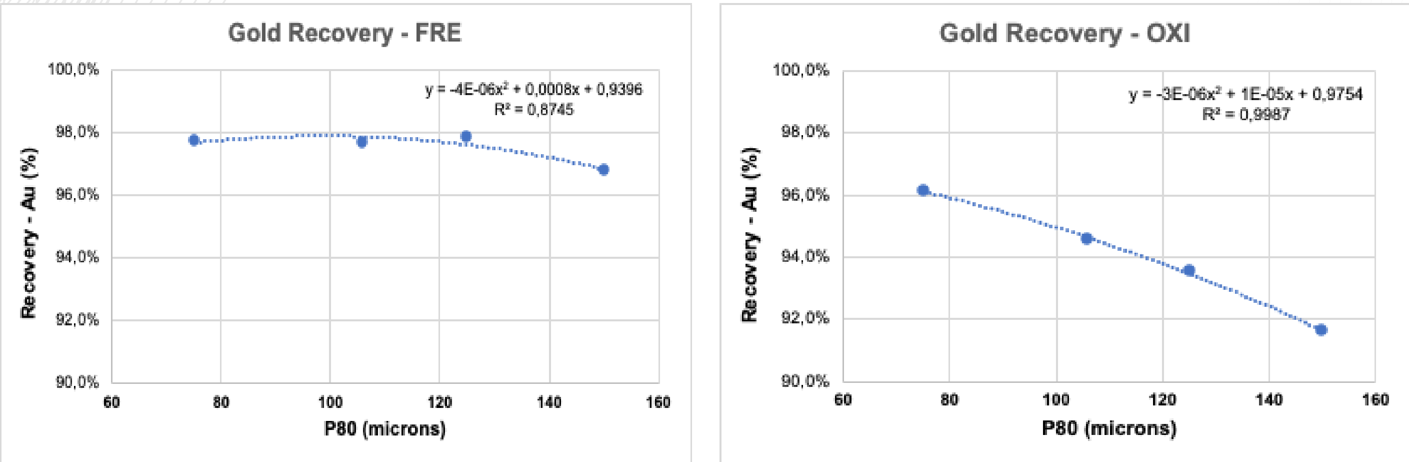
A summary of averaged gold recovery results as obtained per grinding size in both Phase 1 and Phase 2 are listed in Table 13-23 and Table 13-24, as well as in Figure 13-9.

Table 13-23 - Summary of Gold Recovery Results from Leaching Tests – Fresh Rock – Additional Tests.

Sample	P80 (microns)	Gravity Circuit	Pre Lime (2 h)	Circuit	Recov. Average Au (g/t)	Results Considered
Fresh Rock - Ore	150	No	Yes	CIL 24 h	96,76%	Average 2 results
Fresh Rock - Ore	125	No	Yes	CIL 24 h	97,81%	Average 3 results
Fresh Rock - Ore	106	No	Yes	CIL 24 h	97,66%	Average 5 results
Fresh Rock - Ore	75	No	Yes	CIL 24 h	97,70%	Average 2 results
Fresh Rock - E-GRG Tailing	75	Yes	Yes	CIL 24 h	92,12%	Average 2 results
Fresh Rock - Ore	150	No	Yes	CIL 48 h	97,42%	Average 2 results
Fresh Rock - Ore	106	No	Yes	CIL 48 h	98,17%	Average 2 results
Fresh Rock - Ore	75	No	Yes	CIL 48 h	98,17%	Average 2 results

Table 13-24 - Summary of Gold Recovery Results from Leaching Tests - Oxide Ore – Additional Tests.

Sample	P80 (microns)	Gravity Circuit	Pre Lime (2 h)	Circuit	Recov. Average Au (g/t)	Results Considered
Oxide Ore	150	No	Yes	CIL 24 h	91,61%	Average 2 results
Oxide Ore	125	No	Yes	CIL 24 h	93,55%	Average 3 results
Oxide Ore	106	No	Yes	CIL 24 h	94,55%	Average 5 results
Oxide Ore	75	No	Yes	CIL 24 h	96,12%	Average 2 results
Oxide Ore - GRG Tailing	75	Yes	Yes	CIL 24 h	93,68%	Average 2 results
Oxide Ore	150	No	Yes	CIL 48 h	92,75%	Average 2 results
Oxide Ore	106	No	Yes	CIL 48 h	95,56%	Average 2 results
Oxide Ore	75	No	Yes	CIL 48 h	95,56%	Average 2 results



(a) (b)

Figure 13-9 - Summary of Gold Recovery from Direct Leaching Tests - CIL 24 h: (a) Fresh Rock; (b) Oxide Ore – Additional Tests.

Table 13-23 shows that there were no significant differences among gold recovery figures as a function of grinding sizes for the Fresh Rock sample testing. Conversely, the Oxide sample showed an increasing gold recovery with finer grinding sizes.

Based on such trends, as well as on gold recovery values summarized in Table 13-24 and Figure 13-9, the 0.125 mm grinding P_{80} was selected. Such an intermediate grinding size would avoid the relatively high costs associated with fine grinding sizes, together with reductions in gold recovery related to coarse grinding sizes of Oxide sample. The adopted averaged gold recoveries for a 0.125 mm grinding P_{80} are as follows:

- Fresh Rock - Gold recovery of 97.81% (Standard deviation 0.99%).

- Oxide Ore - Gold Recovery of 93.55% (Standard Deviation 0.16%).

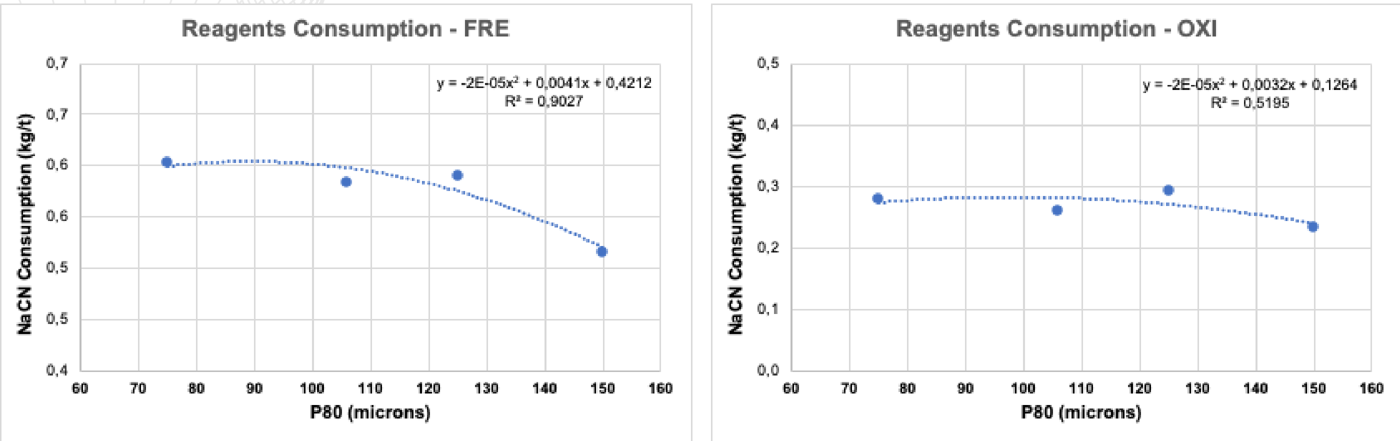
Estimated averaged reagent consumption - sodium cyanide and hydrated lime (calcium hydroxide) are shown in Table 12-25 and Table 13-26, as well as on Figure 13-10 and Figure 13-11.

Table 13-25 - Summary of Reagent Consumptions in Leaching - Fresh Rock.

Sample	P80 (microns)	Gravity Circuit	Pre Lime (2 h)	Circuit	NaCN Consump. (kg/t)	Lime Consump. (kg/t)
Fresh Rock - Ore	150	No	Yes	CIL 24 h	0,52	0,80
Fresh Rock - Ore	125	No	Yes	CIL 24 h	0,59	1,01
Fresh Rock - Ore	106	No	Yes	CIL 24 h	0,58	0,98
Fresh Rock - Ore	75	No	Yes	CIL 24 h	0,60	1,26
Fresh Rock - E-GRG Tailing	75	Yes	Yes	CIL 24 h	0,50	1,01
Fresh Rock - Ore	150	No	Yes	CIL 48 h	0,62	0,99
Fresh Rock - Ore	106	No	Yes	CIL 48 h	0,62	1,10
Fresh Rock - Ore	75	No	Yes	CIL 48 h	0,64	1,52

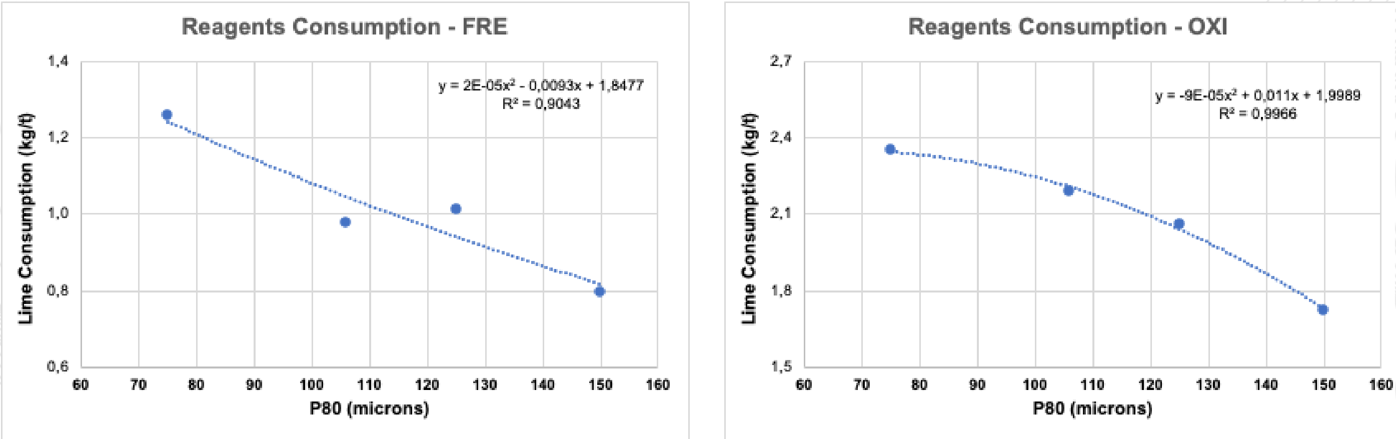
Table 13-26 - Summary of Reagent Consumption in Leaching - Oxide Ore.

Sample	P80 (microns)	Gravity Circuit	Pre Lime (2 h)	Circuit	NaCN Consump. (kg/t)	Lime Consump. (kg/t)
Oxide Ore	150	No	Yes	CIL 24 h	0,23	1,72
Oxide Ore	125	No	Yes	CIL 24 h	0,29	2,06
Oxide Ore	106	No	Yes	CIL 24 h	0,26	2,19
Oxide Ore	75	No	Yes	CIL 24 h	0,28	2,35
Oxide Ore - GRG Tailing	75	Yes	Yes	CIL 24 h	0,25	1,66
Oxide Ore	150	No	Yes	CIL 48 h	0,47	2,01
Oxide Ore	106	No	Yes	CIL 48 h	0,47	2,26
Oxide Ore	75	No	Yes	CIL 48 h	0,50	2,47



(a) (b)

Figure 13-10 - Summary of NaCN Consumption in Leaching - CIL 24 h: (a) Fresh Rock; (b) Oxide Ore.



(a) (b)

Figure 13-11 - Summary of Hydrated Lime Consumption in Leaching - CIL 24 h: (a) Fresh Rock; (b) Oxide Sample.

Figure 13-10 shows that the averaged sodium cyanide consumption was higher for the Fresh Rock sample as compared with the Oxide Ore sample. The opposite was observed in Figure 13-11 with the hydrated lime consumption i.e., higher consumption for the Oxide sample compared to Fresh Rock sample, even though the sulfur content of the former was smaller than the former. Such a fact may result from the higher required dosage of sodium cyanide, a salt derived from a strong base which in turn contributes to a pH increase.

Figure 13-10 also shows that sodium cyanide consumption was little affected by the grinding size, especially in the range from 0.075 mm to 0.125 mm. For hydrated lime, Figure 13-11 shows that the reduction in grinding size resulted in a higher consumption for pH control in both Fresh Rock and Oxide samples.

The adopted reagent consumptions for the 0.125 mm P_{80} grinding size were as follows:

- *Fresh Rock:*
  - NaCN – Consumption of 0.59 kg/t.
  - Hydrated lime – Consumption of 1.01 kg/t.
- *Oxide Ore*:
  - NaCN – Consumption of 0.29 kg/t.
  - Hydrated lime – Consumption of 2.06 kg/t.

13.3.8 GLOBAL GOLD RECOVERY ESTIMATES

The two selected ore types for the Matupá Project represented by Fresh Rock and Oxide samples were considered amenable to direct leaching (cyanidation), as well as gravity tailing leaching, resulting in very high gold recovery figures and low levels of gold in respective tailings (0.02 to 0.10 g/t Au) for the 24-hour leaching period of samples ground to 0.125 mm P_{80}.

Averaged gold recovery figures obtained in the second metallurgical campaign were adopted in the Preliminary Economic Assessment – PEA stage for the two selected ore types of the Matupá Project as listed in Table 13-27.

Table 13-27 - Gold Recovery Results.

Ore Types	Recovery - Au (%)		
	Gravity Recovery (%)	Gravity Taiings Leaching	Leaching Recovery - P80 125 microns
Fresh Rock	36,5%	92,12%	97,81%
Oxide Ore	12,6%	93,68%	93,55%
ILR Efficiency*	98,0%	* Intensive Leaching Reactor.	

Based on the test results summarized in Table 13-27, global recoveries were estimated for: (a) recovery route with gravity circuit, intensive leach reactor ("ILR"), and CIL; (b) recovery route with CIL only. The estimated gold recovery figures associated to each one of these two routes for different Fresh Rock and Oxide sample blends are listed in Table 13-28.

Table 13-28 – Global Gold Recoveries.

BLEND (FRE / OXI)	Global Recovery Estimate (%)	
	Gravity Circuit & ILR & Leaching	Direct Leaching
0 / 100	93,68%	93,55%
50 / 50	92,91%	95,68%
70 / 30	92,60%	96,53%
80 / 20	92,45%	96,96%
90 / 10	92,29%	97,39%
100 / 0	92,14%	97,81%

Apart from the 100% Oxide sample, the calculated gold recovery values listed in Table 13-28 show no benefit in gold recovery with the introduction of gravity and ILR (intensive leach reactor) stages in the recovery circuit. In fact, the gold recovery figures were consistently reduced with such an alternative. Based on such results, the direct leaching (CIL) only route was selected for the Matupá Project.

13.3.9 CYANIDE DESTRUCTION TESTS - DETOX

The SO_2/air method was assessed for destructing cyanide contained in leached tailings. Such a method, also known as the Inco or Detox method, uses sulfur dioxide (SO_2) to remove Wad (weak acidic dissociable - weakly bound) cyanide down to concentrations smaller than 2 mg/L.

Bench scale and continuous tests were carried out in the Testwork Desenvolvimento de Processo Ltda. laboratories in Nova Lima, while assaying was conducted at the SGS-Geosol laboratories, in Vespasiano. The samples used in the tests were all from the Fresh Rock (FRE) sample, as this ore type is predicted to make up 90% of the industrial circuit feed, together with the relatively higher cyanide consumption as adopted to the Project.

13.3.9.1 BENCH SCALE TESTS

Ten bench scale tests were conducted for assessing the efficiency of the selected detox process conducted on tailings from leaching tests.

The tests were carried out in rolling bottles with approximately 1 kg of tailing resulting from leaching of previously ground samples to 0.125 mm P_{80}. The test conditions are described in Table 13-29, which indicates triplicate tests for each condition, together with an additional one. Such a procedure was adopted for quality control.

Table 13-29 - Bench Scale Detox Test Conditions.

Test	gSO2: gCN	Cu (mg/L)	pH Final	Residence Time (h)
Detox Test_01	5,5	50	8,5 - 9,5 w/ lime	2
Detox Test_02	5,5	50	8,5 - 9,5 w/ lime	2
Detox Test_03	5,5	50	8,5 - 9,5 w/ lime	2
Detox Test_04	6	50	8,5 - 9,5 w/ lime	2
Detox Test_05	6	50	8,5 - 9,5 w/ lime	2
Detox Test_06	6	50	8,5 - 9,5 w/ lime	2
Detox Test_07	5,5	25	8,5 - 9,5 w/ lime	2
Detox Test_08	5,5	25	8,5 - 9,5 w/ lime	2
Detox Test_09	5,5	25	8,5 - 9,5 w/ lime	2
Detox Test_10	5,5	130	8,5 - 9,5 w/ lime	2

The chemical analysis methods used were based on SMEWW (Standard Methods for the Examination of Water and Wastewater), as described in Table 13-30.

Table 13-30 - Cyanide Species Analysis Methods.

Parameter		Chemical Assays Methods
Cyanide Free	CNfree	SMEWW 4500 CN- D, E, I
Cianyde Wad	CNwad	SMEWW 4500 CN- D, E, I
Cianyde Total	CNtotal	SMEWW 4500 CN- C, D, E

Averaged results obtained according to the four testing conditions are shown in Table 13-31 which also shows the enhanced effect of the catalyst agent (source of Cu^{2+}) in the Detox process efficiency. Based on CNwad (weak acid dissociable cyanide) parameter, an average recovery of 99.5% was obtained with a catalyst dosage of 50 mg/L.

Table 13-31 - Averaged Detox Results Obtained from Bench Scale Tests.

Test	Ration gSO2/gCN	Concentration Cu2+ (mg/L)	Detox Efficiency (%)		
			CN Total	CN Wad	CN Free
Detox Test_01 @ 03	5,5	50	96,9	98,1	98,0
Detox Test_04 @ 06	6,0	50,0	98,4	99,5	99,5
Detox Test_07 @ 09	5,5	25,0	56,4	64,7	69,7
Detox Test_04 @ 08	5,5	129,6	98,4	97,0	97,2

The average of three CNwad grades obtained in the Detox test feed was 183.2 mg/L, while the CNwad in tailings after the Detox tests are those shown in Figure 13-12. Final CNwad values lower than 2 mg/L are highlighted in the same Figure 13-12 as obtained from tests 4, 5 and 6, whereas values higher than 10 mg/L obtained in tests 7, 8 and 9 were not included in the same graph.

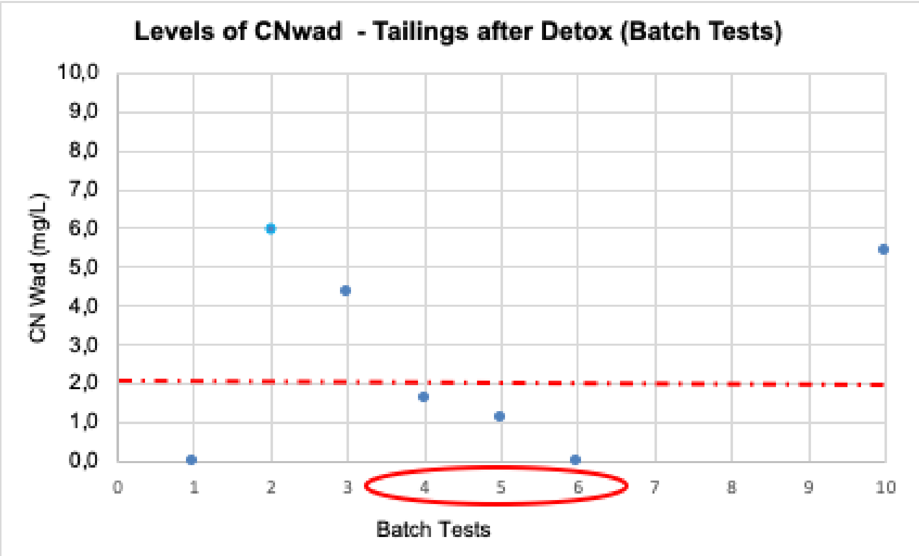


*Figure 13-12 - CN_{wad} Values in the Detox Test Tailings - Bench Scale.**

(*) Considering the control background

13.3.9.2 CONTINUOUS TESTS – PILOT SCALE

Continuous testing conducted on a pilot scale were planned on the basis of the batch test results. The adopted conditions were as follows:

- SO_2 source: sodium metabisulfite - $Na_2S_2O_5$.
- SO_2/air ratio: 6:1.
- Cu^{2+} concentration: 100 mg/L.
- pH in the range of 8.5 - 9.5 using milk of lime (hydrated lime).
- Residence time of 2.0 hours.

The tailings resulted from standard leaching tests were used in the continuous Detox tests. Figure 13-13 shows a diagram representing the SO_2/air process as adopted in the pilot plant testing. Figure 13-14 shows a photo taken from the actual pilot plant installation used for testing.

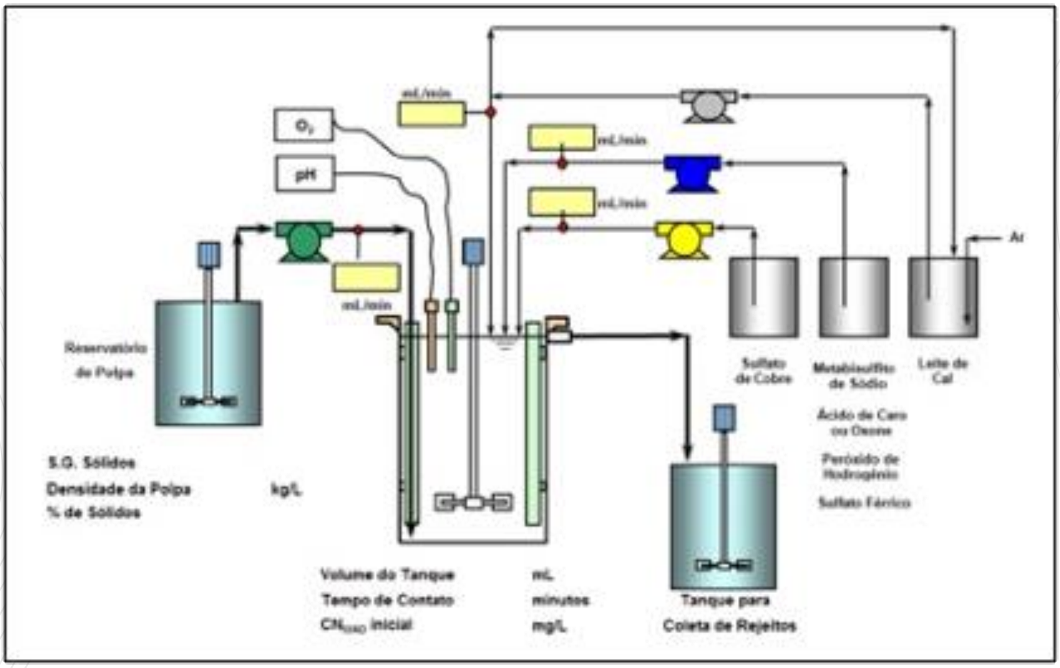


Figure 13-13 - Schematic Diagram of the Process Route Adopted for the Continuous Detox Tests.

Figure 13-14 - Photo of the Pilot Plant Used in the Detox Tests.

The continuous Detox test feed averaged 128.8 mg/L, while the final tailing concentration and efficiency figures for the CNwad parameter are listed in Table 13-32. High efficiency was achieved in the continuous cyanide removal, as all results indicated CNwad grades smaller than 2 mg/L in the final tailings.

Table 13-32 - Results of CN_{wad} in the Continuous Detox Tests.

Matupá Rock - Fresh Rock Sample		CN Wad	
Sample	**Residence Time (h)**	**mg/L**	**Treatment Efficiency (%)**
Test Feed 1	_	126,75	_
Test Feed 2	_	130,75	_
Test - 15 min	0,25	0	100,0%
Test - 30 min	0,50	0	100,0%
Test- 60 min	1,00	1,08	99,2%
Teste - 90 min	1,50	0	100,0%
Test - 120 min	2,00	0	100,0%

'(*) Consideing Background value - Blank Sample.

13.3.10 CONCLUSIONS OF DETOX TESTING

Tests conducted with leaching tailings of the Fresh Rock sample confirmed the amenability of the SO_2/air method to reduce tailings to levels smaller than 2 mg/L of CNwad. The removal efficiency was higher than 95% in both bench and continuous testing. Based on such results the adopted Detox process conditions to the Matupá Project were as follows:

- SO_2 source: sodium metabisulfite - $Na_2S_2O_5$. SO_2/air ratio: 6:1.
- Cu^{2+} source: copper sulfate pentahydrate - $CuSO_4.5H_2O$. Cu^{2+} concentration: 50 mg/L.
- pH: in the range of 8.0 - 9.5 using milk of lime (hydrated lime).
- Residence time: 2.0 hours.

An adequate monitoring and control of the industrial leaching circuit will certainly contribute to optimize the cyanide dosage, which in turn will result in minimizing the residual level of this reagent in the tailings. Such a procedure, together with the obtained efficiency of the SO_2/air method will result in cyanide levels smaller than 2 mg/L in the final tailings.

13.3.11 RHEOLOGY TESTS AND SOLID-LIQUID SEPARATION

Rheology and solid-liquid separation tests were carried out by FLSmidth Brasil, at their laboratories in Votorantim, SP.

Fresh Rock and Oxide samples were prepared at Testwork laboratories and then sent to the FLSmidth laboratory. The preparation of each sample consisted in grinding 20 kg to a P_{80} of 0.125 mm. The results were assessed by FLSmidth Salt Lake City for determining design parameters to both pre-leaching and post-leaching thickeners for the Matupá Project.

13.3.11.1 SEDIMENTATION AND RHEOLOGY TEST RESULTS

Sedimentation tests were conducted for the following unit operations: (a) thickening before the leaching circuit, here referred as pre-leaching; (b) thickening of the tailings before the filtering step, here referred as post-leaching and Detox.

A flocculant screening test was initially carried out using four different reagents. Based on the results, the BASF Magnafloc-10 was selected, which consisted of an anionic polyacrylamide flocculant with a high molecular weight and short charge density.

Assessments of pre-leaching sedimentation tests resulted in a minimum unit area of 0.045 m^2/tpd for the Fresh Rock sample and 0.079 m^2/tpd for Oxide sample. Accordingly, post-leaching testing resulted in minimum unit area of 0.079 m^2/tpd and 0.124 m^2/tpd, respectively for Fresh Rock and Oxide samples.

Data obtained in sedimentation tests and derived parameters are listed in Table 13-33 and Table 13-44, respectively for pre-leaching and post-leaching stages. Equipment design and details are shown in Table 13-33 were based on 161 tph of solids at 35% w/w, which is the predicted feed rate and concentration of solids to the Matupá leaching stage. Table 13-34 design results were also based on the same flowrate of solids, but at a 45% concentration of solids in the pulp.

Table 13-33 - Sedimentation Testing and Thickener Designing Results - Pre-Leaching.

Process Parameters	Fresh	Oxidized
Thickener Model	HRT (Hi-Rate Thickener)	
Solids Conc. Recommend. (wt.%)	10.0	8.0
Underflow		
Underflow solids (wt.%)	45 -50	
Underflow tension (Pa)	5-20	
Required Residence Time (h)	1.0 - 2.0	
Overflow		
Particulate (ppm)	< 200	
Flocculant		
Recommended flocculant	BASF Magnafloc 10	
Recommended Concentration (g/L)	0.1 – 0.25	
Recommended Dosage (g/t)	7- 10	12 – 15
Assay Parameter*		
Unit area (m^2/tpd)	0.045 (min).	0.079 (min.)
Sedimentation rate (tpd/m^2)	19	13
Sedimentation speed Initial (m/h)	16.2	13.8
Recommended Scaling**		
Amount	1	1
Diameter/Height Min.(m)	15/2.3	20/2.3
Recommended Drive Unit	LL-130	LL – 130

Table 13-34 - Sedimentation Testing and Thickener Designing Results - Post-Leaching.

Process Parameters	**Fresh**	**Oxidized**
Thickener Model	HRT (Hi-Rate Thickener)	
Solids Conc. Recommend. (wt.%)	10.0	8.0
Underflow		
Underflow solids (wt.%)	55 -60	
Underflow tension (Pa)	10-60	
Required Residence Time (hrs)	2.0 - 4.0	
Overflow		
Particulate (ppm)	< 200	
Flocculant		
Recommended flocculant	BASF Magnafloc 10	
Recommended Concentration (g/L)	0.1 – 0.25	
Recommended Dosage (g/t)	7- 10	12 – 15
Assay Parameter*		
Unit area (m^2/tpd)	0.079 (min).	0.124 (min.)
Sedimentation rate (tpd/m^2)	19	13
Sedimentation speed Initial (m/h)	16.2	13.8
Recommended Scaling**		
Amount	1	1
Diameter/Height Min.(m)	20/2.7	25/2.7
Recommended Drive Unit	LL-300	LL –750

Rheological test results for the post-leaching stage are summarized in Table 13-35. A significant reduction in the dynamic viscosity with increasing shear rate is observed for both samples in the two tested concentrations of solids. Such data are particularly relevant to pump and pipe design for the industrial circuit.

Table 13-35 - Rheology Testing Results.

Shear Rate	**Dynamic Viscosity (cP)**			
(S^{-1})	**Fresh, 60%**	**Fresh, 55%**	**Oxidized, 60%**	**Oxidized, 55%**
5	4500	3100	3200	1800
25	1281	890	779	452
50	828	504	440	240
75	612	370	309	178
100	518	295	243	138
150	400	207	183	107
200	338	147	151	86
250	280	103	134	77

13.3.11.2 FILTERING TEST RESULTS

Filtration tests were carried out for Fresh Rock and Oxide samples for both vacuum and press alternatives. Pulp concentration of solids in test feeding was based on results obtained for the high-rate thickener underflow of post-leaching stage, as previously shown in Table 13-34.

Table 13-36 and 13-37 summarize test results and design parameters, respectively for vacuum and press filtration. In each case Fresh Rock and Oxide sample data are listed. Design of industrial equipment was based on the leaching circuit nominal feed rate of 161 tph of solids, as well as on 55-60% concentration of solids in the pulp.

Table 13-36 - Summary of Data and Design Parameters for Vacuum Filtration.

Sample	Fresh Rock		Oxide	
Assay Method	**Top**	**Bottom**	**Top**	**Bottom**
Filter Type	Horizontal	Discs	Horizontal	Discs
Solids in feed (wt.%)	55.0 – 60.0		55.0 – 60.0	
Pulp alkalinity (pH)	7.0		9.5 – 10.5	
Operation Vacuum (“Hg)	20 - 21	22 - 23	23 - 24	
Cake thickness (mm)	25 – 30	15 - 20	12 - 17	9 - 14
Cake weight (kg/m^2)	31	20	16	12
Cycle time (min.)	1.1	1.3	0.7	0.8
Cake moisture (wt.%)	18 - 20	19.5 – 21.5	21 - 23	22 - 24
Filtered Solids (ppm)	2,400		2,950	
Filtration rate (kg/m^2/h)	1,372	750	1,031	704
Min. Area Required (m^2)	120	220	160	235

Table 13-37 - Summary of Data and Design Parameters for Press Filtration.

Process Parameters	Fresh Rock	Oxide
Chamber technology	Recess	
Solids in feed (wt.%)	55 – 60	
Pulp alkalinity (pH)	6.5 – 7.5	
Operation pressure (bar)	10 – 15	
Chamber thickness (mm)	50	32
Test cycle time (min.)	3.5	4
Cake moisture (wt.) %)	15 - 17	
Filtered Solids (ppm)	2000	2500
Filtration rate (kg/m^2/hr) *	215	180
Filter model/Number of plates	AFP M2020FBM/108	AFP M2020FBM/144
Amount required	1	1

Adequate consolidation and detachment from filter cloth were observed in cakes resulting from both vacuum and press filtering tests. Cake moisture derived from press filtration tests were in the 15-17% range.

13.4 THIRD METALLURGICAL CAMPAIGN

A third metallurgical campaign was carried out based on the progression of the Life of Mine (LOM) plan as well as the basic engineering of Matupa project. This metallurgical campaign included the following two parts, consolidation and variability.

Consolidation:

- Consolidate a gold metallurgical recovery as a function of gold grade in order to estimate gold production according to the blending predicted by the LOM plan.
- This part was based on blends and gold grades as follows:
  - Year 1 to 3: Blend 1 (COMP 1): 90% Fresh Rock (FRE) and 10% Oxide (OXI).
  - From years 4 onwards: Blend 2 (COMP 2): 90% Fresh Rock (FRE) and 10% Oxide (OXI).
    - Further assessments were carried out on following samples:
  - Fresh Rock and Oxide samples for comminution testing.
  - Blends for physical and geotechnical characterization of tailings for dry stacking:
    - Blend 3 (COMP 3): 90% Fresh Rock (FRE) and 10% Oxide (OXI).
    - Blend 4 (COMP 4): 33% Fresh Rock (FRE) and 67% Oxide (OXI).

Variability:

- Assess the effects of gold grade and metallurgical performance variability for both Fresh Rock and Oxide ore types.
- Four blends were assembled as follows:
  - Variability OXI_LG – Oxide (VAR1) – Au grade lower than 0.6 g/t (Au < 0.6 g/t).
  - Variability OXI_HG – Oxide (VAR2) – Au grade higher than 0.6 g/t (Au > 0.6 g/t).
  - Variability FRE_LG – Fresh Rock (VAR3) – Au grade lower than 1.0 g/t (Au < 1.0 g/t).
  - Variability FRE_HG – Fresh Rock (VAR4) – Au grade higher than 1.0 g/t (Au > 1.0 g/t).

13.4.1 CAMPAIGN PLANNING

The third metallurgical campaign was planned to include the assessments summarized in Table 13-38.

Table 13-38 - Summary of Testing planned for the Matupá project Third Metallurgical Campaign.

Step / Phase	Part	Sample ID	Sample Characteristic	Study / Characterization
3rd Metallurgical Campaign	Consolidation	COMP 1	Blend: Year 1 to Year 3 - 90% FRE_1 & 10% OXI_1	Chemical and metallurgical characterization and tests
		COMP 2	Blend: Year 4 to Year 7 - 90% FRE_2 & 10% OXI_2	Chemical and metallurgical characterization and tests
		COMP 3	Blend 90% FRE_3 & 10% OXI_3	Phisical Characterization / Dry Stacking
		COMP 4	Blend 1/3 FRE_4 & 2/3 OXI_4	Phisical Characterization / Dry Stacking
		FRE	Fresh Rock	Comminution Characterization
		OXI	Oxide Ore	Comminution Characterization
	Variability	VAR 1 or OXI_LG	Variability OXI_LG	Chemical and metallurgical characterization and tests
		VAR 2 or OXI_HG	Variability OXI_HG	Chemical and metallurgical characterization and tests
		VAR 3 or FRE_LG	Variability FRE_LG	Chemical and metallurgical characterization and tests
		VAR 4 or FRE_HG	Variability FRE_HG	Chemical and metallurgical characterization and tests

13.4.2 SAMPLE PREPARATION

Samples from Fresh Rock and Oxide ore types were selected and obtained by the Aura geology team. A total of 812 kg of samples were selected and prepared as listed in Table 13-39 and Table 13-40.

Table 13-39 - Summary of Sample Selection and Preparation for the Matupá Project Third Metallurgical Campaign.

Phase	N.	Sample's Lot	Number of Samples	Reference	Weight (kg)	Weighted Average - Au (g/t) According Geology Assays	
Variability	1	RNM 2021 - 567	29	Variability OXI_LG	50,00	0,35	_
	2	RNM 2021 - 568	25	Variability OXI_HG	50,60	0,80	_
	3	RNM 2021 - 569	33	Variability FRE_LG	50,85	0,74	_
	4	RNM 2021 - 570	27	Variability FRE_HG	49,80	1,51	_
Consolidation	5	RNM 2021 - 571	4	Consolidation - Composite 1 - Oxi	7,05	1,56	1,55
	6	RNM 2021 - 572	46	Consolidation - Composite 1 - FRE	63,15	1,55	
	7	RNM 2021 - 573	4	Consolidation - Composite 2 - OXI	7,30	0,70	0,69
	8	RNM 2021 - 574	39	Consolidation - Composite 2 - FRE	63,35	0,69	
	9	RNM 2021 - 575	20	Comminution - Oxi	51,15	_	_
	10	RNM 2021 - 576	13	Comminution - FRE	51,65	_	_
	11	RNM 2021 - 577	12	Physical Characterization - Composite 3 - Oxi	35,35	_	_
	12	RNM 2021 - 578	76	Physical Characterization - Composite 3 - FRE	300,75	_	_
	13	RNM 2021 - 579	8	Physical Characterization - Composite 4 - OXI	20,50	_	_
	14	RNM 2021 - 580	3	Physical Characterization - Composite 4 - FRE	10,80	_	_
Total			**339**	_	**812,3**	_	_

Table 13-40 – Samples Obtained for the Matupá Third Metallurgical Campaign.

Characterization and Tests		Weight (Kg)
Variability Tests - Typology & Grade		
FRE_HG	Fresh Rock - Au > 1,0 g/t	49,80
FRE_LG	Fresh Rock - Au <1,0 g/t	50,85
OXI_HG	Oxide Ore - Au > 0,6 g/t	50,60
OXI_LG	Oxide Ore - Au < 0,6 g/t	50,00
Total		**201,25**
Characterization and Tests		**Weight (Kg)**
Comminution Characterization - Typlology		
FRE	Fresh Rock	51,65
OXI	Oxide Ore	51,15
Total		**102,8**
Characterization and Tests		**Weight (Kg)**
Consolidation Tests (according LOM) - Composites: Typlology & Grade		
COMP 1	Blend: Year 1 to Year 3 - 10% OXI_1 and 90% FRE_1	70,20
COMP 2	Blend: Year 4 to Year 7 - 10% OXI_2 and 90% FRE_2	70,65
Total		**140,85**
Characterization and Tests		**Quantidade (Kg)**
Physical Characterization / Dry Stacking - Composites: Typlology		
COMP 3	Blend 10% OXI_3 and 90% FRE_3	336,10
COMP 4	Blend 2/3 OXI_4 and 1/3 FRE_4	31,30
Total		**367,4**
Total - Global		**812,3**

Sample preparation was carried out at SGS-Geosol in their Vespasiano laboratories. Derived subsamples were subsequently sent to other laboratories for further testing. QA/QC directions were followed throughout the entire preparation campaign. For metallurgical testing each composite was separately crushed to 2.0 mm (10# Tyler). After homogenization and splitting subsamples were separated for full chemical analysis (Head Assay), as well as for characterization testing.

13.4.3 COMMINUTION TESTING

Comminution tests were carried out on both Fresh Rock and Oxide samples at SGS-Geosol at their Vespasiano (Brazil) and Santiago (Chile) laboratories. The SMC test results were validated by JKTech (Australia). A summary of comminution tests carried out on the third campaign is shown in Table 13-41, while Table 13-42 summarizes the results.

Table 13-41 – Summary of Comminution Tests Carried - Third Metallurgical Campaign.

Sample	Weight Received (kg)	Tests (Commintion Characterization)			
		BWI (Closing sieve)	Ai Bond	SMC ®	SG
Fresh Rock	51,7	Yes (0,150 mm)	Yes	Yes	Yes
Oxide Ore	51,2	Yes (0,150 mm)	Yes	No	Yes

Table 13-42 – Summary of Comminution Test Results - Third Metallurgical Campaign.

Test Characteization	Parameters	Ore (SAMPLES)	
		Fresh Rock - FRE	Oxide Ore - OXI
BWI Test	Bond Ball Mill Work Index (kWh/metric ton) Closing screen - 0,150 mm	19,1	19,9
	P80 - BWI Test (microns)	122	121
SMC Test ® (Parameters derived from SMC Test)	A * b	46,0	_
	t_a	0,43	_
	SCSE (Kwh/t)	9,35	_
Ai Test	Abrasion Index	0,151	0,069

The SMC test results indicated a A*b parameter of 46.0 for the Fresh Rock sample, which is a very close value compared with the corresponding index obtained by Mineral Processing Solutions Ltda. (MinPro Solutions) in the second campaign. This test was not carried out on the Oxide sample as no coarse fragments were available for testing.

The BWI tests carried out with a 0.150 mm closing screen resulted in 19.1 kWh/t for the Fresh Rock sample and 19.9 kWh/t for the Oxide sample.

Further testing for obtaining the specific gravity for both samples was conducted. All three campaign results are listed in Table 13-43.

Table 13-43 – Summary of Specific Gravity Test Results – All Three Metallurgical Campaigns.

Specific Gravity (g/cm3) - SUMMARY					
Phase	Laboratory	Sample	SG (g/m3)	Average	Tests and Sample's Preparation
1st Phase	SGS Mineral Service Lakefield (2019)	High Sulfide	2,89	2,76	SGS Mineral Service Lakefield
		Low Sulfide	2,69		
		Oxide	2,70		
2nd Phase	MinPro (2021)*	Fresh Rock - FRE (DH022) / Sample B	2,76	2,670	MinPro Solutions
		Fresh Rock - FRE (DH024) / Sample C	2,58		
		Oxide Ore - OXI (SEDH06) / Sample B	2,57	2,615	
		Oxide Ore - OXI (SEDH34) / Sample C	2,66		
3rd Phase	SGS Mineral Service Chile (2021/2022)	Fresh Rock Composite	2,75	2,75	- Sample Preparation SGS-Geosol - Tesst SGS Min. Serv. Chile - Evaluation / Report JKTech

* *Results - Average of triplicate.*

13.4.4 CHEMICAL ANALYSIS RESULTS

The composite samples selected for both Consolidation and Variability parts of the third campaign were chemical assayed at SGS-Geosol laboratories according to the methods listed in Table 13-44.

Table 13-44 – Summary of the Methods Adopted for Chemical Analysis of the Selected Elements.

SGS Identification	Chemical Analysis Method
ICP40B	Multi-acid digestion (20 mL) - 0,25 g sample - ICP OES
CSA17V	High sulphur grade - Resistive furnace method
CSA20V	Carbon - Infrared LECO
CVA02B	Mercury - Cold vapour and AAS
AAS12E	Siver - Acid digestion and AAS
AAS40B	Copper - Muti-acid digestion and AAS
FAA505	Gold - Fire Assay - AAS - 50 g fusion
FAASCR_150	Gold Fire Assay / Metalic Screen ground at 0.106 mm (150 Mesh Tyler)

13.4.4.1 RESULTS - CONSOLIDATION

Individual and averaged gold grades obtained from both fire assay and metallic screen methods for the Head Sample are listed in Table 13-45, while Table 13-46 include the calculated standard and mean deviations.

Table 13-45 – Chemical Analysis Results - Head Samples.

Sample	Metallic Screen FAA505 - Au (g/t)			Fire Assay - Au (g/t)			FA Triplicate Average
	Assay 1	Assay 2	Average	Assay 1	Assay 2	Assay 3	Au (g/t)
COMP 1	2,06	1,78	1,92	1,654	2,947	2,595	2,40
COMP 2	0,62	0,72	0,67	0,729	0,646	1,413	0,93
SGS's Certificate: BM2100697; BM2100703; BM2100704							

COMP 1 and COMP 2 samples resulted in gold grades respectively of 1.92 g/t and 0.67 g/t Au, therefore showing the reduction predicted by the LOM plan.

Table 13-46 – Chemical Analysis Deviations - Head Samples.

Sample	Metallic Screen		Fire Assay - FA	
	Standard Deviation	Mean Deviation	Standard Deviation	Mean Deviation
COMP 1	0,199	0,141	0,668	0,496
COMP 2	0,075	0,053	0,421	0,322

Table 13-46 indicates higher deviations for the fire assay method compared with the metallic screen. Further chemical analysis were conducted for the selected elements listed in Table 13-47.

Table 13-47 – Full Chemical Analysis Results - Head Samples.

Methd	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B
Element	Ag	Al	As	Ba	Be	Bi	Ca	Cd	Co	Cr	Cu	Fe	K
Sample	ppm	%	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%	%
COMP 1	6	3,23	<10	743	<3	<20	0,04	<3	10	10	376	3,67	3,19
COMP 2	<3	3,47	<10	659	<3	<20	0,08	<3	8	7	334	3,05	3,19

Methd	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B
Element	La	Li	Mg	Mn	Mo	Na	Ni	P	Pb	S	Sb	Sc	Se
Sample	ppm	ppm	%	%	ppm	%	ppm	%	ppm	%	ppm	ppm	ppm
COMP 1	<20	24	0,18	0,03	39	0,17	3	0,01	185	2,69	<10	<5	<20
COMP 2	<20	24	0,22	0,03	45	0,38	4	0,02	129	1,89	<10	<5	<20

Methd	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B
Element	Sn	Sr	Th	Ti	Tl	U	V	W	Y	Zn	Zr
Sample	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm
COMP 1	<20	5	<20	0,06	<20	<20	12	<20	3	616	64
COMP 2	<20	14	<20	0,06	<20	<20	10	<20	4	242	71

Methd	CVA02B	CSA17V	CSA20V	CSA20V	CSA20V	AAS12E	AAS40B
Element	Hg	S	C_ORG	C_ELE	C_CAR	Ag	Cu
Sample	ppm	%	%	%	%	ppm	ppm
COMP 1	<0,05	2,8	<0,05	<0,05	<0,05	7,5	387
COMP 2	<0,05	2,01	0,05	<0,05	<0,05	3,6	349

SGS's Certificate: BM2100699

Silver grades were respectively 7.5 g/t and 3.6 g/t Ag (AAS12E), which indicates a potential benefit for the Project.

The size distributions curves for both COMP 1 and COMP 2 samples are shown in Figure 13-15, while Figure 13-16 shows the gold distribution by size for the same two samples. These results were described in SGS's Certificate: BM2100828; BM2100829.

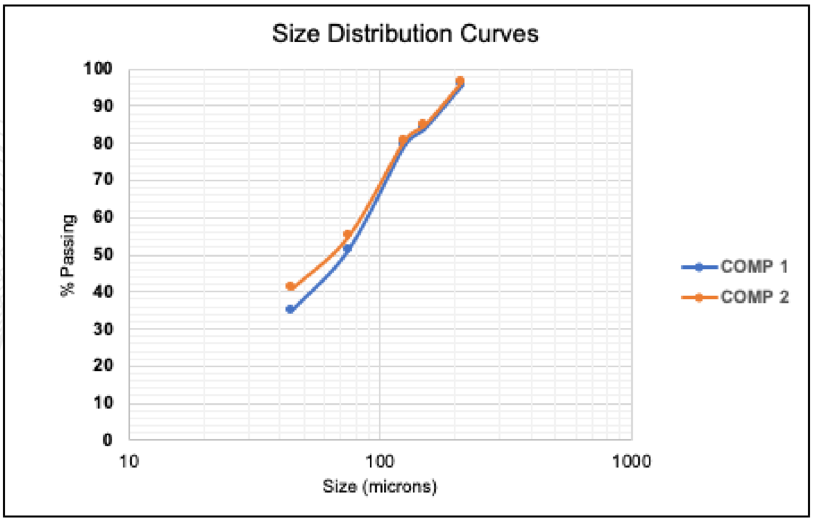


Figure 13-15 – Size Distribution - COMP 1 and COMP 2 Samples.

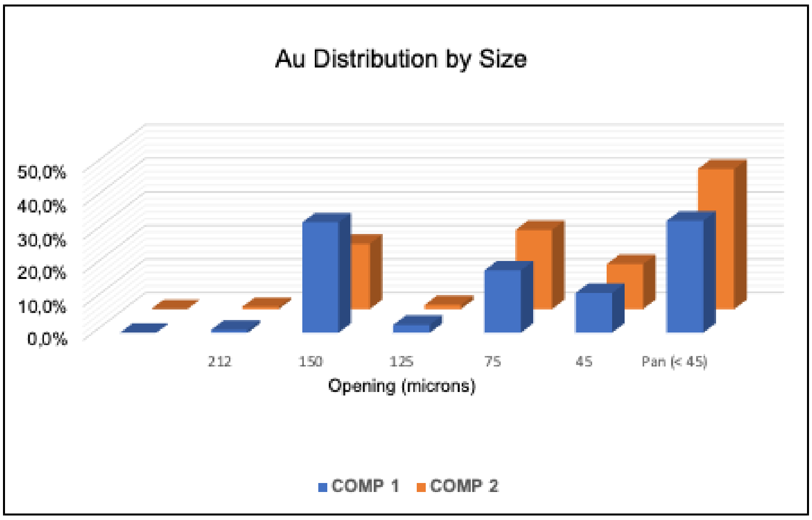


Figure 13-16 – Gold Distribution by Size - COMP 1 and COMP 2 Samples.

13.4.4.2 RESULTS - VARIABILITY

Individual and averaged gold grades obtained from both fire assay and metallic screen methods for the Variability Head Sample are listed in Table 13-48, while Table 13-49 shows the calculated standard and mean deviations.

Table 13-48 – Chemical Analysis Results - Variability Head Samples.

Sample	Metallic Screen FAA505 - Au (g/t)			Fire Assay - Au (g/t)			FA Triplicate Average
	Assay 1	Assay 2	Average	Assay 1	Assay 2	Assay 3	Au (g/t)
VAR 1	0,54	0,51	**0,53**	0,563	1,035	0,650	**0,75**
VAR 2	0,96	1,04	**1,00**	0,774	1,077	1,007	**0,95**
VAR 3	0,91	0,90	**0,91**	0,612	0,725	0,820	**0,72**
VAR 4	1,65	1,63	**1,64**	2,543	1,616	1,576	**1,91**
SGS's Certificate: BM2100696; BM2100702; BM2100701							

Table 13-49 – Chemical Analysis Deviations - Variability Head Samples.

Sample	Metallic Screen		Fire Assay - FA	
	Standard Deviation	Mean Deviation	Standard Deviation	Mean Deviation
VAR 1	0,022	0,016	0,251	0,190
VAR 2	0,055	0,039	0,159	0,119
VAR 3	0,007	0,005	0,104	0,071
VAR 4	0,019	0,013	0,547	0,421

Here too, higher deviations were observed for the fire assay methods compared to the metallic screen one.

Further chemical analyses were conducted for the selected elements, as listed in Table 13-50.

Table 13-50 – Full Chemical Analysis Results - Variability Head Samples.

Methd	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B
Element	Ag	Al	As	Ba	Be	Bi	Ca	Cd	Co	Cr	Cu	Fe	K
Sample	ppm	%	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%	%
VAR 1	<3	5,13	<10	650	<3	<20	0,02	<3	<8	8	36	3,38	2,8
VAR 2	<3	5,08	<10	680	<3	<20	0,02	<3	<8	8	45	3,54	3,12
VAR 3	5	4,29	<10	655	<3	<20	0,12	<3	15	8	339	3,87	3,36
VAR 4	5	4,52	<10	699	<3	<20	0,04	<3	<8	5	656	3,32	3,14

Methd	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B
Element	La	Li	Mg	Mn	Mo	Na	Ni	P	Pb	S	Sb	Sc
Sample	ppm	ppm	%	%	ppm	%	ppm	%	ppm	%	ppm	ppm
VAR 1	<20	23	0,17	0,02	45	0,15	<3	0,02	20	0,12	<10	<5
VAR 2	<20	20	0,15	0,02	46	0,13	<3	0,02	53	0,22	<10	<5
VAR 3	<20	28	0,23	0,04	32	0,48	<3	0,02	261	3,07	<10	<5
VAR 4	<20	19	0,13	0,03	54	0,27	<3	<0,01	247	2,53	<10	<5

Methd	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B	ICP40B
Element	Se	Sn	Sr	Th	Ti	Tl	U	V	W	Y	Zn	Zr
Sample	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm
VAR 1	<20	<20	42	<20	0,07	<20	<20	18	<20	<3	18	66
VAR 2	<20	<20	34	<20	0,07	<20	<20	15	<20	<3	24	73
VAR 3	<20	<20	15	<20	0,06	<20	<20	10	<20	5	977	67
VAR 4	<20	<20	5	<20	0,05	<20	<20	9	<20	<3	503	63

Methd	CVA02B	CSA17V	CSA20V	CSA20V	CSA20V	AAS12E	AAS40B	Sulfate	Sufide
Element	Hg	S	C_ORG	C_ELE	C_CAR	Ag	Cu		
Sample	ppm	%	%	%	%	ppm	ppm	% S	% S
VAR 1	<0,05	0,14	0,06	<0,05	<0,05	0,4	39	0,04	0,10
VAR 2	<0,05	0,24	<0,05	<0,05	<0,05	3	46	0,05	0,20
VAR 3	<0,05	3,31	<0,05	<0,05	0,06	6,2	339	<0,02	2,55
VAR 4	<0,05	2,83	<0,05	<0,05	0,06	6,7	662	0,60	2,46

SGS's Certificate: BA2100086; BM2100698

Silver grades for VAR 3 and VAR 4 samples were both 5.0 g/t Ag (ICP40B), which indicates potential revenue for the Project.

The size distributions curves for all four variability samples are shown in Figure 13-51, while Figure 13-17 shows the gold distribution by size for the same four samples. A relatively large amount of gold is observed in the fraction passing at 0.045 mm (45 µm).

Table 13-51 – Size distributions - Variability Head Samples.

Size Distribution				
Size (µm)	VAR 1 % Accum Passing	VAR 2 % Accum Passing	VAR 3 % Accum Passing	VAR 4 % Accum Passing
212	97,6	97,1	95,3	96,0
150	83,4	82,7	84,3	85,3
125	**79,9**	**80,3**	**79,4**	**80,4**
75	59,1	57,4	58,7	53,7
45	41,4	41,5	44,2	38,3

SGS's Certificate: BM2100736; BM2100780

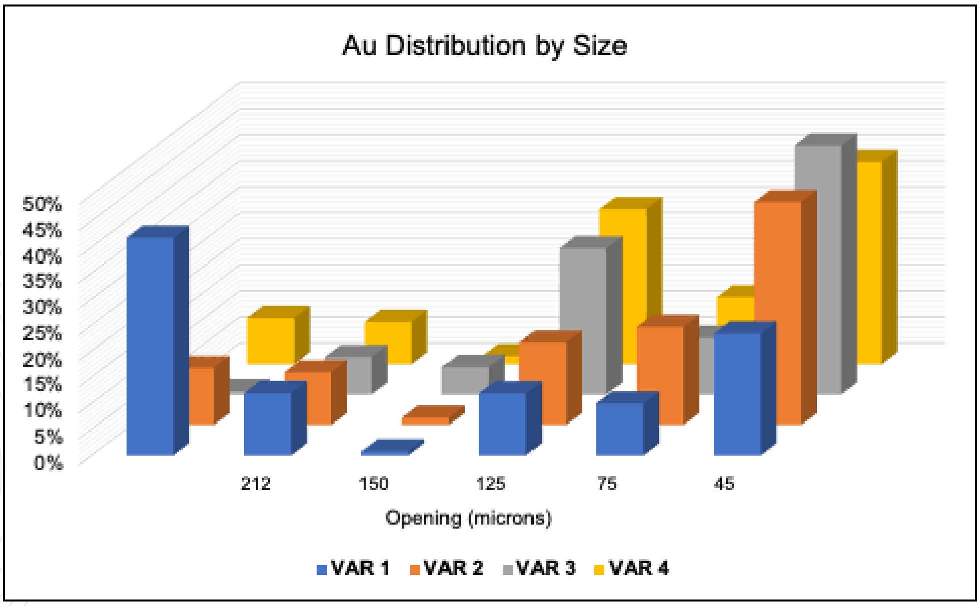


Figure 13-17 – Gold Distribution by Size - Variability Samples.

13.4.5 METALLURGICAL TESTING RESULTS - CONSOLIDATION

Metallurgical testing was carried out using two process flow sheets: (a) direct leaching and (b) gravity concentration in combination with leaching.

13.4.5.1 DIRECT LEACHING TESTS

Direct leaching tests were carried out with COMP1 and COMP2 samples. All tests were conducted in triplicate using approximately 1 kg aliquots. Testing conditions used in all tests are listed below.

- Grinding size: P_{80} of 0.125 mm.
- Total residence time: 28 h.
  - Leaching: 7 h.
  - CIL circuit: 21 h.
- Concentration of solids by weight: 50%.
- Concentration of activated carbon: 25 g/L.
- pH: 10 to 11 obtained with lime (Calcium Hydroxide).
- Concentration of cyanide (NaCN): 1 kg/L.
- Concentration of dissolved Oxygen: >4 mg/L.

Leaching tests were carried out on rolling bottles. Samples were obtained during the tests for monitoring the following parameters: free cyanide (CN_{FREE}), pH, Eh (Oxi-reduction potential) and dissolved oxygen (DO). Adjustments were conducted during the tests when necessary.

Solid/liquid separations were carefully carried out at the end of each test, avoiding contamination and/or losses. Samples as follows were then prepared and sent to SGS-Geosol for assaying.

- Solids (tailings): mass, gold, silver, and copper analysis. Gold analysis was carried out in duplicate by Fire Assay;
- Liquid (solution): volume, gold, silver, and copper analysis. Gold analysis was carried out in duplicate; and
- Carbon (loaded): mass and gold analysis.

Gold and silver recoveries, together with specific consumptions were calculated on the basis of test results, as shown in Table 13-52 and Table 13-53.

Table 13-52 – Leaching Test Results for Gold - Consolidation Head Samples.

Direct Leaching LCIL - Au													
Sample	ID SGS GEOSOL	Au (ppm)						Extraction Efficiency Au (%) Back calculated	Consumption Lime (gpt)	CNfree FINAL (ppm)	Consumption NaCN (gpt)	Average Extraction Efficiency Au (%) Back calculated
		Head Grade Assays Au (ppm)	Head - Back calculated Au (ppm)	Head - Back calculated Au (ppm)	Liquid Tail Au (mg/L)	Loaded Carbon Au (ppm)	Solid Tail Au (ppm)					
COMP 1	TESTE - 37	2,06	1,90	1,95	0,01	56,2	0,08	95,7%	279	232	562	95,60%
	TESTE - 38			2,11	0,01	61,3	0,08	96,4%	281	214	596	
	TESTE - 39			1,64	0,01	46,7	0,09	94,7%	281	154	709	
COMP 2	TESTE - 40	0,71	0,75	0,76	0,01	21,9	0,03	96,2%	281	142	732	95,32%
	TESTE - 41			0,70	0,01	19,9	0,03	95,3%	283	142	732	
	TESTE - 42			0,79	0,01	22,2	0,04	94,5%	280	198	626	
SGS's Certificate: BM2100849; BM2100853; BM2200062												

Table 13-53 – Leaching Test Results for Silver - Consolidation Head Samples.

Direct Leaching LCIL - Ag												
Sample	ID SGS GEOSOL	Ag (ppm)						Extraction Efficiency Ag (%) Back calculated	Consumption Lime (gpt)	CNfree FINAL (ppm)	Consumption NaCN (gpt)	Average Extraction Efficiency Ag (%) Back calculated
		Head Grade Assays Ag (ppm)	Head - Back calculated Ag (ppm)	Head - Back calculated Ag (ppm)	Liquid Tail Ag (mg/L)	Loaded Carbon Ag (ppm)	Solid Tail Ag (ppm)					
COMP 1	TESTE - 37	6,75	5,46	4,80	0,10	103,0	1,30	72,9%	279	232	562	76,58%
	TESTE - 38			5,82	0,10	134,0	1,30	77,7%	281	214	596	
	TESTE - 39			5,76	0,10	135,0	1,20	79,1%	281	154	709	
COMP 2	TESTE - 40	3,31	3,11	3,28	0,10	75,0	0,70	78,6%	281	142	732	78,54%
	TESTE - 41			3,31	0,10	76,0	0,70	78,8%	283	142	732	
	TESTE - 42			2,75	0,10	62,0	0,60	78,2%	280	198	626	
SGS's Certificate: BM2100849; BM2100853; BM2200062												

Averaged gold and silver recoveries were all higher than 95% and 76% respectively. The averaged consumptions were as follows:

- Lime - $Ca(OH)_2$: 280 g/t.
- Sodium cyanide - NaCN: 623 g/t for the COMP1 sample and 697 g/t for the COMP2 sample.

13.4.5.2 GRAVITY CONCENTRATION AND LEACHING TESTS

A laboratory-scale Knelson centrifugal concentrator was used to conduct gravity concentration tests, whose tailings were homogenized and split for leaching tests, the latter following the same procedures as previously adopted and described in Section 13.4.5.1. The obtained results are described in Table 13-54 and Table 13-55.

Table 13-54 – Gravity Concentration Test Results - Consolidation Samples.

Gravity Concentration (KNELSON) - Au						
Sample	ID SGS GEOSOL	Au (ppm)				Effic. Conc. Au (%) Back Calculated
		Head Grade Assays Au (ppm)	Head - Back calculated Au (ppm)	Concentrate Au (ppm)	Tail Au (ppm)	
COMP 1	Test A	**2,06**	1,58	91,97	0,70	**56,0%**
COMP 2	Test B	**0,71**	0,69	41,5	0,29	**58,0%**

Table 13-55 – Leaching Test Results - Consolidation Samples.

Direct Leaching LCIL - Au												
Sample	ID SGS GEOSOL	Au (ppm)						Extraction Efficiency Au (%) Back calculated	Consumption Lime (gpt)	CNfree FINAL (ppm)	Consumption NaCN (gpt)	Average Extraction Efficiency Au (%) Back calculated
		Head Grade Assays Au (ppm)	Head - Back calculated Au (ppm)	Head - Back calculated Au (ppm)	Liquid Tail Au (mg/L)	Loaded Carbon Au (ppm)	Solid Tail Au (ppm)					
COMP 1	TESTE - 43	0,70	0,79	0,81	0,01	22,0	0,07	**91,1%**	284	148	721	91,60%
	TESTE - 44			0,79	0,01	21,6	0,06	**92,2%**	280	142	732	
	TESTE - 45			0,78	0,01	21,4	0,07	**91,5%**	280	142	732	
COMP 2	TESTE - 46	0,29	0,31	0,31	0,01	8,00	0,03	**89,3%**	263	118	777	90,55%
	TESTE - 47			0,31	0,01	8,13	0,03	**90,2%**	268	124	766	
	TESTE - 48			0,31	0,02	8,13	0,02	**92,1%**	276	126	762	

Averaged leaching test results for gold recovery were higher than 90.5%. These gravity test results, Table 13-54, are considered higher than those obtained in the corresponding industrial scale operation based on similar works. In this case, the gravity concentration test was carried out only for obtaining mass for conducting leaching tests on tailings. It is here recommended to adopt the E-GRG (Extended Gravity Recoverable Gold) test results as described in Section 13.3.5, for predicting the performance associated with the industrial gravity concentration circuit.

13.4.5.3 GOLD RECOVERY RELATIONSHIP

Direct Leaching

Direct leaching data were clustered according to back-calculated gold grades resulting from corresponding tests, as listed in Table 13-56. Averaged gold recoveries were associated with each cluster resulting in values listed in Table 13-57, which were plotted in the graph shown in Figure 13-18.

Table 13-56 – Gold Grades and Recoveries Obtained per Test and Derived Clustering - Direct Leaching - Consolidation.

Test SGS ID	Head Back Calc. Au (g/t)	Gold Recovery (%)	Group
TESTE - 38	2,108	96,4%	grupo 1
TESTE - 37	1,949	95,7%	grupo 2
TESTE - 39	1,639	94,7%	grupo 3
TESTE - 40	0,761	96,2%	grupo 4
TESTE - 41	0,700	95,3%	
TESTE - 42	0,787	94,5%	
TESTE - 43	0,809	91,1%	
TESTE - 44	0,785	92,2%	
TESTE - 45	0,783	91,5%	
TESTE - 46	0,307	89,3%	grupo 5
TESTE - 47	0,309	90,2%	
TESTE - 48	0,313	92,1%	

Table 13-57 – Averaged Gold Grades and Recoveries for Each Cluster - Direct Leaching - Consolidation.

Class / Group	Au Grade (g/t) Average	Recov. Au (%)
group 1	2,108	96,4%
group 2	1,949	95,7%
group 3	1,639	94,7%
group 4	0,630	91,5%
group 4	0,313	92,1%

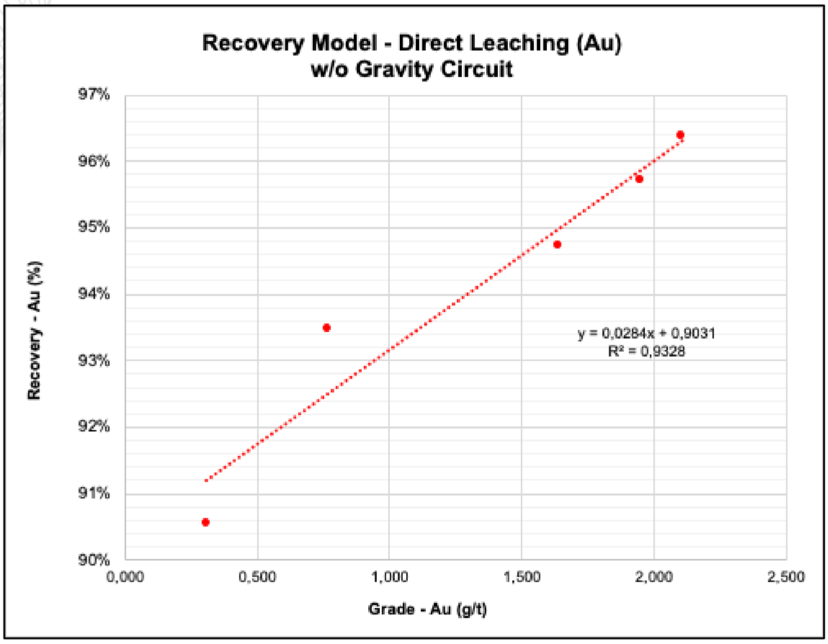


Figure 13-18 – Au Recovery Model - Direct Leaching Alternative - Consolidation.

A linear regression was selected to constrain the gold recovery as a function of gold grade for the direct leaching alternative. The correlation coefficient (R^2) for the derived equation was 0.9328. In this case:

$$\text{Direct Leaching Au Recovery (\%)} = 100 \times [\text{Au Grade (g/t)} \times 0.0284 + 0.9031] \quad (1)$$

Gravity Concentration and Leaching

Leaching data derived from the Gravity/Leaching route were clustered according to back-calculated gold grades resulting from corresponding tests, as listed in Table 13-58. Averaged gold recoveries of leaching tests were associated with each cluster resulting in values listed in Table 13-59, which were plotted in the graph shown in Figure 13-19.

Gravity concentration modelling was based on E-GRG together with the FLSmidth (Knelson – Canada), the latter based on the Amira model and the FLSmidth extensive industrial data. Simulations based on these models were carried out for the 90% Fresh Rock 10% Oxide blend. A value of 98% Au, recommended by FLSmidth based on their data base and experience, was used as the gravity concentrate value in the LOM Adopted value for gold recovery (extraction).

Leaching modelling was based on the results previously described in Section 13.4.5.2., according to the respective gold grades calculated for the tailings of the gravity concentration stage.

Table 13-58 – Leaching Gold Grades and Recoveries Obtained per Test and Derived Clustering - Consolidation.

Test SGS ID	Head Back Calc. Au (g/t)	Gold Recovery (%)	Group
TESTE - 38	1,949	95,71%	Group 1
TESTE - 39	1,639	94,73%	Group 2
TESTE - 43	0,809	91,14%	Group 3
TESTE - 44	0,785	92,19%	
TESTE - 45	0,783	91,48%	
TESTE - 46	0,307	89,35%	Group 4
TESTE - 47	0,309	90,17%	
TESTE - 48	0,313	92,12%	

Table 13-59 – Leaching Averaged Gold Grades and Recoveries for Each Cluster - Consolidation.

Class / Group	Au Grade (g/t) Average	Recov. Au (%)
group 1	1,949	95,7%
group 2	1,639	94,7%
group 3	0,792	91,6%
group 4	0,309	90,5%

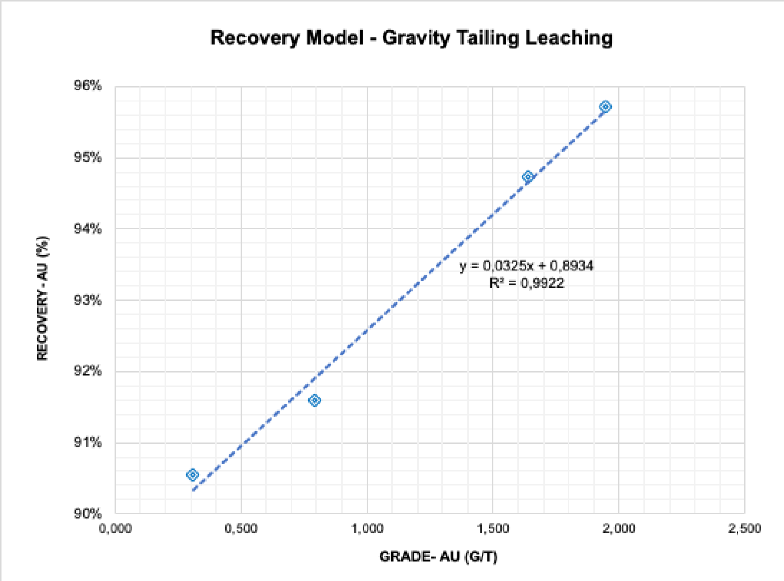


Figure 13-19 – Gold Recovery Model for Leaching in the Gravity/Leaching Route - Consolidation.

A linear regression was selected to parameterize the gold recovery as a function of gold grade for the leaching stage of the gravity/leaching alternative. The correlation coefficient (R^2) for the derived equation was 0.9922. In this case:

Gold Recovery in Leach Tailings (%) = 100 x [Au Grade in Gravity Tailings (g/t) x 0.0325 + 0.8934] (2)

Gravity gold recovery was estimated on the FLSmidth modelling, which was based on a centrifugal concentrator processing the grinding circuit circulating load (cyclone underflow). The results obtained in the second metallurgical campaign indicated the following gold recoveries: 36.5% for the FRE sample, as well as 12.6% for the OXI sample.

Based on a 9:1 (FRE:OXI) blend, the weighted gravity gold recovery resulted in 34.11%. In this case the gold extraction was 98%. The overall gravity gold recovery was thus 33.42% (34.11% x 0.98).

For the gravity-leaching process route the equation developed to estimate the overall gold recovery (OGR) (%) based on Gold Head Grade (GHG) (g/t) was as follows:

$$OGR = 33.42 + (100 - 33.42) \times \{[GHG \,.\, (1 - 0.3342)] \times 0.0325 + 0.8934\} \quad (3)$$

Which is equivalent to:

$$OGR = 92.9 + 1.44 \times GHG \quad (4)$$

The graph shown in Figure 13-20 shows the gold recovery as a function of the processing plant gold head grade for both Direct Leaching (w/o Gravity) and Gravity-Leaching (w/ Gravity) routes. The former distribution was based on equation (1), while the latter was based on equation (4).

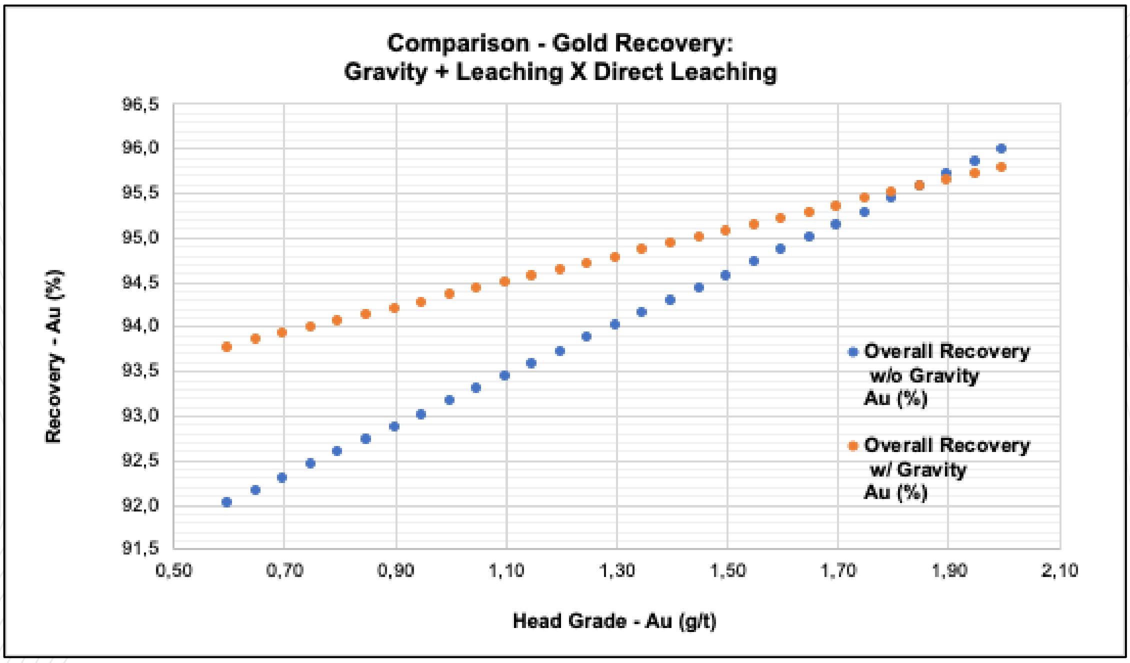


Figure 13-20 – Gold Recovery for Direct Leaching and Gravity/Leaching Routes - Consolidation.

The graph shown in Figure 13-20 indicates higher gold recoveries for the Gravity/Leaching route compared to the Direct Leaching route for Gold Head Grades smaller than 1.85 g/t.

Based on such results, the Gravity/Leaching route should be implemented to the Matupá Project and equation (4) should thus be adopted to estimate the gold recovery for the LOM as a function of gold head grade for the industrial plant feed.

13.4.6 METALLURGICAL TESTING RESULTS - VARIABILITY

While Consolidation metallurgical testing was carried out for estimating the gold recovery for the LOM, a parallel campaign was carried out to estimate gold recovery for various composites, which were as follows:

- VAR1 OXI_LG – Oxide (Low Grade) – Gold grade lower than 0.6 g/t (Au < 0.6 g/t).
- VAR2 OXI_HG – Oxide (High Grade) – Gold grade equal or higher than 0.6 g/t (Au ≥ 0.6 g/t).
- VAR 3 FRE_LG – Fresh Rock (Low Grade) – Gold grade lower than 1.0 g/t (Au < 1.0 g/t).
- VAR4 FRE_HG – Fresh Rock (High Grade) – Gold grade equal or higher than 1.0 g/t (Au ≥ 1.0 g/t).

13.4.6.1 Direct Leaching Tests

Direct leaching tests were carried out in the same conditions as previously described in Section 13.4.5.1. Tests were carried out in triplicates for each one of the composites. All samples had been previously ground to a P_{80} of 0.125 mm. Data obtained from such tests are listed in Table 13-60. Test 3 (Teste-3) gold grade in carbon was abnormally high, which increased the back calculated head grade. These two values were thus considered outliers.

Table 13-60 – Gold Grades and Recoveries Obtained per Test - Direct Leaching - Variability.

Au Direct Leaching LCIL										
ID	ID SGS-Geosol Test Number	Au (ppm)						Extraction Efficiency Au (%), by Head Assays	Extraction Efficiency Au (%), by Back Calculated Grade	Extraction Efficiency Au (%), by Carbon Recovery
		Mean Head Grade Assays Au (ppm)	Mean Back Calculated Head Grades	Back Calculated Head Grades Au (ppm)	Liquid Tail Au (mg/L)	Loaded Carbon Au (ppm)	Solid Tail Au (ppm)			
OXI LG VAR 1	TESTE - 1	0,66	0,70	0,73	0,01	20,8	0,04	95,9%	94,8%	94,0%
	TESTE - 2			0,66	0,01	19,0	0,03	97,1%	95,9%	95,0%
	TESTE - 3			1,69	0,01	40,4	0,35	62,1%	79,1%	78,8%
OXI HG VAR 2	TESTE - 4	0,93	0,89	0,88	0,01	24,9	0,05	95,0%	94,7%	94,0%
	TESTE - 5			0,81	0,01	22,1	0,08	91,8%	90,5%	89,8%
	TESTE - 6			0,98	0,01	27,1	0,08	91,5%	91,9%	91,3%
FRE LG VAR 3	TESTE - 7	0,82	0,80	0,81	0,01	23,7	0,02	97,0%	97,0%	96,3%
	TESTE - 8			0,76	0,01	22,0	0,03	96,5%	96,2%	95,4%
	TESTE - 9			0,81	0,01	23,4	0,03	95,9%	95,9%	95,1%
FRE HG VAR 4	TESTE - 10	1,68	1,61	1,73	0,01	50,0	0,07	95,6%	95,7%	95,4%
	TESTE - 11			1,56	0,01	43,5	0,12	92,8%	92,3%	91,9%
	TESTE - 12			1,55	0,01	43,7	0,10	93,9%	93,4%	93,0%
SGS's Certificate: BM2100735; BM2100738; BM2100781										

Gold recovery calculations were based on back calculated grades i.e., gold contained in tailings and loaded carbon. Figure 13-21 shows a comparison between back calculated gold grades and corresponding values obtained by chemical analysis.

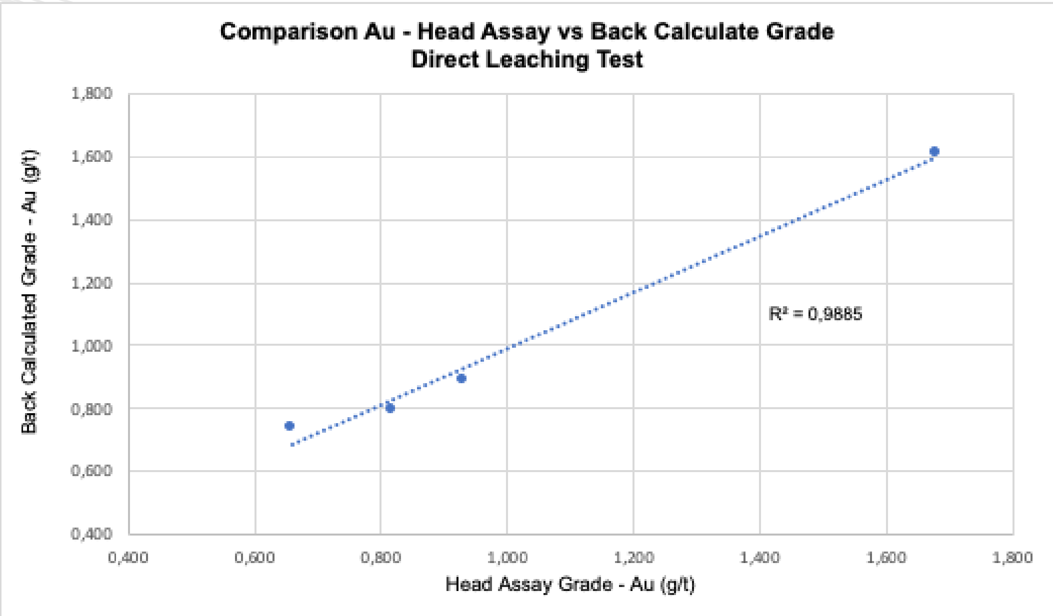


Figure 13-21 – Back Calculated Gold Grades and Assays Obtained by Chemical Analysis - Direct Leaching Alternative - Variability.

Student statistics were used to assess the variability of each composite. Table 13-61 shows the results obtained for 95% confidence level. Variability of gold grades and recoveries were higher for both VAR 2 OXI HG and VAR 4 FRE HG composites.

Table 13-61 – Statistics of Gold Grades and Recoveries - Direct Leaching - Variability.

VAR 1 - OXI_LG			VAR 2 - OXI_HG		
ID SGS-Geosol	Back Calculated Grade Au (g/t)	Efficiency Extraction Au (%), by Back Calculated Head	ID SGS-Geosol	Back Calculated Grade Au (g/t)	Efficiency Extraction Au (%), by Back Calculated Head
TESTE - 1	0,73	94,80%	TESTE - 4	0,88	94,71%
TESTE - 2	0,66	95,92%	TESTE - 5	0,81	90,55%
TESTE - 3	1,69	79,15%	TESTE - 6	0,98	91,90%
Mean	**0,696**	**95,36%**	**Mean**	**0,889**	**92,38%**
St Deviation	_	_	**St Deviation**	**0,085**	**2,12%**
IC 95 %	_	_	**IC 95 %**	**0,212**	**5,27%**
VAR 3 - FRE_LG			**VAR 4 - FRE_HG**		
ID SGS-Geosol	Back Calculated Grade Au (g/t)	Efficiency Extraction Au (%), by Back Calculated Head	ID SGS-Geosol	Back Calculated Grade Au (g/t)	Efficiency Extraction Au (%), by Back Calculated Head
TESTE - 7	0,81	97,00%	TESTE - 10	1,73	95,73%
TESTE - 8	0,76	96,19%	TESTE - 11	1,56	92,29%
TESTE - 9	0,81	95,86%	TESTE - 12	1,55	93,38%
Mean	**0,795**	**96,35%**	**Mean**	**1,615**	**93,80%**
St Deviation	**0,030**	**0,59%**	**St Deviation**	**0,101**	**1,75%**
IC 95 %	**0,074**	**1,47%**	**IC 95 %**	**0,251**	**4,36%**

As previously emphasized, the variability assessment of VAR 1 OXI LG was biased due to Test-3 results. The other three variability results are plotted in Figure 13-22.

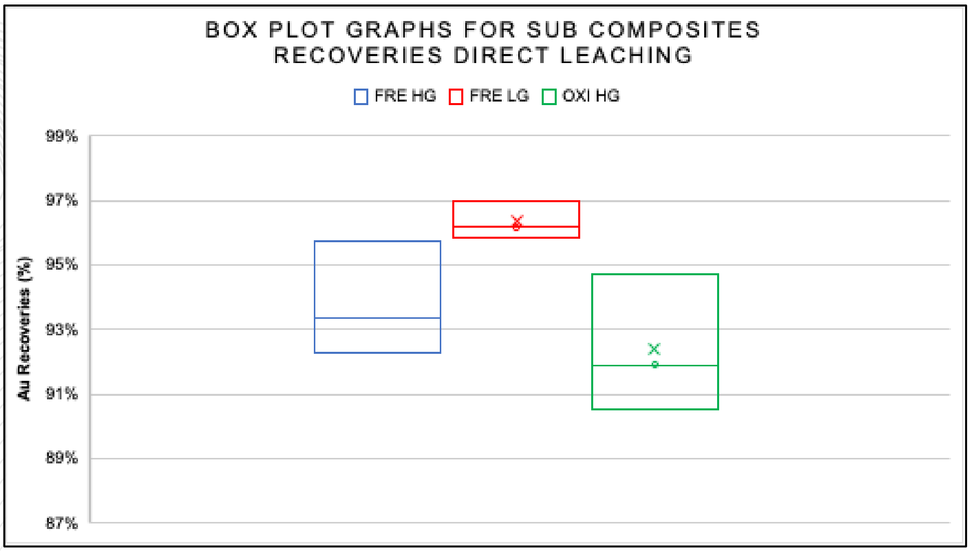


Figure 13-22 – Box Plot of Gold Recovery Variability - Direct Leaching Alternative - Variability.

13.4.6.2 GRAVITY CONCENTRATION TESTS

Single stage gravity concentration tests were carried out in duplicate for all four composites. Even though the amount of concentrate obtained in batch laboratory tests are not directly correlated to industrial conditions, the tests were instrumental to obtain tailings for subsequent leaching tests. Table 13-62 shows the test results, which also contains both back calculated gold grades and head gold assays obtained by chemical analysis.

Table 13-62 – Gravity Concentration Test Results - Variability.

KNELSON Concentration - Au											
ID	ID SGS-Geosol Test Number	Grade - Au (g/t)					Mass			Effic. Conc. Au (%), by Head Assay	Effic. Conc. Au (%), Back Calc.
		Mean Head Assays Au (g/t)	Mean Back Calc. Grade Au (g/t)	Back Calc. Grade Au (g/t)	Concentrate Au (g/t)	Tail Grade Au (g/t)	Feed (g)	Conc (g)	Tail (g)		
OXI LG	1 A	0,66	1,01	0,91	134	0,23	10000	51	9949	104%	74,8%
	1 B			1,11	160	0,26	10000	53	9947	129%	76,3%
OXI HG	2 A	0,93	0,90	0,96	89,4	0,43	10000	60	9940	58%	55,8%
	2 B			0,83	66,2	0,41	10000	65	9935	46%	51,6%
FRE LG	3 A	0,82	0,81	0,81	38,6	0,43	10000	98	9902	46%	46,9%
	3 B			0,82	46,3	0,44	10000	83	9917	47%	46,9%
FRE HG	4 A	1,68	1,73	1,82	137	0,72	10000	81	9919	64%	61,0%
	4 B			1,63	116	0,69	10000	82	9918	57%	58,2%
SGS's Certificate: BM2100740; BM2100750											

Figure 13-23 shows a comparison between back calculated gold grades and corresponding values obtained by chemical analysis.

Composite	ALM Au (g/t) **EXP** (Head Assay)	ALM Au (g/t) **CALC** (Back Calc.)
FRE HG	1,68	1,82
	1,68	1,63
FRE LG	0,82	0,81
	0,82	0,82
OXI HG	0,93	0,96
	0,93	0,83
OXI LG	0,66	0,91
	0,66	1,11

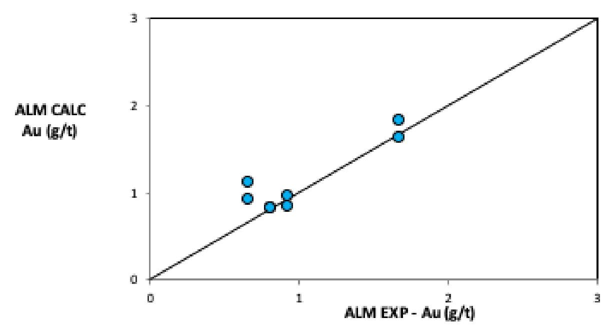


Figure 13-23 – Back Calculated Gold Grades and Assays Obtained by Chemical Analysis – Gravity Concentration Tests - Variability.

Figure 13-23 shows that back calculated values for the OXI_LG were significantly different from corresponding head assays obtained by chemical analysis. These values were adjusted by minimum squares methods as available in the BILCO software. Figure 13-24 shows plotted gravity concentration test gold recoveries as a function of respective back calculated grades. No direct correlation was observed between these two variables.

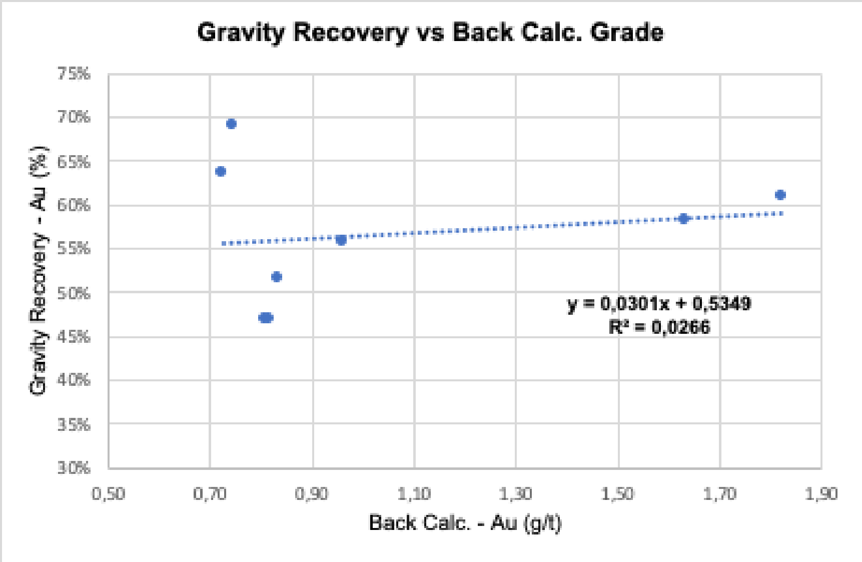


Figure 13-24 – Back Calculated Gold Grades and Gravity Tests Gold Recoveries - Variability.

Figure 13-25 shows the gravity gold recoveries as obtained to each one of the four composites.

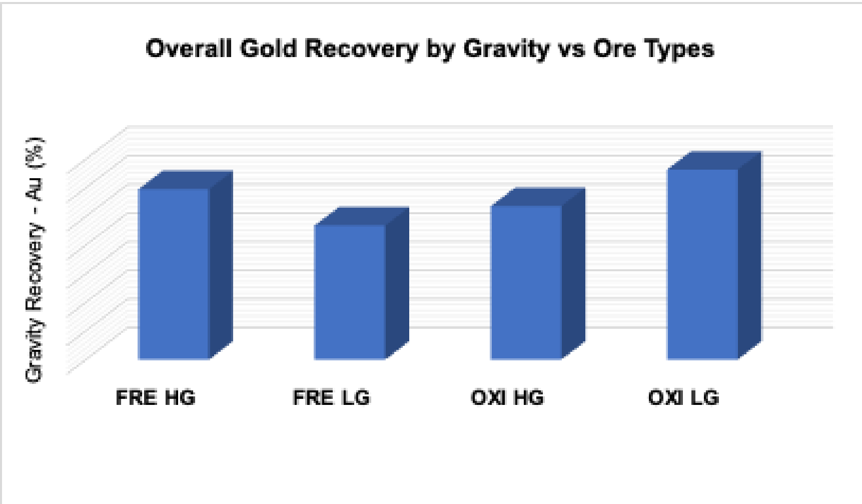


Figure 13-25 – Overall Gold Recovery for each Composite – Gravity Concentration Tests - Variability.

13.4.6.3 LEACHING TESTS OF GRAVITY TAILINGS

Gravity tailings were leached in triplicate, following the conditions previously described in Section 13.4.5.1. These included both A and B samples obtained from each gravity concentration tests, as previously listed in Table 13-62. Each composite tailing was thus tested six times, as listed in Table 13-63 (A samples) and Table 13-64 (B samples).

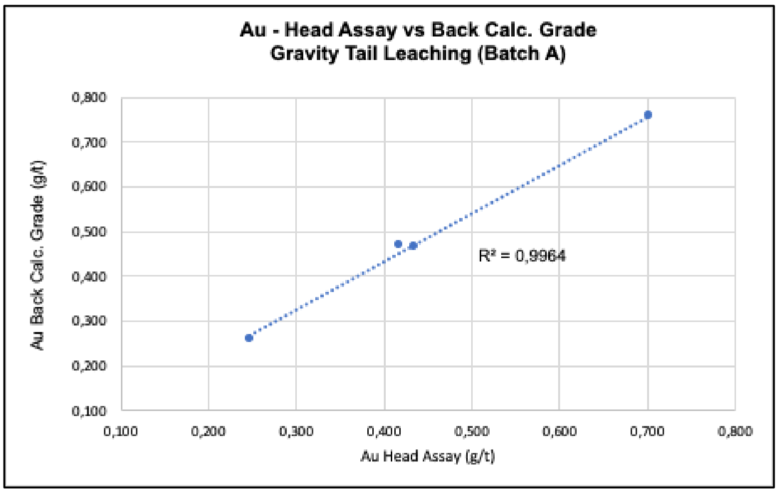
Table 13-63 – Results of Leach Tests Carried Out on Gravity Concentration Tails – Samples A - Variability.

Gravity Tail Leaching - Au (Batch A)										
ID	ID SGS-Geosol Test Number	Au (ppm)						Extraction Efficiency Au (%), by Head Assays	Extraction Efficiency Au (%), by Back Calculated Grade	Extraction Efficiency Au (%), by Carbon Recovery
		Mean Head Grade Assays Au (ppm)	Mean Back Calculated Head Grades	Back Calculated Head Grades Au (ppm)	Liquid Tail Au (mg/L)	Loaded Carbon Au (ppm)	Solid Tail Au (ppm)			
OXI LG VAR 1	TESTE - 13	0,25	0,26	0,26	0,02	6,3	0,03	87,1%	**87,7%**	80,1%
	TESTE - 14			0,26	0,02	6,4	0,03	88,6%	**89,1%**	81,4%
	TESTE - 15			0,26	0,03	6,0	0,03	87,8%	**88,3%**	76,7%
OXI HG VAR 2	TESTE - 16	0,42	0,47	0,46	0,04	11,6	0,04	90,3%	**91,2%**	82,6%
	TESTE - 17			0,47	0,04	11,9	0,04	90,1%	**91,3%**	82,8%
	TESTE - 18			0,47	0,04	11,7	0,05	88,6%	**89,9%**	81,5%
FRE LG VAR 3	TESTE - 19	0,44	0,46	0,47	0,04	12,3	0,02	94,3%	**94,7%**	86,2%
	TESTE - 20			0,46	0,04	11,6	0,04	91,2%	**91,7%**	83,0%
	TESTE - 21			0,46	0,01	12,9	0,03	93,4%	**93,8%**	91,7%
FRE HG VAR 4	TESTE - 22	0,70	0,76	0,80	0,05	20,6	0,07	90,7%	**91,8%**	85,5%
	TESTE - 23			0,75	0,04	19,6	0,06	91,1%	**91,6%**	86,3%
	TESTE - 24			0,72	0,04	19,1	0,05	92,3%	**92,5%**	87,0%
SGS's Certificate: BM2100776; BM2100778; BM2100815										

Table 13-64 – Results of Leach Tests Carried Out on Gravity Concentration Tails – Samples B - Variability.

Gravity Tail Leaching - Au (Batch B)										
ID	ID SGS-Geosol Test Number	Au (ppm)						Extraction Efficiency Au (%), by Head Assays	Extraction Efficiency Au (%), by Back Calculated Grade	Extraction Efficiency Au (%), by Carbon Recovery
		Mean Head Grade Assays Au (ppm)	Mean Back Calculated Head Grades	Back Calculated Head Grades Au (ppm)	Liquid Tail Au (mg/L)	Loaded Carbon Au (ppm)	Solid Tail Au (ppm)			
OXI LG VAR 1	TESTE - 25	0,25	0,27	0,27	0,01	7,1	0,03	89,0%	**89,8%**	87,5%
	TESTE - 26			0,28	0,01	7,5	0,03	88,8%	**90,1%**	88,0%
	TESTE - 27			0,27	0,02	6,7	0,03	87,2%	**88,3%**	81,0%
OXI HG VAR 2	TESTE - 28	0,42	0,46	0,45	0,01	12,2	0,03	92,1%	**92,6%**	90,4%
	TESTE - 29			0,48	0,02	12,7	0,04	91,1%	**92,2%**	88,0%
	TESTE - 30			0,46	0,01	12,5	0,04	90,8%	**91,7%**	89,5%
FRE LG VAR 3	TESTE - 31	0,44	0,46	0,47	0,02	12,8	0,03	94,0%	**94,5%**	90,2%
	TESTE - 32			0,45	0,01	12,6	0,03	93,7%	**93,8%**	92,5%
	TESTE - 33			0,46	0,03	12,2	0,03	93,7%	**94,0%**	87,5%
FRE HG VAR 4	TESTE - 34	0,70	0,75	0,76	0,03	20,9	0,05	93,5%	**94,0%**	90,1%
	TESTE - 35			0,77	0,03	21,2	0,05	93,5%	**94,1%**	90,2%
	TESTE - 36			0,71	0,01	20,0	0,04	94,2%	**94,2%**	93,4%
SGS's Certificate: BM2100802; BM2100803; BM2100835										

Good correlation was observed between back calculated and head assays obtained for gravity concentration tailings, as shown in Figure 13-26. These grades correspond to the feed of the leaching tests.



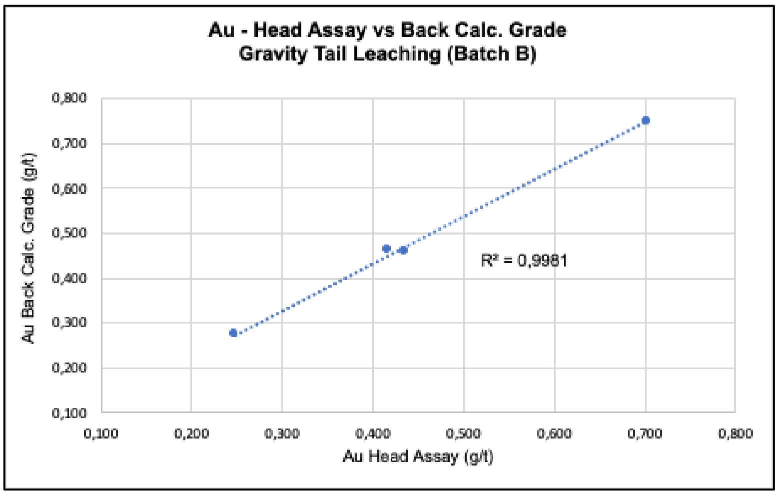


Figure 13-26 – Back Calculated and Head Assays Gold Grades Obtained for Gravity Concentration Test Tailings - Variability.

Student statistics were used to assess the variability of leaching test results carried out on gravity tailings. Table 13-65 shows the results obtained for samples A and B at the 95% confidence level.

Table 13-65 – Statistics of Gold Back calculated Grades and Recoveries – Leaching of Gravity Tailings - Variability.

VAR 1 - OXI_LG (Batch A)			VAR 1 - OXI_LG (Batch B)		
ID SGS-Geosol	Back Calculated Grade Au (g/t)	Efficiency Extraction Au (%), by Back Calculated Head	ID SGS-Geosol	Back Calculated Grade Au (g/t)	Efficiency Extraction Au (%), by Back Calculated Head
TESTE - 13	0,26	87,74%	TESTE - 25	0,27	89,79%
TESTE - 14	0,26	89,08%	TESTE - 26	0,28	90,12%
TESTE - 15	0,26	88,31%	TESTE - 27	0,27	88,34%
Mean	**0,260**	**88,38%**	**Mean**	**0,273**	**89,42%**
St Deviation	**0,001**	**0,67%**	**St Deviation**	**0,006**	**0,94%**
IC 95 %	**0,002**	**1,68%**	**IC 95 %**	**0,016**	**2,35%**
VAR 2 - OXI_HG (Batch A)			**VAR 2 - OXI_HG (Batch B)**		
ID SGS-Geosol	Back Calculated Grade Au (g/t)	Efficiency Extraction Au (%), by Back Calculated Head	ID SGS-Geosol	Back Calculated Grade Au (g/t)	Efficiency Extraction Au (%), by Back Calculated Head
TESTE - 16	0,46	91,25%	TESTE - 28	0,45	92,61%
TESTE - 17	0,47	91,26%	TESTE - 29	0,48	92,16%
TESTE - 18	0,47	89,92%	TESTE - 30	0,46	91,68%
Mean	**0,470**	**90,81%**	**Mean**	**0,461**	**92,15%**
St Deviation	**0,005**	**0,77%**	**St Deviation**	**0,015**	**0,47%**
IC 95 %	**0,012**	**1,92%**	**IC 95 %**	**0,037**	**1,16%**
VAR 3 - FRE_LG (Batch A)			**VAR 3 - FRE_LG (Batch B**		
ID SGS-Geosol	Back Calculated Grade Au (g/t)	Efficiency Extraction Au (%), by Back Calculated Head	ID SGS-Geosol	Back Calculated Grade Au (g/t)	Efficiency Extraction Au (%), by Back Calculated Head
TESTE - 19	0,47	94,75%	TESTE - 31	0,47	94,46%
TESTE - 20	0,46	91,69%	TESTE - 32	0,45	93,83%
TESTE - 21	0,46	93,83%	TESTE - 33	0,46	94,00%
Mean	**0,465**	**93,42%**	**Mean**	**0,460**	**94,10%**
St Deviation	**0,004**	**1,57%**	**St Deviation**	**0,011**	**0,33%**
IC 95 %	**0,010**	**3,90%**	**IC 95 %**	**0,026**	**0,81%**
VAR 4 - FRE_HG (Batch A)			**VAR 4 - FRE_HG (Batch B)**		
ID SGS-Geosol	Back Calculated Grade Au (g/t)	Efficiency Extraction Au (%), by Back Calculated Head	ID SGS-Geosol	Back Calculated Grade Au (g/t)	Efficiency Extraction Au (%), by Back Calculated Head
TESTE - 22	0,80	91,76%	TESTE - 34	0,76	94,02%
TESTE - 23	0,75	91,64%	TESTE - 35	0,77	94,10%
TESTE - 24	0,72	92,54%	TESTE - 36	0,71	94,21%
Mean	**0,757**	**91,98%**	**Mean**	**0,749**	**94,11%**
St Deviation	**0,037**	**0,49%**	**St Deviation**	**0,036**	**0,09%**
IC 95 %	**0,091**	**1,22%**	**IC 95 %**	**0,089**	**0,23%**

A boxplot of gold recovery variabilities is shown in Figure 13-27. Higher variabilities were obtained for both FRE_LG Sample A and OXI_LG Sample B.

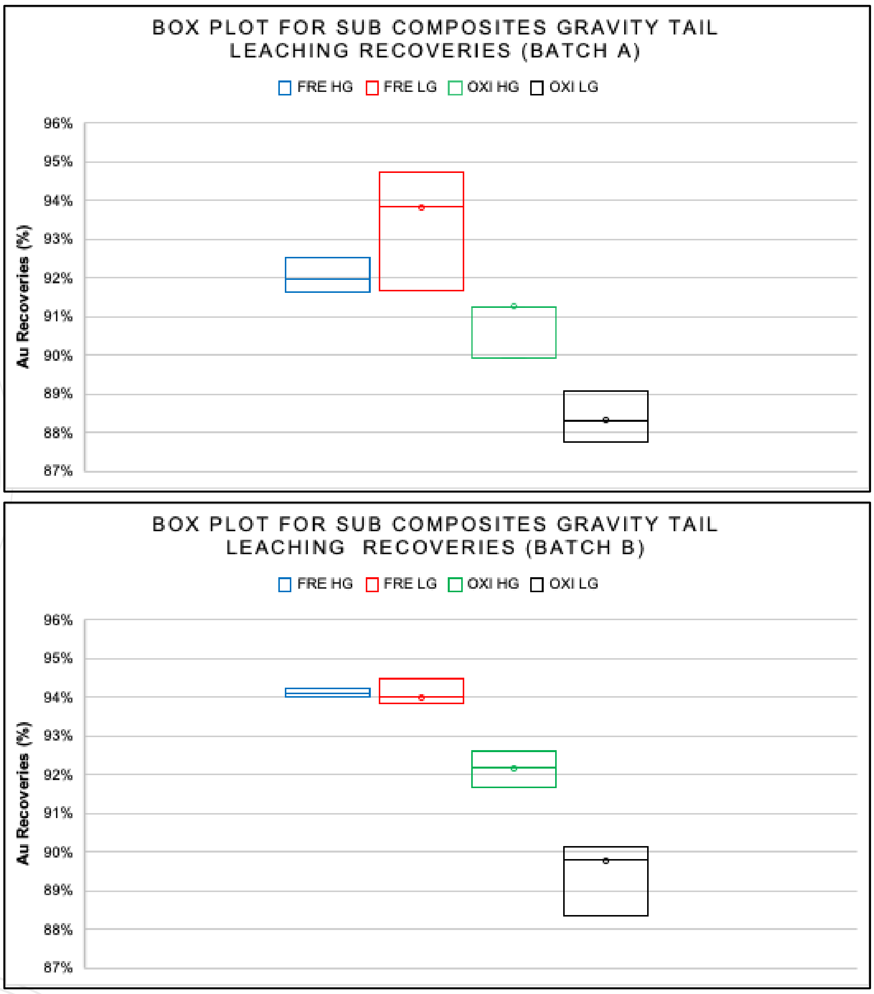


Figure 13-27 – Box Plot of Gold Recovery Variability - Gravity Tailings Leaching - Variability.

Figure 13-28 summarizes results obtained for both gravity concentration tests and tailing leaching tests. It is here emphasized that gold recoveries obtained in gravity tests must be modelled for obtaining realistic values for the industrial circuit.

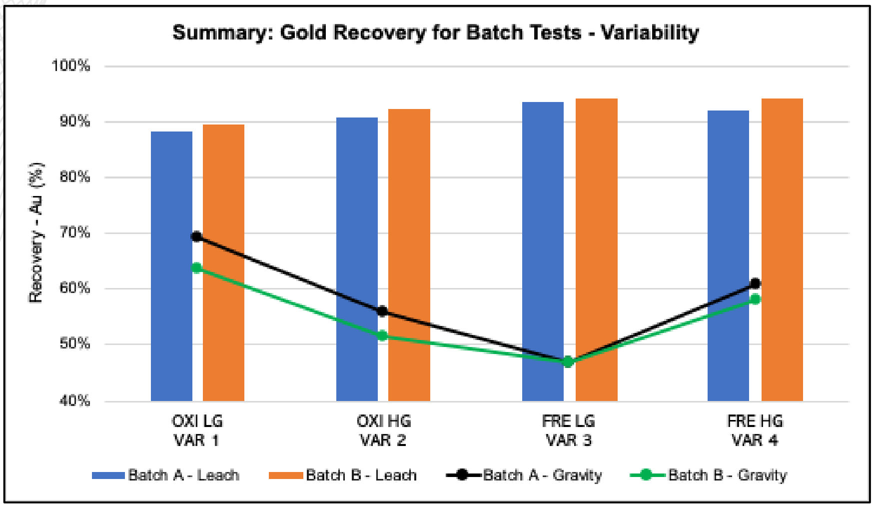


Figure 13-28 – Summary of Testing Results – Gravity/Leaching - Variability.

13.4.7 RHEOLOGY AND SOLID-LIUID SEPARATION TESTING

The third metallurgical campaign also included additional rheology and solid-liquid separation tests carried out by FLSmidth Brasil, at their laboratories in Votorantim, SP. These tests included two samples: the first (COMP3) represented the LOM, while the second (COMP4) represented an increased Oxide ore type in the blend, as follows:

- COMP 3 (90% FRE e 10% OXI).
- COMP 4 (33% FRE e 67% OXI).

Three scenarios for the solid-liquid separation stages were initially considered for the Matupá Project, as listed in Table 13-66. Most of these scenarios were assessed in the second metallurgical testing campaign, as previously described in Section 13.3.11. Additional tests were thus conducted in the third campaign as described throughout this section.

Table 13-66 – Scenarios for the Solid-Liquid Separation.

Solid-Liquid Parameters		Scenario 1	Scenario 2	Scenario 3
Cyclones O/F	% w/w	35*	35*	42
U/F 1st Thickner	% w/w	50	50	_
CIL	% w/w	50	50	42
Detox	% w/w	46-47	46-47	38 - 40
Feed 2nd Thickner	% w/w	46-47	_	38 - 40
U/F 2nd Thickner	% w/w	55 - 60	_	55 - 60
Filtration Feed	% w/w	55 - 60	46-47	55 - 60
Filtration Cake Moisture	% w/w	To be Determined	To be Determined	To be Determined

** HDA Simulation were carried out with 35% solids at Cyclone O/F including pre-CIL thickener (PLAN HDA-Promon - Matupa - 01-21 - Rev 0 - 17 Nov 2021)*

Sample preparation carried out at SGS-Geosol labs consisted of grinding 30 kg to a P_{80} of 0.125 mm. The results were assessed by FLSmidth at Salt Lake City for determining design parameters.

13.4.7.1 RHEOLOGY TEST RESULTS

Rheology tests were conducted for two circuit alternatives:

- Leaching with no previous thickening – 38% solid concentration by weight; and
- Leaching with previous thickening – 50% solid concentration by weight together with post-leaching Detox at 46% solid concentration by weight.

Rheological test results are summarized in Table 13-67, as well as plotted in the graph shown in Figure 13-29.

Table 13-67 – Third Campaign Rheology Testing Results.

Shear Rate (S^{-1})	Dynamic Viscosity (cP)					
	COMP 3 (90% FRE & 10% OXI)			COMP 4 (33% FRE & 67% OXI)		
	38% w/w	46% w/w	50% w/w	38% w/w	46% w/w	50% w/w
50	12	50	78	17	47	122
75	11	40	57	16	37	89
100	11	31	45	16	29	70
150	10	22	35	15	25	51
200	10	20	33	14	21	43
250	10	19	30	14	21	35

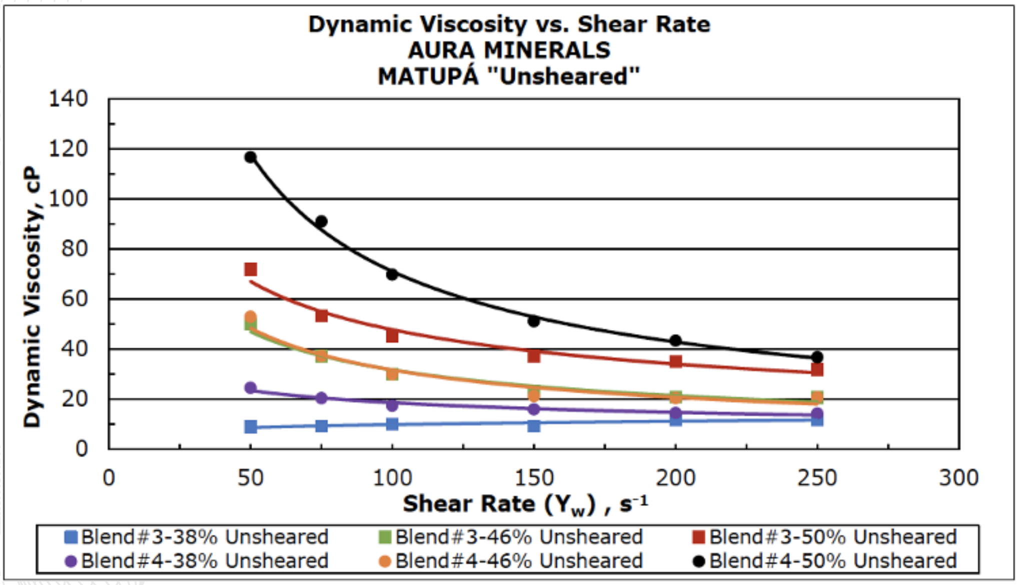


Figure 13-29 – Dynamic Viscosity as a Function of the Shear Rate - Third Campaign.

13.4.7.2 FILTERING TEST RESULTS

Filtration tests were carried out using both Top and Bottom feeding procedures for simulating horizontal belt and disc/drum industrial filters, respectively. Pressure filtering testing was included a specific campaign as an alternative method.

Table 13-68 and 13-69 summarize the test results and design parameters for vacuum and press filtration, respectively, as carried out on COMP 4 sample.

Table 13-68 - Summary of Data and Design Parameters for Vacuum Filtration.

Sample	COMP 4 (33% FRE & 67% OXI)	
Test Method	**Top**	**Bottom**
Filter Type	Horizontal	Discs
Solids in feed (wt.%)	46 – 47	
Pulp alkalinity (pH)	9.5 - 10.5	
Operation Vacuum ("Hg)	24 - 25	
Cake thickness (mm)	9 – 14	4 – 7
Cake weight (kg/m^2)	12	6
Cycle time (min.)	1.2	0.8
Cake moisture (wt.%)	20 - 22	29 - 31
Filtered Solids (ppm)	5000 - 6000	
Filtration rate (kg/m^2/h)	460	82
Min. Area Required (m^2)	350	1970

Table 13-69 - Summary of Data and Design Parameters for Pressure Filtration.

Process Parameters	COMP 4 (33% FRE & 67% OXI)	
Solids in feed (wt.%)	46 – 47	
Pulp alkalinity (pH)	9.5 – 10.5	
Chamber technology	Membrane	
Chamber thickness (mm)	50	32
Membrane pressure - Feed (bar)	7	
Operation pressure - Feed (bar)	15	
Test cycle time (min.)	12.5	11
Cake moisture (wt.) %)	16	15
Weight Filtered (kg/m^2)	43.5	21.5
Filtration rate (kg/m^2/hr)	33	29

Based on test results listed in Table 13-68 and 13-69, as well as those obtained in the second campaign, vacuum filtration in horizontal filters was selected for both pre-leaching and post-leaching stages. The resulting cake moisture was within the predicted 21%-23% interval, calculated on a wet basis.

Individual and averaged gold grades obtained from both fire assay and metallic screen methods for the Head Sample are listed in Table 13-45, while Table 13-46 includes the calculated standard and mean deviations.

13.4.7.3 PHYSICAL AND GEOTECHNICAL TEST RESULTS

Physical and geotechnical testing was conducted on a representative sample of leaching tailings previously thickened and filtered. A COMP 3 sample (90% FRE e 10% OXI) was used throughout the testing campaign. The objective this testing, carried out at Pattrol Investigacoes Geotecnicas Ltda. at Belo Horizonte, Brazil was to obtain data for the dry stacking tailing disposal alternative.

The methods and standards used in the respective tests are listed in Table 13-70.

Table 13-70 – Standards Used in Physical and Geotechnical Tests.

Test	Standard / Method
Size Distribution by Screening and Sedimetation	NBR 7181/2016
Density of Solids	NBR 6458/2016
Liquidity limit	NBR 6459/2016
Plastic limit	NBR 7180/2016
Proctor Compaction	NBR 7182/2016
Variable Head Permeability	NBR 14545/2021
Oedometric Densification	ASTM D2435-11/2020
Consolidated Undrained Triaxial	ASTM D4767-11/2020
Standard Test Method for Unconsolidated-Undrained Triaxial Compression Test On Cohesive Soils	ASTM D2850-15/2016
Standard Test Method for Consolidated Undrained Direct Simple Shear Testing of Fine Grain Soils	ASTM D6528/2017

Table 13-71 – Physical and Geotechnical Tests - Size Distribution Parameters.

Size Distribution (%)												Material
#1"	#3/8"	#4	#10	#40	#200	%G	%CS	%MS	%FS	%S	%C	
100	100	100	100	100	59.9	0	0	6.6	47.8	42.1	3.6	Sandy Loam

%G = Percentage of Gravel
%CS = Percentage of Coarse Sand
%MS = Percentage of Medium Sand
%FS = Percentage of Fine Sand
%S = Percentage of Silt
%C = Percentage of Clay

Table 13-72 – Physical and Geotechnical Tests – Density of Solids.

Sample Identifiction	Density of Solids (g/cm^3)
COMP 3 (90% FRE + 10% OXI)	2.861

Table 13-73 – Physical and Geotechnical Tests – Atterberg Limits.

Sample Identifiction	LL (%)	LP (%)	PI (%)
COMP 3 (90% FRE + 10% OXI)	NL	NP	-

LL = Liquidity Limit
LP = Plastic Limit
PI = Plasticity index

Table 13-74 – Physical and Geotechnical Tests – Proctor Compaction.

Sample Identifiction	Maximum Density (g/cm^3)	Optimal Moisture (%)
COMP 3 (90% FRE + 10% OXI)	1.525	23.3

Table 13-75 – Physical and Geotechnical Tests – Variable Head Permeability and CIU Triaxial Test.

Sample Identifiction	Moulding Condition	Permeability Test K (cm/s) average	Triaxial CIU			
			Total Stress		Effective Stress	
			C (kPa)	Φ (°)	C' (kPa)	Φ' (°)
COMP 3 (90% FRE + 10% OXI)	80% PN	1.22×10^{-3}	2.80	19.1	1.35	29.1
	85% PN	5.49×10^{-4}	2.30	20.7	1.80	29.1
	90% PN	2.53×10^{-4}	2.29	23.3	1.07	31.8

Table 13-76 – Physical and Geotechnical Tests – UU Triaxial Test.

Sample Identifiction	Total Stress	
	C (kPa)	Φ (°)
COMP 3 (90% FRE + 10% OXI)	-	-

One of the main results obtained from the physical and geotechnical tests was the optimal stacking moisture content (23.3%), as shown in Table 13-74. It is important to notethat this result refers to geotechnical moisture content which is calculated on a dry basis, in this case equivalent to 18.9% moisture content when calculated on a wet basis, the latter is commonly used throughout mineral processing and hydrometallurgy.

The resulting cake moisture is within the predicted 21%-23% interval for the selected solid-liquid separation alternative. This is in accordance with the optimal stacking moisture (23.3%) as above described.

13.4.7.4 TAILING SAMPLE FLOW PARAMETERS

Flow property tests were conducted on a representative sample of leaching tailings previously thickened and filtered. A 123 kg COMP 3 sample (90% FRE e 10% OXI) ground to a P_{80} of 0.125 mm at SGS-Geosol laboratories was used throughout the testing campaign, the latter carried out at Jenike & Johanson laboratories – Vinhedo, Brazil.

The tests were carried out at 18%, 22% and 24% moistures contents, all calculated on a wet basis. Photographs of the sample with moisture content are shown in Figure 13-30.

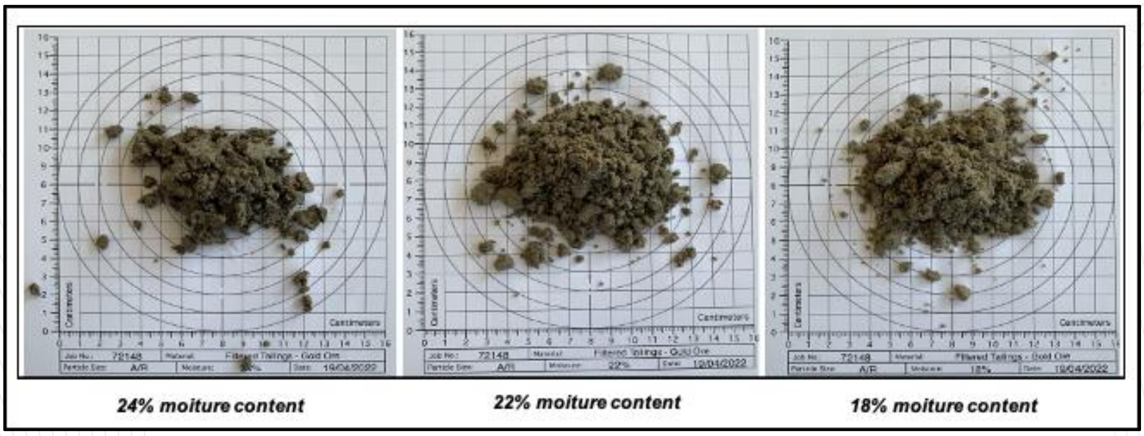


Figure 13-30 – Photos from Sample Prepared at Three Moisture Contents.

A list of the tests carried out and respective application is listed in Table 13-77.

Table 13-77 – Flow Test Application.

Test	Aplication
Cohesive strength tests (arqhing and ratholing dimensions)	Used to calculatiog the bin critical dimensions to prevent bridging and ratholing
Compressibility	Reationship between bulk density and consolidation pressure
Wall frisction angles	Used for example for calculatin mass flow hopper angles, evaluate the material flow througn transfer chutes
Particle Size Analysis (Laser diffraction)	Physical and Flow Properties
Minimum chute angles required to maitain flow after impact	Physical and Flow Properties
Wall fristion tests	Physical and Flow Properties
Angle of repose and drawdown angle test	Physical and Flow Properties

The size distribution properties obtained by laser diffraction are listed in Table 13-78. The 90% passing value (P_{90})for the sample used in all tests was 0.252 mm.

Table 13-78 – Size Distribution Properties.

Material	Particle Size Distribution					
	At 0.5 bar dispersion pressure			At 3.0 bar dispersion pressure		
	D10 (µm)	D50 (µm)	D90 (µm)	D10 (µm)	D50 (µm)	D90 (µm)
COMP 3 (90% FRE & 10% OXI)	16	111	293	15	100	252

The compressive properties and particle density are listed in Table 13-72. The compressive properties obtained from this test was a density of 2.769 kg/m^3, Table 13-79.

Table 13-79 – Compressive Properties and Particle Density.

Material	Moisture content (%)	Measured bulk density range (kg/m3)	Loose / Tapped density (kg/m3)	Particle density (kg/m3)
COMP 3 (90% FRE & 10% OXI)	24	940 - 1718	Not tested	Not tested
	22	862 - 1610	793 / 1477	Not tested
	18	749 - 1490	Not tested	Not tested
	0.22	Not tested	Not tested	2.769

Table 13-80 shows the results obtained for both repose and drawdown angles testing. The former represents the stockpile filling angle while the latter is related to gravity reclaiming of a stockpile. Actual surcharge angle values in the field may be lower due to various methods of pile formation.

Table 13-80 – Results of Repose and Drawdown Angle Tests.

Material	Moisture content (%)	Angle* of repose (average)	Angle* of repose (range)	Drawdown* angle (average)	Drawdown angle* (average)
COMP 3 (90% FRE & 10% OXI)	22	38	33 - 42	68	61 - 74

** Horizontal angles*

An important remark on the Jenike & Johanson report is that the material is cohesive and has a tendency to form a stable "rathole" if stored in a funnel flow type silo.

14. MINERAL RESOURCE ESTIMATES

The Matupá Gold Project Mineral Resource Estimate is limited to the X1 Deposit. The Mineral Resource Estimate updates were performed using all information, as of the effective date of this report, in the validated database. 3-D updated models were constructed in the GEOVIA-Surpac software platform (version 6.3) and Mineral Resources estimated using same platform by Farshid Ghazanfari, P.Geo. and QP for Aura Minerals. In opinion of the QP for this section, the Mineral Resource Estimate meets industry standards and the general guidelines for NI 43-101 compliant resources for Measured, Indicated, and Inferred confidence levels as discussed herein.

Since 2012, there has been no infill drilling carried out by Aura Minerals. Aura Minerals concluded some geotechnical drilling in 2020 for preliminary mine design but the core was not assayed and was only used for geotechnical purposes. Aura Minerals also drilled one exploration hole in the west extension of the X1 Deposit which did not intersect any significant mineralization.

14.1 TOPOGRAPHIC AND DTM SURFACES

The weathering surfaces were interpreted based on lithological descriptions of the drill holes and referenced to the 2012 topographical survey data. The base of the saprolite and base of the altered intrusive were reconstructed as weathering surfaces (DTM surfaces) and used to separate oxide mineralization from sulfide. Blocks were assigned a material type of oxide, or sulfide based on their position relative to these surfaces.

The new ground survey topography was performed in 2021 on X1 at a property wide scale covering all areas surrounding the X1 Deposit that would be subject to any future mining operation. An updated topography surface file was created by a contracted surveying company (Geoline Serviços Minerais Eireli). A drone base topography survey also was performed in 2021 for a larger area covering areas that potential infrastructure and process plants may be located. The detail of both topographical survey methods is described in section 12 this report.

Reported Mineral Resources in this section is based on the new DTM surface for topography.

14.2 X1 DATABASE

The X1 database incorporates different drilling campaigns conducted by companies such as Vale, Santa Elina, Rio Novo and Aura in the Project between 2003 to 2021. The X1 database which was received as part of acquisition of Rio Novo by Aura Minerals is in a relational Microsoft Access database containing all drilling information. This database was validated by Aura Minerals for all the data corresponding to the drill log files, downhole survey files and assay certificates.

The Table 14-1 shows the drilling and sampling quantitative studies summary that were performed during all the drilling campaigns in the Project.

Table 14-1 - Drill Hole Data Status and Statistics

Project	Company	Year	Drill Type	Number of	Meters Drilled	Number of Samples	Meters Sampled
X1	VALE	2003	Diamond	18	3190.05	3190	3189.8
X1	MSE	2003	Diamond	1	90.8	48	90.8
X1	MSE	2007	Diamond	2	493.42	495	493.42
X1	MSE	2008	Diamond	60	13590.52	7663	13590.52
X1	RNG	2010	Diamond	6	1081.3	1011	1081.3
X1	RNG	2011	Diamond	54	10388.36	9307	10355.02
X1	AMI	2021	Diamond	2	686.39	697	686.39

1.

Aura performed data validation and verification on the received database and in QP's opinion, the database and data within are in good standing in terms of data accuracy and can be relied upon for Mineral Resource estimation purposes.

14.3 GEOLOGICAL AND DOMAIN MODELING

Two alteration models were developed based on lithological and alteration logging information of all drill holes that intersected mineralization in X1 property. These alteration models were verified by the QP against the validated X1 database then updated and merged. In the opinion of the QP, these 3-D models effectively encompassed disseminated mineralization in the X1 Deposit and are better than any grade shell to constrain mineralized zone. These two models, with some minor adjustments, were used to estimate the Mineral Resource for the X1 Deposit for this study (Figure 14-1). The alteration models were described as phyllic+ early potassic and strong pervasive phyllic (Figure 14-2) coded as such in the block model. Grade interpolation was performed separately with the hard boundary between two domains.

Based on DTM surfaces, logging, and weathering profile in X1, three 3-D models were created for saprolite, weathered and fresh rocks after grade interpolation. These models coded appropriately in the X1 block model for the Oxide attribute to separate tonnes and grade for oxide and sulfide materials within the model. Figure 14-2 shows DTM models and weathering profiles in the X1 Deposit.

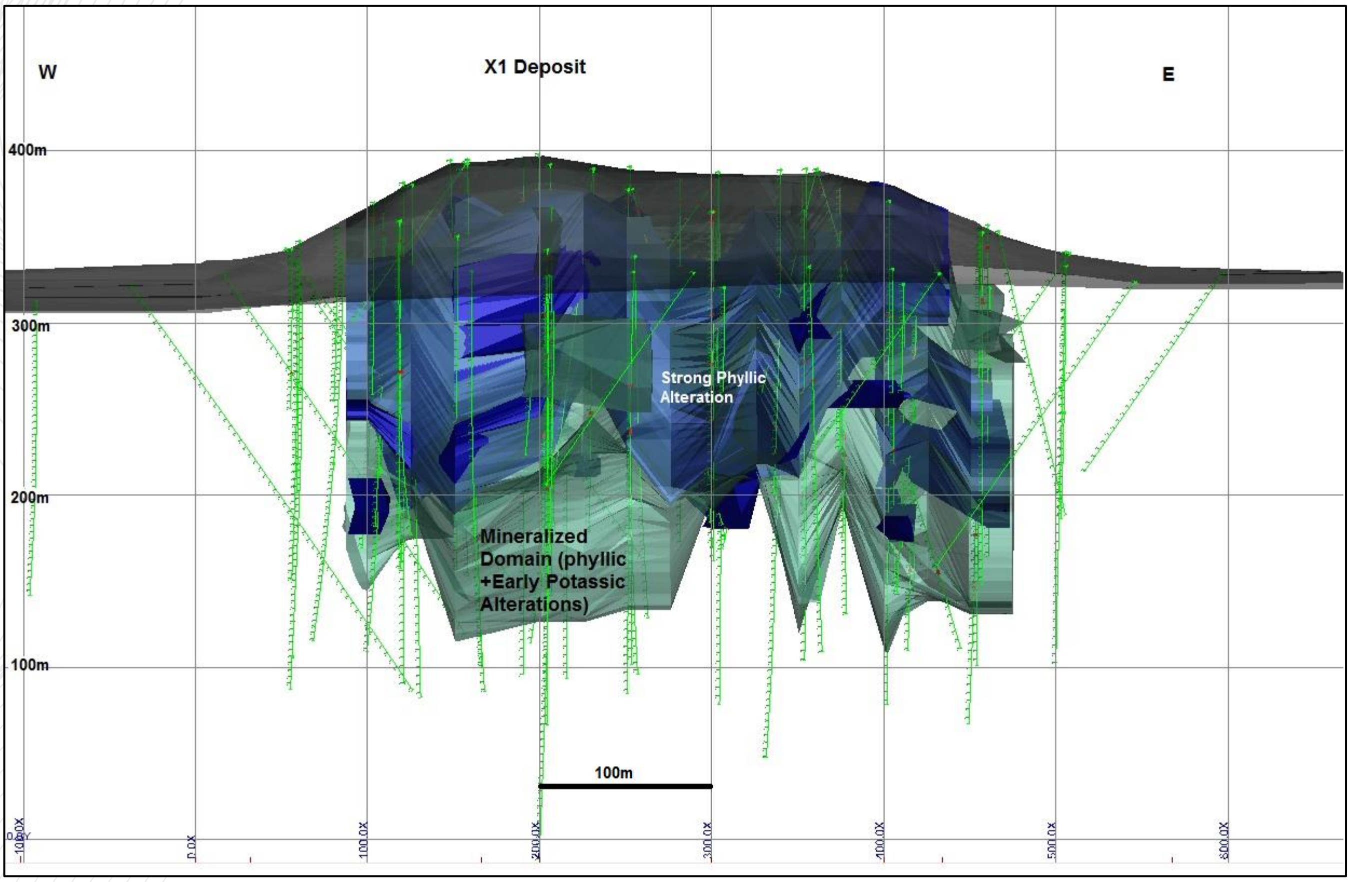


Figure 14-1 - X1 Deposit 3-D Alteration Models and trace of Drill holes.

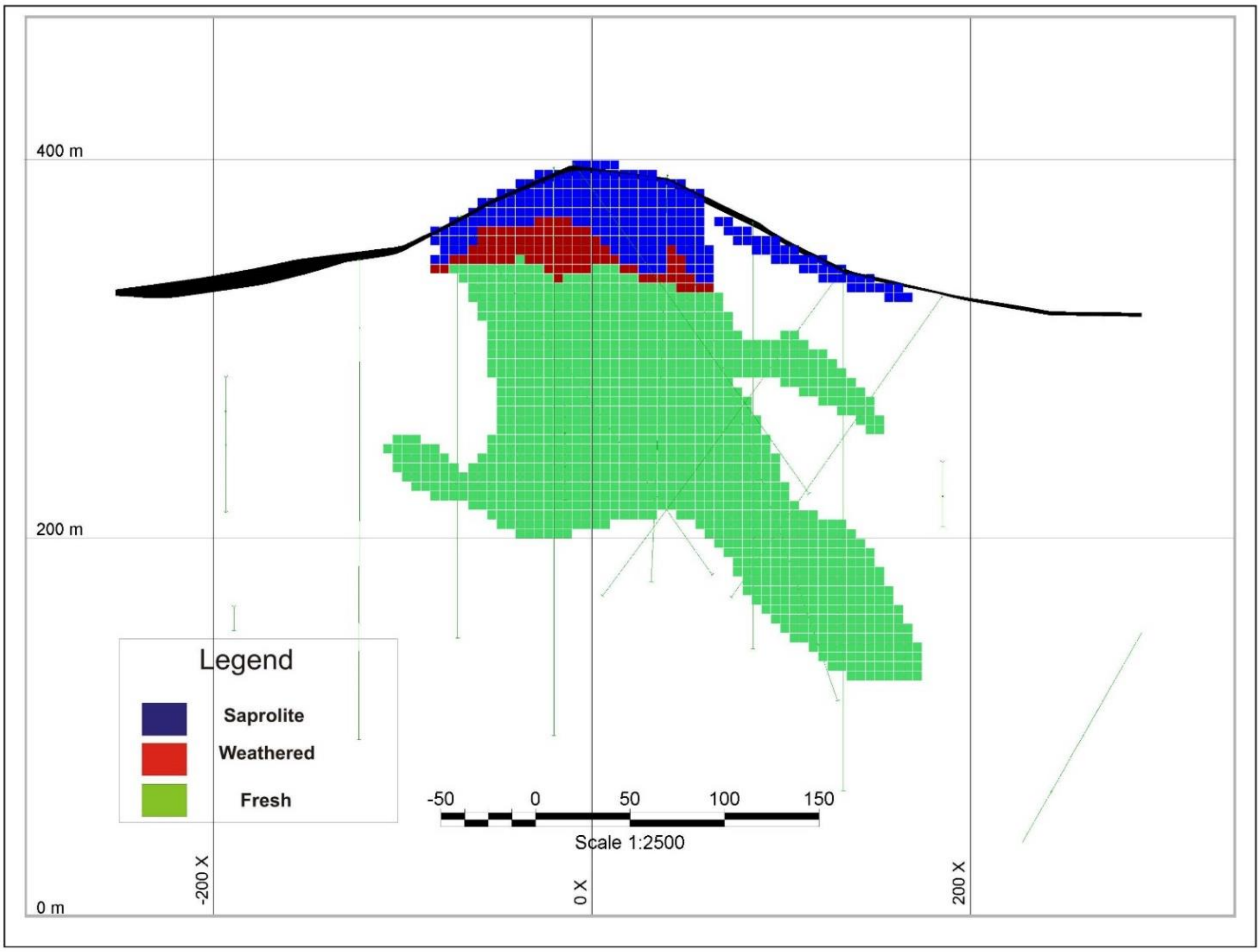


Figure 14-2 - X1 Deposit 3-D Weathering Profiles and Trace of Drill Holes in a Typical Section.

14.4 DENSITY MODEL

Details of density measurements, sampling and testing procedures are discussed in section 11.3 of this report. Here the density values which were selected and assigned to the 3-D model to convert volume to tonnes will be discussed.

The X1 density database is representing different lithologies, mineralization types, and degree of alteration and weathering. The database has 1,261 density measurements for fresh rock on drill core and 68 samples for weathered, soil and saprolite samples. The water immersion method was used by Rio Novo geologists for weathered, saprolite and soil samples, and the porous samples were sealed in plastic bags. The data were analyzed statistically by lithology and outliers removed. The density measurements typically were performed on 10 cm samples.

In the saprolite zone, essentially all rock between surface and above the weathered zone, an average density of 1.51 tonnes per cubic meter (from dry density measurements) was assigned as default density. This value was derived based on the average of 64 density samples taken from the saprolite and soils. For weathered rocks, a total of 28 samples were taken for density measurements and only a few of them are based on dry density measurements. Therefore, the actual wet density values in the database used to estimate density values in the X1 density model for these materials.

Table 14-2 summarizes the average density values which was recorded and stored in in the X1 database.

Table 14-2 - Average of Density for Different Type of Rocks (X1).

Lithology	Density t/m³
Saprolite & Soil	1.51
Weathered	2.44
Fresh Rock	2.71

A density model was interpolated using Ordinary Kriging (OK). The density point data was used, and no compositing done as most of density values are based on 10 cm sample intervals. The values were used as points in the 3-D spacing to interpolate density values inside the ore body. Similar constrains for the alteration model (phyllic and strong phyllic domains) were also assigned for interpolation. The interpolation of density values for each zone used search ellipsoid orientations based on the directions of continuity determined from gold grades. A minimum of 4 and maximum of 24 data were required to make a block estimate, with a maximum of 3 data from any one drill hole. This allows a single drill hole to make a block model density estimate.

The weighted average density of 96,341 blocks for mineralized domains is 2.65 t/m^3 and compares well with average of the 1,067 density data, which is 2.64 t/m^3.

14.5 EXPLORATORY DATA ANALYSIS (X1)

The X1 database contains sufficient data to support a well-informed Mineral Resource Estimate. The X1 database contains 21,663 samples with Au and Ag values equal to or greater than zero. Sample lengths are variable, from 0.34 m to 6.45 m with an average length of 1.33 m.

In order to have a point data set to perform statistical analysis, drill holes intersected against mineralized wireframes as were described in section 14.4 were selected and all assays extracted with their corresponding lengths. The total samples which reside inside of the mineralized domains and used for grade interpolation, are 7,567 samples for Au and 7,420 samples for Ag, with average length of 1.37 m.

Figures 14-3 and 14-4 show a log histogram of all gold and silver assays inside the X1 mineralized wireframes respectively.

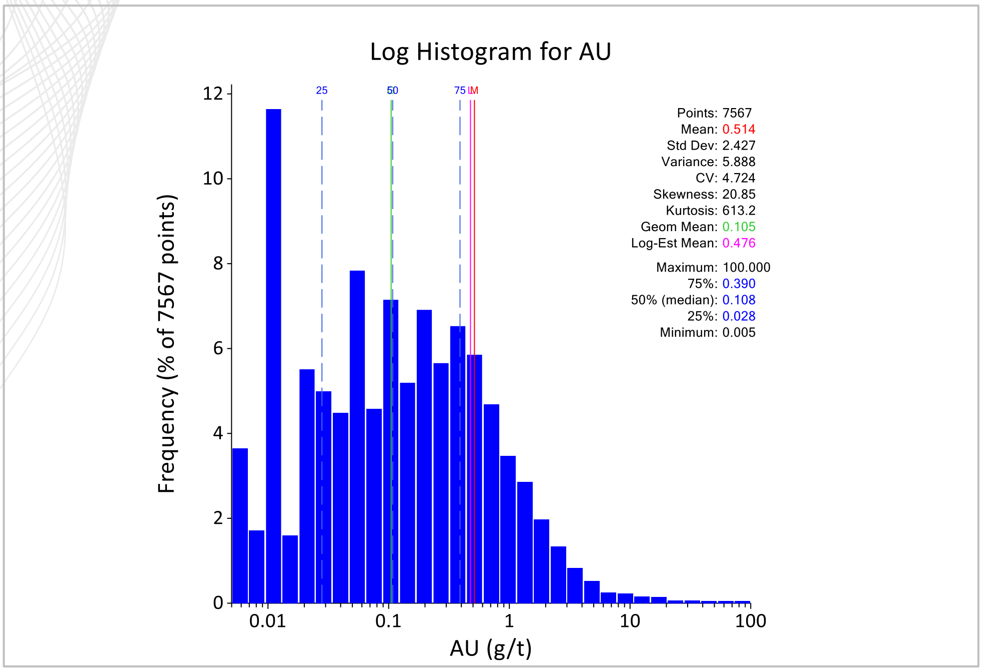


Figure 14-3 - Log Histogram for Gold Assay Values (X1).

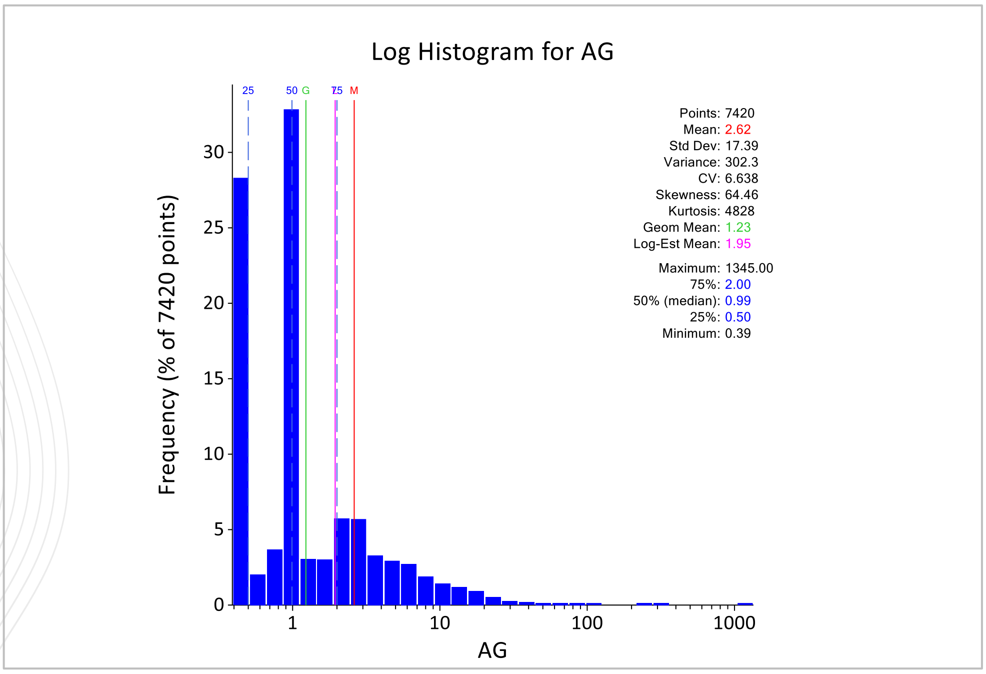


Figure 14-4 - Log Histogram for Silver Assay Values (X1).

Figures 14-5 and 14-6 show a log probability plot for Au and Ag for the same dataset and suggests that the use of a capping grade, on raw data, as high as 15.0 g/t for Au and 44 g/t for Ag could be justified. This is representative of 99.2% of percentile for Au and 99.5% percentile for Ag. This is interpreted from the end of the straight portion of the curve at high grades for the respective domains. For the purpose of this feasibility study 15.0 g/t was adopted for capping gold values and 44 g/t was adopted for silver values.

Samples within the mineralized envelopes were processed into 2.0 m composites and capped, after compositing, at 15.0 g/t Au and 44 g/t Ag. Figures 14-7, and 14-8 show the histogram, univariate statistics and log probability plot for the X1 Au composites within the strong phyllic alteration-high grade envelope before capping. Figures 14-9 and 14-10 show the histogram, univariate statistics and log probability plot for the X1 Au composites within the Low grade alteration phyllic envelope before capping.

Figures 14-11 and 14-12 show the histogram, univariate statistics and log probability plot for the X1 Ag composites within the strong phyllic alteration-high grade envelope before capping. Figures 14-13 and 14-14 show the histogram, univariate statistics and log probability plot for the X1 Ag composites within the Low grade phyllic alteration envelope before capping.

Figures 14-15 and 14-16 show the histogram, univariate statistics and log probability plot for the X1 Sulfur (S) composites within the strong phyllic alteration-high grade envelope. Figures 14-17 and 14-18 show the histogram, univariate statistics and log probability plot for the X1 Sulfur (S) composites within the Low grade phyllic alteration envelope. Sulfur assays are only available for some of the more recent holes in the X1 database.

The composites file contains samples with lengths down to 0.20 m which is 10% of composite length.

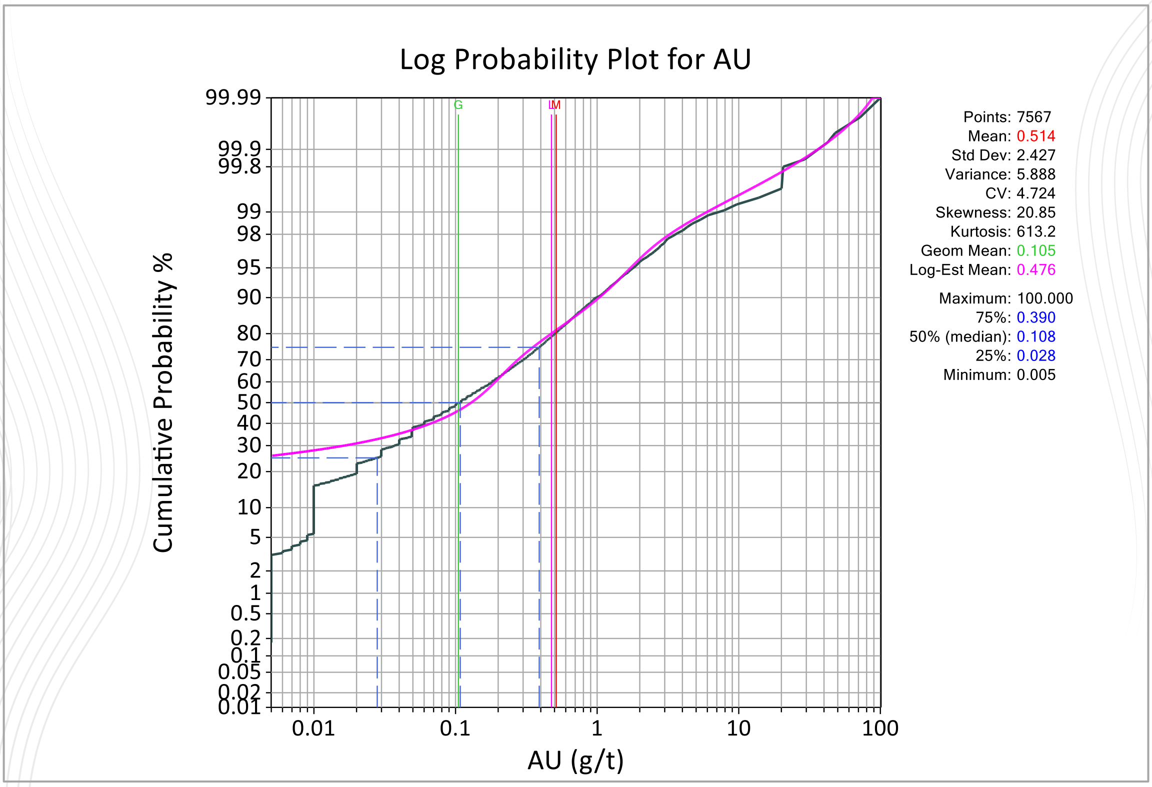


Figure 14-5 - Log Probability Plot for Gold Assay Values (X1).

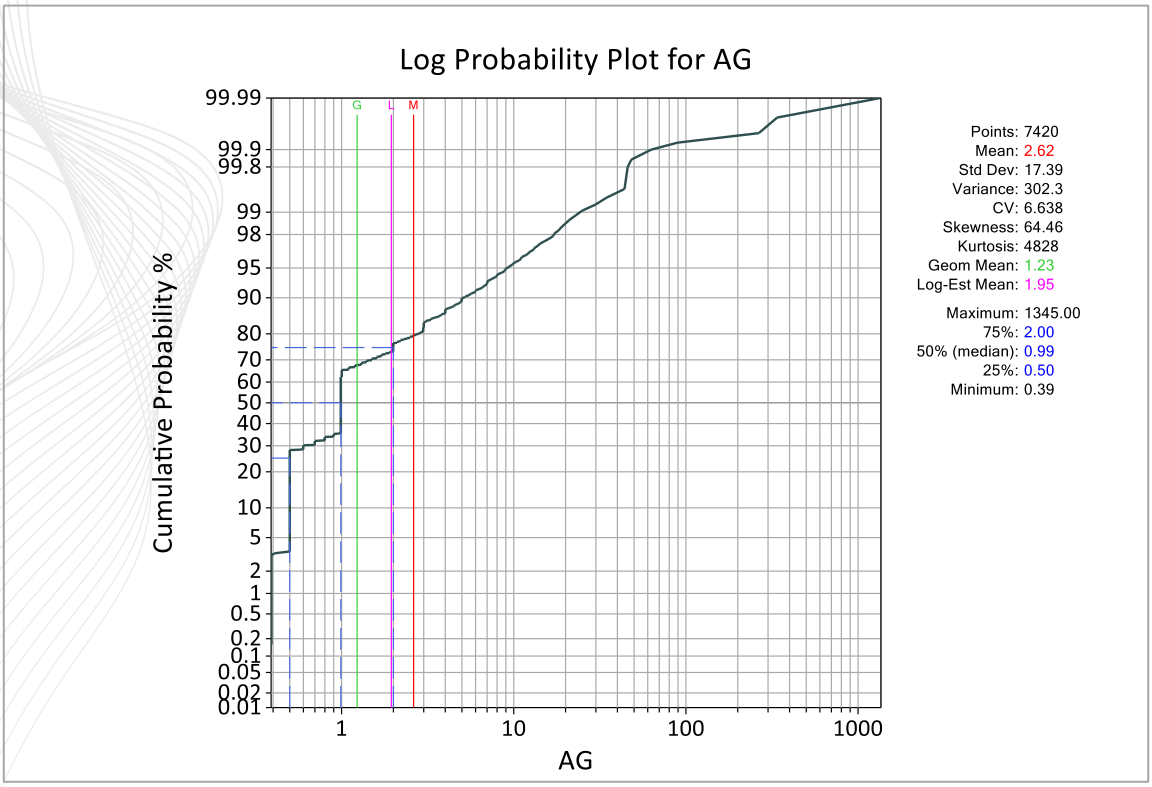


Figure 14-6 - Log probability Plot for Silver Assay Values (X1).

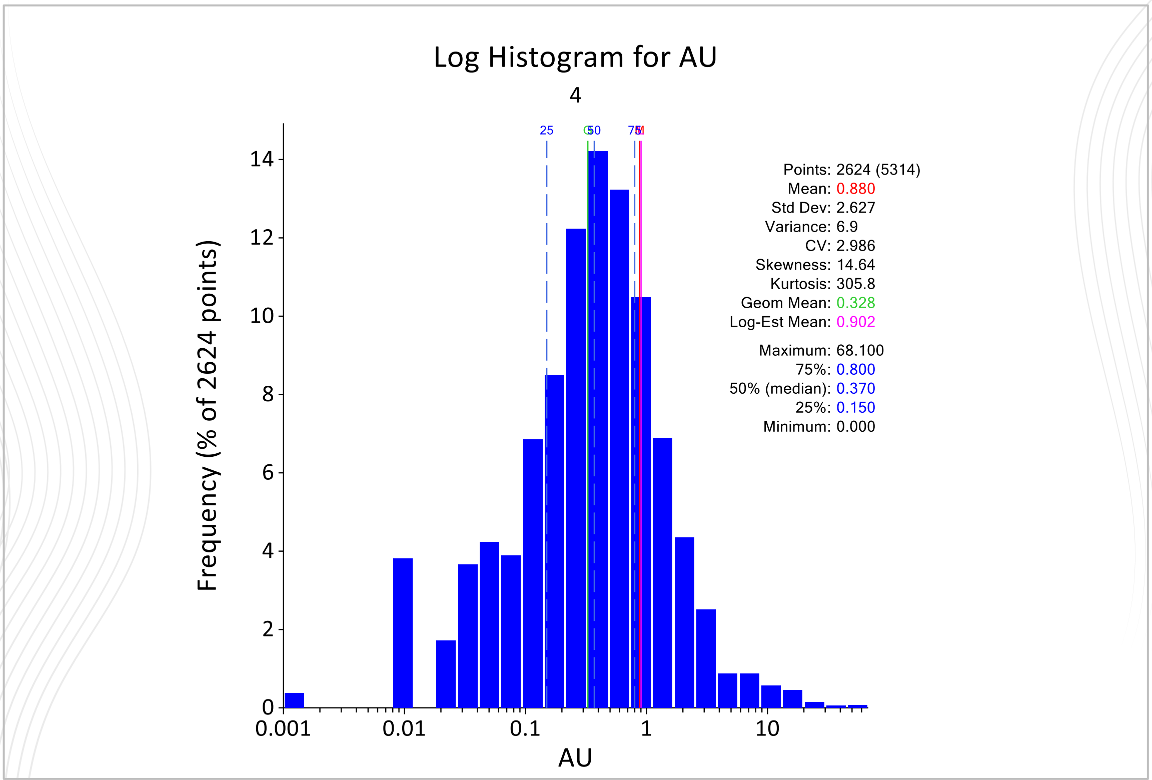


Figure 14-7 - Log Histogram for Composited Gold Values for Domain 4 (Strong Phyllic Alteration domain).

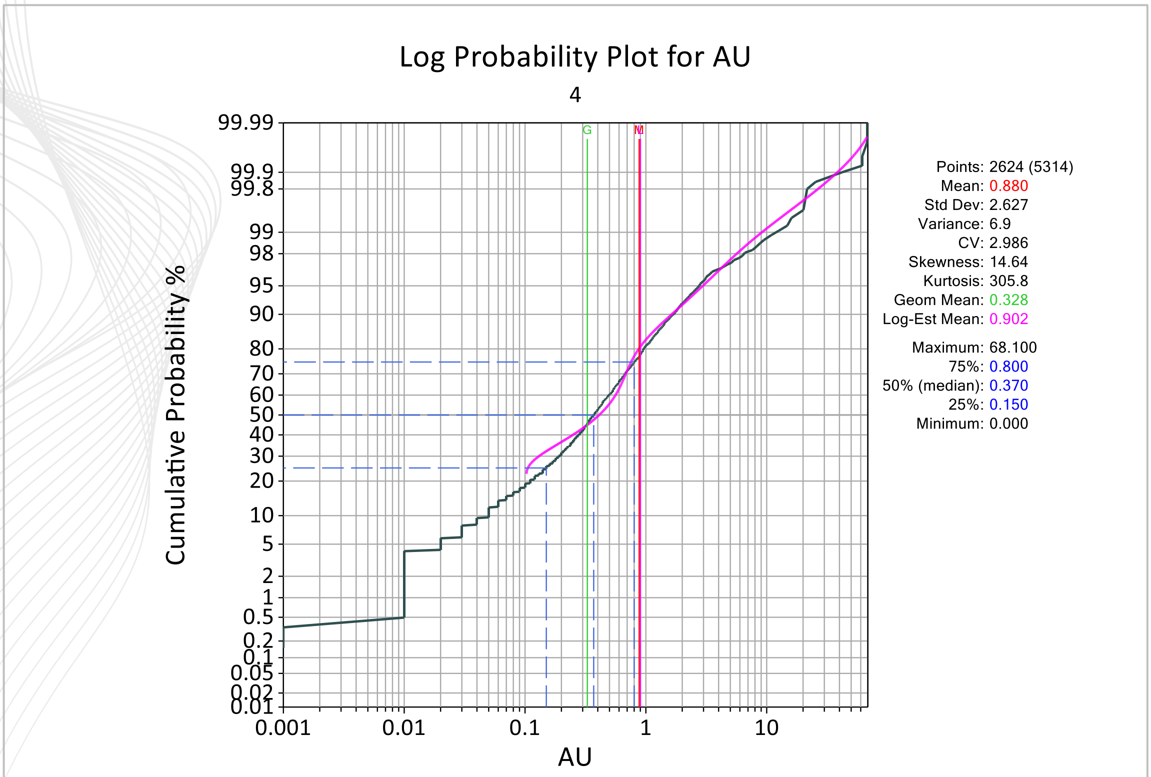


Figure 14-8 - Log Probability Plot of Composited Gold Values for Domain 4 (Strong Phyllic Alteration domain).

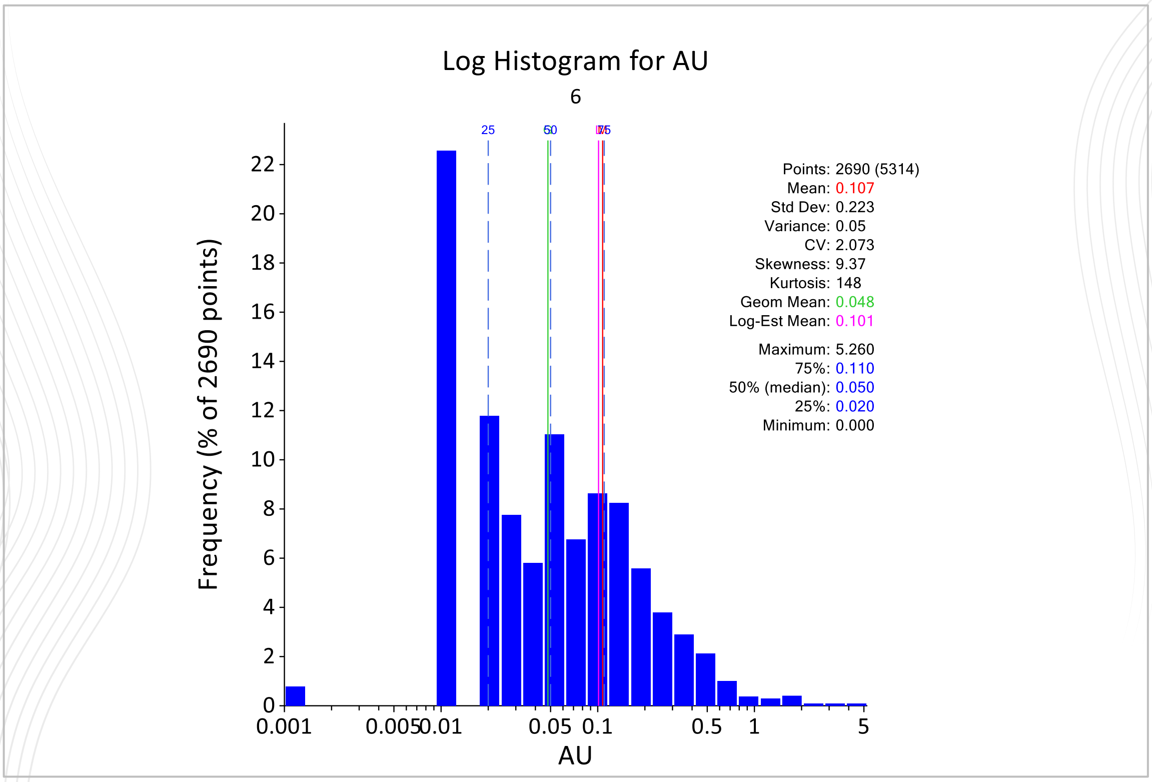


Figure 14-9 - Log Histogram for Composited Gold Values for Domain 6 (Phyllic Alteration domain).

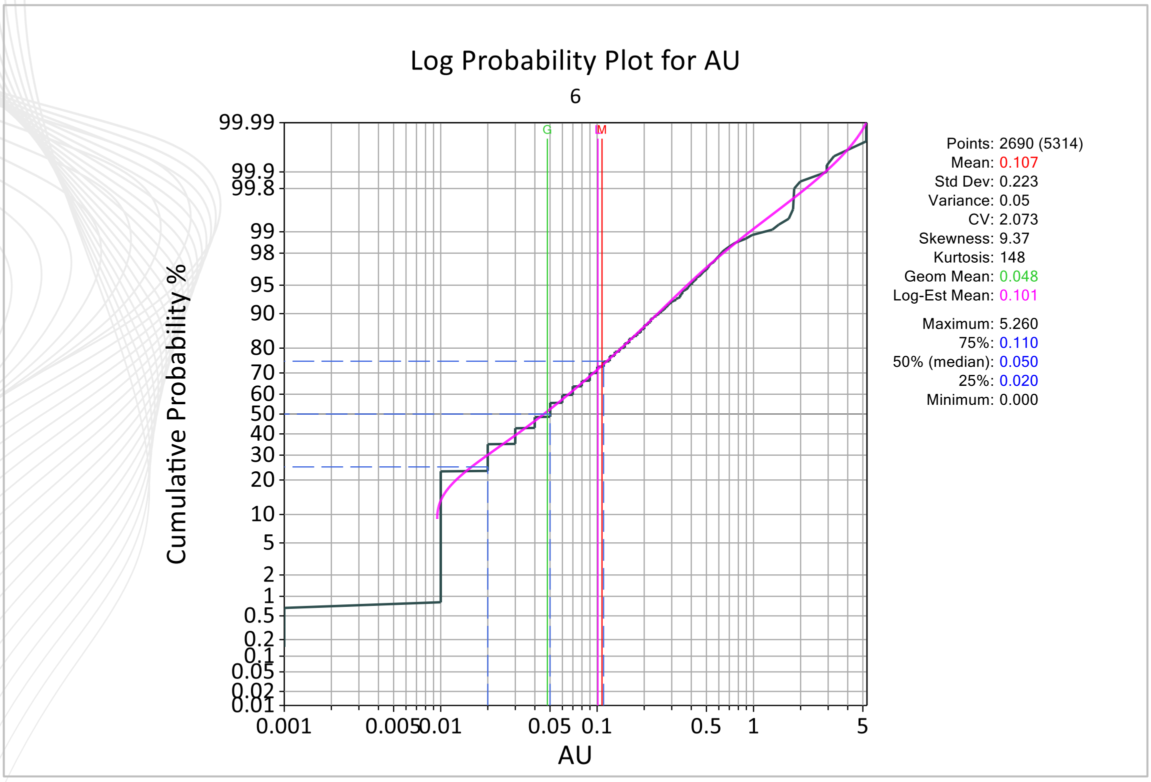


Figure 14-10 - Log Probability Plot of Composited Gold Values for Domain 6 (Phyllic Alteration domain).

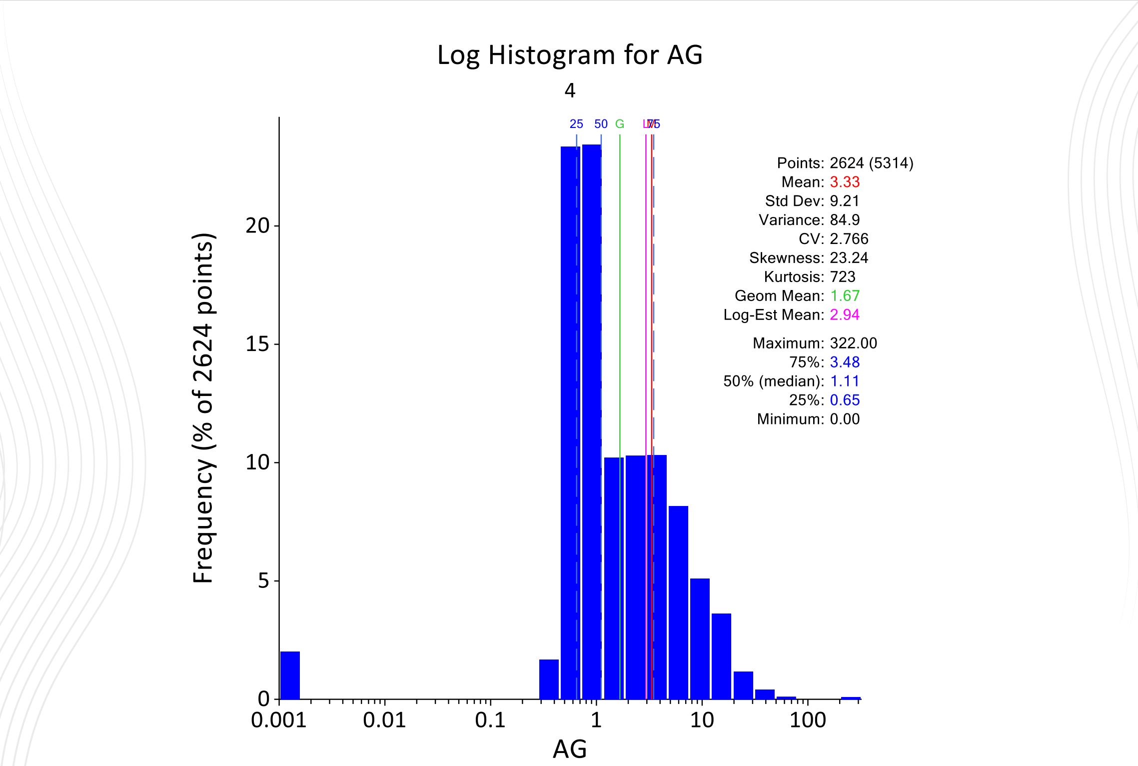


Figure 14-11 - Log Histogram for Composited Silver Values for Domain 4 (Strong Phyllic Alteration domain).

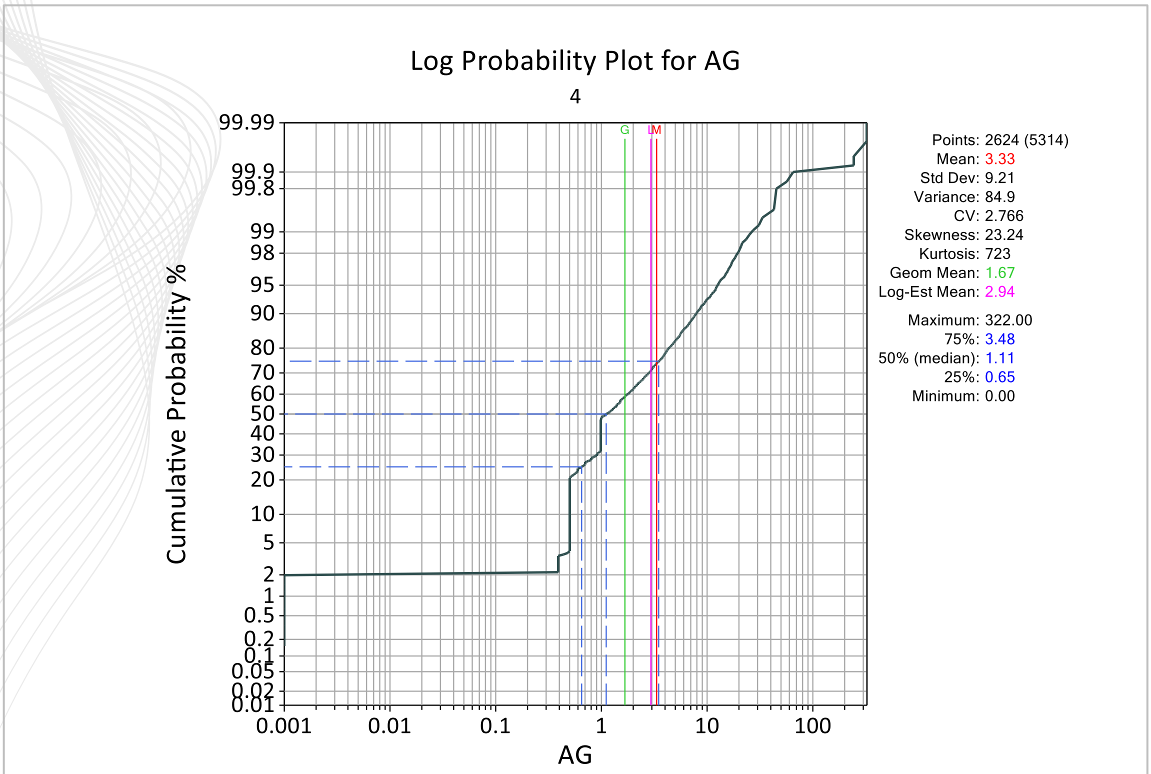


Figure 14-12 - Log Probability Plot of Composited Silver Values for Domain 4 (Strong Phyllic Alteration domain).

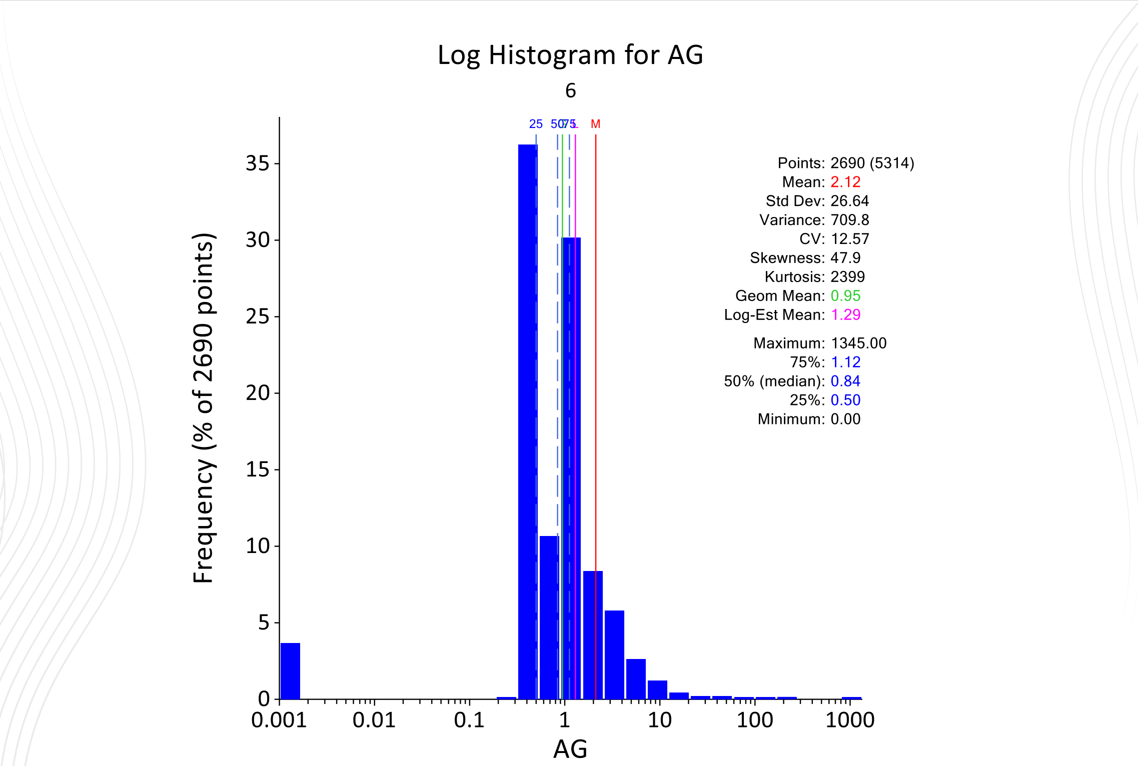


Figure 14-13 - Log Histogram for Composited Silver Values for Domain 6 (Phyllic Alteration domain).

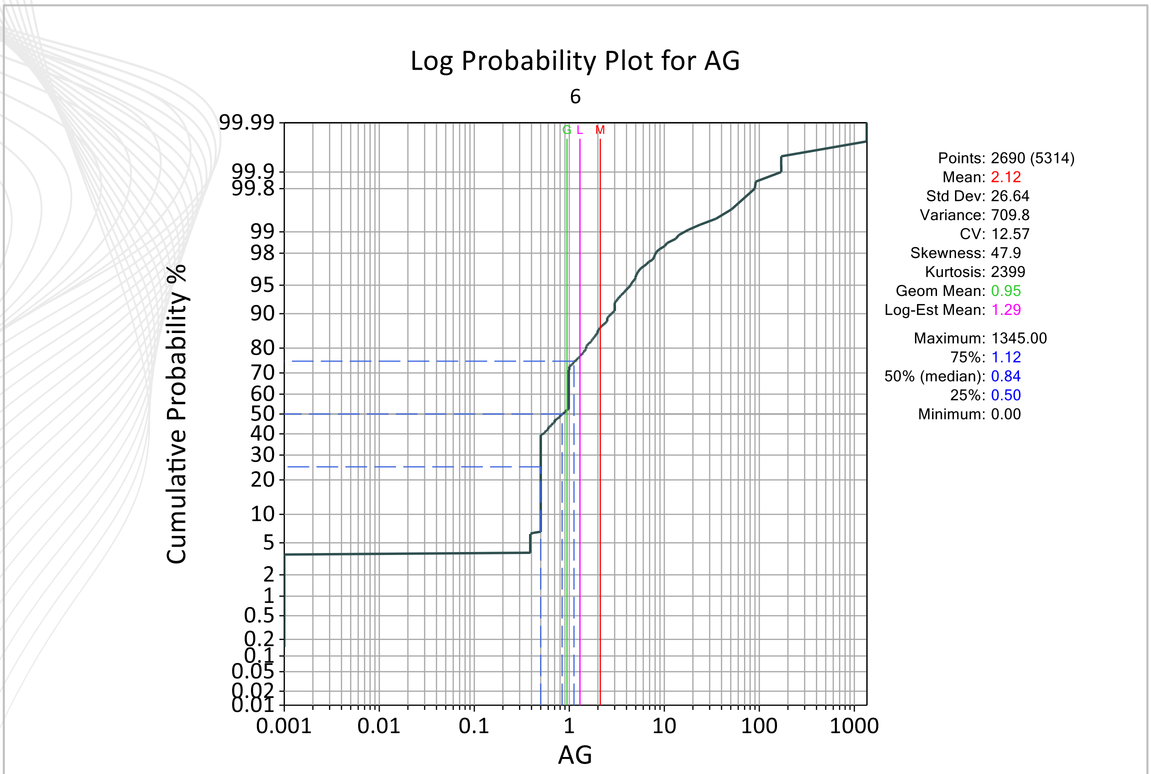


Figure 14-14 - Log Probability Plot of Composited Silver Values for Domain 6 (Phyllic Alteration domain).

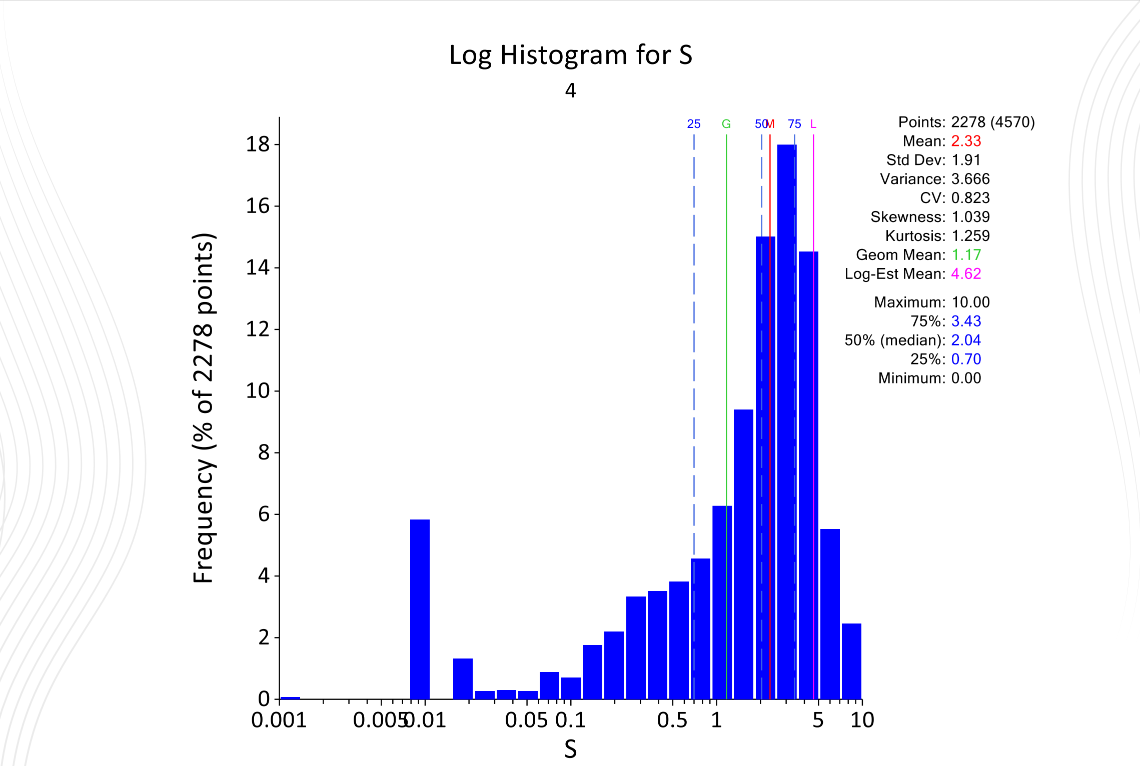


Figure 14-15 - Log Histogram for Composited Sulfur Values for Domain 4 (Strong Phyllic Alteration domain).

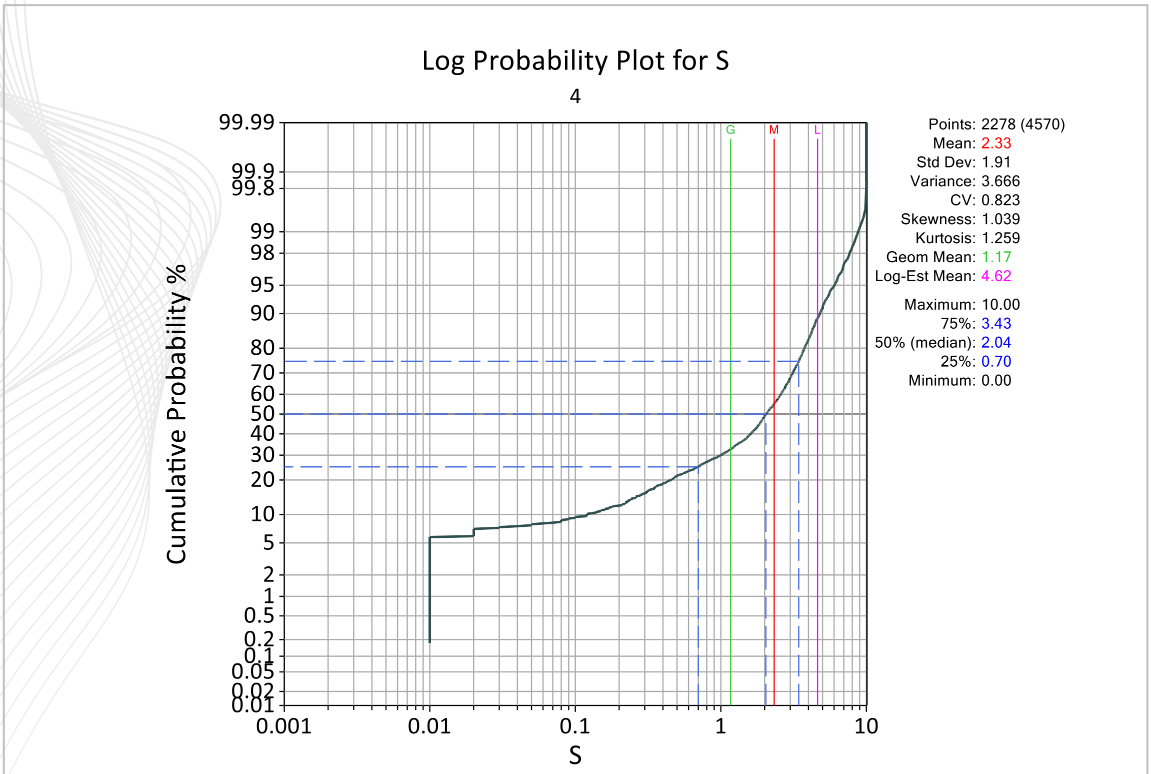


Figure 14-16 - Log Probability Plot of Composited Sulfur Values for Domain 4 (Strong Phyllic Alteration domain).

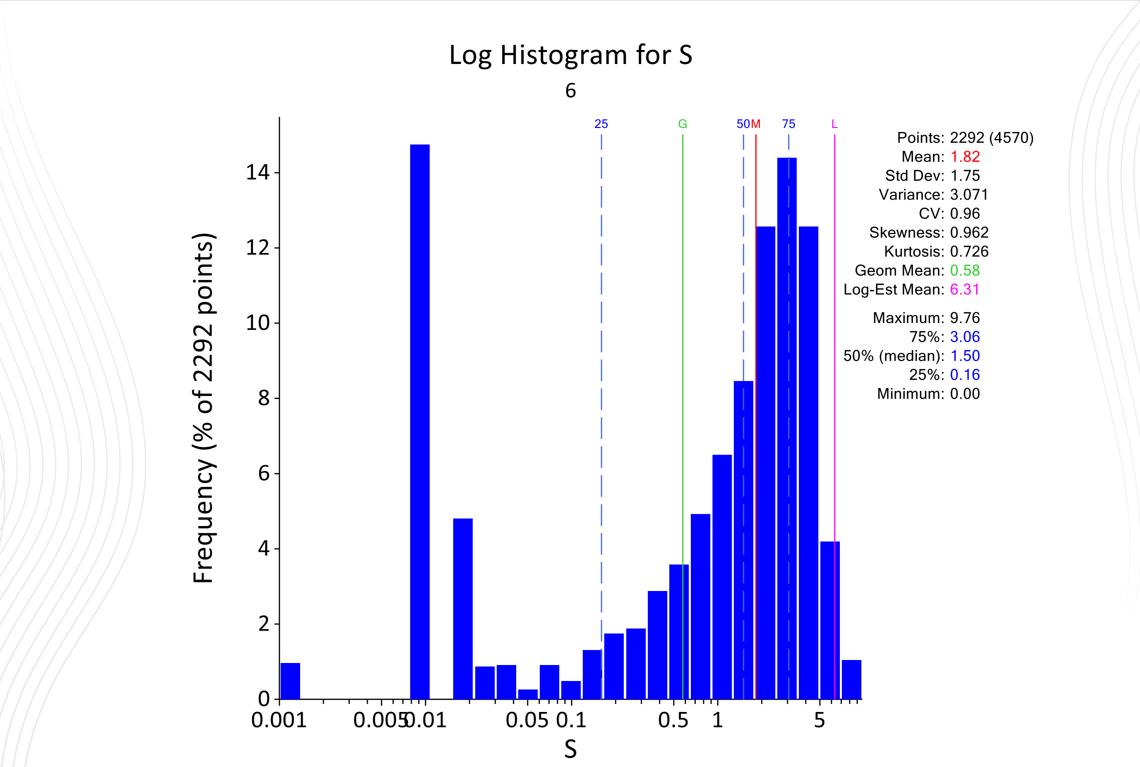


Figure 14-17 - Log Histogram for Composited Sulfur Values for Domain 6 (Phyllic Alteration domain).

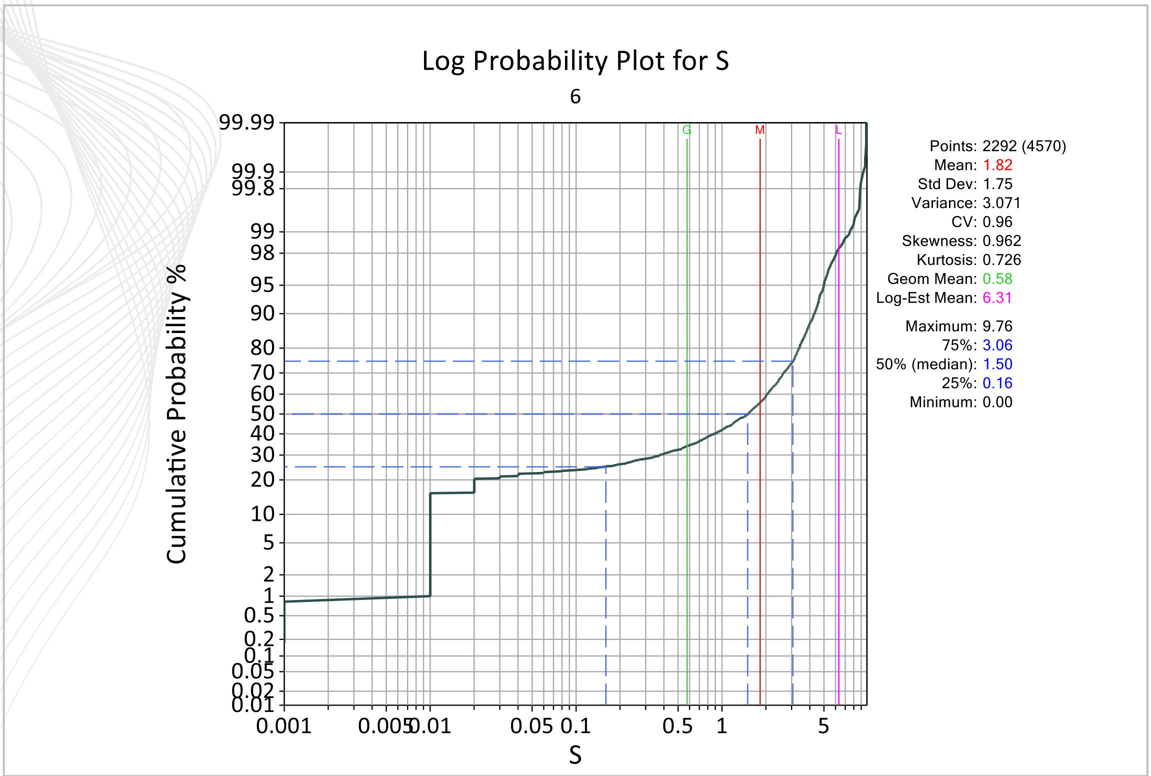


Figure 14-18 - Log Probability Plot of Composited Sulfur Values for Domain 6 (Phyllic Alteration domain).

A statistical summary of the selected samples in different lithotypes is shown in Table 14-3 for Au. Statistics for the generated 2.0 m composites in different lithotypes for Au is also shown in Table 14-4.

A statistical summary of the selected samples in different lithotypes is shown in Table 14-5 for Ag. Statistics for the generated 2.0 m composites in different lithotypes for Ag is also shown in Table 14-6.

Table 14-3 - Summary Statistics – Selected Gold Assay Samples.

LITH	NUMBER	MINIMUM	MAXIMUM	MEAN	VARIANCE	STANDDEV	LOGESTMN	CV
Saprolite	593	0.00	6.11	0.23	0.27	0.52	0.23	2.24
Weathered	765	0.00	34.60	0.35	2.15	1.47	0.34	4.13
Fresh	6209	0.00	100	0.56	6.87	2.62	0.53	4.68
All	**7567**	**0.00**	**100**	**0.51**	**5.89**	**2.43**	**0.48**	**4.72**

Note: CV = coefficient of variation

Table 14-4 - Summary Statistics – 2.0 m Composites (Au).

LITH	NUMBER	MINIMUM	MAXIMUM	MEAN	VARIANCE	STANDDEV	LOGESTMN	CV
Saprolite	453	0.00	5.14	0.23	0.22	0.47	0.24	2.04
Weathered	563	0.00	14.99	0.34	0.76	0.87	0.36	2.55
Fresh	4298	0.00	68.10	0.54	4.29	2.07	0.52	3.87
All	**5314**	**0.00**	**68.10**	**0.49**	**3.58**	**1.89**	**0.48**	**3.87**

Note: CV = coefficient of variation

Table 14-5 - Summary Statistics – Selected Silver Assay Samples.

LITH	NUMBER	MINIMUM	MAXIMUM	MEAN	VARIANCE	STANDDEV	LOGESTMN	CV
Saprolite	589	0.39	1345	5.75	3579	59.83	1.47	10.41
Weathered	753	0.39	109	1.71	39.60	6.29	1.14	3.67
Fresh	6078	0.39	63.0	2.43	16.77	4.10	2.10	1.69
All	**7420**	**0.39**	**1345**	**2.62**	**302.3**	**17.39**	**1.95**	**6.64**

Note: CV = coefficient of variation

Table 14-6 - Summary Statistics – 2.0 m Composites (Ag).

LITH	NUMBER	MINIMUM	MAXIMUM	MEAN	VARIANCE	STANDDEV	LOGESTMN	CV
Saprolite	453	0.00	1345	7.43	4520	67.23	1.84	9.05
Weathered	563	0.00	109	1.73	35.14	5.93	1.27	3.42
Fresh	4298	0.00	45.21	2.35	13.85	3.72	2.15	1.58
All	**5314**	**0.00**	**1345**	**2.72**	**401.5**	**20.04**	**2.04**	**7.37**

Note: CV = coefficient of variation

A statistical summary of the selected samples in different lithotypes is shown in Table 14-7 for Sulfur (S). Statistics for the generated 2.0 m composites in different lithotypes for S is also shown in Table 14-8.

Table 14-7 - Summary Statistics – Selected Sulfur Assay Samples.

LITH	NUMBER	MINIMUM	MAXIMUM	MEAN	VARIANCE	STANDDEV	LOGESTMN	CV
Saprolite	544	0.00	1.23	0.02	0.01	0.08	0.02	3.74
Weathered	663	0.00	6.55	0.31	0.68	0.82	0.25	2.70
Fresh	5528	0.00	10.00	2.40	3.84	1.96	3.20	0.82
All	**6735**	**0.00**	**10.00**	**2.00**	**3.95**	**1.99**	**5.32**	**0.99**

Note: CV = coefficient of variation

Table 14-8 - Summary Statistics – 2.0 m Composites (Sulfur).

LITH	NUMBER	MINIMUM	MAXIMUM	MEAN	VARIANCE	STANDDEV	LOGESTMN	CV
Saprolite	350	0.00	1.32	0.03	0.01	0.10	0.02	3.43
Weathered	458	0.00	6.14	0.33	0.63	0.80	0.23	2.44
Fresh	3762	0.00	10.00	2.48	3.17	1.78	3.01	0.72
All	**4570**	**0.00**	**10.00**	**2.07**	**3.43**	**1.85**	**6.06**	**0.89**

Note: CV = coefficient of variation

14.6 GEOSTATISTICAL ANALYSIS (VARIOGRAMS)

Variography for composited samples was completed using Snowden's Supervisor software. A variography model was fitted for composited data within X1 alteration ore models. For continuity modelling, a normal scores transform was used.

The anisotropy directions are coincident with the deposit shape (geological models). The strike of the deposit was adopted to be azimuth of the major axis. The azimuth of the major axis was selected to be 0° with a plunge of -50°. All data reported are the results from the back-transform of the normal scores. Semi-variograms and correlograms for Au and Ag (Figures 14-19 and 14-20) were calculated to analyze geometric and zonal anisotropy. Figures 14-21 and 14-22 show back-transformed results and summarizes the semi-variogram parameters for gold and silver respectively from the X1 dataset.

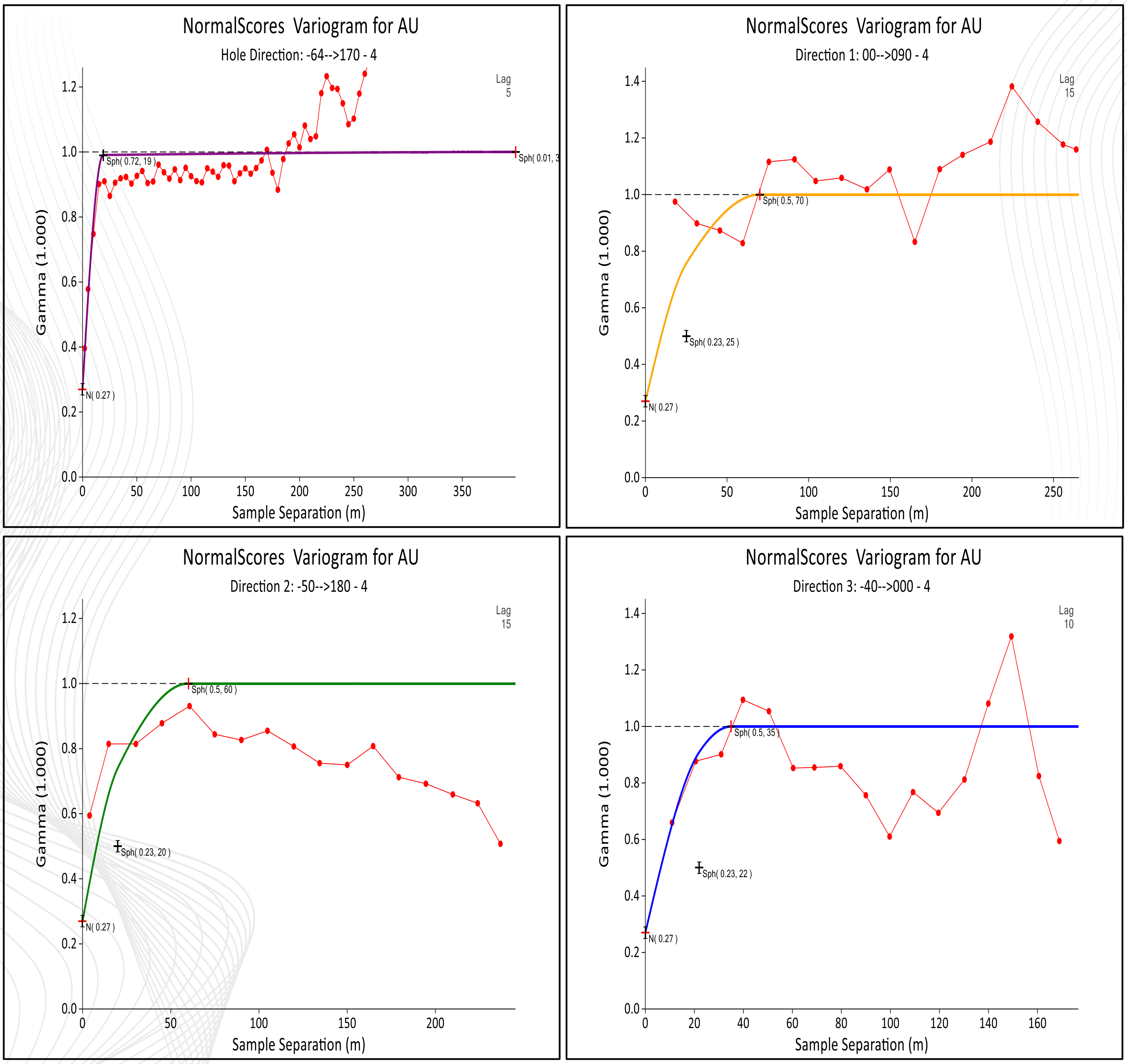


Figure 14-19 - X1 Model Semi-Variograms (Au).

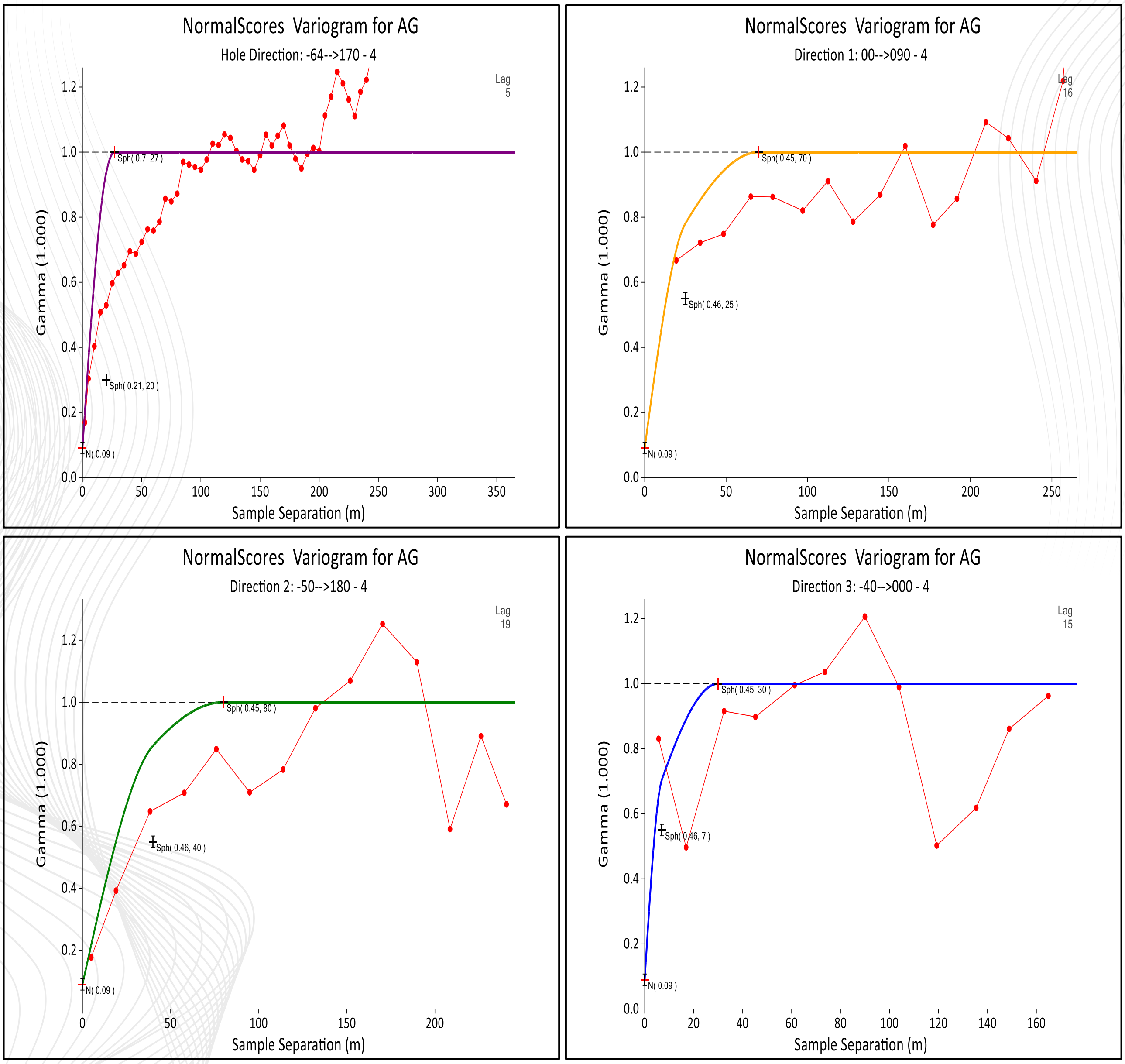


Figure 14-20 - X1 Model Semi-Variograms (Ag).

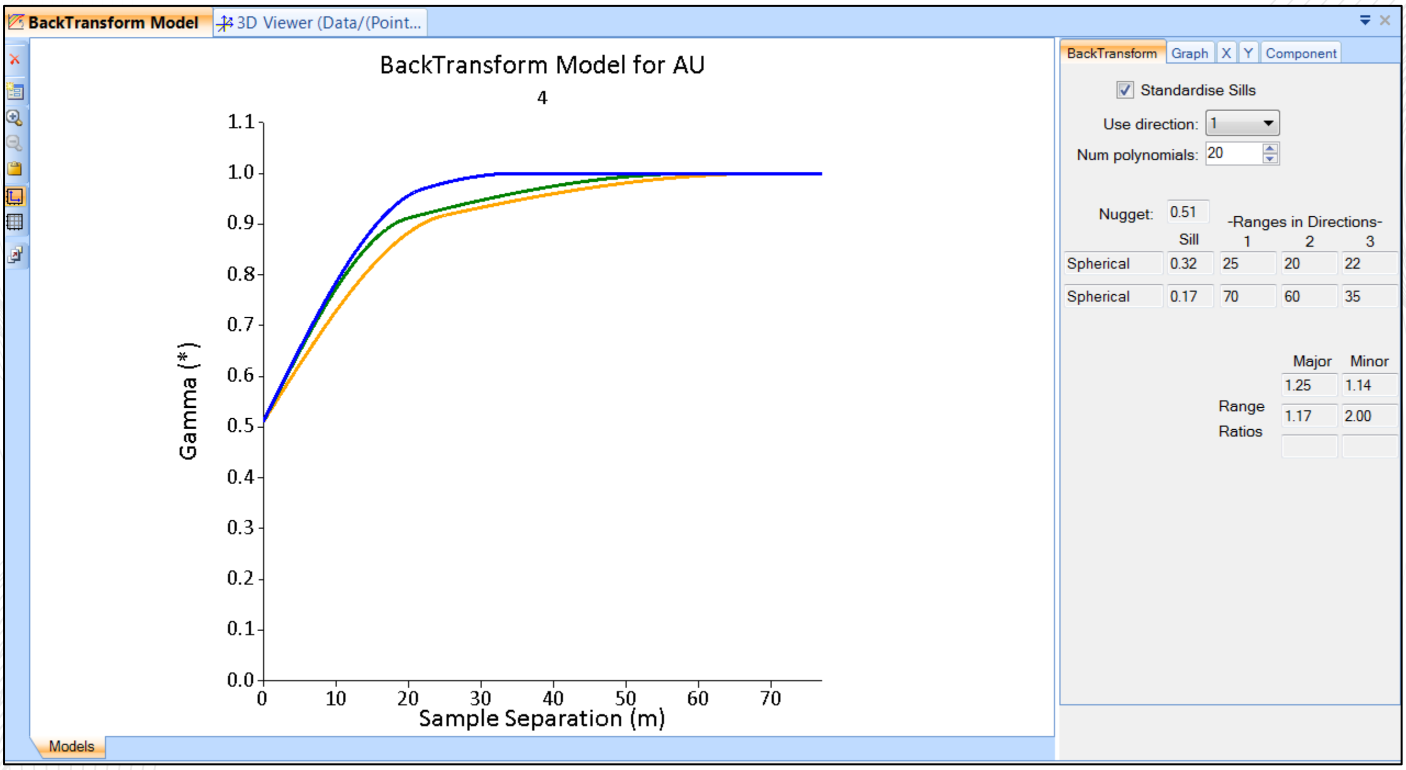


Figure 14-21 - X1 Back Transform Model (Au).

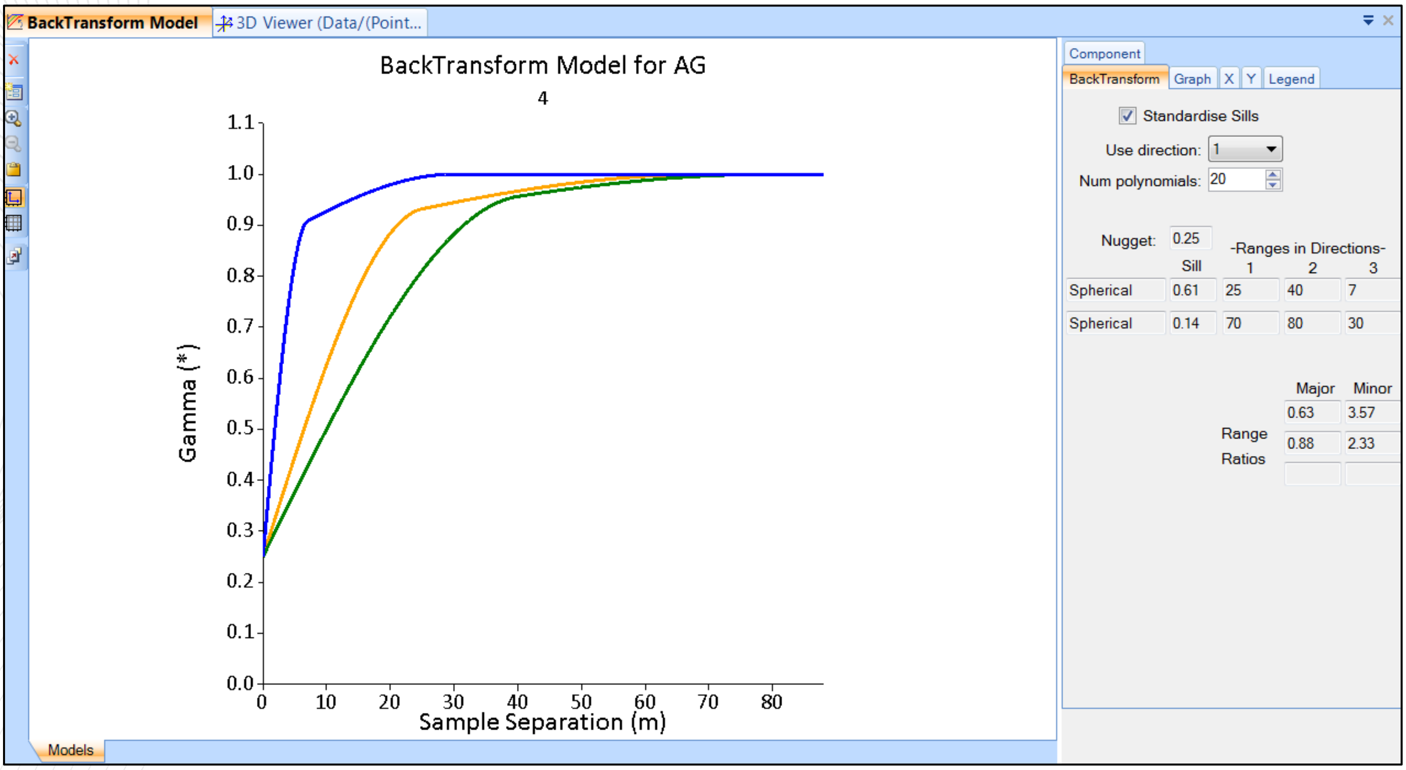


Figure 14-22 - Back Transform Model (Ag).

14.7 BLOCK MODELS SET UP

The block model limits were defined using UTM coordinates, South American Datum (1969), UTM Zone 21 South, and the block size selected for the model was 5 m x 5 m x 5 m. The model was not rotated. The block model definition is given in Table 14-9.

Table 14-9 - Block Model Definition (X1).

	ORIGIN	BLOCK SIZE (m)	NO. BLOCKS
Easting (X)	727,750	5	120
Northing (Y)	8,886,350	5	110
Elevation	425	5	65

14.8 GRADE INTERPOLATION

The grade interpolation used Ordinary Kriging (OK) with variography as set out in Section 14.7. The updated 3-D alteration models coded in the block model, was interpolated, using just the data points from inside that specific zone as the data source. The strong phyllic alteration, coded as rock type 4, and weaker phyllic alteration, coded as rock type 6, and a separate composite data set for each domain was used in the grade interpolation.

These two grade shell domains were modelled using 3 interpolation runs for Au and Ag. Given the current average grid spacing of about 30 m by 30 m in some parts of the X1 Deposit, the short 1st structure in the variograms was adopted for the first pass and used to classify the Mineral Resource in the Measured Resource category. From correlogram model, ellipsoid search distances were obtained using of range of the first structure for first pass and range of second structure of correlogram model for the second pass (Figures 14-21 and 14-22). The third pass adopted 2 times the range of the correlogram model. The entirety of the model was populated by gold and silver grades using the first 3 passes. At least two holes were used to estimate the blocks for first and second passes (defining three composites per hole as a maximum of samples used in grade interpolation). The same grade interpolation strategy also was used to populate the X1 block model with Sulfur (S) and Molybdenum (Mo) grades.

The search parameters for each interpolation run are listed in Table 14-10 and 14-11 for Au and Ag respectively. A typical cross-section with Au and Ag values through the estimated block model is shown in Figure 14-23 and 14-24 respectively.

Table 14-10 - Au Grade Interpolation Parameters (X1).

Search Reference	Search Distances (m)			Minimum No. of Composites	Maximum No. of Composites	No. Drill Holes	Maximum Composites per Hole
	X	Y	Z				
1	25	20	22	4	24	2	3
2	70	60	35	4	24	2	3
3	140	120	70	4	24	3	1

Table 14-11 - Ag Grade Interpolation Parameters (X1).

Search Reference	Search Distances (m)			Minimum No. of Composites	Maximum No. of Composites	No. Drill Holes	Maximum Composites per Hole
	X	Y	Z				
1	25	40	7	4	24	2	3
2	70	80	30	4	24	2	3
3	140	160	60	4	24	3	1

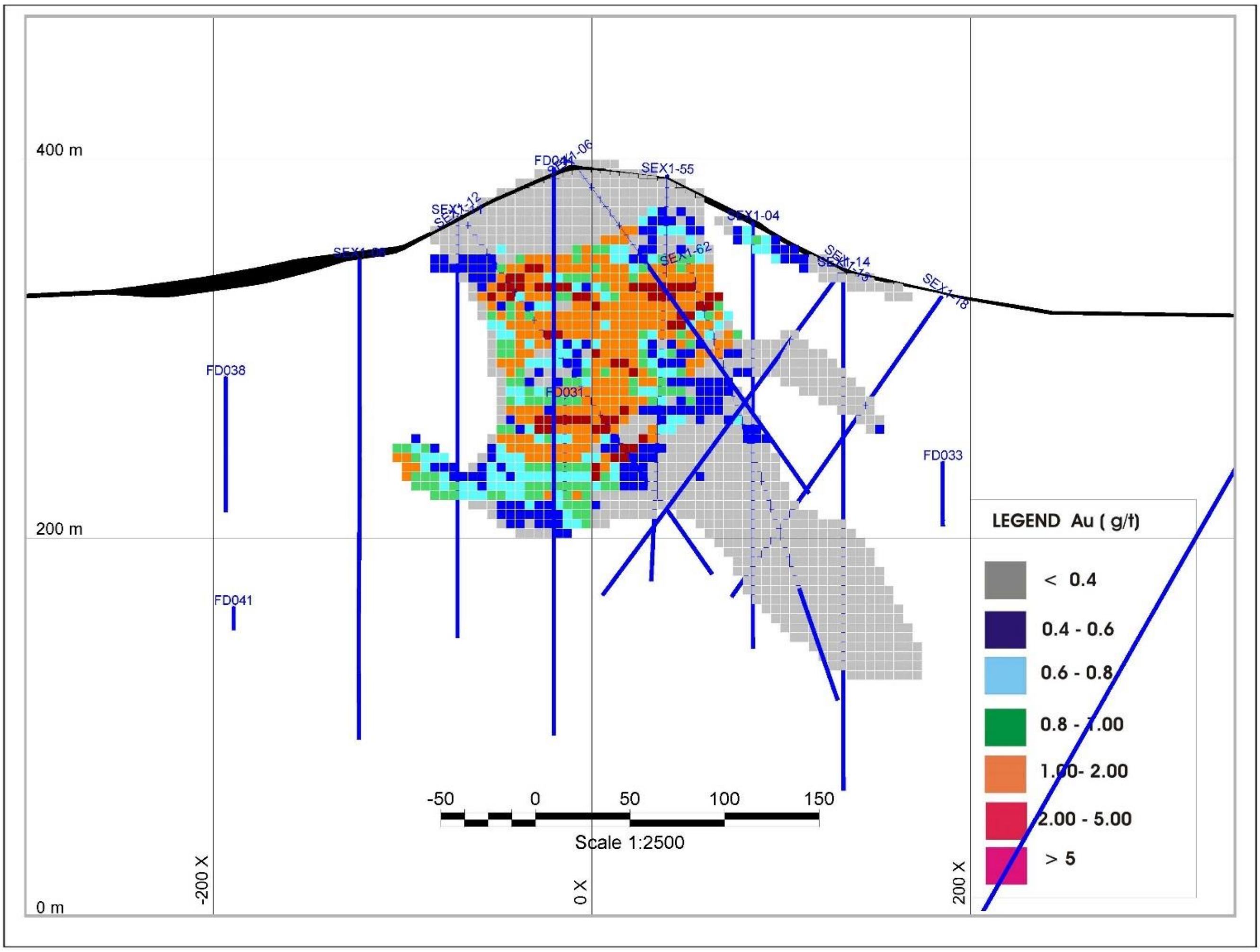


Figure 14-23 - Typical Cross-Section Through the X1 Block Model Estimated Grades (Au) (Looking N).

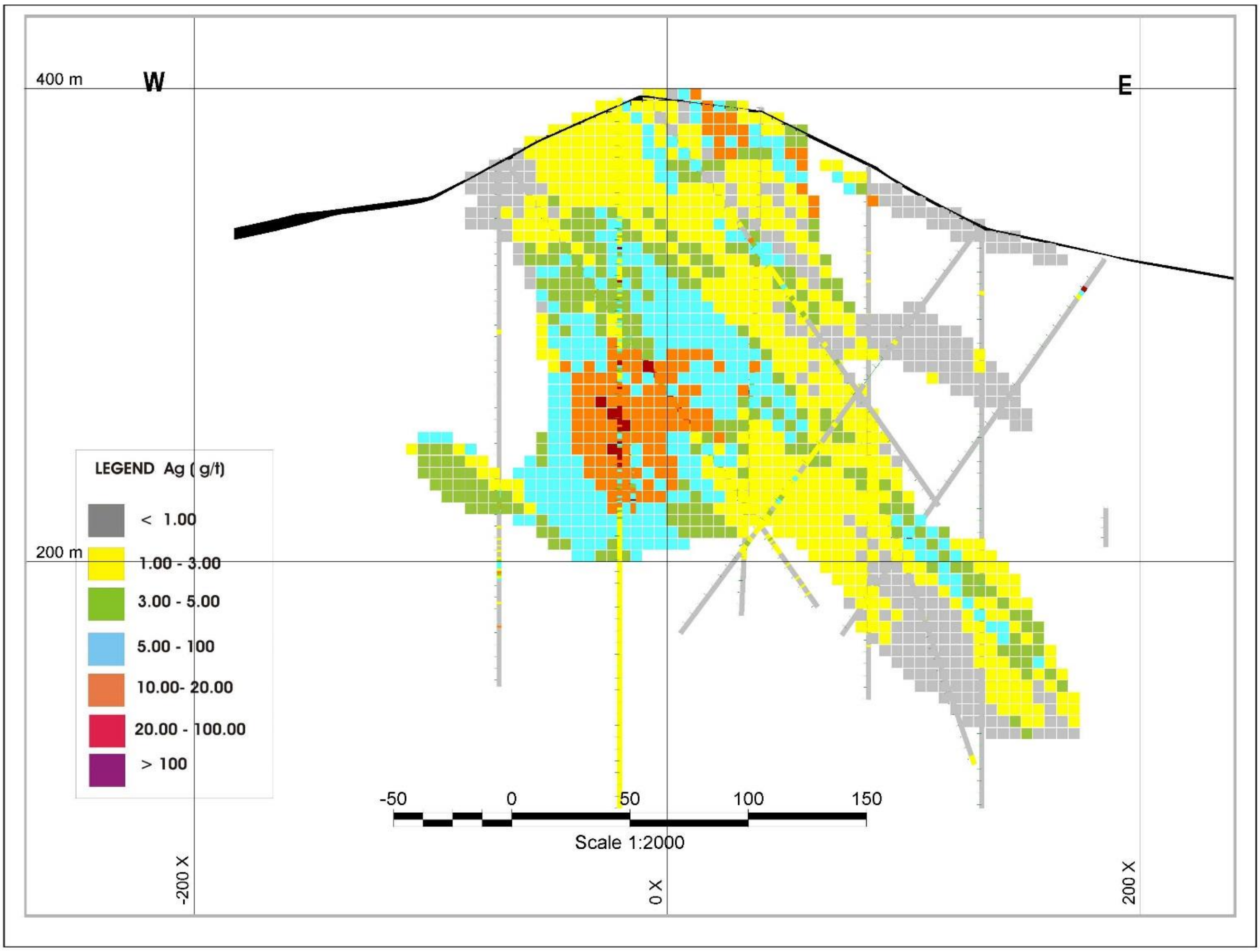


Figure 14-24 -Typical Cross-Section Through the X1 block Model Estimated Grades (Ag) (Looking N).

14.9 BLOCK MODEL VALIDATION

For both silver and gold, grade estimation was done using Ordinary Kriging (OK), Inverse Distance Squared ("ID^2") and Nearest Neighbor ("NN") methods and the results compared against each other for validation purposes (Table 14-12 and 14-13). A global comparison also was made between raw assays, composites which were used for the models and block model average grades at zero cut-off grades.

A visual comparison between composites and blocks was made section by section and level by level to investigate local bias in the Mineral Resource estimation.

Table 14-12 - Comparison of Global Average Au (g/t) Grades.

ASSAYS (Uncut)		BLOCK MODEL AVERAGES*		
Raw Samples	Composites	NN	ID	OK
0.51	0.49	0.43	0.43	0.43

Note*: NN = nearest neighbor
. ID = inverse distance (^2)
. OK - ordinary kriging

Table 14-13 - Comparison of Global Average Ag (g/t) Grades.

ASSAYS (Uncut)		BLOCK MODEL AVERAGES*		
Raw Samples	Composites	NN	ID	OK
2.62	2.72	2.35	2.35	2.37

Note*: NN = nearest neighbor
. ID = inverse distance (^2)
. OK - ordinary kriging

Spatial validation of the updated block model was completed using swath plots. Example swath plots are provided in Figure 14-25 and Figure 14-26. The swath plots show a satisfactory general trend of the various estimation methods to composite data and NN estimates with minor local deviations for estimated blocks based on influence of high yield samples specially in west part of the ore body. The plots show that local spike of grades both for Au and Ag silver composites effectively mitigated by using estimation methods in general.

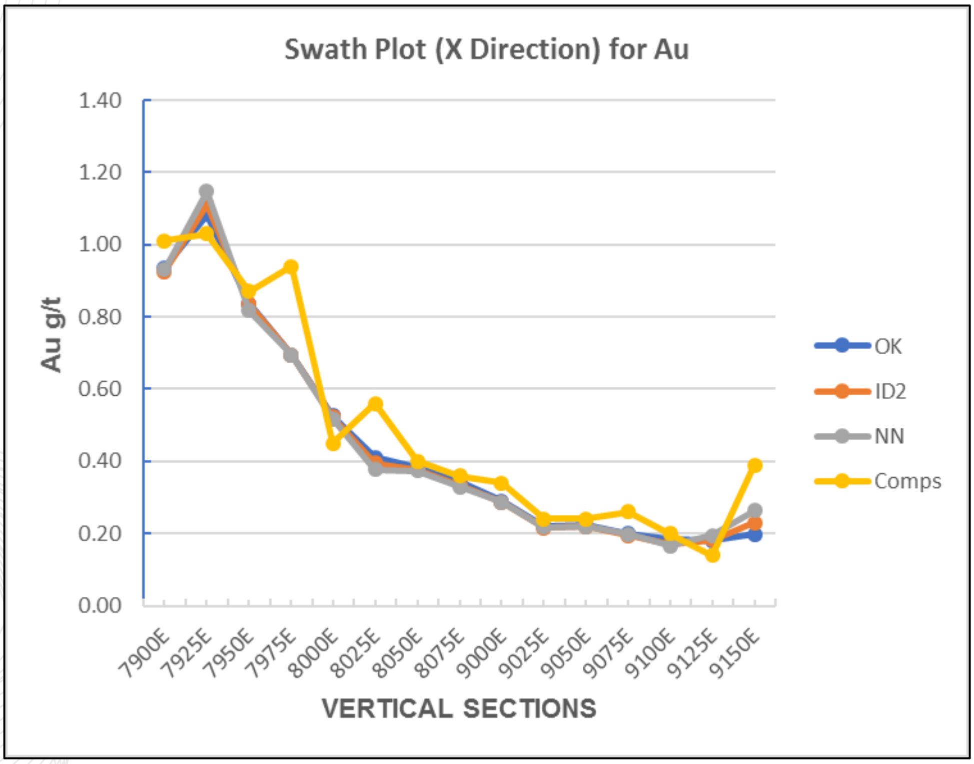


Figure 14-25 - W-E Swath Plot of Au Average Grades by Different Estimation Methods Versus Capped Composites (Comps)

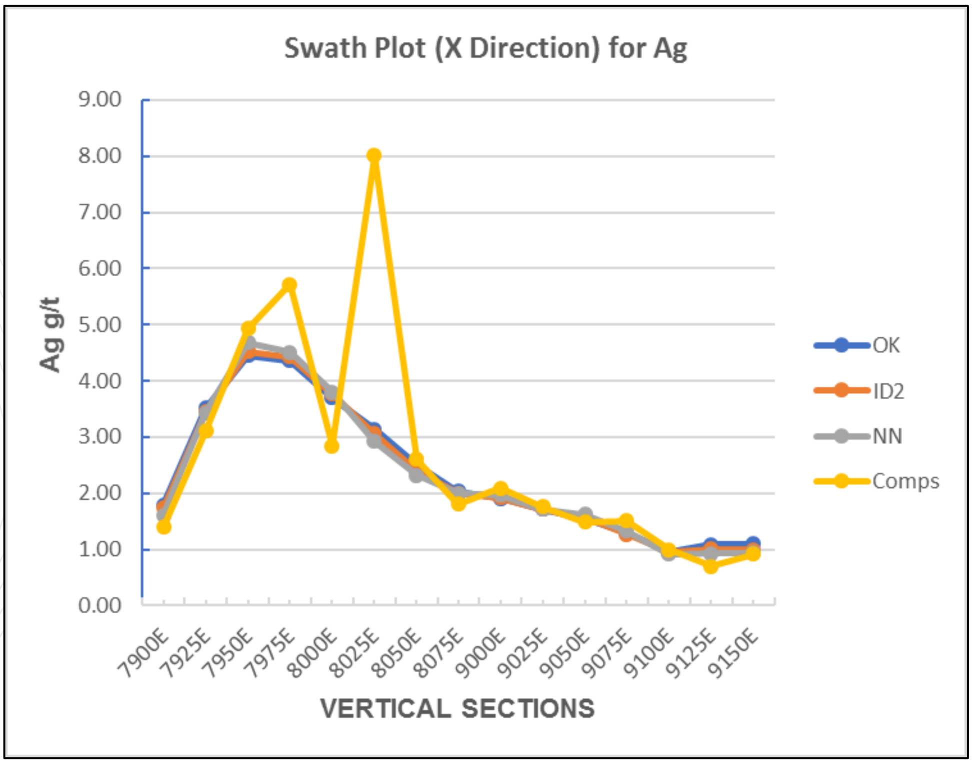


Figure 14-26 - W-E Swath Plot of Ag Average Grades by Different Estimation Methods Versus Capped Composites (Comps).

14.10 MINERAL RESOURCE CLASSIFICATION

The Mineral Resources for the X1 Deposits have been classified in accordance with the CIM definitions and standards for Mineral Resources and Mineral Reserves (CIM, 2014). The classification parameters consider the proximity and number of composite data. The block model then is coded accordingly for Measured (1), Indicated (2) and Inferred (3) for all three deposits.

The X1 model used the first and second passes to assign the Measured and Indicated categories, respectively. The passes were the result of the variography which was discussed in section 14.7. Other criteria such as number of drill holes, distance from drill hole data, and minimum and maximum number of composites were also used to estimate a block. The Mineral Resource classification criteria applied in the current study are those shown in Table 14-14.

Table 14-14 - Classification Criteria (X1).

Category	Pass No. (Okpass)	Approximate Distance (m)	Actual Distance of Composites	Min No. Comps	Max. No. Comps	Max. No. Samples per Hole
Measured	1	≤ 25 m	≤ 40	4	24	3
Indicated	2	>25 and ≤ 70	≤ 40	4	24	3
Inferred	3	No limit	>40	4	24	1

In order to avoid "spotted dogs" in classification, a polyline was constructed section by section for all Measured and Indicated blocks using the criteria above. Then a 3-D model was constructed and all blocks outside this model were assigned to the Inferred classification. Figures 14-27, 14-28 and 14-29 show the classification scheme in 3-D, plan view, and a cross-section in the middle of the X1 Deposit, respectively.

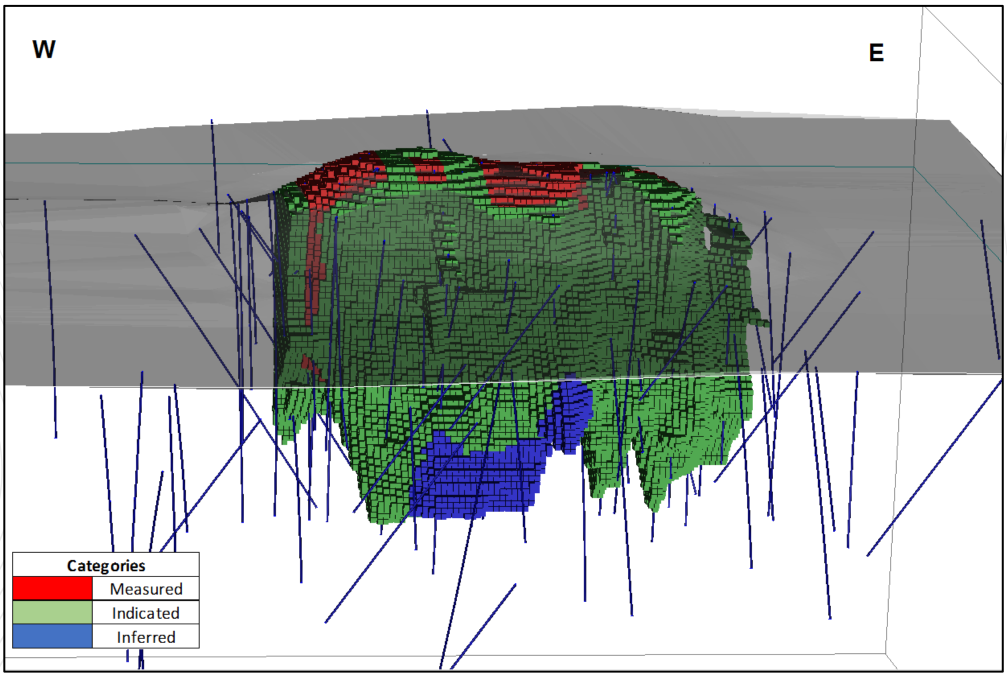


Figure 14-27 - Classification Scheme in 3-D (X1)-Looking N.

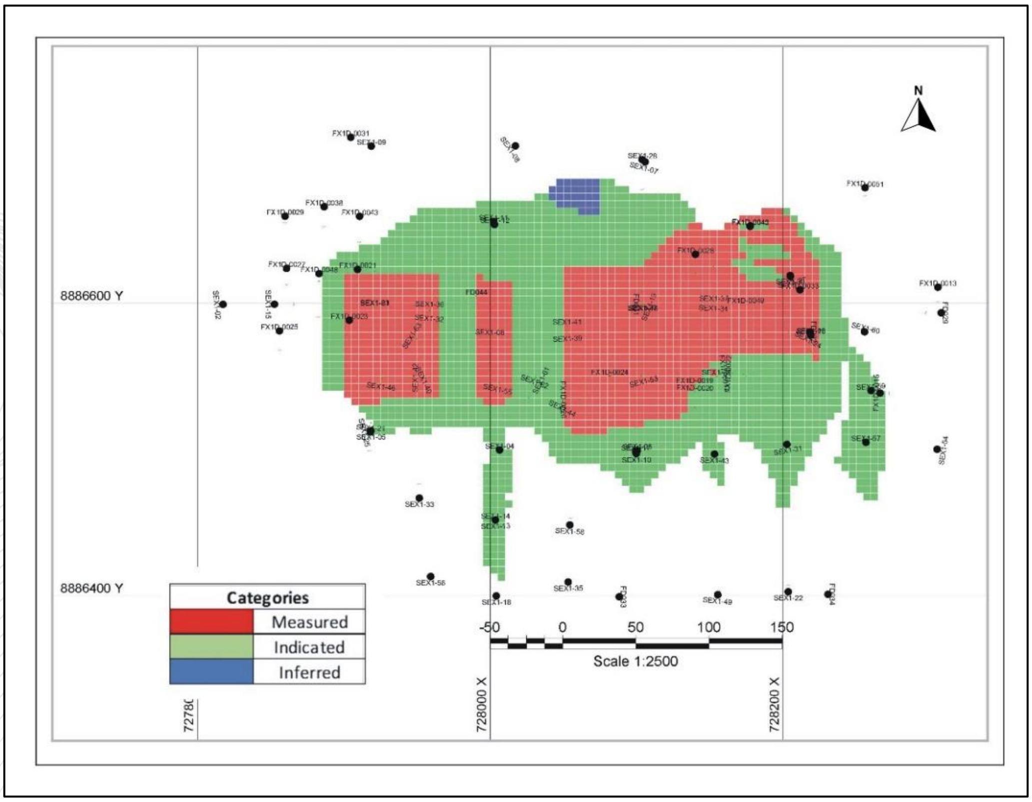


Figure 14-28 - Classification Scheme and Drill Holes in Plan View (X1-350 m elevation).

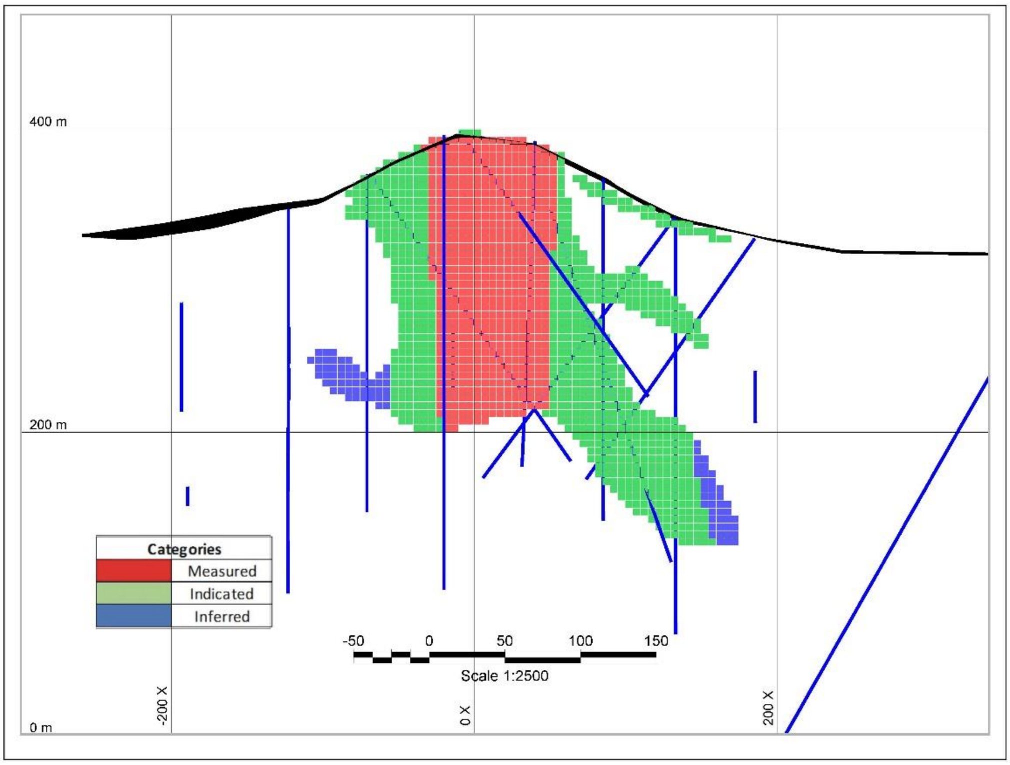


Figure 14-29 - Classification Scheme in Cross-Section View (X1).

14.11 MINERAL RESOURCE ESTIMATE

A Mineral Resource can only be declared for material which is considered to have potential for economic extraction at some point in the future. The cut-off at which a Mineral Resource is reported should also meet this criterion. It should not include material which does not have reasonable potential to be mined and processed.

In the case of the X1 Deposit, the larger envelope of phyllic alteration includes assays as low as 0.00 g/t Au and block grade as low as 0.008 g/t Au, therefore selecting a cut-off grade which determines the lowest grade to be mined is necessary for the disclosure of Mineral Resources. The detail of economic parameters for pit optimization and cut-off grade calculation is discussed in section of 15 of this report. A cut-off grade of 0.35 g/t Au was calculated based on EDEM (mining consultants) estimation. In the opinion of the QP, this is a reasonable assumption compared to other similar gold deposits. Table of 14-15 shows a summary of parameters which was used in establishing of the Mineral Resource and Mineral Reserve cut-off grade.

Table 14-15 - Cut-off Grade Assumptions.

DEPOSIT	CONCENTRATION PROCESS COSTS (US$/t)	G & A (US$/t)	ORE TRANSPORT MINE TO PLANT (US$/t)	TOTAL PLANT COST (US$/t)	Au SALES PRICE (US$/g)	PLANT RECOVERY	CUT-OFF GRADE
X1	15.44	1.64	1.50	18.58	44.21	95%	0.35

The updated Mineral Resource Estimate is based on the alteration models which encompassed all economic gold mineralization in the X1 Deposit. These mineralized domains were analyzed for grade capping and variography and were interpolated using the ordinary kriging method. Once the block model was completed it was classified into Measured, Indicated, and Inferred Mineral Resources. This was followed by a Lerchs-Grossman open pit optimization which resulted in the updated Mineral Resource Estimate presented in Table 14-16. Figure 14-30 shows Mineral Resource blocks within an optimized pit at, using US$1,800/oz gold price, in cross-sectional view. Table 14-17 lists the tonnage and amount of gold and silver within the resource pit shown in Figure 14-28. Figure 14-29 shows the tonnage curve, grade versus tonnes, for the resource pit.

Table 14-16 - X1 Mineral Resources.

Resources Classification	**Tonnes (t)**	**Au (g/t)**	**Au (oz)**	**Ag (g/t)**	**Ag (oz)**
Measured	4,692,520	1.14	172,000	3.85	580,810
Indicated	4,653,150	0.96	143,600	4.39	656,430
Measured + Indicated	**9,345,670**	**1.05**	**315,600**	**4.12**	**1,238,240**
Inferred	77,560	0.78	1,950	1.25	3,120

**Mineral Resource Notes and Assumptions*

1. *The Mineral Resource Estimate has an effective date of August 31, 2022.*
2. *The Mineral Resources are inclusive of Mineral Reserves.*
3. *Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.*
4. *The Mineral Resources in this estimate were calculated with the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions.*
5. *The base case cut-off grade for the estimate of Mineral Resources is 0.35 g/t Au.*
6. *The Measured, Indicated and Inferred Mineral Resources are contained within a limiting pit shell (using US$1,800 per ounce of gold) and comprise a coherent body.*
7. *A density model based on alteration and rock type was established for volume to tonnes conversion averaging 2.76 tonnes /m^3.*
8. *Contained metal figures may not add due to rounding.*
9. *Surface topography as of July 31, 2021.*
10. *The Mineral Resource Estimate for the X1 Deposit was prepared by Farshid Ghazanfari, P.Geo., a Qualified Person as that term is defined in NI 43-101.*

Table 14-15 shows the sensitivity of Mineral Resources in different cut-off grades and Figure 14-31 Grade-Tonnage curve for Measured and Indicated Mineral Resources within US$1800/oz. optimized pit shell.

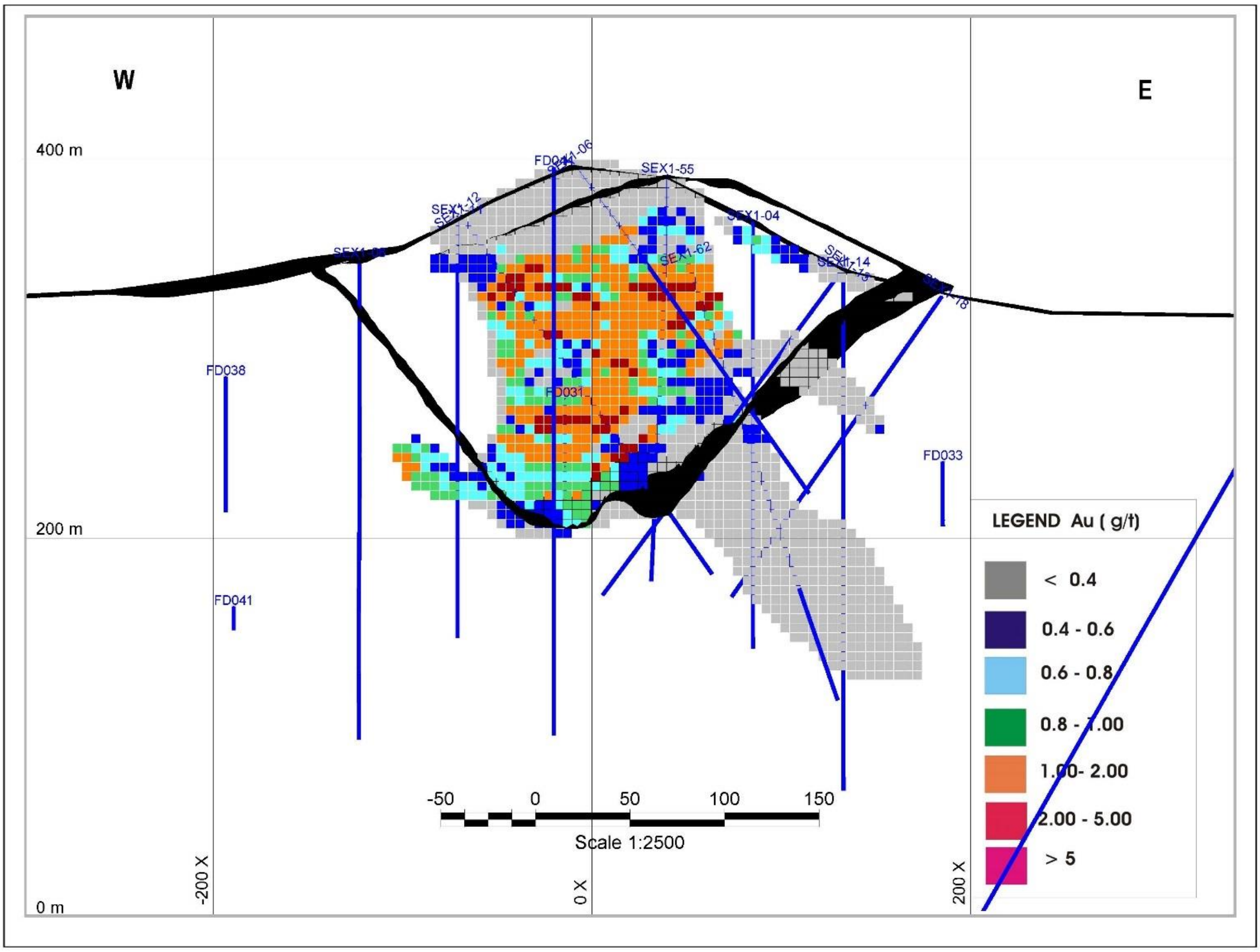


Figure 14-30 - Typical Cross-Section Through the X1 Block Model Estimated Grades (Au) showing resource pit outline in black. Looking N).

Table 14-17 - X1 Grade - Tonnage for Measured and Indicated Mineral Resources (Inside the resource Pit).

Au Cut-Off	TONNES	Au	Au
g/t	tt	g/t	oz
0	16,189	0.67	348,727
0.1	13,734	0.78	344,415
0.2	11,500	0.91	336,457
0.3	10,079	1.00	324,047
0.4	8,669	1.11	309,373
0.5	7,149	1.25	287,307
0.6	5,896	1.39	263,489
0.7	4,909	1.54	243,055
0.8	4,168	1.68	225,127
0.9	3,619	1.81	210,600
1	3,153	1.94	196,660

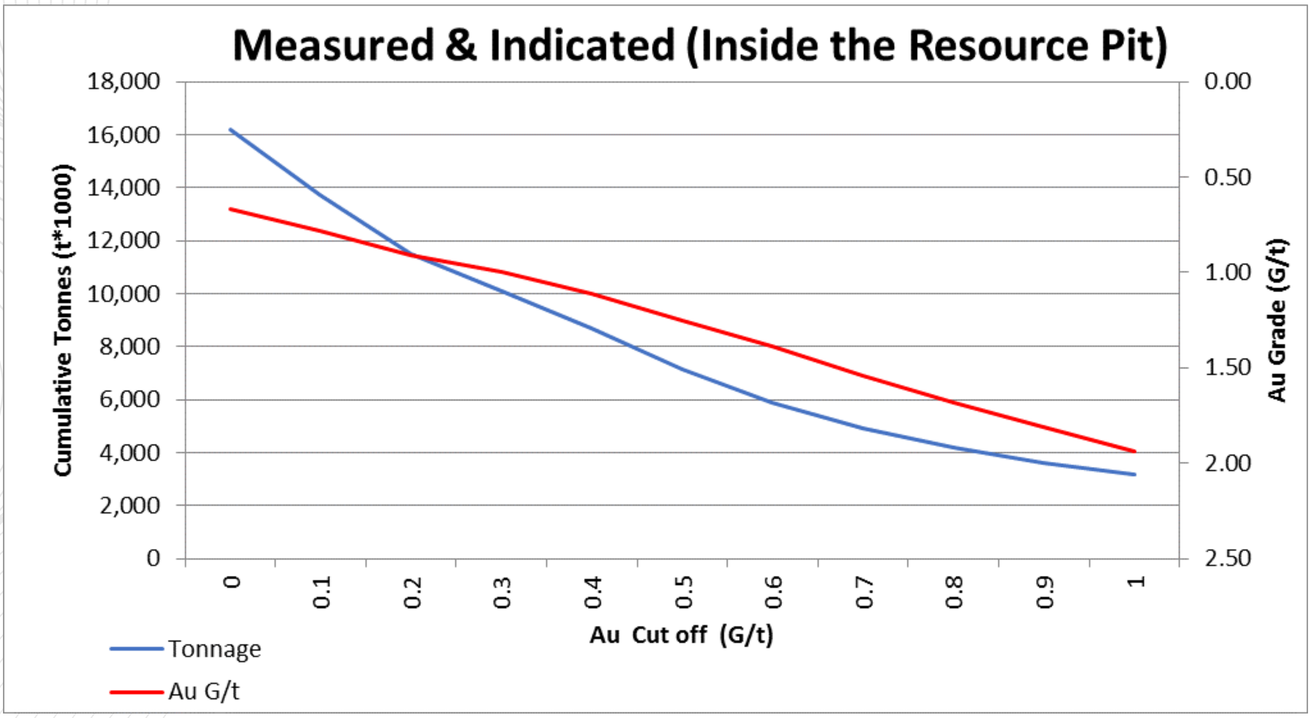


Figure 14-31 - X1 Grade-Tonnage Curve for Measured and Indicated Mineral Resources (Inside the resource Pit).

14.12 FACTORS THAT MAY AFFECT THE MINERAL RESOURCE ESTIMATE

Areas of uncertainty that may materially impact the Mineral Resource Estimate include:

- Changes to long to term metal price assumptions.
- Changes to the input values for mining, processing, and general, and administrative costs to constrain the estimate.

- Changes to local interpretations of mineralization geometry and continuity of mineralized zones.
- Changes to the density values applied to the mineralized zones.
- Changes to metallurgical recovery assumptions.
- Changes in assumptions of marketability of the final product.
- Changes to assumptions with an existing agreement or new agreements.
- Changes to environmental, permitting, and social licence assumptions.

14.13 COMMENTS ON MINERAL RESOURCE ESTIMATE

As described in section 12 of this report, silver values have analytical inaccuracy due to the analytical method that was used during different sampling and assaying campaigns in the Matupá Project, The grade estimation for silver was provided only for the information purposes and estimation of global gold to silver ratio. In the opinion of QP, there is not enough confidence in silver values to be the basis of any economic evaluation for a feasibility level study. As a result, the gold equivalent is not reported for purpose of this report.

The QP is not aware of any environmental, legal, title, taxation, socio to economic, marketing, political or other relevant factors that would materially affect the estimation of Mineral Resources that are not discussed in this Technical Report. The QP is of the opinion that the Mineral Resources were estimated using industry to accepted practices and conform to the 2014 CIM Definition Standards and 2019 CIM Best Practice Guidelines. Technical and economic parameters and assumptions applied to the Mineral Resource Estimate are based on parameters received from other consultants and reviewed by QP to determine if they were appropriate.

15. MINERAL RESERVE ESTIMATION

15.1 INTRODUCTION

Aura Matupá Mineração Inc. (Aura) retained EDEM Engenharia de Minas (EDEM) to complete the mining sections for this Feasibility Study (FS) for the Matupá Project, specifically the X-1 gold mineral deposit located in the State of Mato Grosso, Brazil.

EDEM, through Qualified Person Eng. Luiz Eduardo Campos Pignatari, was hired by Aura Matupá Mineração Ltda to prepare an independent technical report and calculate the Mineral Reserves for the Matupá Project, located between the municipalities of Guarantã do North and Matupá, State of Mato Grosso. This technical report is in accordance with the requirements of Canadian National Instrument NI-43.101 reporting.

The previous technical report, the Preliminary Economic Assessment (PEA) was based on a 1.3 Mtpa feed to the process plant, and forms the basis for this specific study.

The mining issues are summarized in Section 15 (Mineral Reserves) and Section 16 (Mining Method) of this report including:

- Data validation.
- Pit optimization with Lower intermediary slope angle. (45 degree).
- Mining schedule: detailed as follows: the mining pit design-will be on a quarterly basis and monthly calculated in the first two years of production; after the third year the pit design and calculation are to be formed on an annual basis.
- Mining Method.
- Estimates of the fleet and manpower required to achieve the plan.
- Mining operating and capital costs.

The X1 Deposit occupies a topographic high point (hill) and is hosted by the Matupá Intrusive Suite near the contact with the mafic/ultramafic Flor da Serra Suite. The extents of the deposit have dimensions of 400 m east-west and 250 m north-south.

The mineralized rocks are mainly quartz, muscovite/sericite, and pyrite, which is the main sulfide, followed by chalcopyrite partially replaced by covellite, chalcocite, and minor bornite in chalcopyrites as an alteration product of it. At intense hydrothermal alteration zones muscovite can make up to 40% of the rock.

15.2 RESERVES INVENTORY

The bases and procedures for Mineral Reserves estimation for the Matupá Project, for the X1 mineral deposit is presented in this section.

*Table 15-1 - Matupá Project Mineral Reserve**

RESERVE CLASSIFICATION		PROVEN		PROBABLE		TOTAL	
Ore Type	Material Type	Tonnes (kt)	Grade AU (g/t)	Tonnes (kt)	Grade AU (g/t)	Tonnes (kt)	Grade AU (g/t)
LOW-GRADE ORE	Soil Proven	7.0	0.40	-	-	7.0	0.40
	Soil Probable	-	-	4.4	0.40	4.4	0.40
	Saprolite Proven	16.0	0.41	-	-	16.0	0.41
	Saprolite Probable	-	-	17.7	0.40	17.7	0.40
	Fresh Rock Proven	180.2	0.40	-	-	180.2	0.40
	Fresh Rock Probable	-	-	223.2	0.41	223.2	0.41
	SUB-TOTAL	**203.2**	**0.40**	**245.3**	**0.40**	**448.5**	**0.40**
HIGH-GRADE ORE	Soil Proven	71.9	0.73	-	-	71.9	0.73
	Soil Probable	-	-	47.6	0.70	47.6	0.70
	Saprolite Proven	253.7	1.07	-	-	253.7	1.07
	Saprolite Probable	-	-	278.2	0.96	278.2	0.96
	Fresh Rock Proven	3,270	1.40	-	-	3,270.4	1.40
	Fresh Rock Probable	-	-	4,114.5	1.04	4,114.5	1.04
	SUB-TOTAL	**3,596**	**1.36**	**4,440.3**	**1.03**	**8,036.4**	**1.18**
TOTAL MINERAL RESERVE		**3,799**	**1.31**	**4,685.6**	**0.99**	**8,484.9**	**1.14**

**Note:*

1. *The Mineral Reserve estimates were prepared in accordance with the CIM Standards on Mineral Resources and Reserves.*
2. *The Mineral Reserve Estimate has an effective date of August 31, 2022.*
3. *The Mineral Reserve Estimate is based on an updated optimized shell using US$1,500/oz gold price, average dilution of 3%, mining recovery of 100% and break-even cut off grades of 0.35 g/t Au for X1 pit.*
4. *Contained metal figures may not be added due to rounding.*
5. *Surface topography as of July 31, 2021.*
6. *Mineral Reserve estimate for Matupá Project was prepared under the supervision of Luiz Pignatari, P.Eng.,a "qualified person", as that term is defined by NI 43-101.*
7. *The concentration plant recovery was established by Consolidations Tests Recovery model presented in the "technical report".*
8. *The silver grades and metal contents were not considered in the reserve calculation as still there are doubts about the metallurgical recovery during the gold production process.*

15.2.1 RECOVERY MODEL FROM METALLURGICAL CONSOLIDATION TESTS

The Figure 15-1 shows the Metallurgy Recovery Model ascertained from the Consolidations tests that form the basis for these Mineral Reserve calculations.

Model w/o Gravity	
Au (ppm)	1.00
Recov %	93.2%

Model w Gravity	
Au (ppm)	1.00
Recov %	94.34

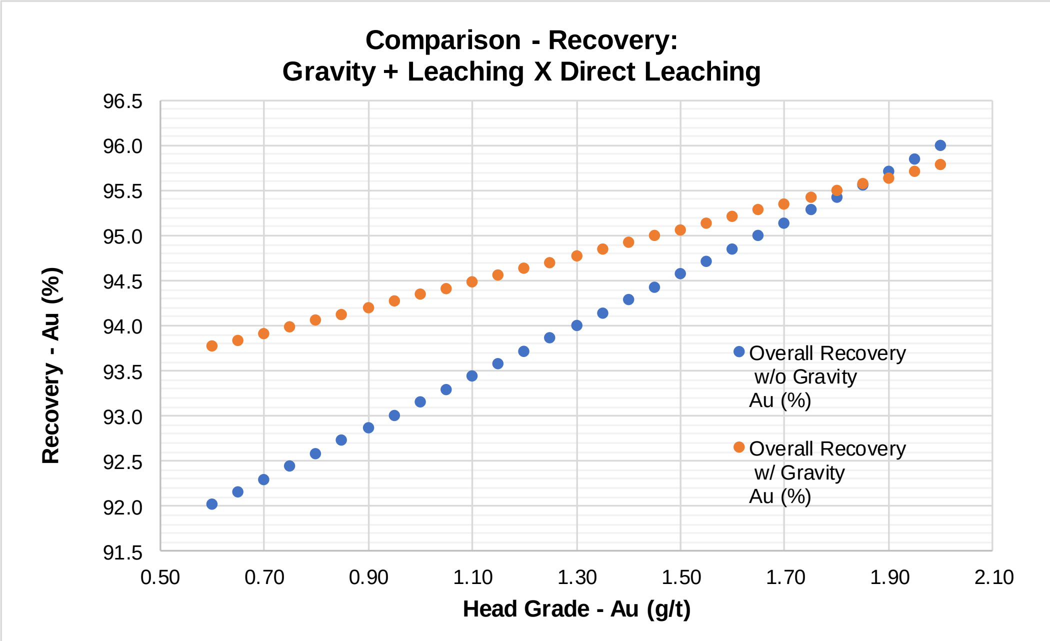


Figure 15-1 - Matupá Project Metallurgical Recovery Model After Consolidations Tests.

15.2.2 DOCUMENTS AND INFORMATION PROVIDED

This study is based on information supplied by Aura and includes:

- Geomechanics Technical Report 2020.
- Exploration Drilling database.
- Block model from the PEA study of 2021 (GE21 Independent Report).
- Tonnage feed plant plan.
- Topographic surface.
- Mineral Resource block models.
- Physical constraints (tenement boundaries, environmental restrictions, etc.).
- Metallurgical recoveries, ramp-up curve, and costs of the processing plant.
- General and administration costs.
- Mining subcontracted operation costs.
- Diesel price per liter in the Project region and salaries costs.

- Exchange and discount rates.
- Technical report.

15.3 GEOTECHNICS STUDIES

The basis for the existing geotechnical and hydrogeological studies of the Matupá mineral deposit were:

- Analysis of existing geological and geotechnical data.
- Description of the drill cores carried out for the purpose of preparing the geological model (profiling and photographs).
- Geotechnical laboratory test results, including uniaxial, triaxial, and diametral compression tests, shear tests, and physical index and permeability.
- Development of a 3-D geomechanics conceptual model.
- Geotechnical sectoring, stability analysis and pit design parameters.
- Hydrogeological studies developed.

The reliability of the geotechnical data was verified through the geomechanical description of 12.67% of the existing exploration drill holes' core which determined: the lithological identification, the water level along the holes, and the visual tactile characterization of the materials; the evaluation of the drill hole core recovery percentage, the calculation of the RQD, the degree of alteration (A), the degree of resistance (R), the degree of coherence (C) and the discontinuities frequencies level.

The Aura's database presents lithological classification associated to the geomechanics information in 21 exploration drill holes for geological purposes. The holes were selected to provide the geomechanics classification by lithology. Table 15-2 presents the features of each lithology present in the X1 Deposit.

The lithology abbreviations to be used with the following tables is listed in the Legend.

Legend

- BGRD: Biotite Granodiorite with K-feldspar (potassium feldspar) porphyry.
- CZGRD: Contact Zone – Granodiorite with fractures infilled by centimeter to decimeter sized porphyritic quartz-feldspar. Irregular, centimeter thick xenoliths of aphanitic mafic rock usually occur in this zone.
- MD: gabbro/diorite mafic intrusions.
- QFP: Porphyritic feldspar quartz.

  Waste: No classification regarding lithology was presented.

Table 15-2 presents the summary of information obtained from the database together with the geomechanical classification developed for each of the lithologies and features.

Table 15-2 - Summary of Geomechanics Rock Classification and Lithology.

CLASSIFICATION	LITHOLOGY				
	BGRG	CZGRD	MD	QFP	WASTE
Soil					X
Saprolite		X			
Altered Rock		X	X	X	X
Fresh Rock	X	X	X	X	X

Therefore, the following subsections present the segregation and characterization developed for each lithology, providing later the adequate grouping of the rock mass fractions.

The initial rock mass geomechanics classification is based on the information presented in the Table 15-3. The discontinuities spacing was obtained via correlation of the mean curve as presented by Bieniawski (1989). The rock alteration factor was adopted as a parameter to determine the index related to the condition of the discontinuities. This factor was related to the condition of the discontinuities due to the presence of water, weathered rocks, etc.Based on the premise and practical recommendations presented by Read & Stacey (2009) for groundwater, the rock mass was determined to be completely dry. Any pore pressure in the rock mass will be considered in the stability analysis.

Table 15-3 - Indexes for Geomechanics Classification.

GEOMECHANICAL CLASSIFICATION	Technical Criteria					
	UCS	RQD	Discontinuities Spacing (1)	Discontinuities Conditions (2)	Underground Waters	TOTAL
BGRD	12	17	8,00	30	15	82
CZGRD_RD	4	3	5,00	20	15	47
CZGRD_RS	12	17	4,00	30	15	48
MD_RD	4	3	0,00	20	15	42
MD_RS	12	13	8,00	30	15	78
QFP_RD	4	3	5,00	0	15	27
QFP_RS	12	17	10,00	30	15	84
WASTE_RD	4	3	5,00	20	15	47
WASTE_RS	12	8	8,00	30	15	73

1. Based on graphic correlation between RQD and Discontinuities Spacing (Bieniawiski – 1989)
2. Based on the altered rock proportion

From these results, rocks with similar mechanical characteristics can be grouped to provide the final sectorization of the X1 pit (Table 15-4).

Table 15-4 - Lithologies Grouping by Geomechanics Classification.

LITHOLOGY	RMR	ROCK MASS CLASSIFICATION
BGRD	82	Class II
QFP_RS	84	
CZGRD_RS	78	Class III
MD_RS	78	
WASTE_RS	73	
CZGRD_RD	47	Class IV
MD_RD	42	
WASTE_RD	47	
QFP_RD	27	

15.3.1 SLOPE ANGLE SELECTION

The slope orientation was determined by geometric configuration, dividing the into five sectors was adopted; it was based on the (Azimuth) of a typical pit for the X1 mineral deposit (Figure15-1). The kinematic analysis defined the sections.

The Failure probability assessment through structural factors was done by considering different face angles for each sector to optimize the angles to be adopted. Kinematic analyzes were developed for each of the evaluated situations.

Table 15-4 presents the risk associated to each rock sector. In this model a maximum probability rupture of 20% was adopted for the X1 mineral deposit pit. As the probabilities do not change significantly with the variation of the rock slope's face angle, the maximum angle of 85° for class II rock was adopted. In addition, the local geological structures were analyzed to determine any potential issues, which gives more reliability to this geotechnical premise.

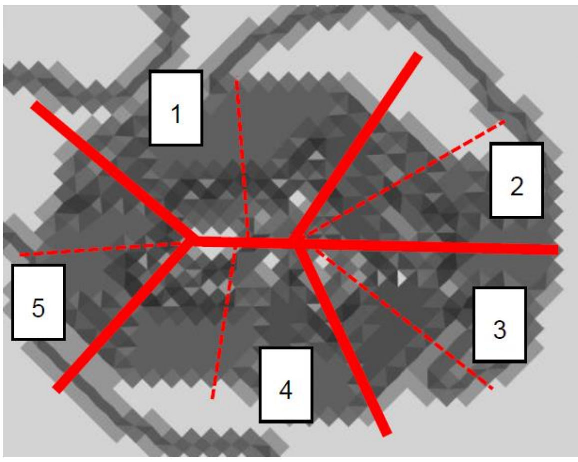


Figure 15-2 - Sectors Defined in X1 Typical Pit and the Sections for the Stability Analysis.

Table 15-5 - Summary Worksheet Risk Based on the Bench's Face Angle.

SECTION	PLANAR (%)			WEDGE (%)			FLEXURAL TOPPLING (%)			DIRECT TOPPLING (%)		
	85º	80º	70º	85º	80º	70º	85º	80º	70º	85º	80º	70º
1	**8,35**	**8,03**	**6,75**	**20,36**	**18,61**	**14,83**	**5,57**	**4,71**	**4,28**	**11,26**	**9,46**	**6,85**
2	**4,60**	**4,18**	**6,75**	**16,53**	**14,59**	**14,83**	**10,39**	**9,31**	**4,28**	**9,89**	**8,39**	**6,85**
3	**4,60**	**4,18**	**6,75**	**16,53**	**14,59**	**14,83**	**10,39**	**9,31**	**4,28**	**9,89**	**8,39**	**6,85**
4	**70,70**	**6,53**	**5,14**	**18,03**	**16,14**	**12,52**	**6,96**	**6,32**	**4,60**	**10,11**	**8,56**	**6,15**
5	**7,17**	**6,53**	**6,00**	**18,57**	**16,82**	**13,75**	**9,42**	**8,89**	**7,60**	**9,06**	**7,51**	**5,05**

The face angle definitions for the different rock types were determined by kinematic analysis and experience from other applications. Considering that the global slope intersects 3 classes of rock mass, the face angle definitions were indicated for each one of the classes, summarized in Table 15-6.

Table 15-6 - Bench's Face Angle by Geotechnical Domain.

PARAMETER	**GEOTECHNICS DOMAINS**		
	1	2	3
Geomechanics Class	**Soil/Saprolite**	**Class III Rock**	**Class II Rock**
Bench Face Angle	**40º**	**70º**	**85º**

15.3.2 GEOTECHNICS DRILLING HOLES

Three additional drill holes were planned to drill the X1 Target of the Matupá Project, totaling 900 m. These holes correspond to drilling orientations (azimuth) that had not yet been performed over the Deposit area. These holes will intercept with different positions of the Deposit and respective rock mass.

The final holes' depths are compatible with the available geological model and mathematical pit presented. The holes' location, direction, and length are represented in Table 15-7.

Table 15-7 - Geotechnics Drilling Holes: Location, Direction and Length.

DRILLING HOLE	COORDINATES			DIRECTION		LENGHT (m)
	EASTING	NORTHING	ELEVATION	STRIKE	DIP	
SGT-OG-01	757,887.00	8,886,536.00	355.061	90º	30º	350
SGT-OG-02	728,079.00	8,886,643,00	371.427	210º	45º	250
SGT-OG-03	728,112.67	8.886,460.77	347.629	330º	30º	300

15.3.3 GEOTECHNICS PARAMETERS

The detailed main technical information report was obtained from the geotechnical geological X1 mineral DEPOSIT database. Database Technical verifications were developed either through statistical validation, through geological-geotechnical descriptions. It there was coherence between the evaluated data, which allows the use of such data as a basis to the orebody X1 geomechanically classification.

The geomechanically classification identified the most significant parameters that influence the behavior of the rock mass, allowing a groupings division in classes. The rock mass that surrounds the rock mass involving X1 mineral deposit can be divided into 4 distinct geomechanics classes, as expressed in Table 15-8.

Table 15-8 - Geotechnics' Classification to the Rock Masses for the X1 Mineral Deposit.

<table>
<tr><th>LITHOLOGY</th><th>GEOMECHANICS CLASSIFICATION</th></tr>
<tr><td>BGRD</td><td rowspan="2">CLASS III</td></tr>
<tr><td>QFP_RS</td></tr>
<tr><td>CZGRD_RS</td><td rowspan="3">CLASS III</td></tr>
<tr><td>MD_RS</td></tr>
<tr><td>WASTE_RS</td></tr>
<tr><td>CZGRD_RD</td><td rowspan="4">CLASS IV</td></tr>
<tr><td>MD_RD</td></tr>
<tr><td>WASTE_RD</td></tr>
<tr><td>QFP_RD</td></tr>
<tr><td>CZGRD_SAP</td><td rowspan="3">CLASS V</td></tr>
<tr><td>WASTE_SOIL</td></tr>
<tr><td>WASTE_SAP</td></tr>
</table>

Note: Lithology codes are listed in section 5.3 Legend

Geomechanical classes IV and V were treated together in the models. The determination of the geometric parameters of the slopes were considered for the discontinuities to the resistance of the rock mass. Therefore the safety factors obtained in the stability analysis, using the limit equilibrium method, demonstrate that the geotechnical parameters provided for each of the analyzed sectors will allow the pit to be developed with geotechnical safety. Thus, the sectorization of the pit was divided into five sectors. These sectors are geometrically divided in the pit as a function of the direction of the global slope. After the detailed analyzes by sector and by geomechanically classes, the design criteria for this phase of the study were defined, as shown in Table 15-9.

Table 15-9 - Geotechnics' Design Criteria by Sector to the Rock Masses for the X1 Orebody.

SECTION	GEOMECHANICS CLASSIFICATION	FACE ANGLE	BERM WIDTH	BENCH HIGH		GENERAL ANGLE (1)	IRA (2)	OSA(3)
				OPERATIONAL	CLOSURE			
1	II	85º	5 m	5 m	10 m	55 º	60º	45º
	III	70º				46 º	49º	
	SOIL/SAP	40º				-	30º	
2	II	85º	5 m	5 m	10 m	55 º	60º	49º
	III	70º				46 º	49º	
	SOIL/SAP	40º				-	30º	
3	II	85º	5 m	5 m	10 m	55 º	60º	49º
	III	70º				46 º	49º	
	SOIL/SAP	40º				-	30º	
4	II	85º	5 m	5 m	10 m	55 º	60º	47º
	III	70º				46 º	49º	
	SOIL/SAP	40º				-	30º	
5	II	85º	5 m	5 m	10 m	55 º	60º	51º
	III	70º				46 º	49º	
	SOIL/SAP	40º				-	30º	

1. [1] Maximum overall angle per geomechanical class.
2. [2] Angle between ramps by geomechanical class
3. [3] Maximum global angle per sector.
4. Consider a 7-meter berm in the contact between domains 1 and 2.
5. ** Simulated with a ramp width of 15 meters.

There is a real possibility to increase the pit angles due to the high safety factors calculated into the current design and the good rock mass quality. The geometry limitation is related to the equipment operation and fleet capacity limitations. It is recommended that studies using different fleet and production configurations are carried out to find the best geometric configuration to allow for mining resource optimization.

For optimal pit simulation in the NPV software an angle average value of 50 degrees was used.

15.3.4 CLIMATE AND HYDROLOGY

The X1 mineral deposit region is in a humid continental equatorial climate unit (IB3): low latitude (8° to 9° south latitude) with altitudes between 100 and 300 meters which creates a mega thermal condition, where annual average temperatures fluctuate between 25 .7 and 24.7°C, with the maximum between approximately 32.0 and 33.0°C and the minimum between 19.5 and 21.0°C. The greatest thermal differences (amplitude) are associated with the day and night cycle and are not seasonal, that is, the daily thermal amplitude varies between 10° and 12°C, while the annual amplitude temperature is between 1° and 2°C.

The average total rainfall varies between 2,000 and 2,500 mm. The rainy season lasts for 8 months (October to April), with a high hydric balance, ranging from 100 to 1,200 mm. The dry season occurs between June and September (4 months) with a deficit hydric balance of 200 to 250 mm.

Annually, the hydric balance surplus is around 1,197 mm. The monthly distribution of this surplus is at a maximum during the months of December (207 mm), January (266 mm), February (299 mm), and March (197 mm). The months of November and April also present considerable water surpluses, being 129 mm and 90 mm, respectively. The annual water deficit is 244.2 mm, starting in May and continuing until September.

There is a rainfall station close to the X1 Deposit area, in the municipality of Peixoto de Azevedo at the intersection of BR-163 (Estrada Cuiabá-Santarém) with the Peixoto de Azevedo River. The station, under the responsibility of ANA (National Water Agency) and operated by CPRM (Geological Service of Brazil), provides historical rainfall data for the years from 2004 to 2020.

Figures 15-3 and 15-4 present graphs based on this historical data for the annual and monthly averages of rainfall in the rainfall season in the municipality of Peixoto de Azevedo.

The historical data from the pluviometry station are representative of the area when compared to the regional rainfall regime, with average annual volumes of around 2,000 mm of rain. The monthly distribution clearly demonstrates the dry season, between the months of May and September, and the rainy season, between the months of October and April. The months from December to March present, on average, volumes of more than 300 mm of rain, being the period of the year with the highest hydric balance surplus.

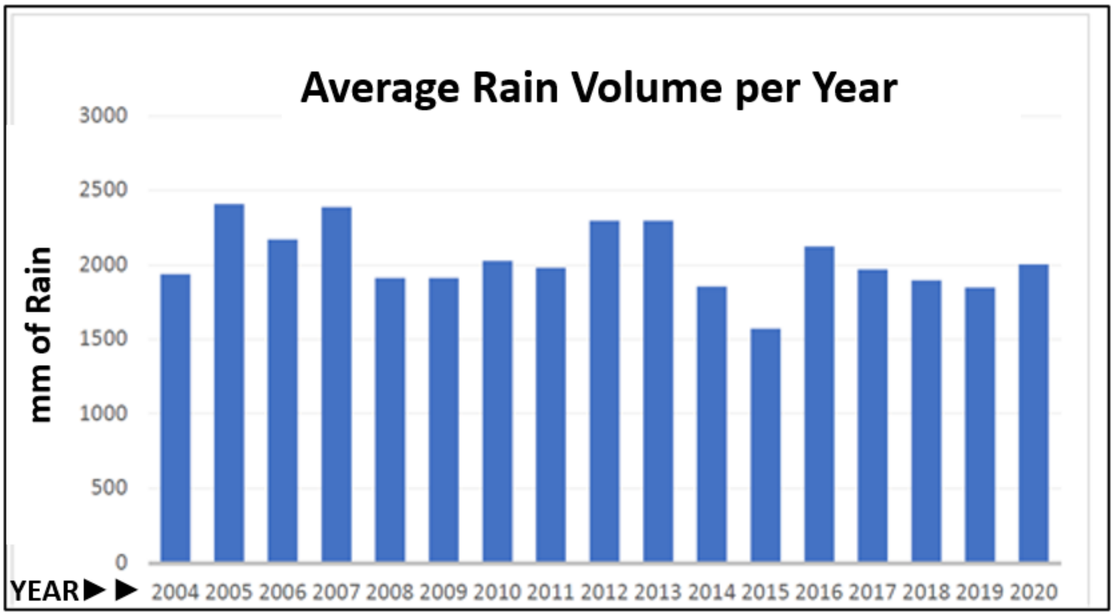


Figure 15-3 - Annual Average of Rainfall Data, Years 2004 to 2020, 105500-pluviometry station, municipality of Peixoto de Azevedo.

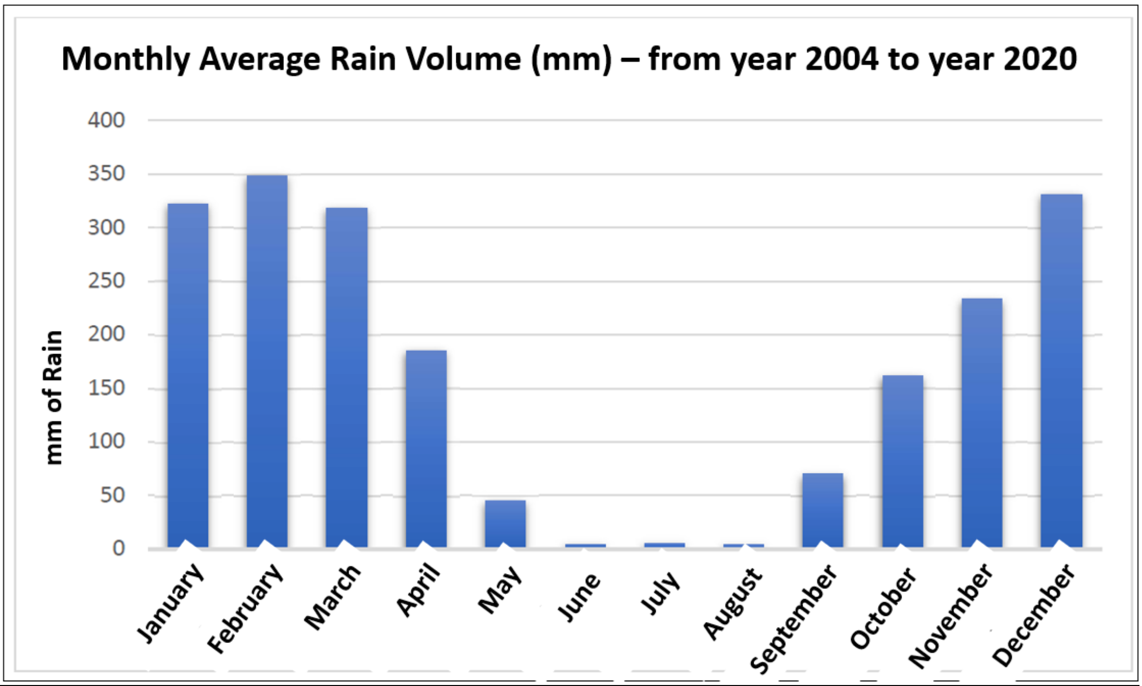


Figure 15-4 - Monthly Average Rainfall Data, years 2004 to 2020, 105500-pluviometry season, municipality of Peixoto de Azevedo.

15.4 MINE PLANNING FOR RESERVE ESTIMATION

The Matupá Project X1 orebody is planned to be mined by the open pit method in three stages. Typically, this type of mining operation uses a combination of 4.5 m^3 hydraulic excavators (backhoe buckets assembled model), front-end loaders, and 35-tonne haul trucks (vocational) as the primary mining equipment.

Based on the mine optimization analysis, ultimate pits were designed to the specific X1 mineral deposit. The recommended mine schedule results in a 1.3 Mt annual average run-of-mine (ROM) production rate at 1.13 g/t Au and 14.7 Mt of waste over the 7 year life of the Project at full capacity.

The Mineral Reserves will be mined from the X1 mineral deposit. The pit optimization was performed using Whittle Lerchs-Grossmann shell analysis. Whittle is a software package that uses Lerchs-Grossmann algorithms to determine the approximate shape of a near optimal pit shell based on applied cut-off grade criteria and pit slopes. These shells are generated from the geologic grade models, economic and physical criteria. For the Whittle analysis, the geological block model includes grade, lithology, rock density, and Mineral Resource classification (Measured, Indicated, and Inferred) information for the X1 Deposit. The pit optimization involved analysis of potential gold cut-off grades.

Based on the mine schedule, capital and operating costs were estimated.

15.4.1 AGREED OPTIMIZATION PARAMETERS

Mining costs were based on a contracted operation. Mining cost adjustment factors were applied to reflect specific requirements for blasting and grade control. A fixed ore haul cost was added to account for specific long-haul distances from the pits to the processing plant.

In addition, mining dilution and mining recovery factors were estimated by EDEM and were applied during the pit optimization and mining planning.

EDEM completed a pit optimization design with a ROM annual production rate of 1.3 Au Mtpa.

The input optimization parameters were provided by Aura and are listed in Table 15-10. Gold prices are in US Dollars and costs are in Brazilian Reals.

Table 15-10 - Pit Optimization Parameters.

Geometric and Economic Parameters for Pit Optimization (for NPV Calculation)					
TYPE	ITEM	DESCRIPTION	UNITS	X1 Deposit	
				ORE	WASTE
		Gold price by oz	USD/oz	1,500.00	n/a
		Gold price by g	USD/g	48.28	n/a
		Au Royalty CFEM	% of Gross	2.15%	2.15%
		Currebcy FX	R$/US$	5.58	5.58
		Discount Rate	Discount Rate (%)	5.0%	5.0%
Physicals	ROM Quality Limits	Bulk density	t/m3	In Model	In Model
		Au Grade	g/t	In Model	n/a
	Mining	Mining recovery	%	98%	n/a
		Mining Dilution	%	3%	n/a
	Block Model	Dimensiion X	m	5.0	n/a
		Dimensiion Y	m	5.00	n/a
		Dimensiion Z	m	5.00	n/a
	Pit Slopes	Soil	degree	32.0	32.0
		Saprolite rock	degree	36.0	36.0
		Fresh Rock	degree	52.0	52.0
	Plant Troughput	Full Capacity	Mtpa	1.30	n/a
	Plant Recovery	Gold Metal	%	95%	n/a
Cost	G & A	Total	US$/t	1.64	n/a
	Plant	Processing	US$/t	15.44	n/a
	Mining	Soil	US$/t	1.35	1.35
		Sprolite Rock	US$/t	1.51	1.43
		Fresh Rock	US$/t	1.65	1.57

Note: USD = US$, R$ Real $

Figure 15-5 shows the pit generated NPV and the best pit alternative, the Net value, that supports the optimal pit selection.

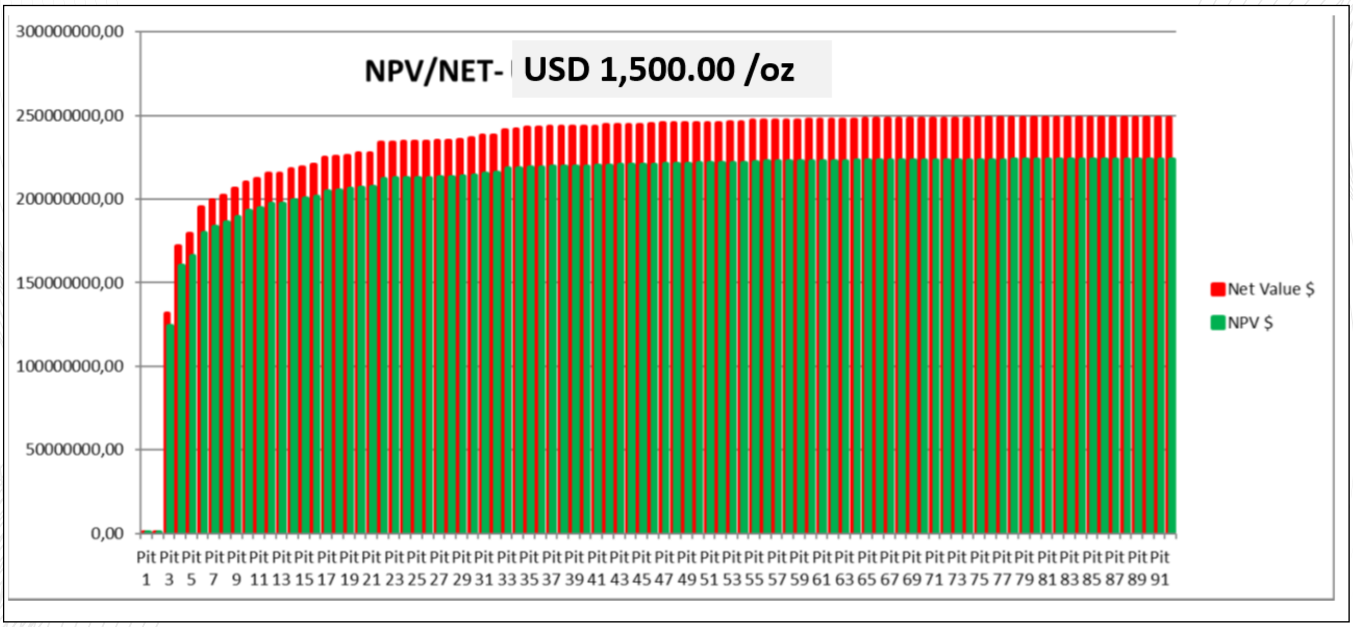


Figure 15-5 - NPV Value Generated to Support the Optimum Pit Selection.

After selecting the final pit, an operationalization was performed according to the top view of the final pit shown in Figure 15-6, and sections A-A and B-B, longitudinal and transversal profiles respectively illustrated in Figures 15-7 and 15-8.

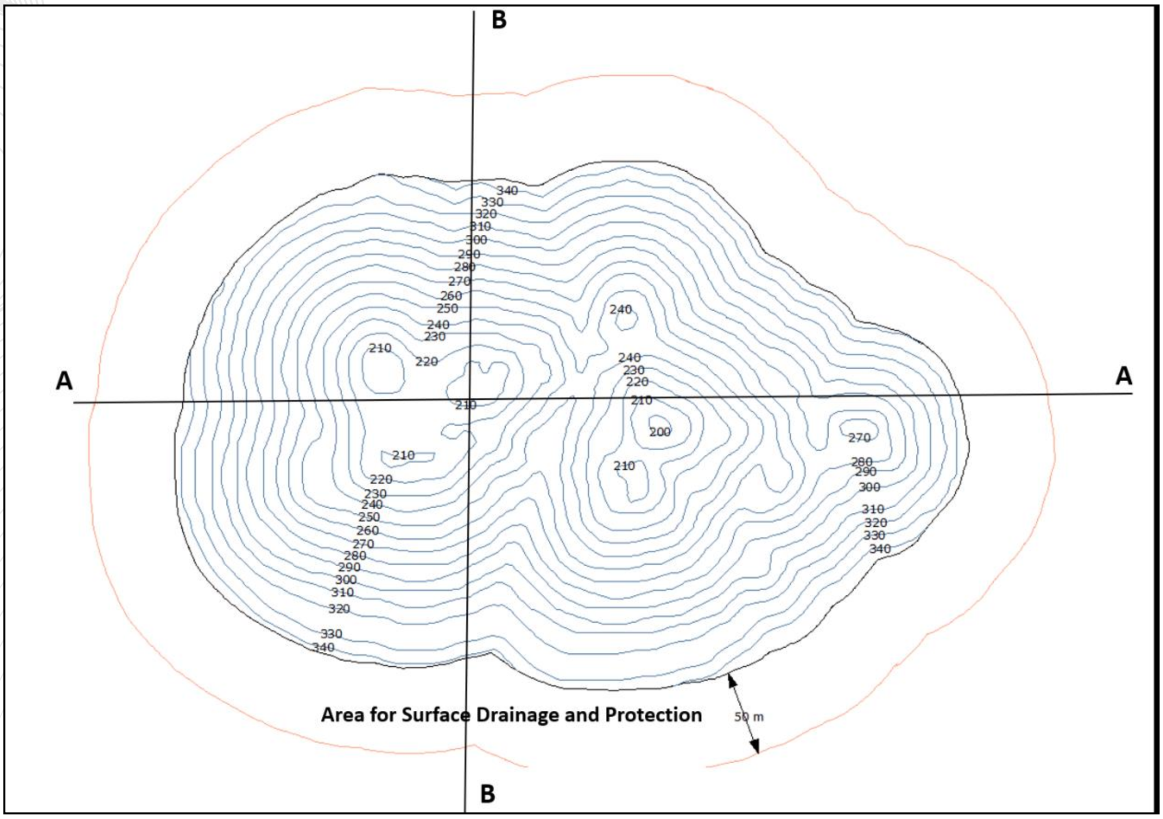


Figure 15-6 - NPV Value Generated from Many Pits to Support the Optimum Pit Selection.

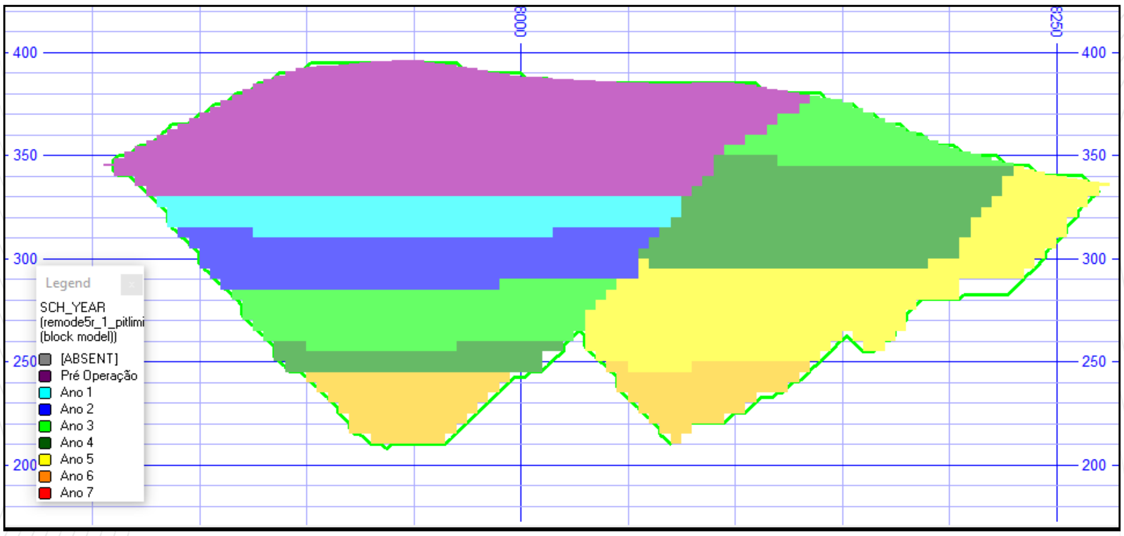


Figure 15-7 - A-A Profile Longitudinal View from Open Pit Design (Figure15-5).

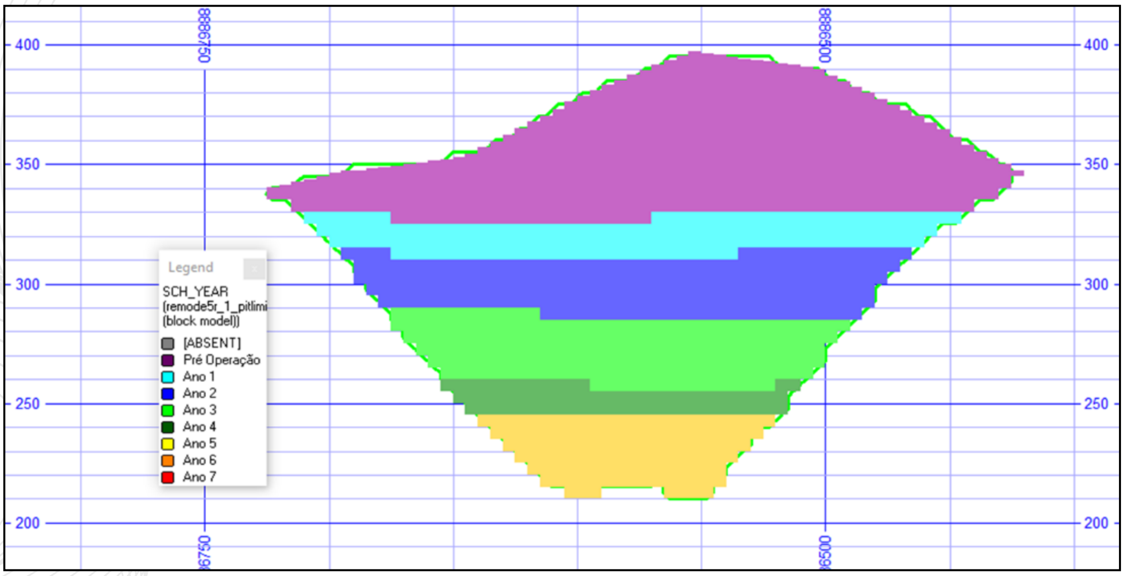


Figure 15-8 - A-A Profile Vertical CrossSection from the Open Pit Design (Figure 15.5).

Scheduling assumptions and constraints were based on a processing plant start-up on September 1st, 2023, and a ramp-up as follows:

- September: 40%
- October: 60%
- November: 80%
- December: 100%

Several geometrical constraints were applied to ensure a practical sequence:

- Maximum vertical advance : 40 meters per year (for each phase).

- Good quality blasting of final walls and intermediate cutbacks will be critical to achieve a good performance, planned dilution and safety.

15.4.2 DILUTION

The grade block model for the X1 Deposit was recalculated using the weighted gold grade which incorporates grade dilution into the block model resulting in a more accurate model. The original model used ore percentiles inside each block.

Since the orebody is massive, it is not necessary to add a dilution factor in the NPV Schedule simulation.

15.4.3 PIT OPTIMIZATION RESULTS

Many pit shells were generated for a range of revenue factors on the gold price. Preliminary cash flows are estimated by the optimizer based on a 5% discount rate and a nominal gold price of US$ 1,500/oz.

Optimization scenarios are automatically generated by Whittle software determining the:

- Best case - based on an increasing pit shell extraction sequence.
- Worst case - following a bench-by-bench mining sequence.
- Specified case - extraction sequence created based upon predefined pushback geometries. This scenario is considered as the closest to the operational and economic reality.

A summary of the optimization results by deposit are described in the following sections below.

The cut-off grade was calculated considering mining dilution, processing costs, metallurgical recovery, metal price and royalties. Processing costs include G&A and reclamation costs.

A range of pit shells were developed to determine the Project sensitivity and the basis for the designed ultimate pit. As the cut-off grades decreased, the rock tonnage increased, that resulted in a reduction of the average grade.

15.4.3.1 FINAL PIT DESIGNS

Detailed designing was performed on the selected ultimate pits and intermediate pushbacks for each deposit including accesses and ramps.

It was agreed by Aura and EDEM that the intermediate phases would be designed using gentler slope angles (45 degree) than the ultimate pits. This aids in pit safety, allowing for drilling, and blasting to be performed to ensure good performance and safety.

The geometric parameters are presented in the Table 15-11.

Table 15-11 - Geometrical Parameters for the Matupá Project.

MINE	LITHOLOGY	BENCH HEIGHT (m)	BENCH WIDTH (m)	BENCH FACE ANGLE (°)	INTER RAMP (toe-to-toe) (°)	OVERALL [1] (toe to crest) (°)
X1	Soil	10	8	45	32	32
	Saprolite	10	8	65	45	36
	Fresh Rock	20	10	75	60	52

Notes:

1. Overall slope angles to be applied for pit optimization.
2. Ramps or safety berms 15-m width must be included to meet the proposed overall slope angles.
3. Double bench operation (20-m bench height) provided the batter and overall slope angles be met.

The agreed assumptions for the pits and mining phases designs are:

- Geometric parameters from Table 15-11.
- Road and ramp width: 15 m.
- Maximum ramp gradient: 10%.
- Minimum distance between phases: 40 m.
- Minimum bottom width: 20 m.

A summary of in pit tonnage and grades by deposit is presented in Table 15-12.

Table 15-12 - Mining Inventories of X1 Deposit

Desciption	Matupá-X1
Total Ore (tx10^3-dry)	8.485
Grade Au (g/t)	1.12
Au Metal Contend (oz)	305.395
Gold Recovered (oz)	293.226
Waste rock (tx10^3-dry)	14.692
Waste/Ore ratio- WO (t:t)	1.73
Total rock moved (tx10^3-dry)	23.177
Life of Mine - LOM (years)	7
Dilution (Planned + Operational)	**6.0%**

15.4.3.2 MINING SCHEDULE

Table 15-11 shows the distribution of total ROM material moved from the open pit mine. As can be seen in the mining sequencing, after year six the concentration plant is planned to be fed almost exclusively from stockpiles.

The Pre-Stripping Operation (Pre-operation (PRE-OP)), that is planned to happen up to June 2023, volumes are presented in Table 15-13, which also shows the ROM origin to be fed to the concentration plant.

Table 15-13 - Concentration Plant Ore Planned to be Fed, by Origin.

Rock Type		Ore			Waste			TOTAL BY LITHOLOGY		
		Fresh Rock	Soil	Saprolite	Fresh Rock	Soil	Saprolite	Fresh Rock	Soil	Saprolite
YEAR	Quarter	kt	kt	kt	Kt	kt	kt	kt	kt	kt
	Pre-Op.	101.4	115.0	204.0	202.7	2,027.9	604.3	304.0	2,142.9	808.3
1	1	234.6	13.4	285.7	66.5	711.0	229.2	301.2	724.4	514.9
	2	292.5	-	16.9	194.5	146.0	256.0	487.0	146.0	272.9
	3	303.0	-	-	241.2	95.3	244.0	544.1	95.3	244.0
	4	305.6	-	-	459.8	10.1	90.1	765.4	10.1	90.1
	Sub- total	1,135.7	13.4	302.6	962.1	962.5	819.3	2,097.7	975.9	1,122.0
2	1	304.1	-	-	503.5	-	44.4	807.6	-	44.4
	2	305.3	-	-	535.7	-	11.6	840.9	-	11.6
	3	310.0	-	-	555.5	-	-	865.6	-	-
	4	322.6	-	-	410.0	28.0	144.6	732.6	28.0	144.6
	Sub- total	1,242.0	-	-	2,004.7	28.0	200.5	3,246.7	28.0	200.5
3	1	315.2	-	-	389.8	30.0	120.0	705.0	30.0	120.0
	2	307.9	-	-	339.5	40.0	100.0	647.4	40.0	100.0
	3	305.3	-	-	338.1	80.0	100.0	643.3	80.0	100.0
	4	292.3	2.6	2.6	304.9	118.3	119.0	597.1	120.8	121.6
	Sub- total	1,220.6	2.6	2.6	1,372.3	268.3	439.0	2,592.9	270.8	441.6
4	1	289.7	-	5.5	396.9	46.9	90.6	686.6	46.9	96.1
	2	305.6	-	20.5	408.7	54.3	74.7	714.2	54.3	95.2
	3	339.3	-	5.3	514.5	93.5	177.9	853.8	93.5	183.2
	4	339.9	-	20.5	648.1	4.6	73.2	988.0	4.6	93.6
	Sub- total	1,274.5	-	51.8	1,968.2	199.3	416.3	3,242.6	199.3	468.1
5	1	318.8	-	-	536.3	-	2.7	855.1	-	2.7
	2	333.5	-	-	530.7	-	-	864.3	-	-
	3	329.1	-	1.3	489.4	-	28.7	818.5	-	30.0
	4	324.6	-	2.3	313.6	-	-	638.2	-	2.3
	Sub- total	1,306.1	-	3.6	1,870.0	-	31.3	3,176.1	-	35.0
6	1	309.2	-	-	78.5	-	-	387.7	-	-
	2	307.2	-	-	44.1	-	-	351.3	-	-
	3	311.2	-	-	54.4	-	-	365.6	-	-
	4	328.9	-	-	107.0	-	-	435.9	-	-
	Sub- total	1,256.6	-	-	284.0	-	-	1,540.6	-	-
7	1	239.2	-	-	31.3	-	-	270.5	-	-
	2	-	-	-	-	-	-	-	-	-
	3	-	-	-	-	-	-	-	-	-
	4	-	-	-	-	-	-	-	-	-
	Sub- total	239.2	-	-	31.3	-	-	270.5	-	-

15.5 X1 MINE PIT DESIGN

The pit optimization of the Matupá Project X1 Deposit is designed to achieve, for each quarter period, approximately, 330,000 t of high-grade ore as shown in Table 15-14. The low-grade mined ore will initially be stockpiled to feed the process plant at the end of the LOM. This procedure allows a more accurate approach to optimize the waste removal, and the pre-strip volumes. The volume calculation, done by quarter, shows ore grades and the waste/ore strip ratio in Table 15-14.

Table 15-14 - X-1- Orebody Pit Optimization Results.

		TOTAL ROCK MOVIMENT						
		All Lithologies	Soil	Saprolite	Fresh Rock	Plant Feeding	Ore	Waste
Year	Quarter	tx10³	tx10³	tx10³	tx10³	tx10³	tx10³	tx10³
	Pre-Op.	3,255	2,143	808	304	-	420	2,835
1	1	1,541	724	515	301	195	534	1,007
	2	906	146	273	487	325	309	597
	3	883	95	244	544	325	303	580
	4	866	10	90	765	325	306	560
	Sub- total	4,196	976	1,122	2,098	1,170	1,452	2,744
2	1	852	-	44	808	325	304	548
	2	853	-	12	841	325	305	547
	3	866	-	-	866	325	310	556
	4	905	28	145	733	325	323	583
	Sub- total	3,475	28	201	3,247	1,300	1,242	2,233
3	1	855	30	120	705	325	315	540
	2	787	40	100	647	325	308	480
	3	823	80	100	643	325	305	518
	4	853	121	135	597	325	311	542
	Sub- total	3,319	271	455	2,593	1,300	1,239	2,080
4	1	849	47	116	687	325	315	534
	2	844	54	76	714	325	307	538
	3	1,151	93	204	854	325	365	786
	4	1,066	5	73	988	325	340	726
	Sub- total	3,910	199	468	3,243	1,300	1,326	2,584
5	1	858	-	3	855	325	319	539
	2	864	-	-	864	325	334	531
	3	851	-	32	818	325	333	518
	4	638	-	-	638	325	325	314
	Sub- total	3,211	-	35	3,176	1,300	1,310	1,901
6	1	388	-	-	388	325	309	78
	2	351	-	-	351	325	307	44
	3	366	-	-	366	325	311	54
	4	436	-	-	436	325	329	107
	Sub- total	1,541	-	-	1,541	1,300	1,257	284
7	1	284	-	-	525	325	239	31
	2	271	-	-	271	325	-	-
	3	-	-	-	-	165	-	-
	4	-	-	-	-	-	-	31
	Sub- total	555	-	-	796	815	239	63

15.6 METAL CONTENTS

The Table 15-15 shows, by year and origin, the gold content in the ore planned to be fed to the plant.

Table 15-15 - Au Contained in Yearly Planned Feed to the Concentration Plant.

YEAR	Ore fed in the plant		Stock (Pre- Operation only)		Waste	REM	Conten Au Fed in the Plant	Au Recoved from Plant
	Kt x 10^3 (dry)	Grade (g/t Au)	t x 10^3 (dry)	Grade (g/t Au)	Kt x 10^3 (seca)	t:t	Au oz	Au oz
Pré Operation	-	-	420	0.93	2,835	6.74	-	-
1	1,170	1.41	-	-	2,744	1.89	53,143	50,466
2	1,300	1.49	-	-	2,233	1.81	62,317	59,273
3	1,300	1.43	-	-	2,080	1.68	59,840	56,854
4	1,300	1.32	-	-	2,584	1.97	55,227	52,525
5	1,300	0.76	-	-	1,901	1.51	31,795	29,890
6	1,300	0.79	-	-	284	0.25	32,912	30,954
7	814	0.39	-	-	31	-	10,160	13,262
Total /Average	8,484	1.12	420	0.93	14,692	1.73	305,395	293,226

Table 15-16 shows the silver values calculated for the mining production plan, planned yearly feed; however, silver is not considered as part of the Mineral Reserve as there is some concern regarding the actual metallurgical recovery amount.

Table 15-16 - Ag Contained in Yearly Planned Feed to the Concentration Plant.

	Alim. Planta	AG	AG
ANO	t	g/t	Kg
1	1.170.000	4,28	5.008
2	1.300.000	4,26	5.538
3	1.300.000	4,26	5.538
4	1.300.000	4,08	5.304
5	1.300.000	3,78	4.914
6	1.300.000	3,78	4.914
7	814.902	3,11	2.534
TOTAL	**8.484.902**	**3,98**	**33.750**

15.7 MINE MAIN INFRASTRUCTURES

The general mine site design plan, Figure 15-9, shows a schematic general layout with the main infrastructure required for the planned mining operation for the Matupá Project. The structures designed to meet the total movement of rock related to the X1 mining operations are: final pit, waste pile, low grade or pile, and accesses.

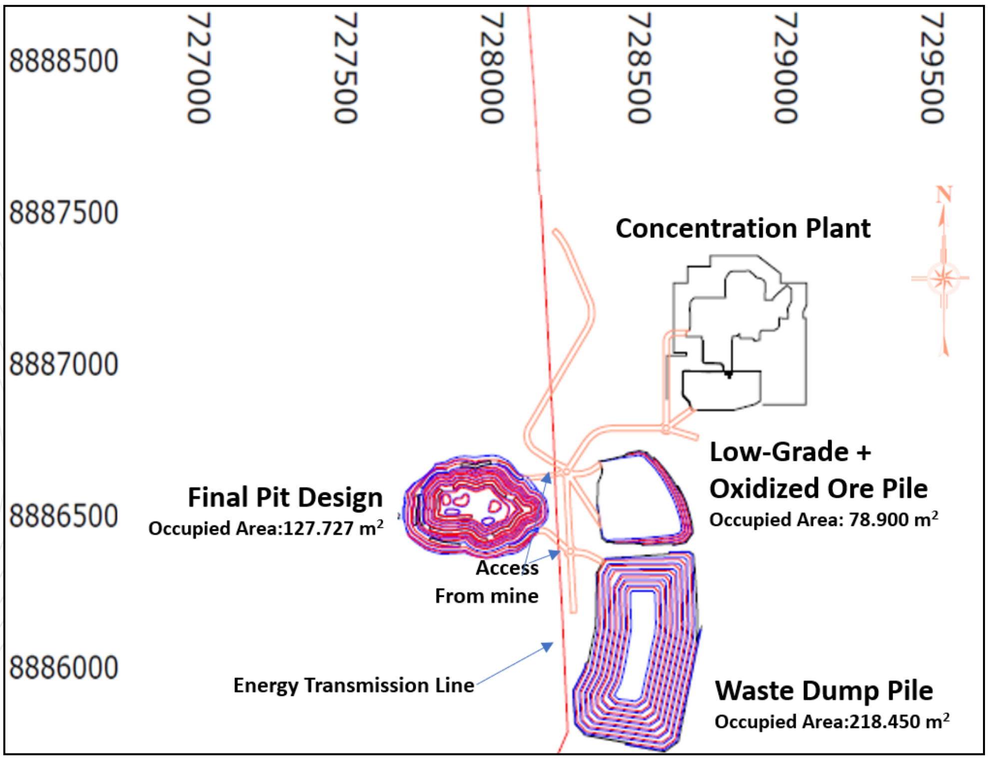


Figure 15-9 - General Layout Including Final Waste Dump Pile, Oxidized Ore and Low-Grade Ore Pile and Final Pit Design.

Mine infrastructure in Figure 15-10, copied from Drawing MTP-B-DS-5010 -GHT-G-0065 by GeoHydroTech (GHT), shows in more detail the final designed figures for: waste dump piles, oxidized ore and Low-Grade ore pile, and mine pit design.

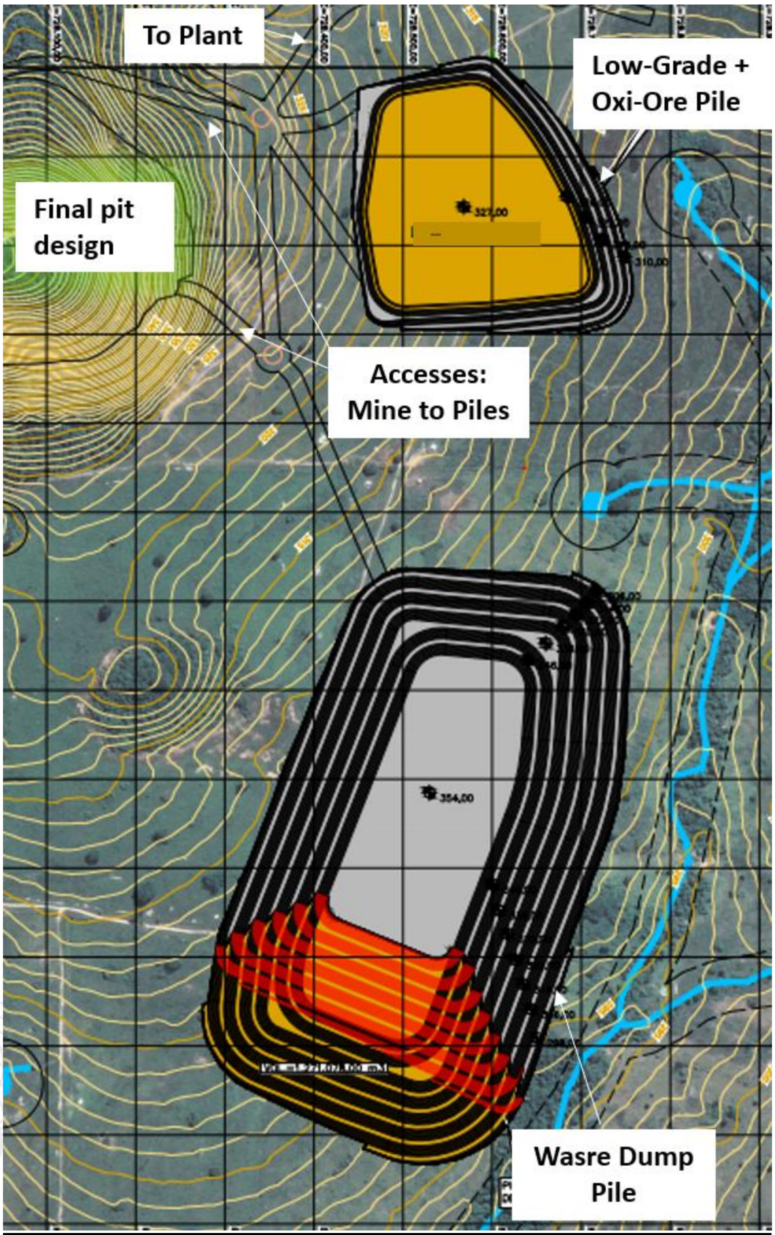


Figure 15-10 - Details of the Final Designed Plans for: Waste Dump Piles, Oxidized Ore and Low-Grade Ore Pile, and Mine Pit Design.

15.7.1 OPERATIONAL PIT DESIGN

The ultimate final pit design and accesses are shown in Figure 15-11, the equivalent 3-D solid is presented in Figure 15-12. The Pre-operation and the first-year operation pits are presented in Figure 15-13 and Figure 15-14.

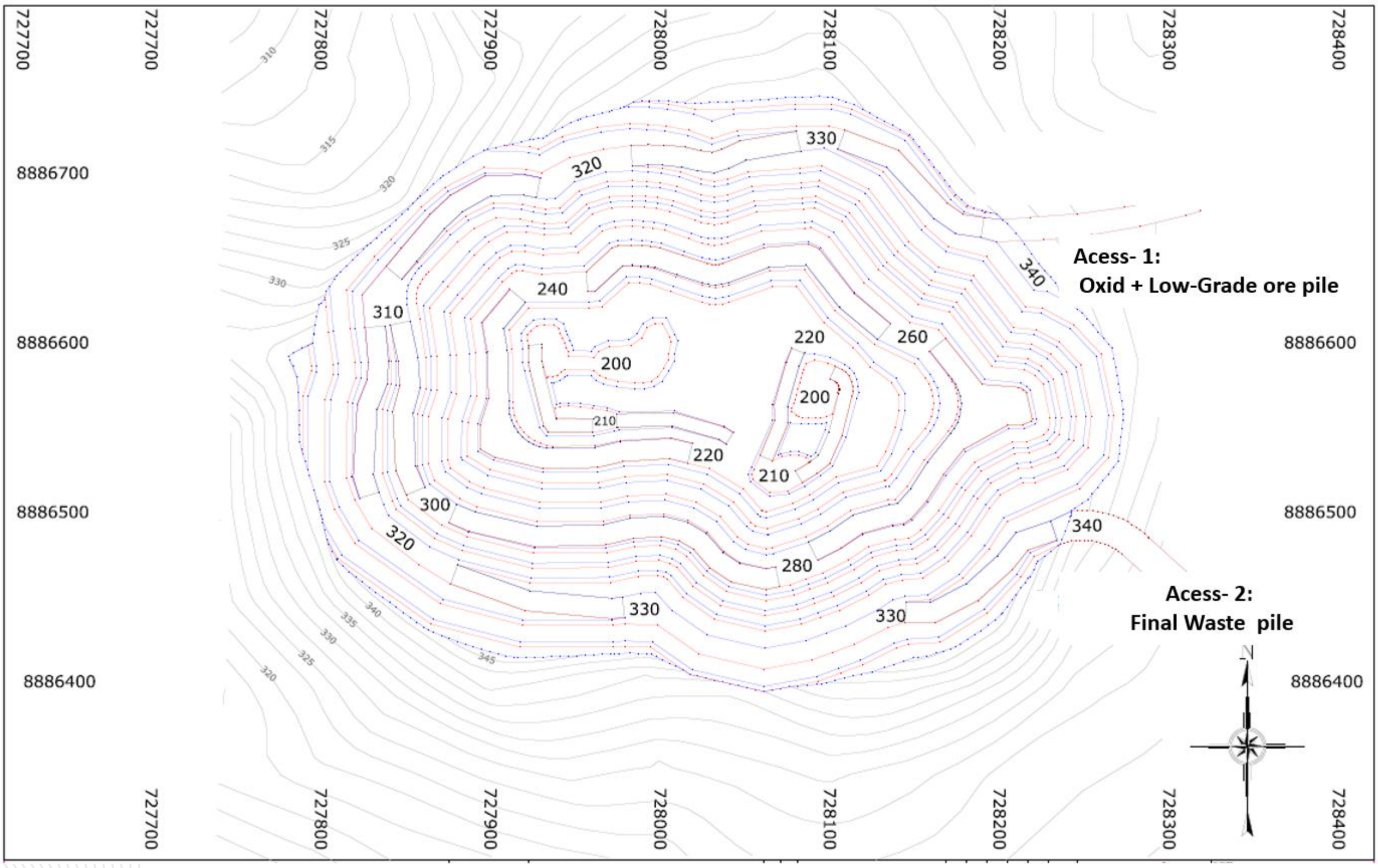


Figure 15-11 - Final Pit Design with Accesses.

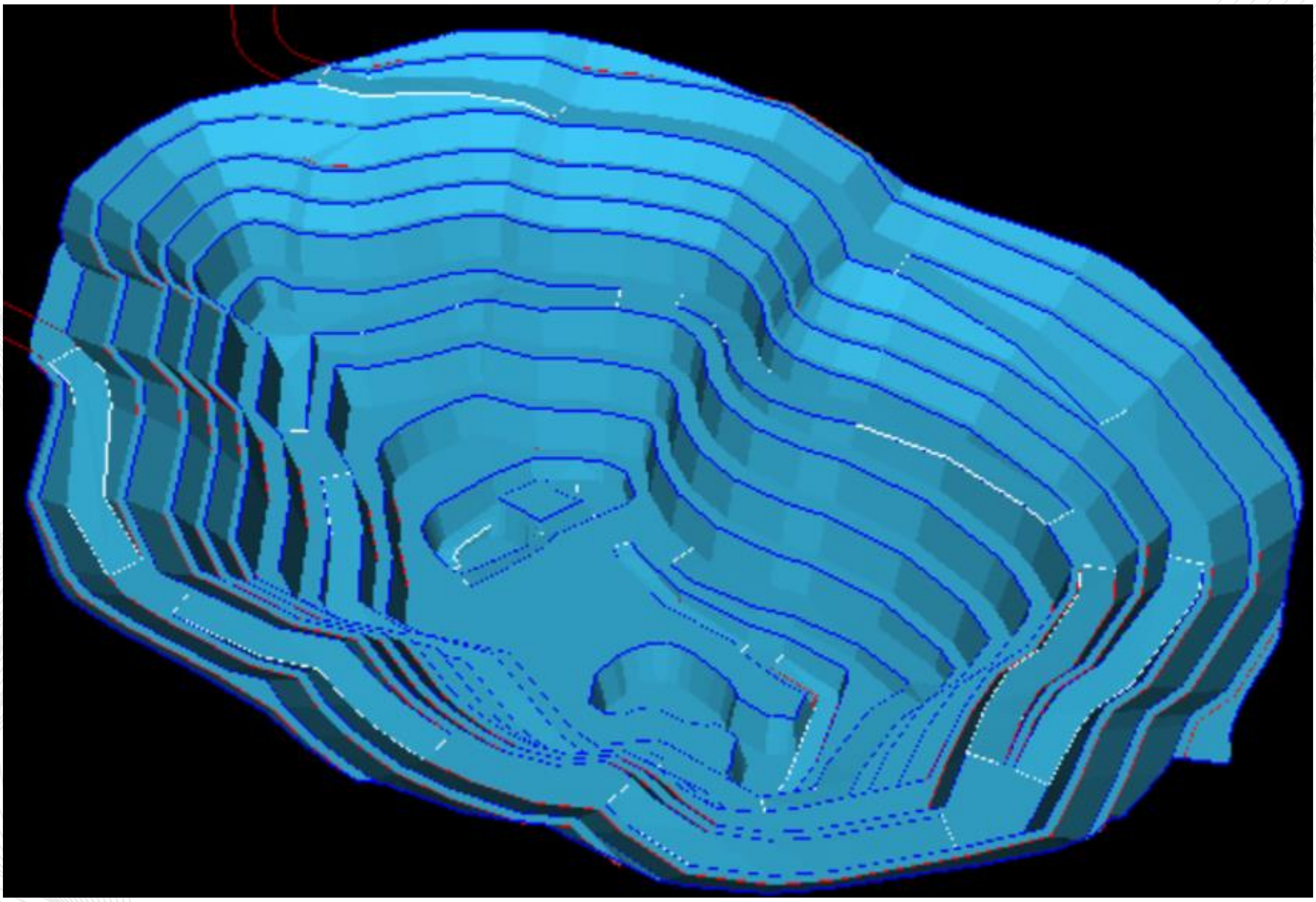


Figure 15-12 - Final Pit Design Solid.

To support the initial mine operation, the Pre-operation pit design was developed for the first- and second-year operations when the low-grade ore will be mined and stockpiled to feed the process plant at the end of the LOM. The Pre-operation design is shown in Figure 15-13 and the related 3-D solid is in Figure 15-14.

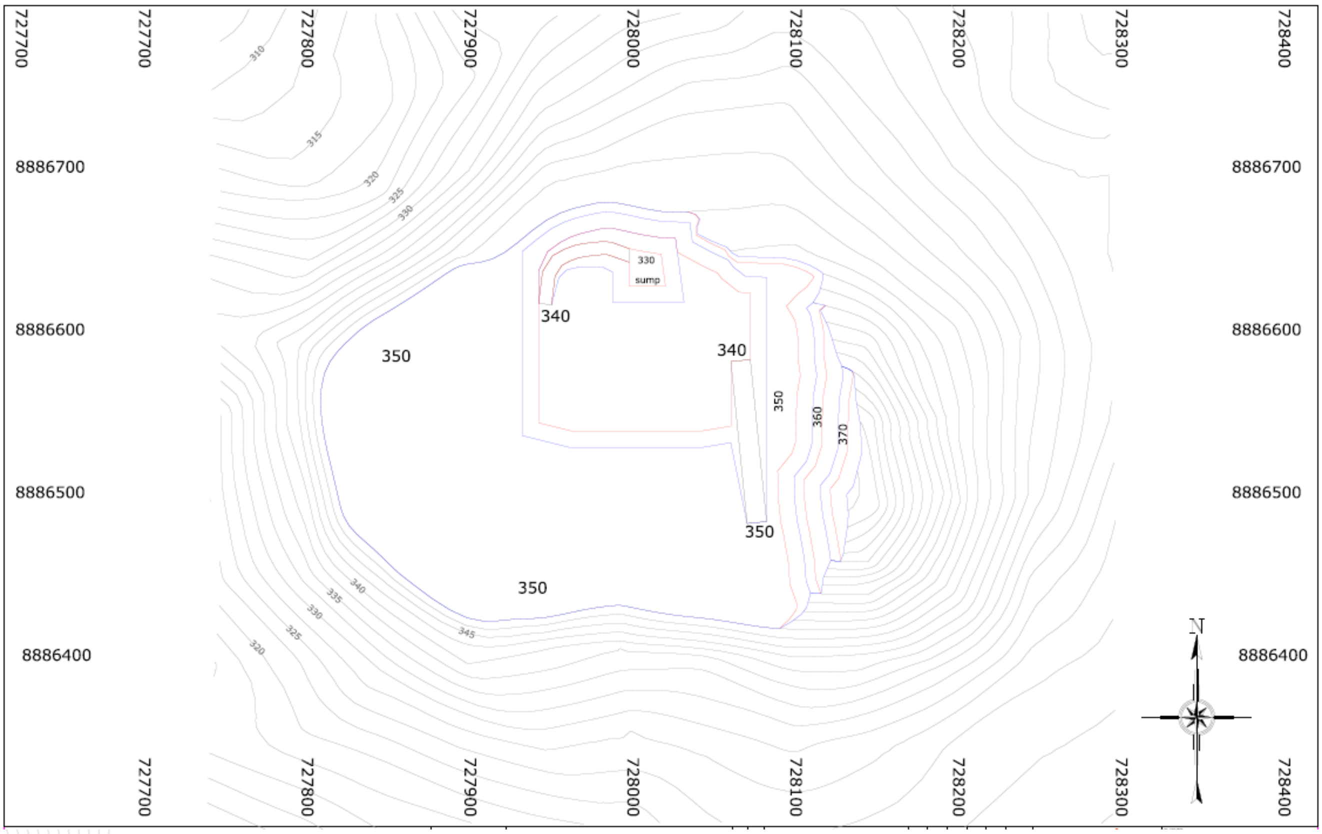


Figure 15-13 - Pre-Operational Pit Design.

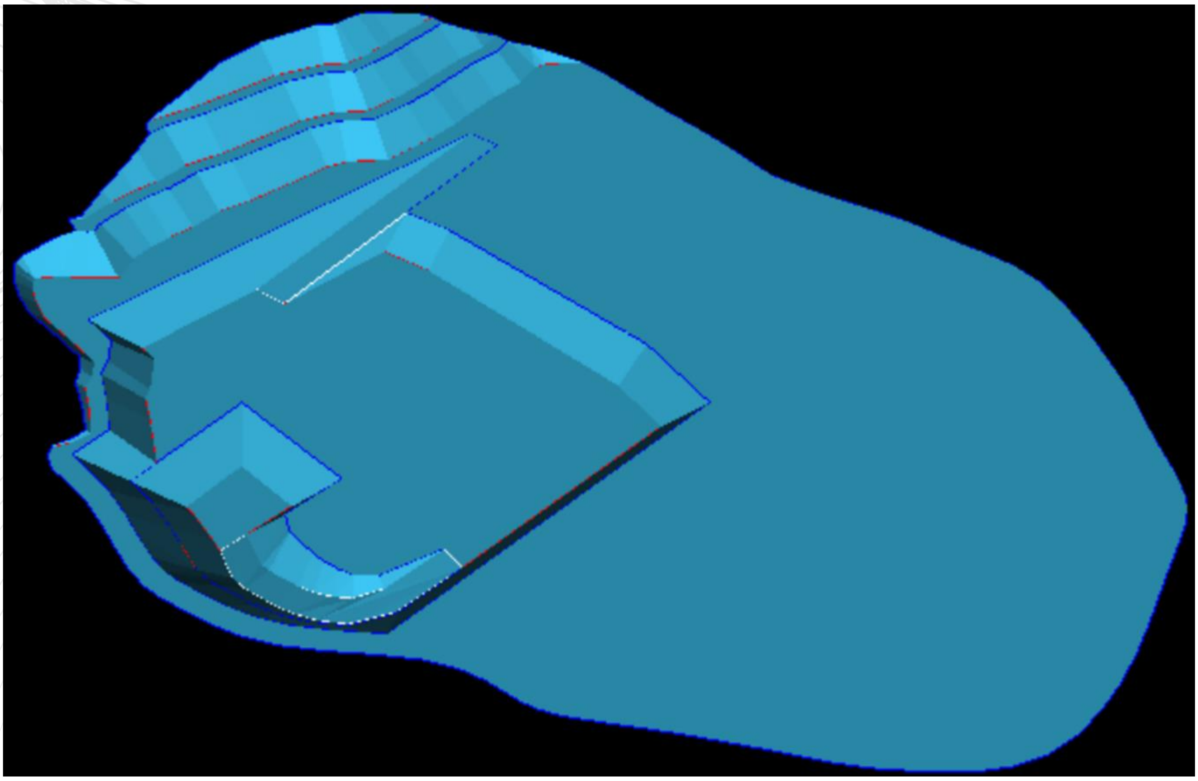


Figure 15-14 - Pre-Operational Pit Design Solid.

Figures 15-15 and 15-16 display the planned pit design for the end of the first year of operation and related 3-D solid drawing.

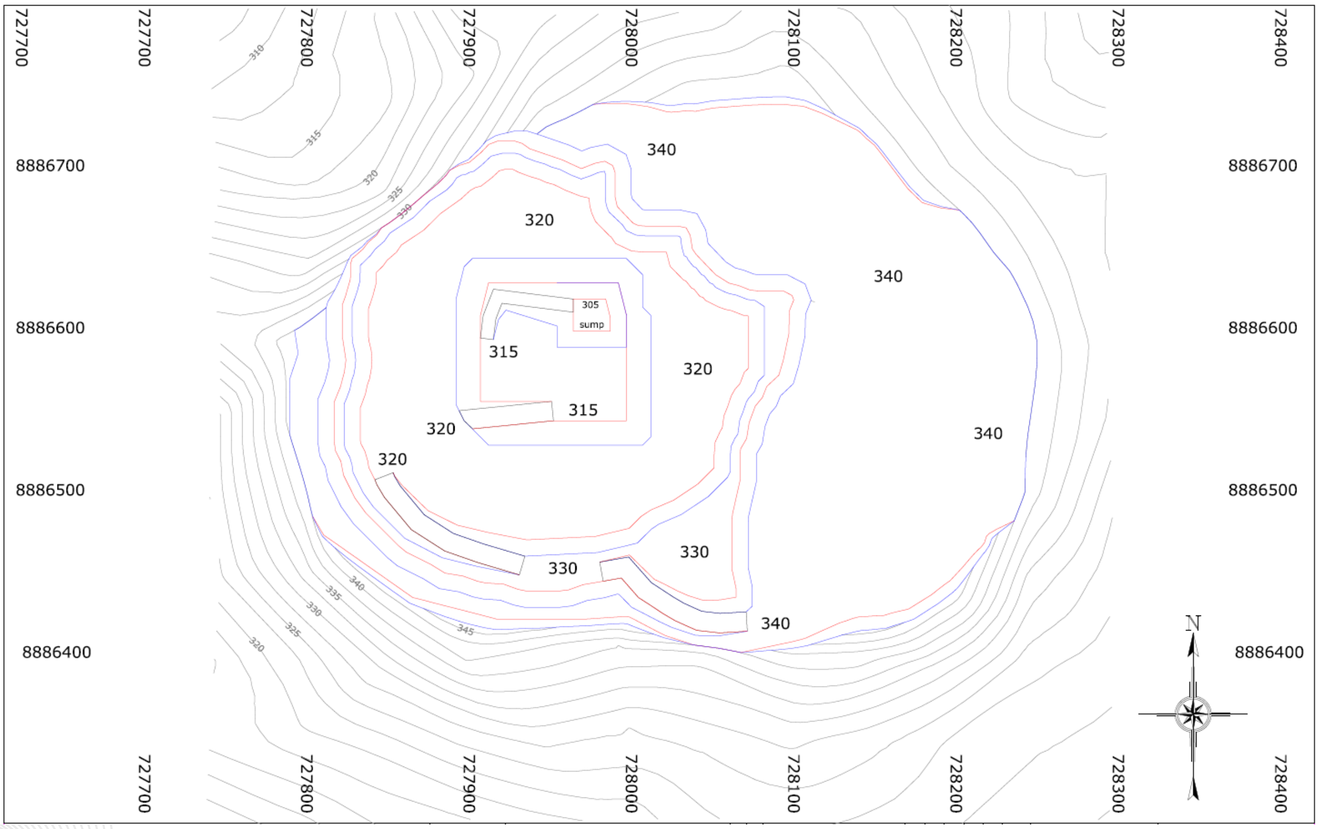


Figure 15-15 - End of First Year Operational Pit Design.

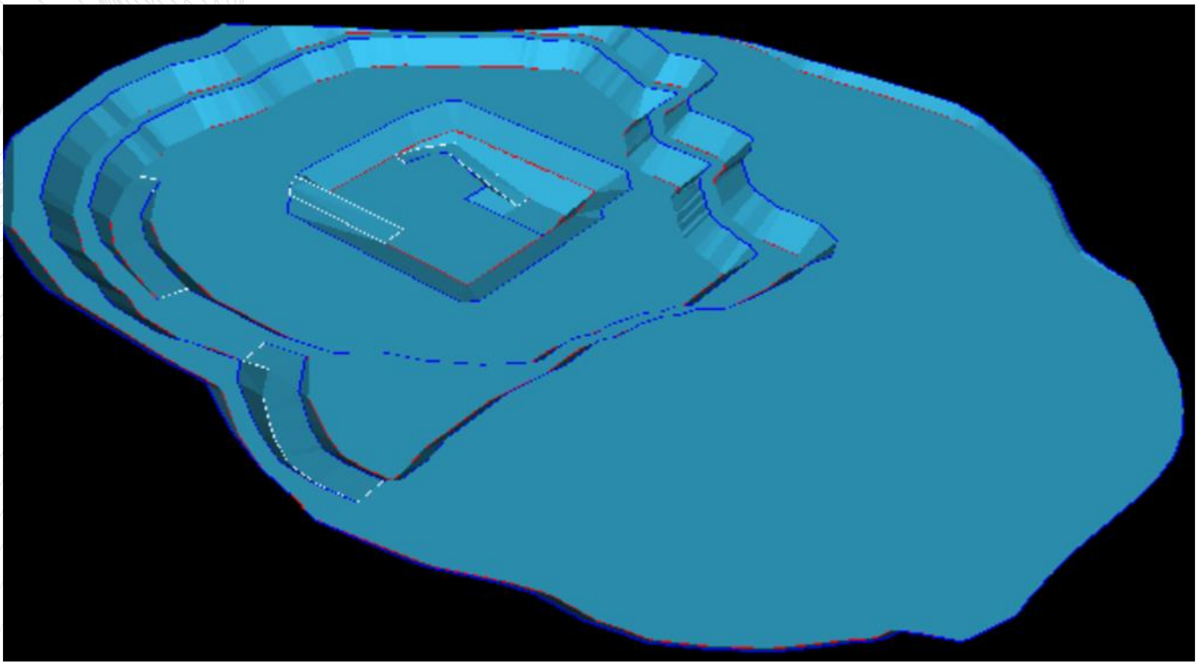


Figure 15-16 - End of First Year Operational Pit Design Solid.

The Figures 15-17 and 15-18 display the planned end of the second year of operations pit design and related 3-D solid drawing.

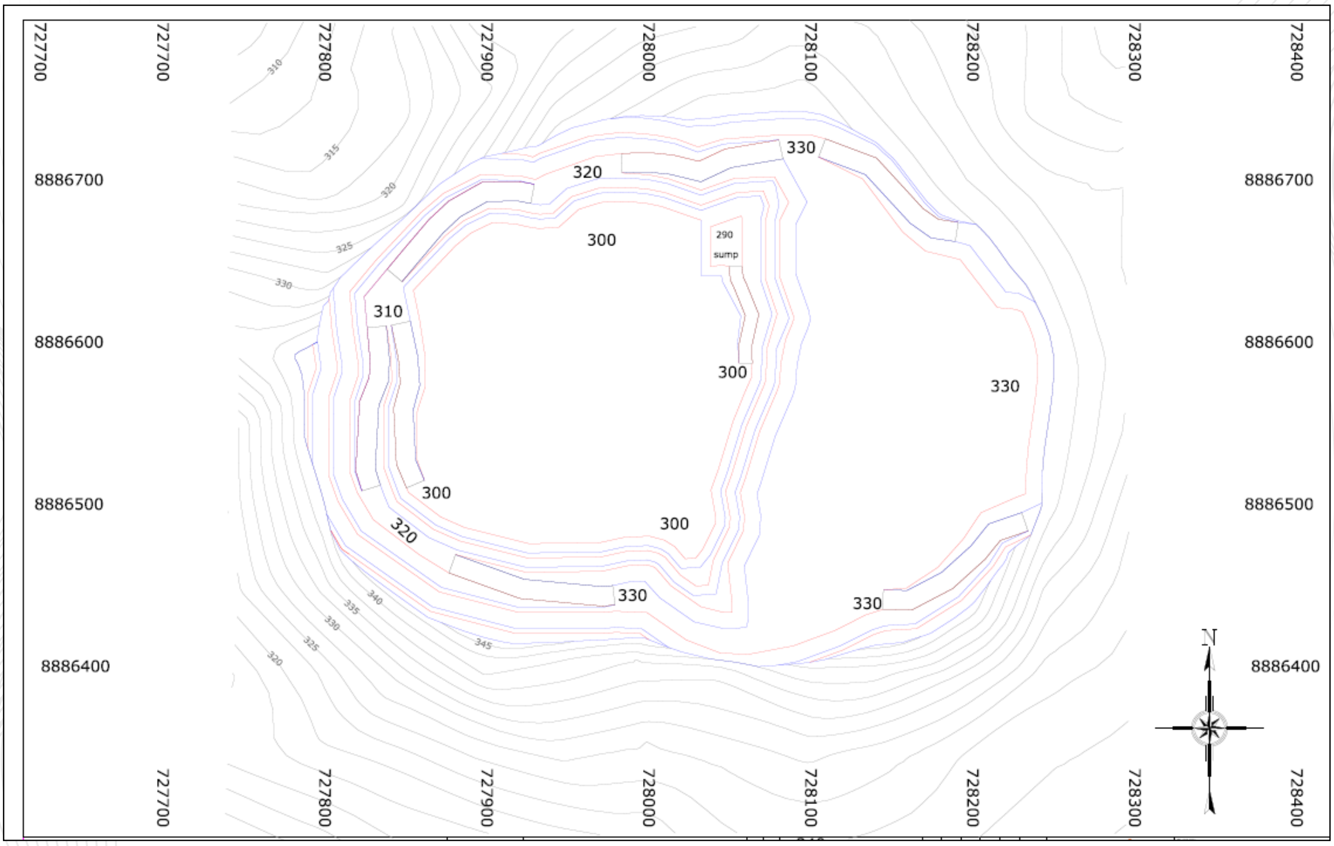


Figure 15-17 - End of Second Year Operational Pit Design.

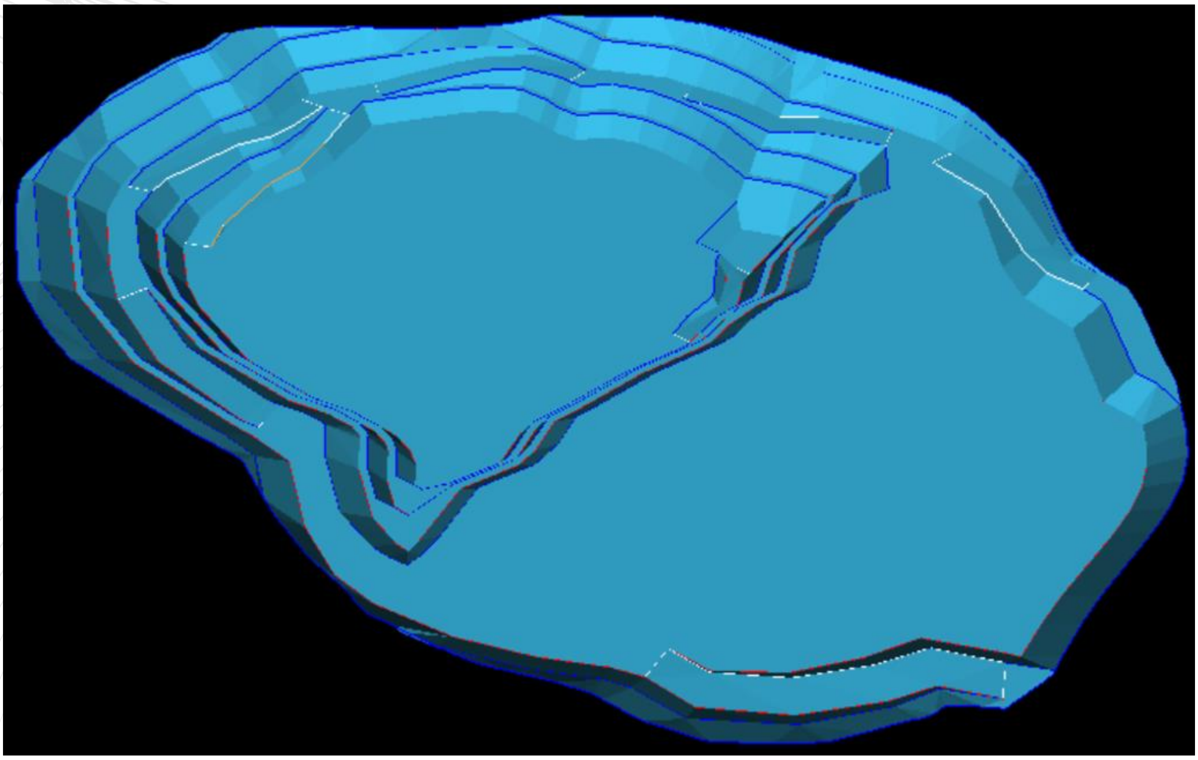


Figure 15-18 - End of Second Year Operational Pit Design Solid.

15.7.2 OXIDIZED ORE AND LOW-GRADE ORE PILE

Figure 15-19 presents the low-grade ore stockpile design and location in the same pile location as the Oxidized ore.

The Oxidized ore extraction is planned primarily for the beginning of the operation and will be stored in the stockpile until use. A limited portion of the Oxidized ore will be part of the process plant feed be used to blend the ore at a planned rate of 25.000 t per quarter or 100.000 t per year.

The Low-Grade ore volume is not representative of the general ore grade and it is planned to be used only in the end of the LOM, or when additional feed is required. The two ore types will be stockpiled in the same stockpile area while kept physically separated in different sides of the pile, as showed in Figure 15-19.

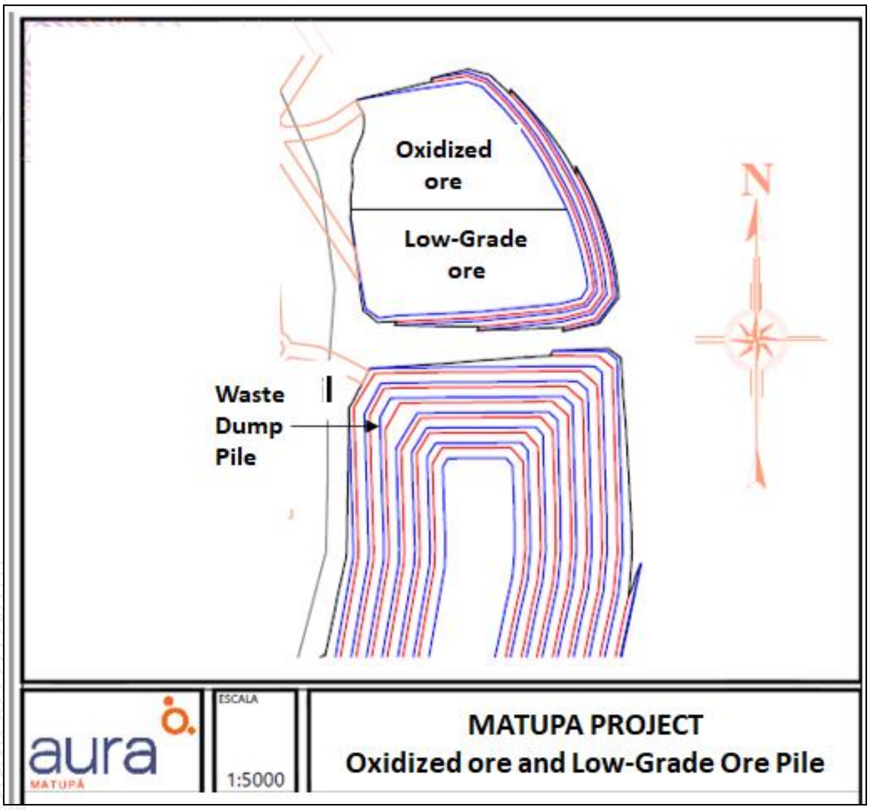


Figure 15-19 - Low-Grade and Oxidized Ores Schematic Stockpile Design.

Figure 15-20, image copied from Drawing MTP-B-DS-5010 -GHT-G-0065 by GeoHydroTech (GHT), presents more details about the shared stockpile area for the oxidized ore and low-grade ore.

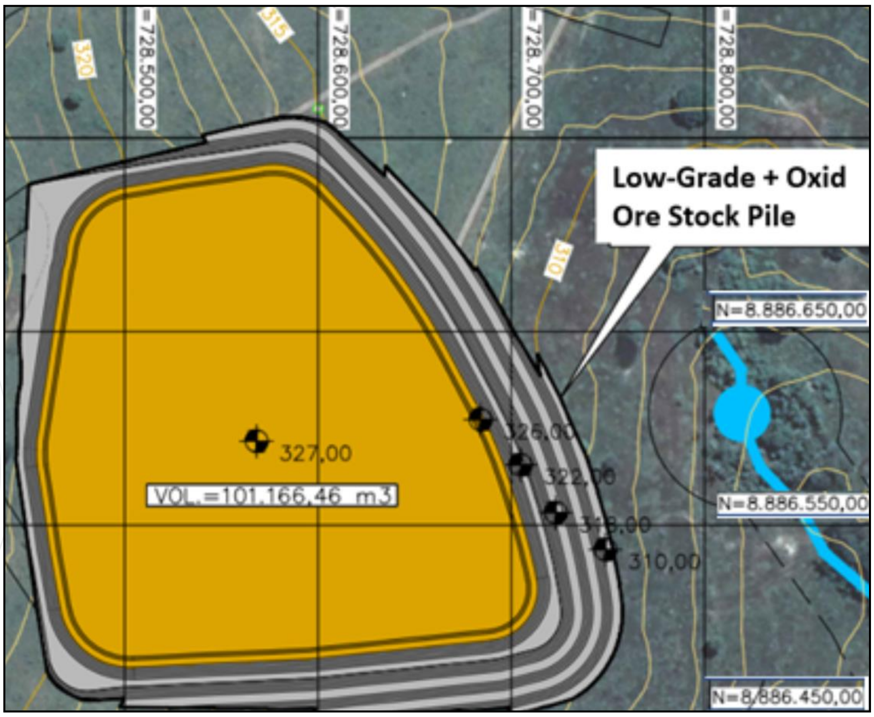


Figure 15-20 - Oxidized Ore + Low-Grade Ore Stockpile Design.

15.7.3 WASTE DUMP PILE

The parameters used for the waste dump design are listed below and are based on the Geotechnical studies supplied by Aura in 2021:

- Bench height: 10 m.
- Bench width: 10 m.
- Bench face angle: 37º.
- Roads and ramps width: 12 m.
- Maximum ramp gradient: 10%.
- Swell factor: 25%.
- Additional volume factor: 5%.

The calculated volumes of the waste dumps are presented in Table 15-15.

Figure 15-21 shows the waste dump pile size and shape related to the operations at the end year one and year four.

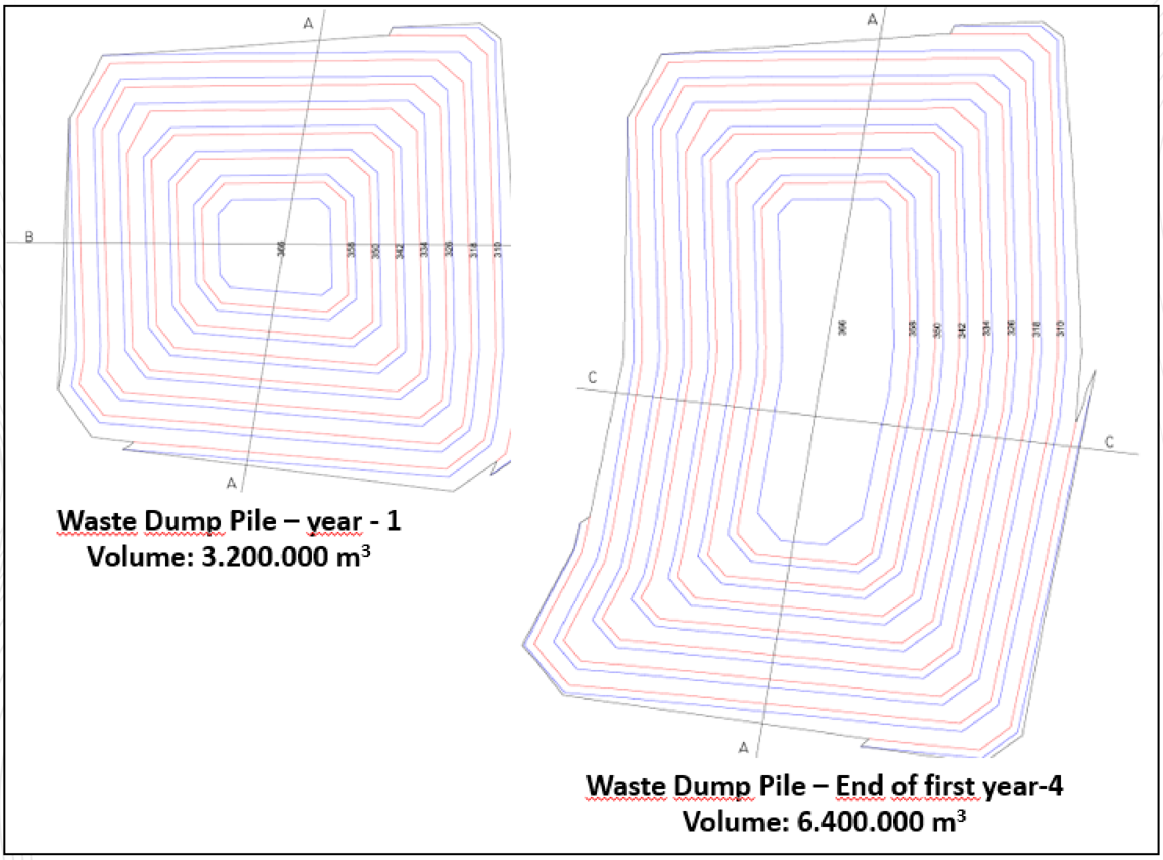


Figure 15-21 - Waste Dump Pile, intermediate dumping phase related to operations year one and year four.

The waste dump pile for the end of the LOM and its total volume capacity is shown in Figure 15-22, image copied from Drawing MTP-B-DS-5010 -GHT-G-0065 by GeoHydroTech (GHT).

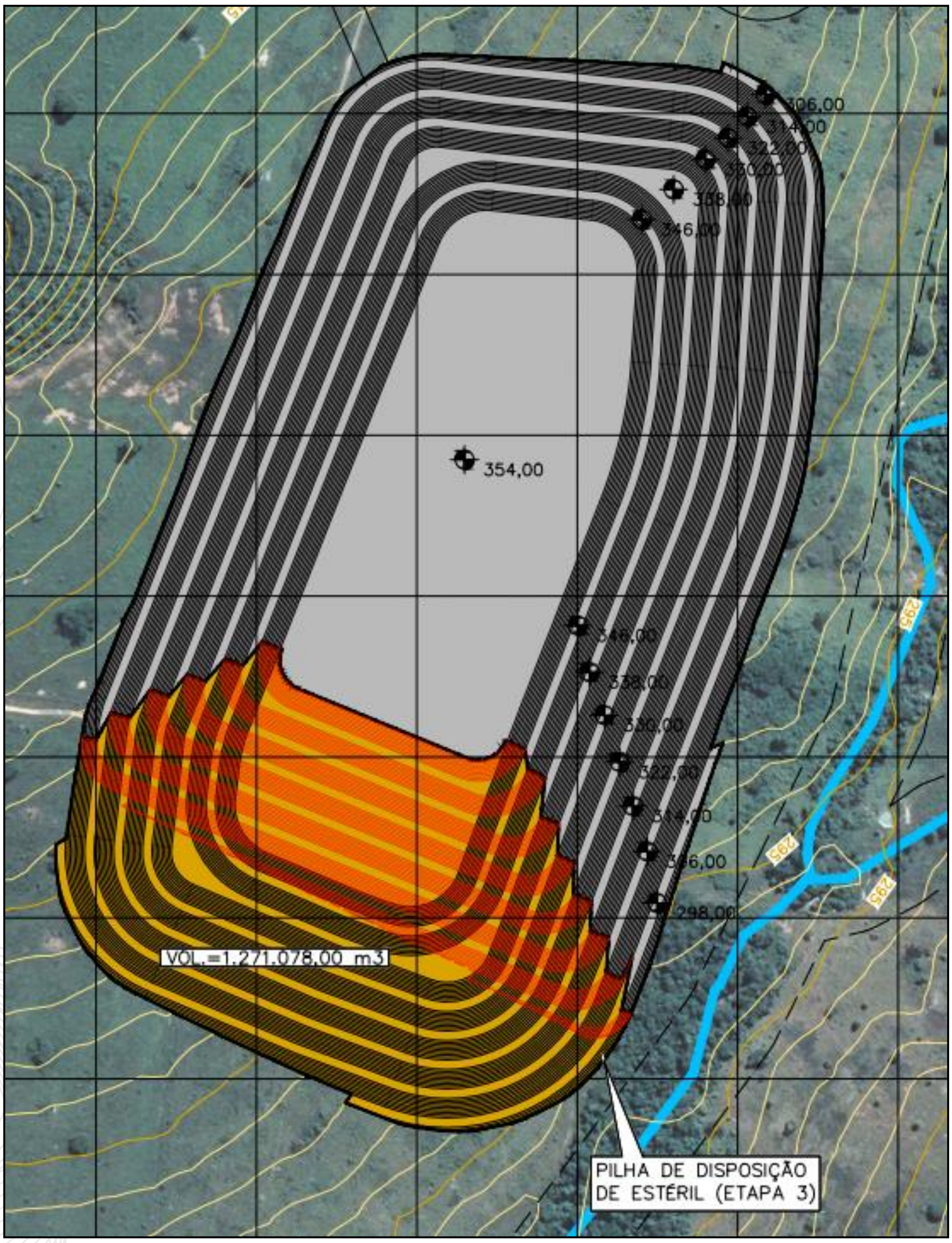


Figure 15-22 - Waste Dump Pile for the End of LOM.

15.8 REMARKS

15.8.1 DRILLING AND BLASTING

The studies have considered the objective of obtaining a blasting size curve according to the top size of 600 mm, P_{80} of 250 mm, and P_{50} of 80 mm.

The analyzes considered the available data from the X1 mineral deposit rock mass and compared this to other similar applications.

Considering a drill diameter of 5.0 inches we estimate a load rate of 410 g/t. Depending on the size of the Crusher, some operational difficulties that could be expected are related to the maximum feed size ("top size"). However, the control of the block size will have a widely favorable procedure since the resumption of the ore from the ore yard that is planned to be done by a loader, an additional operational control point to the crusher feed, once the first top size control is done in mine front with the hydraulic excavators before loading the trucks.

A mined block size larger than 600 mm is estimated to occur about 3% of the time. Fragmentation of these blocks could be accomplished by using a properly selected hydraulic breaker mounted onto a hydraulic excavator; a hydraulic breaker greater than 2,400 kg work in the primary crushing yard is recommended.

The blastholes diameter to fragmentation analysis was based on 5½ inches, in 10-m high benches for waste rock and 5-m high benches for ore. All dimensioning for the Matupá Project uses 5.0-inch blasthole diameters, this builds a safety margin into the calculation. The selected equipment model, and the size of the fleet is designed to meet both possibilities. The explosive charge ratio has been maintained, and therefore the cost considerations remain the same.

The explosives charge ratio considered in this study is 410 g/t, with the use of electronic cap fittings to detonate the explosives for the ore. As the grinding costs are significantly higher than blasting costs, better fragmentation of the blasting rock will benefit processing costs. In the waste rock mine areas, far from the ore/waste contact, the explosives load rate will be 220 g/t with non-electrical accessories. The drilling and blasting unit operations are based on many empirical aspects and can be optimized during operation.

15.8.2 MINERAL RESERVES

Mineral Reserves were calculatedreliably, in accordance with CIM and NI 43-101 regulations, using adequate operational parameters.

The mining operation with a bench height of 10 m and the 5 m high bench for the ore/waste contacts, to improve mining selectivity, appears to be a favorable solution. Waste franc, rock blasting will take place on benches 10 meters high.

Although there is a room to recalculate the cut grade, considering current metal prices and the high dollar exchange rate, this should only increase the Low Grade ore.

Adopting an operational slope of 45 degrees, to define the push backs in the LOM sequencing, guarantees a better performance from the fleet, allowing a greater operational space within the mineralized areas and in the waste franc, far from the ore/waste contact.

15.8.3 COSTS

The costs adopted in the Mineral Reserve Estimate are realistic and provide a robust analysis of the Project's economy. The cost base used is utilized by Apoena, in operation of the Ernesto open pit mine.

The explosives charge rate and the use of electronic caps in the ore and waste rock blasting for the Matupá Project, will have low cost increases and the benefits will extend to all other unit operations of the metal concentration processing. In addition, the proposed rock blasting will help to minimize the mass vibrations in the process plant, improve the ore selectivity, and ore mining recovery. Thus, it is EDEM's opinion that the additional costs in blasting to assure obtaining a P_{80} <6.0 inches are not significant compared to the benefits that can be obtained in cost reductions to the ROM loading, transport, crushing and grinding.

16. MINING METHODS

Mining costs are based on the specific rock requirements for each lithology including drilling, blasting, loading, transport to crushing patio and specific piles; also including infrastructure required for mining production.

The tonnages planned to be mined and the basis for all costs calculation are presented as follow.

A third metallurgical campaign was carried out based on the progression of the Life of Mine (LOM) plan as well as the engineering project. This campaign included the following two parts: consolidation and variability.

16.1 MINING MOVED ROCK

The Table 16-1 shows the planned gold metal recovery from the processing plant, yearly ore tonnages and their respective average gold grade, and the total waste to be removed and dumped in the stockpile.

Table 16-1 - Yearly X1's Pit Volumes: ore tonnage feed to the concentration plant with respective gold grade, gold metal recovered in the plant, and total tonnage moved and to the dump piles.

YEAR	Ore fed in the plant		Stock (Pre- Operation only)		Waste	REM	Conten Au Fed in the Plant	Au Recoved from Plant
	Kt x 10^3 (dry)	Grade (g/t Au)	t x 10^3 (dry)	Grade (g/t Au)	Kt x 10^3 (seca)	t:t	Au oz	Au oz
Pré Operation	-	-	420	0.93	2,835	6.74	-	-
1	1,170	1.41	-	-	2,744	1.89	53,143	50,466
2	1,300	1.49	-	-	2,233	1.81	62,317	59,273
3	1,300	1.43	-	-	2,080	1.68	59,840	56,854
4	1,300	1.32	-	-	2,584	1.97	55,227	52,525
5	1,300	0.76	-	-	1,901	1.51	31,795	29,890
6	1,300	0.79	-	-	284	0.25	32,912	30,954
7	814	0.39	-	-	31	-	10,160	13,262
Total /Average	8,484	1.12	420	0.93	14,692	1.73	305,395	293,226

16.1.1 MINING VOLUMES MOVED FROM THE MINE

The mining operation feed to the concentration plant by lithology for the Matupá Project X1 Deposit is presented in the Table 16-2.

Table 16-2 - Quarterly Planned Ore from X1's Pit by Lithologyfor Concentration Plant Feed.

		TOTAL ROCK TO BE FED IN PLANT				
		All Lithologies		High-Grade Ore		Low-Grade Ore
Year	Quarter	Total	Teor Médio	Rocha Sã	Solo + Saprolito	Todas Litologias
		kt	g/t	kt	kt	kt
1	1	195	1.21	175.5	19.5	-
	2	325	1.38	300.3	24.7	-
	3	325	1.32	299.9	25.1	-
	4	325	1.66	300.1	24.9	-
	Sub- total	1,170	1.39	1,076	94.3	-
2	1	325	1.70	300.0	25.0	-
	2	325	1.54	299.8	25.2	-
	3	325	1.67	300.1	24.9	-
	4	325	1.05	300.1	24.9	-
	Sub- total	1,300	1.49	1,200	100.1	-
3	1	325	1.22	299.9	25.1	-
	2	325	1.30	300.1	24.9	-
	3	325	1.45	299.9	25.1	-
	4	325	1.76	300.5	24.5	-
	Sub- total	1,300	1.43	1,200	99.5	-
4	1	325	2.10	300.6	24.4	-
	2	325	1.49	300.0	25.0	-
	3	325	0.95	299.7	25.3	-
	4	325	0.75	300.0	25.0	-
	Sub- total	1,300	1.32	1,200	99.8	-
5	1	325	0.84	300.0	25.0	-
	2	325	0.67	299.8	25.2	-
	3	325	0.82	297.9	27.1	-
	4	325	0.71	300.0	25.0	-
	Sub- total	1,300	0.76	1,198	102.4	-
6	1	325	0.68	299.9	25.1	-
	2	325	0.88	300.0	25.0	-
	3	325	0.84	299.9	25.1	-
	4	325	0.76	300.1	24.9	-
	Sub- total	1,300	0.79	1,200	100.1	-
7	1	325	0.70	302.8	22.2	-
	2	325	0.45	100.0	34.2	190.8
	3	265	0.40	-	-	264.9
	4	-	-	-	-	-
	Sub- total	915	0.39	403	56.4	456

16.1.2 RUN OF MINE (ROM) DESTINATION

The mined ROM destinations are:

- High-Grade ore stockpile to crushing plant area: will be taken up by a Frontal End Loader to feed the crushing plant. Alternatively, a high-grade ore volume is stored in the same area to be fed later in case a higher grade is needed to improve the gold production in the concentration plant.

- Low-grade ore stockpile: following a strategy to maximize the net present value, the low-grade value will be assigned to the low-grade ore stockpile located close to the concentration plant. The low-grade ore will be taken to feed the processing plant at the end of the LOM.

- Waste dump piles: will be located close to pit it was extracted from and will be treated as a part of the environment reclamation at the end of LOM.

The ROM by origin, its destination, and Average Transport Distance (“ATD”) are shown in Table 16-3.

Table 16-3 - Quarterly ROM Volumes Mined in the Pit, Destination and Average Transport Distance by Lithology.

	Origin ►	Mine						Ore Patio		Oxidized Ore Pile		Low-Grade Ore	
	Destin ►	Ore Patio/Piles						Crushing Plant		Crushing Plant		Crushing Plant	
	Ore/Waste ►	High-Grade Ore		Low-Grade Ore		Waste		High-Grade Ore		Low-Grade Ore		Low-Grade Ore	
YEAR ▼	Quarter▼	Tonnage	ATD	Tonnage	ATD	Tonnage	ATD	Tonnage	ATD	Tonnage	ATD	Tonnage	ATD
		kt	m	kt	m	kt	m	kt	m	kt	m	kt	m
	Pre-Oper,	400	1,373	20	1,505	2,835	1,335		-	-	-	-	-
1	1	500	1,283	34	1,649	1,007	1,431	195	30	20	800		-
	2	300	1,283	9	1,649	597	1,431	325	30	25	800		-
	3	300	1,283	3	1,649	580	1,431	325	30	25	800		-
	4	300	1,283	5	1,649	560	1,431	325	30	25	800		-
	Sub- total	1,400	1,283	52	1,649	2,744	1,431	1,170	30	94	800		-
2	1	300	1,392	4	1,780	548	1,491	325	30	25	800		-
	2	300	1,392	6	1,780	547	1,491	325	30	25	800		-
	3	300	1,392	10	1,780	556	1,491	325	30	25	800		-
	4	300	1,392	23	1,780	583	1,491	325	30	25	800		-
	Sub- total	1,200	1,392	42	1,780	2,233	1,491	1,300	30	100	800		-
3	1	300	1,731	15	2,084	540	1,890	325	30	25	800		-
	2	300	1,731	8	2,084	480	1,890	325	30	25	800		-
	3	300	1,731	5	2,084	518	1,890	325	30	25	800		-
	4	287	1,731	11	2,084	542	1,890	325	30	24	800		-
	Sub- total	1,187	1,731	39	2,084	2,080	1,890	1,300	30	100	800		-
4	1	281	1,301	15	1,476	534	1,291	325	30	24	800		-
	2	319	1,301	7	1,476	538	1,291	325	30	25	800		-
	3	280	1,301	65	1,476	786	1,291	325	30	25	800		-
	4	320	1,301	40	1,476	726	1,291	325	30	25	800		-
	Sub- total	1,200	1,301	126	1,476	2,584	1,291	1,300	30	100	800		-
5	1	300	1,335	19	1,658	539	1,472	325	30	25	800		-
	2	300	1,335	34	1,658	531	1,472	325	30	25	800		-
	3	298	1,335	33	1,658	518	1,472	325	30	27	800		-
	4	302	1,335	25	1,658	314	1,472	325	30	25	800		-
	Sub- total	1,200	1,335	110	1,658	1,901	1,472	1,300	30	102	800		-
6	1	300	1,339	9	1,732	78	1,475	325	30	25	800		-
	2	300	1,339	7	1,732	44	1,475	325	30	25	800	-	-
	3	300	1,339	11	1,732	54	1,475	325	30	25	800	-	-
	4	300	1,339	29	1,732	107	1,475	325	30	25	800	-	-
	Sub- total	1,200	1,339	57	1,732	284	1,475	1,300	30	100		-	-
7	1	229	1,890	10	2,356	31	2,556	325	30	22	800	-	-
	2	-	-	-	-	-	-	325	30	34	800	190.8	900
	3	-	-	-	-	-	-	265	30	-		264.9	900
	4	-	-	-	-	-	-	-	-	-		-	-
	Sub- total	229	1,890	10	2,356	31	2,556	915	30			455.7	900

16.2 OPERATIONAL PRODUCTION MINING

The mining operation concept for the Matupá Project is conventional open pit mining with a production schedule that provides an initial 0.4 Mt stock of ROM with 0.93 g/t Au content, equivalent to three months production. Commercial operation is scheduled to start up in September 2023 with a ramp up until December 2023. For commissioning purposes the Low-Grade ore will also be available as process plant feed.

The mine development is planned to allow access to those grade levels required to maximize gold production and provide operational flexibility by mining several benches simultaneously.

The waste rock comprises soil, saprolite, altered rock mass, and fresh rock. The planned excavation of these deposits is to drill and blast, with explosives, all fresh rock and 30% of saprolite. Load and haulage will be performed mainly by hydraulic excavators, backhoe configuration, a complement of front-end loaders, and on-road transportation by a fleet of trucks (vocational).

Benches will be configured as follows:

- A minimum mining width of 30 m on a 10 m-high bench has been used. The X1 pit will include a final bench access incorporating an operational mining width of 15 m to maximize access to the mineralized zone.
- The waste and ore benches will be mined as 5 m thick layers, leaving a designed 10 m maximum bench height.
- The ore and waste zones have been analyzed and it is possible to operate with a proper berm width and in-pit dumping operational space.
- The benches will have a slight decline from crest to the toe for the upper bench face slope, in the direction of the open side to drain rainfall and to maintain designed slope angles. A good drainage design inside the pit and in rainwater collection contribution areas around the pit, allow for the minimization of operational disturbances during heavy rain.

The processing plant is located about 1.0 km from the X1 pit.

The mining faces will be accessed by 15-m wide double lane roads with a 10% gradient. All roads will have 2.0 cm/m transversal gradient, from the center to the lateral edge of the road, with drainage ditches along the roads. Road conditions must be compatible with good practices for the safe operation of mining equipment.

The Matupá Project's gold mining concept is based on the application of conventional techniques for surface rock mass excavation with a maximum level of mechanization:

- Grade control with dedicated drilling: sample collecting to provide good support to the grade control models and short-term mining plan. The technology being considered is Down the Hole hammer with reverse circulation.
- Blastholes: the holes are probably going to be drilled by a hydraulic Top Hammer drilling rig.
- Primary rock blasting: most of the rock, ore, and waste, will be fragmented by explosives. The ore fragmentation has special requirements, specifically the ore will require the use of electronic caps.
- Rock mechanical excavation: must be performed by bulldozers or directly by hydraulic excavators.
- Loading operation: will be carried out, preferentially, by retro bucket profile hydraulic excavator, and complemented by front end loaders ("FEL").
- Rock transport: will be by conventional on-road trucks, 8 x 4 PBT (Total Gross Weight) bigger than 88 tonnes.
- Mine development and preparation: will be undertaken by bulldozers, motor grader, road roller, water tank truck. As important as the production equipment are the ancillary equipment for the preparation and development of the mine: crawler tractor, motor graders, water tank trucks. Without proper mine development / preparation the production, and/or costs/t, would increase significantly.
- Bulldozer selection: must consider activities requuired for blasted rocks fronts and spreading blasted rock in waste dump piles. This requires tractors of greater weight than soil-cutting tractors.
- Soft rock: will be excavated and loaded directly onto the trucks by hydraulic back-hoe excavators. Where the layers of altered rock are thicker, track dozers can be used to complement the excavation work.

The destinations of mined materials are:

- ROM stockpile at the primary crusher area.

- Waste dump pile.
- Low-grade stockpile.

The ore will be re-handled from the ore stockyard using a front loader (FEL) that will feed the primary crusher. No direct feed by trucks to the crusher is considered.

The present review considers that the mining operation will be carried out by a contractor using 70-t class operating weight hydraulic excavators in backhoe configuration, which will load 8 x 4 trucks with 22 m^3 dump box size (Struck). This means about 10% more for heaped capacity per truck load using a 58 t Total Gross Weight (“PBT” in Brazil) truck but if a bigger truck capacity was used, a bigger dump box can be considered: the market already has a 66 t PBT capacity.

Mining is planned to be carried out in 10-m high benches. However, along the ore / waste contacts mining will be undertaken using 5 m high benches to improve selectivity.

When necessary, the material from low-grade stockpiles will be rehandled and fed into the processing plant. Risk mitigation strategies during the rainy season should be developed; otherwise, issues with potential loss of access to mining areas and operational difficulties may occur.

Short term grade control will be performed by a dedicated team that will be responsible for collecting samples and analyzing the ore quality upfront ore feeding.

The time for pit development and stockpile preparation is estimated at six months in advance before the process plant start up.

The water pump system selected is driven by diesel engine (so called "Moto-Bomba"), with 200 m^3/hour water pumping capacity at 50 HP for the rainwater drainage system in the mine pit. The system was quoted and included in capital costs. The pumping system is scheduled to start operating in the beginning, in the first year of mining operation, and will be able to accommodate the rainwater pump drainage system needs during all the pit Life of Mine (LOM). The final depth from the ultimate pit design is going to reach 150 meters deep, in the level 200 m asl.

The necessary pipes calculated for the rainwater pumping system, to pump the rainwater from the mine pit to the water treatment station, reached a length of 600 meters, using 6 inch diameter pipes.

16.3 MINE EQUIPMENT SELECTION

Like Aura’s other operations in Brazil, mining will be contracted. Aura will be responsible for contractor’s management to achieve the necessary production following the mine plan.

A mining fleet calculation exercise was performed.

16.3.1 EQUIPMENT SELECTION STRATEGY

As already mentioned, for the ore loading operations, a hydraulic excavator like a Caterpillar CAT374D equipped with a 3.5 m^3 bucket is proposed. The unit will be used to excavate and load 25 m^3 (35 tonnes) capacity trucks. A front-end loader, like a Caterpillar 966H equipped with a 3.5 m^3 bucket, will also be needed for the processing plant loading operations. Similar loading machines are going to be applied to waste rock loading.

The digging depth of these machines allows for top loading vocational trucks to be used for 2.5 benches and the ability to maintain the rock wall slope angles at the prescribed bench heights. In terms of operational flexibility, the excavator enables controlled excavation for mid-benches where the soil, saprolite and oxide ore zones intersect.

16.3.2 BLASTHOLE DRILLING AND BLASTING

The blasting pattern parameters for this pit operation are based on similar mine operations. Fragmentation quality monitoring should take place during the operation to optimize the total rock excavation in the mine and the grinding process.

The blasting pattern parameters used to estimate the explosives requirements are summarized in Table 16-4. A review was carried out regarding fragmentation information required for calculations.

Blastholes of 5.1/2inches in diameter are used for the 10-m high waste rock bench and 5- or 10-m high ore bench, depending on the local mineralization. For ore blasting, an explosive charge rate of 410 g/t is being considered, with electronic caps. In waste rock, the explosive charge rate is 220 g/t, with non-electrical caps.

Table 16-4 - Blasting Pattern Parameters According to the Height of the Benches, Ore and Waste.

BLASTING PATTERN PARAMETERS	UNIT	5 m BENCH HEIGHT		10 m BENCH HEIGHT	
MATERIAL	#	ore	waste	ore	waste
BENCH HEIGHT	m	5.0	5.0	10.0	10.0
% ROCK COLUMN BY BENCH HEIGHT	%	25%	25%	75%	75%
BLASTHOLE DIAMETER	(")	5.0	5.0	5.0	5.0
	m	0.127	0.127	0.127	0.127
BURDEN (B)	m	3	3.5	3	4.1
SPACING (E)	m	3.4	4.5	3.5	4.8
SPACING/BURDEN RATIO (E/B)	#	1.13	1.3	1.2	1.17
SUB DRILLING	m	0.5	0.5	1.0	1.0
HOLE INCLINATION (FROM HORIZONTAL)	(°)	90	90	90	90
TOTAL HOLE LENGTH	m	5.5	5.5	11.0	11.0
STEMMING	m	1.6	2	3.1	3.5
EXPLOSIVE DENSITY	g/cm^3	1.15	1.15	1.15	1.15
EXPLOSIVE CHARGE/METER	kg/m	14.6	14.6	14.6	14.6
EXPLOSIVE CHARGE PER BLASTHOLE	kg/hole	56.9	51.1	115.3	109.5
"IN SITU" ROCK VOLUME PER BLASTHOLE	m^3	51	79	105	197
SPECIFIC ROCK DENSITY "IN SITU"	t/m^3	2.67	2.44	2.67	2.44
"IN SITU" ROCK TONNAGE PER HOLE	t	136	192	280	480
SPECIFIC EXPLOSIVE CHARGE PER VOLUME	g/m^3	1.116	649	1.098	556
SPECIFIC EXPLOSIVE CHARGE PER MASS	g/t	419	266	411	228

Table 16-5 shows, by year, the volumes of rocks to be drilled and blasted, separating the fresh rock from the saprolite and soil. We considered that 30% of the saprolite will be drilled and blasted. Different blasting patterns will be used for waste and ore. Knowing the lithology, it is possible to calculate the blasting requirements and allows for the fleet selection of blasthole drilling rigs.

Table 16-5 - Annual Rock Tonnage Separated by Main Lithology or Waste.

		Ore			Waste			TOTAL BY LITHOLOGY		
		Fresh Rock	Soil	Saprolite	Fresh Rock	Soil	Saprolite	Fresh Rock	Soil	Saprolite
Ano	**Quarter**	**kt**	**kt**	**kt**	**kt**	**kt**	**kt**	**kt**	**kt**	**kt**
	Pre-Op.	**101.4**	**115.0**	**204.0**	**202.7**	**2,027.9**	**604.3**	**304.0**	**2,142.9**	**808.3**
1	**1**	234.6	13.4	285.7	66.5	711.0	229.2	301.2	724.4	514.9
	2	292.5	-	16.9	194.5	146.0	256.0	487.0	146.0	272.9
	3	303.0	-	-	241.2	95.3	244.0	544.1	95.3	244.0
	4	305.6	-	-	459.8	10.1	90.1	765.4	10.1	90.1
	Sub- total	**1,135.7**	**13.4**	**302.6**	**962.1**	**962.5**	**819.3**	**2,097.7**	**975.9**	**1,122.0**
2	**1**	304.1	-	-	503.5	-	44.4	807.6	-	44.4
	2	305.3	-	-	535.7	-	11.6	840.9	-	11.6
	3	310.0	-	-	555.5	-	-	865.6	-	-
	4	322.6	-	-	410.0	28.0	144.6	732.6	28.0	144.6
	Sub- total	**1,242.0**	**-**	**-**	**2,004.7**	**28.0**	**200.5**	**3,246.7**	**28.0**	**200.5**
3	**1**	315.2	-	-	389.8	30.0	120.0	705.0	30.0	120.0
	2	307.9	-	-	339.5	40.0	100.0	647.4	40.0	100.0
	3	305.3	-	-	338.1	80.0	100.0	643.3	80.0	100.0
	4	292.3	2.6	2.6	304.9	118.3	119.0	597.1	120.8	121.6
	Sub- total	**1,220.6**	**2.6**	**2.6**	**1,372.3**	**268.3**	**439.0**	**2,592.9**	**270.8**	**441.6**
4	**1**	289.7	-	5.5	396.9	46.9	90.6	686.6	46.9	96.1
	2	305.6	-	20.5	408.7	54.3	74.7	714.2	54.3	95.2
	3	339.3	-	5.3	514.5	93.5	177.9	853.8	93.5	183.2
	4	339.9	-	20.5	648.1	4.6	73.2	988.0	4.6	93.6
	Sub- total	**1,274.5**	**-**	**51.8**	**1,968.2**	**199.3**	**416.3**	**3,242.6**	**199.3**	**468.1**
5	**1**	318.8	-	-	536.3	-	2.7	855.1	-	2.7
	2	333.5	-	-	530.7	-	-	864.3	-	-
	3	329.1	-	1.3	489.4	-	28.7	818.5	-	30.0
	4	324.6	-	2.3	313.6	-	-	638.2	-	2.3
	Sub- total	**1,306.1**	**-**	**3.6**	**1,870.0**	**-**	**31.3**	**3,176.1**	**-**	**35.0**
6	**1**	309.2	-	-	78.5	-	-	387.7	-	-
	2	307.2	-	-	44.1	-	-	351.3	-	-
	3	311.2	-	-	54.4	-	-	365.6	-	-
	4	328.9	-	-	107.0	-	-	435.9	-	-
	Sub- total	**1,256.6**	**-**	**-**	**284.0**	**-**	**-**	**1,540.6**	**-**	**-**
7	**1**	239.2	-	-	31.3	-	-	270.5	-	-
	2	-	-	-	-	-	-	-	-	-
	3	-	-	-	-	-	-	-	-	-
	4	-	-	-	-	-	-	-	-	-
	Sub- total	**239.2**	**-**	**-**	**31.3**	**-**	**-**	**270.5**	**-**	**-**

Note: Ano = year, kt = kilotonnes or thousands of tonnes.

The drilling fleet selection parameters to calculate the blasthole needs for rock excavation are summarized in Table 16-6.

Table 16-6 - Blasting Pattern Parameters Depending on the Height of the Benches, Ore and Waste Rock.

DRILLING PARAMETERS	Unit	Bench Height = 5 m		Bench height = 10 m	
MATERIAL	#	ore	Waste	Ore	Waste
DRILLING RIG SIMILAR MODEL	Sandvik	DP 1500	DP1500	DP1500	DP 1500
HAMMER TYPE	#	Top Hammer	Top Hammer	Top Hammer	Top Hammer
ANNUAL CALENDAR HOURS	h	8.76	8.76	8.76	8.76
PLANNED ANNUAL HOURS	h	8.52	8.52	8.52	8.52
MECHANICAL AVAILABILITY	%	73%	73%	73%	73%
UTILIZATION	%	83%	83%	83%	83%
TOTAL UPTIME RATIO	%	65%	65%	65%	65%
ANNUAL WORKING HOURS	h	4.043	4.043	4.043	4.043
ANNUAL ENGINE HOURS	h	3.369	3.369	3.369	3.369
DRILLING RATED METER	m/h	25	25	25	25
% BENCH HEIGHT	%	25%	25%	75%	75%
"IN SITU" DENSITY	t/m^3	2.67	2.44	2.67	2.44
"IN SITU" ROCK TONNAGE PER HOLE	t	136	192	280	480
PRODUCTION CAPACITY PER RIG	Mtpa	1.02	1.43	2.09	3.58

It is planned that 30% of total Saprolite is going to be drilled and blasted, therefore the plan assumes that the drilling and blasting cost for saprolite is going to be 30% of the fresh rock drilling and blasting costs. Table 16-7 shows the drilling rig quantities calculated for production blastholes.

Table 16-7 - Drilling Rig Quantities Needed Per Year, Calculated Considering the Different Lithologies.

	ROCK TO BE BLASTED WITH EXPLOSIVES					
	TOTAL FRESH ROCK		TOTAL SAPROLITE		DRILLING FLEET	
YEAR	TONNAGE	DRILLING FLEET CALCULATED	TONNAGE	DRILLING FLEET CALCULATED	TOTAL CALCULATED	TOTAL SELECTED
	Kt	Quantity	kt	Quantity	Quantity	Quantity
Pre-Oper.	**304**	**1.2**	**808**	**1.6**	**2.8**	**3**
1	**2,098**	**2.1**	**1,122**	**0.6**	**2.7**	**3**
2	**3,247**	**3.2**	**201**	**0.1**	**3.3**	**4**
3	**2,593**	**2.6**	**442**	**0.2**	**2.8**	**3**
4	**3,243**	**3.2**	**468**	**0.2**	**3.5**	**4**
5	**3,176**	**3.2**	**35**	**0.0**	**3.2**	**4**
6	**1,541**	**1.5**	**-**	**-**	**1.5**	**2**
7	**271**	**0.3**	**-**	**-**	**0.3**	**1**

The drilling fleet considered equipment like the DP 1500 to drill holes of 5.0 inches and 5½ inches in diameter.

The Pre-operation time is planned to be a 3-month period, after which drilling will increase to a reasonable productivity rate, once the superficial soil has been removed by stripping. The Drilling Rigs mobilization is planned to start after the pre-operation period, followed by the startup of the other fleets.

In practice, normally 45% to 50% of the drilling rigs are in use at one time based on experience with similar applications. This is low utilization point for unit operation which can transform drilling operations into one of the mine operation bottlenecks. Considering a total cost analysis, the highest costs are transport and grinding. In conclusion, it is recommended to maintain some extra capacity for drilling equipment for reliable availability.

The hydraulic drilling rigs are a high productivity alternative and provide the possibility of technologies that control dust generation, with high efficiency "dust collector" systems. The generation of dust by the operation could affect environmental problems, in addition to significantly increasing equipment maintenance costs.

The equipment can help to reduce the dilution of the ore by improving the selectivity of the mine with GPS systems for correct drilling of the holes and more accurate control of the inclination of the holes, even with the irregularities in surfaces where the drilling rigs usually work.

As GPS technology for drilling rigs, although available on the market, is not widely used; the equipment must have a good hole depth measurement system ("depth meter") and a very functional and easy learned hole angle system. The depth meter can significantly assist in the quality of the pit floor resulting in reduced Bulldozer work hours. An uneven floor makes it difficult to collar and drill the holes.

A precision angle indicator ("inclinometer"), in addition to helping to decrease the ore dilution, allows for much greater adherence of the holes to the blasting patterns and consequently minimizes the generation of vibration in the rock mass.

Figure 16-1 below shows the most common causes of drilling errors that can be minimized with the use of a depth meter and an inclinometer.

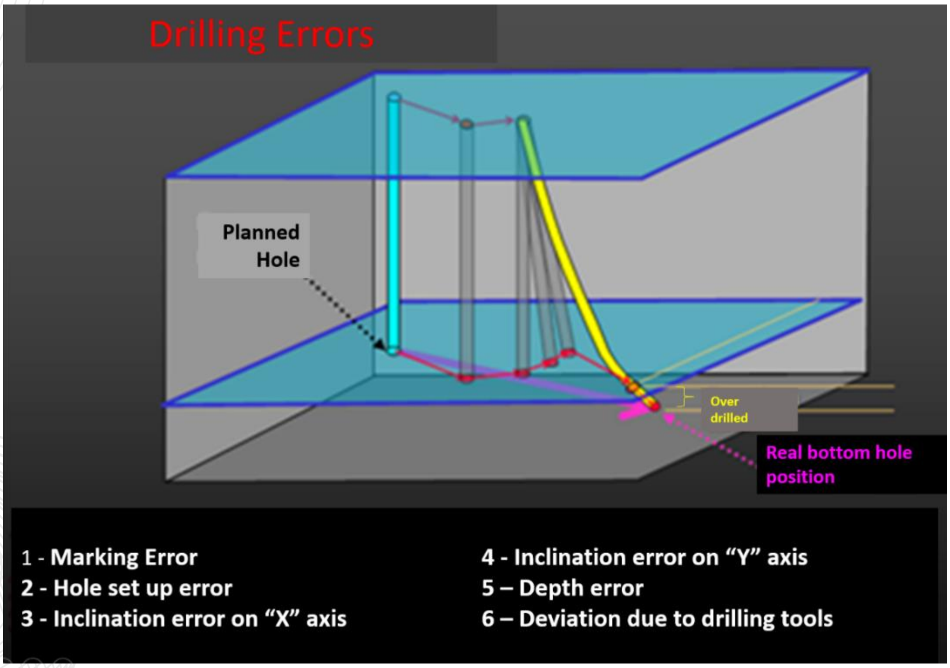


Figure 16-1 - Most Frequent Reasons for Error on Drill Hole Positioning, Related to Blasting Pattern.

Detail of the fleet variation during the LOM for the mining operation, considering the production of ore at 1.3 Mtpy is shown in Table 16-16.

16.3.2.1 ORE BLASTING FRAGMENTATION TO FACE THE PLANT REQUEST

The studies completed consider the objective of achieving an ore blasting operation with a distribution size curve targeting a "top size" rock fragment of 600 mm, P_{80} lower than 250 mm, P50 lower than 80 mm.

The analysis considered the available data of rock mass from the X1 mineral deposit compared with other similar applications.

Considering a drilling diameter of 5 1/2 inches, an explosive charge ratio of 410 g/t will be required.

Depending on the size of the Crusher, the ROM top size must be evaluated to control the size of the blocks to be fed from the crusher. The planned way to feed the crusher creates a favorable condition to control the block size: the ore is going to be from the ore storage through a Front-End Loader, not straight from the mine. In this way, the operation will have an additional operational control point to the crusher's feed.

The estimation of blasting fragments larger than 600 mm size is around 3%; this size could be reduced by using a suitably selected hydraulic breaker, greater than 2,400 kg, mounted on a hydraulic excavator working in the ore yard.

The blasthole diameter for the fragmentation size required is 5½ inches, for 10 m high benches in the waste rock and 5 m high benches in the ore. In this report a 5 inch hole diameter was considered, that means an additional safety margin. The equipment model selected, and the size of the fleet, meet both possibilities. In both cases, the explosive charge ratio was maintained and that means the cost considerations remain the same.

The explosives charge rate for the ore in this report is 410 g/t, with the use of accessories with electronic caps. The costs of the grinding operation are significantly higher than drilling and blasting, but it can be greater if the fragmentation of the blasting operation is not adequate. In waste material, the explosive charge ratio will be 220 g/t, with non-electrical accessories. Obviously, because they are unit operations based on many empirical aspects, drilling and blasting must be optimized during operation.

16.3.3 GRADE CONTROL DRILLING RIG

Grade Control Drilling is used to update the short-term mining plan to carry out grade control in gold mines. The use of reverse circulation has been shown to be the most effective for supporting mining reconciliation. It is strongly recommended to use this technology.

Reverse circulation (“RC”) drilling has become standard practice in most mines around the world for grade control for mining reconciliation. First developed in Australia in the 1970s, the drilling technique was originally applied as a drilling solution technique to drill hole difficulties encountered in soft iron ore and mineral sands. The first RC drill rods were adapted from the US oil industry and manufactured in Western Australia in 1972 by Bruce Metzke and John Humphries.

The drill cuttings are transferred to the surface inside of the drill rods, which are linked together to create a ‘drill string’. Drill bits attached to the end of the hammer are made from tungsten steel. These also have metal nodules attached at the end to allow cutting through particularly tough rock.

Most RC drilling rigs use dual pipe drill rods, with one tube inside another. The tubes inside overlap and provide a path for drilled rock from the ground to reach the surface. See Figure 16-2 for an illustration from the Boart Long Year Catalog.

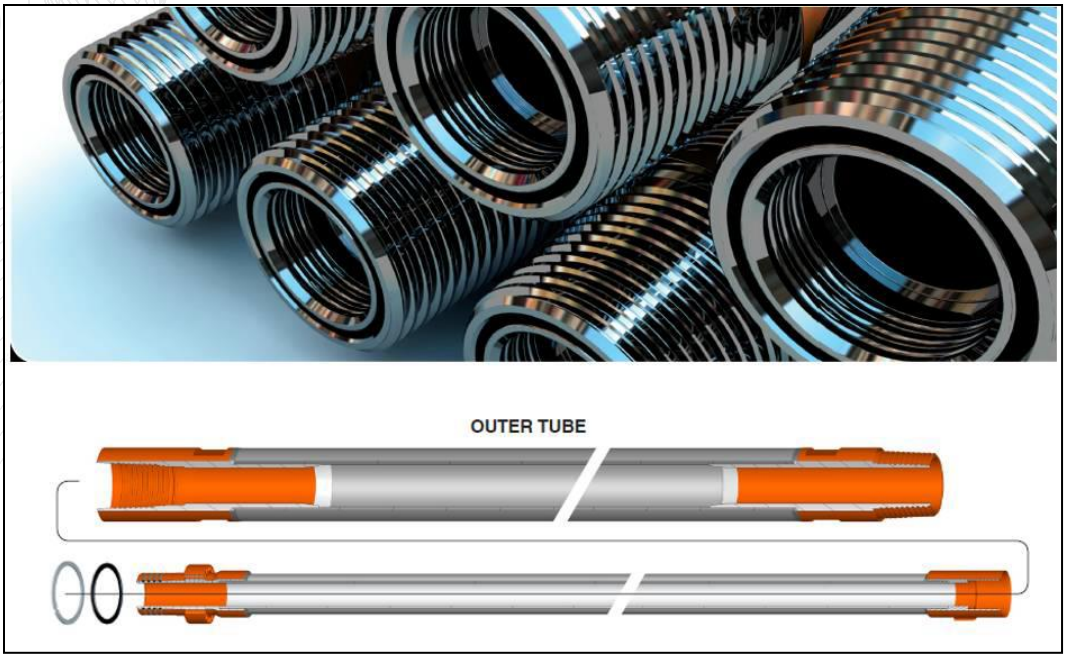


Figure 16-2 - RC Dual Pipe Rods Images from Boart Longyear – Global Product Catalogue – Reverse Circulation, 2009.

The machine uses regular Down-the-Hole drilling, like the Epiroc FlexiDrill machine, including the special rotary head to allow the collection of drill cuttings and conveying them to a quarter system. The Down the Hole Hammer is specifically designed to allow the drill cuttings to pass through the hammer, as shown in Figure 16-3.

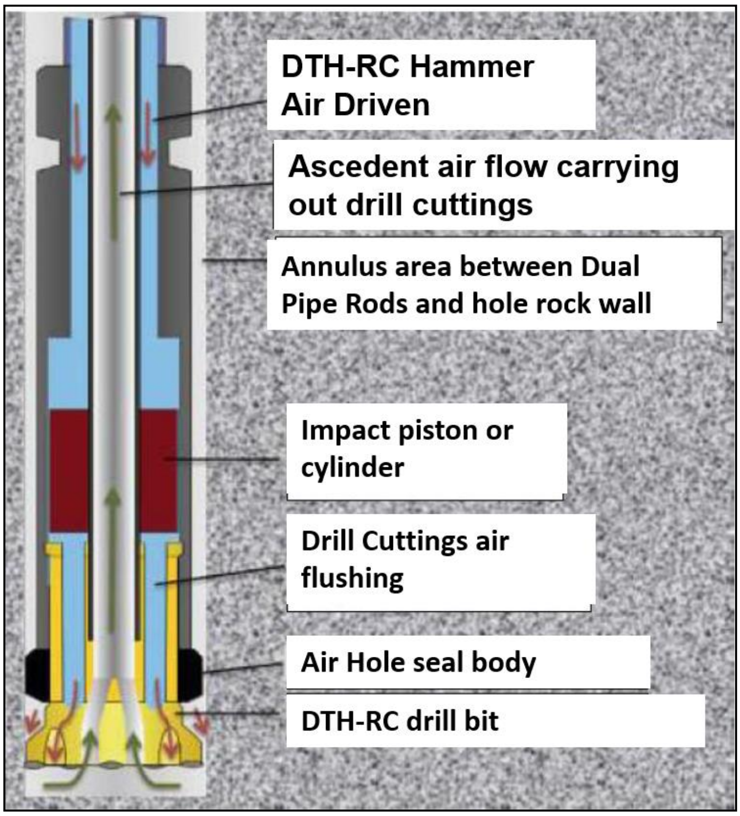


Figure 16-3 - Schematic RC DTH Hammer for Grade Control Drilling Recommended Epiroc–Product Catalogue.

Figure 16-4 shows the main components to complement the regular DTH drill rig and convert it to an RC drilling rig for grade control – Epiroc Product Catalogue.

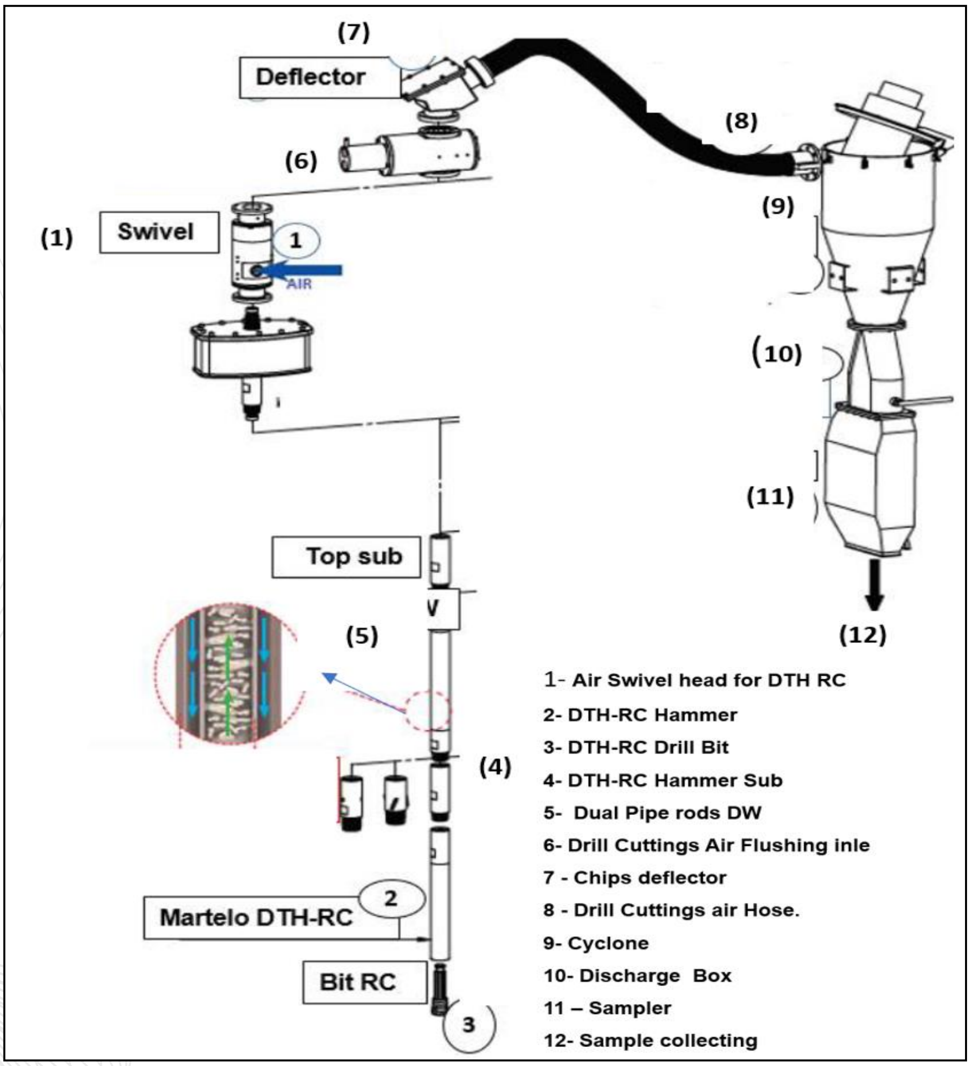


Figure 16-4 - Shows all the Main Components that Complement the Drilling Rig to Covert in RC Machine for Grade Control.

The type of sampler regularly recommended for grade control is the conical type as illustrated in Figure 16-5.

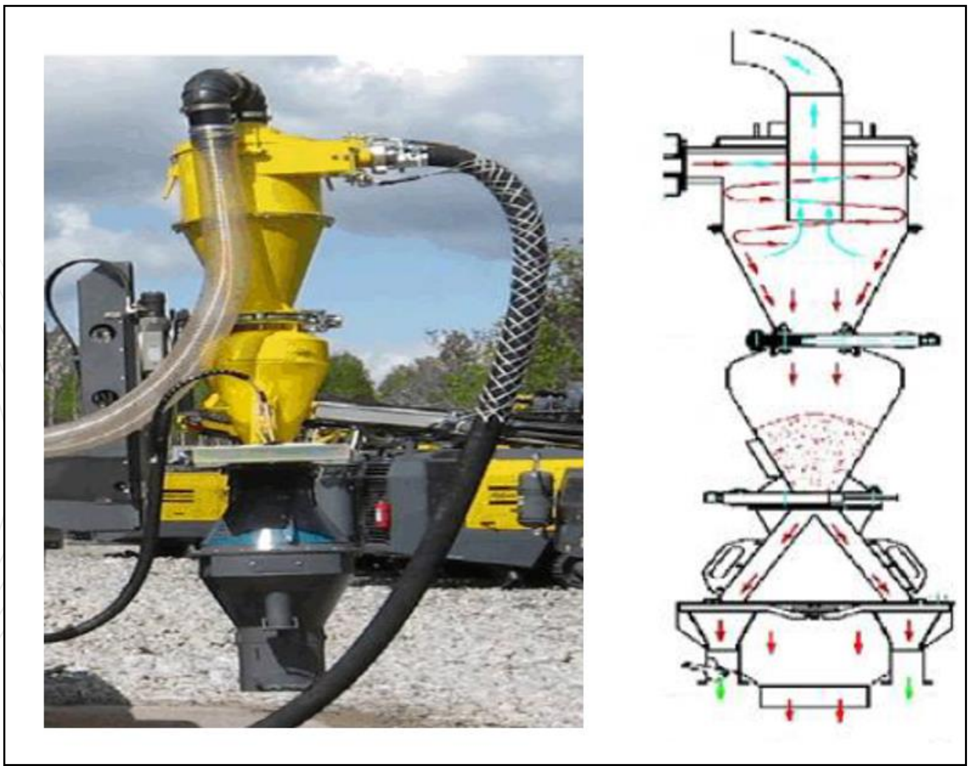


Figure 16-5 - Conical Sampler Illustration, Normally Recommended for Grade Control in Surface Mining Operations.

The parameters proposed to select the fleet for grade control rigs are:

- Grade control drilling mesh: 10 x 10 m.
- Drill hole depth: 30 m.
- Diameter hole: 5¾ inces.
- Two shifts operation.

Table 16-8 presents the yearly drilling rigs fleet calculation needs exclusively for grade engineering control.

Table 16-8 - Drill Control Fleet Needs Per Year.

	ORE VOLUME							Drilling meters
	Fresh Rock		Soil		Saprolite		Total	
Density	2.7	t/m^3	2.1	t/m^3	2.3	t/m^3	Volume	10 X 10 m
Year	kt	m^3x10^3	kt	m^3x10^3	kt	m^3x10^3	m^3x10^3	m
Pré Op.	234.6	86.9	13.4	6.4	285.7	124.2	217.5	2,900
1	901.0	333.7	-	-	16.9	7.3	341.1	3,411
2	937.9	347.4	-	-	-	-	347.4	3,474
3	984.7	364.7	-	-	46.2	20.1	384.8	3,848
4	324.6	120.2	-	-	2.3	1.0	121.2	1,212
5	947.3	350.9	-	-	-	-	350.9	3,509
6	-	-	-	-	-	-	-	-
7	-	-	-	-	-	-		-

The beginning of the operation is expected to be a challenging situation when too many meters will be needed to be drilled in a short period. For the rest of the mining period, the plan is for a one unit, well-managed drilling operation for engineering control, although Table 16-16 shows a number slightly above the one unit.

16.3.4 LOADING RUN OF MINE ROCKS EQUIPMENT

Table 16-9 - Presents the Parameters Used to Select the Loading Fleet for the ROM on the Mining Fronts.

Parameters	unit	VOCATIONAL TRUCK	
Rock type	#	Fresh Rock	Soil/Saprolite
Trucks Dump Box Heaped load	m^3	24	24
Minimum truck 8x4 PBT	t	58	58
Rock Tranport truck capacity	t	39	36
Hydraulic Excavator typical model	#	CAT 374	CAT 374
Excavator bucket size	m^3	4.3	4.3
Tonnagem excavator loadind capacity	t	6.7	7.3
Calculated loading cycle quantity	Qty	5.8	4.9
Roundd loading cycle quantity	n.º	5	5
Truck time manoeuvre in loading front	min	0.5	0.5
Truck loading Time	min	1.1	1.1
Truck queue waiting time	min	1	1
Loading Cycle time	min	0.58	0.58
Total Excavator Loading cycle time	min	3.4	3.4
Total Truck loading time	min	5.5	5.5
Hydraulic Excavator Productivity	t/h	425	393

Table 16-10 shows the working hours need for the loaders.

Table 16-10 - Truck Working Hours Considering all Origins and Destinations to Transport all Planned Rock Moved in all Periods of LOM.

	LOADING WORKING HOURS NEEDED (h)													
	MINE → CRUSHING YARD STOCKING / PILES									From Piles		TOTAL		
	High-Grade Ore			Low-Grade Ore			Waste			To Crushing Yard				
LITHOLOGY	Fresh Rock	Soil	Sapr.	Fresh Rock	Soil	Sapr.	Fresh Rock	Soil	Sapr.	soil + Sapr.	Low Grade	Fresh Rock	soil + Sapr.	All
YEAR▼	h	h	h	h	h	h	h	h	h	h	h	h	h	h
Pre Oper.	238	265	496	-	28	23	476	5,164	1,539	-	-	715	7,515	8,229
1	2,595	34	720	74	-	51	2,261	2,451	2,086	240	-	4,931	5,582	10,512
2	2,820	-	-	99	-	-	4,712	71	511	255	-	7,631	837	8,468
3	2,784	6	-	85	1	7	3,225	683	1,118	253	-	6,094	2,067	8,162
4	2,727	-	102	268	-	30	4,626	507	1,060	254	-	7,622	1,953	9,575
5	2,815	-	6	255	-	3	4,395	-	80	261	-	7,465	350	7,815
6	2,820	-	-	133	-	-	668	-	-	255	-	3,621	255	3,876
7	538	-	-	24	-	-	74	-	-	144	1,071	1,707	144	1,851

Note: h = hours.

The cost of transport is usually much more significant than the cost of loading. Thus, it is important to ensure the optimization of the most expensive mining operation unit. It is recommended that Aura ensures a realistic additional capacity to the loading fleet, and all the fleet related to mine infrastructure, to provide continuous, good transport conditions.

An electronic dispatch system can benefit ROM loading to minimize queues when loading.

Details of the fleet size variation over time for this proposed mining operation, considering the production of ore to 1.3 Mtpy is shown in Table 16-16.

16.3.5 TRUCKS FOR ROM TRANSPORT

The conventional 8 x 4 road truck type (vocational) with a 22 m^3 dump box suitable for transporting primary rock is the equipment selected. Figure 16-6 schematically illustrates a "sketch" of the 8 x 4 truck with blasted rock dump box type, manufactured by

Rossetti and with a struck capacity of 22 m³. Unlike Caterpillar off-road trucks, the volume of the bucket is struck and not heaped. This fact means that the dump box heaped load of the truck should be between 24 to 25 m³.

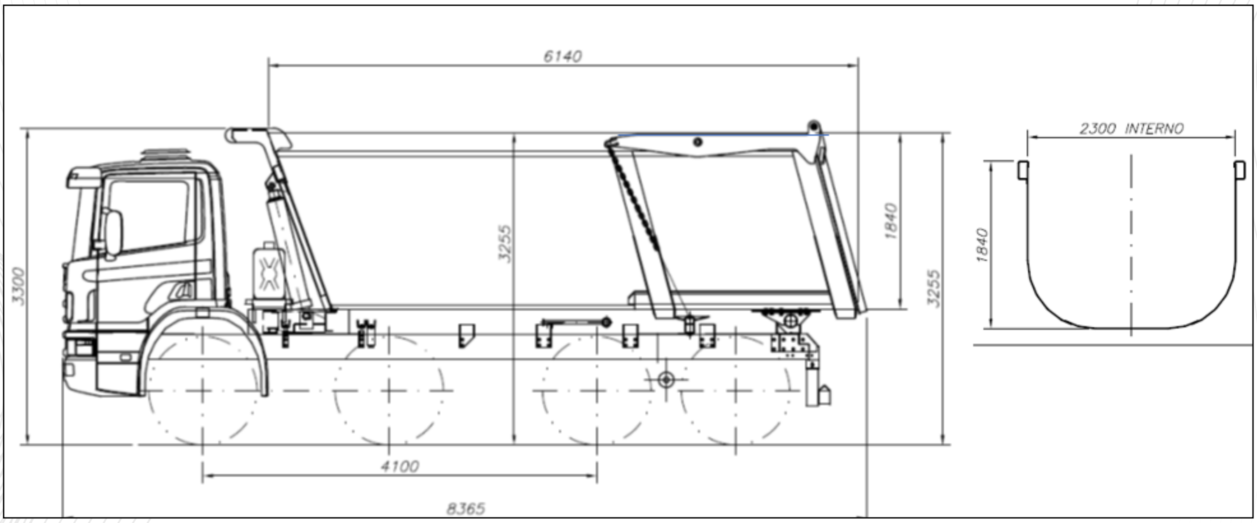


Figure 16-6 - Schematic Drawing of the 8 x 4 Truck with typical Bucket for Rossetti Mining Operations.

If the primary blasted rock density is 1.7 to 1.8 t/m³, the truck load is going to be around 39 tonnes; five cycles will be needed for the 4.3 m³ bucket size excavator to fill it.

In terms of specification, the trucks have a PBT (Total Gross Weight) of 58,000 kg. Discounting the tare of 19,000 kg (truck = 12 t + bucket = 7 t) there would be 39 t left for the net load. This overload is a situation already observed in other similar operations.

Safety issues should be the biggest concern with transport. The main items to be observed by the maintenance personnel are the brakes, the steering, and the suspension systems. Obviously, on the roads, higher speeds will require greater demands from the brakes and steering system. Similar truck manufacturers (Scania and Volvo) are already manufacturing this equipment with a PBT of 66,000 t.

For the truck fleet selection, average truck speeds considered are shown in Table 16-11.

Table 16-11 - Average Transport Speeds for 8 x 4 Vocational Trucks.

TRUCK AVERAGE SPEED			
AVERAGE TRUCK SPEED Km/h		**IN THE PIT**	**OUT OF PIT**
Loaded	**up ramp**	11	11
	horizontal	30	30
	down ramp	20	20
Empty	**up ramp**	30	30
	horizontal	30	35
	down ramp	16	16

An average fixed total time of six minutes for loading by hydraulic excavator, as per Table 16-6, includes: maneuvering in the loading area, maneuvering, and tipping time at the discharge and waiting time in the queue.

The transport cycle time variable, for trucks, was calculated based on the average transport distances for all the LOM periods shown in Table 16-12

Table 16-12 - X1 Pit: Average Transport Distance (ATD) in meters Year by Year B, Origin and Destination of Each Route.

AVERGAGE TRANSPORT DISTANCE (ATD)						
Origin	Mine			Crushing Yard	Ples	
Destin	Crushing Yard / Piles			Crusher	Crushing Yard	
Lithology	High Grade	Low Grade	Waste	Ore	Oxid	Low Grade
YEAR	m	m	m	m	m	m
Pré Oper	1,373	1,505	1,335	-		
1	1,283	1,649	1,431	30	800	-
2	1,392	1,780	1,491	30	800	-
3	1,731	2,084	1,890	30	800	-
4	1,301	1,476	1,291	30	800	-
5	1,335	1,658	1,472	30	800	-
6	1,339	1,732	1,475	30	-	-
7	1,890	2,356	2,556	30	-	900

Table 16-13 shows in the pit sequenced period: average transport distances to calculate the truck productivities.

Table 16-13 - X1 Pit: Calculated Trips per Hour (TPH), Year by Year, by Origin and Destination of Each Route.

Trips per Hour (TPH)						
Origin	Mine			Crushing Yard	Piles	
Destin	Crushing Yard / Piles			Crusher	Crushing Yard	
Lithology	High Grade	Low Grade	Waste	Ore	Oxid	Low Grade
YEAR	TPH	TPH	TPH	TPH	TPH	TPH
Pré Oper	3.5	3.3	3.6	Loader	-	-
1	3.6	3.2	3.4	Loader	4.5	-
2	3.5	3.0	3.3	Loader	4.5	-
3	3.1	2.7	2.9	Loader	4.5	-
4	3.6	3.4	3.6	Loader	4.5	-
5	3.6	3.1	3.4	Loader	4.5	-
6	3.5	3.1	3.4	Loader	-	-
7	2.9	2.5	2.4	Loader	-	4.3

Note: TPH = tonnes per hour.

Table 16-14 shows in the pit sequenced period: working hours of Trucks for transport all the ROM and rock rehandling from piles.

Table 16-14 - Transport Calculated Working Hours Yearly Needed to Move All Rock, Year by Year, by Origin and Destination of Each Route.

TRUCKS WORKING HOURS BY YEAR NEED FOR ALL ROCK MOVED (h)							
Origin	Mine			Crushing Yard	Piles		All
Destin	Crushing Yard / Piles			Crusher	Crushing Yard		All
Lithology	High Grade	Low Grade	Waste	Ore	Oxid	Low Grade	TOTAL
YEAR	h	h	h	h	h	h	h
Pré Oper	3,269	173	22,814	Loader	-	-	26,256
1	11,038	467	22,919	Loader	599	-	35,022
2	9,875	-	19,082	Loader	635	-	29,592
3	11,041	407	20,400	Loader	632	-	32,480
4	9,527	1,071	20,436	Loader	633	-	31,667
5	9,657	996	16,128	Loader	650	-	27,431
6	9,672	527	2,412	Loader	-	-	12,611
7	2,247	115	373	Loader	-	3,038	5,773

Note: h = hours.

Table 16-16 presents the necessary truck working hours, by year, for all the transport requirements related to the total rock moved in the LOM plan.

16.3.6 MINING INFRASTRUCTURES AND AUXILIARY EQUIPMENT

16.3.6.1 MOTO-GRADER

The selection of motor graders should not only consider the sum of the lengths of all accesses to be maintained, but also the multivariate tasks that the grader performs in a mining operation. It is also important to consider equipment of adequate size for mining operations, with a minimum blade width of 14 feet (CAT 140). For the beginning of the operation, 2 units are selected, but it is estimated the need will increase by at least one more unit when the mine operation occurs in two pits simultaneously.

16.3.6.2 LOW BED TRUCK

Considering the various crawler-propelled equipment that need to tram constantly over longer distances, either because of the arrangement of the operation fronts, or because of the needs of maintenance in the shop, it is recommended that at least one low bed truck unit be acquired to meet the mining operation. This unit can also be used to transport large components of the concentration plant during large scale maintenance. The low bed truck width should consider the ability to transport safely the widest crawler equipment in the mining operation.

16.3.6.3 WATER TANK TRUCK

Despite the high-level rain rates in the region, the dry season periods are prolonged and will require the use of many hours of water trucks. Two water truck units are recommended for one pit, more units will be needed when the operation involves two pits simultaneously. Usually, the water sprinkler system used is the “peacock tail” properly regulated to moisten the floor and not “wash” the floor. It is important that water trucks have a nozzle system to help put out fires during the dry season.

16.3.6.4 BULLDOZERS

At current production levels, 3 tractors weighing 35 tonnes (CAT D8) would be required to ensure the availability of two units most of the time. This need is accentuated in the rainy season.

One of the fundamental parameters for this type of application is the crawler tractor’s weight. The 20-tonne crawler tractor (CAT D6) is lightweight and is able to carry out the necessary work in the mining area and is suitable for the spreading activities in the waste dump areas. However, if there are large areas of deposition this can be an issue. CAT D6 tractors could be extremely useful in mine road maintenance when the heaviest jobs are difficult or become impractical for the graders. Due to the size of the equipment and investment value, it is rare tp find a wheel tractor (like to CAT 824 model) in mining operations in Brazil. One CAT D6 per pit on operation and one unit at the end of the operation is recommended.

16.3.7 EQUIPMENT WORKING HOURS CALCULATION

Table 16-15 shows the key parameters assumed for equipment fleet sizing, including the mining Infrastructure and main auxiliary equipment selected. The table presents the mine fleet size selected per mining unit operation and by unit rig: shift/day planned, utilization, yearly working hours calculated and diesel consumption estimated.

Table 16-15 - Equipment Calculated Working Hours and Diesel Consumption per Equipment Unit and Type.

Equipament	Shifts per day	Availability	Utilization	Working hour / year	Diesel (liters/h)
Truck, 8x4 PBT > 58t	3	75%	65%	-	16
Hydraulic Excavator, 70t class	3	85%	65%	-	55
Loader with 3,5 m3, similar to CAT 966	3	80%	70%	-	30
Trator de Esteiras 35 ton, similar to CAT D8	3	85%	50%	3,723	38
Grader similar to CAT 140	3	85%	60%	2,978	18
Explosives Truck	1	85%	45%	1,104	14
Water tank truck	3	85%	60%	4,415	14
Back hoe loader	3	85%	45%	1,104	12
Drilling rog similar ro Sandvik DP1500	3	85%	45%	5,151	35
Grade Eng. Drilling rig similar ► Epiroc D65 RC	2	85%	50%	5,151	40
Hydraulic Breaker + Hydraulic Escavator	3	85%	50%	5,151	35
Lube + Diesel tank trucks	3	85%	45%	3,311	12
Workshop Truck	1	85%	15%	1,104	12
Lighting tower	2	85%	30%	1,472	2
Pick ups	3	85%	40%	2,943	5
Low Boy + Truck	3	85%	15%	1,104	45

16.3.8 SUMMARY OF MINING EQUIPMENT FLEET

For optimized operations it is essential for the complete synchronization of:

- Grade control drilling.
- Drilling for production.
- Mining development and preparation by Bulldozers/Graders/ Water tank trucks and others.
- Loading by excavators.

It is common in open pit mining operations to have a proper fleet selection for drilling, blasting, loading and transport but the auxiliary equipment for mine infrastructure is not suitable or are undersized. The improper selection for auxiliary equipment can significantly affect the costs of the operations of loading, transportation, crushing and grinding.

The required fleet selected for mining operations is shown in Table 16-16. The fleet calculations are rounded to define the fleet necessary to accomplish the mining sequence planned for the X1 Deposit, Matupá Project. The parameters considered are presented in Tables 16- 1 to 16-15. The summary of the selected fleet for the Matupá Project, mine X1 mining operation, to feed ore volume per year of 1.3 Mt is presented in the Table 16-16.

The estimated fleet for ROM transport starts with a fleet of 8 trucks and reaches a peak in year 3. As the equipment is made in Brazil, and quite common in the local market, the most appropriate investments can be made to vary production levels. Equipment that can be easily acquired on the market to be sent to a rock-type dump box supplier, where, in addition to the dump box, several safety items are installed. The waiting period for these alterations is about 90 days. Minimizing the waiting time on excavators increases the productivity of trucks and can decrease the size of the fleet.

The roads conditions will certainly affect the speed and safety conditions for the ROM transport.

As the unloading of the ore is planned to take place in a yard, a Front-End Loader loads the ore and feeds the concentration plant. No queues are expected at the ore truck unloading point.

Details of the fleet variation over time for this mining operation, considering the production of ore to 1.3 Mtpy, is shown in Table 16-16.

Table 16-16 - Summary from Calculated and Rounded Fleet to Accomplish the Production of 1.3 Million Tonnes of Ore Per Year.

Equipment	Referrence Model	YEAR							
		Pre Op.	1	2	3	4	5	6	7
Loading ROM									
Hydraulic Excavator	CAT 374	3	3	3	2	2	2	1	1
Front End Loader	CAT966L	1	2	2	2	2	2	2	1
Transport ROM									
Vocational Truck 8x8	Scania HT G500	9	9	9	7	7	6	4	4
Drilling Rigs									
Grade Eng. Sampling	Epiroc D65 RC	1	1	1	1	1	1	1	0
Top Hammer Hydraulic	Sandvik DP 1500	3	3	4	3	4	4	2	1
Mine Infraestructure + Service Vehilcle									
Buldozers	CAT D8	2	2	2	2	2	1	1	1
Graders	CAT 140	2	2	2	2	2	2	1	1
Explosive Trucks	MB-6x4	1	1	1	1	1	1	1	1
Water Tank Trucks	MB-6x4	2	2	2	2	2	2	2	2
Back Hoe Loaders	CAT 416	1	1	1	1	1	1	1	1
Hydraulic Breakes + Hydr. Escavators	CAT 330	1	1	1	1	1	1	1	1
Lube + Fuel tank Truck	MB-6x4	2	2	2	2	2	2	2	2
Maintenace Truck	MB-6x4	1	1	1	1	1	1	1	1
Lightning tower	Terex RL 4K	7	7	7	7	7	7	2	2
Pick ups	Hilux	10	10	10	10	10	10	7	5
Low Bed Truck	to be specified	1	1	1	1	1	1	1	1
Total Fleet		47	48	49	45	46	44	30	25

16.4 OPERATIONAL MINING COSTS

As mentioned before, the operation concept discussed with Aura is outsourced operation, i.e., all the mining operation is going to be contracted.

The basis for costs calculations are the contracts from Mineração Apoena, for the Ernesto Mine – MT. The contract was signed on October 2020 and revied in December 2021 and involves the following:

- Drilling, Loading and Transport the ROM, Dozer for spreading the material, prepare and maintain all mine infrastructure. The diesel is supplied by Aura Minerals, and it is deducted from monthly payment to the contractor.
- Explosive and accessories supplying. The explosives and accessories are supplied by Aura Minerals and deducted from monthly payment to the contractor.
- Grade Control Drilling, contracted by meter. The diesel is supplied by Aura Minerals and deducted from monthly payment to the contractor.

The diesel price by liter used was the price in Matupá/Guarantã do Norte's region, Mato Grosso state in December 2021

Tables 16-17 present, by pit, the costs in American dollars and Brazilian reais per tonne to the mining operations described above, and the G&A and Processing Costs that are going to be the basis for costs calculations. The US Dollar rate to the Brazilian Real applied is: US$1.00 equivalent to R$ 5.58.

Table 16-17 - G&A, Processing and Mining Costs by Lithology.

	AREA	Ore/Waste	Lithology	Currency: US$/t		Currency: R$ /t	
COSTS	G & A	Ore	All	US$/t ore	1.64	R$/t ore	9.15
	Processing	Ore	All	US$/t ore	15.44	R$/t ore	86.16
	Mining	Ore: High Grade + Low-Grade	Soil	US$/t	1.35	R$/t	8.26
			Saprolite Rock	US$/t	1.51	R$/t	9.30
			Fresh Rock	US$/t	1.65	R$/t	10.16
		Waste	Soil	US$/t	1.35	R$/t	7.46
			Saprolite Rock	US$/t	1.43	R$/t	7.91
			Fresh Rock	US$/t	1.57	R$/t	8.69
	Stockpile	Reaclaim	All	US$/t	0.60	R$/t	3.35

Based on the numbers in Table 16-17 and Table 16.5 showing the total rock moved by: quarter, origin, destination, lithology; it was decided to calculate the mining costs by quarter as summarized in the Table 16-19.

Table 16-18 shows the grade control costs considered for grade control engineering.

Table 16-18 - Grade Control Costs to be Applied in Ore and Waste.

GRADE ENGINEERING	R$ / t
Geology/Mining Plan/Grade Control	**1.40**

Table 16-19 presents a Matupá Project Mining Costs Summary by quarter.

Table 16-19 - Matupá' s Gold Project Summary Mining Costs.

MATUPA'S PROJECT MINING COSTS - Million BRL (R$*1,000,000.00)																
TIME PERIOD		HIGH GRADE ORE (HG)				LOW GRADE ORE (LG)				TOTAL ORE HG + LG	RECLAIM LG/ OXID ORE	WASTE				GRAND TOTAL
BY YEAR ▼	QUARTER ▼	SOIL	SAPRO-LITE	FRESH ROCK	TOTAL	SOIL	SAPRO-LITE	FRESH ROCK	TOTAL			SOIL	SAPRO-LITE	FRESH ROCK	TOTAL	
		R$	R$	R$	R$	R$	R$	R$	R$	R$	R$	R$	R$	R$	R$	R$
PRE OPERATION▶		0.89	1.86	1.03	3.78	0.09	0.09	-	0.18	3.96	-	15.13	4.78	1.76	21.67	25.63
YEAR - 1	Commission.	0.11	2.54	2.24	4.90	-	0.19	0.14	0.34	5.23	0.07	5.30	1.81	0.58	7.70	13.00
	2	-	0.16	2.88	3.04	-	-	0.10	0.10	3.14	0.08	1.09	2.03	1.69	4.80	8.02
	3	-	-	3.05	3.05	-	-	0.03	0.03	3.08	0.08	0.71	1.93	2.10	4.74	7.90
	4	-	-	3.05	3.05	-	-	0.06	0.06	3.11	0.08	0.08	0.71	4.00	4.78	7.97
	Sub-total	0.11	2.70	11.22	14.03	-	0.19	0.33	0.52	14.56	0.32	7.18	6.48	8.36	22.02	36.89
YEAR - 2	1	-	-	3.05	3.05	-	-	0.04	0.04	3.09	0.08	-	0.35	4.38	4.73	7.90
	2	-	-	3.05	3.05	-	-	0.06	0.06	3.10	0.08	-	0.09	4.65	4.75	7.93
	3	-	-	3.05	3.05	-	-	0.10	0.10	3.15	0.08	-	-	4.83	4.83	8.06
	4	-	-	3.05	3.05	-	-	0.23	0.23	3.28	0.08	0.21	1.14	3.56	4.92	8.28
	Sub-total	-	-	12.19	12.19	-	-	0.44	0.44	12.63	0.34	0.21	1.59	17.42	19.22	32.18
YEAR -3	1	-	-	3.21	3.21	-	-	0.17	0.17	3.38	0.08	0.23	0.98	3.49	4.70	8.17
	2	-	-	3.21	3.21	-	-	0.08	0.08	3.30	0.08	0.31	0.82	3.04	4.17	7.55
	3	-	-	3.21	3.21	-	-	0.06	0.06	3.27	0.08	0.62	0.82	3.03	4.46	7.82
	4	0.02	0.12	3.05	3.19	0.00	0.03	0.08	0.11	3.30	0.08	0.91	0.97	2.73	4.62	8.00
	Sub-total	0.02	0.12	12.69	12.83	0.00	0.03	0.39	0.42	13.25	0.33	2.07	3.59	12.29	17.95	31.53
YEAR -4	1	-	0.18	2.85	3.03	-	0.06	0.09	0.15	3.18	0.08	0.35	0.72	3.45	4.52	7.78
	2	-	-	3.05	3.05	-	0.01	0.06	0.07	3.11	0.08	0.40	0.59	3.55	4.55	7.74
	3	-	0.19	2.84	3.03	-	0.05	0.60	0.65	3.69	0.08	0.70	1.41	4.47	6.58	10.35
	4	-	-	3.05	3.05	-	-	0.41	0.41	3.45	0.08	0.03	0.58	5.63	6.24	9.78

MATUPA'S PROJECT MINING COSTS - Million BRL (R$*1,000,000.00)																
TIME PERIOD		HIGH GRADE ORE (HG)				LOW GRADE ORE (LG)				TOTAL ORE HG + LG	RECLAIM LG/ OXID ORE	WASTE				GRAND TOTAL
BY YEAR ▼	QUARTER ▼	SOIL	SAPRO-LITE	FRESH ROCK	TOTAL	SOIL	SAPRO-LITE	FRESH ROCK	TOTAL			SOIL	SAPRO-LITE	FRESH ROCK	TOTAL	
		R$	R$	R$	R$	R$	R$	R$	R$	R$	R$	R$	R$	R$	R$	R$
	Sub-total	-	0.37	11.79	12.16	-	0.11	1.16	1.27	13.43	0.33	1.49	3.29	17.10	21.88	35.65
YEAR -5	1	-	-	3.05	3.05	-	-	0.20	0.20	3.24	0.08	-	0.02	4.66	4.68	8.01
	2	-	-	3.05	3.05	-	-	0.35	0.35	3.40	0.08	-	-	4.61	4.61	8.09
	3	-	0.02	3.03	3.05	-	0.01	0.33	0.34	3.39	0.09	-	0.23	4.25	4.48	7.96
	4	-	-	3.05	3.05	-	-	0.26	0.26	3.30	0.08	-	-	2.73	2.73	6.11
	Sub-total	-	0.02	12.17	12.19	-	0.01	1.13	1.14	13.33	0.34	-	0.25	16.25	16.50	30.17
YEAR -6	1	-	-	3.05	3.05	-	-	0.10	0.10	3.14	0.08	-	-	0.68	0.68	3.91
	2	-	-	3.05	3.05	-	-	0.08	0.08	3.12	0.08	-	-	0.38	0.38	3.59
	3	-	-	3.05	3.05	-	-	0.12	0.12	3.16	0.08	-	-	0.47	0.47	3.72
	4	-	-	3.05	3.05	-	-	0.30	0.30	3.35	0.08	-	-	0.93	0.93	4.36
	Sub-total	-	-	12.19	12.19	-	-	0.59	0.59	12.78	0.34	-	-	2.47	2.47	15.59
YEAR – 7	1	-	-	2.45	2.45	-	-	0.11	0.11	2.57	0.07	-	-	0.30	0.30	2.95
	2	-	-	-	-	-	-	-	-	-	1.09	-	-	-	-	1.09
	3	-	-	-	-	-	-	-	-	-	0.55	-	-	-	-	0.55
	4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Sub-total	-	-	2.45	2.45	-	-	0.11	0.11	2.57	1.72	-	-	0.30	0.30	4.59
Total		1.02	5.08	75.73	81.83	0.10	0.43	4.16	4.68	86.51	3.71	37.02	35.17	149.84	122.01	212.23

16.4.1 WORKFORCE

As the mine operation is intended to be sub-contracted, the mine labor force is related to management, grade control and mine planning.

The people that need to be contracted are described in Table 16-20.

Table 16-20 - Workforce Requirements for the Outsourced Mining Operation Alternative.

SECTOR	JOB TITLE	FORMATION	QUANTITY
Management	Mine manager	Mining Engineer	1
Grade Control	Department Chief	Geologist Sr	1
	Coordinator	Geologist full	1
	Grade control technician	Mining Technician	2
Mine Planning	Department Chief	Mining Engineer Sr	1
	Mine planning engineer	Mining Engineer full	1
	Topography Specialist	Topographer	1
	Mine planning engineer	Mining Technician	2
Mine Production	Department chief	Mining Engineer Sr	1
	Production Engineer	Mining Engineer	1
	Production Supervisor	Mining Technician	4

16.5 REMARKS

16.5.1 MINING COSTS

As the operational costs are based on outsourced services, and the costs applied come from the real costs experienced by Aura in one similar operation within the company, the operational costs are believed to be realistic and are robust.

16.5.2 GRADE CONTROL ENGINEERING

The reverse circulation using DTH drilling method is the best technology available to assure the right content of gold to concentration planning, the author emphasizes the importance of this technology to help achieve the numbers calculated in this report.

17. RECOVERY METHODS

17.1 PROCESS SUMMARY

The process flow sheet for the Matupá Project was based on the conclusions as described in Chapter 13 and resulted in a robust route, extensively adopted throughout the gold industry. Third campaign testing, described in Chapter 13, resulted in a higher metallurgical performance for the gravity concentration and leaching process alternative, as compared with the direct leaching option. The former process route was thus adopted for the Matupá Project.

The Matupá ore treatment flow sheet consists of a primary crushing circuit, followed by a single stage SAG mill in a closed configuration with hydrocyclones designed to a product with a P_{80} of 0.125 mm. The hydrocyclones combined underflow will be split into two fractions. The first will flow through the gravity concentration stage, whose tailings will flow to the SAG mill feed. The second fraction will flow straight back to the SAG mill feed. The gravity concentration circuit will include a scalp screen, a centrifugal concentrator, and an intensive leaching reactor. The grinding circuit product i.e., the hydrocyclones combined overflow will be directed to a thickener prior to processing in a L-CIL (leaching-carbon in leach) circuit for gold leaching and carbon adsorption. Loaded carbon from CIL circuit will follow elution, electrowinning and smelting for obtaining gold doré. Tailings from the CIL circuit will be directed to cyanide neutralization (Detox) and further dewatered in thickening and filtering circuits for final disposal in tailings piles. Water recovered in the thickener and filtering stages will be recirculated in the industrial processing plant.

The main design criteria adopted in designing the processing circuit are listed as follows:

- Nominal processing rate of 3,562 tpd, which is equivalent to 1.3 Mtpa;
- Crusher circuit availability: 70%;
- Grinding and extraction circuit availability: 92%; and
- Filtration circuit availability: 92%.

17.2 DESCRIPTION OF THE PROCESS PLANT

The selected metallurgical process flow sheet for the industrial processing of the Matupá Project shown in Figure 17-1 is comprised of the following circuits:

- Crushing of run of mine (ROM) ore;
- Storage system for crushed ore:
    - Ore storage silo to feed the grinding circuit;
    - Emergency stockpile to store excess crushed ore;
- Grinding system with SAG mill and hydrocyclones;
- Gravity concentration circuit, including a scalp screen, a centrifugal concentrator, and an intensive leaching reactor. Electrowinning stage dedicated to process the pregnant solution resulting from intensive leaching stage is in analysis;
- Trash screening;

- Pre leaching thickener;
- Leaching and absorption circuit (L-CIL);
- Activated carbon acid wash and Zadra Elution;
- Electrowinning and smelting of doré gold bullion;
- Regeneration of activated carbon;
- Neutralization of residual cyanide present in tailings (SO_2/Air method);
- Carbon safety screening;
- Tailing preparation system for disposal in piles (Dry Stacking), including:
  - Thickening;
  - Storage of thickened pulp;
  - Filtration;
  - Water recirculation system; and
- Reagent storage, preparation, and dosage systems.

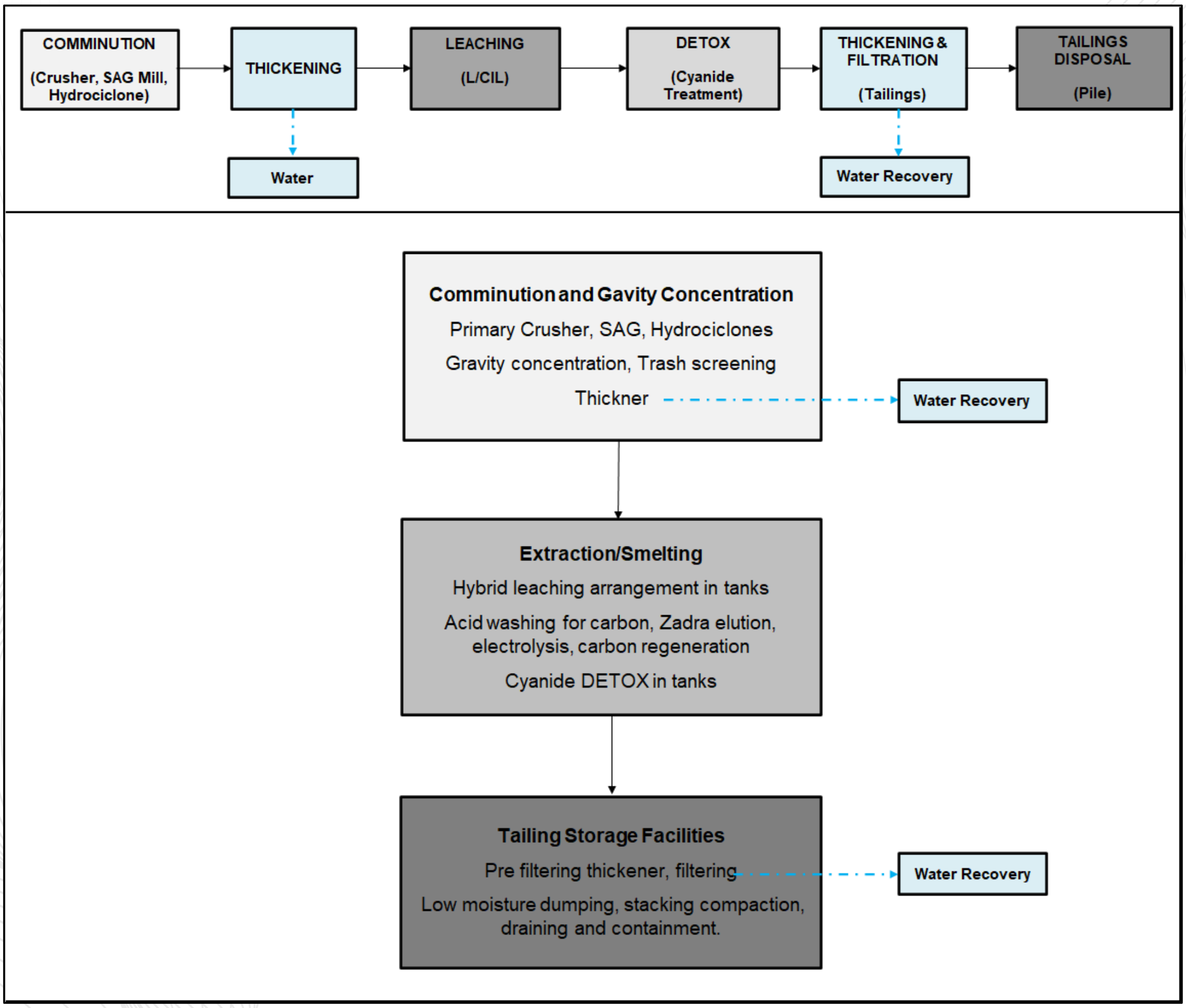


Figure 17-1 - Overall Process Flow Sheet - Matupá Project.

17.2.1 DESIGN CRITERIA

Table 17-1 summarizes the main process and project criteria used for the Matupá Project.

Table 17-1 - Process and Design Criteria for Matupá Project.

Design Parameter	Units	Value
Plant Throughput	t/d	3562
Head Grade Design	gAu/t	1,12
Crushing Plant Availability	%	70
Mill Availability	%	92
BWI Bond Work Index - Fresh Rock	kWh/t	14,5
BWI Bond Work Index - Oxide Ore	kWh/t	16,8 – 18,6
JK (Axb) - 90% Fresh Rock & 10% Oxide Ore		55
Ai Bond Abrasion index - 90% Fresh Rock & 10% Oxide Ore	g	0,31
Primary Crusher		C116 or equivalent
Material Specific Gravity - project	t/m^3	2,66
Angle of Repose	°	37
Ore Moisture Content - project	%	5
SAG Mill Dimensions	mxm	6,55 m diam x 5,18 EGL
SAG Mill Installed Power	MW	3,5
SAG Mill Discharge Density	%w/w	70
SAG Mill Ball Charge	%v/v	18
Primary Grind Size (P80)	µm	125
Gravity Concentrator Recovery - average	% Au from new feed	34,11
Intensive Leaching (ILR) Recovery	% Au from Gravity	98,0
Tail Thickener settling Rate	$t/d/m^2$	13-19
Tail Thickener Flocullant Addition	g/t	10
Thickener Underflow Density	%w/w	50
L / CIL Residence Time	h	28
L / CIL Extraction	Au %	According recovery model
L / CILOperating Density	% w/w	50
L / CIL Dissolved Oxigen Target	mg/L	5~8
L / CIL pH Target	_	10,5~11,0
CIL Carbon Concentration	g/L	< 20
L CIL Sodium Cyanide Addition	g/t	620
L CIL Hydrated Lime Addition	kg/t	1,5
(L / CIL) Tanks	#	2+6
Elution Process		Zadra
Elution Circuit Capacity	t	3
Detox Residence Time	min	120
Detox Oxygen Addition Rate	gO_2:gSO_2	3
Detox Feed Cyanide Concentration (CNwad) - nominal	mg/L	100
Detox Cyanide Discharge Target (CNwad)	mg/L	<2
Detox Copper (Cu2+) Addition	mg/L	50
Detox SO_2 Addition	gSO_2:gCNwad	6
Detox Lime Addition	g(CaO):gSO_2	1
Tail Thickener settling Rate	$t/d/m^2$	13-19
Tail Thickener Flocullant Addition	g/t	25
Tail Thickener Feed Density	% w/w	49
Tail Thickener Underflow Density	%w/w	55-60
Filtering rate	$kg/m^2/h$	1031
Cake Filtering Moisture (Dry basis)	%w/w	22

17.2.2 CRUSHING CIRCUIT

The crushing circuit is designed for a nominal capacity of 3,562 tpd and 70% availability. These figures represent a total of 6,132 annual operating hours, as well as 212 tph crushing circuit nominal throughput.

Run-of-mine (ROM) ore will be hauled and dumped in stockpiles and reclaimed with a front-end loader into the crushing feed hopper, that is equipped with a static grizzly for retaining the oversize material, while a mobile rock breaker is used to break up larger rocks. From the hopper a vibrating grizzly feeder modulates the feeding flow rate, as well as separating the coarse (oversize) and relatively fine (undersize) fractions. The former flows by gravity to the primary jaw crusher chamber, while the latter together with the primary crusher discharge is conveyed to a surge bin. Given that the crushing and milling circuits are designed according to different availabilities, excess crushed material will result when the crushing plant is fully operational. This excess material will be piled in a dedicated stockpile and reclaimed by a front-end loader to a reclaim bin equipped with a vibrating feeder that also feeds the milling circuit. Based on selected ROM size distribution, equipment design and circuit simulations, the predicted crushing circuit P80 is 90 mm.

The main equipment associated with the handling and crushing circuit are as follows:

- ROM hopper equipped with a static grizzly;
- Vibrating grizzly feeder;
- Primary jaw crusher;
- Surge bin;
- Mill vibrating feeders equipped with a variable speed device;
- Suspended conveyor magnet and magnetic detectors; and
- Material handling equipment.

17.2.3 GRINDING CIRCUIT

The single stage griding circuit will include a high-aspect semi-autogenous (SAG) mill operating in a closed configuration with hydrocyclones. The grinding circuit was designed on the basis of feed and product at P_{80} of 90 mm and 0.125 mm respectively. The fresh feed reclaimed from crushing plant surge bin is conveyed to the SAG mill, whose discharge pulp flows to a dedicated trommel screen. The material retained in the trommel screen (pebbles) is conveyed back to the SAG mill feed, whereas the trommel undersize gravitates to an underneath sump, from which it is pumped to a single hydrocyclones nest. The relatively coarse fraction (underflow) will be split into two fractions. The first will flow through the gravity concentration stage, whose tailings will flow to the SAG mill feed. The second fraction will flow straight back to the SAG mill feed. The gravity concentration circuit will include a scalp screen, a centrifugal concentrator, and an intensive leaching reactor. The hydrocyclones nest overflow is the grinding circuit product. Water is added at the SAG mill feed and sump for adjusting the pulp dilution to 70% and 61% w/w, respectively. The hydrocyclones overflow will be directed to a trash screen, whose undersize will flow to a thickener for increasing the concentration of solids to 50%, prior to processing in a L-CIL (leaching-carbon in leach) circuit. The main equipment designed to the grinding circuit is as follows:

- SAG mill;
- Hydrocyclones sump-pumps;

- Hydrocyclones;
- Trash screen;
- Hi-rate thickener;
- Centrifugal concentrator;
- Intensive leaching reactor; and
- Material handling equipment.

17.2.4 GRAVITY CONCENTRATE RECOVERY CIRCUIT

The gravity circuit comprises one centrifugal concentrator complete with a feed scalping screen. Feed to the circuit is directed from the cyclone underflow to the scalping screen. Gravity scalping screen oversize material at +2 mm reports to the gravity tails pump box, from where the gravity tails pump directs the material back to feed the SAG mill.

Scalping screen undersize is fed to the centrifugal concentrator. Operation of the gravity concentrator is semi-batch and the gravity concentrate is collected in the concentrate storage cone and subsequently leached by the intensive cyanidation reactor circuit. The tails from the gravity concentrator also reports to the gravity tails pump box.

The gravity recovery circuit includes the following key equipment:

- Gravity feed scalping screen;
- Gravity concentrator; and
- Gravity tails sump and pump.

17.2.5 INTENSIVE LEACH REACTOR

Concentrate from the gravity circuit is sent to the intensive leach reactor (ILR) to extract the contained gold by intensive cyanidation. The concentrate from the gravity concentrator is directed to the ILR gravity concentrate storage cone and de-slimed before transfer to the ILR.

ILR leach solution (a mixture of NaCN, NaOH and LeachAid® - an oxidant) is made up within the heated ILR reactor vessel feed tank. From the feed tank, the leach solution is circulated though the reaction vessel, then drained back into the feed tank. The leached residue within the reaction vessel is washed, with wash water recovered to the reaction vessel feed tank, and then the solid gravity leach tailings are pumped to the L-CIL circuit.

The ILR pregnant leach solution is pumped from the reaction vessel feed tank to the ILR pregnant solution tank located in the gold room.

ILR pregnant solution is treated in the gold room for gold recovery as gold sludge using a dedicated electrowinning cell. The sludge is combined with the sludge from the carbon elution electrowinning cells and smelted. It can also be smelted separately for metallurgical accounting purposes.

The ILR circuit includes the following key equipment:

- Gravity concentrate storage cone;
- Intensive leaching reactor - ILR;
- ILR pregnant solution tank; and
- ILR electrowinning cell.

17.2.6 PRE-LEACH THICKENING

Trash screen undersize material feeds the pre-leach thickener, which increases the solids concentration to 50% (w/w) prior to the L-CIL circuit. Flocculant is added in emulsion (35% w/v) at a dosage of 10 g/t to the thickener feed to improve solids settling in the thickener. The thickener overflow is reused as process water throughout the plant – mainly at the cyclone feed sump.

The pre-leach thickening circuit includes the following key equipment:

- Pre-leach thickener; and
- Pre-leach thickener underflow sump-pump.

17.2.7 LEACHING AND ADSORPTION CIRCUIT

The leach-adsorption circuit will consist of two leach tanks and six carbon-in-leach (CIL) tanks. The circuit will be fed by the grinding circuit product with a P_{80} of 0.125 mm in a pulp with a solid concentration of 50% w/w.

Air will be sparged to each tank for maintaining a dissolved oxygen level at 5 mg/L required for leaching. Hydrated lime will be added to adjust the operating pH to set point of 10.5. Cyanide solution will be added to the first leach tank according to a dosage of 620 g/t of ore. Mechanical agitation installed in all tanks will maintain the suspension of solids, as well as an adequate reagent homogenization. Fresh and regenerated carbon from the carbon regeneration circuit will be added to the last tank of the CIL circuit at an average concentration of 15-20 g/L of pulp for adequate gold adsorption. Carbon will flow counter-currentto the slurry flow by pumping slurry and carbon. Slurry from the last CIL tank will gravitate to the cyanide detoxification tanks. Once a day, the pulp from the first carbon tank will be pumped into a dedicated screen to separate the gold loaded carbon from the pulp, followed by transferring of the former to the acid washing and elution circuit. After regeneration, the carbon will return to the circuit passing through a dewatering screen. The slurry from the last tank will gravitate to the cyanide detoxification tanks. The leach and carbon adsorption circuit includes the following key equipment:

- Leach/CIL tanks and double impeller agitators;
- Loaded carbon screen;
- Intertank carbon screens;
- Carbon sizing screen;
- Carbon transfer pumps;

- Cyanide concentration and dosage control system;
- Lime dosage and pH control system; and
- Air injection system.

17.2.8 CYANIDE NEUTRALIZATION CIRCUIT

The pulp from the leaching and adsorption circuit will flow by gravity to the cyanide neutralization (Detox) circuit.

The Detox circuit will consist of two tanks with a 60-minute residence time each for reducing weak acid dissociable cyanide (CNwad) from 100-150 mg/L to less than 2 mg/L by using the SO_2/air method.

The required reagents for the Detox process are as follows:

- Sodium metabisulfite (source of SO_2); and
- Copper sulfate pentahydrate (source of copper ions - reaction catalyst) and hydrated lime to adjust pH and o Eh required for the reaction.

A sufficient amount of air (O_2 source) will be injected in spargers into the tanks for reaction purposes. Tanks will be equipped with agitators to ensure that the oxygen and reagents are thoroughly mixed with the tailing.

The pulp from the neutralization circuit will flow to the safety screen to retain any loaded carbon, which will be stored for recirculation in the CIL circuit. The screen undersize will be pumped to the tailing thickener.

The Detox circuit will include the following key equipment:

- Neutralization tanks equipped with double impeller agitators;
- Air injection system;
- In-line samplers for measuring the CNwad level at both the entrance and exit of the circuit, thus establishing the dosage of reagents and the efficiency of the treatment;
- Carbon safety screen; and
- Tailing box and pumps.

17.2.9 CARBON ACID WASH, ELUTION, ELECTROWINNING AND SMELTING

The loaded carbon from the CIL circuit will be repulped and pumped to an acid washing column, to remove calcium adsorbed by the carbon. Such a process occurs in the column using a 3% w/v hydrochloric acid solution. The acid solution resulting from a further rinsing stage will be pumped to the Detox circuit.

After the acid wash and rinsing, carbon will be transferred to the elution column. The selected elution method is the Zadra, which was selected on the basis of the available water quality. For this method, the eluent solution will be prepared at 1% Sodium Hydroxide (NaOH) and 0.1% Sodium Cyanide (NaCN). The resulting solution will be heated up to 150°C and injected at the bottom of the elution column.

The amount of heated water injected into the column will correspond to a volume equivalent of 8 to 10 times the corresponding volume of coal BV - Bed Volume. The eluted solution will be pumped to the pregnant solution tank for feeding the electrowinning stage, whose solution will be recirculated to the eluent tank through heat exchangers and tanks. At the end of each elution cycle, a third of the electrowinning will be transferred to the L-CIL circuit.

The sludge gold-rich cathodes will be washed, filtered, and dried. The dry material obtained will be mixed with smelting fluxes (borax, nitrate, carbonate, and silica) and smelted in a liquefied petroleum gas (“LPG”) furnace at 1,100°C to produce gold doré (bullion).

The key equipment related to carbon acid wash, elution, electrowinning, and smelting are as follows:

- 3-tonne capacity acid wash column;
- 3-tonne capacity elution column;
- Heating and heat exchange system;
- Pregnant and barren solution tanks;
- Water and solution pumping systems;
- Carbon transfer system;
- Electrowinning cell;
- Rectifier device for electrowinning;
- Filter, drying oven and flux mixer;
- Liquefied petroleum gas (LPG) smelting furnace with bullion moulds and slag handling system; and
- System for exhaustion and washing of gases and metallic fumes.

17.2.10 CARBON REACTIVATION

After the elution process the carbon will be reactivated in a rotary kiln. A screening stage will dewater and separate carbon at 1 mm mesh. The oversize fraction will be stored and fed to the kiln, while the undersize material will be stored for further treatment.

The rotary kiln will reach 750°C at an atmosphere of superheated steam to restore the activity of the carbon. Kiln discharge will be quenched and screened, any oversize material will be directed to the CIL circuit, while the undersize will be stored and sold. The new carbon will be added to the circuit in the same conditioning tank and transfer system as the regenerated carbon. The key equipment related to carbon reactivation circuit is as follows:

- Carbon dewatering screen;

- Carbon kiln including hopper and screw feeder;
- Carbon quench screen;
- Carbon quench tank; and
- Pumping system for transferring carbon to the leaching circuit.

17.2.11 TAILING THICKENING AND FILTERING CIRCUIT

Tailings resulting from the Detox circuit with a solids concentration of 48-49% w/w will be transferred to a high-rate thickener. Further dilution occurs in the thickener feedwell where flocculant will be added in a 35% p/v emulsion at a concentration of 25 g/t.

The 60% w/w pulp in the thickener underflow will be pumped to the filtering circuit. Due to different availabilities stipulated to grinding/leaching/thickening and filtering circuits, a dedicated tank will be installed to receive the thickened pulp for equalizing the daily basis operation.

The filtration circuit will include a horizontal vacuum filter for reducing the cake moisture to 21-23% (dry basis). Filtering water, together with thickening resulting water will be recirculated within the processing plant, whereas the filtered product will be transferred to disposal piles. Water runoff from piles will also be recirculated in the processing plant.

The key equipment related to thickening and filtering circuits are as follows:

- High-rate thickener;
- Thickened slurry tank for filtering circuit feed;
- Thickened slurry transfer pump;
- Horizontal vacuum filter;
- Reclaimed water tank;
- Pump for transferring water to plant; and
- Pond to store runoff/percolation water from the tailing piles.

17.3 REAGENT HANDLING, STORAGE AND PREPARATION SYSTEMS

The following reagent systems are required for Matupá processing circuit:

- Sodium cyanide;
- Hydrated lime;
- Sodium hydroxide;
- Hydrochloric acid;

- Sodium metabisulfite;
- Copper sulfate pentahydrate;
- Flocculant;
- Activated carbon;
- Anti—scalant; and
- Smelting fluxes.

Reagents will be received and stored in appropriate facilities. Each reagent will be prepared in accordance with occupational/environmental safety standards, preventing incompatible reagents from mixing. 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, eyewash stations, and Material Safety Data Sheet (MSDS) stations will be located throughout the facilities. Sumps and sump pumps will be provided for spillage control.

17.3.1 SODIUM CYANIDE

The solid briquetted/powdered 98% sodium cyanide (NaCN) will be received in 1-t reagent bags and stored in the warehouse following the International Cyanide Management Institute – ICM guidelines.

The reagent will be transferred to the preparation area, where it will be diluted in a basic solution to a 20% w/w solution in a stirred tank. The diluted solution will be transferred to the cyanide dosage tank using a transfer pump. Dedicated pumps will distribute the solution to such an installation, the latter equipped with an exhausting system to remove particulate solids dispersed in air. The resulting solution will be pumped to the leach/adsorption and elution circuits.

17.3.2 HYDRATED LIME

Hydrated lime ($Ca(OH)_2$), will be used to ensure the basic pH required for the reactions occurring in the leaching/adsorption.

The granular solid reagent will be received in dedicated trucks and stored in the warehouse. The solid reagent will be diluted in the mixing/storage agitated tank to a 25% w/w solution, then pumped through distributing points to the leaching/adsorption, cyanide destruction (Detox), and treating facilities of effluent generated in waste, low grade, and tailing piles.

17.3.3 SODIUM HYDROXIDE

Sodium hydroxide (NaOH) will be used to pH adjusting in the: (a) preparation of sodium cyanide solution and (b) elution and electrowinning processes.

The 50% w/w solution of sodium hydroxide will be received in 1,000 L IBC, stored in the warehouse and pumped to the dosing area from which the solution will be directed to the consumption points.

17.3.4 HYDROCHLORIC ACID

Hydrochloric acid (HCl) will be used in carbon washing, prior to the elution process. The main purpose of acid washing is to remove Calcium (Ca) ions from carbon.

The 33% solution of hydrochloric acid will be received in tank trucks and stored in the warehouse. From the warehouse the reagent will be transferred to the dosing area, from which the solution will be directed to the acid washing by a positive displacement pump.

17.3.5 SODIUM METABISULFITE

Sodium metabisulfite - NaMBT or SMBS ($Na_2S_2O_5$) is used in the cyanide detoxification (Detox) as a source of SO_2.

The 97.5% solid SMBS will be supplied in 1 t bags and stored in the warehouse. The reagent will be transferred to the preparation area, where it will be diluted to a 20% w/w solution in a stirred tank. This installation will be equipped with an exhaust system to remove particulate solids dispersed in air. The resulting solution will be pumped to a dosing tank, from which it will be pumped to the Detox circuit.

17.3.6 COPPER SULFATE PENTAHYDRATE

The 99.5% solid copper sulfate pentahydrate ($CuSO_4.5H_2O$) will be supplied in 500 kg plastic bags for using in the Detox system.

From the warehouse the reagent will be transferred to the preparation area, where it will be diluted to a 20% w/w solution in a stirred tank. The resulting solution will be pumped to a dosing tank, from which it will be pumped to the Detox circuit.

17.3.7 FLOCCULANT

Flocculant will be received in 950 L IBC as a 35% w/w emulsion and stored in the warehouse. From the warehouse the reagent will be transferred to the preparation area, where it will be diluted and pumped to high-rate thickeners.

17.3.8 ACTIVATED CARBON

Solid granular activated carbon will be received in 0.5 t bulk bags. The fresh carbon will be transferred to the carbon quench tank, or directly to the final CIL tank.

17.3.9 ANTI-SCALANT

Anti-scalant solution will be received in 1 m^3 containers and stored in the warehouse. From the storage area the containers will be transferred to tanks, from which the solution will be dosed by positive displacement pumps to addition points.

17.3.10 SMELTING FLUXES

Gold smelting fluxes such as borax, silica, sodium nitrate and sodium carbonate will be delivered in small bags for storing in the warehouse, from there they will be transferred for use in the gold smelting process.

17.4 UTILITIES AND WATER

17.4.1 SERVICES, INSTRUMENTS AND PROCESS AIR

17.4.1.1 PROCESS AND INSTRUMENT AIR

High-pressure air at 700 kPag will be produced by compressors to meet plant needs. Drying devices will be installed for supplying the instrument air demand.

17.4.1.2 PROCESS AIR

Industrial air blowers will generate the necessary air flow at 400 kPag to the L-CIL and Detox circuit tanks. The purpose of air injecting into the tanks is oxygen supplying to the cyanide gold complexation reaction, as well as in the cyanide neutralization.

17.4.2 WATER SUPPLY

17.4.2.1 RAW WATER

An estimated 25 m^3/h of raw water will be consumed in the processing plant. Such a flowrate will be pumped from the Porcão River, which is located close to the future industrial installations. The raw (make-up) water characteristics of low salinity, low conductivity, and low particulate solids are particularly suited to be used in the following applications:

- Pump sealing;
- Mill system sealing;
- Cooling the mill's hydraulic system;
- Preparing reagents;
- Carbon acid washing and elution;
- Safety devices such as eye washing;
- Dust suppression;
- Sprinkler and hydrant fire preventing systems; and
- Safety showers and other similar applications.

17.4.2.2 RECIRCULATE WATER

The majority of water consumed in the processing plant is designed to derive from recirculation within the industrial installation. Accordingly, a calculated 206 m^3/h water flow rate obtained from tailing thickening and filtering will recirculate to supply mainly the grinding stage.

17.5 PROCESS FLOW SHEETS

Process flow sheets adapted to the Matupá processing plant are described in Figure 17-2 to Figure 17-4.

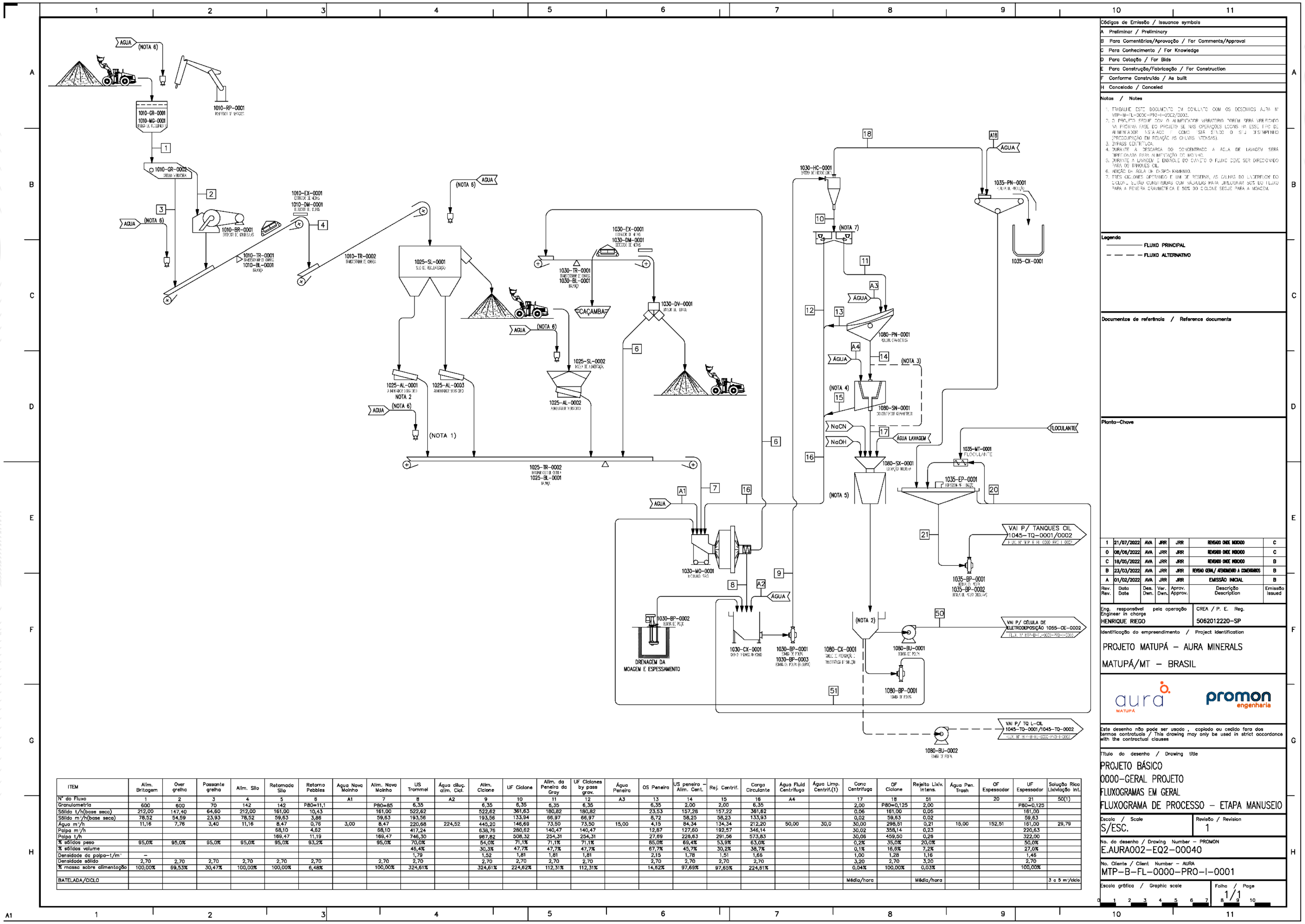
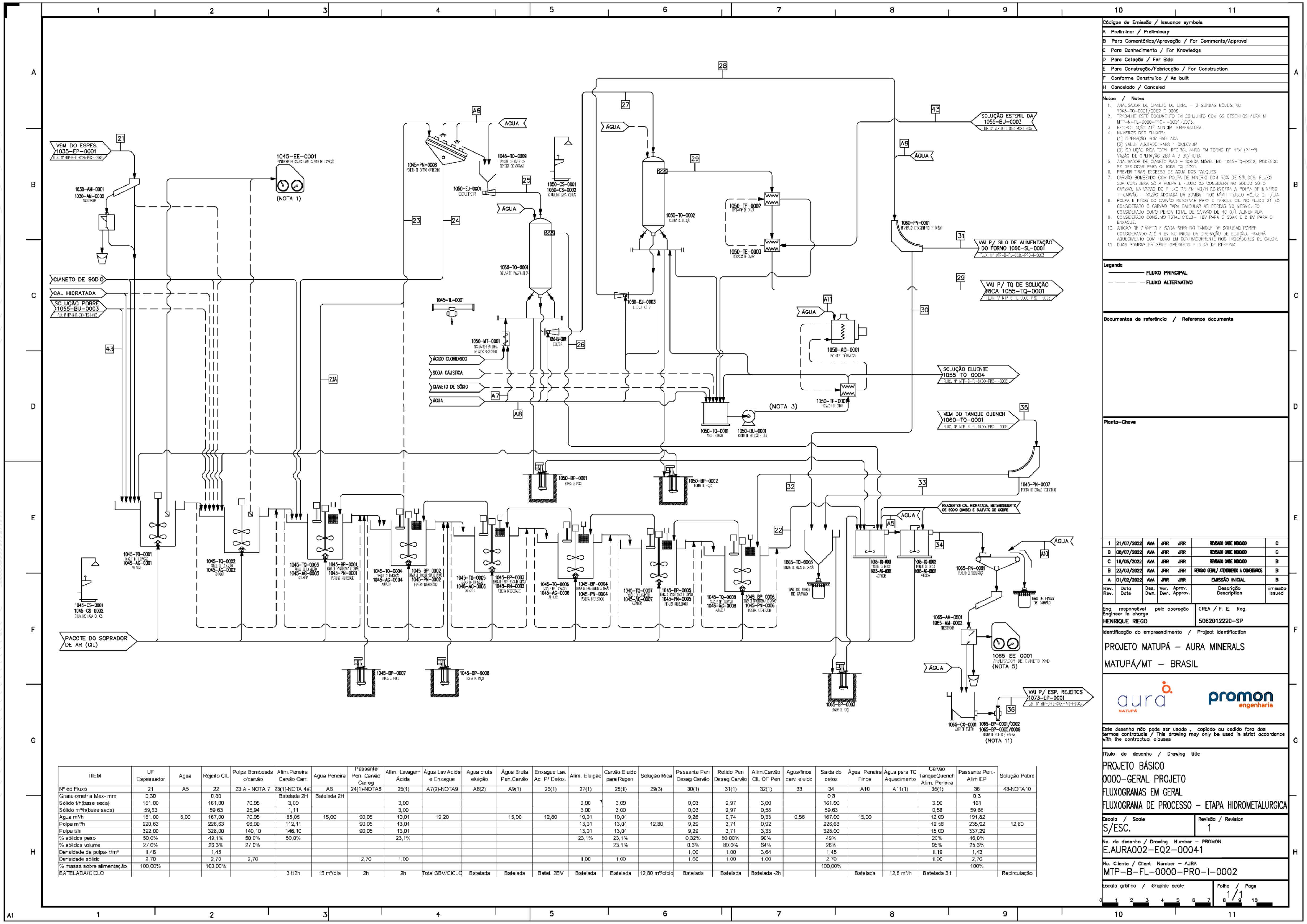


Figure 17-2 - Process Flow Sheet Part I - Matupá Project (MTP-B-FL-0000-PRO-I-0001 - Promon).

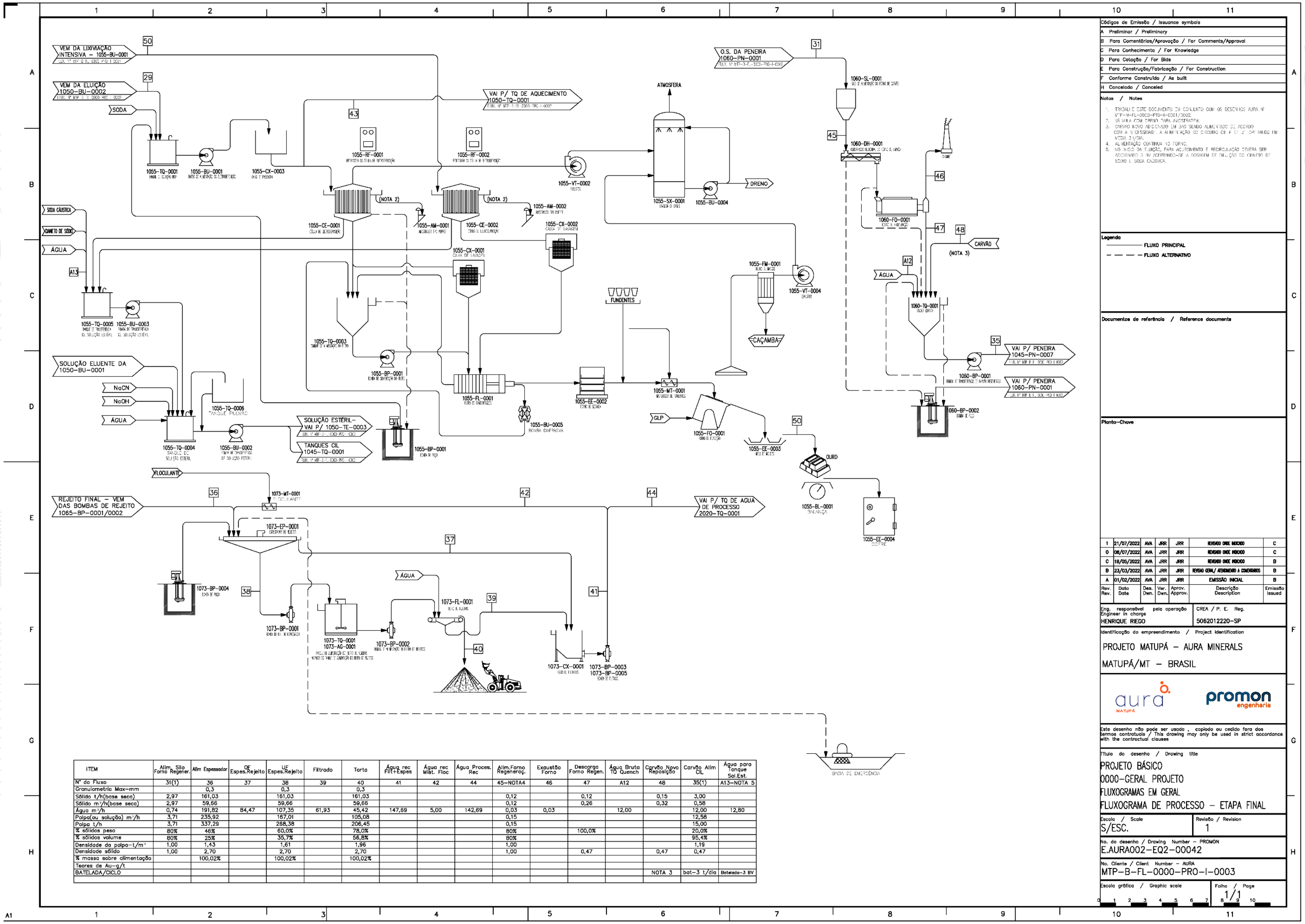


ITEM	UF Espessador	Agua	Rejeito CIL	Polpa Bombeada c/carvão	Alim.Peneira. Carvão Carr.	Agua Peneira	Passante Pen. Carvão Carreg	Alim. Lavagem Ácida
Nº do Fluxo	21	A5	22	23 A - NOTA 7	23(1)-NOTA 4e7	A6	24(1)-NOTA8	25(1)
Granulometria Max- mm	0,30		0,30		Batelada 2H	Batelada 2H		
Sólido t/h(base seca)	161,00		161,00	70,05	3,00			3,00
Sólido m³/h(base seca)	59,63		59,63	25,94	1,11			3,00
Água m³/h	161,00	6,00	167,00	70,05	85,05	15,00	90,05	10,01
Polpa m³/h	220,63		226,63	96,00	112,11		90,05	13,01
Polpa t/h	322,00		328,00	140,10	146,10		90,05	13,01
% sólidos peso	50,0%		49,1%	50,0%	50,0%			23,1%
% sólidos volume	27.0%		26,3%	27,0%				
Densidade da polpa- t/m³	1,46		1,45					
Densidade sólido	2,70		2,70	2,70			2,70	1.00
% massa sobre alimentação	100,00%		100,00%					
BATELADA/CICLO					3 t/2h	15 m³/dia	2h	2h

ITEM	Água Lav Acida e Enxague	Agua bruta eluição	Água Bruta Pen.Carvão	Enxague Lav. Ac P/ Detox.	Alim. Eluição	Carvão Eluido para Regen.	Solução Rica	Passante Pen Desag Carvão	Retido Pen Desag Carvão
Nº do Fluxo	A7(2)-NOTA9	A8(2)	A9(1)	26(1)	27(1)	28(1)	29(3)	30(1)	31(1)
Granulometria Max- mm									
Sólido t/h(base seca)					3,00	3,00		0,03	2,97
Sólido m³/h(base seca)					3,00	3,00		0,03	2,97
Água m³/h	19,20		15,00	12,80	10,01	10,01		9,26	0,74
Polpa m³/h					13,01	13,01	12,80	9,29	3,71
Polpa t/h					13,01	13,01		9,29	3,71
% sólidos peso					23,1%	23,1%		0,32%	80,00%
% sólidos volume						23.1%		0,3%	80.0%
Densidade da polpa- t/m³								1,00	1,00
Densidade sólido					1,00	1.00		1.60	1.00
% massa sobre alimentação									
BATELADA/CICLO	Total:3BV/CICLO	Batelada	Batelada	Batel. 2BV	Batelada	Batelada	12,80 m³/ciclo	Batelada	Batelada

ITEM	Alim.Carvão CIL OF Pen	Agua/finos carv. eluido	Saida do detox	Água Peneira Finos	Água para TQ Aquecimento	Carvão TanqueQuench Alim. Peneira	Passante Pen.-Alim EP	Solução Pobre
Nº do Fluxo	32(1)	33	34	A10	A11(1)	35(1)	36	43-NOTA10
Granulometria Max- mm			0,3				0,3	
Sólido t/h(base seca)	3,00		161,00			3,00	161	
Sólido m³/h(base seca)	0,58		59,63			0,58	59,66	
Água m³/h	0,33	0,56	167,00	15,00		12,00	191,82	
Polpa m³/h	0,92		226,63			12,58	235,92	12,80
Polpa t/h	3,33		328,00			15,00	337,29	
% sólidos peso	90%		49%			20%	46,0%	
% sólidos volume	64%		26%			95%	25,3%	
Densidade da polpa- t/m³	3,64		1,45			1,19	1,43	
Densidade sólido	1.00		2,70			1,00	2,70	
% massa sobre alimentação			100,00%				100%	
BATELADA/CICLO	Batelada -2h			Batelada	12,8 m³/h	Batelada 3 t		Recirculação

Figure 17-3 - Process Flow Sheet Part II - Matupá Project (MTP-B-FL-0000-PRO-I-0002 - Promon).



ITEM	Alim. Silo Forno Regener.	Alim Espessador	OF Espes.Rejeito	UF Espes.Rejeito	Filtrado	Torta	Água rec Filt+Espes	Água rec Mist. Floc	Água Proces. Rec	Alim.Forno Regeneraç.	Exaustão Forno	Descarga Forno Regen.	Água Bruta TQ Quench	Carvão Novo Reposição	Carvão Alim CIL	Água para Tanque Sol.Est.
N° do Fluxo	31(1)	36	37	38	39	40	41	42	44	45–NOTA4	46	47	A12	48	35(1)	A13–NOTA 5
Granulometria Max–mm		0,3		0,3		0,3										
Sólido t/h(base seca)	2,97	161,03		161,03		161,03				0,12		0,12		0,15	3,00	
Sólido m³/h(base seca)	2,97	59,66		59,66		59,66				0,12		0,26		0,32	0,58	
Água m³/h	0,74	191,82	84,47	107,35	61,93	45,42	147,69	5,00	142,69	0,03	0,03		12,00		12,00	12,80
Polpa(ou solução) m³/h	3,71	235,92		167,01		105,08				0,15					12,58	
Polpa t/h	3,71	337,29		268,38		206,45				0,15					15,00	
% sólidos peso	80%	46%		60,0%		78,0%				80%		100,0%			20,0%	
% sólidos volume	80%	25%		35,7%		56,8%				80%					95,4%	
Densidade da polpa–t/m³	1,00	1,43		1,61		1,96				1,00					1,19	
Densidade sólido	1,00	2,70		2,70		2,70				1,00		0,47		0,47	0,47	
% massa sobre alimentação		100,02%		100,02%		100,02%										
Teores de Au–g/t																
BATELADA/CICLO														NOTA 3	bat–3 t/dia	Batelada–3 BV

Figure 17-4 - Process Flow Sheet Part III - Matupá Project (MTP-B-FL-0000-PRO-I-0003 - Promon).

18. PROJECT INFRASTRUCTURE

18.1 OVERALL SITE

The overall site plan (see Figure 18-1) shows the major project facilities, including the open pit mines, tailings management facility (TMF), waste rock facilities, mine services, and access roads. Access to the facility is from the east side of the property from the existing access road. Main access will be via the security gate near the process plant.

The site will be fenced to deter access by unauthorised people. The process plant is located east of the X1 Deposit.

Site selection took into consideration the following factors:

- locate mining, administration and processing plant staff offices close together to limit walking distances between them; and
- locate the ready line close to the mining admin/office area and changehouse.

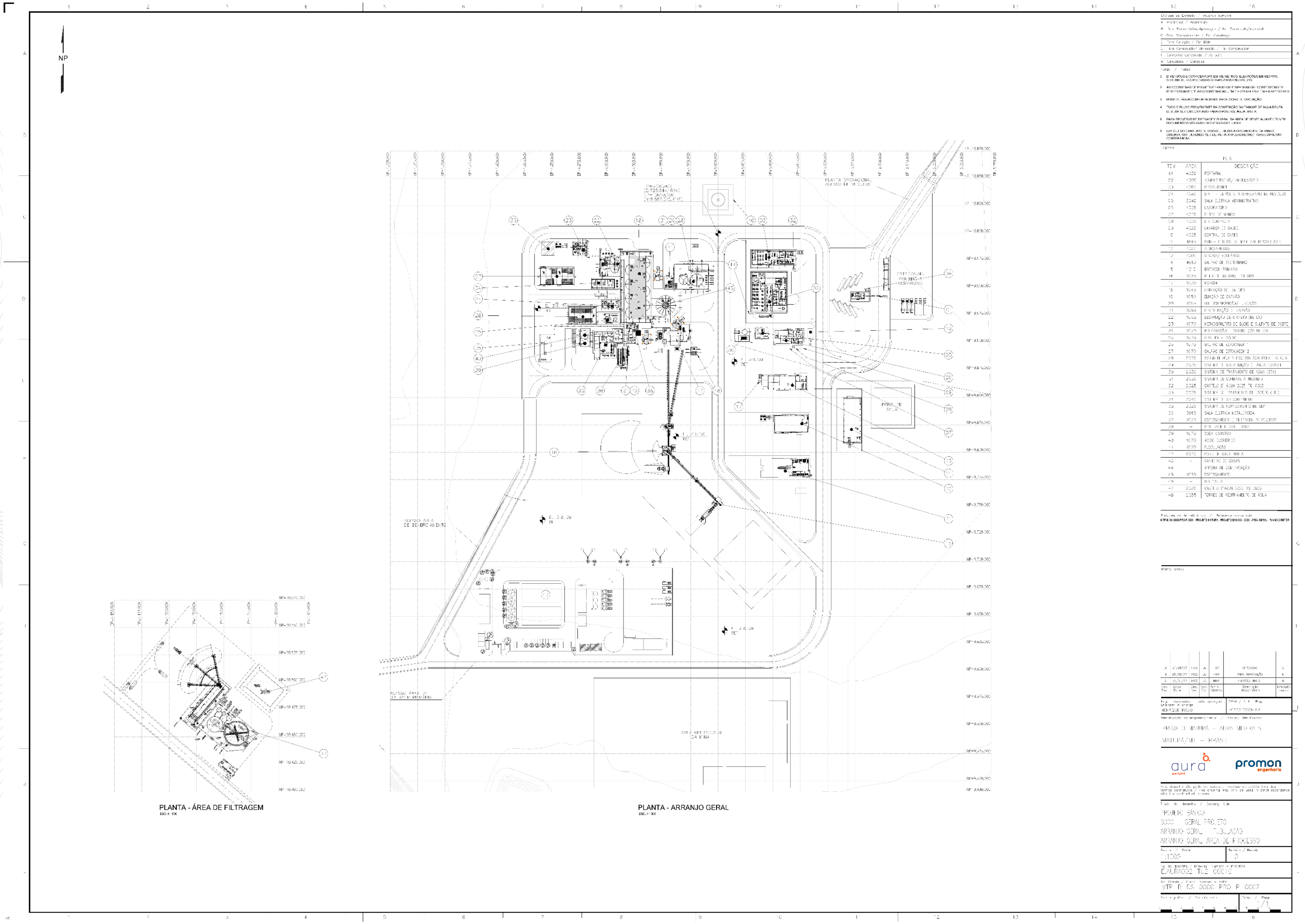


Figure 18-1 - Overall Site Plan (MTP-B-DS-0000-P-0002-RB)

18.2 ROADS

18.2.1 ACCESS TO SITE

From the municipality of Matupá (state of Mato Grosso, Brazil) to site, access is via 12 km of road (BR-163), as shown in Figure 18-2. For the Project, a new access road to the to the mine site from the road is recommended.

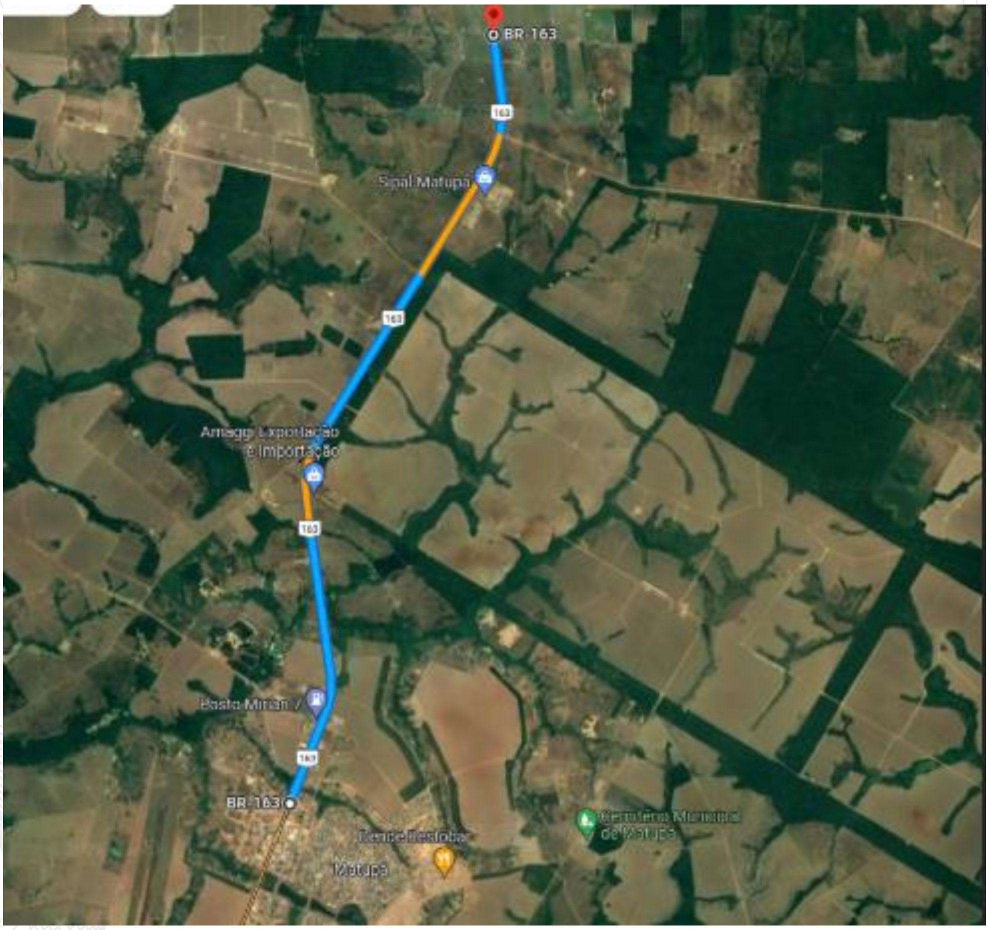


Figure 18-2 - Site Access (Google Maps).

18.2.2 PLANT SITE ROADS

The plant internal accesses are approximately 8 and 10 m wide, designed using primary covering, drainage, and appropriate signage.

On the sides where there is risk of vehicles falling, barriers will be built with a minimum height of half the diameter of the largest vehicle tire that will use that access.

The internal roads will allow access between the administrative and operational installations, construction site, beneficiation plant, crushing area, mine pit, waste deposit and low-grade stockpile. See Figure 18-3.

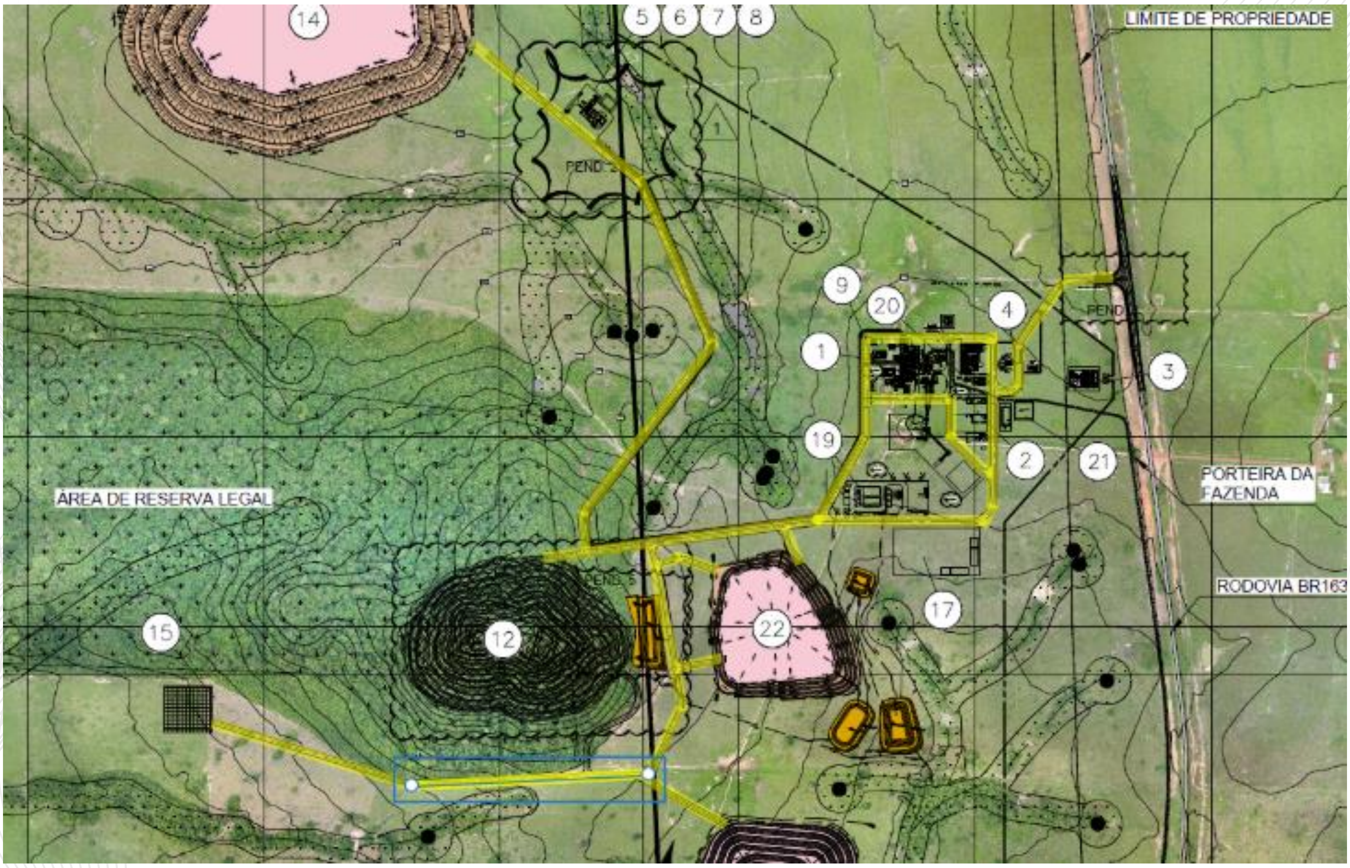


Figure 18-3 - Internal Accesses (MTP-B-DS-0000-PRO-P-0001-R1).

18.3 POWER SUPPLY

18.3.1 ELECTRICAL POWER SOURCE

Power will be provided from a new sectioning sub-station, which will section the existing 138 kV Transmission line Matupá – Guarantã. The 138 kV Transmission Line crosses the area designed for the process plant, therefore, it will be necessary to adapt the header point of the new LT sections.

The new sectioning sub-station will be located on the east of the process plant, close to the administrative area. The project's main substation will be positioned adjacent to the sectioning substation and will be responsible for transforming the voltage level from 138 kV to 13.8 kV.

The sectioning sub-station and the headers of the Transmission Line will be built by a package contracted directly by ENERGISA that will be responsible for engineering, environmental licensing, construction, and commissioning. This will reduce the deadlines as it is a Public Utility construction and as it is built by the local energy concessionaire.

The 138/13.8 kV substation site will contain an incoming structure and isolation switch, main circuit breaker, provision for utility metering, bus work to deliver 138 kV power to a 10/12.5 MVA stepdown transformer complete with primary circuit breaker, and isolating switches. This transformer will feed associated secondary switchgear and is arranged to provide 13.8 kV power to the main processing plant, the filtering plant, the Administration Area, and other remote areas. Provision is included for automatically switched capacitor banks to assist with site power factor correction. The electrical power for the raw water supply area will be provided directly by the local energy concessionaire, at a level of 13.8 kV.

18.3.2 ELECTRICAL DISTRIBUTION

The primary distribution voltage will be radial, at 13.8 kV, three phase, 60 Hz, from the main substation.

Feed distribution from the main substation will be via three-phase powerlines and power poles and underground conduits for the secondary substations. Distribution from the secondary substations to the loads and panels in the field will be via cable rack and conduits, as required.

The conventional three-phase powerlines and power poles network will be supplied as a turn-key, including pole-mounted transformers.

18.3.3 MAIN SUBSTATION

The main substation will include an electrical room and the associated high-voltage equipment. The substation will have a 10/12.5 MVA ONAN/ONAF transformer from 138 to 13.8 kV.

18.3.4 SECONDARY SUBSTATIONS

Site electrical supply was selected and designed around the major load centers as shown in Table 18-1.

Table 18-1 - Plant Substations.

TAG NUMBER	TYPE	CHARACTERISTICS	POWER DISTRIBUTION FROM MAIN SE
3015-SE-0001 (Metallurgy)	E-room	Feed: 13.8 kV-25 kA Process loads: 480 V-50 kA Lighting: 380/220 V-50 kA	Conventional aerial network - 380 m
3060-SE-0001 (Filtering)	E-room	Feed: 13.8 kV-25 kA Process loads: 480 V-50 kA Lighting: 380/220 V-50 kA	Conventional aerial network – 2,600 m
3040-SE-0001 (Administrative)	E-room	Feed: 13.8 kV-25 kA Process loads and lighting: 380/220 V-50 kA	Conventional aerial network - 320 m
3050-SE-0001 (Raw water capture)	Skid	Feed: 13.8 kV Process loads and lighting: 380/220 V-50 kA	Directy Feed by the local energy concessionaire.

The substations will feed the following areas:

- 3015-SE-0001: Grinding, thickening, gravity, leach, detox, elution and electrowinning, reagents, compressed air system, Primary crushing, stockpile/surge bin and water distribution systems:
- 3040-SE-0001: Administrative buildings and laboratory:
- 3050-SE-0001: Raw water capture system: and
- 3060-SE-0001: Waste Filtering system.

18.3.5 EMERGENCY POWER

Three diesel generators will be provided to feed critical process loads, administrative buildings, and security systems. Each diesel generator is located near the designated electrical room and will be connected to the adequate motor control center.

18.4 SUPPORT BUILDINGS

The Figure 18-4 shows a 3-D image of the process plant.

Figure 18-4 - Rendered Image of the Process Plant (Promon, 2022).

18.4.1 PRIMARY CRUSHING AREA

The primary crushing area will be located south of the process plant. The crushing stage will be composed of a mobile crushing skid, containing a vibrating grizzly feeder, a primary jaw crusher, chutes, platework and a discharge conveyor. The process equipment will be serviced by mobile cranes as required.

18.4.2 GRINDING AREA

The grinding area will be unenclosed and includes the SAG mill, classification cyclones, cyclone feed hopper and pumps, trash screen and gravity circuit equipment, also including a liner handler.

The grinding building will be an 18 m (long) x 32 m (wide) steel structure building with a ground floor, one elevated concrete floor, and multiple equipment access platforms. The process equipment will be serviced by a 4-tonne hoist. Any heavier loads need to be serviced by the mobile crane.

18.4.3 LEACH AND DETOX AREAS

The L-CIL/elution area will be 65.5 m (long) x 28.5 m (wide) and will include two 10-m diameter leach tanks and six 10-m diameter CIL tanks, including tank platforms, and the area is completely limited by a containment bund with a volumetric capacity equivalent to 110% of the largest tank contained. There is a separate structure in the area for screen maintenance purposes. The area will be serviced by a 7.5-tonne hoist on a monorail to access the tank pumps, and screens. For agitator maintenance, a mobile crane will be required.

To the west of the L-CIL tanks is the detoxification and tailings area. This area will include two 7-m diameter detoxification tanks and will be 30 m (long) x 12 m (wide). It will also include the tailings hopper and pumps to the TMF.

18.4.4 GOLD ROOM

The gold room will be an 18 m (long) x 14.5 m (wide) two-storey, pre-cast concrete building that will house the electrowinning cells, sludge hopper/filter, drying oven, furnace, vault, and security room, complete with a five-tonne monorail. It will be located in a fenced area with restricted access that also encompasses the pregnant solution tank and its containment bund.

18.4.5 REAGENT AREAS

The reagent preparation and storage systems are separate around the plant as per the location of dosage. The lime system area is 17 m (long) x 9 m (wide). The sodium cyanide preparation and storage area is a fenced area with restricted access. The containment bund and equipment area is 15 m (long) x 7.5 m (wide). The flocculant is also separate from the other reagents, to be closer to the thickener and minimize piping, in an area that is 10.5 m (long) x 10 m (wide). The area for the detox reagents (i.e., sodium metabisulfite and copper sulfate) is next to the detox tanks, to the east of the leach tanks, and the contained area is 12.5 m (long) x 12 m (wide).

18.4.6 MINE SUPPORT AREA / TRUCK SHOP / TRUCK WASH

The operation of the mine will be outsourced, so it is not foreseen by the project's engineering team to build a mine support structure.

In the contract for the outsourcing of the mine operation, it will be stated that the contractor builds its own necessary support structure. This will allow the contracted company to adapt the facilities according to the size of the equipment in its fleet.

Aura will supply water and electricity at the contracted company's facilities.

18.4.7 WASTE MATERIAL WAREHOUSE

The Disposable Materials Center will be a shed of 225 m² in metal structure, masonry and metal covering, for storage of hazardous waste and waste classes I and II, separated by bays. The shed will be accessed from both sides, by forklifts or electric pallet racks. Class II waste shall be pressed, stacked, and stored in bales on pallets. Hazardous waste will be packed in drums on pallets and hospital and laboratory supplies will be in capped containers.

18.4.8 WAREHOUSE

The Warehouse will be a shed in metallic structure, masonry and metal covering, of approximately 561.70 m², surrounded by a patio with approximately a 2,366.30 m² area. The shed will have an anteroom for dispatching materials and an office with two jobs.

18.4.9 MAINTENANCE SHOPS AND CHANGEROOM

The Maintenance Workshop will be a building in metal structure, masonry and metal covering, with 541.90 m² of area for boxes for the boiler, machining, mechanical and electrical, as well as tooling area for storage of lubricants, air compressors and washing of parts. Attached to the workshop, there will be a building on two floors, with 124.40 m² each, covering the men's and women's changing rooms on the ground floor, with capacity for 96 and 18 bins/lockers, respectively.

On the upper floor, on the changing rooms, will be the workshop office, with 10 jobs, a room for shift technicians, with 2 posts, the instrumentation laboratory, a canopy, and a meeting room.

All floor drainage will be directed to an SAO (Water/Oil Separator).

18.4.10 CORE SHED

The Shed of Testimonies and Geology will have 592.70 m² and will be built in metallic structure, masonry and metal covering. It will have an office for technicians and geologists, with two jobs and accesses on both sides of the shed, through sliding gates.

18.4.11 STORAGE SHED - REAGENTS

The Reagent Storage Shed will be a building in metal structure and closure in masonry and metal tiles, with approximately 985.30 m² of area for storage of bags of hydrated lime, flocculant, activated carbon, anti escalant, copper sulfate and sodium metabisulfite, as well as areas for storage of sodium hydroxide and hydrochloric acid, in tot bins.

18.4.12 STORAGE SHED - CYANIDE

The Cyanide Storage Shed will be a building in metal structure and masonry closure, with approximately 383.00 m² of area for storage of cyanide bags. The shed area will be surrounded by.

18.4.13 EXPLOSIVES STORAGE AND HANDLING

The mine explosive magazines will be located in an isolated area away from the process plant and the pit entrance. The buildings will be located approximately 200 m apart. Each building will have a 3.5 m high compacted earth berm on three sides. The fourth side of each building will be open to allow access.

Aura will adopt the same method as carried out at the Apoena Unit, which is a subsidiary of Aura Minerals in the state of Mato Grosso in Brazil, where one company was contracted to set up an emulsion factory and another company was contracted to set up the explosives and accessories storage facilities according to NORMA R105 (Regulation for Inspection of Controlled Substances) issued by the Brazilian Ministry of Defence.

It will be necessary to set up the emulsion factory with storage tanks, an explosives magazine, and an accessories magazine. The installations will be in independent masonry buildings.

The entire area will be security fenced with a guard post at the entrance.

18.4.14 FUEL STATION

As informed in the previous section 18.4.6, the entire mine support infrastructure, including the Fuel Station will be contracted to contractors. The contracted company must also supply the fuel tanks to supply its equipment.

Aura's mobile equipment will be supplied by a mobile supply train with fuel supplied by the gas stations in the city of Matupá.

Vehicles fuel supply will also take place at gas stations in the city of Matupá.

18.4.15 PLANT ADMINISTRATION BUILDING

In the administrative plateau will be located the buildings that will house the Main Ordinance, the Administrative Building (where the Office, the Ambulatory / Fire Brigade, the Control Room, the Training Room and the Living Area, as well as toilets and canopy that serves the entire area will be located), and the Restaurant.

18.4.16 MAIN GATEHOUSE

The Main Ordinance will be a building of approximately 139.40 m², containing a waiting room / training with accessible male and female sanitary facilities and area with bins/lockers for guarding the belongings of visitors and suppliers, reception with two security guards and a supervisor, pantry and magazine room.

18.4.17 ADMINISTRATIVE BUILDING

The Administrative Building will be a building of 1,270.90 m² of approximate area, in a U shape. On one side will be the Administrative Office and the Control Room, with 52 jobs and meeting rooms. On the other side, there will be a Training Room for 34 people, a room for HSEC with 16 jobs, and the Outpatient Clinic, composed of an individualized office, an immunization room, a nursing center, a collective room, areas for purging and hygiene of patients, as well as an area for keeping stretchers and wheelchair, and a garbage dump, with external access. There will also be a covered area for the ambulance shelter and a fire brigade's personal protective equipment (PPE) depot. Connecting the two buildings, in the center of the "U", there will be a central square and a covered area, which will be the Living Area, in addition to common areas such as male, female toilets and one for people with disabilities, canopy and a storage room of cleaning material (DML).

18.4.18 MESS HALL

The Mess Hall building will consist of the following areas:

- Industrial kitchen, with a receiving area for control of supplies, food storage, disposable waste deposit, and storage of cleaning materials, cold rooms, cooking area, cleaning and storage of pots areas, employee access area with female and male changing rooms and clothing area, as well as separate areas for the preparation of snacks and desserts, meats and hortifruti and a room for a nutritionist.
- Cafeteria, with washbasins for hygiene on the access balcony and lounge, with a capacity for 72 people (serving peak of 207 employees in the largest shift, considering outsourced and visits), with a service ramp.
- Return and cleaning of trays, with an air-conditioned garbage tank.

18.4.19 LABORATORY

The Laboratory, will be a building of 309.30 m², and will have analysis and testing areas – rooms for receiving samples, physical preparation, fire assay with access to archive, sample registration, preparation and addition of wet, wet way, area for Leco and cyanide analysis, room for atomic absorption (AA), weighing of bullion and partition / acid digestion, in addition to an office for two jobs, a small area for a canopy, and male and female toilets and one with disability access.

Annexes to the building, will have areas for a mango filter, compact ETE, gas washer, LPG plant and DIR - intermediate waste deposit.

18.5 SITE GEOTECHNICAL

A geotechnical survey was performed to evaluate the terrain for installation of the process plant, along with the support buildings. Different areas of the plant site were analysed using the most appropriate drill holes as per their location, a total of 14 percussion drill holes and 3 mixed drill holes.

In general, analysis of the geotechnical survey results showed that foundations are supported at 23 m for all areas of the plant. The highlights to this are the mill and crushing circuit foundations, which are typically more critical due to the greater static and dynamic loads for these equipments.

18.6 WATER MANAGEMENT AND STERILE, LOW-GRADE ORE AND TAILING PILES STORAGE FACILITIES

18.6.1 PROJECT WATER BALANCE

To maintain the water balance for the Matupá plant area, drainage catchment from pile ponds strategically distributed on the Low-grade Ore Pile ("PMBT"), Sterile Pile ("PDE"), and Tailing Pile ("PDR"), with a total storage volume of 296.641 cu.m, will be used to catch pluvial waters. This surface rainfall capture will be reutilized as of part of the storage volume from the PMBT and PDE pile ponds as raw water for the processing plant, with an approximate volume of 23.217 cu.m, which will help prevent water overflow on ponds. The rest of the effluent will be discharged onto a hydric body using separate safety devices from the ones used for the reuse water, with an approximate volume of 96.629 cu.m, that originated at the PMBT, PDE and PDR pile ponds. For the complete report for this study, see document MTP-B-RL-1007-PRO-B-0001.

18.6.2 PROJECT STERILE, LOW-GRADE ORE AND TAILING PILES STORAGE FACILITIES

GOAL

This presentation aims to provide the main information regarding the following projects developed by GeoHydroTech:

- Sterile Pile (**PDE**);
- Low-grade Ore Pile (**PMBT**); and
- Tailing Pile (**PDR**).

STRUCTURE OVERVIEW - PDE AND PMBT

- PDE has an expected project life of 7 years at which point it is expected to have reached its filling volumetric capacity of 6,827,377.92 m³.
- PMBT, has an expected project life of 7 years at which point it is expected to have reached its filling volumetric capacity of 685,156.81 m³.
- Waste disposal will progress from north to south due to the better construction cost ratio due to the construction of the massif's internal drainage devices.

SEQUENCING OVERVIEW - PDE and PMBT

Pre-operation and first year of operation:

- Vol. PDE:
  - 2,593,614.86 m³.
- Vol. PMBT:
  - 419,450.74 m³.

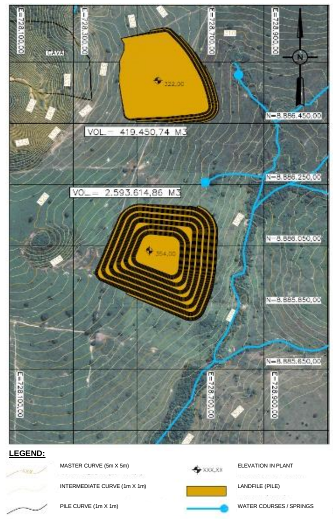


Figure 18-5 – Pre-operation and first year of operation (MTP-B-DS-5010-GHT-G-0003, GHT, 2022)

Second year until the end of the fourth year of operation:

- Vol. PDE:
  - 2,962,684.85 m³.
  - **TOTAL:** 5,556,299.71 m³.
- Vol. PMBT:
  - 164,539.61 m³.
  - **TOTAL:** 583,990.35 m³.

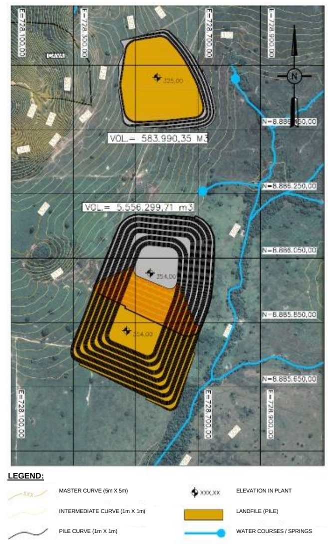


Figure 18-6 - Second year until the end of the fourth year of operation (MTP-B-DS-5010-GHT-G-0004, GHT, 2022)

Fifth year until the end of the seventh year of operation:

- Vol. PDE:
    - 1,271,078.21 m³.
    - **TOTAL: 6,827,377.92 m³.**
- Vol. PMBT:
    - 101,166.46 m³.
    - **TOTAL: 685,156.81 m³.**

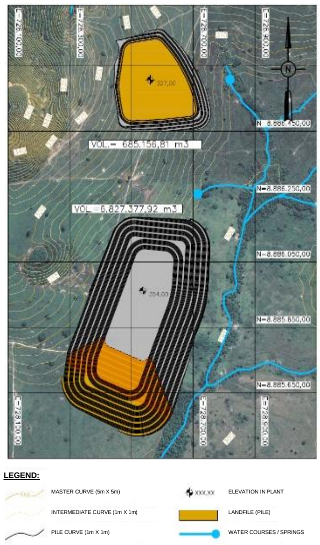


Figure 18-7 - Fifth year until the end of the seventh year of operation (MTP-B-DS-5010-GHT-G-0005, GHT, 2022)

- The closing and covering of the pile corresponding to the first stage with a transition layer overlaid by clay with a low degree of permeability and planting of grasses and legumes is planned for the second stage;
- After closing a section of the plie, it is important to emphasize that the surface drainage devices must be adjusted so that the water that falls on the closed portion and the portion in execution do not mix; and
- Thus, the contamination from rainwater must be eliminated. Rainwater will be collected in the sump to the north of the pile. The water can be pumped into the local environment without going through recirculation in the plant or specific treatments.

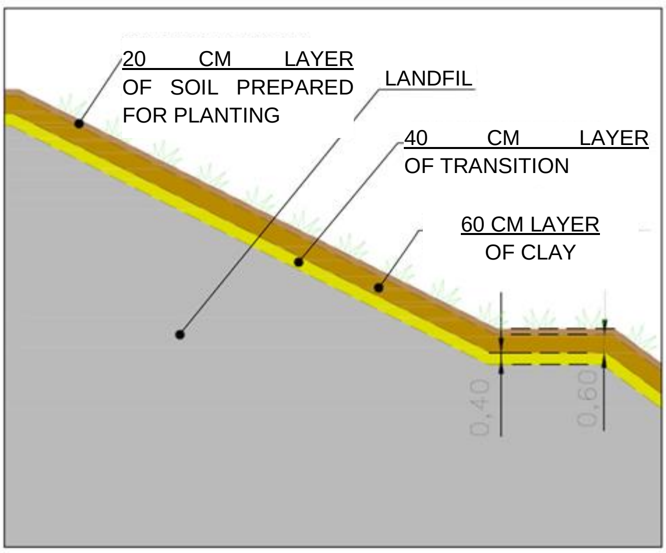


Figure 18-8 – Typical section with transition layer overlaid by clay with low degree, permeability and planting grasses

GEOMETRIC AND GEOTECHNICAL CRITERIA - PDE and PMBT

Storage capacity:

- 685,156.81 m³ - Low-grade and Oxidized Ore Pile (PMBT).
- 6,827,377.86 m³ - Sterile Pile (PDE).

Occupation area:

- 76,869.49 m² - Low-grade and Oxidized Ore Pile (PMBT).
- 228,163.81 m² - Sterile Pile (PDE).

Final stack height:

- 18 m - Low-grade and Oxidized Ore Pile (PMBT).
- 60 m - Sterile Pile (PDE).

Bench height:

- 8 meters.

Width of berms between benches:

- 6 meters.

Inclination of slopes:

- 1V:1.5H.

Crowning quota:

- El. 327.00 m - Low-grade rade and Oxidized Grade Ore Pile (PMBT).
- El. 354.00 m - Sterile Pile (PDE).

- The PDE and PMBT piles to be developed have acid drainage potential, thus, sterile material with low sulfide content will be used for waterproofing the base of the structure and for its covering. The central portion of the structure will be composed of sterile sulfide;
- The sumps will have a slope of 2H:1V for the internal slopes, in order to contain the sediment generated through the piles and allow the treatment of residual water; and

- The sumps will be coated with high-density polyethylene ("HDPE"), 1.00 mm thick, to avoid infiltration of low pH water into the ground.

DISPOSAL RATE - PDE and PMBT

The disposal rate for the PDE and PMBT piles is summarized in Table 18-2.

Table 18-2 -.

Item	Q	LOW GRADE ORE				STERILE			
		Soil (HG)	Saprolite (HG)	Solo (LG}	Saprolite (LG)	Sulf. (LG)	Low Sulfide Content		Sulfide
							Soil	Saprolite	
Density	-	1,9 t/m^3		1.9 t/m^3			1,8 t/m^3	1.9 t/m^3	2,0 t/m^3
Pre Stripping	-	61,641	91,005	6,498	5,158	0	1,112,769	308,827	11,482
Commissioning	**1**	8,067	171,577	779	4,184	5,074	120,088	121,182	73,545
	2	0		0	181	3,258	160,371	100,132	93,535
	3	0	0	0	380	3,093	100,289	59,267	96,137
	4	0	0	0	0	1,086	120,056	44,806	111,137
ANO 1	-	**8,067**	**171,577**	**779**	**4,745**	**12,511**	**500,804**	**325,387**	**374,354**
	5	0	0	0	0	2,702	52,039	80,178	131,695
	6	0	0	0	0	1,876	48,002	62,744	161,450
	7	0	0	0	0	4,222	0	105,961	169,122
	8	0	0	0	0	3,719	0	78,947	192,628
ANO 2	-	**0**	**0**	**0**	**0**	**12,519**	**100,041**	**327,829**	**655,894**
	9	0	0	0	0	5,958	0	78,947	199,989
	10	0	0	0	0	2,800	0	78,947	188,242
	11	0	0	0	0	742	0	46,746	185,819
	12	0	0	0	0	3,697	0	42,105	184,030
ANO 3	-	**0**	**0**	**0**	**0**	**13,197**	**0**	**246,746**	**758,080**
	13	0	0	0	0	4,778	0	43,464	170,258
	14	5,985	49,390	794	14,262	28,792	0		171,248
	15	0	0	0	177	24 232	0		188,535
	16	0	0	0	0	11 687	0	11,855	165,796
ANO 4		**5,985**	**49,390**	**794**	**14,439**	**69,489**	**0**	**55,319**	**695,836**
	17	0	0	0	0	11 781	0	3,487	165,796
	18	0	0	0	0	13,303	0	0	160,642
	19	0	0	0	0	15,913	0	0	184,463
	20	0	0	0	0	13,381	0	0	161,266
ANO 5		**0**	**0**	**0**	**0**	**54,378**	**0**	**3,487**	**672,165**
	21	0	0	0	0	16,792	0	0	138,948
	22	0	0	0	0	10,757	0	0	81,818
	23	0	0	0	0	6,955	0	0	71,553
	24	0	0	0	0	2,731	0	0	73,102
ANO 6		**0**	**0**	**0**	**0**	**37,234**	**0**	**0**	**365,420**

SLOPE STABILITY - PDE and PMBT

The slope stablitiy for the PDE and PMBT stockpiles is shown in Figure 18-9. Figures 18-10 to 12

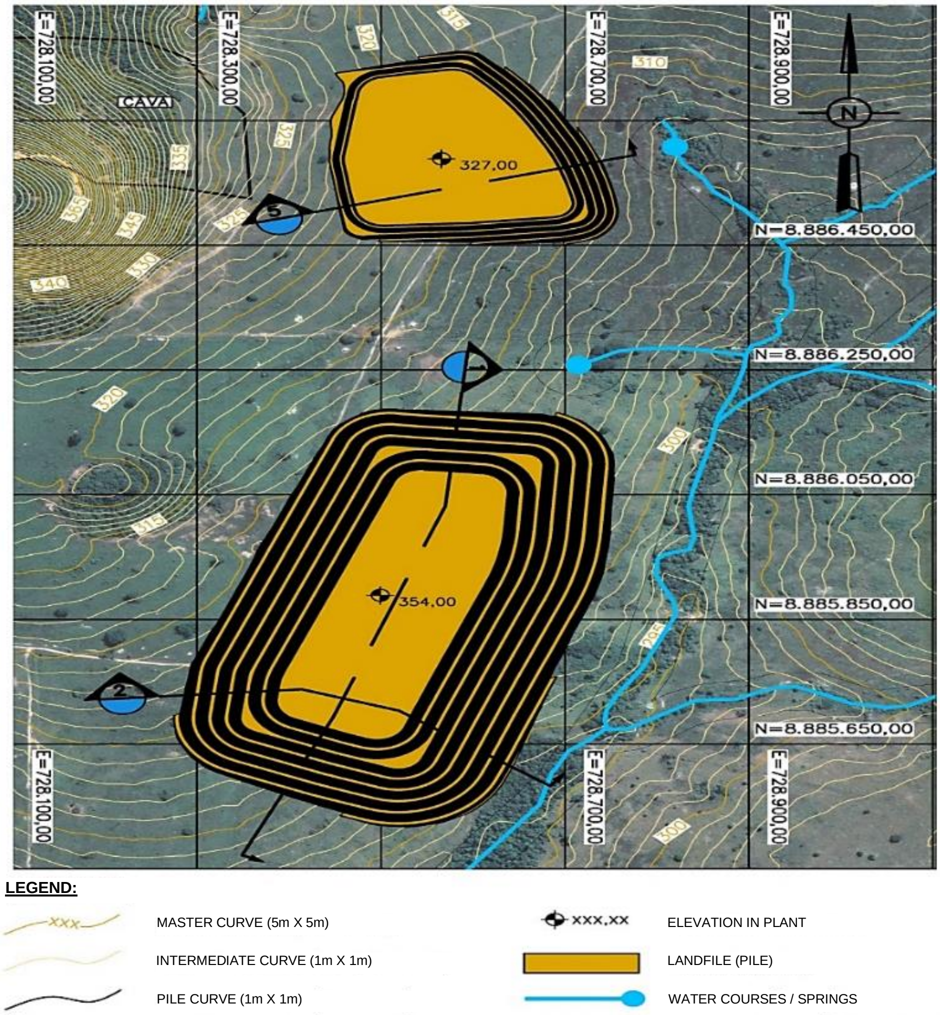


Figure 18-9 - Slope stablitiy for the PDE and PMBT stockpiles (GHT, 2022)

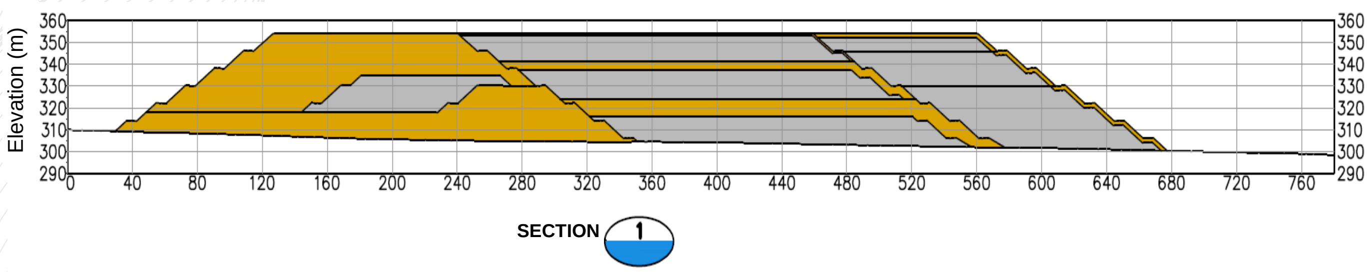


Figure 18-10 - - Section slope stablitiy for the PDE stockpiles (MTP-B-DS-5010-GHT-G-0013 - GHT, 2022)

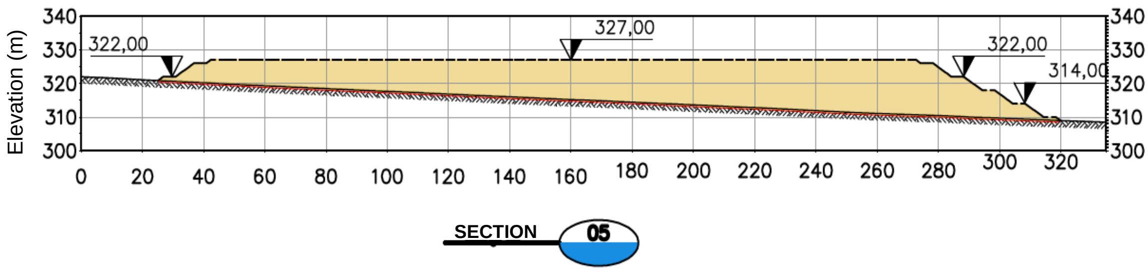


Figure 18-11 - Section 05 lope stablitiy for the PMBT stockpiles (MTP-B-DS-5010-GHT-G-0013 - GHT, 2022)

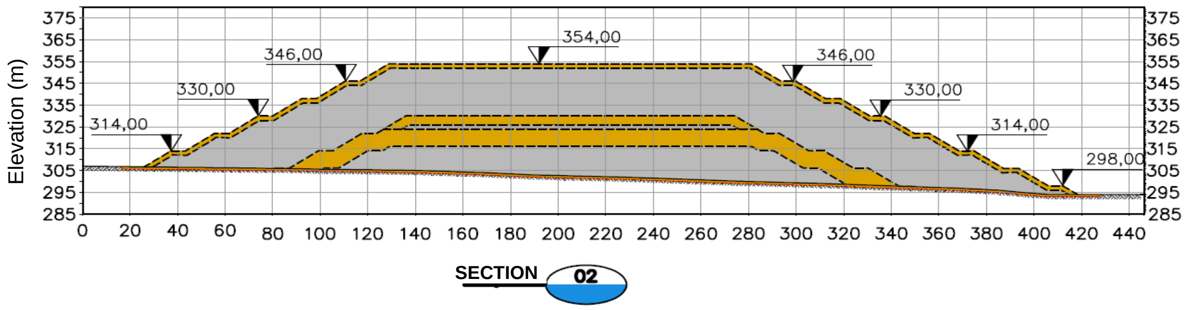


Figure 18-12 - Section 02 slope stablitiy for the PMBT stockpiles (MTP-B-DS-5010-GHT-G-0013 - GHT, 2022)

GEOTECHNICAL INSTRUMENTATION - PDE and PMBT

- 08 vibrating string piezometers (PE);
- 04 water level meters (MNA);
- 08 surface displacement marks (MS);
- 04 topographic reference points (“Benchmark”) (BM);
- 04 inclinometers (INC);
- 03 inspection wells (PC).

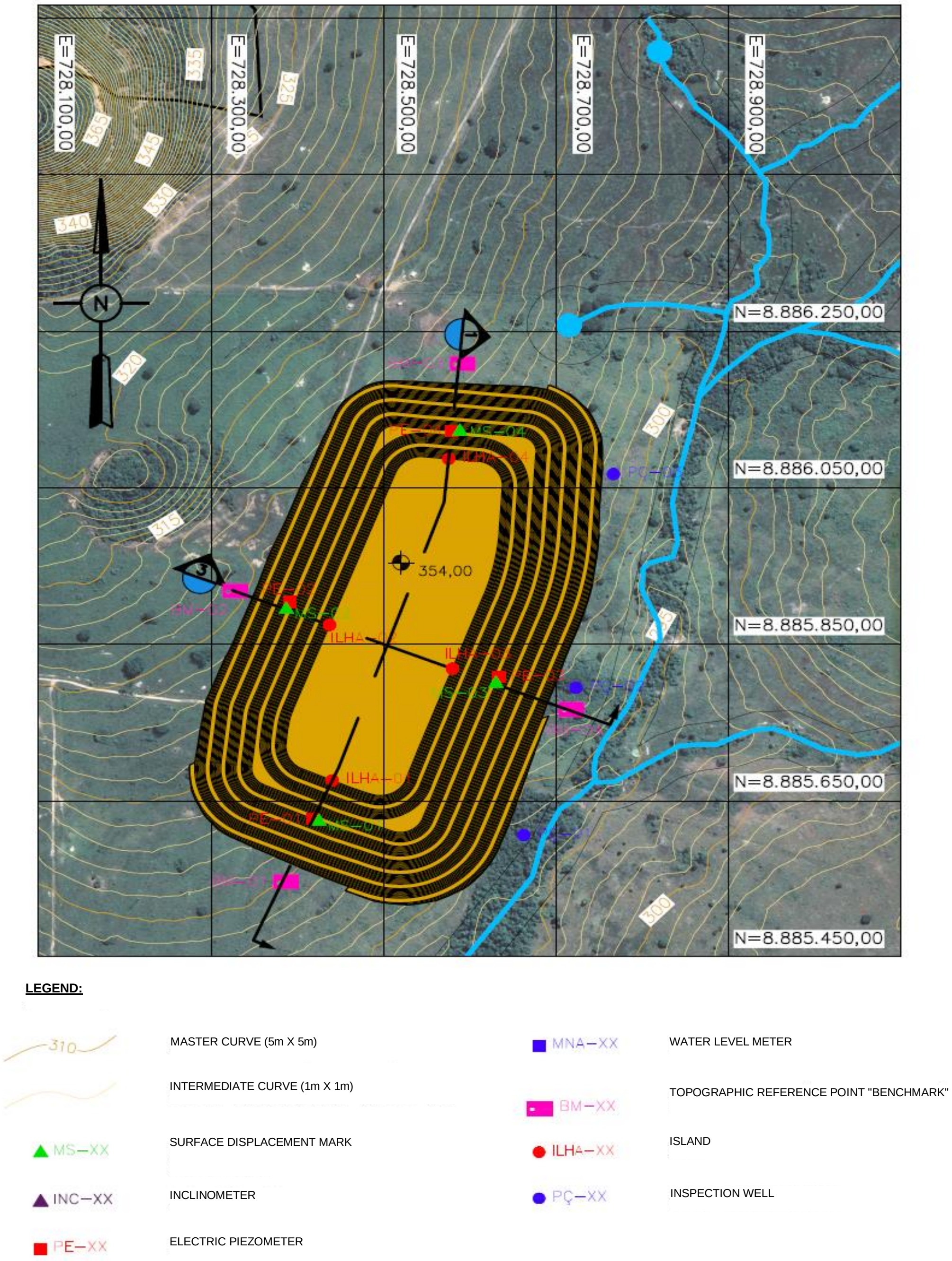


Figure 18-13 - Geotechnical Instrumentation PDE (MTP-B-DS-5010-GHT-G-0017, GHT, 2022)

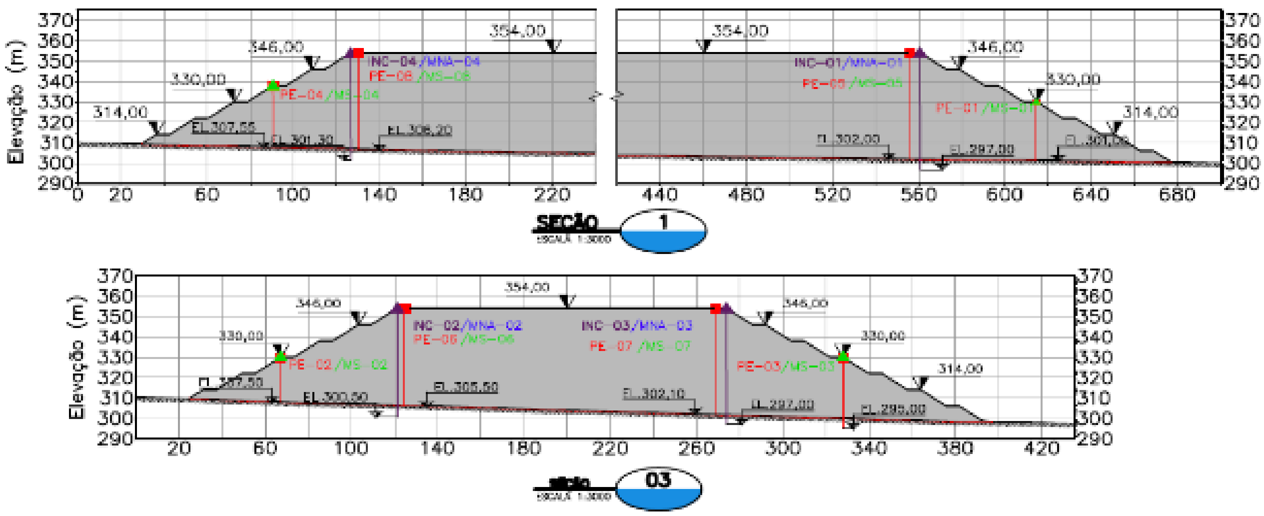


Figure 18-14 – Sections 01 and 03 Geotechnical Instrumentation PDE (MTP-B-DS-5010-GHT-G-0018, GHT, 2022)

GEOTECHNICAL INSTRUMENTATION - PDE and PMBT

PDE Monitoring:

- Vibrating string piezometers – daily readings;
- Surface displacement milestones – weekly readings during execution, then monthly readings;
- Water level gauges – weekly readings;
- Inclinometers – monthly readings; and
- Monitoring wells – monthly collections.

PMBT Monitoring:

- For the PMBT, monitoring must be carried out every 15 days with the topography team, since there will be no installation of instruments in this pile; and
- Monitoring must be carried out for geometric control, analyzing the width of the berms, and the slope and height of the slopes to ensure adherence to the project.

CONCLUSIONS AND RECOMMENDATIONS - PDE and PMBT

- For the pile areas foundation, an excavation of up to 0.50 meters in depth to clean the surface material will be carried out, which must be validated during the execution;

- For the maintenance of the hydraulic structures, a program of inspection and continuous cleaning must be carried out, thus ensuring the proper behavior for the surface and deep drainage, and sumps; and
- The PDE and PMBT piles will have 3 sequencing steps, totaling a volume of 6,827,377.86 m^3 for the sterile pile and 685,156.81 m^3 for the low-grade pile.

STRUCTURE OVERVIEW - **PDR**

- The total volume which is served by the two piles is 6,421,432.42 m³;
- Plant: tailings moisture will be 28.2%; and
- Saturation degree adopted will be 90.0%.

Figures 18-15 to 16

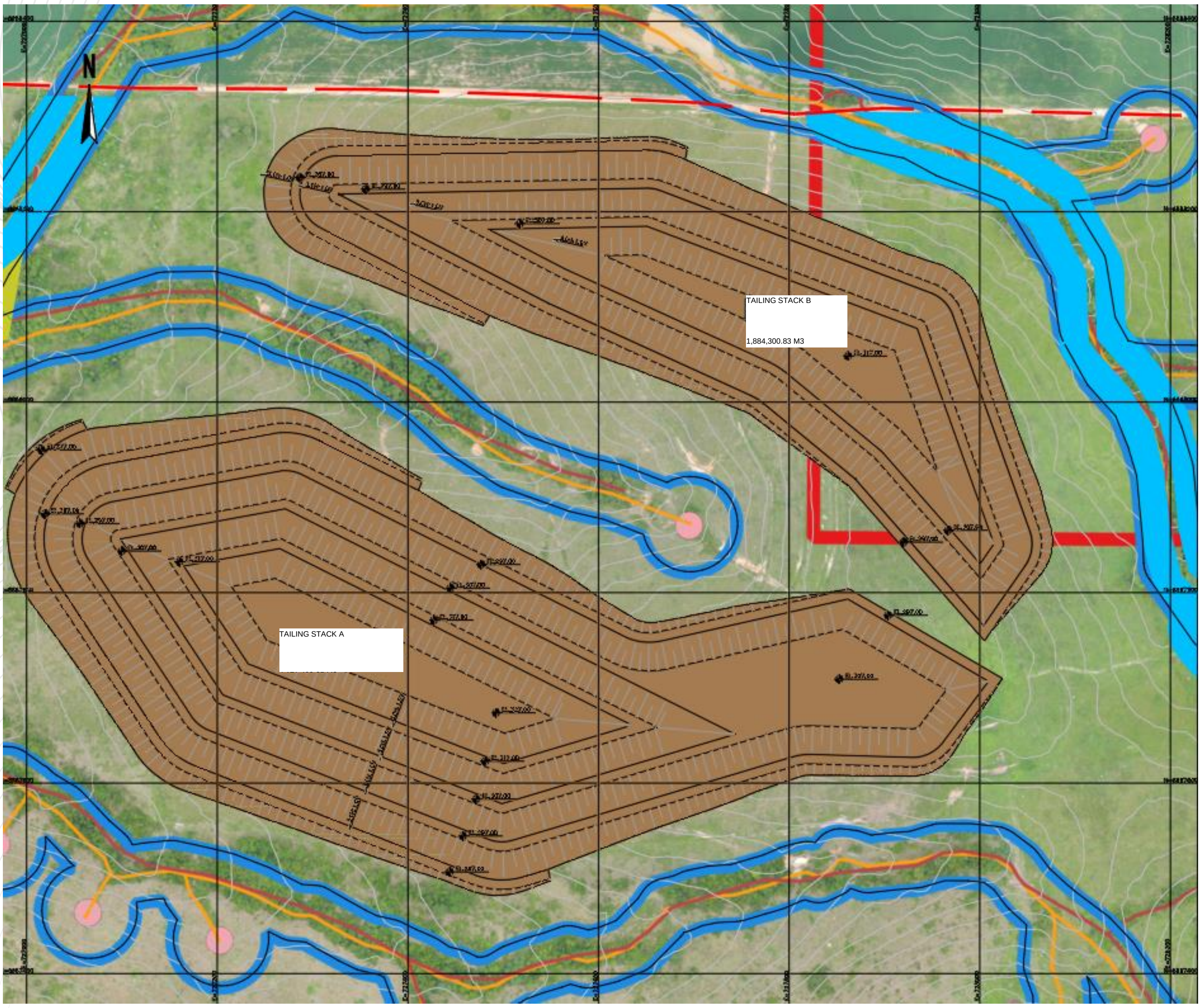


Figure 18-15 – Waste stockpile plants (MTP-C-DS-6020-GHT-G-0037, GHT, 2022)

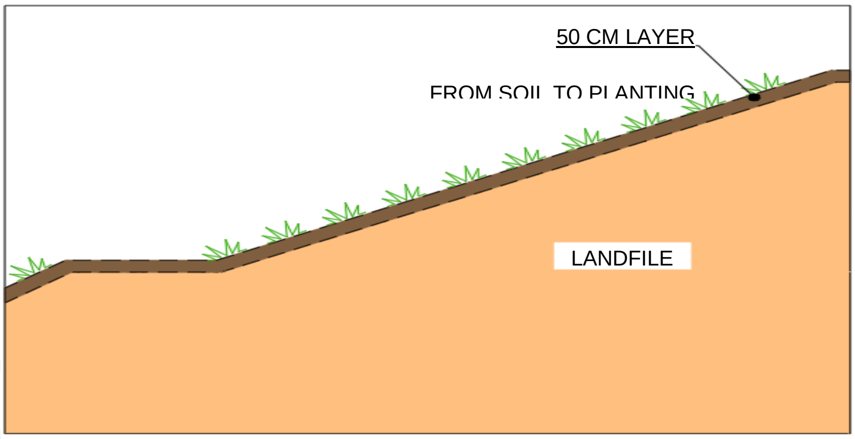


Figure 18-16 – Typical section waste stockpile (GHT, 2022)

- Arrangement, scattering and compaction of the pile – Constructive sequencing;
- Parallel execution of surface drainage structures;
- Execution in parallel of the clayey closing layers on the sides of the pile slopes;
- Execution in parallel of the vegetation covering on the slopes where the closing clay layer was executed; and
- Execution and installation of instruments.

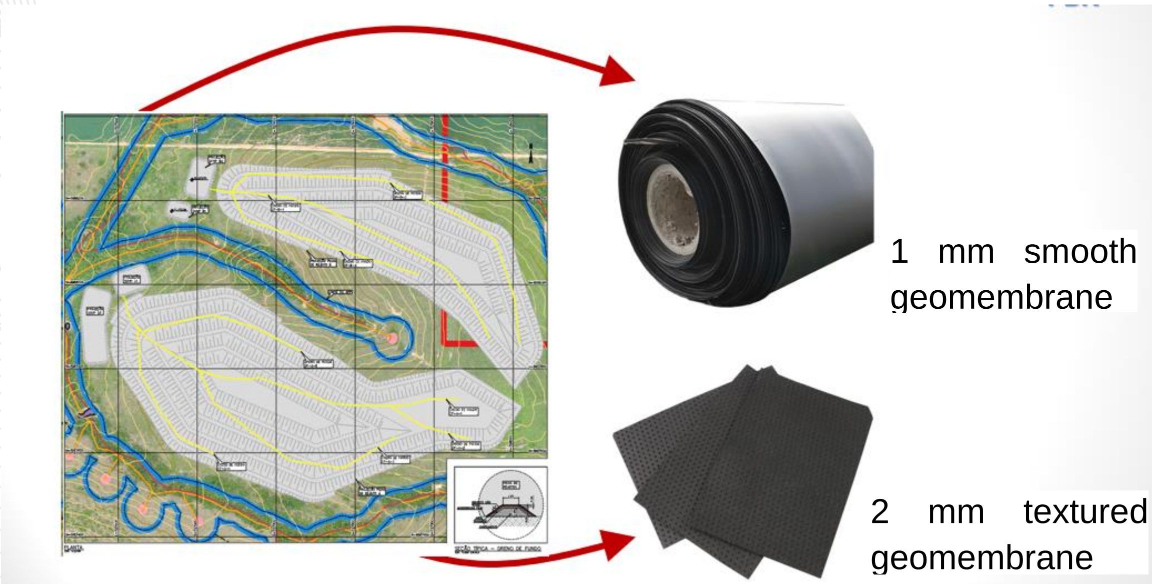


Figure 18-17 – Typical drainage structures (GHT, 2022)

SURFACE DRAINAGE

- **Freeboard** criterion of 30% (minimum) of the height of the walls of the devices in relation to the water depth, except for water descents, in which 40% was adopted;

- Dimensioning of channels in excavated soil for sections **between stages**; and
- Sizing of top gutters, berm gutters, water drops, culverts, peripheral channels, and energy sinks for the **final stack configuration**.

RESULTS - SURFACE DRAINAGE – **PDE**

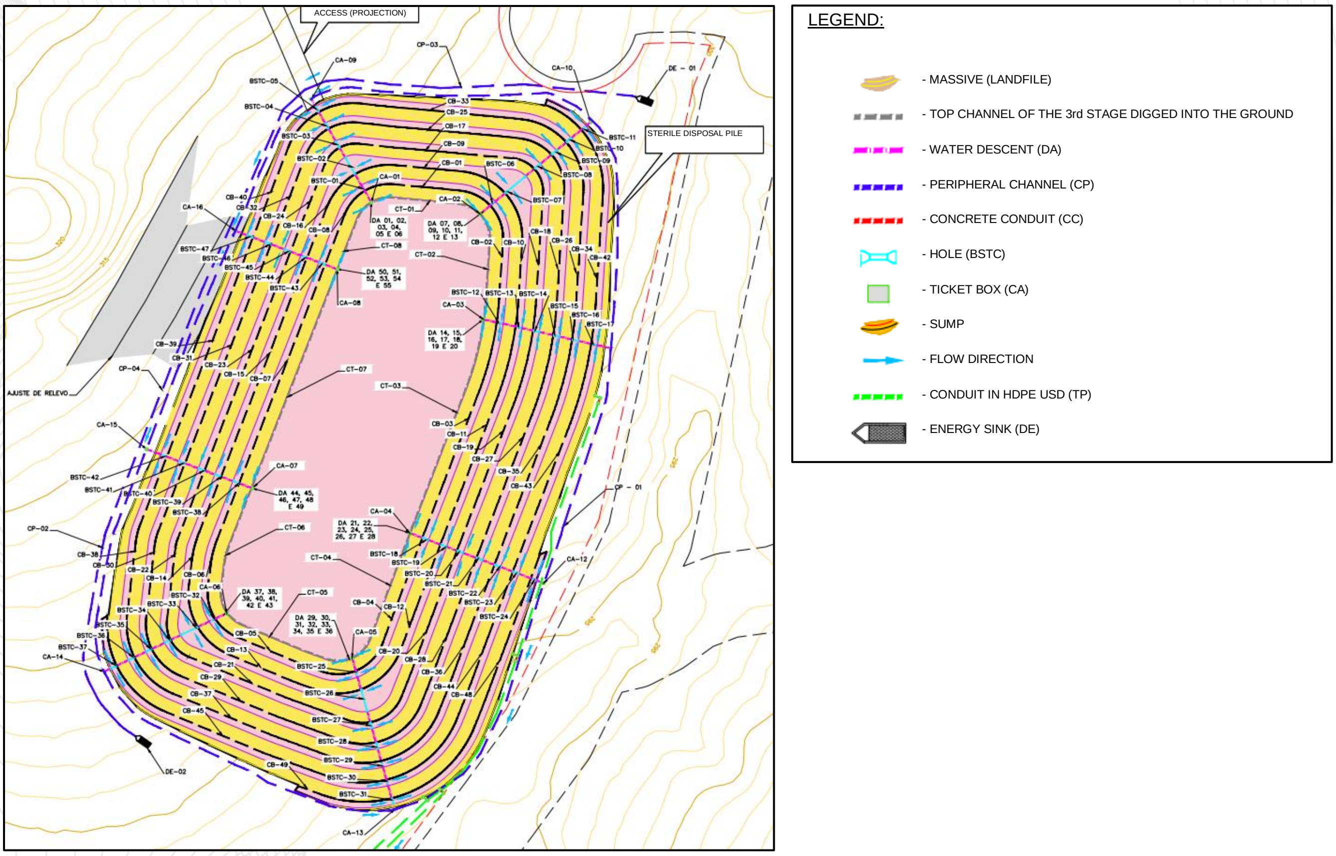


Figure 18-18 – Plant Surface drainage sterile stockpile - PDE (GHT, 2022)

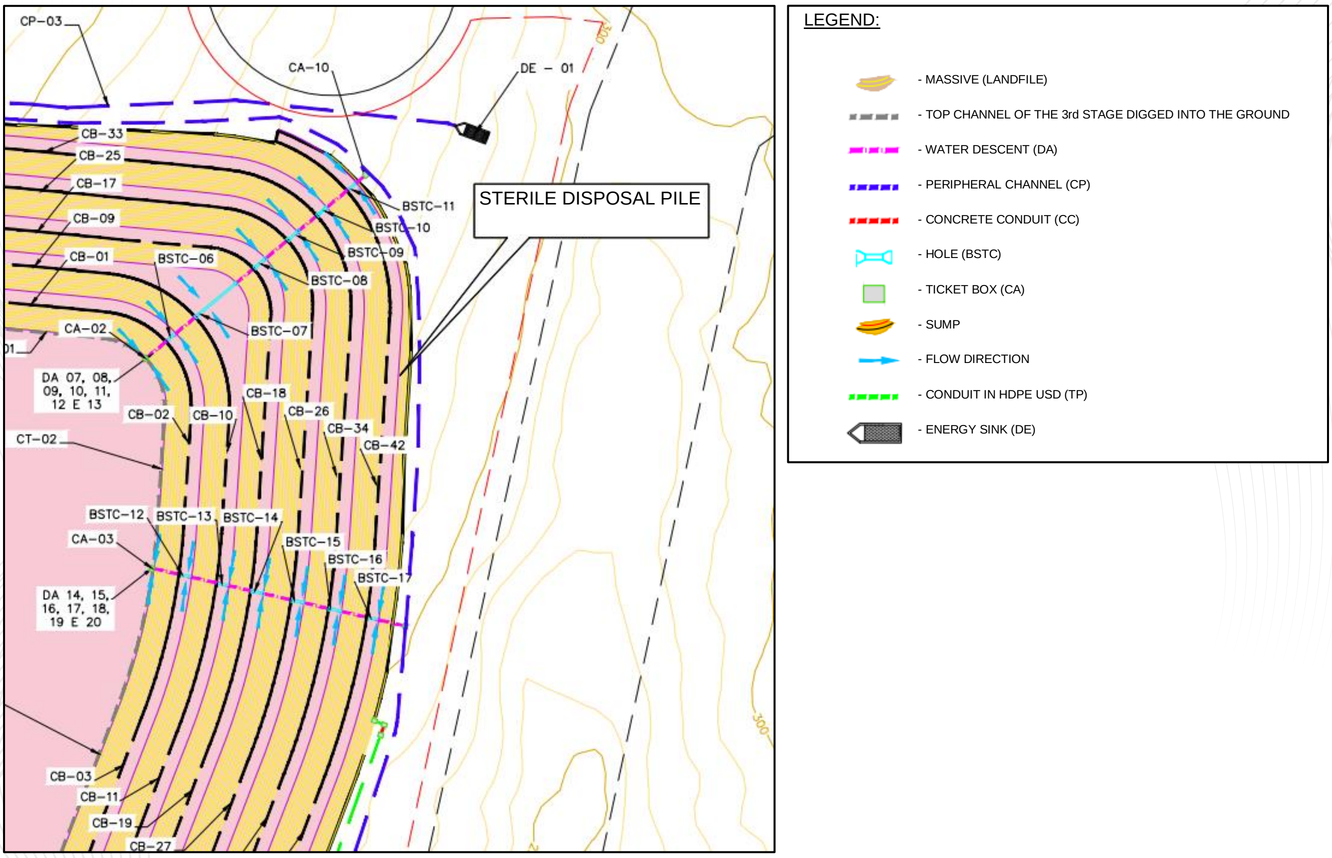


Figure 18-19 – Detail Surface drainage sterile stockpile - PDE (GHT, 2022)

RESULTS - SURFACE DRAINAGE – **PMBT**

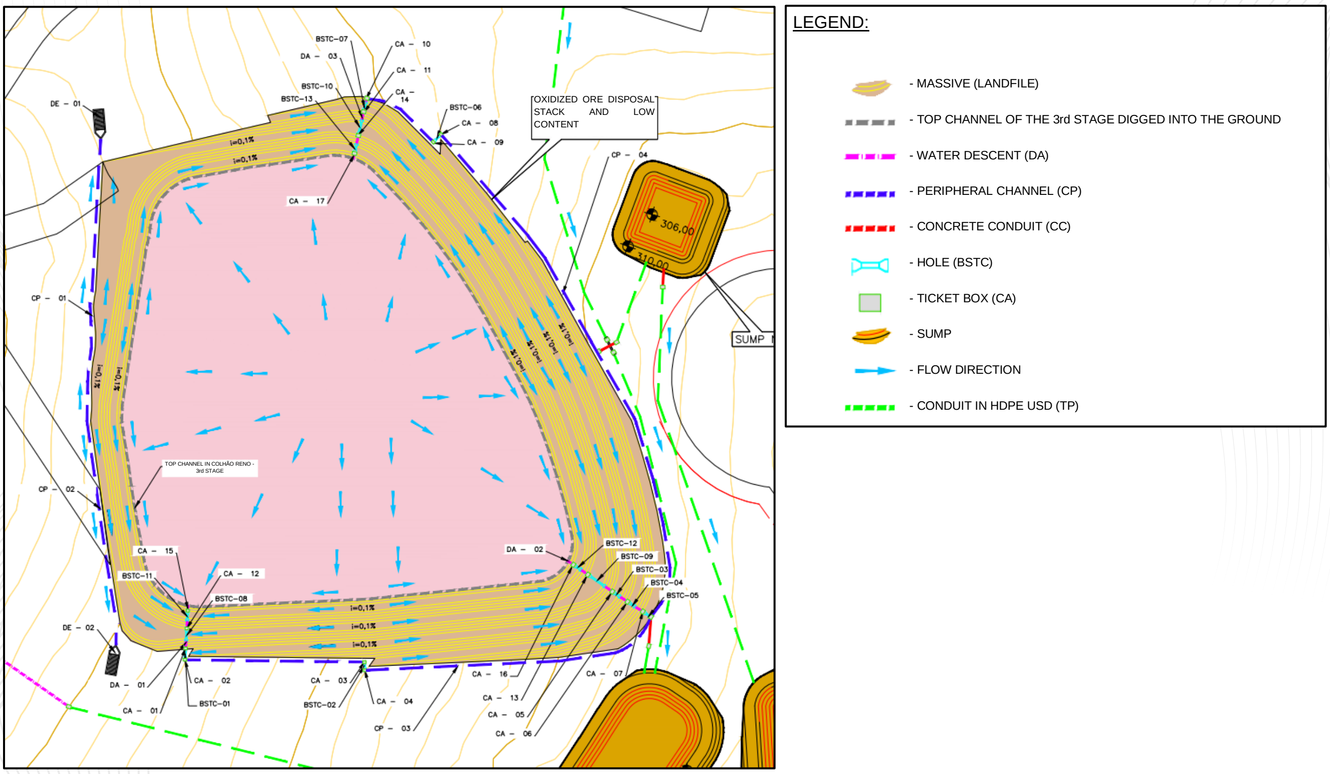


Figure 18-20 - Plant surface drainage Low Grade Ore stockpile - PMBT (GHT, 2022)

DEEP DRAINAGE

- Dimensions made for **PDR, PDE and PMBT** batteries;
- Application of **Wilkins Methodology**, applied to turbulent flow conditions;
- Minimum FS of 2.5, according to the method of obtaining the flow, in accordance with **NBR 13.029/2017**; and
- Draining section will be in **rock blocks (rockfill)**, surrounded by a transition layer of coarse material and another layer of medium to fine sand.

RESULTS - DEEP DRAINAGE – **PDE**

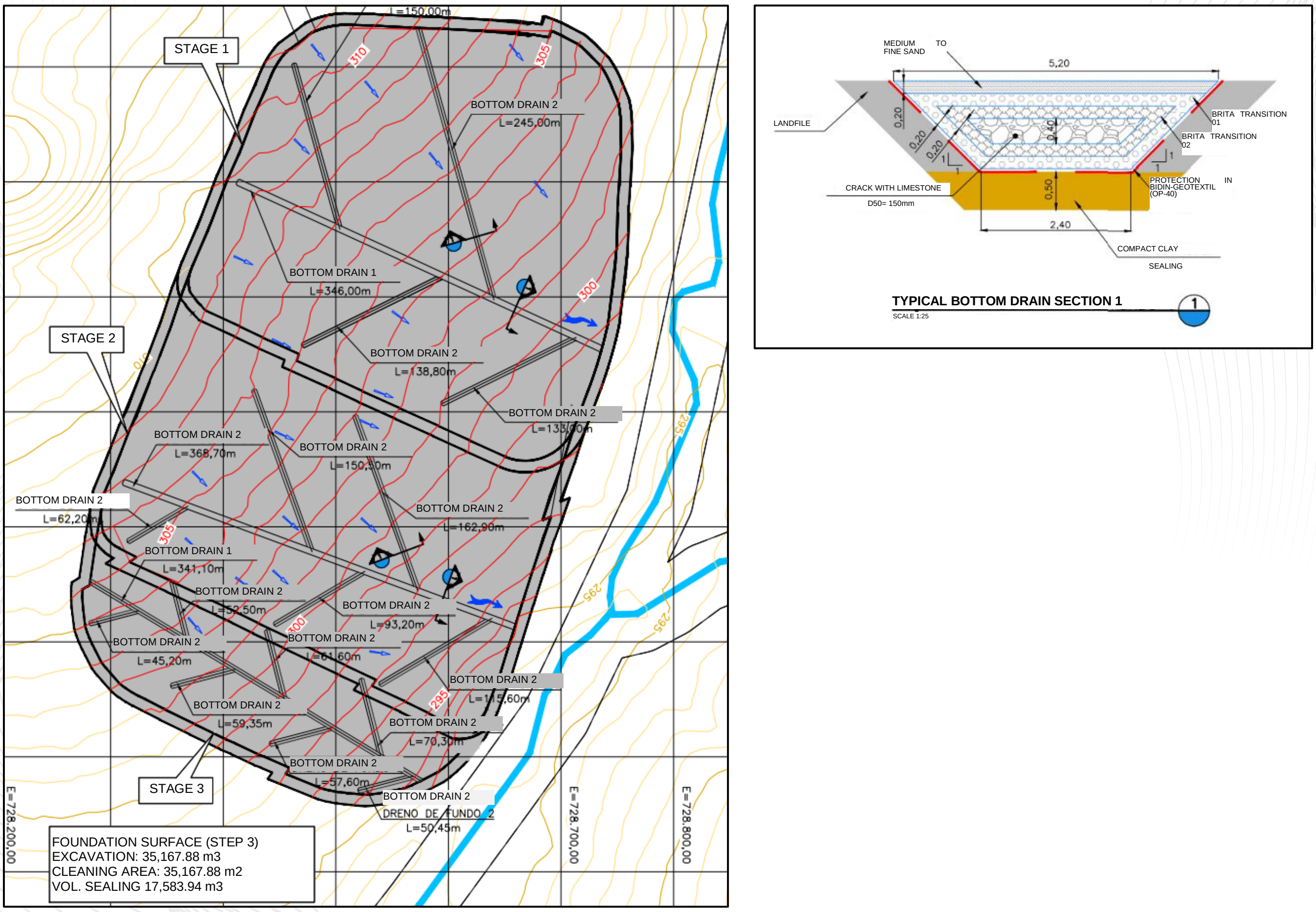


Figure 18-21 – Plant deep drainage sterile stockpile - PDE (GHT, 2022)

RESULTS - DEEP DRAINAGE – **PMBT**

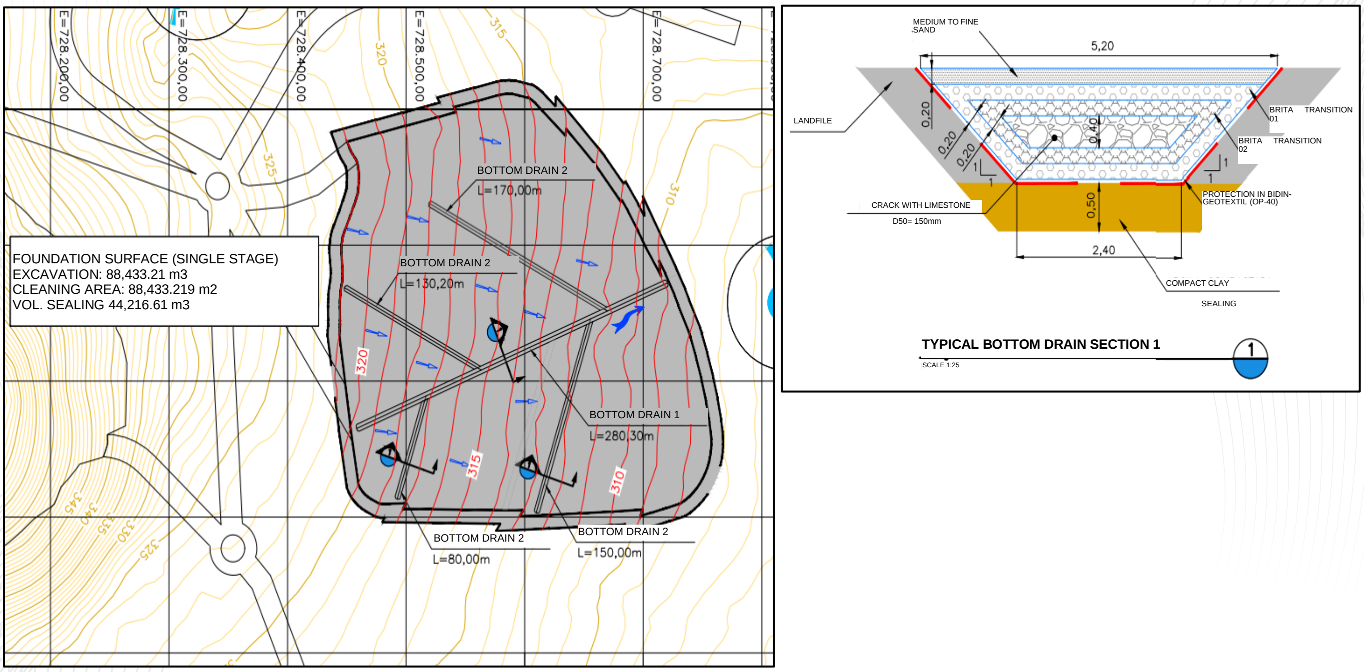


Figure 18-22 - Plant deep drainage Low Grade Ore stockpile - PMBT (GHT, 2022)

FINE CONTAINMENT STRUCTURE

- Sumps sized separately, with different purposes, for **PDE, PMBT and PDR** structures;
- Payback period of 2 years and duration of 48 hours (**assumed**);
- Extravasation system of each sump sized for a return period of 500 years, according to **NBR 13.029/2017**;
- Dimensioning of channels, water descents, culverts and pipes between the sumps, structures and tributaries of the Córrego Porcão, as well as energy dissipators in the final stretches, with the flow being carried out by gravity in all stretches (**assumed**);
- The treated effluent will be released into an affluent water body of the Córrego Porcão **(assumed**). It is noteworthy that this release must follow the effluent release standards and the classification of surface water bodies, as recommended by CONAMA **Resolution No. 357/2005**;
- Systems with **operation in gates** were proposed to direct the flows (applicable to PMBT); and
- The volumes coming from the Beneficiation Plant (**process water**) were disregarded in the water balance of the sumps.

RESULTS - LEASE OF SUMPS – PDE

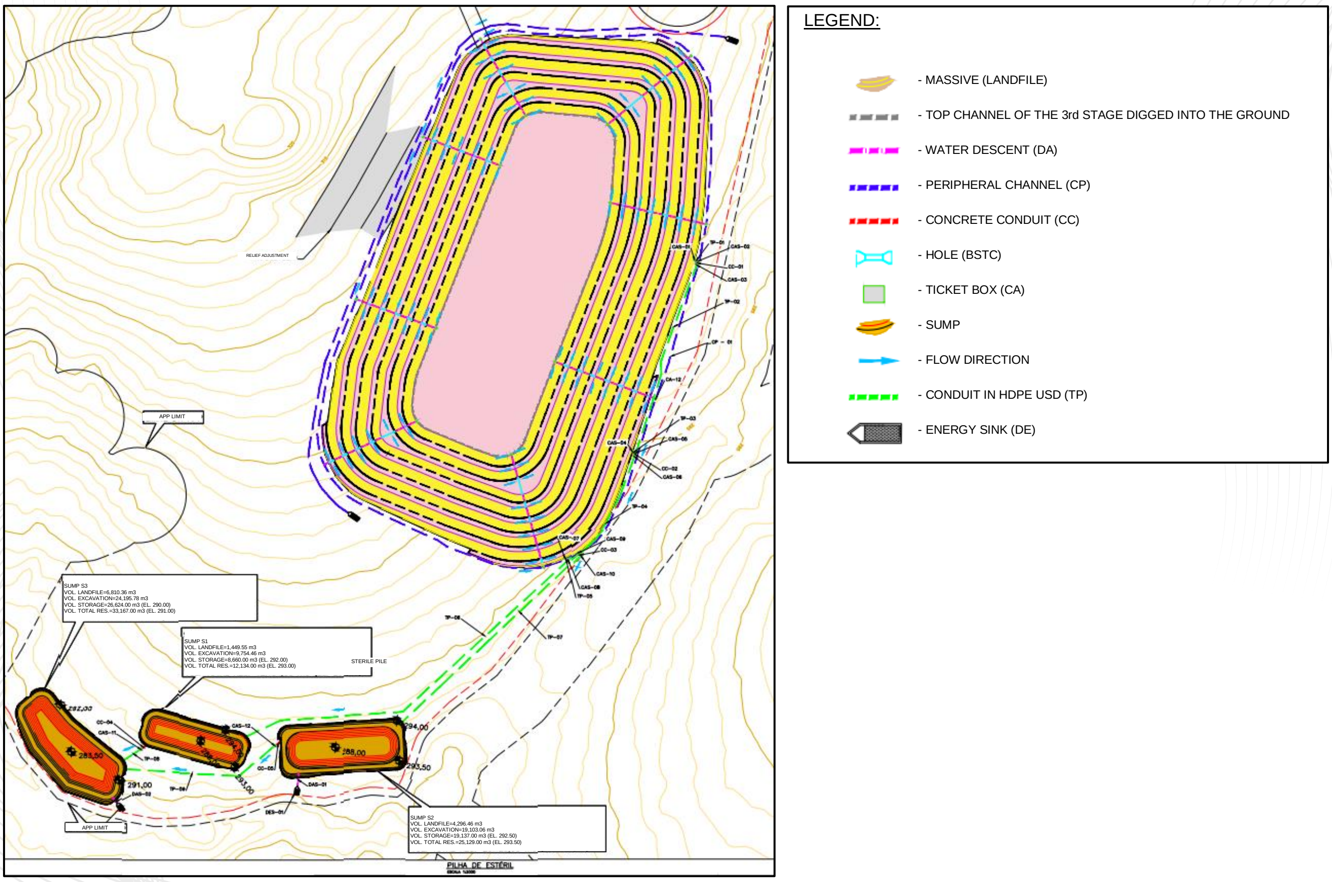


Figure 18-23 - Plant lease of sumps drainage sterile stockpile - PDE (GHT, 2022)

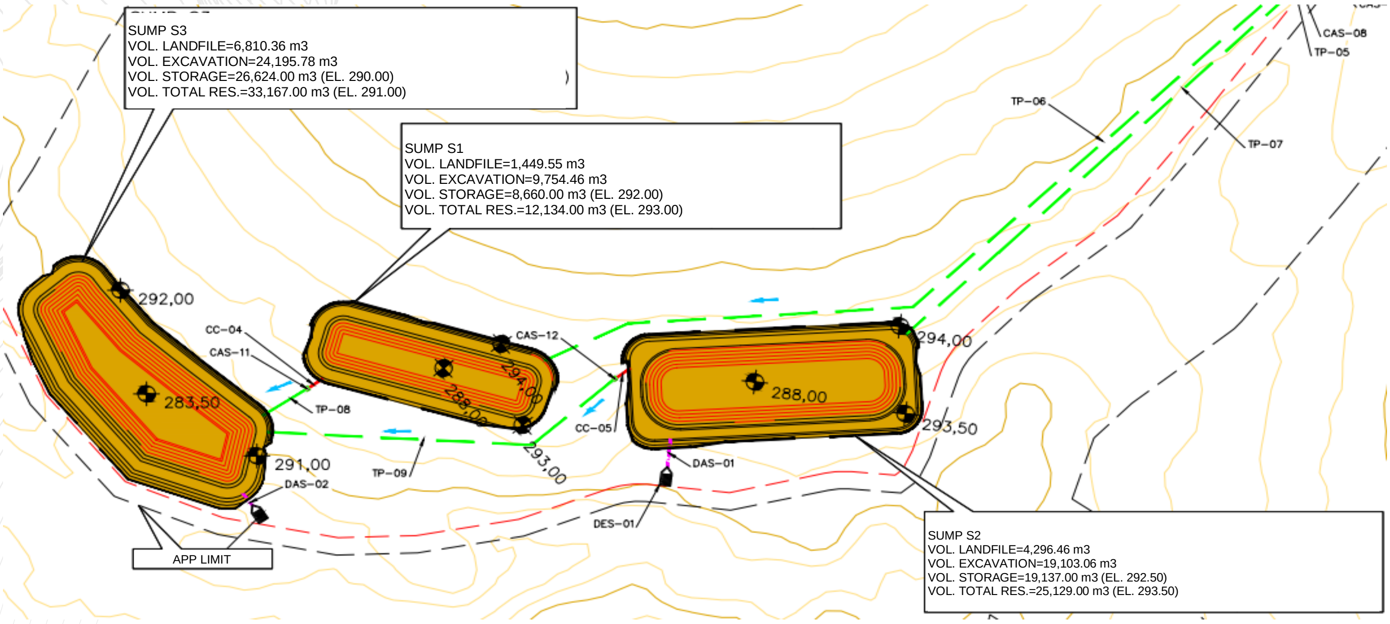


Figure 18-24 - Plant volume of sumps drainage sterile stockpile - PDE (GHT, 2022)

RESULTS - LEASE OF SUMPS – PMBT

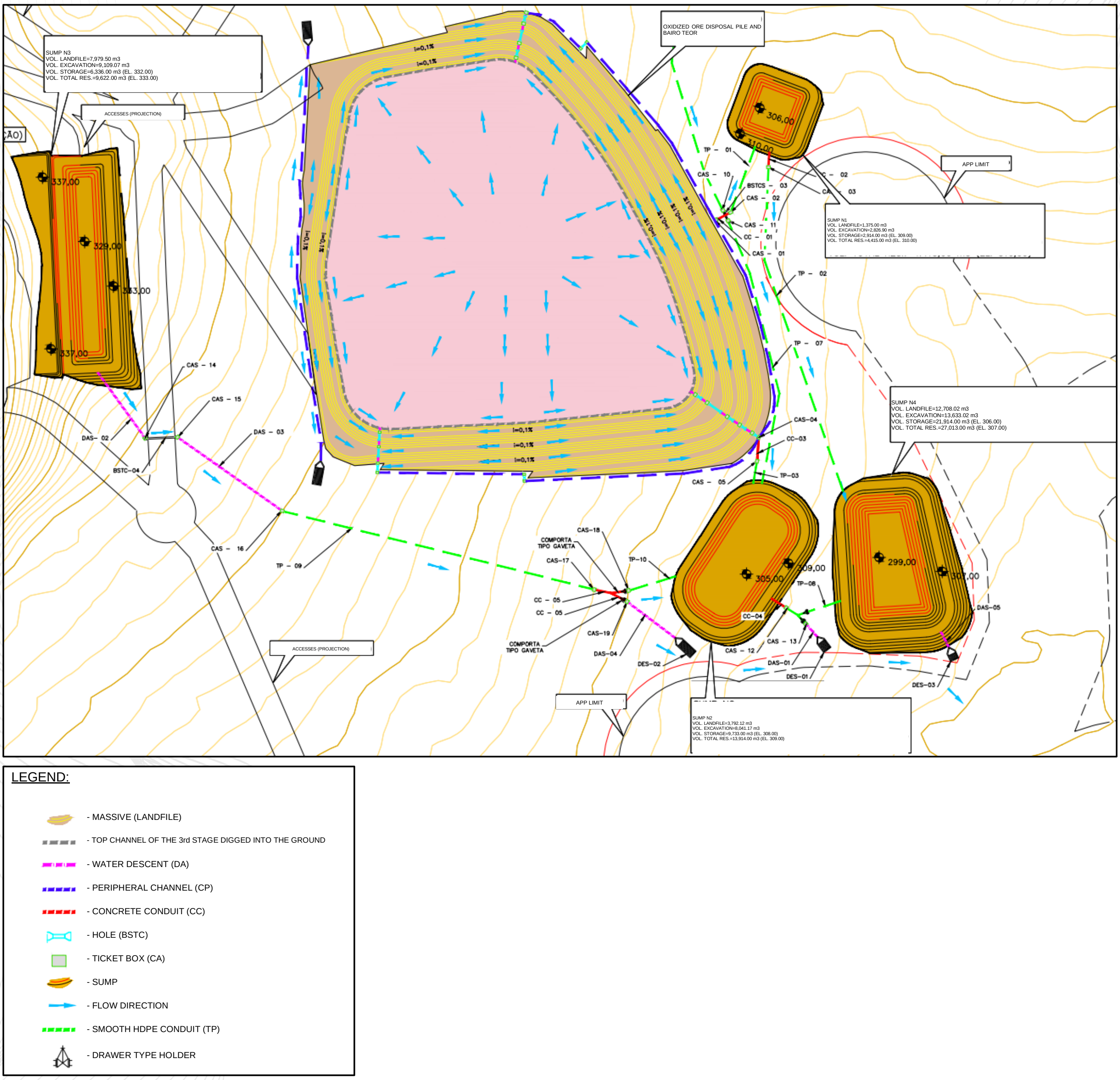


Figure 18-25 - Volume and lease plant of sumps drainage Low Grade Ore stockpile - PMBT (GHT, 2022)

SUMPS CLEANING SYSTEM - PDE, PMBT and PDR

- The sump cleaning system can be carried out by water blasting from a water truck, with the removal of the flow formed in the sump by means of its own pump for suction of sediments.
- The material extracted from the sumps can be deposited in the piles themselves.

CONCLUSIONS AND RECOMMENDATIONS - PDE, PMBT and PDR

- The provision of rockfill at the base of the piles must be considered together with the implementation of drainage systems, in order to avoid erosion in rainy seasons;
- The implementation of protective windrows around the service plaza is also considered to prevent material thrown in due to rain;
- For the surface protection of the piles against erosion caused by rain, the preparation of a layer for planting grasses and legumes is foreseen (only PDE and PDR). This overlay will also allow for a reduction in the contribution of sediments being carried to the sumps;
- For the dimensioning of the sumps, the slope of the excavated slopes was considered equivalent to 2H:1V;
- For the PMBT, the surface drainage collection sump, in addition to collecting the contributions from the surface drainage of the pile, will also receive contributions from the drainage areas of the plateau of crushed material and the crushing square (5,000 m^2 of drained area);
- For PMBT, the treatment sump will allow the pumping of collected surface water for reuse in the Beneficiation Plant and to aid treatment in the deep drainage sump;
- Sumps must be cleaned periodically (annually) to allow for an increase in the lifespan of the structure. The sumps are designed to withstand an annual period sediment load;
- The surface of the sumps will be coated with HDPE geomembrane, with a thickness of 1.00 mm, with smooth edges; and
- For the maintenance of hydraulic structures, a program of inspection and continuous cleaning must be carried out, thus ensuring the proper behavior of surface and deep drainage and sumps.

18.7 WATER SYSTEMS

18.7.1 RAW WATER SUPPLY SYSTEM

The design considered that raw water will be captured from the Porcão River. It will be directed to the raw water tank, from which it will be distributed to required points in the plant, such as gland water and reagent preparation, feed the potable water treatment system feed, and used as a make-up source for process water. The bottom section of the raw water tank will be dedicated for the fire water system.

The raw water pond will receive water from the Porcão River, serving as a technical reserve of water for the site in the event of the unavailability of the water pumping system.

The design also considered four underground wells that will feed the portable water treatment system during periods of lack of rain.

18.7.2 POTABLE WATER SUPPLY

The quality requirement for the potable water treatment plant will match the local drinking water guidelines. Raw water will be sourced from the raw water pump and processed through the potable water treatment skid before being stored in the potable water tank. This water will feed all safety showers and administrative buildings.

18.7.3 FIRE SUPPRESSION SYSTEM

All facilities will have a fire suppression system in accordance with the structure’s function. For the most part, fire water will be used with an underground ring main network around the facilities. All buildings will have hose cabinets and handheld fire extinguishers. Electrical and control rooms will be equipped with dry-type fire extinguishers. Ancillary buildings will be provided with automatic sprinkler systems. For the reagents, appropriate fire suppression systems will be included according to their material safety datasheets.

18.7.4 SEWAGE COLLECTION

A sewage treatment plant package will be supplied at the plant to treat all sewage collected within the site. The collection network will be underground. Depending on the type of chemical waste from the laboratory, it is either recycled to the plant or stored for off-site disposal. Office and domestic waste are collected and disposed of offsite in accordance with applicable regulations.

18.8 ACCOMMODATIONS CAMP

The construction of accommodation for the team during construction will be performed by Aura in Mautpá City.

The construction concept of using modular equipment, compact electrical rooms in containers, pre-moulded structures will allow fewer people to be mobilized for the site.

The work is expected to reach less than 400 workers at its peak and the cities mentioned above have the capacity to accommodate this team without major problems.

19. MARKET STUDIES AND CONTRACTS

19.1 INTRODUCTION

Promon Engenharia Ltda. (Promon) was engaged by Aura Minerals Inc. (Aura) to prepare an Independent Technical Report ("ITR") containing a Market Study on Gold for the Matupá Gold Project, located in the State of Mato Grosso, Brazil.

Aura is a publicly traded company listed on the stock exchanges of Toronto, Canada, and on the stock exchange of São Paulo, Brazil. The Company's business focus is the exploration, development and operation of gold, copper and other metals projects across the Americas.

The purpose of this report ("Report" or "Technical Report") is to provide background and supporting information on the economic potential for the Matupá Gold Project. This Report and the results herein comply with the requirements of the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101") – Standard of Disclosure for Mineral Projects and Form 43-101F1 – Technical Report. This Report will be part of Aura's FS Report for the Matupá Project.

19.2 MARKET STUDY

The World Bank forecast indicates an increase in the average price of gold to US$1,698/oz in 2020 from an average of US$1,392/oz in 2019. In the next ten years, the gold price is expected to reach around US$1,400/oz in 2030, as presented in Table 19-1.

Table 19-1 - Forecast of Gold Price.

WORLD BANK COMMODITIES PRICE FORECAST (NOMINAL US DOLLARS)							RELEASED: OCTOBER 22, 2020			
FORECASTS										
Commodity		Unit	2019	2020	2021	2022	2023	2024	2025	2030
Precious Metals	Gold	$/oz	1,392	1,775	1,740	1,698	1,658	1,618	1,580	1,400
	Silver	$/oz	16.2	21.0	18.1	18.1	18.1	18.1	18.1	18.0
	Platinum	$/oz	864	875	870	906	943	982	1,022	1,250

Source: World Bank – Commodities Market Outlook, October – 2020 – Commodities Prices Forecasts.

In the first month of 2020, gold price averaged US$1,560/oz, which was about 6% higher than December 2019. Throughout the year of 2020 the spot price of gold reached approximately US$2,000/oz, which represents a growth of more than 27% during the same year.

Gold is known as a precious metal, highly ductile and malleable. It is used for making jewelry, developing electronic equipment, medicines, and for investment purpose around the globe.

The demand for gold is growing as investors increase their focus on long-term investments and this causes the price of gold to rise as well, but the key factor that is fueling the demand for the precious metal is a high level of uncertainty observed in the global economy due to the Coronavirus situation.

As the analysis of the World Gold Council shows, the gold value during its periods of low-interest rates are twice as high as their historical average. Moreover, gold seems to be more effective in portfolio diversification, mitigation of risk, and long-term returns compared with government bonds. So, in current conditions of low-to-negative interest rates, demand on gold from investors and the Central Bank is going to continue strengthening, moving prices up.

The value of US$ 1,699/oz was adopted for the Economic Model.

19.3 CONTRACTS

There are no material contracts or agreements in place as of the effective date of this Technical Report. Refining contracts are typically put in place with well-organized international refineries and sales are made based on spot gold prices.

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 INTRODUCTION

The main objective of this section is to present the studies conducted so far to develop a baseline for the Matupá Gold Project - X1 Deposit and also to support the Environmental Impact Assessment and Environmental Impact Assessment Report (EIA/RIMA) required for environmental licensing of the Matupá Gold Project (X1 Deposit).

20.2 LOCATION

The study area is located in the northern region of the state of Mato Grosso, on the limits of the municipalities of Matupá and Guarantã do Norte. The city of Matupá is located about 720 km from the state capital Cuiabá and borders the State of Pará and Mato Grosso municipalities of Guarantã do Norte, Nova Guarita, Novo Mundo and Peixoto de Azevedo. The X1 Deposit is located about 13 km north of the city of Matupá, towards Guarantã do Norte, about 1 km from the left bank of the Federal Highway BR 163, rural area of the municipality of Matupá, in the following geographical coordinates: 10° 03’ 56” S, 54° 55’ 05” W.

20.3 GENERAL OVERVIEW

With the objective of starting the licensing of the Matupá Gold Project, X1 Deposit, Aura Minerals filed, in January 2019, the request for a Term of Reference (“TR”) with SEMA-MT (Secretary of the Environment of Mato Grosso) for the elaboration of an Environmental Impact Assessment and Environmental Impact Assessment Report (EIA/RIMA). The Term of Reference was issued on September 20, 2020 and from that moment on, proposals were requested from several specialized companies and the winning company was Mineral Engenharia (São Paulo), with good experience and results from projects in the state of Mato Grosso.

Field surveys and secondary data required by the TR were carried out between April and September 2021, covering the 2 climatic seasons (rainy and dry) existing in the state of Mato Grosso. Among the field studies carried out, assessment of mammals, birds, fish, reptiles, flora, hydrology and hydrogeology, sampling of surface and underground waters and sediments, historical and archaeological heritage, socioeconomic diagnosis of the region and traditional populations, identification of areas for conservation, etc. In addition to these studies, tailings and waste samples were collected to assess the acid mine drainage potential.

The EIA/RIMA was filed with the Secretary of State for the Environment of Mato Grosso -SEMA-MT on November 30, 2021 and was forwarded to the Coordination of Licensing and Environmental Impact Study -CLEIA, a sector of SEMA-MT specialized in analysis of Environmental Impact Studies.

20.4 ENVIRONMENTAL PERMITTING

20.4.1 BRAZILIAN AND STATE REGULATORY SCENARIO

Mining activities require preliminary Environmental Permitting, regardless of the necessary procedures with ANM (National Mining Agency), as defined by Federal Law No. 6,938/81, which established the National Environmental Policy.

Annex I of CONAMA Resolution No. 237/97 lists the activities and undertakings that use environmental resources, effectively or potentially polluting, that are subject to Environmental Permitting. The Federal Law No. 6,938 / 81 and CONAMA Resolution No. 237/97 define three (3) types of environmental license, namely:

- Preliminary License

Issued in the preliminary stage of the project planning. It validates the location and design, attesting to the environmental feasibility, and establishing the basic requirements and conditions to be met in the next phases of its implementation.

- Installation License:

  Authorizes the installation of the enterprise in accordance with the specifications of the approved plans, programs, and projects, including environmental control measures and other conditions.

- Operation License:

  Authorizes the operation of the enterprise, after verifying the effective fulfillment of previous licenses, environmental control measures, and conditions determined for the operation.

The Federal Law No. 6,938/81 assigned to the STATES the power to license activities located within their regional limits. If the undertaking develops activities in more than one state, or if the environmental impacts exceed the territorial limits, IBAMA will be the body responsible for the grant of permits.

In the case of the Matupá Gold Project, located in the state of Mato Grosso, SEMA-MT (Secretaria de Meio Ambiente do Estado do Mato Grosso) is the agency responsible for environmental permitting and issuance of all Permits and Authorizations. Other play in the state scenario is the CONSEMA (Environment Counsel of Mato Grosso State) which is in charge of endorsing the grant of Preliminary License.

20.4.2 LICENSING STATUS

The EIA/RIMA was filed with SEMA-MT on November 30, 2021 in compliance with the Terms of Reference issued by SEMA-MT. Aura made a presentation of the Matupá Gold Project to the technical team of the environmental agency on March 15, 2022.

The Public Hearing, for the presentation of the Matupá Gold Project to the local community, followed the instructions of SEMA-MT and was held on May 10, 2022 in Cuiabá, with a virtual transmission on Youtube by SEMA-MT and Aura Minerals, and with a face-to-face point at the Matupá City Council. In general, the Matupá Gold Project was well received by the population and also by the mayors of Matupá and Guarantã do Norte.

The next steps in environmental licensing are the visit of the SEMA-MT technical team to the Project area, analysis of the EIA, and the demands of the public hearing and, finally, the issuance of the Technical Opinion and Preliminary License.

It is estimated that the Preliminary License (LP) will be issued by SEMA-MT and endorsed by the Mato Grosso State Environment Council (CONSEMA), around the third and fourth quarter (Q3, Q4) of 2022. The Installation License will be required between the third and fourth quarter of 2022 and its issuance is expected by the end of the first half of 2023. The Operating License will be required 4 months prior to completion of construction.

During this process, accessory licenses will be required, such as the Water Use Grant and Effluents Release Grant, Vegetation Clearing Permit, and others that may be required to complete the licensing process. The studies and projects necessary to support the application for these licenses are in progress for proceeding in the first half of 2022.

20.5 ENVIRONMENTAL AND SOCIAL BASELINE

20.5.1 CLIMATE

The climate in the region of the Project is characterized as hot and humid, with little thermal variability throughout the year, with average monthly temperatures ranging between 26.3°C and 24.4°C and the highest temperatures recorded between the months of June, 33.1°C, and October, 33.5°C. It has 2 well-defined seasons: the dry season, between June and September, and the rainy season, between October and April. Average annual rainfall between 2,000 and 2,500 mm, with high water surplus, and dry season characterized by water deficiency. There is an annual water surplus in the order of 1,197 mm, with the maximum surplus in the months of December (207 mm), January (266 mm), February (299 mm) and March (197 mm). The highest records of evaporation and relative air humidity follow the annual seasonality of the dry and rainy seasons.

20.5.2 WATER RESOURCES

20.5.2.1 HYDROLOGY

The X1 Deposit is located in the Amazon River Basin, in which the Tapajós River is one of the main affluents. The main watercourse in the Project area is the Peixoto de Azevedo River, a tributary of the Teles Pires River, which in turn flows into the Tapajós River. Locally, the X1 Deposit is fully within the Porcão River basin, which is about 29 km long and has a drainage area of approximately 154.36 km², with its source in the municipality of Guarantã do Norte and its mouth in the municipality of Matupá, flowing into in the Peixoto de Azevedo River.

In the surroundings of the enterprise, 6 watersheds were demarcated and 44 springs were mapped. The drainages have a radial pattern, converging to the west, and empty into the Porcão River. The main channels are perennial; however, some springs dry up completely in the dry season and the volumes and flows of streams reduce significantly when compared to the peaks of the rainy season. Figure 20-1, shows the drainages and microbasins of the Matupá Project area.

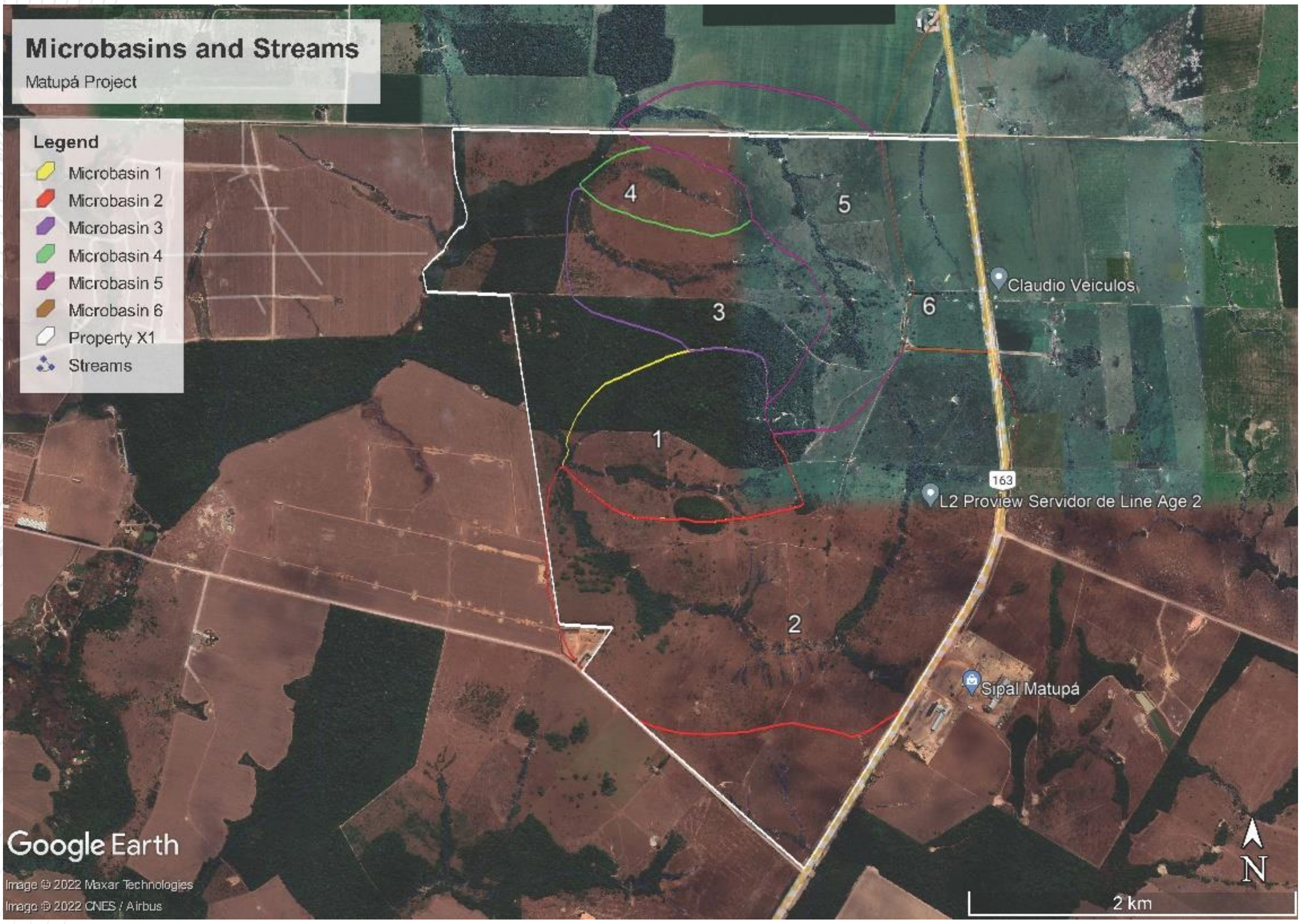


Figure 20-1 – Microbasins and Streams in the Matupá Gold Project Site.

In order to assess the possible impacts of the Project on the Porcão River basin, a query was carried out in the SIMLAM application (Integrated Environmental Monitoring and Licensing System) on the SEMA/MT (Secretary of State for the Environment) website regarding water availability in 3 Porcão River basin locations:

- Location 1: Pickup point planned for the future project;
- Location 2: Mouth of the Porcão River; and
- Location 3: Peixoto de Azevedo River, immediately after the mouth of the Porcão River.

The raw water demand for the operation will be around 56 m^3/h (0.01556 m^3/s). Simulations of abstraction of 56 m^3/h in the Porcão River show that the available water capacity of the basin is sufficient to meet the requested flow in all months of the year. Even at the planned catchment point, which is located further upstream of the basin, the volume captured commits 2.66% of Q95 (corresponds to the flow that is present in the river during at least 95% of the time) in that stretch. The total impact on the Porcão River basin will be smaller, corresponding to 1.36% of the river's Q95. The impact on the Peixoto de Azevedo River basin represents 0.017% of the Q95 of this body of water.

Figures 20-2, 20-3 and 20-4 show, respectively, the comparative graph of the volume that will be captured versus Q95 at the catchment point, at the mouth of Porcão River and at the Peixoto de Azevedo River, immediately after the mouth of the Porcão River.

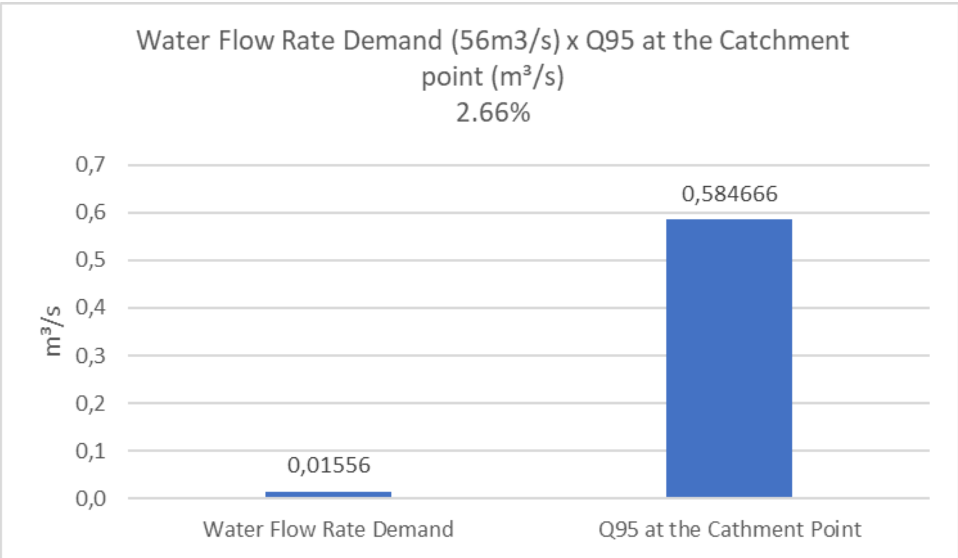


Figure 20-2 -WaterCcatchment Rate Versus Q95 Catchment Point.

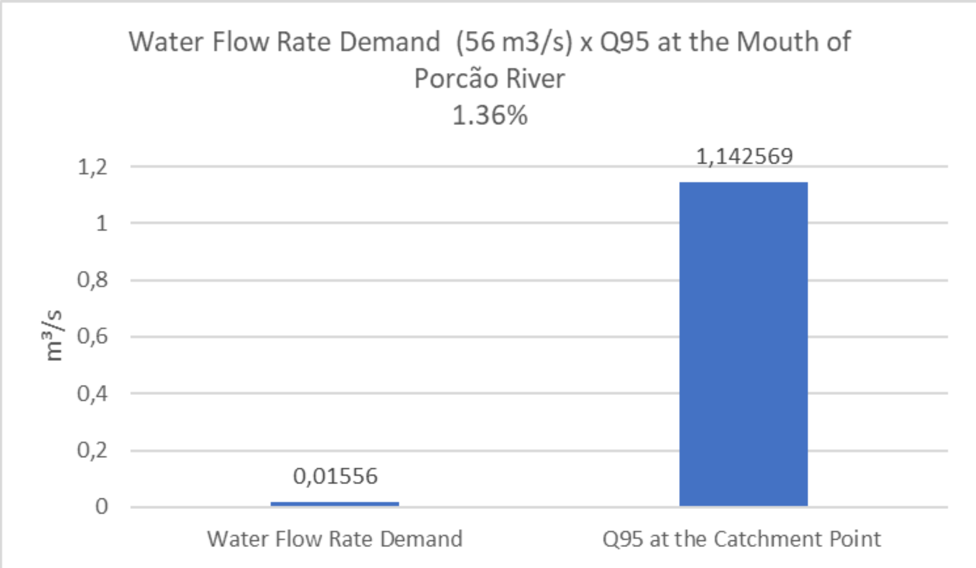


Figure 20-3 - Water Catchment Rate Versus Q95 at the Mouth of Porcão River.

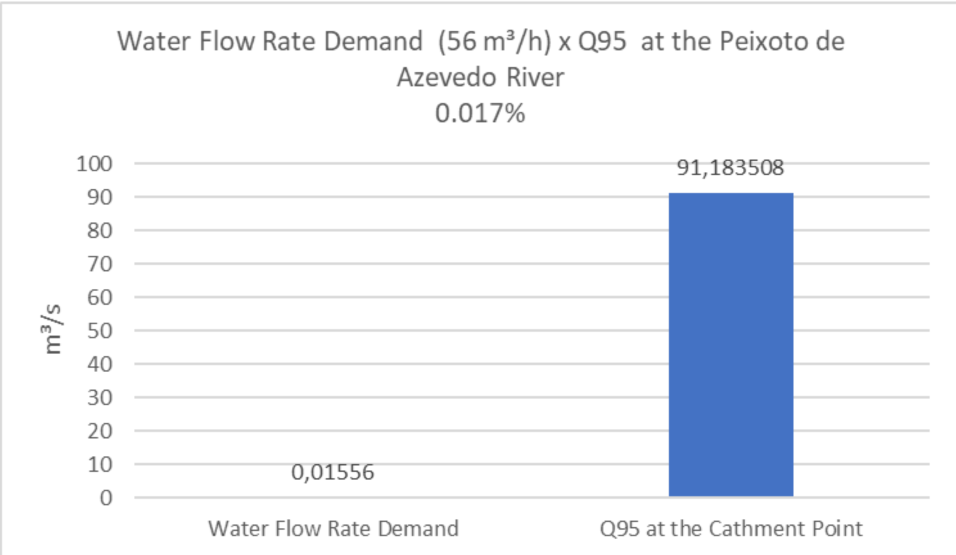


Figure 20-4 - Water Catchment Rate Versus Q95 at the Peixoto Azevedo River.

Therefore, it was decided to collect water from the Porcão River because it does not compromise the flow of the river, it is not necessary to suppress native vegetation and because of the lower interference in local properties when compared to the capture in Peixoto Azevedo River.

20.5.2.2 SURFACE WATER QUALITY

Surface water samples were collected in the two climatic periods (rainy and dry seasons) in the area of influence of the enterprise, with some specific changes for thermotolerant coliforms, probably related to animal waste that use the vicinity of the collection points for drinking water, as well as changes in the samples related to the parameters Phenols and Polycyclic Aromatic

Hydrocarbons (PAH) in low concentrations, which may be related to the fires that are very common in the dry season. In none of the samples were significant changes in pH.

20.5.2.3 WATER BALANCE AND WATER MANAGEMENT

As there is a surplus of water during the rainy season, the project provides for containment structures (sumps) located around the tailings, waste rock and low-grade ore piles. A drainage system will be implemented both in the pit and in the piles that will direct the water to the sumps to prevent it from falling directly into the water courses without proper treatment, if monitoring indicates the need.

The set of sumps will play a fundamental role in water management, working, in the dry season, as a water reservoir for maintenance of the operation and, in the rainy season, absorbing the water surplus. Another important function is to act as the place for the treatment of effluents from the piles and pit, in case the acid drainage potential is confirmed. Thus, the treatment will be carried out in the sumps close to the generating sources, using Lime Milk to neutralize the acidity and abatement of metals and the water will only be discharged into the streams after meeting the water quality standards and effluent discharge (CONAMA Resolution No. 357/2005 and CONAMA Resolution No. 430/2011). Section 18 presents the description and drawings of the drainage system and sumps for the pit, tailings, waste rock and low-grade ore piles.

In order to evaluate the hydraulic behavior of the sumps for the waste rock, low-grade ore and dry staking tailings piles during the rainy season, Promon prepared the water balance for this system, analyzing the volume of water stored, the volume released into the environment and the volume to be reused by the mill plant.

To carry out the study, the concept of intense rainfall with a return period of 25 years was used. Three scenarios were simulated for each sump with 2 rains, prioritizing the alternatives with a lower volume of water released into nature and a greater volume of water returned to the mill plant.

Regarding the volume of rainwater for reuse in the process, only the final Sumps N4 (low-grade ore pile) and S3 (waste rock pile) were considered, due to their proximity to the beneficiation plant. Approximately 931 m³ was counted for reuse in the first intense rainfall and 22,286 m³ in the second intense rainfall period.

As for the volume of excess water to be released into the environment, Sumps N2, N3, N4 (low-grade ore pile), S2, S3 (waste rock pile) and R3 (dry staking tailings pile) were considered. In this case, the volumes are 23,446.50 m³ in the first intense rainfall, and 73,182.50 m³ in the seconnd intense rainfall period.

For the release of excess water to the environment, it will be necessary to apply to the SEMA-MT for an Effluent Release Grant for each water discharge point. These permits are part of the accessory licenses process described in section 20.4.2 – Licensing Status.

20.5.2.4 EFFLUENT TREATABILITY STUDY

Effluent treatability tests are in progress at Campo Laboratory in Paracatu, Minas Gerais, and are being carried out in a Jar Test with two different effluents, produced based on the solutions used in the Net Acid Generation Test ("NAG") prediction tests, in samples of waste rock and tailings. The idea is to simulate effluents, in their worst composition in terms of acid drainage generation, coming from waste rock and tailings piles. The treatment chemical substance used in the tests will be Cal, $Ca(OH)_2$.

The main function of these tests is to evaluate the capacity of lime - $Ca(OH)_2$ - to fit the produced effluents to Brazilian legal standards, aiming at a simplified and efficient treatment operation.

The tests will be carried out in 3 stages:

- Phase 1 (Demonstration of the chemical principle).

- Phase 2 (Optimization of chemical equation components).
- Phase 3 (Optimization of the pH of the chemical reaction).

In the first phase, tests are carried out covering a wide range of $Ca(OH)_2$ addition to determine approximately how much lime will be needed to remove heavy metals and raise the acidic pH of the water.

In the second phase, the results of the first phase will be optimized, aiming to determine the best lime dosage for the effluents.

The third phase will be conducted to determine the maximum pH that can be reached with the optimal dosage of lime determined, in addition to determining the optimal contact time for mixing lime with the effluents. In this phase, the optimal conditions for precipitation of oxides and hydroxides formed during the treatment of effluents will also be determined.

The end of the laboratory tests is expected for September 2022. With the results in hand, a check can be made with the treatment assumption adopted by engineering, correcting the dosage, if necessary.

20.5.2.5 HYDROGEOLOGY

Mathematical flow modeling studies were carried out for the X1 Deposit area with the objective of evaluating the possible influences of the opening of the pit and its effects on the springs located in adjacent areas, as well as the hydrodynamics of the surroundings close to the proposed mineral project.

In the development of the modeling, in addition to the information on the physical environment obtained through bibliographic references, parameters obtained directly from the hydraulic characterization were used from the installation of one (1) pumping well, 6 monitoring piezometers and the performance of a pumping test. The data thus obtained were applied in the construction of the model and in its calibration.

The simulations obtained indicate that the hydrodynamics of the surroundings near the pit should not be altered by the drawdown activity, due to the low interaction between the characterized aquifer systems and the contrast of their hydraulic properties.

The flows measured in the field and modeled remain practically unchanged in relation to the downgrade scenario. These data confirm that the volumes that support the flows from the springs close to the X1 Deposit are the result of direct surface runoff and the transfer of discharge volumes from the granular aquifer system, supported by rock alteration.

20.5.3 ACID MINE DRAINAGE POTENTIAL

To assess the potential for generating acidity, static prediction tests (MABA and NAG) were conducted on 107 representative samples of waste rock material and 2 representative samples of tailings at the SGS/GEOSOL Laboratory during the first half of 2021.

The static tests indicated that the two tailings samples and an important percentage (71-84%) of the waste rock material samples showed potential for acidity generation. Most of the sulfur contained in the samples is associated in the form of sulfides. The carbonate contents are very low in the sample as a whole. In more than 46% of the samples, the carbonate contents were below the detection limit.

According to the NAG tests, in terms of predicting the acidity potential, there is possibly a cut-off around 1% for sulfur sulfides, which could be a cut-off content for generating and non-generating materials. This concentration of sulfide can be used in the block model to evaluate the possibility of segregation of materials, in order to use these materials for the encapsulation of generating materials.

The results of the static tests indicated the need to carry out kinetic tests to evaluate the reactivity of the materials (waste rock and tailings) and also the quality of the drained water.

Both tailings and waste rock samples were sent to the SGS/GEOSOL Laboratory for the Free Draining Kinetic Column Leach test to confirm the potential for generating acidity and mobilizing metals, especially for those samples with contents lower than 1%. of sulfur sulfide content.

Regarding the tailings, the tests started at the end of September 2021, with two samples, one derived from fresh rock (MATUF-RJ1-DT1) and the other from oxidized material (MATUO-RJ1-DT1), in addition to a blend formed by the two original samples in the proportion (by mass) of 90% MATUF-RJ1-DT1 and 10% MATUO-RJ1-DT1, which probably represented the final tailing to be formed according to current data.

The results obtained from these samples in the static tests point to the potential for generating acid associated with the sulfide material. However, "Free Draining Kinetic Column Leach" tests did not confirm the generation of acid drainage suggested in the static tests, nor even the relevant and persistent metal leaching.

For the waste rock, 9 samples were selected from those tested in the static tests and used in the kinetic tests which started in December 2021. The 9 samples were selected by choosing the most representative and abundant lithotypes of the mine in order to confirm the results of the static tests, which indicated the possibility of using a cut-off content of sulfides around 1% in the separation of generating and non-generating materials of acidity. The results of the waste rock tests suggest that only one sample is potentially generating acidity so far, with only significant concentrations of leached metals (copper and manganese) being observed.

The scenario presented in the kinetic tests, both for the tailings and for the waste rock, is more positive than that previously suggested by the static tests. However, as the specialized literature recommends that the evaluation of the acid drainage potential, through kinetic tests, be at least 20 weeks, the specialized consultancy suggests an additional 20-week cycle for the tailings and waste rock samples tested.

If, after 20 weeks, the samples do not generate acid drainage and metal leaching, it is suggested that these same samples be evaluated in relation to their static potential, as well as the chemical and mineralogical composition focused on the sulfides present and neutralizing minerals for a more consistent conclusion.

20.5.4 VEGETATION

The Project is located in the Amazon Biome, with the Submontane Open Ombrophilous Forest being the representative vegetation of the region. Over the years, part of the forest formations were replaced by pasture and agriculture. The diagnosis of vegetation cover carried out in the area of the Matupá Project, X1 Deposit identified pasture (82.97%), Ombrophilous Forest (11.25%) and Riparian Forest (5.75%) as the main vegetation formations, demonstrating that most of the area where the Matupá Project, X1 Deposit will be developed is used for livestock. Table 20-1 shows land use in the area of direct influence of the enterprise.

In the existing forest fragments, 34 species belonging to 23 botanical families were recorded, the most abundant species being Siparuna guianensis (negramina). Isolated arboreal individuals were also identified amidst the pasture in the Project area, being two endangered species Bertolletia excelsa (Brazil Nut Tree) and Cedrela fissilis (cedar).

Table 20-1 - Land Use in the Area of Direct Influence of the Matupá project.

Land Use and Occupation	Area (ha)	%
Pasture	98.50	82.97
Riparian Forest	6.82	5.75
Submontane Open Ombrophilous Forest	13.36	11.25
Bodies of Water	0.04	0.04

20.5.5 WILDLIFE

Field surveys were carried out in the two climatic seasons (rainy and dry) for the identification and evaluation of the terrestrial and aquatic fauna that occurs in the surroundings of the enterprise.

Regarding medium and large mammals, 23 species were recorded in the forest fragments of the study area. Some are considered common and widely dispersed, such as the marmoset (Mico sp) and capybara (Hydrochoerus hydrochaeris). Others are more demanding, top predators of restricted occurrence, such as black-faced spider monkey (Ateles chamek) and puma (Puma concolor). Among the endangered species are the puma (Puma concolor), tapir (terrestrial tapirus), black-faced spider monkey (Ateles chamek) and peccary (Tayssu pecare).

With regard to birds, 183 species were recorded, and nationally, the endangered Monasa morphoeus and the vulnerable Capito dayi (captain-de-girdle) stand out, and regionally Pteroglossus bitorquatus (red-necked aracari). In the international sphere of conservation, the species Pteroglossus bitorquatus is classified as endangered and the birds Capito dayi and Ramphastos vitellinus (white-backed toucan) as vulnerable.

During the 2 field campaigns, 203 individuals of reptiles and amphibians were recorded, distributed in 23 species. No endangered species were identified in the region, however boa constrictor (Boa constrictur) and teiuaçu lizard (tupinambis teguixin) are registered in the appendices of CITIES, 2021 (International Trade in Endangered Species of Wild Fauna and Flora) as non-endangered species that could be endangered if traded without controls. Three species of unique occurrence in the Amazon were also recorded, such as the frog (Lithobates palmipes) and the tree frog (Ostheocephalus taurinus) and (Callimedusa tomopterna), the latter being considered rare.

With regard to fish, 51 specimens were recorded that occur in the Project area, through fishing and collection, which are distributed in 21 species. Despite the occurrence of 10 species restricted to the Amazon region, they are not considered endemic to the area of the Project and no endangered species were found.

20.5.6 SOCIOECONOMIC ASPECTS

The Project area is located in the rural area of the municipality of Matupá, surrounded by rural properties.

The municipalities of Matupá, Gurantã do Norte and Peixoto de Azevedo were considered in the diagnosis and assessment of the impacts of the EIA/RIMA as an area of influence for the Project, with Matupá, where the project will be installed, as an area of direct influence.

The population growth rate in the municipality of Matupá is considered high. The Demographic Density of the municipality is 3.21 inhabitants/km², being the points of greatest population concentration, predominantly in urban areas and the urbanization rate of 77.09%.

In the area of influence of the enterprise and in the municipality of Matupá, the social groups are distributed among farmers, extractivists (garimpeiros), settlers, and fishermen. Only the miners and rural farmers classes are represented by unions. The miners are part of the Cooperativa dos Garimpeiros do Vale do Rio Peixoto (Coogavepe). In the area directly affected by the Project, there are only social groups that are part of the universe of rural farmers.

With regard to educational services, there are 16 teaching establishments in the municipality of Matupá, with 4,584 enrollments, with a teaching staff of 320 teachers, distributed between kindergarten, elementary and high school. The municipality reached an average above those projected by the Brazilian Basic Education Development Index (Ideb). However, according to IBGE (Brazilian Institute of Geography and Statistics) data, 12.8% of the population aged 15 and over is not literate.

With regard to the basic health structure, the municipality of Matupá has health facilities that are part of the existing care network in the municipalities in the area of influence of the Project, belonging to the federal, state and municipal spheres, being part of the Region of Health 11- Peixoto Azevedo Regional.

Matupá city hall has 15 health establishments, a low coverage of hospital beds, with 54 beds, without ICU. The number of staff is low, with only 11 health professionals with higher education. These data show that the health structure found in the municipality does not follow the population growth in the region, which is a national reality.

Health conditions for endemic diseases are currently stable in the municipality of Matupá, however, it has a high incidence of Dengue, according to data from Bulletin No. 16 Ed.01 SE-18/2017 of the state of Mato Grosso. It is also worth mentioning some cases of malaria in the area of influence of the Project, which was corroborated by the study of the potential for malaria carried out in 2021 for the Environmental Impact Assessment, which confirmed positive cases in the 3 years analyzed, thus having the possibility of malaria in the Project site region.

An important and positive fact is that the infant mortality rate for live births decreased from 15.87 for 2017 to 11.2 for 2019.

Basic sanitation and energy supply are primarily in the hands of private companies. There is good service in relation to water supply, with service for 98.59% of the population of the urban area of Matupá. The municipality has an ETA (Water Treatment Station) under construction and about 15.38% of sewage network coverage in the structuring phase. Garbage collection is carried out by city hall by means of a compactor truck and a team composed of employees of the city hall itself, 3 times a week during the day, and serves 12 urban locations. For energy, 100% of the municipality benefits from the distribution of electricity by the private company Energisa.

The security sector had a considerable increase in the Project area, with the crimes of robbery and theft as the highest growth in the region. The area of influence of the Project is served according to the organizational structure of the military and civil police.

The total employed population of Matupá in 2019 was 4,662, 85.19% of these were salaried people with an average monthly salary of 2.2 times the minimum wage.

The economic activities of Matupá and the region are mainly based on agricultural production with a specialty in soy and corn “commodities”, demanding production infrastructure, machinery, inputs and specialized advice involving, in addition to the primary sector, the secondary and tertiary sectors, linked to the processing of the so-called agro-industry, which in 2018 represented 19.86% of the total GDP of Matupá.

20.5.7 LAND USE AND OCCUPATION

The municipality of Matupá is mostly occupied by forest formations (58.60%), followed by pasture areas (30.70%). Temporary crops occupy about 14,000 hectares, mineral extraction areas concentrate approximately 6,800 hectares, and urbanized areas a total of 809 hectares. The surroundings of the area where the Matupá Project will be developed are mostly made up of pastures (39.49%), agriculture (28.87%), and industrial use (12.16%).

Figure 20-5 shows the vegetation cover and the use and occupation of the land in the area of direct influence of the Project.

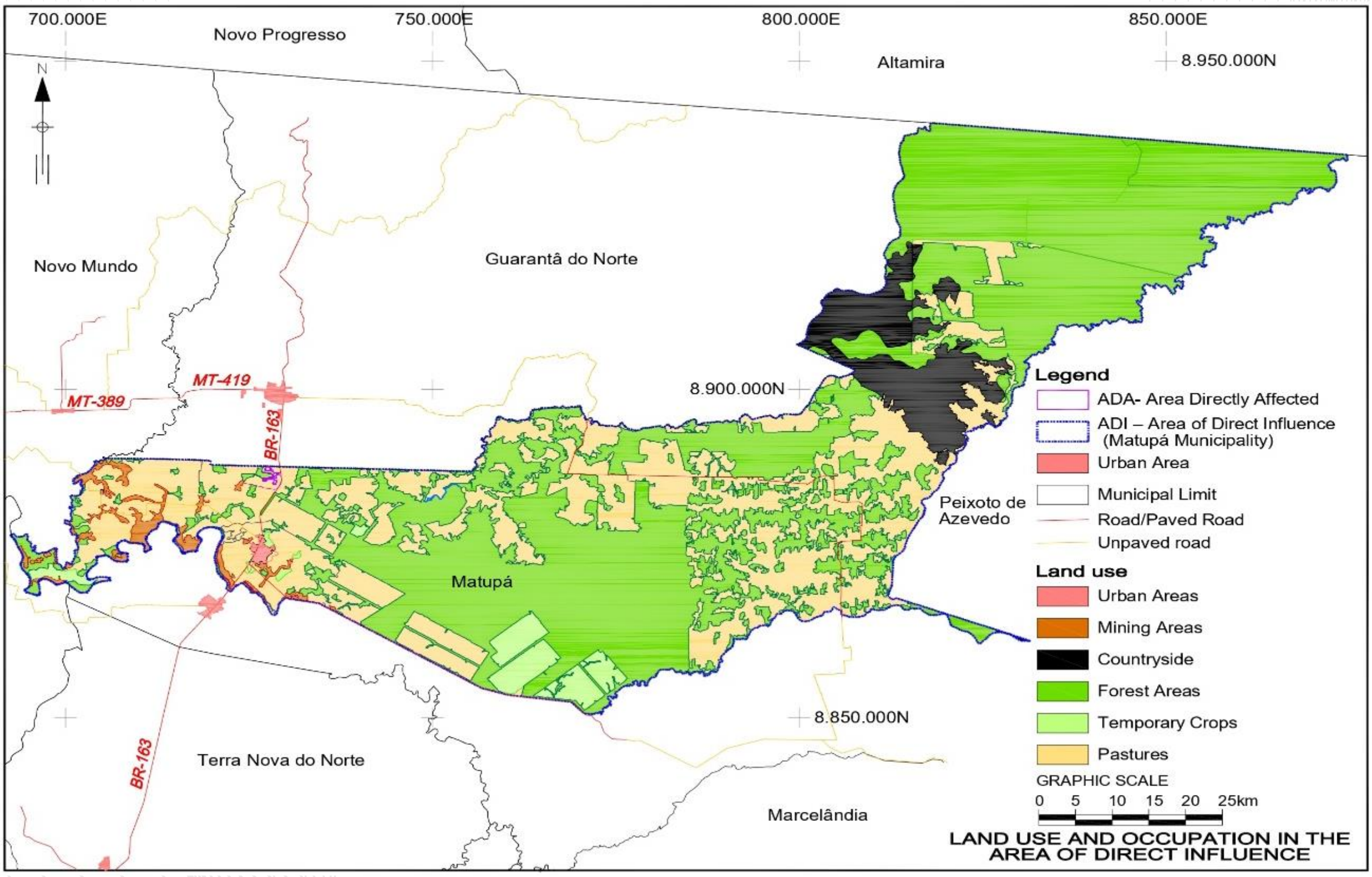


Figure 20-5 - Map of Land Use of Matupá Municipality.

20.5.8 LEGALLY PROTECTED AREAS AND TRADITIONAL POPULATIONS

The Project area is not located within Conservation Units (Parks, Environmental Protection Areas, Ecological Stations, etc.) nor in their buffer zones. Indigenous Lands, Quilombola Communities and settlement projects in their area of direct influence were also not identified as part of the Project area.

20.5.9 ARCHAEOLOGICAL HERITAGE

The execution of the Impact Assessment on Archaeological Heritage - AIPA, was authorized by Institute of National Historic and Artistic Heritage ("IPHAN") through Ordinance No. 47 of August 6, 2021, published after analysis of the research project with the objective of complying with the Specific Term of Reference (TRE) No. 05/IPHAN-MT, of 05/Feb/2021. The criteria and methodologies adopted by this Assessment sought to comply with Normative Instruction (NI) No. 01 of March 25, 2015, which establishes the administrative procedures to be observed by IPHAN in environmental licensing processes and other legislation in force.

The Archaeological Research and Impact Assessment Project on the Archaeological Heritage was under the responsibility of the consulting company *A Lasca*, from the city of São Paulo, SP, but with extensive experience in projects in Mato Grosso.

The Impact Assessment on Archaeological Heritage - AIPA, in the Directly Affected Area - ADA, of the Matupá Project, X1 Deposit, was carried out in 2021 between August/30 – September/10 and did not result in the identification of archaeological remains.

The Final Impact Assessment Report on Archaeological Heritage was filed with IPHAN in early October 2021, which, after evaluating the aforementioned report, issued Official Letter No. 878/2021/IPHAN-MT on October 19, expressing its approval of the report and favorable to the issuance of the Preliminary, Installation and Operation Licenses for the enterprise within the scope of the environmental licensing in progress at the Environmental Agency of Mato Grosso - SEMA-MT.

20.6 MAIN IMPACTS

The impacts listed below deserve to be highlighted, between positive and negative:

- Stimulating the local and regional economy;
- Tax collection;
- Job and income generation;
- Interference in the processes of superficial soil dynamics;
- Lowering of the water table;
- Interventions in drainage and intermittent water holes;
- Native vegetation loss and fragmentation;
- Disturbance of wildlife;
- Loss of habitats and alteration in ecological processes; and
- Acid Drainage Generation Potential.

The positive impacts identified for the Matupá Project, X1 Deposit, both in the implementation and operation phases, are those related to the socioeconomic environment, such as the dynamics of the local and regional economy, tax collection and generation of employment and income, bringing benefits not only for the municipality of Matupá, but also for Peixoto Azevedo and Guarantã do Norte.

Among the negative impacts, those related to the clearing native vegetation, water resources, and the potential for generating acid drainage, stand out.

As already described in section 20.5.4 Vegetation, the area and region where the Matupá Project, X1 Deposit, will be developed, has, over the last few years, undergone a process of replacing the vegetation cover for pasture and agriculture, thus, for the implementation of the Matupá Project, X1 Deposit, it is estimated the removal of 14.72 ha of the remaining Submontane Dense Ombrophylous Forest (12.38% of the Directly Affected Area) and 4.97 ha of the remaining riparian forest (4.18% of the Directly

Affected Area). In addition to the clearing in the forest fragments mentioned above, there will be the removal of 148 isolated arboreal individuals.

Such environments, forested or not, constitute important areas of foraging, shelter, refuge, and reproduction for several species of animals. However, it should be noted that the implementation of the Project in question will not cause a significant increase in the fragmentation of the regional landscape, and that the species of animals found in the field surveys in the area of the Matupá Project were also verified in fragments located in the areas of influence of the Project.

Direct interventions in water courses and intermittent water bodies can result in a negative impact on the recharge of surface waters. However, the hydrogeological studies carried out have verified that the activities of lowering the water table, necessary for the opening of the pit, will not imply changes in the water availability of the springs and water points of the area of direct influence of the Matupá Project, since the volumes that sustain the flows of the springs close to the X1 Deposit are the result of direct surface runoff and the transfer of discharge volumes from the aquifer system granular, supported by rock weathering. This possible impact will be systematically monitored throughout the implementation and operation of the enterprise.

The potential for generating acid drainage suggested in the static tests was not confirmed in the kinetic tests, however, preventive and control measures are already being adopted in the engineering project, as presented in section 18, such as lining at the base of the piles, drainage systems and sumps for containment of solids, and eventual treatment with milk of lime to remove metals at the level of water quality from water courses, according to current environmental legislation.

20.7 MITIGATING MEASURES AND PLANS

For each environmental impact identified for the Matupá Project in the EIA/RIMA, measures were proposed for the prevention, control, minimization, and compensation of negative impacts, as well as the enhancement of positive impacts.

These measures are organized into Environmental Plans and Programs that must be carried out by the Company during the different phases of theProject. It is important to highlight that the proposed actions correspond to a first instrument of environmental management and planning for the Matupá Project and that they should be detailed and expanded throughout the planning, implementation, and operation phases of the Project.

Plans and Programs are listed in Table 20-2 below.

Table 20-2 - Social and Environmental Plans and Programs.

Programs	Planning	Implantation	Operation
Environmental Management Program (EMP)			
Social Communication Program (SCP)			
Environmental Education Program (EEP)			
Environmental Plan for Construction (EPC)			
Degraded Area Recovery Program (DARP)			
Risk Management Program (RMP)			
Water and Sediment Quality Monitoring Program			
Program for the Control and Monitoring of Erosive and Sedimentation Processes			
Noise and Vibration Monitoring Program			
Air Quality Monitoring Program			
Hydrogeological Monitoring Program			
Hydrological Monitoring Program			
Geotechnical Monitoring Program			

Programs	Planning	Implantation	Operation
Terrestrial Fauna Monitoring Program			
Fish and Aquatic Biota Monitoring Program			
Terrestrial Wildlife Rescue Program			
Terrestrial Wildlife Signaling Program			
Program for Monitoring the Clearing of Native Vegetation and Rescue of Germplasm and Epiphytes			
Forest Replacement Program			
Workforce Training and Qualification Program			
Program of Actions with the Community and Government			

20.8 MINE RECOVERY AND CLOSURE

The Degraded Area Recovery Plan ("DARP") and Closure Plan, presented in the Environmental Impact Assessment of the Matupá Project, establish guidelines for planning recovery and closure, in addition to general recovery measures to be taken during and after mining, to ensure progressive rehabilitation by making the site close to pre-mining conditions. In addition to revegetation efforts, other important recovery measures to be implemented include land topography regularization, drainage, and slope stabilization.

The following is a summary of the main actions to be carried out:

- Pit: recovery of the final slopes in the soil of the pit with the use of up to two species of grasses and two species of native vines of fast growth and intercropping, so that they promote the rapid colonization of the slopes, since there will be no physicochemical conditions to sustain a more complex plant community composed of tree and shrub species;
- Mill Plant and auxiliary facilities: dismantling, demolition of civil facilities, removal of infrastructure and foundations, regularization of the land surface and drainage for rainwater drainage, covering with organic soil and revegetation;
- Waste Rock Pile: reconfiguration of slopes, review of the surface and internal drainage system, covering with clay, organic soil and revegetation;
- Dry Staking Pile: cover with clay layer, organic soil layer and revegetation;
- Access roads: accesses that are not expected to be used in new activities in the area will be restored after their use has ended, through subsoiling techniques, unpacking, and subsequent planting of grasses and legumes;
- water and tailings pipelines: dismantling, removal and recovery of areas along the water supply line with grasses, legumes and native tree species;
- Pumping system for capturing raw water and pumps: demolition and removal;
- Explosives emulsion factory and auxiliary facilities: demolition, removal, land regularization, soil preparation, cover with organic soil, and revegetation with grasses, legumes and native tree species;
- Removal of towers, substations and lines, including all foundations, land surface regularization, soil preparation, cover with organic soil, and revegetation with grasses and native tree species;

- Recovery of all degraded areas, from mining activities, within the limits of the mineral exploitation property.

- All recyclable/reusable materials will be processed and/or disposed of in accordance with Brazilian legislation.

After the closure, a program will be implemented to monitor the recovered areas and revegetation, monitoring of surface and groundwater, geotechnical monitoring and erosive processes monitoring of waste rock and dry staking piles, and wildlife monitoring to identify any possible alteration or contamination that could affect the environment and make the necessary corrections. The costs for recovery, closing, post-closing and monitoring involving all the actions described above are estimated at US$ 203,965,718.50, as summarized in Table 20-3 below.

Table 20-3 - Closure Costs Summary.

Description	Value (US$)
Administration	558,286.89
Reclamation Executive Project	234,615.38
Dismantling and Demolition	3,593,684.76
Tographic reconfiguration and drainage system	438,755.94
Revegetation	1,615,188.51
Comunication Program and Monitoring	818,500.00
Contingency	508,132.20
Total	**7,767,163.69**

21. CAPITAL AND OPERATING COSTS

21.1 CAPITAL COSTS

The CapEx estimation contains all costs related to assembly, construction, equipment, materials necessary for the implementation of the Project, as shown in Table 21-1. The variation of the CapEx estimative is over 15% and less than 10% of the total estimated investment. This section is divided into services, supplies, mine, piles, transmission line, and indirect costs. The estimates were based on quoted, estimated, or historical values, together with values provided by Aura, based on experience in the sector and similar projects. All tables are quoted in U.S. dollars, based on an exchange rate of R$ 5.20 (Brazilian reais) for $1.00 (U.S. dollar). The values presented consider the tax benefits reported by EY Consulting. The technical items were evaluated and validated by Aura. The complete table of values stated in document MTP-B-RL-0000-PRO-V-0003 indicates those responsible for each considered value.

Table 21-1 - Overall CapEx Estimative.

Item	FEASIBILITY	
	Cost (US$)	%
Services	$32,177,943.88	30%
CIV01 - Administrative buildings (project, material, and construction)	$2,495,265.06	2%
CIV02 - Processing Plant (materials and construction)	$11,019,472.65	10%
CIV03 - Temporary installations, earthworks, access, and drainage.	$2,992,888.63	3%
MON-01 - MONTAGEM ELETROMECÂNICA	$14,960,778.20	14%
MON-02 - MONTAGEM PEAD OFF SITE	$709,539.35	1%
Supplies	$40,526,051.16	38%
Electric	$10,974,822.83	10%
Mechanics	$23,544,312.75	22%
Instrumentation	$1,877,981.23	2%
Civil	$1,410,456.95	1%
Pipes	$2,430,015.87	2%
Fire detection and alarm system	$288,461.54	0%
Mine, Piles, and Transmission Line	$14,109,618.30	13%
CIV04 - MINE	$4,930,250.94	5%
CIV05 - REJECT PILE	$2,888,330.62	3%
CIV06 - STERILE PILE	$1,521,765.53	1%
CIV07 - LOW CONTENT PILE	$1,182,728.94	1%
MEC-XX - SAMPLE LABORATORY	$619,527.29	1%
ELE-TK-XX - SUBSTATION SUITABILITY AND TRANSMISSION LINE	$2,808,522.23	3%
ENERGY FOR TEMPORARY INSTALLATIONS	$120,031.20	0%
Deep capture wells	$38,461.54	0%
Indirect Costs	$13,008,309.51	12%
IND-01 - EPCM	$4,794,304.45	4%
IND-01 - Assembly supervision	$156,505.77	0%
IND-01 - Spare and Special Items	$410,147.58	0%
IND-01 - Owner Cost	$4,026,760.12	4%
IND-01 - shipping	$-	0%
IND-01 - First Fill	$426,435.75	0%
Starter spares	$24,990.00	0%
IND-01 - Indirect Field Construction	$1,715,473.35	2%
IND-01 - Engineering Risk Insurance	$1,367,154.04	1%
IND-01 - Expediting and inspection	$86,538.46	0%
Subtotal	**$99,821,922.86**	**93%**
Contingency	$7,300,848.08	7%
Total Investment	**$107,122,770.94**	**100%**
Less Limit (-10%)	$96,410,493.85	
Upper Limit (+15%)	$123,191,186.58	

Table 21-1 indicates the source of information for each item, together with its corresponding value, both in terms of US$ and percent of total CapEx. The complete CapEx document issued by Promon describes values per item, the selected supplier, costing method, referred tax classification (NCM), quantities, units, description, responsible for the reported value, tag, among other information.

The assembly and construction costs were budgeted with suppliers specialized in this type of work. The costs related to the acquisition of electrical, mechanical, and instrumental equipment were budgeted by more than one supplier. In all cases, the criteria for selecting the supplier were the lowest cost, following a previous technical assessment. The same occurred with the selection of material suppliers. The costs of mine, piles, and transmission lines were provided by specialized consultants, as hired by Aura. Indirect costs were estimated based on the plant size, team, amount of equipment, budgeted items, among others. In addition to these costs, the estimative included costs associated with construction management, supervision of assemblies, spare parts, first filling, insurance, and other indirect values.

21.2 SERVICES

The items referring to services listed in Table 21-1 are associated with contracts for the execution of administrative buildings, process plan, temporary constructions, and electromechanical assemblies. The costs of all these items included assembly/construction labor, materials and indirect costs required for the scope, as provided by specialized suppliers.

21.3 SUPPLIES

The electrical, mechanical, and instrument equipment were budgeted with specialized suppliers, together with respective costs of freight, fees, including values for international deliveries. In each item, a technical assessment was carried out for selecting the corresponding instrument. In all cases, the criteria for selecting the supplier were the lowest cost, following a previous technical assessment. Part of material, fire detection, and alarm system was estimated based on Promon's historical database. The tax benefits were calculated according to the NCM of each item, as instructed by EY Consulting, the values are presented separately in the complete document Estudos Financeiros – Relatório de Estimativa de Investimento (CAPEX), MTP-B-RL-0000-PRO-V-0003. 2022..

21.4 MINE, PILE AND TRANSMISSION LINE

As costs of mining, piles, transmission lines, laboratory, energy for temporary installations are specific to mining companies, such items were assessed out by Aura, together with the support of specialized consultants. The costs for digging deep wells were estimated.

21.5 INDIRECT COSTS

Indirect costs include engineering, procurement, and construction management (EPCM), where the contracted company develops the project, purchase equipment and materials, as well as manages the construction process. Also included are the values associated with assembly supervision, costs for spare and starting items, costs related to the owner of theProject, freight, first supply, indirect costs of field, risk insurance engineering, expediting and inspection. These indirect costs were calculated together with Aura Minerals, which has a database related to indirect costs for existing plants similar to Matupá.

21.6 TAXES

All taxes included in the suppliers' proposals were considered, in accordance with current Brazilian tax laws. Taxes included in the capital estimate include:

- ISS (Imposto Sobre Serviços – Tax on Services);

- ICMS (Imposto sobre Circulação de Mercadorias e Serviços – Tax on Circulation of Merchandise and Services);
- PIS/COFINS (Programa de Integração Social/Contribuição para Financiamento da Seguridade Social – Social Integration Program/Contribution for Financing Social Security);
- DIFAL (Diferencial de alíquota do ICMS – Differential from ICMS), if applicable;
- IPI (Imposto sobre os Produtos Industrializados – Tax on Industrialized Products): as per fiscal classification of supply; and
- II (Imposto de Importação – Importation Tax) and applicable fees.

For CapEx, the following tax benefits were considered:

- Ex-Tariff - consists of a reduction of the Import Tax levied on imports of new capital goods;
- RECAP - consists of the suspension of the incidence of PIS and COFINS on the import and acquisition of capital goods whose NCMs is listed; and
- CONFAZ - consists of reducing the ICMS calculation base for listed NCMs.

21.7 TAXES BENEFITS

For OpEx only the percentage of reduction of 75% of the Income Tax (IR) was applied.

For CapEx, the following tax benefits were considered:

- Ex-Tariff - consists of a reduction of the Import Tax levied on imports of new capital goods;
- RECAP - consists of the suspension of the incidence of PIS and COFINS on the import and acquisition of capital goods whose NCMs is listed; and
- CONFAZ - consists of reducing the ICMS calculation base for listed NCMs.

The studies of tax benefits applicable to this project and its supplies were carried out by specialized consulting.

21.8 OPERATIONG COSTS

The 7-year period of plant operation was considered for the OpEx estimative. The values indicated in Table 21-2 are in dollars per tonne (of gold per year) of ore, run of mine.

Table 21-2 - OpEx for Matupá Project.

Item	Cost (US$) US$ / t (metric) ROM	%
	$22.71	100%
Labor (Fixed Costs)	$3.53	16%
G&A (Fixed Cost)	$1.69	7%

Item	Cost (US$)	%
	US$ / t (metric) ROM	
	$22.71	100%
Laboratory (Fixed Cost)	$1.26	6%
Access Maintenance (Fixed Cost)	$-	0%
Equipment rental (Fixed Cost)	$0.02	0%
Energy (Variable Costs)	$2.18	10%
Reagents and Consumables (Variable Costs)	$7.74	34%
Maintenance	$1.12	5%
Water and sewage treatment plant	$0.01	0%
Pile	$1.30	6%
Mine	$3.84	17%

The operation costs related to reagents and consumables were validated with Mineração Apoena's, a subsidiary of Aura, mining operation. The energy cost was calculated based on the new demands, validated by the Aura team. Values related to labor, G&A, and laboratory were calculated according to the Extended National Consumer Price Index ("IPCA") adjustment for the period. Values for access maintenance and equipment rental were provided by the Aura team, while mining and piling values were provided by specialized consultants (GHT and EDEM). Maintenance costs were calculated based on the CapEx costs of mechanical and electrical equipment. For water and sewage treatment stations, operating prices quoted by a specialized supplier were considered.

21.9 LABOR

Labor costs indicate operating costs for mining plant teams, such as managers, management areas, geology and mine planning. Plant management and maintenance teams include leaders, technicians, assistants, supervisors, engineers, and managers. The estimative also includes personnel dedicated to work safety, health, environment, property security, information technology, warehouse, supplies, controllers, and other teams. PEA estimations as well as IPCA variation were the basis for calculating the labor costs, as shown in Table 21-3.

Table 21-3 - Labor Cost Estimations.

Item	OPEX –FEASIBILITY	
	Cost (US$)	%
	US$ / t (metric) ROM	
Labor		
General Manager	$0.19	5%
Mine in the open	$0.60	17%
Plant maintenance	$1.76	50%
SSMAC	$0.36	10%
Administrative	$0.62	17%
Labor (Fixed Costs)	$3.53	16%

21.10 G&A

G&A costs shown in Table 21-4 include all costs relating to travel, transfers, consultants, individual safety equipment, exams, uniforms, environmental permits, as well as all employee benefits such as transportation, food, training. This item also includes the costs of cleaning and conservation of the building, vehicles, software licenses and other costs.

Table 21-4 - G&A Cost Estimations.

Item	OPEX – FEASIBILITY	
	Cost (US$)	%
	US$ / t (metric) ROM	
G&A		
Overheads	$0.07	4%
Health, Safety and Environment	$0.18	10%
Human resources	$0.36	21%
Administration	$0.35	21%
Outsourced Services	$0.73	43%
Other	$0.01	1%
G&A (Fixed Cost)	$1.69	7%

21.11 LABORATORY

Laboratory costs were provided by Aura and used the same calculations as the initial estimate corrected by the IPCA variation for the period.

21.12 ACCESS MAINTENANCE

The costs associated with access maintenance were included in mining costs.

21.13 EQUIPMENT RENTAL

The costs associated equipment rental were provided by Aura.

21.14 ENERGY

The energy consumption was calculated according to the specific demands of the Project. The unit values were obtained from Energiza, a local energy distributor. Table 21-5 shows the energy consumption costs for the Matupá Project.

Table 21-5 - Energy Consumption Cost Estimations.

Item	OPEX – FEASIBILITY	
	Cost (US$)	%
	US$ / t (metric) ROM	
Energy		
Metallurgy Substation	$1.70	78%
Administrative Building Substation	$0.20	9%
New Water Catchment Substation	$0.04	2%
Filtering Substation	$0.23	11%
Energy (Variable Costs)	$2.18	10%

21.15 REAGENTS AND CONSUMABLES

Quantities associated with reagents and consumables required for the operation were calculated by Promon based on process information, as validated by Aura. The costs related to this item were provided by Aura, based on the Apoena Mine operation. Table 21-6 shows the respective calculated values.

Table 21-6 - Reagents and Consumables Cost Estimations.

Item	OPEX – FEASIBILITY	
	Cost (US$)	**%**
	US$ / t (metric) ROM	
Reagents and Consumables		
Hydrated Lime	$0.18	2%
Sodium Cyanide	$1.95	25%
Sodium Hydroxide - 50% w/w	$0.24	3%
Hydrochloric Acid - 33%	$0.12	2%
Copper Sulfate Pentahydrate	$0.79	10%
Sodium Metabisulfite	$0.82	11%
Flocculant	$0.10	1%
Activated Carbon	$0.13	2%
Leachaid - Intensive Leaching	$0.01	0%
Hydrated Lime - Treating of Effluent	$0.00	0%
Smelting Fluxes	$-	
Borax	$0.00	0%
Silica	$0.00	0%
Sodium Nitrate	$0.00	0%
Sodium Carbonate	$0.00	0%
Crucibles	$0.01	0%
Consumables	$-	
Ball Mill	$1.69	22%
Mill Lining	$0.88	11%
Jaw Crusher	$-	0%
Fixed Jaw - Wear Plate	$0.01	0%
Moving Jaw - Wear Plate	$0.01	0%
Upper Side Coating Left Side	$0.00	0%
Lower Side Coating Left Side	$0.00	0%
Upper Right Side Coating	$0.00	0%
Lower Side Coating Right Side	$0.00	0%
Filter Cloths	$0.25	3%
Liquefied Petroleum Gas LGP - Plant	$0.43	6%
Liquefied Petroleum Gas LGP – Mess Hall	$0.02	0%
Liquefied Petroleum Gas LGP - Laboratory	$0.07	1%
Reagents and Consumables (Variable Costs)	**$7.74**	**34%**

21.16 MAINTENCE

Maintenance costs are directly linked to percentages of CapEx values of mechanical and electrical equipment, as follows: 3.5% for maintenance parts, 1.2% for consumables, and 1% for fuel and lubricants. Table 21-7 shows the maintenance cost estimations.

Table 21-7 - Maintenance Cost Estimations.

Item	OPEX – FEASIBILITY	
	Cost (US$)	**%**
	US$ / t (metric) ROM	
Maintenance		
Parts and Maintenance Materials	$0.72	65%
Consumables	$0.22	19%
Fuels and Lubricants	$0.18	16%
Maintenance	**$1.12**	**5%**

21.17 WATER AND SEWAGE TREATMENT PLANT

The costs related to the water and sewage treatment plant operation were reported by a specialized supplier. The final value includes the operation of the structure, such as chemical products and maintenance, as well as the required operation team.

21.18 PILE, MINE AND SUSTAINING

The costs related to the operation of piles were provided by Aura, together with the specialist advisor GHT. The results are listed in Table 21-8.

The mining operating costs were provided by Aura, along with specialist consultant EDEM. The studies indicated the costs for each year of mine operation, gold produced and recovered.

Table 21-8 - Piling and mining cost estimations.

Item	OPEX – FEASIBILITY	
	Cost (US$)	%
	US$ / t (metric) ROM	
Pile, Mine		
Reject Pile	$1.30	100%
Mine	$3.84	17%

22. ECONOMIC ANALYSIS

22.1 INTRODUCTION

This section presents the development of the economic modeling for the Matupá Project, which includes information provided by Promon, Aura, EDEM and EY, and the information in section 21, Capital and Operating costs.

This section describes the economic evaluation and financial metric methodologies used to establish the financial model for the Matupá Project Feasibility Study.

The economic model has been developed by Promon to support the evaluation of potential options and develop an optimal path forward for the Project. The main contributors to the total economic model are presented in Table 22-1.

Table 22-1 - Contributors and Their Roles in Developing the Total Economic Model.

CONTRIBUTOR	ROLE
Aura Minerals (Owner)	• Oversee the administration of the economic model including establishing governing parameters such as discount rate, tax regime, commodity rates, etc. • Develop the operating and owner expenses for the various project areas for which gaps exist. • Review the contributing inputs and their suitability for the Project.
Promon	• Develop the feasibility capital cost estimates for the processing plant, support infrastructure, and civil infrastructure. • Provide equipment data into the overall model based on the feasibility equipment list developed for the Project. • Provide cost data for equipment operations including labour, operating cost factors, and maintenance cost factors. • Provide other cost data and factors to support the execution of the total economic model. • Develop the feasibility capital cost estimate for the mining development and infrastructure. • Provide equipment data into the overall model for the mining operations. • Provide cost-of-life data for equipment including labour, operating cost factors, and maintenance cost factors for mining operations. • Execute the economic model and provide results to the project team to establish the optimal path forward.
EY	• Analysis of tax incentives
EDEM	• Responsible for the Mine Plan and mine operating costs

The economic model was developed in an Excel spreadsheet-based on a financial model composed of several worksheets.

All currency in this Section is provided in United States Dollars (US$), unless otherwise indicated. The average exchange rate used is US$1.00 = R$5.19.

Table 22-2 presents the financial model main premises and indicators, and Table 22-3 presents the summary results of the financial model that will be detailed in this section.

Table 22-2 - Main Premises and Indicators of the Financial Mode.

		Year -1	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total or Average
Exchange Rate	US$/US$	5.2	5.15	5.15	5.2	5.2	5.2	5.2	5.2	5.2	5.19
Plant Feed	Ton x 1.000			1,452	1,242	1,239	1,326	1,310	1,257	239	8,065
Production / Sales Volume	OZ			50,466	59,273	56,854	52,525	29,890	30,954	13,262	293,226
Gold Price	US$/OZ			1,749	1,650	1,650	1,650	1,650	1,650	1,650	1,667
Net Revenue	US$ x 1.000			88,256	97,799	93,808	86,665	49,317	51,074	21,882	488,800
Cash Cost	US$/OZ			600.8	468.2	485.1	549.3	908.9	736.5	683.7	591.8
Cash Cost	US$/Ton			20.9	22.3	22.3	21.8	20.7	18.1	37.9	21.5
AISC	US$/OZ			758.1	627.8	726.8	717.1	1038.8	851.3	872.8	762.0
Cash Cost	US$/OZ			600.8	468.2	485.1	549.3	908.9	736.5	683.7	591.8
Freight	US$/OZ			5.7	5.7	5.7	5.7	5.7	5.7	5.7	5.7
Refining	US$/OZ			0.6	0.6	0.6	0.6	0.6	0.6	0.6	0.6
SG&A	US$/OZ			55.1	46.4	48.4	44.6	59.9	48.9	114.2	52.7
CFEM	US$/OZ			26.2	24.7	24.7	24.7	24.7	24.7	24.7	25.0
Royalties	US$/OZ			36.9	34.8	34.8	34.8	34.8	34.8	34.8	35.2
Capex Sustaining	US$/OZ			32.8	47.2	127.3	57.4	4.2	0.0	9.1	51.0
Gross Margin	%			47.9%	55.3%	53.2%	46.7%	9.2%	20.9%	-46.5%	39.3%
SG&A	US$ x 1.000			2,780	2,753	2,753	2,340	1,790	1,514	1,514	15,445
SG&A / Net Revenue	%			3.1%	2.8%	2.9%	2.7%	3.6%	3.0%	6.9%	3.2%
Net Profit	US$ x 1.000			29,236	39,266	36,009	29,330	838	6,188	(12,484)	128,381
Net Margin	%			33.1%	40.1%	38.4%	33.8%	1.7%	12.1%	-57.1%	26.3%
EBITDA	US$ x 1.000			51,652	63,386	59,726	52,013	18,391	24,722	10,427	280,318
EBITDA Margin	%			58.5%	64.8%	63.7%	60.0%	37.3%	48.4%	47.7%	57.3%
CAPEX	US$ x 1.000	42,849	64,274	1,656	2,797	7,239	3,015	125	-	121	122,076
Capex	US$ x 1.000	42,849	64,274	-	-	-	-	-	-	-	107,123
Capex Sustaining	US$ x 1.000	-	-	1,656	2,797	7,239	3,015	125	-	121	14,953
Working Capital Need	US$ x 1.000	-	-	(5,850)	59	115	(16)	794	338	4,560	4,560
Financial Cycle	Days	-	-	-	-	-	-	-	-	-	-
Cash Flow (FCFF)	US$ x 1.000	(42,849)	(64,274)	39,632	54,234	46,124	43,707	18,911	23,948	24,296	143,729
Accumulated Cash Flow (FCFF)	US$ x 1.000	(42,849)	(107,123)	(67,490)	(13,256)	32,867	76,574	95,485	119,433	143,729	143,729

Table 22-3 - Summary Results of the Financial Model.

		MATUPÁ PROJECT
LOM	Years	0
Sales / Production Volume	OZ	293,226
Exchange Rate	R$/US$	5.19
Exchange Rate	CAD/US$	1.25935
Average Selling Price	US$/OZ	1,667
Royalties	%	2.15%
CFEM	%	1.50%
Average Cash Cost	US$/OZ	591.8
Average Cash Cost	US$/Ton	21.5
Average AISC	US$/OZ	762.0
Capex	US$ x 1.000	107,123
Sustaining Capex (Year 1 to 7)	US$ x 1.000	14,953
EBITDA - Annual Average	US$ x 1.000	40,045
EBITDA Margin - Annual Average	%	57.3%
Maximum Working Capital Need	US$ x 1.000	4,560
Financial Cycle - Average	days	-
Tax Regime	-	Real Profit
Income Tax and Social Contribution	%	34.0%
Discount Rate Calculated	%	15.53%
Discount Rate Used	%	5.00%
IRR - Internal Rate of Return	%	27.48%
NPV - Net Present Value	US$ x 1.000	96,128
Profitability Index	Index	1.95
Discounted Payback	Years	4.6
Simple Payback (Including Start-Up)	Years	4.3
Simple Payback (After Start-Up)	Years	2.0

22.2 ASSUMPTIONS

The following section summarizes the main assumptions used in the Project's financial analysis, including the mine production plan, product logistics, capital and operating expenditures, revenues, taxation, depreciation, royalties, and other general parameters.

22.2.1 PRODUCT

Only gold is considered for production minerals – specified in grade and ounces (oz).

22.2.2 PRODUCTION

The period of the construction and production plans is based on Project years. The construction period begins in year 2023. The first output of saleable gold is planned to begin in year 2025. Mining activity is planned to finish in year 2031, to summarize, the planned operation of the processing plant life is 7 years.

The metallurgical recovery for the contained gold is expected to be 94.57%, over a 7-year processing life results in 293,225.83 oz Au.

Table 22-4 summarizes the annual feed to the process plant with the tonnes of ROM, mineral grades, and gold content recover.

Table 22-4 - Summary of Production Plan.

		Year -1	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Plant Feed (ROM)	ton			1,451,678	1,242,049	1,239,040	1,326,232	1,309,718	1,256,552	239,239
Grade	g/ton			1.14	1.56	1.50	1.29	0.75	0.82	1.85
Au Content	oz			58,141	68,297	65,599	60,298	34,856	36,156	15,629
Availability Factor	%			91.4%	91.2%	91.2%	91.6%	91.2%	91.0%	90.5%
Recovery Ratio	%			94.96%	95.12%	95.01%	95.11%	94.01%	94.05%	93.73%
Au Recovered by period	oz			50,466.49	59,273.40	56,854.24	52,525.46	29,889.84	30,954.34	13,262.06

22.2.3 CAPITAL INVESTMENT

The initial capital cost amounts to US$ 107,123 million, which includes an allowance for contingencies.

For CapEx and, consequently, economic modelling, no type of applicable tax benefit was considered.

Table 22-5 summarizes the initial capital cost expenditure by commodity and disbursement schedule considered on the model.

Table 22-5 - Initial Capital Cost Summary and Disbursement Schedule.

ITEM	TOTAL	2021	2022
DESCRIPTION	**US$ X 1,000**		
Equipment	35,418	14,167	21,251
Materials	5,108	2,043	3,065
Construction & Erection	32,178	12,871	19,307
Aura Information	14,110	5,644	8,466
Indirects Costs	13,008	5,203	7,805
Contingency	7,301	2,920	4,381
Total - CapEx	**107,123**	**42,849**	**64,235**

The working capital requirement was calculated based on the assumptions of average terms for receivables, accounts payable, and inventories. The average terms considered are shown below:

- Accounts receivable: 10 days;
- Inventories: 60 days;
- Accounts Payable: 30 days; and

- Tax Liabilities: 15 days of Taxes.

According to the above premises, the capital requirement for the Project's working capital is around US$ 32,464 million. Table 22-6 presents the working capital summary:

Table 22-6 - Sustaining Capital Summary.

		Year -1	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Working Capital Variation	R$ x 1.000			(30,129)	306	596	(81)	4,129	1,759	23,713
	US$ x 1.000			(5,850)	59	115	(16)	794	338	4,560
Working Capital Need	R$ x 1.000									
	US$ x 1.000			(5,850)	(5,792)	(5,677)	(5,692)	(4,898)	(4,560)	-
Financial Cycle	Days			-	-	-	-	-	-	-
Accounts Receivable	R$ x 1.000	-	-	12,625	14,127	13,550	12,518	7,124	7,377	-
	US$ x 1.000	-	-	2,452	2,717	2,606	2,407	1,370	1,419	-
Inventories	R$ x 1.000	-	-	26,023	24,054	23,904	25,004	23,544	19,758	-
	US$ x 1.000	-	-	5,053	4,626	4,597	4,808	4,528	3,800	-
Other Inventories	R$ x 1.000	-	-	8,158	8,238	8,238	8,238	8,238	8,238	-
	US$ x 1.000	-	-	1,584	1,584	1,584	1,584	1,584	1,584	-
Accounts Payable	R$ x 1.000	-	-	15,709	14,912	14,769	15,016	13,401	11,419	-
	US$ x 1.000	-	-	3,050	2,868	2,840	2,888	2,577	2,196	-
Tax Liabilities	R$ x 1.000	-	-	968	1,390	1,404	1,143	32	241	-
	US$ x 1.000	-	-	188	267	270	220	6	46	-

22.2.4 OPERATING COSTS

The average operation cash cost for on-site mining, processing, general and administrative operational activities, and a 0,00% contingency over the processing costs is US$ 591.8/oz produced. The total operating costs, including non-recoverable taxes and refining, and transport and royalties, is US$ 762.0/oz produced.

For OpEx and, consequently, economic modelling, no type of applicable tax benefit was considered.

Recoverable taxes (PIS and COFINS) for non-exempt items, although paid at the time of purchase of inputs, services, and other resources, are assumed recovered in the short term and are not included.

Table 22-7 presents the operating cost summary.

Table 22-7 - Operating Costs Summary.

		Year -1	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Cash Cost	**R$ x 1.000**			**(156,138)**	**(144,322)**	**(143,424)**	**(150,022)**	**(141,266)**	**(118,549)**	**(47,148)**
	US$ x 1.000			**(30,318)**	**(27,754)**	**(27,582)**	**(28,850)**	**(27,167)**	**(22,798)**	**(9,067)**
Mining Costs	R$ / ton									
	R$ x 1.000			(32,272)	(32,166)	(31,532)	(35,633)	(30,159)	(15,577)	(4,585)
	US$ x 1.000			(6,266)	(6,186)	(6,064)	(6,853)	(5,800)	(2,996)	(882)
Processing Costs	R$ x 1.000			(123,865)	(112,156)	(111,893)	(114,389)	(111,107)	(102,972)	(42,563)
	US$ x 1.000			(24,052)	(21,568)	(21,518)	(21,998)	(21,367)	(19,802)	(8,185)
Labor	R$ x 1.000			(23,623)	(23,623)	(23,623)	(20,670)	(19,419)	(17,512)	(15,917)
	US$ x 1.000			(4,587)	(4,543)	(4,543)	(3,975)	(3,734)	(3,368)	(3,061)
Laboratory	R$ x 1.000			(8,524)	(8,524)	(8,524)	(7,246)	(7,246)	(4,688)	(4,688)
	US$ x 1.000			(1,655)	(1,639)	(1,639)	(1,393)	(1,393)	(902)	(902)
Access Maintenance	R$ x 1.000			-	-	-	-	-	-	-
	US$ x 1.000			-	-	-	-	-	-	-
Equipment Rental	R$ x 1.000			(158)	(158)	(158)	(158)	(158)	(158)	(158)
	US$ x 1.000			(31)	(30)	(30)	(30)	(30)	(30)	(30)
Reagents and Consumables	R$ x 1.000			(58,421)	(49,984)	(49,863)	(53,372)	(52,708)	(50,568)	(9,628)
	US$ x 1.000			(11,344)	(9,612)	(9,589)	(10,264)	(10,136)	(9,725)	(1,852)
Maintenance	R$ x 1.000			(7,591)	(7,252)	(7,235)	(7,744)	(7,647)	(7,337)	(1,397)
	US$ x 1.000			(1,474)	(1,395)	(1,391)	(1,489)	(1,471)	(1,411)	(269)
Energy	R$ x 1.000			(16,446)	(14,071)	(14,037)	(15,025)	(14,838)	(14,236)	(2,710)
	US$ x 1.000			(3,193)	(2,706)	(2,699)	(2,889)	(2,853)	(2,738)	(521)
ETE / ETA	R$ x 1.000			(101)	(87)	(86)	(92)	(91)	(88)	(17)
	US$ x 1.000			(20)	(17)	(17)	(18)	(18)	(17)	(3)
Dry Stack Tailing	R$ x 1.000			(9,001)	(8,456)	(8,366)	(10,082)	(9,001)	(8,385)	(8,048)
	US$ x 1.000			(1,748)	(1,626)	(1,609)	(1,939)	(1,731)	(1,613)	(1,548)
Others	R$ x 1.000			-	-	-	-	-	-	-
	US$ x 1.000			-	-	-	-	-	-	-
Contingencies	0%									
	R$ x 1.000			-	-	-	-	-	-	-
	US$ x 1.000			-	-	-	-	-	-	-
SG&A | Despesas Gerais e Administrativas	R$ x 1.000			(14,317)	(14,317)	(14,317)	(12,169)	(9,306)	(7,874)	(7,874)
	US$ x 1.000			(2,835)	(2,835)	(2,835)	(2,410)	(1,843)	(1,559)	(1,559)
Despesas Gerais	R$ x 1.000			(594)	(594)	(594)	(505)	(386)	(326)	(326)
	US$ x 1.000			(118)	(118)	(118)	(100)	(76)	(65)	(65)
Saúde, Segurança e Meio Ambiente	R$ x 1.000			(1,489)	(1,489)	(1,489)	(1,266)	(968)	(819)	(819)
	US$ x 1.000			(295)	(295)	(295)	(251)	(192)	(162)	(162)
Recursos Humanos	R$ x 1.000			(3,040)	(3,040)	(3,040)	(2,584)	(1,976)	(1,672)	(1,672)
	US$ x 1.000			(602)	(602)	(602)	(512)	(391)	(331)	(331)
Administração	R$ x 1.000			(2,946)	(2,946)	(2,946)	(2,504)	(1,915)	(1,620)	(1,620)
	US$ x 1.000			(583)	(583)	(583)	(496)	(379)	(321)	(321)
Serviços Terceirizados	R$ x 1.000			(6,173)	(6,173)	(6,173)	(5,247)	(4,012)	(3,395)	(3,395)
	US$ x 1.000			(1,222)	(1,222)	(1,222)	(1,039)	(794)	(672)	(672)
Outros	R$ x 1.000			(75)	(75)	(75)	(64)	(49)	(41)	(41)
	US$ x 1.000			(15)	(15)	(15)	(13)	(10)	(8)	(8)
Contingências	0%			-	-	-	-	-	-	-
				-	-	-	-	-	-	-

The mine closure is an additional cost that Aura considered will be US$ 6,961 million. The value presented by Aura considered a 9-year projection, but for the economic model, it was considered as present value applied to the last year of the projection.

22.2.5 REVENUE

Projections of net revenue are based on the quantity of gold to be delivered (293,226 kt oz) at an average long-term gold price of US$ 1,664/oz Au. Third-party services for treatment and refining are fixed at US$ 0.60/oz Au, while the transportation of the doré from site to refinery has an average unit cost of US$ 5.74/oz.

Annual average net revenue is US$ 69,829 million from year 1 (full run rate production period) to year 7.

Annual projections are shown in Table 22-8.

Table 22-8 - Annual Revenue.

		2023	2024	2025	2026	2027	2028	2029	2030	2031
		Year -1	**Year 0**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**
		0.5	1.5	2.5	3.5	4.5	5.5	6.5	7.5	8.5
Plant Feed (ROM)	ton			1,451,678	1,242,049	1,239,040	1,326,232	1,309,718	1,256,552	239,239
Grade	g/ton			1.14	1.56	1.50	1.29	0.75	0.82	1.85
Au Content	oz			58,141	68,297	65,599	60,298	34,856	36,156	15,629
Availability Factor	%			91.4%	91.2%	91.2%	91.6%	91.2%	91.0%	90.5%
Recovery Ratio	%			94.96%	95.12%	95.01%	95.11%	94.01%	94.05%	93.73%
Au Recovered by period	oz			50,466.49	59,273.40	56,854.24	52,525.46	29,889.84	30,954.34	13,262.06
Au Price (Anual Average)	US$/oz	1,796	1,762	1,749	1,650	1,650	1,650	1,650	1,650	1,650
Exchange Rate (Anual Average)	R$/US$	5.20	5.15	5.15	5.20	5.20	5.20	5.20	5.20	5.20
Gross Revenue	**R$ x 1.000**			**454,517**	**508,555**	**487,799**	**450,659**	**256,449**	**265,583**	**113,786**
	US$ x 1.000			**88,256**	**97,799**	**93,808**	**86,665**	**49,317**	**51,074**	**21,882**

22.2.6 ROYALTIES

Royalty Payable to the Federal Government – CFEM (Compensação Financeira pela Exploração de Recursos Minerais).

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 (including extraction permits). The royalty rate for gold is 1.5% of gross revenue of the mineral product, less revenue taxes on the mineral product, transportation, and insurance costs.

Royalty Payable to the Landowners of the Mined Areas

The royalty rates over the gross revenue of the gold for each landowner/tenant of the mined areas are 2.15% of gross revenue of the mineral product, less transportation/freight, refining, and CFEM.

22.2.7 TAXATION

Taxes that are due for the Matupá Project were estimated considering existing tax laws, with application to revenues associated with the Project's production.

Tax Regime: The tax regime applied to the economic model was Real Profit ("Lucro Real" in Portuguese).

CSLL – Social Contribution: The social contribution tax is 9% calculated based on pre-tax profit.

IRPJ – Income Tax: A tax rate of 25% would be applied to pre-tax income, however, the tax benefit of a 75% reduction was considered as an exploration profit benefit.

PIS, COFINS and ICMS: These taxes were not applied in this analysis since all production is directed for exportation.

22.2.8 ALL-IN SUSTAINING COSTS

CapEx sustaining was considered in the OpEx projection.

Table 22-9 details the expenditures in the operations phase of the Project in accordance with the definition of All-In-Sustaining Costs ("AISC") as proposed by the World Gold Council's Guidance Note of June 27, 2013. Unit costs per ounce reflect the varying costs of producing gold over the LOM. Figure 22-1 presents the comparison between Price x Cash Costs x AISC.

Table 22-9 - All-In Sustaining Costs.

		Year -1	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total or Average
AISC	US$/OZ			758.1	627.8	726.8	717.1	1038.8	851.3	872.8	762.0
Cash Cost	US$/OZ			600.8	468.2	485.1	549.3	908.9	736.5	683.7	591.8
Freight	US$/OZ			5.7	5.7	5.7	5.7	5.7	5.7	5.7	5.7
Refining	US$/OZ			0.6	0.6	0.6	0.6	0.6	0.6	0.6	0.6
SG&A	US$/OZ			55.1	46.4	48.4	44.6	59.9	48.9	114.2	52.7
CFEM	US$/OZ			26.2	24.7	24.7	24.7	24.7	24.7	24.7	25.0
Royalties	US$/OZ			36.9	34.8	34.8	34.8	34.8	34.8	34.8	35.2
Capex Sustaining	US$/OZ			32.8	47.2	127.3	57.4	4.2	0.0	9.1	51.0

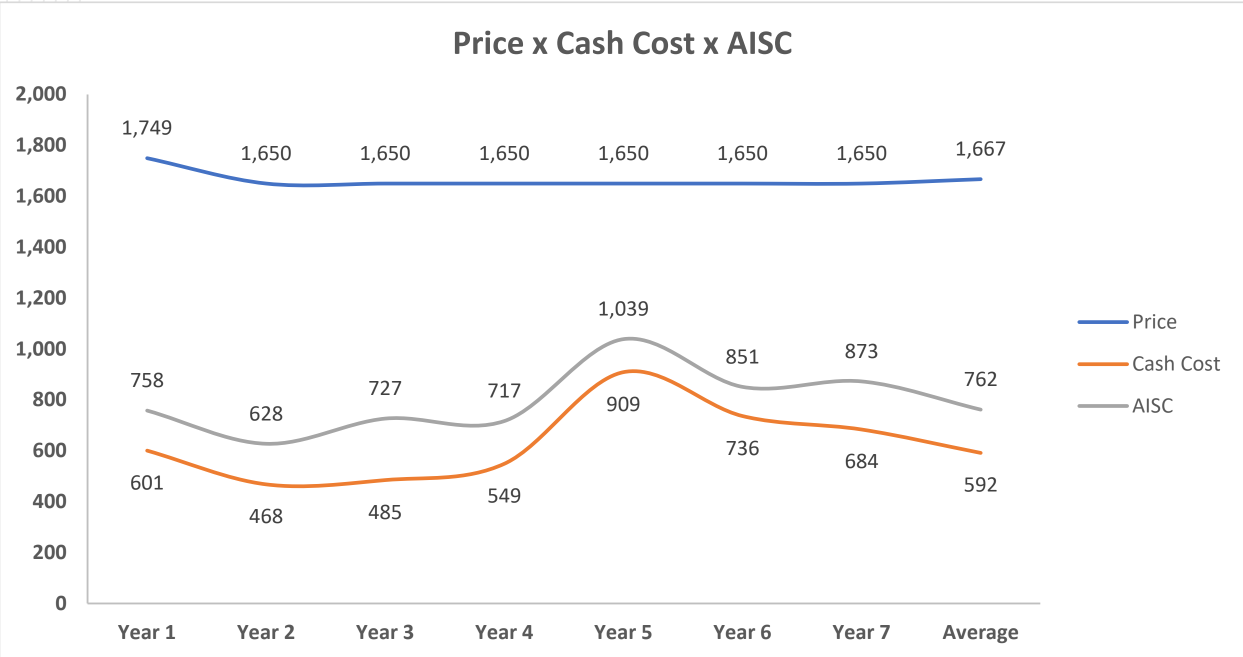


Figure 22-1 - Price x Cash Cost x AISC Graph.

22.3 FINANCIAL ANALYSIS

22.3.1 ANALYZE CONSIDERING WACC RECOMMENDED BY AURA

The financial model adopts the concept of project free cash flow, in which all the Project's cash generation capacity is evaluated by countering this flow with a weighted discount rate ("WACC") which reflects the average cost of sources of funds (cost of equity and third parties). The amounts in the cash flow were expressed in thousand Brazilian reais (US$ x 1,000) and on a real basis (without inflation).

Based on the assumptions adopted, the post-tax net present value (“NPV”) of Aura Minerals Gold Almas Project base case amounts to US$ 96,128 million, at a Discount Rate of 5.0%.

The internal rate of return (“IRR”) is 27.5% and the annual average EBITDA (from year 1 to year 7, full run rate production period) is US$ 280,318 million. Payback after the start-up of operations is 2 years.

The leveraged IRR calculation was performed considering a debt of 50% leverage and the calculated value was 49.9%.

The results are summarized in Table 22-10 and the operating income statement, and the project cash flow are respectively presented in Table 22-11 and Table 22-12.

Table 22-10 - Financial Results Summary (Post tax).

VALUATION - BASIC PROJECT		
NPV	96,128	US$ x 1.000
IRR	27.5%	
Leveraged IRR	49.9%	
Profitability Index	1.95	
Discounted Payback	4.56	years
Simple Payback (Including Start-Up)	4.29	years
Simple Payback (After Start-Up)	2.04	years
Discont Rate	5.0%	
Invest in the Project?	Yes	

Table 22-11 - Operating Income Statement.

		Year -1	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Gross Revenue	**R$ x 1.000**			**454,517**	**508,555**	**487,799**	**450,659**	**256,449**	**265,583**	**113,786**
	US$ x 1.000			**88,256**	**97,799**	**93,808**	**86,665**	**49,317**	**51,074**	**21,882**
Cash Cost	**R$ x 1.000**			**(156,138)**	**(144,322)**	**(143,424)**	**(150,022)**	**(141,266)**	**(118,549)**	**(47,148)**
	US$ x 1.000			**(30,318)**	**(27,754)**	**(27,582)**	**(28,850)**	**(27,167)**	**(22,798)**	**(9,067)**
Mining Costs	R$ / ton									
	R$ x 1.000			(32,272)	(32,166)	(31,532)	(35,633)	(30,159)	(15,577)	(4,585)
	US$ x 1.000			(6,266)	(6,186)	(6,064)	(6,853)	(5,800)	(2,996)	(882)
Processing Costs	R$ x 1.000			(123,865)	(112,156)	(111,893)	(114,389)	(111,107)	(102,972)	(42,563)
	US$ x 1.000			(24,052)	(21,568)	(21,518)	(21,998)	(21,367)	(19,802)	(8,185)
Contingencies	0%									
	R$ x 1.000			-	-	-	-	-	-	-
	US$ x 1.000			-	-	-	-	-	-	-
Freight / Refining	**R$ x 1.000**			**(1,648)**	**(1,954)**	**(1,874)**	**(1,732)**	**(985)**	**(1,021)**	**(437)**
	US$ x 1.000			**(320)**	**(376)**	**(360)**	**(333)**	**(190)**	**(196)**	**(84)**
Freight to Refinery	US$ / oz			5.74	5.74	5.74	5.74	5.74	5.74	5.74
Refining	US$ / oz			0.6	0.6	0.6	0.6	0.6	0.6	0.6
Gross Profit (considering Depreciation)	**R$ x 1.000**			**217,920**	**281,472**	**259,616**	**210,643**	**23,695**	**55,418**	**(52,937)**
	US$ x 1.000			**42,315**	**54,129**	**49,926**	**40,508**	**4,557**	**10,657**	**(10,180)**
SG&A | Despesas Gerais e Administrativas	R$ x 1.000			(14,317)	(14,317)	(14,317)	(12,169)	(9,306)	(7,874)	(7,874)
	US$ x 1.000			(2,780)	(2,753)	(2,753)	(2,340)	(1,790)	(1,514)	(1,514)
CFEM	R$ x 1.000			(6,818)	(7,628)	(7,317)	(6,760)	(3,847)	(3,984)	(1,707)
	US$ x 1.000			(1,324)	(1,467)	(1,407)	(1,300)	(740)	(766)	(328)
Royalties	R$ x 1.000			(9,590)	(10,728)	(10,290)	(9,507)	(5,410)	(5,602)	(2,400)
	US$ x 1.000			(1,862)	(2,063)	(1,979)	(1,828)	(1,040)	(1,077)	(462)
Basis Calculation	US$ x 1.000			86,612	95,956	92,040	85,032	48,388	50,111	21,470
Gross Revenue	US$ x 1.000			88,256	97,799	93,808	86,665	49,317	51,074	21,882
Freight	US$ x 1.000			(290)	(340)	(326)	(301)	(172)	(178)	(76)
Refining	US$ x 1.000			(30)	(36)	(34)	(32)	(18)	(19)	(8)
Taxes (CFEM)	US$ x 1.000			(1,324)	(1,467)	(1,407)	(1,300)	(740)	(766)	(328)
Interest expenses	R$ x 1.000			(13,390)	(11,267)	(6,760)	(2,253)	-	-	-
	US$ x 1.000			(2,600)	(2,167)	(1,300)	(433)	-	-	-
EBIT	**R$ x 1.000**			**173,806**	**237,532**	**220,932**	**179,954**	**5,133**	**37,958**	**(64,918)**
	US$ x 1.000			**33,749**	**45,679**	**42,487**	**34,607**	**987**	**7,300**	**(12,484)**
Depreciação & Amortização	R$ x 1.000			78,812	80,807	82,885	88,263	90,502	90,595	119,138
	US$ x 1.000			15,303	15,540	15,939	16,974	17,404	17,422	22,911
EBITDA	**R$ x 1.000**			**252,617**	**318,339**	**303,817**	**268,217**	**95,636**	**128,553**	**54,220**
	US$ x 1.000			**51,652**	**63,386**	**59,726**	**52,013**	**18,391**	**24,722**	**10,427**

Table 22-12 - Project Cash Flow.

		Year -1	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
EBITDA	**R$ x 1.000**			**252,617**	**318,339**	**303,817**	**268,217**	**95,636**	**128,553**	**54,220**
	US$ x 1.000			**51,652**	**63,386**	**59,726**	**52,013**	**18,391**	**24,722**	**10,427**
CAPEX	R$ x 1.000	(220,673)	(331,009)	-	-	-	-	-	-	-
	US$ x 1.000	(42,849)	(64,274)	-	-	-	-	-	-	-
CAPEX Sustaining	R$ x 1.000			(8,530)	(14,545)	(37,644)	(15,678)	(650)	-	(627)
	US$ x 1.000			(1,656)	(2,797)	(7,239)	(3,015)	(125)	-	(121)
Working Capital Variation	R$ x 1.000			(30,129)	306	596	(81)	4,129	1,759	23,713
	US$ x 1.000			(5,850)	59	115	(16)	794	338	4,560
Mine Closure Cost (Present Value form Aura info - 9 years projection)	R$ x 1.000			-	-	-	-	-	-	(36,196)
	US$ x 1.000			-	-	-	-	-	-	(6,961)
Salvage Value	R$ x 1.000			-	-	-	-	-	-	85,232
	US$ x 1.000			-	-	-	-	-	-	16,391
Income Tax / Social Contribution	R$ x 1.000			(23,242)	(33,351)	(33,686)	(27,437)	(777)	(5,783)	-
	US$ x 1.000			(4,513)	(6,414)	(6,478)	(5,276)	(149)	(1,112)	-
Capex Tax Recovery	R$ x 1.000			-	-	-	-	-	-	-
	US$ x 1.000			-	-	-	-	-	-	-
FCFF Nominal	**R$ x 1.000**	**(220,673)**	**(331,009)**	**204,106**	**282,017**	**239,843**	**227,274**	**98,337**	**124,530**	**126,341**
	US$ x 1.000	**(42,849)**	**(64,274)**	**39,632**	**54,234**	**46,124**	**43,707**	**18,911**	**23,948**	**24,296**
WACC	5.00%	0.98	0.93	0.89	0.84	0.80	0.76	0.73	0.69	0.66
FCFF Discounted	**R$ x 1.000**	**(215,354)**	**(307,649)**	**180,668**	**237,745**	**192,564**	**173,784**	**71,612**	**86,368**	**83,452**
	US$ x 1.000	**(41,816)**	**(59,738)**	**35,081**	**45,720**	**37,032**	**33,420**	**13,772**	**16,609**	**16,048**
FCFF Discounted Acumulated	**R$ x 1.000**	**(215,354)**	**(523,004)**	**(342,335)**	**(104,590)**	**87,975**	**261,758**	**333,371**	**419,739**	**503,190**
	US$ x 1.000	**(41,816)**	**(101,554)**	**(66,473)**	**(20,753)**	**16,279**	**49,699**	**63,471**	**80,080**	**96,128**

22.4 SENSITIVITY ANALYSIS

22.4.1 SPIDER GRAPH ANALYSIS

The sensitivity analysis shows the impact of the variation of the gold price, exchange rates, operating costs (OpEx), Recovery Ratio, WACC and capital costs (CapEx) upon the Project NPV and IRR. The analysis encompasses the following range of variation in the key inputs:

- Gold price: ±20%.
- Exchange Rate: ±20%.
- OpEx (Cost): ±20%.
- WACC (Discount Rate): ±20%.
- CapEx: ±20%.

In assessing the sensitivity of the Project returns, each of these parameters is varied independently of the others. Scenarios combining beneficial or adverse variations simultaneously in two or more variables will have a more marked effect on the economics of the Project than will the individual variations considered. The sensitivity analysis has been conducted assuming no change to the mine plan or schedule.

Figure 22-2 illustrates the results of the sensitivity analysis for Project NPV (after tax) and these effects for each of the critical variables and Figure 22-3 presents the same scenario for the IRR. NPV results are reported at a discount rate of 5.0%.

NPV (US$ MM)	80%	90%	95%	100%	105%	110%	120%
Gold Price	31.21	63.88	80.08	96.13	112.11	128.04	159.92
Exchange Rate	63.70	81.80	89.42	96.13	102.22	107.73	117.36
OpEx (Costs)	121.60	108.88	102.53	96.13	89.76	83.25	70.21
Recovery Ratio	31.46	64.00	80.14	96.13	112.05	127.92	159.67
WACC	104.43	100.23	98.18	96.13	94.19	92.24	88.45
CapEx	111.77	103.97	100.07	96.13	92.27	88.31	80.34

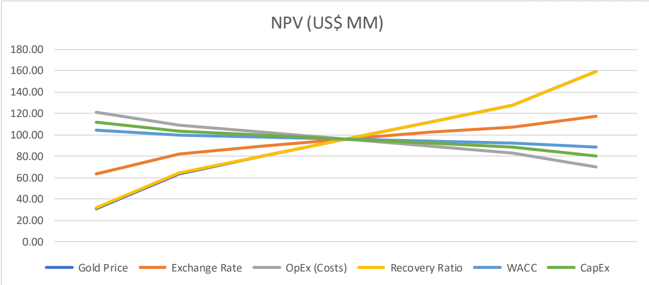


Figure 22-2 - Sensitivity Analysis Graph – NPV.

IRR (%)	80%	90%	95%	100%	105%	110%	120%
Gold Price	13.06%	20.63%	24.14%	27.48%	30.72%	33.84%	39.82%
Exchange Rate	20.72%	24.57%	26.13%	27.48%	28.70%	29.79%	31.65%
OpEx (Costs)	32.46%	30.01%	28.76%	27.48%	26.20%	24.87%	22.13%
Recovery Ratio	13.13%	20.65%	24.15%	27.48%	30.70%	33.81%	39.77%
WACC	27.49%	27.49%	27.49%	27.48%	27.49%	27.49%	27.49%
CapEx	36.25%	31.47%	29.40%	27.48%	25.74%	24.11%	21.20%

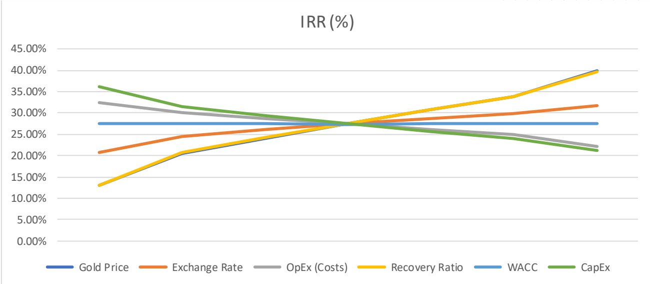


Figure 22-3 - Sensitivity Analysis Graph – IRR.

22.4.2 TWO PARAMETERS ANALYSIS

Additionally, secondary sensitivity analysis were made varying two parameters simultaneously to assess the impact on the IRR, NPV, Discounted Payback, and Simple Payback (Including Start-Up):

- Gold Price x Exchange Rate (Table 22-13).
- Gold Price x OpEx (Table 22-14).
- Gold Price x Discount Rate (Table 22-15).
- Gold Price x CAPEX (Table 22-16).
- OpEx x CapEx (Table 22-17).

Table 22-13 - Sensitivity Analysis Graph – Price x Exchange Rate – IRR / NPV / Discounted Payback / Simple Payback - Including Start-Up.

WACC = 5%

IRR (%)	GOLD PRICE (U$ / Oz) - Percent Variation						
EXCHANGE RATE (R$ / US$)	80%	90%	95%	100%	105%	110%	120%
4.16	4.43%	13.08%	17.01%	20.72%	24.25%	27.63%	34.01%
4.68	9.40%	17.40%	21.07%	24.57%	27.93%	31.14%	37.28%
4.94	11.36%	19.12%	22.70%	26.13%	29.41%	32.57%	38.62%
5.20	13.06%	20.63%	24.14%	27.48%	30.72%	33.84%	39.82%
5.46	14.57%	21.96%	25.41%	28.70%	31.88%	34.97%	40.89%
5.72	15.90%	23.15%	26.54%	29.79%	32.93%	35.99%	41.85%
6.24	18.15%	25.18%	28.47%	31.65%	34.74%	37.74%	43.52%

WACC = 5%

NPV (US$ MM)	GOLD PRICE (U$ / Oz) - Percent Variation						
EXCHANGE RATE (R$ / US$)	80%	90%	95%	100%	105%	110%	120%
4.16	-2.02	30.90	47.36	63.70	79.90	96.10	128.14
4.68	16.44	49.36	65.60	81.80	97.98	113.92	145.79
4.94	24.21	57.02	73.22	89.42	105.42	121.35	153.23
5.20	31.21	63.88	80.08	96.13	112.11	128.04	159.92
5.46	37.54	70.08	86.29	102.22	118.16	134.10	165.97
5.72	43.29	75.73	91.79	107.73	123.66	139.60	171.48
6.24	53.20	85.48	101.42	117.36	133.30	149.23	181.11

WACC = 5%

Discounted Payback (Years)	GOLD PRICE (U$ / Oz) - Percent Variation						
EXCHANGE RATE (R$ / US$)	80%	90%	95%	100%	105%	110%	120%
4.16	WP	5.88	5.41	5.04	4.76	4.52	4.16
4.68	7.54	5.41	5.03	4.75	4.52	4.32	4.00
4.94	6.70	5.24	4.91	4.65	4.43	4.25	3.95
5.20	5.97	5.10	4.80	4.56	4.36	4.18	3.91
5.46	5.79	4.98	4.71	4.48	4.29	4.12	3.87
5.72	5.64	4.89	4.64	4.42	4.23	4.07	3.84
6.24	5.40	4.75	4.51	4.31	4.14	3.99	3.79

WACC = 5%

Simple Payback - Including Start-Up (Years)	GOLD PRICE (U$ / Oz) - Percent Variation						
EXCHANGE RATE (R$ / US$)	80%	90%	95%	100%	105%	110%	120%
4.16	7.84	5.34	4.97	4.69	4.45	4.26	3.96
4.68	5.81	4.97	4.68	4.45	4.25	4.09	3.84
4.94	5.60	4.84	4.58	4.36	4.18	4.02	3.80
5.20	5.42	4.73	4.49	4.29	4.11	3.97	3.76
5.46	5.28	4.64	4.42	4.22	4.06	3.93	3.73
5.72	5.16	4.57	4.35	4.17	4.01	3.89	3.70
6.24	4.97	4.44	4.24	4.08	3.94	3.83	3.65

Table 22-14 - Sensitivity Analysis Graph – Price x OpEx – IRR / NPV / Discounted Payback / Simple Payback - Including Start-Up.

WACC = 5%

IRR (%)	GOLD PRICE (U$ / Oz) - Percent Variation						
OpEx (%)	80%	90%	95%	100%	105%	110%	120%
80%	19.11%	26.05%	29.31%	32.46%	35.53%	38.51%	44.24%
90%	16.18%	23.40%	26.77%	30.01%	33.15%	36.20%	42.05%
95%	14.64%	22.03%	25.48%	28.76%	31.94%	35.03%	40.94%
100%	13.06%	20.63%	24.14%	27.48%	30.72%	33.84%	39.82%
105%	11.44%	19.20%	22.77%	26.20%	29.47%	32.63%	38.68%
110%	9.78%	17.74%	21.38%	24.87%	28.21%	31.42%	37.53%
120%	6.28%	14.67%	18.51%	22.13%	25.59%	28.93%	35.20%

WACC = 5%

NPV (US$ MM)	GOLD PRICE (U$ / Oz) - Percent Variation						
OpEx (%)	80%	90%	95%	100%	105%	110%	120%
80%	57.54	89.72	105.66	121.60	137.53	153.47	185.35
90%	44.50	76.91	92.95	108.88	124.82	140.76	172.63
95%	37.85	70.40	86.59	102.53	118.46	134.40	166.28
100%	31.21	63.88	80.08	96.13	112.11	128.04	159.92
105%	24.56	57.36	73.56	89.76	105.75	121.69	153.56
110%	17.92	50.84	67.05	83.25	99.39	115.33	147.21
120%	4.63	37.55	54.01	70.21	86.41	102.61	134.49

WACC = 5%

Discounted Payback (Years)	GOLD PRICE (U$ / Oz) - Percent Variation						
OpEx (%)	80%	90%	95%	100%	105%	110%	120%
80%	5.30	4.68	4.46	4.27	4.10	3.97	3.76
90%	5.61	4.88	4.62	4.41	4.22	4.06	3.84
95%	5.78	4.98	4.71	4.48	4.29	4.12	3.87
100%	5.97	5.10	4.80	4.56	4.36	4.18	3.91
105%	6.65	5.23	4.90	4.64	4.43	4.24	3.95
110%	7.39	5.38	5.01	4.73	4.50	4.31	3.99
120%	8.61	5.70	5.27	4.93	4.67	4.45	4.10

WACC = 5%

Simple Payback - Including Start-Up (Years)	GOLD PRICE (U$ / Oz) - Percent Variation						
OpEx (%)	80%	90%	95%	100%	105%	110%	120%
80%	4.89	4.39	4.20	4.04	3.91	3.81	3.63
90%	5.13	4.55	4.34	4.16	4.00	3.89	3.69
95%	5.27	4.64	4.41	4.22	4.06	3.93	3.72
100%	5.42	4.73	4.49	4.29	4.11	3.97	3.76
105%	5.59	4.84	4.57	4.36	4.18	4.02	3.80
110%	5.77	4.94	4.66	4.43	4.24	4.07	3.83
120%	7.08	5.20	4.86	4.60	4.38	4.19	3.91

Table 22-15 - Sensitivity Analysis Graph – Price x Discount Rate – IRR / NPV / Discounted Payback / Simple Payback - Including Start-Up.

IRR (%)	GOLD PRICE (U$ / Oz) - Percent Variation						
DISCOUNT RATE (%)	80%	90%	95%	100%	105%	110%	120%
80%	13.06%	20.63%	24.14%	27.49%	30.72%	33.84%	39.82%
90%	13.06%	20.63%	24.14%	27.49%	30.72%	33.84%	39.82%
95%	13.06%	20.63%	24.14%	27.49%	30.72%	33.84%	39.82%
100%	13.06%	20.63%	24.14%	27.48%	30.72%	33.84%	39.82%
105%	13.06%	20.63%	24.14%	27.49%	30.72%	33.84%	39.82%
110%	13.06%	20.63%	24.14%	27.49%	30.72%	33.84%	39.82%
120%	13.06%	20.63%	24.14%	27.49%	30.72%	33.84%	39.82%

NPV (US$ MM)	GOLD PRICE (U$ / Oz) - Percent Variation						
DISCOUNT RATE (%)	80%	90%	95%	100%	105%	110%	120%
80%	36.44	70.64	87.59	104.43	121.10	137.78	171.12
90%	33.78	67.20	83.77	100.23	116.53	132.83	165.43
95%	32.48	65.52	81.91	98.18	114.30	130.42	162.65
100%	31.21	63.88	80.08	96.13	112.11	128.04	159.92
105%	29.96	62.26	78.28	94.19	109.95	125.71	157.23
110%	28.73	60.67	76.51	92.24	107.83	123.42	154.59
120%	26.33	57.57	73.06	88.45	103.69	118.94	149.42

Discounted Payback (Years)	GOLD PRICE (U$ / Oz) - Percent Variation						
DISCOUNT RATE (%)	80%	90%	95%	100%	105%	110%	120%
80%	5.85	5.02	4.74	4.50	4.30	4.13	3.88
90%	5.91	5.06	4.77	4.53	4.33	4.16	3.90
95%	5.94	5.08	4.78	4.55	4.34	4.17	3.90
100%	5.97	5.10	4.80	4.56	4.36	4.18	3.91
105%	6.01	5.12	4.82	4.57	4.37	4.19	3.92
110%	6.11	5.14	4.84	4.59	4.38	4.20	3.93
120%	6.32	5.19	4.87	4.62	4.41	4.23	3.94

Simple Payback - Including Start-Up (Years)	GOLD PRICE (U$ / Oz) - Percent Variation						
DISCOUNT RATE (%)	80%	90%	95%	100%	105%	110%	120%
80%	5.42	4.73	4.49	4.29	4.11	3.97	3.76
90%	5.42	4.73	4.49	4.29	4.11	3.97	3.76
95%	5.42	4.73	4.49	4.29	4.11	3.97	3.76
100%	5.42	4.73	4.49	4.29	4.11	3.97	3.76
105%	5.42	4.73	4.49	4.29	4.11	3.97	3.76
110%	5.42	4.73	4.49	4.29	4.11	3.97	3.76
120%	5.42	4.73	4.49	4.29	4.11	3.97	3.76

Table 22-16 - Sensitivity Analysis Graph – Price x CapEx – IRR / NPV / Discounted Payback / Simple Payback - Including Start-Up.

WACC = 5%

IRR (%)	GOLD PRICE (U$ / Oz) - Percent Variation						
CAPEX (%)	80%	90%	95%	100%	105%	110%	120%
80%	19.75%	28.41%	32.42%	36.25%	39.96%	43.54%	50.40%
90%	16.10%	24.16%	27.91%	31.47%	34.92%	38.25%	44.63%
95%	14.52%	22.32%	25.94%	29.40%	32.73%	35.95%	42.12%
100%	13.06%	20.63%	24.14%	27.48%	30.72%	33.84%	39.82%
105%	11.73%	19.07%	22.47%	25.74%	28.87%	31.89%	37.69%
110%	10.50%	17.64%	20.94%	24.11%	27.15%	30.09%	35.73%
120%	8.31%	15.06%	18.19%	21.20%	24.09%	26.87%	32.20%

WACC = 5%

NPV (US$ MM)	GOLD PRICE (U$ / Oz) - Percent Variation						
CAPEX (%)	80%	90%	95%	100%	105%	110%	120%
80%	47.41	79.81	95.83	111.77	127.70	143.64	175.52
90%	39.34	71.85	88.03	103.97	119.91	135.84	167.72
95%	35.27	67.86	84.06	100.07	116.01	131.94	163.82
100%	31.21	63.88	80.08	96.13	112.11	128.04	159.92
105%	27.14	59.89	76.09	92.27	108.21	124.15	156.02
110%	23.08	55.91	72.11	88.31	104.31	120.25	152.12
120%	14.95	47.87	64.14	80.34	96.51	112.45	144.32

WACC = 5%

Discounted Payback (Years)	GOLD PRICE (U$ / Oz) - Percent Variation						
CAPEX (%)	80%	90%	95%	100%	105%	110%	120%
80%	5.14	4.48	4.24	4.04	3.90	3.78	3.58
90%	5.56	4.79	4.52	4.30	4.12	3.96	3.75
95%	5.77	4.94	4.66	4.43	4.24	4.07	3.83
100%	5.97	5.10	4.80	4.56	4.36	4.18	3.91
105%	6.57	5.27	4.94	4.69	4.47	4.29	3.99
110%	7.16	5.43	5.09	4.81	4.59	4.40	4.09
120%	8.02	5.76	5.39	5.07	4.83	4.62	4.28

WACC = 5%

Simple Payback - Including Start-Up (Years)	GOLD PRICE (U$ / Oz) - Percent Variation						
CAPEX (%)	80%	90%	95%	100%	105%	110%	120%
80%	4.76	4.21	4.01	3.87	3.75	3.64	3.46
90%	5.09	4.48	4.25	4.07	3.92	3.81	3.61
95%	5.26	4.61	4.37	4.18	4.01	3.89	3.69
100%	5.42	4.73	4.49	4.29	4.11	3.97	3.76
105%	5.59	4.86	4.61	4.40	4.21	4.06	3.83
110%	5.75	4.99	4.72	4.50	4.31	4.15	3.90
120%	6.22	5.25	4.96	4.72	4.51	4.34	4.05

Table 22-17 - Sensitivity Analysis Graph –CapEx x OpEx – IRR / NPV / Discounted Payback / Simple Payback - Including Start-Up.

WACC = 5%

IRR (%)		OPEX (%)						
		80%	90%	95%	100%	105%	110%	120%
CAPEX (%)	80%	41.94%	39.14%	37.71%	36.25%	34.78%	33.28%	30.18%
	90%	36.77%	34.16%	32.83%	31.47%	30.10%	28.70%	25.78%
	95%	34.53%	32.00%	30.71%	29.40%	28.07%	26.70%	23.88%
	100%	32.46%	30.01%	28.76%	27.48%	26.20%	24.87%	22.13%
	105%	30.57%	28.19%	26.97%	25.74%	24.47%	23.18%	20.52%
	110%	28.81%	26.50%	25.32%	24.11%	22.88%	21.62%	19.03%
	120%	25.67%	23.48%	22.35%	21.20%	20.02%	18.83%	16.37%

WACC = 5%

NPV (US$ MM)		OPEX (%)						
		80%	90%	95%	100%	105%	110%	120%
CAPEX (%)	80%	137.19	124.48	118.12	111.77	105.41	99.05	86.15
	90%	129.39	116.68	110.33	103.97	97.61	91.21	78.18
	95%	125.50	112.78	106.43	100.07	93.71	87.23	74.20
	100%	121.60	108.88	102.53	96.13	89.76	83.25	70.21
	105%	117.70	104.98	98.63	92.27	85.78	79.26	66.23
	110%	113.80	101.08	94.73	88.31	81.79	75.28	62.24
	120%	106.00	93.29	86.86	80.34	73.83	67.31	54.21

WACC = 5%

Discounted Payback (Years)		OPEX (%)						
		80%	90%	95%	100%	105%	110%	120%
CAPEX (%)	80%	3.83	3.93	3.98	4.04	4.11	4.18	4.35
	90%	4.03	4.16	4.23	4.30	4.38	4.46	4.64
	95%	4.15	4.28	4.36	4.43	4.51	4.60	4.78
	100%	4.27	4.41	4.48	4.56	4.64	4.73	4.93
	105%	4.38	4.53	4.61	4.69	4.77	4.87	5.08
	110%	4.50	4.65	4.73	4.81	4.91	5.00	5.24
	120%	4.73	4.89	4.98	5.07	5.18	5.30	5.55

WACC = 5%

Simple Payback - Including Start-Up (Years)		OPEX (%)						
		80%	90%	95%	100%	105%	110%	120%
CAPEX (%)	80%	3.69	3.78	3.82	3.87	3.92	3.97	4.11
	90%	3.86	3.96	4.01	4.07	4.13	4.20	4.35
	95%	3.95	4.05	4.11	4.18	4.25	4.32	4.48
	100%	4.04	4.16	4.22	4.29	4.36	4.43	4.60
	105%	4.14	4.26	4.33	4.40	4.47	4.55	4.72
	110%	4.23	4.36	4.43	4.50	4.58	4.66	4.84
	120%	4.43	4.56	4.64	4.72	4.80	4.88	5.08

22.4.3 CONCLUSION

The financial model for the Matupá Project was prepared using capital costs, operating expenditures, and production schedule with inputs provided by Aura, Promon, EY and EDEM.

The financial model adopts the concept of project free cash flow, in which all the project's cash generation capacity is evaluated by countering this flow with a WACC, which reflects the average cost of sources of funds (cost of equity and third parties). The financial model considers the Real Profit tax regime.

After the evaluation, the NPV Post-tax, at a WACC rate of 5% per year, resulted in US$ 96,128 million, with an IRR of 27,5% and a 4.56-years Single Payback Time. For this scenario, the gold price adopted was US$1,664/oz (average per year) and the average exchange rate used was 5.19 (R$/US$).

A series of analysis were made varying Gold Price, Exchange Rate, CapEx, Discount Rate, and OpEx to assess the impact of these variables on the NPV, IRR, Discounted Payback, and Simple Payback (Including Start-Up).

Based on the results of the Sensitivity Analysis the project profitability is most affected by the gold price and exchange rate, followed by CapEx and Discount Rate.

Considering the WACC of 5% and the feasibility analysis of the Project through the applied methodology, the rate of return and the NPV are good enough to consider the viability of the Project. However, further analysis is possible to improve the perception of the viability.

23. ADJACENT PROPERTIES

Several other exploration companies maintain land positions near Aura's Matupá Project area (Figure 23-1). Recent activity on these adjacent properties has been high, especially since 2017. As of November 2021, adjacent properties are held by Codelco do Brazil, Nexa Resources, Fides Gold Mineradora S.A., P.A. Gold Mineração, Mineração Santa Elina, Anglo American, Yamana Gold, Iamgold-Brazil, many other small companies and Coogavepe, the greater cooperative of garimpo in Brazil.

23.1 ADJACENT EXPLORATION PROPERTIES

Yamana Gold holds 2 blocks of exploration licenses around the Matupá Project and is currently active in the district. Anglo American, Nexa Resources, and Codeldo hold several blocks of exploration licenses in the district and are currently active in the district. Iamgold-Brasil holds 5 blocks of exploration licenses in the district and is currently active in the district. Fides Mining holds over 55,000 hectares of mineral rights in the district and is currently active.

23.2 ADJACENT OPERATING PROPERTIES

Fides Mining operates the União Mine, located around 100 km east from the X1 Deposit by road and around 90 km from Peixoto de Azevedo town. The operation consists of open pits with a heap leach plant and tailings and disposal facilities. The capacity and production of the operation is not known by Aura.

Recently P.A. Gold is implementing the Filão do Paraíba Mine, located around 60 km southwest from X1 Deposit by road. The project will consist of underground operations of a high-grade lode quartz rich in gold and copper.

Additionally, hundreds of garimpo operations are currently active all around the Project area and in the district, mostly under the Coogavepe, the district Cooperativa of Garimpo.

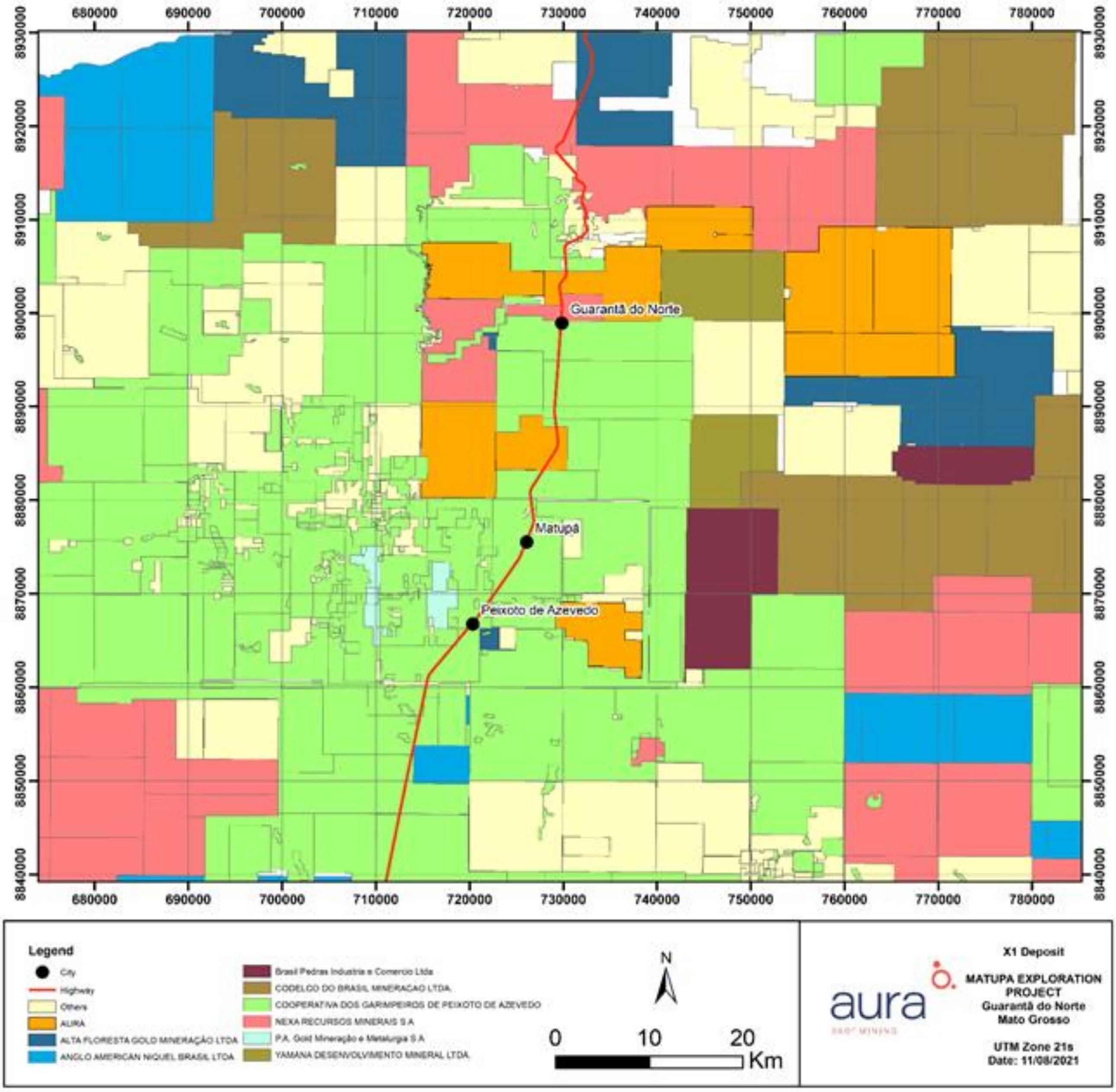


Figure 23-1 - Adjacent Properties Around X1 Matupá Gold Project with Active Exploration or Operation Activities.

24. OTHER RELEVANT DATA AND INFORMATION

There is no other relevant information available at the time of this report.

25. INTERPRETATION AND CONCLUSIONS

25.1 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACTS

The Matupá Project is in the initial stage of environmental licensing and is expected to obtain the main Licenses and Authorizations throughout 2022 and the end of the first half of 2023 to start construction. The Operating License will be required 4 months before the end of the works.

As preventive measures to avoid soil and water contamination, the Project provides for a drainage system (internal and superficial) and a set of structures (sumps) for solid containment and treatment, in the areas of the pit, waste rock pile, dry staking tailings pile, and low grade ore pile for effective water management in the rainy season. In addition, the base of the piles will be lined to avoid any contact of effluents with water courses.

The Matupá Project is not located in sensitive areas such as indigenous lands, territories of traditional populations and settlement projects, nor in areas of Parks, Ecological Reserves, and other Conservation Units. Neither archaeological nor speleological remains were identified in the area of influence of the Project.

Finally, the Environmental Impact Assessment (EIA) concluded that the Matupá Project is environmentally viable, provided that the negative impacts identified are not presented as impediments to the implementation of the Project, and it was well received by the authorities and population of Matupá city and region in the Public Hearing held on May 10, 2022.

25.2 GEOLOGY AND MINERAL RESOURCES

The Matupá Project is located on the prolific Juruena-Teles Pires Gold Province, specifically in the Peixoto de Azevedo District where many gold deposits and occurrences exist.

The mineralization and alteration occurs in Paleo-Proterozoic granitic host rocks (1.87± 12 Ma) belonging to the Matupá Intrusive Suite, including biotite monzogranites and biotite granodiorites both porphyritic (K-feldspar porphyries), derived from oxidized magmas, type I, from calcium-alkaline to sub-alkaline, metaluminous to peraluminous, medium to high potassium and magnesian to slightly ferrous affinity (Assis et al., 2014).

Early potassic Alteration (1.77± 5.7 Ma) was later overprinted by pervasive phyllic alteration. Gold mineralization in the X1 Deposit is mainly associated with strong phyllic alteration represented by coarse muscovite. In the Serrinhas Target, most of the gold mineralization is associated with early potassic alteration.

The X1 Deposit occupies a topographic high point (hill) in the area of the Project and is hosted by the Matupá Intrusive Suite near the contact with the mafic/ultramafic Flor da Serra Suite. The X1 Deposit extends 400 meters along strike from east to west and 150 meters from north to south. The main host rock is porphyritic granodiorite which is intruded by a quartz feldspar porphyry (QFP). Gold occurs in the disseminated form associated with intense phyllic alteration (quartz + muscovite + pyrite) in the QFP. The X1 Deposit strike extension is cut-off by two possibly normal faults which resulted in the exposure of the Deposit to further weathering due to higher topographic relief.

X1 mineralization is mainly the sulfidic type and oxide (saprolite and weathered intrusive) is limited to the weathering profile on top of the hill. The depth of the weathering profile varies between 10 m to 50 m, averaging 30 m.

The Matupá Feasibility Study includes an updated Mineral Resource Estimate for the X1 Deposit, which was completed to incorporate all geological data from previous drilling campaigns at the X1 Deposit since January 2012. The updated Mineral Resource Estimate is based on the alteration models which encompassed all economic gold mineralization in the X1 Deposit. The alteration models were described as phyllic and early potassic alterations. A high-grade shell model constructed for strong, pervasive, phyllic-alteration grade interpolation was performed separately, with a hard boundary between the phyllic and the strong pervasive phyllic alteration domains. These mineralized domains were analyzed for grade capping and variography, and was interpolated using the ordinary kriging method. Once the block model was completed it was classified into Measured, Indicated, and Inferred Mineral Resources, followed by a Lerchs-Grossman open pit optimization which resulted in the Mineral Resource Estimate presented in Table 25-1.

Table 25-1 - Matupá Gold Project – X1 Deposit Mineral Resource Estimate.*

Classification	Tonnage (t)	Au (g/t)	Au (oz)	Ag (g/t)	Ag (oz)
Measured	4,692,520	1.14	172,000	3.85	580.810
Indicated	4,653,150	0.96	143,600	4.39	656,430
Measured + Indicated	**9,345,670**	**1.05**	**315,600**	**4.12**	**1,238,240**
Inferred	77,560	0.78	1,950	1.25	3,120

**Mineral Resources Notes and Assumptions*

1. *The Mineral Resource Estimate has an effective date of August 31, 2022.*
2. *Mineral Resources are inclusive of Mineral Reserves.*
3. *Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.*
4. *The Mineral Resources in this estimate were calculated with the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions.*
5. *The base case cut-off grade for the estimate of Mineral Resources is 0.35 g/t Au.*
6. *The Measured and Indicated Mineral Resources are contained within a limiting pit shell (using US$1,800 per ounce of gold) and comprise a coherent body.*
7. *A density model based on alteration and rock type was established for volume to tonnes conversion averaging 2.76 tonne/m^3.*
8. *Contained metal figures may not add due to rounding.*
9. *Surface topography as of July 31, 2021.*
10. *The Mineral Resource Estimate for the X1 Deposit was prepared by Farshid Ghazanfari, P.Geo., a Qualified Person as that term is defined in NI 43-101.*

25.3 MINING AND MINERAL RESERVES

25.3.1 DRILLING AND BLASTING

The blasting rock fragmentation studies selected a primary blasting pattern to accomplish the size distribution curve with the following parameters: top size of 600 mm, P_{80} of 250 mm, P50 of 80 mm.

The analysis are based on the X1 orebody rock mass data available compared to the other mining operations, in similar conditions to the rock characteristics and operational conditions.

The distribution size curve analysis presented in this report considers a blasting pattern to the ore rocks based on a 5.0-inch blasthole diameter and 410 g/t explosive charge ratio. When compared to the blasting pattern fragmentation analysis reference a blasthole diameter of 5½ inches was chosen for the Matupá Project, which provides an additional safety margin due to the smaller blasthole diameter. The planned bench is 10 m high for the waste rock and 5 m high for the orebody.

In addition to the 410 g/t explosive charge ratio to the ore rock blasting pattern, electronic caps were selected to assure the proper fragmentation for the grinding operation. The grinding costs are much higher than drilling and blasting costs and grinding costs will be even higher if the blasted rock fragmentation is poor. In the waste rock, the explosive charge ratio will be 220 g/t, with non-electric accessories.

Drilling and blasting are based on many empirical aspects, and they can be optimized during operation.

The primary crusher feeding system will be supplied via a Front Wheel Loader, allowing control block size selection and avoiding operational problems with big boulders generated by the primary rock blasting operation in mine, that were not separated by the excavator in the mine by the ROM loading operation phase. After truck transport the ROM will be dumped in the pile, close to the crusher area. The Front End Loader can select the proper block size to be fed in the primary crusher. The operation has an additional block size selection to avoid crusher operational problems.

The volume estimation to block size bigger than 600 mm is around 3% of total ore blasted volume. To break the boulders in the primary crusher yard a hydraulic breaker was selected, bigger than 2.400 kg, mounted onto a hydraulic excavator.

25.3.2 REVERSE CIRCULATION WITH DOWN THE HOLE HAMMER (DTH-RC) RIG FOR GRADE CONTROL

Reverse circulation (RC) drilling has become a standard practice in most of mines worldwide for grade control to support mining reconciliation. The first developments were in Australia in the 1970s. This drilling technique was originally applied as a solution to face drilling difficulties encountered in soft iron ore and oil mineral sands. The first RC drilling equipment were adapted from the United States oil industry and manufactured in Western Australia in 1972 by Bruce Metzke and John Humphries.

A reliable sampling method is necessary to provide good grade control. The grade control system is the way to control the operation and minimize the ore being discarded in the waste dump piles and waste rocks from being fed to the concentration plant.

Complementary to the proper equipment selection for sample collection, it is important to have an experienced team for a quick reaction to grade control changes and to achieve the targets for the existing mineral development from the mineral deposit. A proper grade control system is the way to ensure the profitability to the entire mine project operation. The Qualified Person (QP) responsible for the mining operation disciplines strongly recommends the use of this specific technology.

25.3.3 COSTS

The costs applied are realistic and provide a robust basis to the project economic analysis. The data related to the mine costs are from one actual gold mine in similar mining operation: Aura's Apoena gold mine in MT State. The main source for these costs was the contract, signed on May of 2022, to hire a contractor for Apoena's Ernesto gold mine operation. Also, the rock blasting cost estimates, were from the Ernesto Mine operation.

25.4 METALLURGY AND PROCESSING

The three metallurgical test work campaigns were carried out on samples representing the two main ore types of the Matupá X1 Deposit. The first two campaigns used representative samples for the resources, while the third included two samples representing the LOM for both consolidation and variability assessments. In all three campaigns the test work included dedicated testing for comminution, flotation, gravity concentration, leaching, cyanide destruction, solid-liquid (rheology, sedimentation and filtering), dry stacking, the latter including handling and physical/geotechnical characterization testing.

The third testing campaign included LOM samples representing (a) pay-back period – year 1 to 3 and (b) remaining period – year 4 onwards. The blending between the two ore types used in the consolidation testing was 90% fresh rock and 10% oxide, as predicted by the production plan.

The consolidation test work results indicated a 1% increase in gold recovery by including a gravity concentration stage in the processing circuit, as well as benefits in gold recovery for variability samples. Therefore, the selected process for the Matupá industrial circuit includes gravity concentration and intensive leaching of gravity concentrate, followed by leaching/absorption Leach-Carbon in Leach (L-CIL) route.

The adopted capacity for the Matupá industrial plant was 1.3 Mtpa for a ground product to a P_{80} of 0.125 mm.

The third metallurgical test work campaign resulted in a model for predicting overall gold recovery as a function of gold grade for the entire LOM period.

Even though the ore reserve is not specific for silver, the recovery of such a metal was assessed in the consolidation tests. At present there is no indication that silver would interfere in the extraction and elution method selected for the Matupá Project. As silver recovery represents an opportunity for additional revenue for the Matupá Project it should be further assessed.

The selected method for cyanide neutralization (SO_2/air) resulted in adequate performance according to environmental regulatory specifications. Online reagent control for assessing the cyanide neutralization in real time is part of the Matupá Project processing plan.

Even though the LOM blend included 10% of the oxide ore type, the design of thickening and filtering operations was based on higher proportions of oxide ore, which are likely to occur during regular operation.

The adopted method for tailing disposal (dry stacking) will avoid building dedicated dams, together with providing a relatively high rate of recirculated water for the Matupá metallurgical processing plant.

The process criteria and process route selected for the Matupá Project were solidly based on the metallurgical test work conducted, as well as on standard practices of similar industrial circuits.

25.5 CAPITAL AND OPERATING COSTS

Capital and operating costs were estimated for a feasibility study. Total initial project capital costs total US$107M and operating costs are $22.71/t year processed or an average annual expenditure of US$28.73M.

25.6 INFRASTRUCTURE

The designed infrastructure for the Matupá Project meets operational requirements; minor technical recommendations were described for next phase. The Matupá Project infrastructure includes the required access, power supply, water supply, tailings storage, and support facilities to support ore production.

25.7 MARKET STUDIES AND ECONOMIC ANALYSIS

GE21 analyzed the gold market study prepared by World Bank concluded that the demand for gold is growing as investors increase their focus on long-term investments. This causes the price of gold to rise accordingly. However, the key factor that is fueling the demand for the precious metal is a high level of uncertainty observed in the global economy due to the Coronavirus situation and war between Russia and Ukraine.

The financial model adopts the concept of project free cash flow, in which all of the project's cash generation capacity is evaluated by countering this flow with a weighted discount rate ("WACC") which reflects the average cost of sources of funds (cost of equity and third parties). The amounts in the cash flow were expressed in thousand dollars (US$ x 1,000) and on a real basis (without inflation).

Based on the assumptions adopted, the post-tax net present value ("NPV") of Aura Minerals Matupá Gold Project base case scenario amounts to US$ 96,128 million, at a Discount Rate of 5.0%.

The internal rate of return (IRR) is 27,5% and the annual average EBITDA (from year 1 to year 7, full run rate production period) is US$ 40,045 million. Payback after the start-up of operations is 2.04 years. The results of the economic analysis are summarized in Table 25-2.

Table 25-2 - Financial Results Summary.

VALUATION - BASIC PROJECT		
NPV	96,128	US$ x 1.000
IRR	27.5%	
Leveraged IRR	49.9%	
Profitability Index	1.95	
Discounted Payback	4.56	years
Simple Payback (Including Start-Up)	4.29	years
Simple Payback (After Start-Up)	2.04	years
Discont Rate	5.0%	
Invest in the Project?	Yes	

26. RECOMMENDATION

26.1 EXPLORATION

In terms of exploration, Aura strongly believes that there is an exploration upside in the Project area to expand and improve the Mineral Resources, and consequently, the future Project's Life of Mine (LOM). Based on this, Aura has increased its mineral rights position in 2020 and 2021 from 28,674 hectares to 62,505 hectares (118% increase). Aura is planning to maintain the aggressive exploration plan started in early 2021 in to 2022 and beyond.

Several 100% owned satellite targets in a 30 km radius of the X1 Deposit have been identified so far, the most advanced is the Serrinhas Target, which is located 27 km from X1 by paved highway. The target consists of 10 km northwestern trending hills with a series of former artisanal small pits, and notable gold anomalies in the soil.

A total of 62 historical, diamond drill holes from the 1990s and 2000s, totaling 9,020 m exist within the Serrinhas Target area. Positive gold intersections are present within these drill holes, indicating the possible existence of near surface mineralization zones. Aura is currently testing those zones with new drilling. Aura is conducting a QA/QC validation program of historical holes, consisting of resampling existing core and a twin-hole program, which will also provide samples for a metallurgical characterization determination. A total of 12,074 m in 54 holes were drilled in Serrinhas, as of the effective date of this report.

Continued exploration and drilling are recommended for the Serrinhas Target area to try to further delineate the lithology and mineralization in the area. This recommendation is summarized in Table 26-1 which lists the type of work recommended and the expected costs involved as a proposed exploration budget for 2022 and 2023.

Table 26-1 – 2022-2023 Proposed Exploration Budget.

Activity	Type	Estimated Cost (US$)	DD Drilling (m)	RC Drilling (m)
MATUPÁ				
SERRINHAS (Phase 1): Infill + Extension + Exploration Drilling: Validation, extension and infill of mineralization delineated by historical drilling (MP2 Zone). Geological mapping / sampling and exploration drilling along the remaining gold anomalous trend.	EARLY STAGE	2,500,000	20,000	6,000
SERRINHAS (Phase 2): Infill and step out drilling over the mineralization discovered in the Phase 1.	ADVANCED	2,300,000	22,000	-
Matupá District: Geological Mapping and surface Sampling following by exploration drilling in other satellite early-stage gold target within Aura mineral rights / Concession holding costs	EARLY STAGE	550,000	5,000	4,000
TOTAL MATUPÁ		**3,150,000**	**21,000**	**6,000**

26.2 GEOLOGY AND MINERAL RESOURCES (X1 DEPOSIT)

Matupá Project Feasibility study is limited to the X1 Deposit Mineral Resources at the effective date of this technical report and consequently has a relatively short Life of Mine (LOM).

The X1 Deposit upside is limited as the deposit is constrained by faulted blocks of barren intrusive to the east and west of the known mineralization. Rio Novo drilling (2011-2012) and Aura drilling (2020-2021) was unsuccessful in identifying any mineralization along strike or in the vicinity of the X1 Deposit so far. Down dip extension of the Deposit is marked by weak phylllic alteration which has resulted in lower grades at depth. Most of the Inferred Mineral Resources are located below the resource pit, therefore converting these Mineral Resources to Measured and Indicated classifications are not recommended at this time. The amount of Inferred Resource within the optimized pit is small in volume and not material to the Project. Therefore, additional infill drilling within the resource pit not recommended at this time, and it can be performed during ramp up to production.

However, in the opinion of the QP, the additional satellite deposits such as Serrinhas as described in this section need to be verified and further explored to increase the Life of Mine (LOM) of the Matupá Project. The Serrinhas Target has the potential to double up the existing Mineral Resources for the Matupá Project and could be a significant material change for the Project.

Below are further recommendations that need to be considered regarding the geology and Mineral Resources of the X1 Deposit exclusively:

- Further bulk density measurements need to be done on altered and saprolite zones by measuring the moisture. Dry densities need to be determined on additional samples with different levels of weathering profiles and preferably on larger sample sizes.

- Relogging of some of the previous drill holes especially in higher grade part of the X1 Deposit for purpose of modeling the mafic dykes to incorporate into the Mineral Resource model as a separate mineralized domain, if this is possible.

- Re-assaying of silver values for selected drill holes (preferably using existing pulps) by Fire Assay method.

- Further re-sampling of drill holes related to Mineração Santa Elina (MSE) drill campaigns need to be done, especially for drill holes with elevated sulfur grade in the west part of the X1 Deposit which hosts most of the contained ounces.

- The post mineralization fault zones need to be mapped in detail on the surface, their motions need to be understood and tied in to drill holes, if it possible, to construct a 3-D model of all fault zones.

- A mineralogical study using TIMA_X and QEMSCAN by SGS Canada is recommended for the X1 Deposit to understand the relationship between the gold and copper and also determine gold liberation in both strong and weak phyllic alteration zones.

26.3 MINERAL RESERVE ESTIMATION

EDEM considers the Geotechnical information, at this stage, enough to start the mining operation, but it is important to start the operation with an experienced, and dedicated, geotechnical team to assure a good geotechnical monitoring program to give a good support to all the operations. The geotechnical team must be able to, eventually, revise the Matupá Open Pit mining plan according to the mine operation requirements.

An electronic dispatch is highly recommendable for all the operation control, including mining operation contractors.

EDEM recommends a constant oxidized ore blending to be fed in the plant at 10% of the total ore. This blending policy is recommended for all of the mine life (LOM).

Create a low-grade ore stock, in a separated ore pile, to be fed in the last months of the LOM operation period.

26.4 MINING METHOD

- EDEM recommends grade control drilling with Down-The-Hole reverse circulation drills to support the grade control engineering.

- The grade control drilling, to feed the initial concentration plant operation, must be done to complete the mining medium term plan modeling before starting feeding the plant. Afterwards, continuing the grade control drilling and mining model updates to have the grade control engineering carefully studied in advance

- Bulldozer size for the mining operation is recommended to be heavier than 35 tonne or bigger; similar to a CAT D8.

- EDEM recommends having one Wheel tractor, similar to a CAT 824, for every three units of hydraulic excavator to complement the grader work and to keep the loading fronts clean and prepared for a proper loading trucks operation. Alternatively, but not as efficiently, consider using a bulldozer, similar to a CAT D6, because it is easier to find them in the Brazilian market.

- As the ROM transport is the most expensive operation unit when considering the total rock exploitation, it is recommended to have an extra capacity for the loading fleet, and for all the equipment fleets, that affect the ROM transport operation, assuring the trucks are in good operational condition.

- An electronic dispatch system is highly recommended for all operational control.

26.5 MINERAL PROCESSING AND METALLURGICAL TESTING

Changes that occur between the Basic Project and the Detailed Project must comply with a change management protocol, auditable and with risk assessments considering aspects of the production process, environmental and occupational safety. The risks and their interrelatedness with the different Project disciplines must be identified and a mitigation plan drawn up.

In the Executive Project phase, the commissioning, startup (ramp up) and operation training plans must be developed. These plans must be ready at the end of construction as well as the qualified team properly trained according to the manuals and practices for each operation unit.

Tests for control loops and interlocks for all operations, especially for grinding, must be exhaustively performed in order to identify and cover all possible deficiencies and failures of interlocks, response times, and control loop adjustments - PID.

It is recommended to have a process team, already working in the final phase of construction, for the development of geometallurgical studies to predict the results and conduct operational adjustments.

The predicted make-up water consists of raw water, with low hardness, for use in elution, reagent preparation, cooling systems, sealing, and dedusting. The reuse of water accumulated in effluent treatability ponds from the piles and pit must have its chemical and physical characteristics, such as hardness, evaluated before its application, especially for possible replacement of raw water during periods of drought.

A critical step that must be carefully evaluated is the dry deposition of the tailings (low humidity). This operation will demand that the dosage of reagents and monitoring of the residual content of CNwad be under control in real time, guaranteeing the efficiency of the Detox - SO_2/air treatment, in order to guarantee compliance with occupational and environmental parameters.

The bullion produced by the Matupá industrial operation will have silver in its composition. The predicted amount of recovered silver is three to four times the recovered gold amount. Such an assessment was based on the samples tested in the consolidation phase as representative of the LOM. A silver recovery model was developed as a function of silver grade in the plant feed. The additional cost for producing the silver will only occur in the bullion refining phase. Risk assessment should refer to the taxation of CFEM (Financial Compensation for Mineral Exploration) and any royalties due. This situation derives from the lack of declaration of silver from ore reserves, which contrasts with silver production. It is thus recommended to anticipate evaluations and negotiations with ANM, for avoiding legal issues.

26.6 RECOVERY METHODS

Relevant recommendations regarding recovery methods were previously described in Section 25.5 together with mineral processing and metallurgical testing issues.

26.7 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITIES IMPACTS

The recommendations for licensing, environmental and social management of the Matupá Gold Project are presented below:

- Strong monitoring of all stages of licensing with SEMA-MT to obtain the licenses and ancillary authorizations necessary for the installation and operation of the Matupá Project;
- Prepare a Physical Financial Schedule for all stages of licensing, with all environmental and social studies and plans required for all licenses/authorizations until the issuance of the Installation License to minimize construction delays;
- Intensify social studies and mapping of stakeholders to support the social management of the enterprise;
- Mining planning must verify the possibility of segregating waste rock material with sulfide contents up to 1%;
- Once the mining activities are started, a program to evaluate the potential for generating acidity under field conditions should also be started;
- Avoid the use of materials from mineralized areas, especially waste rock, in works until the conclusion of the Acid Mine Drainage tests;
- Conduct studies to evaluate mixing zones in water courses adjacent to waste rock piles, tailings, low-grade ore, and pit for managing the release of effluents from them;
- Water and sediment monitoring (baseline) should be intensified, in order to obtain consistent and sufficient pre-operational data for the assessment of base metal levels;
- Discussions about water management or treatment should remain on the agenda, composing an effluents management plan from the piles and mine;
- Evaluate the possibility of reusing the wastewater from the piles in order to reduce the demand for new water;
- The waters from the pit operation should be considered in the ongoing assessments on water management, as they will probably require some type of treatment and/or management; and

- Preventive engineering measures should be adjusted according to the results of acid drainage potential studies.

SIGNATURE PAGE

Feasibility Study Technical Report (NI 43-101) for the Matupá Gold Project, Matupá Municipality, Mato Grosso, Brazil

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28. CERTIFICATES

CERTIFICATE OF QUALIFIED PERSON

FARSHID GHAZANFARI, P.GEO.

As an author of this report titled "Matupá - Technical Report feasibility Study - NI 43101 for the Matupá Gold Project, Matupá Municipality, Mato Grosso, Brazil" dated October 07, 2022, I, Farshid Ghazanfari, P.Geo., do hereby certify that:

1. I am Director of Geology and Mineral Resources for Aura Minerals residing at 2135 Heidi Ave., Burlington, Ontario, L7M 3P4, Tel.: (905) 483-6272 and carried out this assignment on behalf of Aura Minerals.

2. I am a graduate of the Tehran University (Iran) having been awarded a M.Sc. (Hons.) Degree in Geology in 1992.

3. I have worked as geologist in mineral industry for 30 years. My work experience include five years for Geological Survey of Iran as geologist and mineralogist, six years as exploration geologist with major mining companies for gold and base metals including two years of underground experience in Northwest Ontario, Canada, six years as resource geologist consultant for junior mining sector estimating range of mineral deposits from base and precious metals to industrial minerals. I also practiced 3 years as an independent consultant in mining industry. I was involved with Aura Minerals with my current role since 2015.

4. I am a Professional Geologist in good standing with the Association of Professional Geologists of Ontario, License #1702.

5. I am the QP responsible for sections 3, 4, 6, 7, 8 , 9, 10, 11, 12, 14 and 23, and summaries there from in sections 1 and 25 of the technical report entitled "Matupá - Technical Report Feasibility Study - NI 43101 for the Matupá Gold Project, Matupá Municipality, Mato Grosso, Brazil"

6. I visited the Matupá Gold Project in Mato Grosso State Brazil and Aura's core logging facility in many occasions between 2018 to 2021 and my last visit was between October 16 to 18, 2021.

7. I have had prior involvement with the properties that are subject to the Technical Report.

8. I have read NI 43-101 and Form 43-101F1 and the Report and the portion of the report for which I am responsible has been prepared in compliance therewith.

9. I am a "qualified person" for the purposes of NI 43-101 due to my experience and current affiliation with a professional organization (Professional Geologists of Ontario) as defined in NI 43-101.

10. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signing Date: November 18 2022
Effective Date: August 31, 2022

"Farshid Ghazanfari" {signed and sealed}
Farshid Ghazanfari, P.Geo.

LUIZ EDUARDO PIGNATARI, P.ENG.

As an author of this report titled “Matupá - Technical Report Pre-feasibility Study - NI 43101 for the Matupá Gold Project, Matupá Municipality, Mato Grosso, Brazil” dated August 30, 2022, I, Luiz Eduardo Pignatari, P.Eng., do hereby certify that:

1. I am a QP consultant by, Chilean Commission for the Qualification of Competencies in Resources and Reserves) – CH 20.235 nº 288.

2. I graduated with degree in Mining Engineer from the University of São Paulo (1978) with Post Graduation in Mining Operations from the same institution.

3. I have been worked continuously since I have concluded my graduation as a Mining Engineer, and I have a large mining operation experience and its mineral processing, in mineral exploration, technical evaluation for many mining enterprises with economic financial feasibility studies, always with a focus on the most advanced technology and operational intelligence. I spent a significant amount of time working for gold mining, phosphate, and cement manufacturing, including, also, major corporations such as Bunge Fertilizers, Yamana Gold and Camargo Correa Cement.

4. I have read the definition of “Qualified Person” as set out in the National instrument 43-101 and certify that I am a Qualified Person according to Comisión Minera CH-20.235 nº 288, accepted by NI 43-101 and JORC.

5. I have visited the Matupá Gold Project in Mato Grosso State Brazil and Aura’s core logging facility and warehouse in the town of Matupá on July, 2022.

6. I am responsible for Sections 3, 4, 5, 15, 16, 18 and 20 of this Technical Report and summaries therefrom in Section 1 and 25.

7. I am independent of Aura Minerals Inc., as defined in Section 1.5 of NI 43-101.

8. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make this report not be misleading.

Signing Date: November 18 2022
Effective Date: August 31, 2022

“Luiz Eduardo Pignatari” {signed and sealed}
Luiz Eduardo Pignatari, P.Eng.

HOMERO DELBONI JR, B.E., M.Eng.Sc., Ph.D., MAusIMM CP (Metallurgy).

As an author of this report titled “Matupá - Technical Report feasibility Study - NI 43101 for the Matupá Gold Project, Matupá Municipality, Mato Grosso, Brazil” dated October 07, 2022, I, Homero Delboni Jr, Ph.D., MAusIMM CP (Metallurgy), do hereby certify that:

1. I am a Senior Consultant of HDA Serviços S/S Ltda., residing at Alameda Casa Branca, 755 cj. 161 São Paulo, SP 01408-001 Brazil, Tel +55 11 98383-4678.
2. I graduated with a Bachelor of Engineering Degree in Mining and Mineral Processing from The University of Sao Paulo (Brazil) in 1983, concluded a Masters in Engineering in Minerals Processing in The University of Sao Paulo (Brazil) in 1989 and obtained a Ph.D. in Minerals Processing Engineering at The University of Queensland – Julius Kruttschnitt Mineral Research Centre, Brisbane (Australia) in 1999. I am a Member (#112813) and Chartered Professional in Metallurgy of the Australasian Institute of Mining and Metallurgy – MAusIMM – CP (Metallurgy) I have worked as a Minerals Processing engineer for a total of 39 years since my graduation from university.
3. I have read the definition of Qualified Person set out in the National Instrument 43-101 (Instrument) and certify that by reason of my education, affiliation with a professional association and past relevant work experiences, I fulfil the requirement to be an independent qualified person for the purposes of NI 43-101.
4. I have read NI 43-101 and Form 43-101F1 and the Report and the portion of the report for which I am responsible has been prepared in compliance therewith.
5. This certificate applies to the technical report titled “Matupá - Technical Report Feasibility Study - NI 43101 for the Matupá Gold Project, Matupá Municipality, Mato Grosso, Brazil” dated October 07, 2022.
6. I have not visited the Matupá Gold Project.
7. I am responsible for Sections 2, 5, 13, 17, 21 and 22 and summaries there from in Sections 1 and 25 of this Technical Report.
8. I am independent of Aura Minerals Inc., as defined in Section 1.5 of NI 43-101.
9. I have been involved with the Matupá Gold Project since 2021, during the preparation of the preliminary economic assessment study.
10. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with that instrument and form.
11. As of the effective date of this certificate, to the best of my knowledge, information and belief, the technical sections of the Technical Report, for which I am responsible contains all scientific and technical information that is required to be disclosed to make this report not be misleading.

Signing Date: November 18, 2022

Effective Date: August 31, 2022

“Homero Delboni Jr” {signed and sealed}

Homero Delboni Jr, Ph.D., MAusIMM CP (Metallurgy)