Updated Feasibility Study Technical Report (NI 43-101) for the Almas Gold Project, Almas Municipality, Tocantins, Brazil

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TABLE OF CONTENTS

List of Tables

List of Figures

1 EXECUTIVE SUMMARY

The Almas Gold Project is located in the municipality of Almas, in Tocantins State, Brazil (Figure 1-1). The Project consists of three separate open pit mining areas and a central processing facility. The Almas Project's three main gold deposits, Paiol, Cata Funda and Vira Saia are located along a 15 km long corridor of the Almas Greenstone Belt, a Paleoproterozoic volcano-sedimentary sequence which hosts numerous orogenic gold occurrences. The project development begins with the construction of a CIL mill facility at Paiol. The Paiol open pit will be prepared for production during the construction period, produce for approximately three years followed by development and production from Cata Funda and Vira Saia. Mined material will be trucked from the satellite deposits to Paiol to maintain the annual plan.

Figure 1-1 The Almas Gold Project Location

Aura completed an updated Feasibility study with the collaboration of other engineering and geological companies including Ausenco, EDEM and Micon and other individual consultants on the Almas Gold Project.

The study consisted mainly of the update of the Mineral Resource Model and further metallurgical test work, pit design, mine schedule, tailings management technologies, costs, and economic model, providing an increased level of accuracy in the Project estimates with respect to the previous 2016 feasibility Study for Rio Novo by RPM.

The Mineral Resource and Mineral Reserves disclosed in this report supersede all previous estimates for the Almas Project. This Executive Summary highlights the work undertaken between 2018 and 2020 and outlines the material changes between this study and previous studies.

1.1 PROPERTY DESCRIPTION AND OWNERSHIP

The Almas Gold Project area lies south of Almas, a small town approximately 300 km southeast of Palmas, the Tocantins State Capitol, and 45 km west of Dianópolis, a regional commercial center. The Almas Gold Project refers to Rio Novo's past exploration and current economic evaluation and planned development by surface mining of gold deposits. This report focuses on the Paiol, Cata Funda and Vira Saia gold deposits that are distributed along a 15 km long segment of the Almas Greenstone Belt, south of the town of Almas. This segment of the belt contains numerous, small scale, artisanal gold mining sites, locally termed Garimpos, whose development preceded Rio Novo's exploration activities.

The Paiol deposit, which was previously mined, and the undeveloped Cata Funda deposit, are situated, respectively, on two inactive Mining Concessions previously assigned to VALE. The Vira Saia deposit is on two exploration permits whose acquisition from a third party was finalized by Rio Novo in 2012.

The Almas Gold Project includes the historic open pit/heap leach Paiol Project operated by VALE from 1996 until 2001, which produced approximately 86,000 ounces of gold. The former open pit is flooded, and the waste dump and heap leach pad have been reclaimed. Most of the VALE facilities have been removed fulfilling reclamation requirements of the Tocantins state environmental authority (NATURATINS).

Rio Novo's mineral rights covering the principal areas of interest, including the Paiol and Cata Funda gold deposits, which are controlled, respectively, by two Mining Concessions (9,137 ha). The Vira Saia deposit is held by two Mining Concession Applications (4,483.75 ha) submitted on March 5, 2013.

The Almas Gold Project includes the properties covered by the Mining Concession ANM number 860.128/1983– Paiol (mined in the past by VALE); the property under ANM number 862.224/1980 – Cata Funda which is undeveloped, and the Application of Mining Concession coincident with de Vira Saia deposit (864.083/2006, 860.373/1988).

1.2 GEOLOGY AND EXPLORATION

The Almas Gold Project area is situated within the Almas-Dianopolis Greenstone Belt (AGB) of Archean-to- Paleoproterzoic age. The greenstone belt lies within the Almas-Conceicao Terrane on the western block of the Goias Massif.

The Paleoproterozoic granite-greenstone terrane is composed of gneissic granite domes with folded, narrow domains of metabasic and metasedimentary rocks including tholeiitic metabasalts and calc-alkaline metatonalites that have been subjected to strong regional metamorphism. The metamorphism resulted in deep-seated, shear-hosted, mesothermal, gold deposits which can be considered as orogenic gold deposits. The gold-mineralized zone occurs in the core of hydrothermal alteration zones, generally associated with variable amounts of quartz, carbonate, albite, sericite and sulphide minerals.

Gold in the Almas Greenstone Belt occurs in four different geological settings:

- Gold associated with hydrothermally-altered shear zones in basic to intermediate volcanic rocks (e.g. Paiol and Cata Funda);
- Gold associated with hydrothermally-altered shear zones in felsic tuff;
- Gold associated with hydrothermally-altered banded iron formation (e.g. Morro de Carniero);
- Gold associated with smoky quartz veins in sheared granite gneiss (e.g. Vira Saia).

The main Paiol ore body has overall dimensions of approximately 650 m in the down dip direction, 1,250 m along strike and averages 30 m in thickness. The Cata Funda ore body has overall dimensions of approximately 240 m in the down dip direction, 230 m along strike and averages 10 m in thickness. The Vira Saia ore body has overall dimensions of approximately 200 m down dip, 350 m along strike and averages 15 m in thickness. At Vira Saia gold is closely associated with sulfide-bearing, quartz-sericite rich ultra-mylonitic formed in the core of shear zones developed in granodiorite. In the mineralized zone chalcopyrite and galena are rare. The intensity of the hydrothermal alteration is proportional to the progressive deformation in the shear zone.

Exploration within the Almas Gold Project dates back to 1977 when VALE identified prospective terrain in the greenstone belts around Almas. Workers in the area have used a combination of geophysics, geochemistry and geologic mapping to discover numerous gold anomalies. The Paiol deposit was discovered in 1987. The Paiol discovery was significant in that the deposit did not crop out, and the discovery was based on a weak soil anomaly and geophysics.

Rio Novo continued to conduct geological, geochemical, and geophysical surveys during exploration of areas adjacent to the known deposits. These surveys led to the discovery of the Vira Saia deposit in 2011 as well as a few other prospects still in the exploration stage.

It is important to note that exploration thus far has been primarily designed to identify near-surface prospects. The deeper, covered areas of the district have yet to be explored. Due to the generally flat terrain and thick soil or saprolite cover, only a small portion of the district has been adequately covered by exploration. None of the three deposits has been truly drilled off and opportunity exist to extend them along strike and down dip beyond the current footprints.

1.3 DRILLING, SAMPLING & ASSAYING

At Paiol, the known extents of mineralization have been drilled out on nominal 25 m centers. Drilling covers an area of about 2,000 m along strike and 300 m across. Additional scout holes have been drilled around the perimeter. The deposit is primarily drilled out to a vertical depth of 250 m to 300 m, although individual drill holes have been drilled as deep as 500 m (vertical depth). In total, there have been 467 diamond core holes drilled in the Paiol area, for approximately 72,500 m. VALE drilled 519 and Rio Novo drilled 33 shallow reverse circulation holes in property.

At Cata Funda, the deposit has been drilled out at nominal 25 m x 25 m centers. The drilling covers an area of about 700 m along strike and 250 m across strike. The deposit is drilled to a vertical depth of about 80 m to 100 m, with an average down hole drilling length of 120 m and the deepest holes reaching vertical depths of 150 m to 170 m. A total of 183 core holes totaling 21,400 m were drilled between 1996 and 2011 and were used to generate the Cata Funda 3D model. Reverse circulation drilling by VALE was not used in the models.

During 2011 and early 2012, a drilling campaign was completed at the Vira Saia discovery. In total, 194 diamond core holes were completed totaling approximately 26,500 m. The main drilling was oriented 045 degrees (N45E), perpendicular to the overall strike of the deposit. The deposit has been drilled to a vertical depth of 150 m to 180 m. Drill hole spacing in the resource area is nominally 25 m x 35 m.

At the Paiol Leach Pad, 92 reverse circulation holes and 166 auger holes were completed.

Rio Novo had a detailed QA/QC protocol which met or exceeded industry best practice using standards, blanks and duplicates as well as a primary and a secondary lab. The primary analytical laboratory used by Rio Novo for the Almas Project was the SGS Geosol laboratory, located in Vespasiano, Minas Gerais State, Brazil. The laboratory has ISO 9001 certification and ISO 14001:2004, ISO 17025:2009 certification for environmental chemical analyses.

SGS Geosol employs modern, industry standard techniques and analytical methods. For the purpose of routine gold analysis in the Almas Gold Project, fire assay with atomic absorption (AA) finish was used most frequently. Multielement analyses on 34 elements were determined by ICP subsequent to digestion of samples either in aqua regia acid or in four-acids. The second laboratory used by Rio Novo for check assays was ALS Chemex which prepped the samples in Vespasiano, Minas Gerais State and Goiania, Goias State, Brazil and completed the analyses at their lab in Lima, Peru.

It is the QP's opinion that the drilling, assaying and QA/QC protocols are sufficient to support a resource model at a feasibility level.

1.4 DATA VERIFICATION

The mineralization, logging, assaying, core storage and QA/QC procedures used, and their results, have been reviewed by Micon. The presence of gold at Paiol is supported by Vale's previous mining experience from 1996 to 2001 when approximately 86,000 ounces were produced. As well there are small scale open pit workings at the other two deposits Cata Funda (previously called Arroz) and Vira Saia.

In the QP's opinion the sampling, security and QA/QC procedures employed at the Almas Gold Project, and their results, are sufficient to produce data adequate for the purposes used in this technical report.

1.5 METALLURGICAL TESTING

The Almas Gold Project samples selected for metallurgical testing represented various ore types and lithologies within the different deposits. In addition, an overall composite representing the first three years of operation has been tested. Sufficient sample mass has been submitted for testing, so that tests were performed on a sufficient amount of material. The samples tested were not refractory and the mineralization was clean with low concentrations of cyanicides and other cyanide consumers present, suggesting that there will be no obvious environmental concerns.

The processing flowsheet selected for the Almas Gold Project incorporated proven technologies for the recovery of gold from ores. Metallurgical testwork completed on the project included a comminution study; gravity recoverable gold and gravity separation tests; evaluation of bulk sulphide flotation; cyanide leaching in the CIL and CIP circuit configurations, cyanide destruction with final effluent analysis, review of potential for deleterious elements; and solid-liquid separation testing.

The projected average overall recovery for the individual ore types tested was in the range of 93-95% and for the 3-Year Composite – 93%. The selected process design criteria included overall gold recovery of 92.5% at a grind of k80 = 75 microns

Cyanide and lime consumptions were low, which reflected the lack of cyanide consuming species present in the ore. Metal dissolution during cyanide leaching was found to be low, and there were no obvious concerns with the presence of environmentally deleterious elements.

1.6 MINERAL RESOURCES

The Almas Gold Project resources contain three mineral deposits Paiol, Cata Funda, and Vira Saia, and one heap leach pad (from historical production at the Paiol deposit). The resource estimate updates were performed for all three deposits plus historical leach pad materials.

For all three deposits, a nominal cut of grade 0.3 g/t Au and favorable lithologies were used to constrain mineralization models within structural and altered corridors for each respective deposit. The Paiol leach pad model was constrained between the topographical surface and a constructed bottom surface based on RC bore hole logging.

The 3D models were initially created on paper cross sections, perpendicular to the main strike, on generally 25 m centers (in places 35 m or 50 m). Cata Funda and Vira Saia models were created in Leapfrog software and the Paiol, Vira Saia extension and Leach Pad model were created using Gemcom Software.

The resource estimate for Paiol is based on both RC and diamond drill holes and for Cata Fund and Vira Saia based only on diamond drill holes. The resource model for the Paiol Leach Pad is based on assays from reverse circulation and auger drilling.

The resources were estimated based on Ordinary Kriging (OK). Resources were classified in accordance with the CIM Definitions and Standards for Mineral Resources and Mineral Reserves (CIM, 2014). The classification parameters were drawn from three different estimation passes and considered the proximity and number of composite data as well. The resource models used the first and second passes to assign the measured and indicated categories, respectively. To avoid "spotted dogs" in classification, a polyline was constructed section by section for all Measured and Indicated blocks using the above criteria. Moreover, historical drill holes without certificates, surveys or QA/QC were not used to define Measured and Indicated Resources. RC holes were also excluded for purpose of classification of resources (except for the Paiol Heap Leach Pad).

Table 1-1 shows the measured and indicated mineral resources which were constrained by respective optimized pits in different cut-off grades. The detail of each deposit cut-off grade assumption is discussed in section 14 of this report.

*Table 1-1 Almas Gold Project Mineral Resources **

ALMAS MINERAL RESOURCE		Tonnes	Au (g/t)	Au (Oz)
PAIOL	MEASURED (M)	4,366,950	1.03	144,870
	INDICATED (I)	13,181,190	0.96	407,590
	M&I	17,548,140	0.98	552,460
CATA FUNDA	MEASURED(M)	482,000	1.97	30,540
	INDICATED (I)	356,000	1.39	15,920
	M&I	838,000	1.72	46,460
VIRA SAIA	MEASURED(M)	566,910	1.24	22,600
	INDICATED(I)	2,787,780	0.91	81,245
	M&I	3,354,690	0.96	103,845
Heap Leach Pad (HLP)	INDICATED (I)	1,510,090	0.88	42,680
GRAND TOTAL (M&I)		23,250,920	1.00	745,445

*Note:

1. The Mineral Resource estimates were prepared in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by the CIM Council on May 10, 2014, and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, adopted by CIM Council on November 29, 2019, using geostatistical and/or classical methods, plus economic and mining parameters appropriate to the deposit.
2. The Mineral Resource Estimate is based on an updated optimized shell using 1800 $/oz gold price and cut-off grades of 0.29 g/t, 0.34 g/t and 0.31 g/t for Paiol, Cata Funda and Vira Saia respectively.
3. Contained metal figures may not add due to rounding.
4. Surface topography based on December 31st, 2016.
5. Mineral Resources are inclusive of Mineral Reserves.
6. The Mineral Resource estimate for the Cata Funda deposit was prepared by Adam Wheeler, C.Eng., a Qualified Person as that term is defined in NI 43-101.
7. The Mineral Resource estimate for the Paiol and Vira Saia deposits and HLP were prepared Farshid Ghazanfari, P.Geo., a Qualified Person as that term is defined in NI 43-101.

1.7 MINERAL RESERVE

The Almas Gold Project design includes three mineral deposits: Paiol, Cata Funda and Vira Saia; and the Heap leach pad from the previous Vale operation in the past (Figure 1-2).

The mineral reserves estimation was prepared using industry standard methods and provides an acceptable representation of the deposit. The Qualified Person (Reserves) reviewed the reported resources, production schedules, and modifying 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 ultimate pits can be classified as Proven and Probable Mineral Reserves respectively in accordance with the NI 43-101 definitions.

A Mineral Reserve of 21.8 Mt (dry) at an average grade of 0.92 g/t Au. The detailed breakdown of the Mineral Reserve by category and deposit is presented in Table 1-2. The Mineral Reserve is estimated on the basis of the current available information. The reserve classification reflects the level of accuracy of the Feasibility Study.

*Table 1-2 Almas Gold Project Mineral Reserves Summary**

ALMAS RESERVE		Tonnes	Au (g/t)	Au (Oz)
PAIOL	PROVEN	5,357,974	0.89	152,683
	PROBABLE	10,780,501	0.88	304,446
	TOTAL	16,138,475	0.88	457,129
CATA FUNDA	PROVEN	438,612	1.89	26,711
	PROBABLE	250,163	1.79	14,412
	TOTAL	688,775	1.86	41,123
VIRA SAIA	PROVEN	646,016	0.88	18,363
	PROBABLE	3,134,066	0.91	91,758
	TOTAL	3,780,082	0.91	110,122
GRAND TOTAL		20,607,332	0.92	608,373

HEAP LEACH STOCKPILE	PROVEN	-	-	-
	PROBABLE	1,275,233	0.90	36,900
	TOTAL	**1,275,233**	**0.90**	**36,900**

*Note:
1. The Mineral Reserve estimates were prepared in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by the CIM Council on May 10, 2014, and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, adopted by CIM Council on November 29, 2019, using economic and mining parameters appropriate to the deposit.
2. The Mineral Reserve Estimate is based on an updated optimized shell using 1,500 $/oz gold price, average dilution of 20% , mining recovery of 100% and break-even cut off grades of 0.29 g/t Au for Paiol, 0.31 g/t Au for Vira Saia and 0.34 g/t Au for Cata Funda.
3. Contained metal figures may not add due to rounding.
4. Surface topography based on December 31st, 2016.
5. Mineral Reserve estimate for Almas Gold Project was prepared under the supervision of Luiz Pignatari, P.Eng. as a Qualified Person, competent to sign as defined by NI 43-101.

Figure 1-2 Paiol, Cata Funda and Vira Saia pits

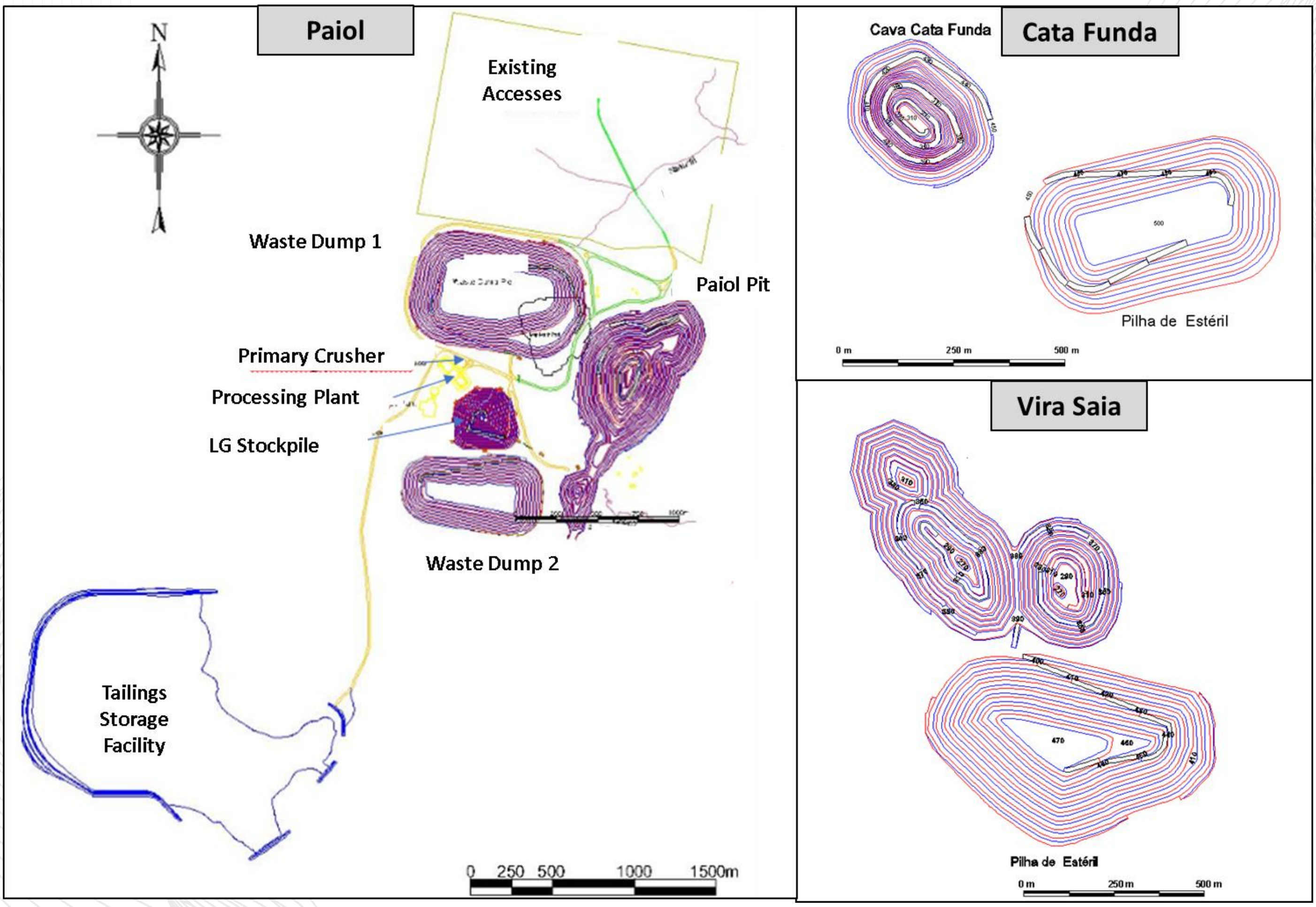


Table 1-3 Almas Gold Project Mineral Reserves Estimation Parameters

INPUT PARAMETER	UNITS	VALUE		
Mine	#	Paiol	Cata Funda	Vira Saia
Ore Mining Costs	USD/t	2.00	4.00	2.60
Waste Mining Costs	USD/t	1.80	1.80	1.80
Dilution	%	(*)	20%	(*)
Low-Grade Cut-Off Grade	g/t	0.29	0.34	0.31
High-Grade Cut-Off Grade	g/t	0.60	0.60	0.60
Pit Wall Overall angle (Assumed)	(°)	50	50	50
Mine deepening ratio limit by year	m	40	40	40
Mining Recovery	%	100%	100%	100%
Concentration Process	USD/t	9.50	9.50	9.50
G & A	USD/t	3.50	3.50	3.50
Total Cost Plant G&A	USD/t	13.00	13.00	13.00
Plant Recovery	%	92.5%	92.5%	92.5%
Gold	USD/oz	1,500.00	1,500.00	1,500.00
Discount Rate	%	5.0	5.0	5.0

(*)The models from Paiol and Vira Saia were re-blocked to an operational dimension block (5x5x5m) and the dilution is already included in the simulations

The economic analysis of the Life of Mine (LOM) plan generates a positive cash flow.

1.8 MINING METHOD

The mining operation concept for the Almas Gold Project is a conventional open pit and is scheduled to start up in July, 2022 ramping up until October, 2022.

The contracted mining fleet involves small backhoe excavators (74-t op. weight) coupled with on-road mining trucks (22 m3 capacity). The materials will be drilled by top-hammer drill rigs in 10-m and 5-m benches.

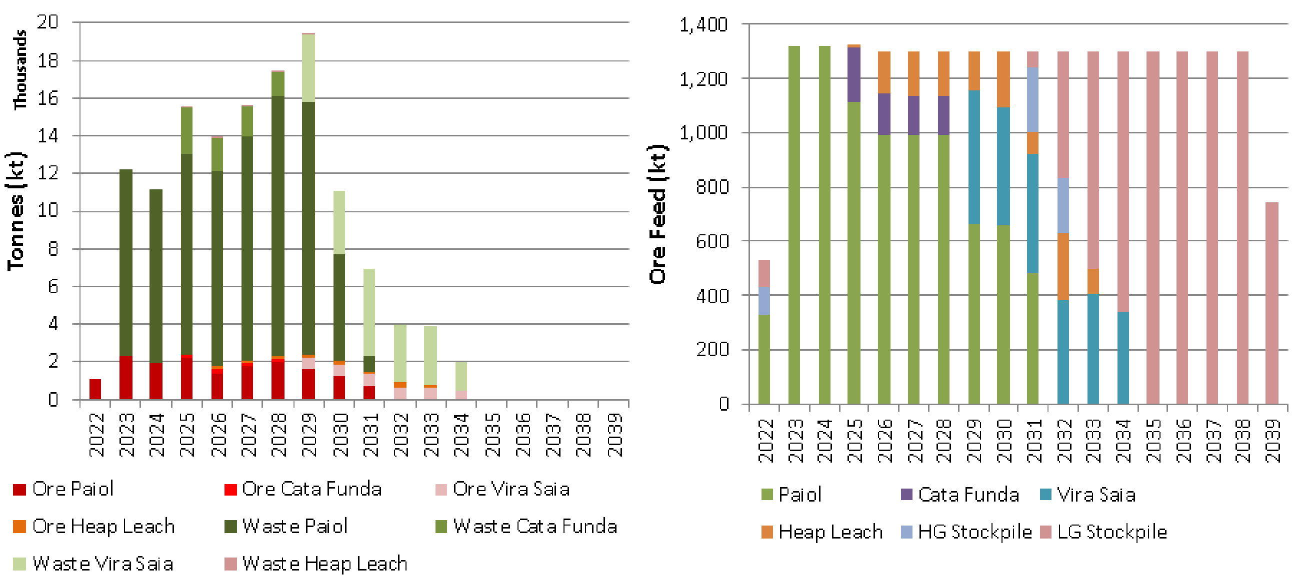
The rock types comprises soil, saprolite, weathered and hard rock. The excavation of these deposits requires the use of drilling and blasting for safe and efficient mining for all material except most of the saprolite, weathered rock and soil which will be direct mined by excavators and track dozers. The powder factor is 410 g/t. Direct mining will be also applied to the heap leach materials.

The ore will be hauled by trucks to a RoM stockpile located near the primary crusher for later re-handling using a front-end loader. No direct feeding is envisaged. The long term blending strategy includes the provision for low and high grade stockpiles throughout the life-of-mine. The waste will be hauled to waste dumps located near the pits.

Grade control with exclusive drilling will provide good support to engineering and short-term mining plans. The technology considered is Down the Hole with reverse circulation. Primary rock blasting will be fragmented by using explosives, and specifically to the ore we are considering the use of electronic caps. Blastholes are going to be drilled by a hydraulic Top Hammer drilling rig. Rock mechanical excavation: must be made by bulldozers or directly by hydraulic excavators. Loading operations will be done, preferentially, by retro bucket profile hydraulic excavator, and complemented by front end loaders (FEL). Rock transport will be done by vocational trucks.

The mine schedule achieved a production target of 1.3 Mtpa with a maximum annual rock movement of 19.4 Mtpa (Figure 1-3). A variable cut-off grade strategy was implemented thereby the high grades are mined in the early periods while leaving the low grades for the end of the mining sequence. The LOM sequence encompasses a 9-month pre-stripping phase at Paiol followed by 13 years of primary mining and, finally, 5 years of re-handling the low grade ore.

Figure 1-3 Mine scheduling



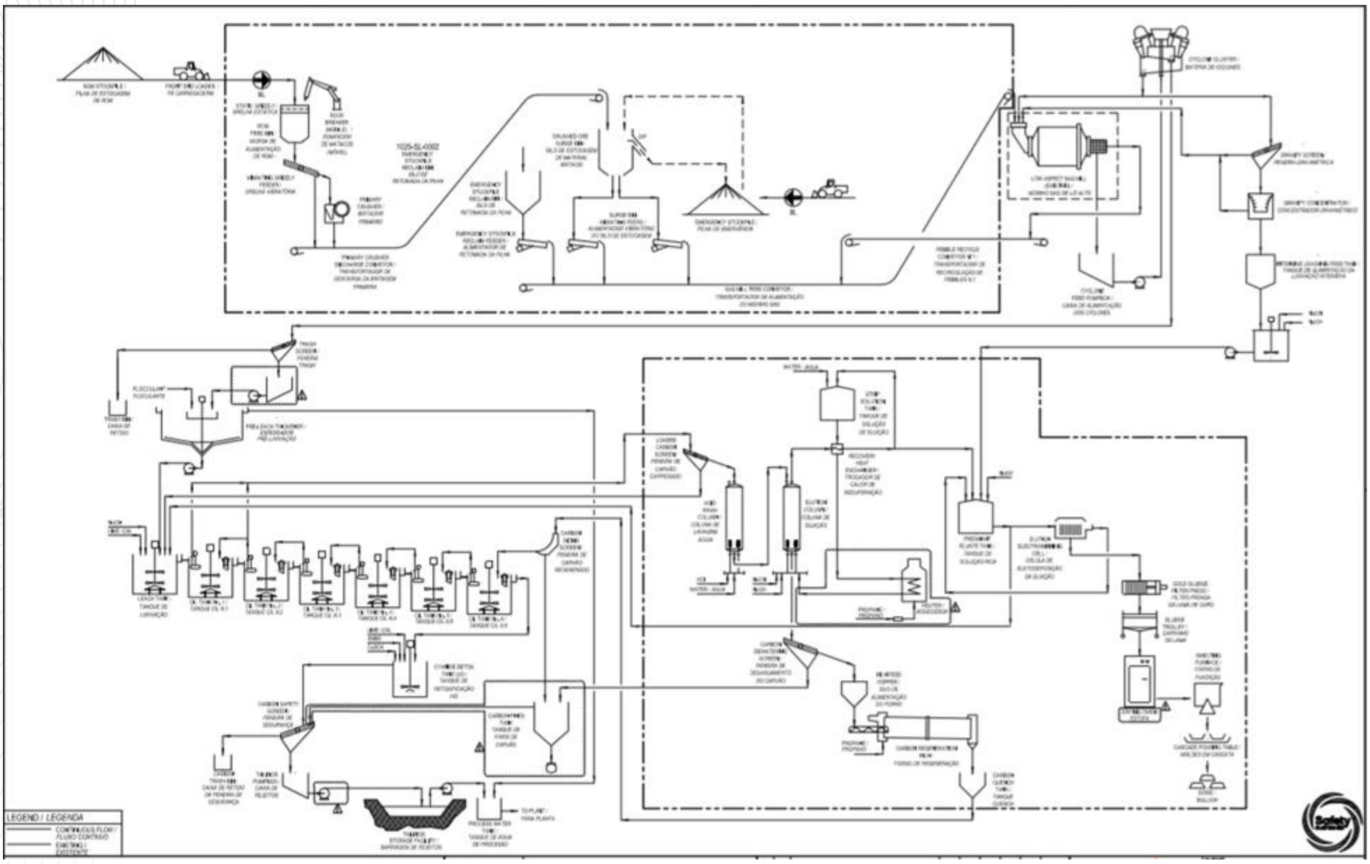
1.9 RECOVERY METHODS

The process plant was designed using conventional processing unit operations. It will treat 3,560 t/d or 1.3 Mt/y based on an availability of 8,059 hours per annum or 92%. The crushing section design is set at 70% availability and the gold room availability is set at 52 weeks per year including two operating days and one smelting day per week. The plant will operate with two shifts per day, 365 days per year, and will produce doré bars.

The plant feed will be hauled from the open pits or stockpiles to a mobile crushing facility that will include a ROM bin and jaw crusher. Crushed ore is conveyed to a surge bin that provides 3 hours retention time. The crushed ore will be ground by a single stage SAG mill in closed circuit with a hydrocyclone cluster. A portion of the hydrocyclone underflow is directed to a gravity concentration circuit for removal of fee gold. The hydrocyclone overflow with an 80% passing size of 75 µm will flow over a trash screen and then to the pre-leach thickener. Thickener underflow will go to a leach ‑ carbon-in-leach (CIL) recovery circuit. Gold and silver leached in the CIL circuit will be recovered onto activated carbon and eluted in a pressurized AARL-style elution circuit and then recovered by electrowinning in the gold room. Gravity concentrate is processed in an intensive leach reactor and the dissolved gold is recovered in a dedicated electrowinning cell in the gold room. The combined gold ‑ silver electrowinning sludge will be dried and then mixed with fluxes and smelted in a furnace to pour doré bars. Carbon will be re-activated in a carbon regeneration kiln before being returned to the CIL circuit. CIL tails will be treated for cyanide destruction using the SO_2/air process prior to pumping to the tailings storage facility (TSF) for disposal.

The simplified Almas Gold Project flowsheet is shown in Figure 1-4.

Figure 1-4 Simplified Almas Gold Project Flowsheet



The installed power for the process plant will be 6,931 kW and the power consumption is estimated to be 28.3 kWh/t processed. Raw water will be pumped from the Manuel Alves River to a raw-water storage tank. Potable water will be sourced from the raw water tank and treated in a potable water treatment plant. Gland water will be supplied from the raw-water tank. Process water will primarily consist of TSF reclaim water. Reagents will include lime, sodium cyanide, sodium hydroxide, copper sulfate, hydrochloric acid and sodium metabisulfite.

1.10 PROJECT INFRASTRUCTURE

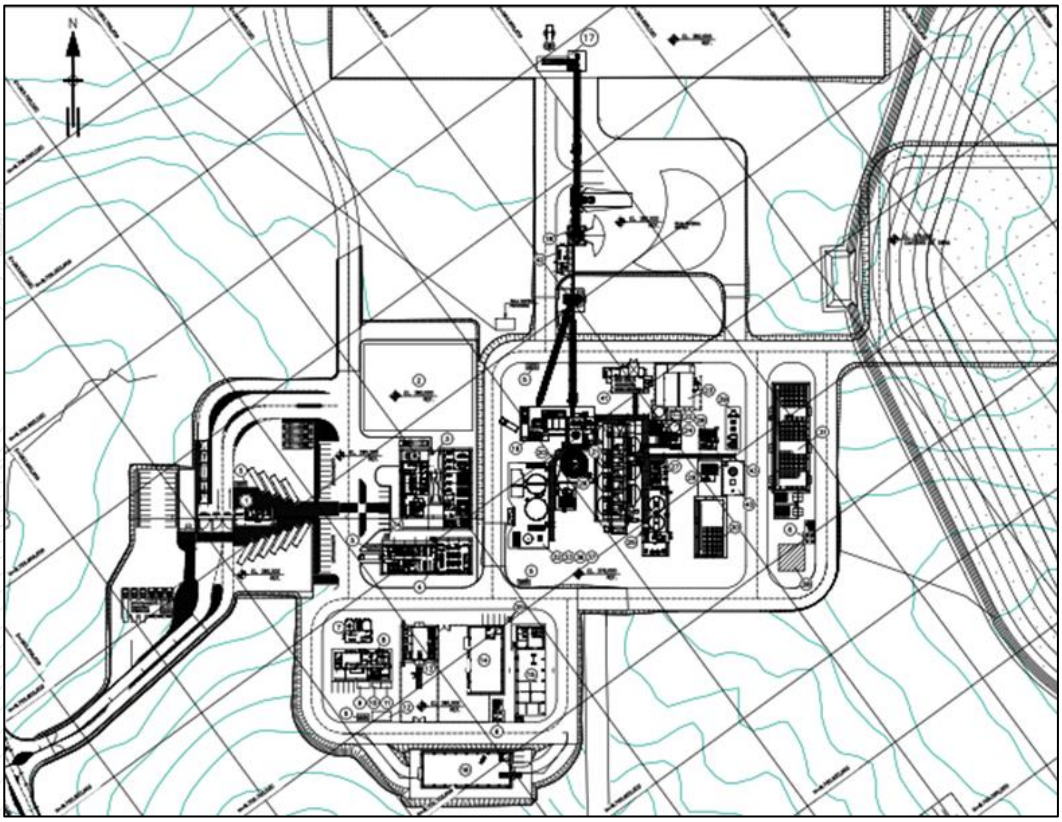
The overall site plan is shown in Figure 1-5. The major project facilities include the open pit mines, tailings management facility (TMF), waste rock facilities, mine services and access roads. Access to the facility is from the west side of the property from the existing access road. Main access will be via the security gate near the process plant.

The site will be fenced to deter access by unauthorized people. The process plant is located west of the Paiol deposit, with the TMF to the southwest.

Site selection took into consideration the following factors:

- locate the major process equipment foundations on competent bedrock and utilize rock anchors for foundations design
- upgrade and utilize the existing access road to reach the site
- locate mining, administration and processing plant staff offices close together to limit walking distances between them
- locate the ready line close to the mining admin/office area and change room facilities

Figure 1-5 Almas Project Site Plan



1.11 MARKET STUDIES AND CONTRACTS

The World Bank forecast indicates an increase in the average price of gold to US$/oz 1,775 in 2020 from an average of US$/oz 1,392 in 2019. In the next ten years, the gold price is expected to reach around US$/oz 1,400 in 2030. In the first month of 2020, gold price averaged US$/oz 1,560, which was about 6% higher than December 2019. Throughout the year 2020 the spot price of gold reached approximately US$/oz 2,000, 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 purposes all over 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. 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.

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.

1.12 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The historical pit at Paiol is currently flooded with water. The existing waste dump and heap leach facilities have been reclaimed. Most of the process equipment and other facilities have been removed in conformance with the reclamation requirements of the State's environmental authority.

After the decommissioning of the Paiol Mine in the 2000s, the mining rights of the project were acquired by Rio Novo Mineração in 2010, which conducted a number of feasibility studies, engineering design and environmental and social studies for the resumption of operation.

Most of the environmental and social studies were carried out between 2010 and 2012, among them the Environmental Assessment (EA) required for the simplified permitting of the Paiol Mine, according to the instructions of the Instituto Natureza do Tocantins – NATURATINS, which is the state's regulatory environmental authority. Although the EA has contemplated the socioeconomic aspects of the Vira-Saia and Cata Funda deposits, it is worth noting that the study was carried out exclusively for the permitting of the Paiol Mine. The EA was conducted and prepared by the Consultancy firm Conestoga-Rovers & Associados (CRA) from São Paulo, in 2011.

From this permitting, the Paiol Mine obtained the Installation License No. 5437/2011 (Licença de Instalação –LI), which has already expired, and subsequently the Preliminary License No. 286/2017 (Licença Prévia –LP) and Installation License No. 297/2017 (Licença de Instalação – LI), which is undergoing analysis for renewal by the technical staff of NATURATINS.

Currently, as part of the resumption of the Almas Gold Project by Aura Minerals, additional studies, including, but not limited to, Geochemistry Tests, Water Quality Characterization, Forest Inventory, Detox Tests, Updated Plan for Monitoring and Rescue of Fish and Wildlife are being carried out to support both the renewal of the Installation License as well as other required Permits to complete the Paiol Mine permitting process.

For the permitting of the mineral deposits Vira-Saia and Cata Funda, another Environmental Assessment (EA) was recommended by NATURATINS, since Illegal artisanal mining ("Garimpo") has already degraded the areas over the years and the potential for negative impacts is low. It is estimated that this study will start in early 2021. The protocol and application for the Preliminary and Installation licenses with NATURATINS is expected in the 4th quarter of 2021, and the forecast for obtaining Licenses is between the 3rd and 4th quarter of 2022.

The estimated reclamation and mine closure costs for the Almas Gold Project, encompassing the 3 mineral deposits, is US $ 5.5 million.

1.13 CAPITAL AND OPERATING COSTS

The estimate conforms to Class 3 guidelines for a feasibility study level estimate with accuracy according to the Association for the Advancement of Cost Engineering International (AACE International). The estimate includes the cost to complete the design, procurement, construction and commissioning of all the identified facilities. The estimate was based on the traditional engineering, procurement and construction management (EPCM) approach where the EPCM contractor would oversee the delivery of the completed project from detailed engineering and procurement to handover of a working facility.

The estimate was derived from a number of fundamental assumptions as indicated in process flow diagrams, general arrangements, mechanical equipment list, electrical equipment list, material take offs (MTOs), cable schedules, scope definition and a work breakdown structure. The estimate included all associated infrastructure as defined by the scope of work.

The initial capital cost estimate is summarized in Table 1-4.

Table 1-4 Initial Capital Cost Estimate Summary (direct and indirect)

COST TYPE	DESCRIPTION	US$ M
Direct	Mine	4.0
	Construction and Erection	18.6
	Mineral processing	25.1
	Tailings facility	2.4
	Power line	1.8
	Laboratory	0.6
	Direct Subtotal	*52.6*
Indirect	Indirect	8.1
	Contingency	4.6
	Owners Costs	7.4
	Indirect Subtotal	*20.2*
Project Total – Initial Capital		***72.8***

The operating cost estimate is presented in Table 1-5 and is based on Q4 2020 United States dollars (USD). The estimate includes mining, processing, general and administration (G&A), and accommodations costs.

Table 1-5 Almas Operating Costs

DESCRIPTION	YEARLY COST (M$USD)	YEARLY COST (USD/t)
Mining	8.86	7.31
Process	14.88	11.44
General and Administration	2.50	1.93
Total	26.24	20.68

1.14 ECONOMIC ANALYSIS

A full financial model was prepared for the Almas Project including capital costs, operating expenditures and production schedule with inputs provided by Aura, Ernst & Young (EY), Ausenco and EDEM.

Based on the assumptions adopted, the post-tax net present value ("NPV") of Aura Minerals Gold Almas Project base case achieved US$183M (R$660M) at 5% discount rate. The internal rate of return ("IRR") reached 43.9% and the annual average EBITDA (from year 1 to year 16, full run rate production period) is US$27M. Payback after the start-up of operations is 2.1 years.

The results of the financial model are summarized in the Table 1-6, the project cash flow is presented in the Table 1-7 and an operating income statement in the Table 1-8.

Table 1-6 Financial Results Summary (Post tax)

ITEM	UNIT	VALUE
DISCOUNT RATE (WACC)	%	5.0%
NET PRESENT VALUE – NPV	US$M	182.7
CAPEX NPV	US$M	(87.8)
Operational NPV	US$M	270.5
PROJECT IRR	%	43.9%
PROJECT PROFITABILITY INDEX		3.1
DISCOUNTED PROJECT PAYBACK	Years	3.7
SIMPLE PAYBACK (including start-up)	Years	3.5
SIMPLE PAYBACK (after start-up)	Years	2.0

*Table 1-7 Project Cash Flow (US $ *1,000)*

DESCRIPTION	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039
EBIT	0	15,757	41,960	42,338	32,616	26,151	20,829	16,830	20,948	25,811	26,414	20,597	18,906	14,385	9,173	9,643	9,101	9,751	5,423
(+) Depreciation	0	3,639	7,277	7,701	7,848	8,405	8,552	9,346	9,090	9,109	9,189	5,570	1,774	1,774	2,244	1,774	2,317	1,808	1,769
(=) EBITDA	0	19,395	49,237	50,039	40,464	34,555	29,381	26,176	30,038	34,920	35,603	26,167	20,680	16,159	11,418	11,418	11,418	11,559	7,192
(-) CAPEX	(36,099)	(36,675)	(2,116)	(737)	(2,784)	(737)	(3,969)	(834)	(834)	(3,185)	(834)	(3,185)	(834)	(3,185)	(834)	(3,546)	(640)	(640)	(192)
(+-) Working Capital Variation	0	(2,718)	(2,223)	102	(406)	73	(162)	(240)	(250)	972	527	529	80	277	341	7	(8)	3	3,097
(-) Mine Closure Cost	0	0	0	0	0	0	0	0	0	0	0	0	0	0	(1,100)	(1,100)	(1,100)	(1,100)	(1,100)
(-) Income Tax / Social Contribution	0	(5,353)	(6,394)	(6,452)	(4,969)	(3,983)	(3,172)	(2,562)	(3,190)	(3,932)	(4,023)	(3,136)	(6,423)	(4,886)	(3,114)	(3,274)	(3,090)	(3,311)	(1,839)
(=) Free Cash Flow to Firm (FCFF)	**(36,099)**	**(25,350)**	**38,504**	**42,951**	**32,304**	**29,908**	**22,079**	**22,540**	**25,764**	**28,776**	**31,272**	**20,375**	**13,503**	**8,365**	**6,710**	**3,504**	**6,580**	**6,510**	**7,158**
(=) Accumulated Free Cash Flow to Firm	**(36,099)**	**(61,450)**	**(22,945)**	**20,006**	**52,310**	**82,218**	**104,296**	**126,836**	**152,600**	**181,376**	**212,649**	**233,024**	**246,527**	**254,892**	**261,601**	**265,105**	**271,685**	**278,196**	**285,354**

Table 1-8 Operating Income Statement (US $ x 1,000)

DESCRIPTION	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039
Gross Operating Revenue	0	31,706	83,476	83,192	77,851	71,558	68,249	67,450	73,572	67,061	61,675	47,502	42,068	34,906	27,014	27,014	27,014	27,105	13,494
Deductions from Operating Revenue	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Net Operating Revenue	0	31,706	83,476	83,192	77,851	71,558	68,249	67,450	73,572	67,061	61,675	47,502	42,068	34,906	27,014	27,014	27,014	27,105	13,494
Cash Cost	0	(9,206)	(27,104)	(26,032)	(30,651)	(30,522)	(32,526)	(34,964)	(36,923)	(26,890)	(21,642)	(17,967)	(18,273)	(16,102)	(13,356)	(13,356)	(13,356)	(13,356)	(5,265)
Freight / Refining	0	(129)	(340)	(339)	(317)	(291)	(278)	(274)	(299)	(273)	(251)	(193)	(171)	(142)	(110)	(110)	(110)	(110)	(55)
Depreciation and Exhaustion	0	(3,639)	(7,277)	(7,701)	(7,848)	(8,405)	(8,552)	(9,346)	(9,090)	(9,109)	(9,189)	(5,570)	(1,774)	(1,774)	(2,244)	(1,774)	(2,317)	(1,808)	(1,769)
Gross Profit	0	18,733	48,754	49,121	39,035	32,340	26,894	22,865	27,260	30,789	30,592	23,772	21,849	16,887	11,303	11,773	11,231	11,831	6,405
Gross margin (without depreciation)	0.0%	59.1%	58.4%	59.0%	50.1%	45.2%	39.4%	33.9%	37.1%	45.9%	49.6%	50.0%	51.9%	48.4%	41.8%	43.6%	41.6%	43.6%	47.5%
SG&A	0	(1,644)	(3,289)	(3,289)	(3,289)	(3,289)	(3,289)	(3,289)	(3,289)	(2,209)	(1,631)	(1,217)	(1,217)	(1,066)	(996)	(996)	(996)	(996)	(415)
SG&A - Depreciation	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
SG & A / Net Revenue	0.0%	5.2%	3.9%	4.0%	4.2%	4.6%	4.8%	4.9%	4.5%	3.3%	2.6%	2.6%	2.9%	3.1%	3.7%	3.7%	3.7%	3.7%	3.1%
CFEM	0	(476)	(1,252)	(1,248)	(1,168)	(1,073)	(1,024)	(1,012)	(1,104)	(1,006)	(925)	(713)	(631)	(524)	(405)	(405)	(405)	(407)	(202)
Royalties	0	(856)	(2,254)	(2,246)	(1,963)	(1,827)	(1,752)	(1,734)	(1,920)	(1,763)	(1,623)	(1,245)	(1,095)	(913)	(729)	(729)	(729)	(678)	(364)
Income before Income Tax / Social Contribution	0	15,757	41,960	42,338	32,616	26,151	20,829	16,830	20,948	25,811	26,414	20,597	18,906	14,385	9,173	9,643	9,101	9,751	5,423

DESCRIPTION	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039
Income Tax	0	(2,364)	(6,294)	(6,351)	(4,892)	(3,923)	(3,124)	(2,525)	(3,142)	(3,872)	(3,962)	(3,090)	(2,836)	(2,158)	(1,376)	(1,446)	(1,365)	(1,463)	(813)
Income Tax (over R$ 60 thousand in the quarter)	0	(1,571)	(4,191)	(4,229)	(3,257)	(2,610)	(2,078)	(1,678)	(2,090)	(2,576)	(2,637)	(2,055)	(1,886)	(1,434)	(913)	(960)	(905)	(970)	(538)
Income Tax - Benefit	0	0	7,867	7,938	6,115	4,903	3,905	3,156	3,928	4,840	4,953	3,862	0	0	0	0	0	0	0
Social Contribution	0	(1,418)	(3,776)	(3,810)	(2,935)	(2,354)	(1,875)	(1,515)	(1,885)	(2,323)	(2,377)	(1,854)	(1,701)	(1,295)	(826)	(868)	(819)	(878)	(488)
Net Income	0	10,404	35,565	35,886	27,647	22,167	17,657	14,268	17,758	21,879	22,390	17,461	12,482	9,499	6,059	6,369	6,011	6,440	3,584
Net Margin	0.0%	32.8%	42.6%	43.1%	35.5%	31.0%	25.9%	21.2%	24.1%	32.6%	36.3%	36.8%	29.7%	27.2%	22.4%	23.6%	22.3%	23.8%	26.6%
EBITDA	0	19,395	49,237	50,039	40,464	34,555	29,381	26,176	30,038	34,920	35,603	26,167	20,680	16,159	11,418	11,418	11,418	11,559	7,192
EBITDA margin	0.0%	61.2%	59.0%	60.1%	52.0%	48.3%	43.0%	38.8%	40.8%	52.1%	57.7%	55.1%	49.2%	46.3%	42.3%	42.3%	42.3%	42.6%	53.3%

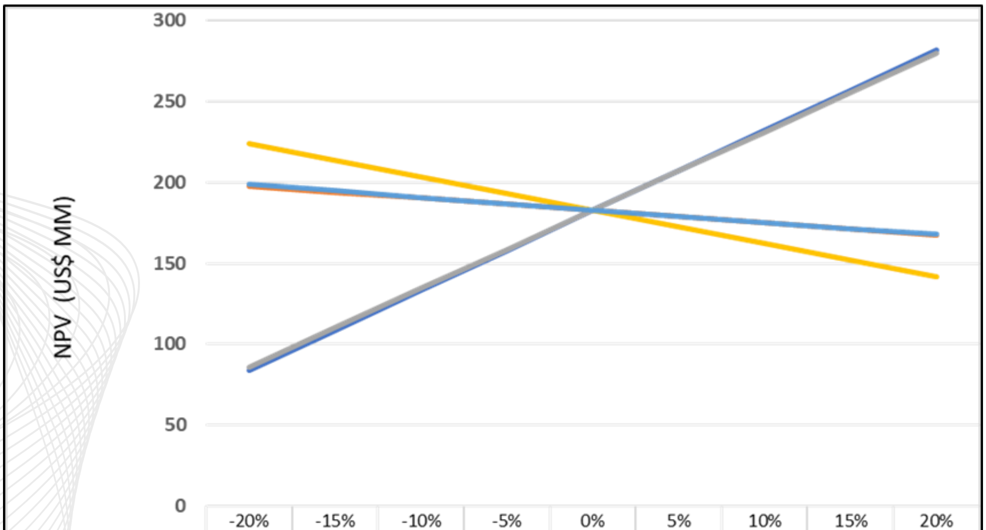
The sensitivity analysis shows the impact of the variation of the gold price, exchange rates, operating and capital costs upon the Project NPV and IRR. The analysis encompasses the following range of variation in the key inputs:

- Gold price: ±20%.
- CAPEX: ±20%.
- Exchange Rate: ±20%.
- Cash Cost: ±20%.
- Discount Rate: ±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.

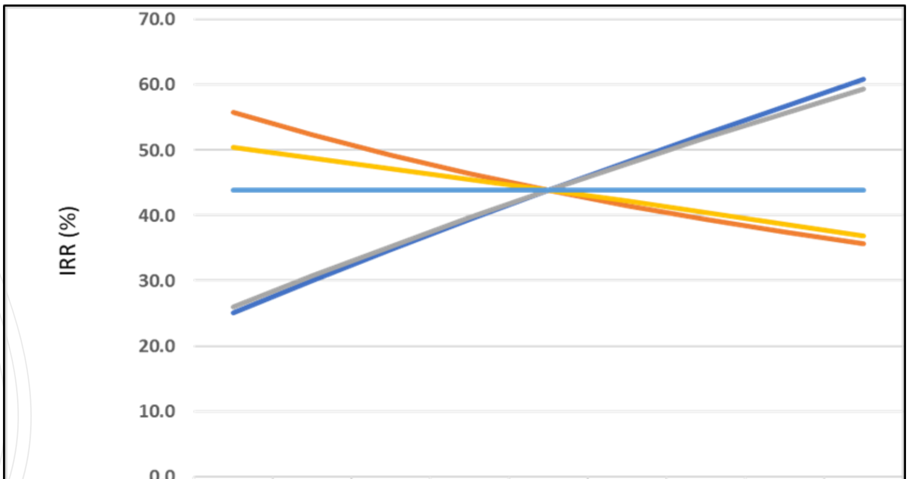
Figure 1-6 illustrates the results of the sensitivity analysis for Project NPV (after tax) and these effects for each of the critical variables and Figure 1-7 presents the same scenario for the IRR. NPV results are reported at a discount rate of 5.0%

Figure 1-6 Sensitivity Analysis Graph – NPV



	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%
Gold Price	84	108	133	158	183	208	232	257	282
CAPEX	198	194	190	187	183	179	175	171	168
Exchange Rate	86	110	134	159	183	207	231	255	280
Cash Cost	224	214	203	193	183	172	162	152	141
Discount Rate	199	195	191	187	183	179	175	171	168

Figure 1-7 Sensitivity Analysis Graph – IRR



		-20%	-15%	-10%	-5%	0%	5%	10%	15%
Gold Price	25.1	30.1	34.8	39.4	43.9	48.2	52.5	56.7	60.8
CAPEX	55.8	52.3	49.2	46.4	43.9	41.5	39.4	37.4	35.6
Exchange Rate	25.9	30.7	35.3	39.7	43.9	47.9	51.9	55.7	59.4
Cash Cost	50.4	48.8	47.2	45.5	43.9	42.2	40.4	38.7	36.9
Discount Rate	43.9	43.9	43.9	43.9	43.9	43.9	43.9	43.9	43.9

GE21 understands that the Almas Gold Project is economically viable and attractive based on these results.

The main risks associated with the economic model results are:

- Financial risk – price: There is a low risk regarding the price used reflected by the current consensus price applied to the project. Exchange rates can affect the ratio of Price/Cost as well;
- Financial risk – fiscal benefits: There is a low risk regarding the fiscal benefits applied to the project, since not all of them have been granted yet.

1.15 CONCLUSION

At the conclusion of this Feasibility Study on the Almas Gold Project it was demonstrated that at a gold price of $1,588/oz, an investment of US$73M would be required, principally to build a processing plant and associated facilities, to treat ores from 3 open pits at a rate of 1.3Mt/yr, which over a mine life of 13 years would yield a return on investment of 44%. At a discounted rate of 5% the "all equity" Net Present Value after taxation is $183M. Average annual gold production is expected to be 35,560 oz. The break even all in cash (AISC) cost has been calculated to be $828/oz.

1.16 RECOMMENDATIONS

- Additional infill drilling is required to convert more resources both in Paiol and Vira Saia from inferred to M&I categories. Multiple narrow shear zones can be identified in HW and FW for all three deposits. Additional infill drilling can delineate and test the continuity of these splay shear structures and related ore shoots.
- At Paiol, the deposit narrows down toward the south but is open towards north with multiple shearing targets. Additional infill drilling will probably delineate more ounces which are not modelled and estimated in the current feasibility study. The northern part of deposit had more ounces due to the presence of high grade where there is a chance of finding more mineralized zones.
- Existing lithological and alteration databases need to be reviewed and revised, and refined lithological models need to be established. Future resource estimates need to consider these updated litho-alteration models for all three deposits.
- A maximum two pits operating simultaneously is recommended, otherwise the operation will become more complex and certainly costs would increased.

- A single low grade ore stockpile close to processing plant is considered a better strategy from a logistic viewpoint and simplifies the low- rade ore pile re-handled after 2031.
- While the geothecnical information at this stage is considered sufficient to start the operation, it is important to start the operation with an experienced dedicated geotecnical team to assure a good monitoring geotechnical program to give good support to the operation and eventually revise the Paiol pit design accordingly, after third and fourth year operation.
- EDEM recommends grade control drilling with Down-The-Hole reverse circulation drills to support the grade control engineering.
- Additional testwork should be considered to define the geometallurgical sample variability in more detail.
- Additional leach testing is recommended to optimize leach conditions and cyanide consumption.   Additional continuous cyanide detoxification tests are recommended to optimize retention time and reagent additional rates.
- Continuous monitoring of the renewal schedule for the Installation License- LI for the Paiol Mine, and other permits with the environmental regulatory body - NATURATINS so there are no delays in the issuance of the environmental permits;
- Priority is given to programs that present a social scope, such as Updating the Social Diagnosis, Mapping Stakeholders, Social Management Plan, Social Communication Program and Defining Partnerships with the communities affected by the Almas Gold Project.

2 INTRODUCTION AND TERMS OF REFERENCE

The Almas Gold Project was acquired by Aura from its previous owner, Rio Novo Mineracao Ltda., in 2018. The previous feasibility study reports were reviewed by Aura and some deficiencies identified . There are also changes in exchange rates, gold price and the costs of goods and services that needed to be implemented in the new report.

Aura, in collaboration with Micon, EDEM, Ausenco, GE21 and a few independent consultants prepared an NI 43-101-compliant Technical Report. This report is a new and updated report for Almas Gold Project considering all required changes in technical information and reflecting the new 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.

The Almas Gold Project is in the municipality of Almas, in Tocantins State, Brazil. The Project consists of three separate mining areas envisioned to be mined and the ore transported to a central processing facility. Exploration and development programs have produced an extensive database of information which forms the foundation for this report.

2.1 PROJECT BACKGROUND

The Almas Gold Project includes three main gold deposits: Paiol, Cata Funda and Vira Saia. The three deposits are located along a 15-km long corridor of the Almas Greenstone Belt, a Paleoproterozoic volcano-sedimentary sequence which hosts numerous orogenic gold occurrences. Several other exploration targets occur in the vicinity but are not the subject of this report.

The Project includes a former historic open pit and a spent heap leach stockpile at Paiol. The project was formerly operated by VALE from 1996 until 2001 and produced 86,000 ounces of gold. The former open pit is currently filled with water and the waste dump and heap leach facilities have been reclaimed. Most of the process equipment and other facilities have been removed in conformance with the reclamation requirements of the state environmental authority.

In January 2013, both Paiol and Cata Funda received approval from ANM authorizing renewal of mining activities. Previously they had status of “Suspended Operation” with the ANM. The process is well documented by ANM and is defined as a request to actively mine again (Requerimento para Retomada de Lavra) under Section 20.2.3 of the Regulatory Norms for Mining (Normas Reguladoras de Mineração “NRM” Suspensão, Fechamento de Mina e Retomada das Operações Mineiras).

To operate the new project at the Paiol mine, Rio Novo was required to obtain a new environmental license under the standards set forth by the Tocantins State environmental authority (NATURATINS). The Environmental Authority (NATURATINS) has accepted the Environmental Assessment Report (EA) for the Paiol mine area and granted the Installation License No. 5437/2011 (Licença de Instalação or LI) on December 2, 2011, which has already expired. Based on this permitting process other Licenses were issued in 2017 as Preliminary License No. 286/2017 (Licença Prévia - LP) and Installation License No. 297/2017 (Licença de Instalação - LI). The renewal of Installation License - LI was required on December 14, 2018 and is undergoing analyses by the technical staff of NATURATINS. It is expected that Aura will get the renewal in the first quarter of 2021.

The Vira Saia deposit is held by two exploration licenses (Process No. 864.083/2006, 860.373/1988) that were assigned to Rio Novo per the terms of an Option Agreement. Two critical steps in the process of granting a Mining Decree to an operator is the acceptance by the ANM of the operator’s Final Exploration Report and the PEA Report.

For the permitting of mineral deposits of Cata Funda and Vira-Saia, another Environmental Assessment (EA) was required by NATURATINS, since Illegal artisanal mining (“Garimpo”) has already degraded the areas over the years and the potential for negative impacts is low. It is estimated that this study will start in early 2021, the protocol and application for the Preliminary and Installation licenses with NATURATINS in the 4th quarter of 2021, and the forecast for obtaining Licenses between the 3rd and 4th quarter of 2022.

The project development concept currently being considered begins with the construction of a CIL mill facility at Paiol. The Paiol open pit will be prepared for production during the construction period, produce for approximately three years followed by development and production from Cata Funda and Vira Saia. Mined material will be trucked from the satellite deposits to Paiol to maintain the annual plan.

In order to achieve the project development above, the following activities and project developments were completed by Aura between from 2018 to 2020:

- Resource estimation updates for Paiol, Cata Funda ,Vira Saua and the Paiol Heap Leach Pad.
- Two mining studies to support production and LOM.
- The 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 and to develop a process for the recovery of gold;
- Completion of a basic engineering design of the process facilities for the recovery gold, tailings storage, and all support services to maintain a production facility adjacent to the Paiol open pit;
- Estimates of capital and operating expenditures for the project, a discounted cash flow for the life of the project and a plan for the project implementation and the ultimate rehabilitation of the site when the project is decommissioned;
- Detailed studies of the environmental impact of the project with submission to the State environmental authorities for approval and the granting of licenses to operate;
- Community relations programs, stakeholder consultation and sustainable development initiatives.

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 2, 3, 4 , 5, 6, 7, 8 , 9, 10, 14, 23, 24 and summaries there from in sections 1 , 25 and 26.
- Terry Hennessey, (P.Geo), Senior Associate Geologidtwith Micon International (Canada), Member of the Association of Professional Geologists of Ontario, Canada (PGO), is the QP responsible for Sections 11, 12, and summaries there from in Sections 1 and 26.
- Inna Dymov, (P.Eng), Professional Engineer (Ontario), Independent Senior Consultant (Metallurgy), is the QP responsible for Sections 13, and summaries there from in sections 1, 25 and 26.
- Robert Raponi, (P.Eng), Professional Engineer (Ontario), Ausenco Principal Consultant (Metallurgy), is the QP responsible for Sections 17, 18, 21, and summaries there from in sections 1, 25 and 26.
- Luis Pignatari, (P.Eng), Professional Engineer, EDEM Mining Consultants (Engenharia de Minas ME), is the QP responsible for Sections 15, 16, and summaries there from in sections 1, 25 and 26 .
- Adam Wheeler, (C.Eng), Professional Engineer , Adam Wheeler Mining Consultant Limited, is the QP responsible for a portion of section 14 related to the resource estimateat the Cata Funda deposit.
- Porfirio Cabaleiro Rodriguez, (FAIG), Professional Engineer , Fellow of the Austrialian Institute of Geoscientists, GE21 (Consultalria Mineral), is the QP responsible for section of 19 and 22 and summaries there from in section 1.

2.3 QUALIFIED PERSONS SITE VISITS

Mr. Farshid Ghazanfari, QP (Aura ,Geology and Mineral Resources) was involved with Almas Gold Project since 2017 and during the due diligence prior to acquisition . He visited the Almas Gold Project on numerous occasions in the past three years. His last two site visits were in April and May 2019.

Mr. Luis Pignatari, QP (EDEM , Mining) visited the Almas Gold Project between Jun 6 to June 08 of 2017 prior to acquisition to do a review of mine plans for Rio Novo . He has been involved in project occasionally since then in different capacities.

Mr. Terrence Hennessey, P.Geo., QP (Micon, Geology) travelled to Brazil on May 2, 2019 and visited the project sites and warehouse in the town of Palmas on May 3 and 4.

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 (Agencia Nacional de Mineração de Brazil).
- Ausenco refers to Ausenco Mining Consultants (Toronto Office, Canada).
- RPM refers to RungePincockMinarco (RPM), a division of the Runge Corporation.
- Rio Novo refers to Rio Novo Mineracao Ltda.
- Micon refers to Micon International Limited (Toronto Office, Canada).
- EDEM refers to Engenharia de Minas ME (Sao Paulo, Brazil).
- GE21 refers to GE21 Consultoria Mineral Ltda. (Belo Horizonte, Brazil).
- EY refers to Ernst & Young Global Limited.

Aura has based all measurements in the metric system, and has identified exceptions to this, notably when listing both English and Metric standards. Currencies are generally based on the October 22, 2020 US Dollar, with the conversion exchange rate of 5.155 Brazilian Reals per 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).

The following abbreviations are used in this report:

Table 2-1 Units, Symbols and Abbreviations

UNITS, SYMBOLS AND ABBREVIATIONS	
%	Percent(age)
$ / USD / US$	United States Dollars
AA/AAS	Atomic Adsorption Spectroscopy
AARL	Anglo American Research Laboratories
Ai	Abrasion index
AISC	All-In-Sustaining Costs
AIG	Australian Institute of Geoscientists
AFRIMM	Additional of Freight
ANM	National Mining Agency (*Agência Nacional de Mineração*)
AT	Assay Ton

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	Cyanidation
CNWAD	Weak acid dissociable cyanide
CPG	Certified professional geologist
CRM	Certified reference material
CSLL	Social Contribution
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
F80	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 ton
G&A	General and Administrative
GPS	Global positioning system
GRG	Gravity Recoverable Gold
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
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 ton
km	kilometer
kPag	kilopascals, gauge
kV	kilovolts
kW	kilowatts
kWh/t	kilowatt-hour per metric tonne
LI	Installation License

LMC	Linear co-regionalization model
LO	Operating License
LOM	Life of Mine
LP	Preliminary License
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	Meters of Column of Water
mg/L	milligram per liter
Mm	millimeters
Mt or mt	Million tonnes
Mt/a	Million tonnes per annum (year)
mtpy	Million tonnes per year
mv	millivolt
MW	Megawatts
NI 43-101	Canadian National Instrument 43-101
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
P80	Product- 80% passing particle size
PIS and COFINS	Recoverable taxes
ppb	parts per billion

ppm	parts per million
QA/QC	Quality assurance/Quality control
QP	Qualified person
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-bisulphite
SMC test	SAG mill comminution test
SO_2	Sulphur dioxide
SUDAM	Amazon Development Superintendent Agency (*Superintendência de Desenvolvimento da Amazônia*)
T or t	Metric Tonne (1,000 kg or 2,204.6 lbs)
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
TMF	Tailings Management Facility .
toz.	Troy Ounce
Tpa or tpy	Tonnes per annum/year
tph/m²	ton per hour per square meter
TSF	Tailings Storage Facility
TSS	Total Suspended Solids
UTM	Universal Transverse Mercator coordinate system
VAT	Value-added tax
WACC	Weighted Average Cost of Capital

XRF	X-Ray Fluorescence
y	Year
yd^3	cubic yards
µm	micron or micrometer

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

2.5 UNITS

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 %	1000 ppb

3 RELIANCE ON OTHER EXPERTS

This report was prepared by Aura and is based in part on information presented in the 2016 NI 43-101 report under “title of Updated Feasibility Study Technical Report for Almas Gold Project by RPM”, 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, were conducted by persons who are not QPs. Mr. Luiz Pignatari, P.Eng. (QP , Mining) and Mr. Farshid Ghazanfari. P.Geo. (QP for Geology and Resources) have relied on this data, as necessary, to complete this report.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 PROPERTY LOCATION

Rio Novo Mineração Ltda. ("Rio Novo") explored several gold targets in the Almas Greenstone Belt in the south-central region of Tocantins State, Brazil (Figure 4-1). The project area lies south of Almas, a small town approximately 300 km southeast of Palmas, the Tocantins State Capitol and 45 km west of Dianópolis, a regional commercial center. The Almas Gold Project refers to Rio Novo's and Aura's on-going exploration, economic evaluation and planned development by surface mining of gold deposits in the Belt.

This report focuses on the Paiol, Cata Funda and Vira Saia gold deposits that are distributed along a 15 km long segment of the Almas Greenstone Belt, south of the town of Almas. This segment of the belt contains numerous, small scale, artisanal gold mining sites, locally-termed Garimpos, whose development preceded Rio Novo's exploration activities (Figure 4-2). The preponderance of gold deposits of the historical Garimpos are associated with metabasic rocks, including Paiol and Cata Funda whilst the Vira Saia deposit is hosted in mylonitic granodiorite west of the metabasic rocks.

The approximate centers of the three principal deposits in the project area are given below in coordinates referenced to the South American Datum (1969), UTM Zone 23 South – a map projection used throughout this report.

- Paiol 265025.3m East, 8705719.1m North
- Cata Funda 264579.4m East, 8719215.5m North
- Vira Saia 264792.7m East, 8710681.9m North

The two main deposits on the property are; the Paiol deposit, which was previously mined and the undeveloped Cata Funda deposit, situated, respectively, on two inactive Mining Concessions previously assigned to VALE. The Vira Saia deposit is on two exploration permits whose acquisition from a third party was finalized by Rio Novo in 2012. Exploration drilling on all three deposits has been completed. The Cata Funda deposit lies immediately outside of the Almas town site, approximately 15 km north of Paiol, along the regional strike of the Greenstone Belt. The Vira Saia deposit is approximately 5 km northwest of Paiol. Rio Novo has investigated other gold targets in the Greenstone Belt which are at earlier stages of exploration.

The Almas Gold Project includes the historic open pit/heap leach Paiol Project operated by VALE from 1996 until 2001, which produced approximately 86,000 ounces of gold. The former open pit is flooded, and the waste dump and heap leach pad have been reclaimed. Most of the VALE facilities have been removed fulfilling reclamation requirements of the Tocantins state environmental authority (NATURATINS).

Figure 4-1 The Almas Gold Project Location

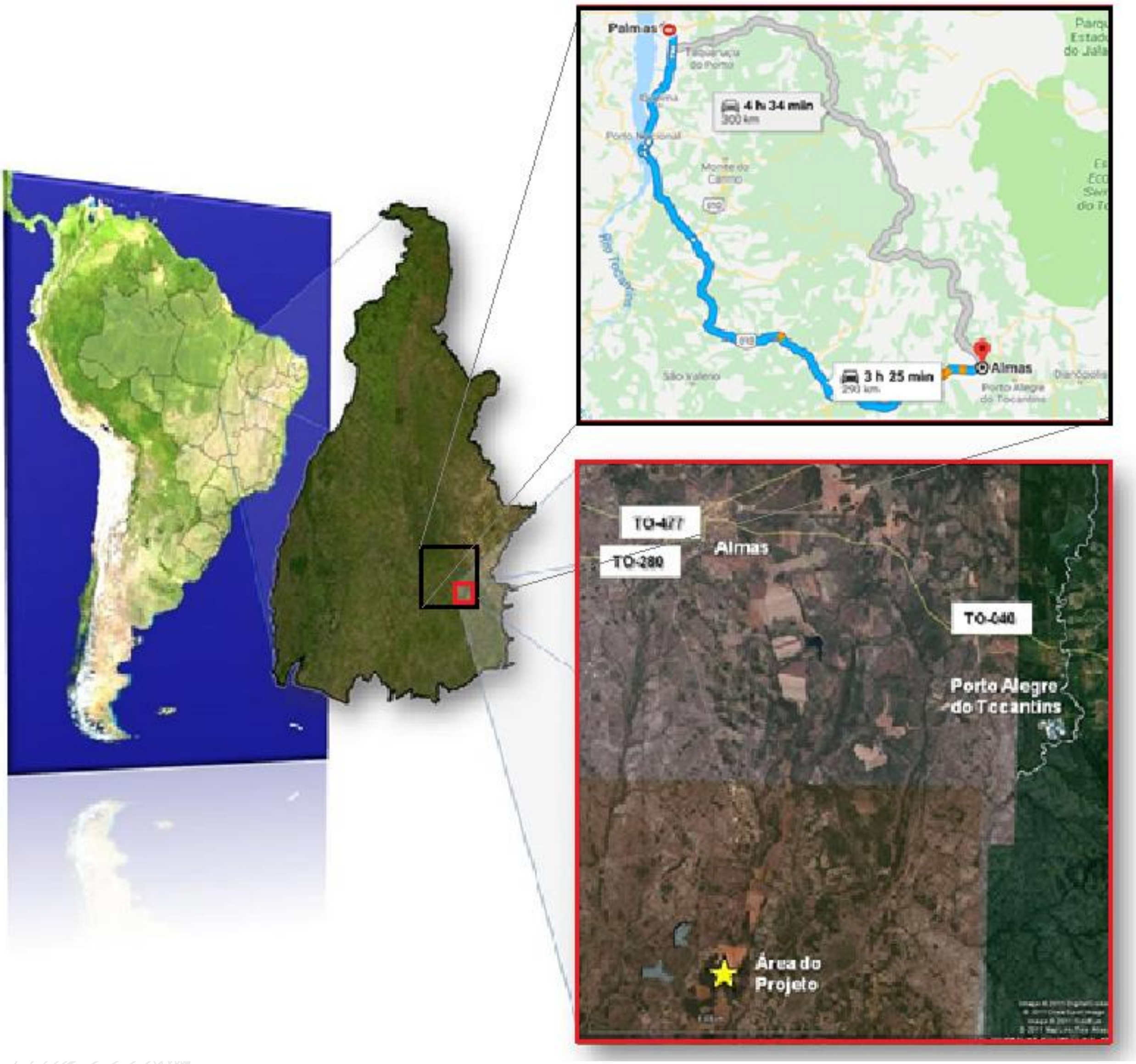
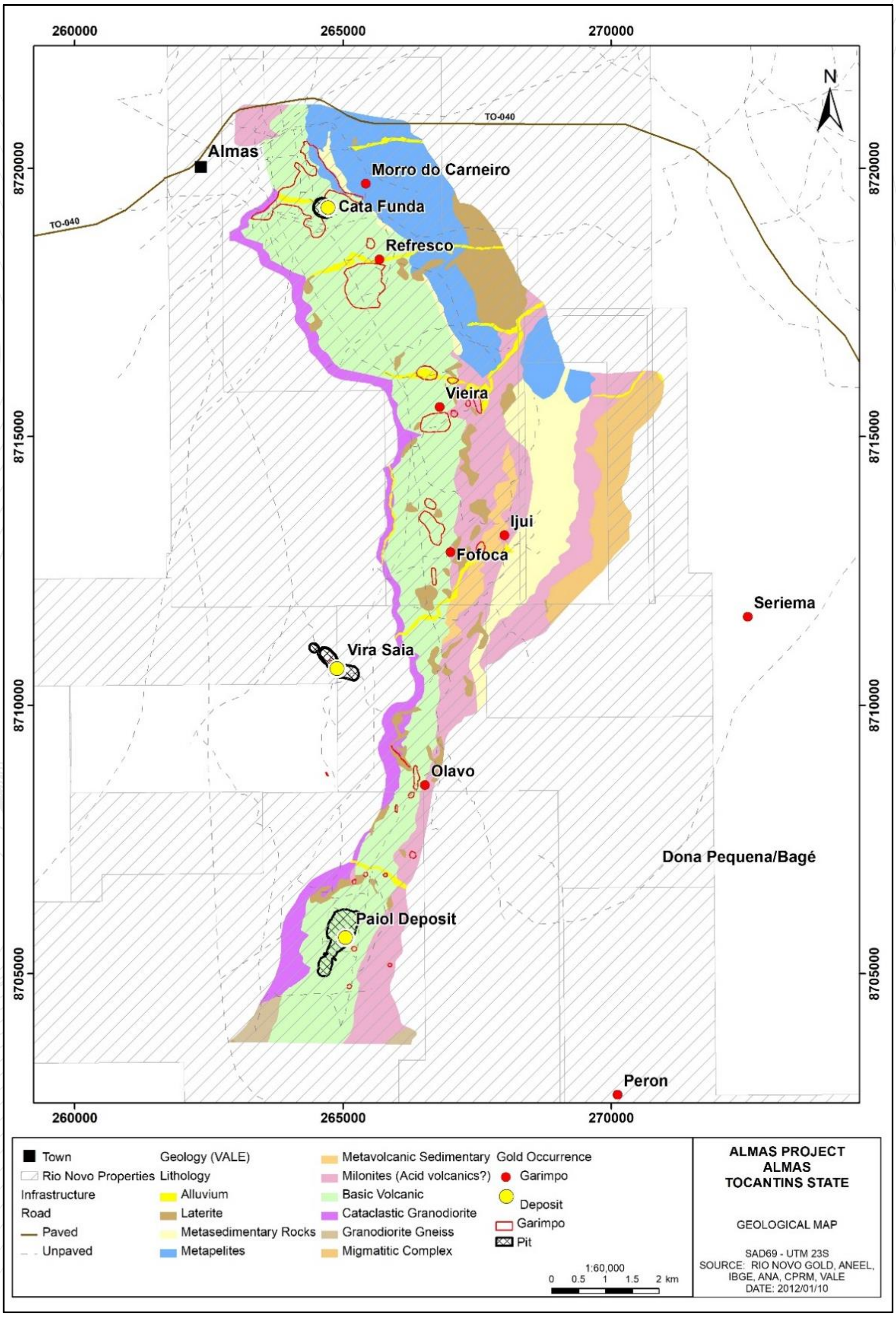


Figure 4-2 Target Locations



4.2 MINERAL RIGHTS

The status of Rio Novo's Exploration Permits ("Autorizações de Pesquisa"), Application Mining Concession ("Requerimento de Concessão de Lavra") and Mining Concessions ("Portarias de Lavra") as of December 07, 2020 are summarized in Table 4-1 and visualized in Figure 4-3 to Figure 4-6.

Table 4-1 Concession and Exploration License Status, December 7, 2020

CONCESSION STATUS - RIO NOVO (Almas) - December 2020				
Nº_ANM	**AREA (HECTARES)**	**STATUS**	**COMMENTS**	**EXPIRATION DATE**
862224/1980	3,962.00	Mining Concession	Start of mining extension requested	Indeterminate
860128/1983	5,175.00	Mining Concession	Start of mining extension requested	Indeterminate
864083/2006	1,759.29	Application for Mining Concession	Application for Mining Concession	Awaiting ANM analysis
860373/1988	2,724.46	Application for Mining Concession	Application for Mining Concession	Awaiting ANM analysis
864613/1994	6,186.80	Exploration Permit	Renewal application on 25-Sep-2018	Awaiting ANM analysis
864415/2011	2,991.38	Exploration Permit	Renewal application on 16-Jul-2018	Awaiting ANM analysis
864417/2011	508.87	Exploration Permit	Renewal application on 16-Jul-2018	Awaiting ANM analysis
864416/2011	1,458.22	Exploration Permit	Renewal application on 16-Jul-2018	Awaiting ANM analysis
864110/2012	4,701.64	Exploration Permit	Renewal application on 14-Aug-2018	Awaiting ANM analysis
864014/2013	7,717.38	Exploration Permit	Renewal application on 16-Aug-2019	Awaiting ANM analysis
864041/2013	8,919.92	Exploration Permit	Renewal application on 16-Aug-2019	Awaiting ANM analysis
864015/2013	6,376.66	Exploration Permit	Renewal application on 16-Aug-2019	Awaiting ANM analysis
864026/2015	8,927.47	Exploration Permit	Renewal application on 25-Sep-2018	Awaiting ANM analysis
864226/2015	4,402.21	Exploration Permit	Renewal application on 22-Apr-2020	Awaiting ANM analysis

CONCESSION STATUS - RIO NOVO (Almas) - December 2020				
Nº_ANM	AREA (HECTARES)	STATUS	COMMENTS	EXPIRATION DATE
864143/2011	7,550.37	Exploration Permit	Renewal Exploration Permit on 18-Oct-2019	18-Oct-2022
864011/2016	361.14	Exploration Permit	Exploration Permit on 07-Feb-2018	07-Feb-2021
864004/2016	630.53	Exploration Permit	Exploration Permit on 07-Feb-2018	07-Feb-2021
864246/2016	5,298.31	Exploration Permit	Exploration Permit on 09-Feb-2018	09-Feb-2021
864027/2017	49.55	Exploration Permit	Exploration Permit on 09-Feb-2018	19-Feb-2021
864002/2018	178.62	Exploration Permit	Exploration Permit on 23-Oct-2018	23-Oct-2021
864005/2018	6,604.67	Exploration Permit	Exploration Permit on 23-Oct-2018	23-Oct-2021
864003/2018	1,700.24	Exploration Permit	Exploration Permit on 23-Oct-2018	23-Oct-2021
864008/2016	445.47	Exploration Permit	Exploration Permit on 07-Feb-2018	07-Feb-2021
864004/2018	6,784.71	Exploration Permit	Exploration Permit on 23-Oct-2018	23-Oct-2021
864019/2016	6,691.32	Exploration Permit	Exploration Permit on 07-Feb-2018	07-Feb-2021

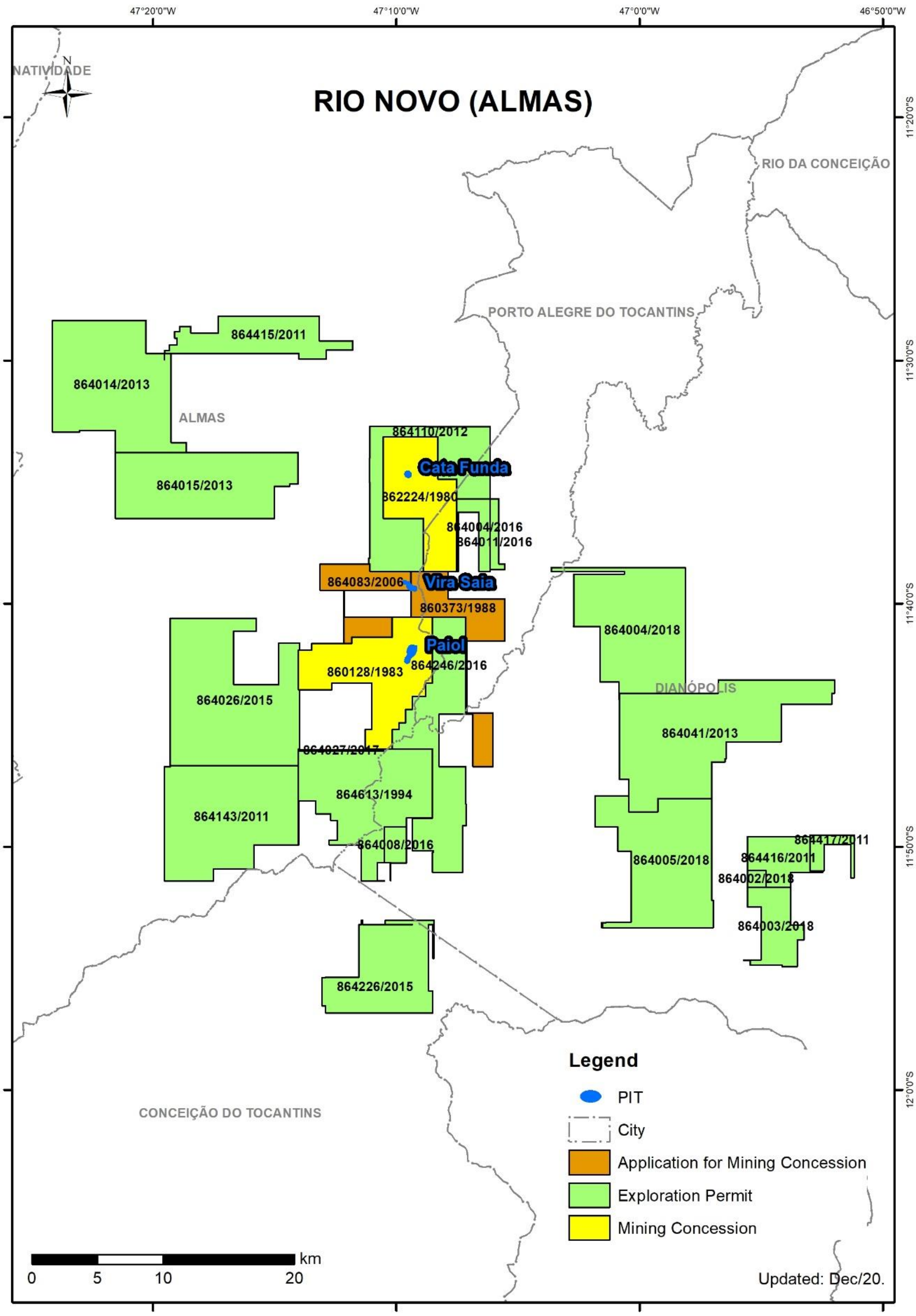
Rio Novo's mineral rights covering the principal areas of interest including the Paiol and Cata Funda gold deposits are controlled, respectively, by two Mining Concessions (9,137 ha). The Vira Saia deposit is held by two Application Mining Concession (4,483.75 ha) submitted on March 5, 2013 (Figure 4-2 for the main targets).

With respect to mineral rights outside of the principal areas of interest, Rio Novo holds twenty (21) Exploration Permits totaling 88,485.48 ha.

A map showing the distribution and classification of Rio Novo's mineral holdings in the Almas Greenstone Belt as of December 2020 is presented on Figure 4-3.

It is important to mention that Resolution 50/2020 was not used for the deadlines of the Exploration Permits. This Resolution extends some deadlines for the pandemic situation. In this way, some deadlines allow some additional time.

Figure 4-3 Concession and Exploration License December 07, 2020



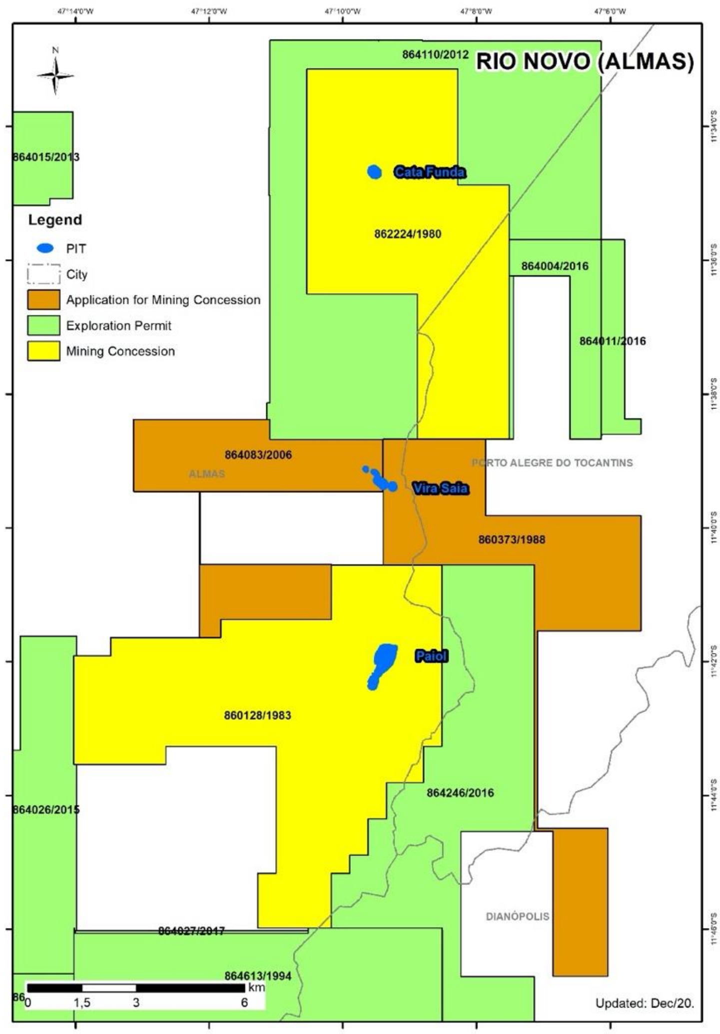
4.3 MINING CONCESSION AND APPLICATION OF MINING CONCESSION

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 passage, 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 4-4 is a map showing the status of their mineral licenses in the vicinity of the Paiol, Cata Funda and Vira Saia properties. The map shown on Figure 4-4 is current as of December 2020.

Figure 4-4 Concession and Exploration License in detail December 07, 2020



The Almas Gold Project includes the properties covered by the Mining Concession ANM number 860.128/1983– Paiol (mined in the past by VALE)"; the property under ANM number 862.224/1980 – Cata Funda which is undeveloped, and the Application of Mining Concession coincident with the Vira Saia deposit (864.083/2006, 860.373/1988).

The Mining Concession processes have a request for an extension of the start of mining activities submitted on April 6, 2018. To operate the new project at the Paiol mine, Rio Novo was required to obtain a new environmental license under the standards set forth by the Tocantins State environmental authority (NATURATINS). The Environmental Authority (NATURATINS) has accepted the Environmental Assessment Report (EA) for the Paiol mine area and granted the Installation License No. 5437/2011 (Licença de Instalação or LI) on December 2, 2011, which has already expired. Based on this permitting process other Licenses were issued in 2017 as Preliminary License No. 286/2017 (Licença Prévia - LP) and Installation License No 297/2017 (Licença de Instalação - LI). The renewal of the Installation License - LI was required on December 14, 2018 and is undergoing analyses by technical staff of NATURATINS. It is expected that the renewal will be received in the first quarter of 2021.

The Vira Saia deposit is held by two Application Mining Concessions (Process No. 864.083/2006, 860.373/1988). Recently the ANM published two requirements for this request. The requirements are related to the presentation of a document certifying the Financial Capacity and the Environmental License.

On February 13, 2020 it was presented by the Aura´s Financial Capacity and Trial Balance, and in reference to the Environmental License, a new deadline for the presentation was requested since it is in progress. For the permitting of the mineral deposits of Cata Funda and Vira-Saia, another Environmental Assessment (EA) was required by NATURATINS, since illegal artisanal mining ("Garimpo") has already degraded the areas over the years and the potential for negative impacts is low. It is estimated that this study will start in early 2021, the protocol and application for the Preliminary and Installation licenses with NATURATINS in the 4th quarter of 2021, and the forecast for obtaining Licenses between the 3rd and 4th quarter of 2022.

4.4 SURFACE RIGHTS: ACCESS TO LAND

4.4.1 FOREIGN OWNERSHIP OF RURAL LANDS

Prior to initiating negotiations for acquisition of land, Rio Novo developed procedures for consultation with the affected parties and criteria for valuation of 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.

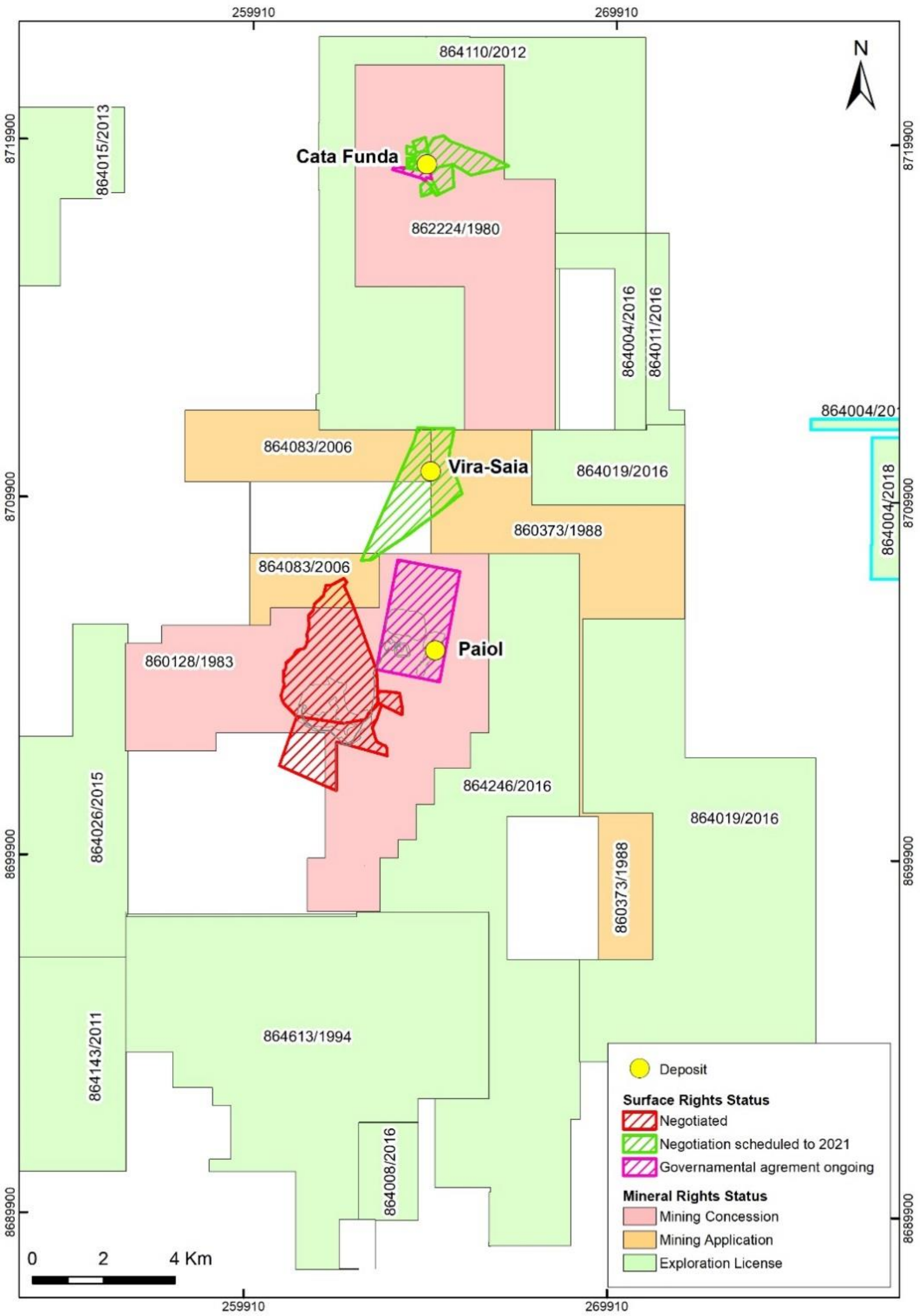
According to the engineering updates for the new production scale, the Project's occupation area has been optimized by reducing the amount of land to be negotiated.

The company Integratio was hired in July 2019 to update the values for the land negotiations necessary for the implementation of the Project.

The office of lawyer Hercules Jackson from Palmas city was hired to survey all the documentation of the surfaces required for the Project and is in the conclusion phase until December 2020. Minimum and maximum values were estimated for each property based on a detailed and well-documented appraisal of local real-estate values in the Almas region of Tocantins State. This work was conducted to ensure maximum transparency and fair valuations according to the local real estate market.

The status of Rio Novo's surface and mineral rights in the three target areas, as of December 2020, is given on Figure 4-5.

Figure 4-5 Mineral and Surface Rights status



4.2.1 PAIOL MINE AREA

The status of Rio Novo's surface and mineral rights in the vicinity of the Paiol mine, as of December 2020, is given on Figure 4-6.

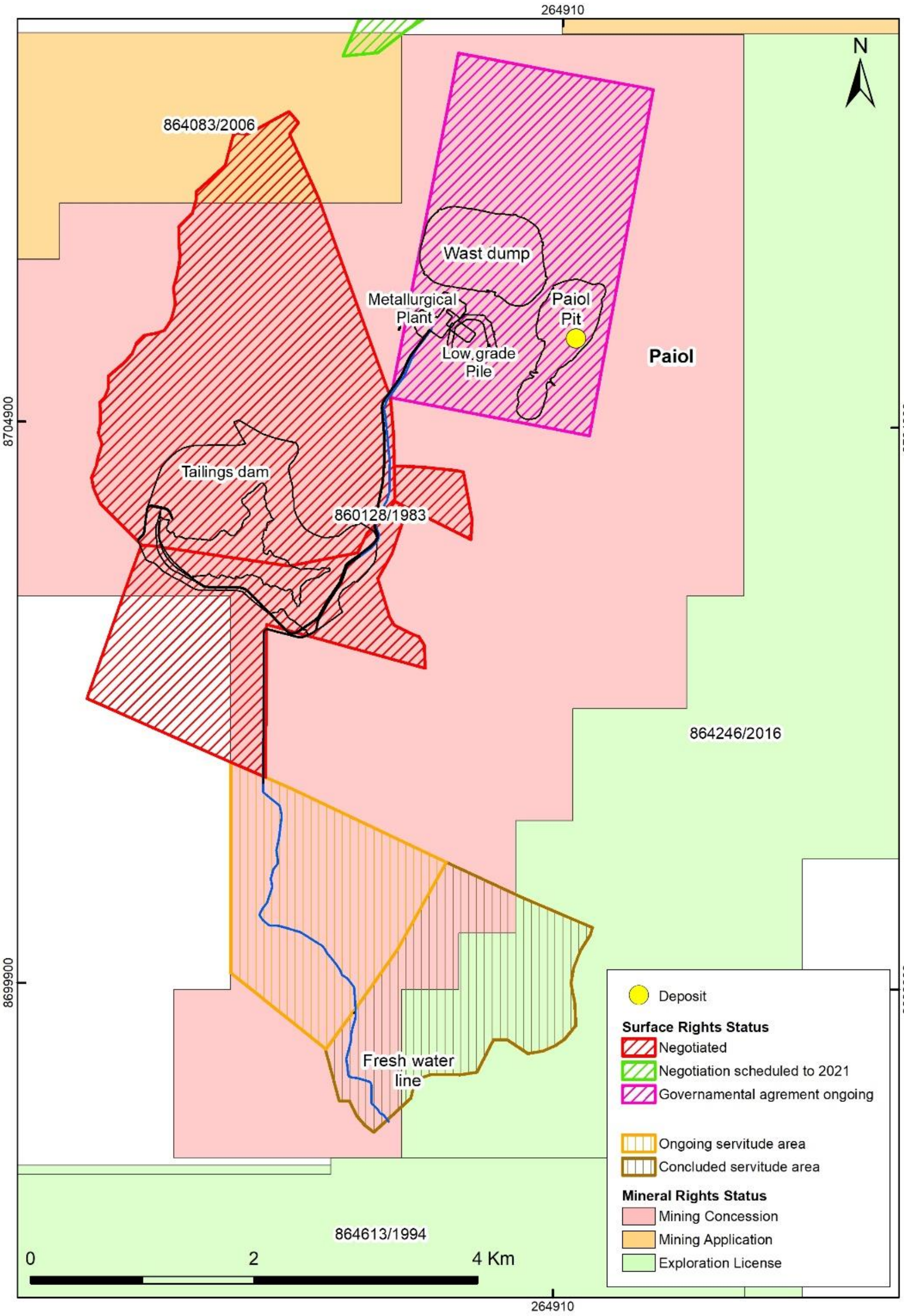
The surface rights covering the former Paiol mine and the surrounding areas previously utilized for the heap leach and other production facilities were donated by VALE to the state of Tocantins upon closure of the former operations. The state subsequently formed a private company, MINERATINS, to hold these surface rights.

On the State lands, Rio Novo is not required to compensate MINERATINS for use of the property other than for damages to existing buildings donated by VALE. These buildings may be reused as temporary facilities or during operations but may be demolished later as the open pit expands.

It will be necessary to pay Royalties to MINERATINS it being the owner of the surface rights. Royalties, according to Brazilian law, must be 50% of the CFEM rate, which in this case will be 0.75% of the gross revenue of the ore produced in the area of this surface. An administrative process to have the agreement with MINERATINS is underway and is being monitored by Aura's legal team and specialized law firms.

As shown on Figure 4-6, surface rights have been acquired for three properties, two of which are sufficient for the construction of the tailings dam.

Figure 4-6 Paiol's Mineral and Surface Rights status



Rio Novo received the approval of the Project's Economic Assessment Report (Plano de Aproveitamento Econômico, PAE) by ANM, which granted the inclusion of the property into the Mineral Servitude Area, thus enabling the company to establish an easement process, if required, as proscribed under the Brazilian legal system.

The remaining surface rights required for future operations at Cata Funda and Vira Saia are held by various private landowners. As shown on Figure 4-5, negotiations for the surface rights at the Vira Saia property and Cata Funda have not been initiated as of December 07, 2020. Rio novo intends to start negotiations for these properties in the first half of 2021, as these mine operations will be in the future according to the mining plan.

4.5 ROYALTIES AND EXPLOITATION TAXES

The ANM imposes a one percent 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% for the States or the Federal District, and 12% to DNPM. DNPM, 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.

Additionally, there is a royalty of 1.2% of revenue from the sale of any mineral production, minus refining charges, transportation and insurance costs, taxes, and sales charges, to be paid by Rio Novo to Mineração Santa Elina Indústria e Comércio S.A. (MSE) for production from tenements transferred from MSE to Rio Novo at the time of the IPO. For the purposes of this report this will apply to production from the Paiol and Cata Funda deposits.

Rio Novo must also pay MINERATINS, 0.75% royalties referring to the production of the Paiol deposit, as MINERATINS is the property's surface rights holder.

Production from the Vira Saia deposit will be subject to a 2.5% NSR royalty payable to Mineradora Santo Expedito Ltda., and Terra Goyana Mineradora Ltda.

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

5.1 ACCESS

The Almas Gold Project is situated in southeastern Tocantins State in the municipality of Almas, south of the city of Almas (population 7,000) in central Brazil. The Almas town site is accessed by State Highways TO-010, TO-070 and TO-050 from the state capital of Palmas, via Porto Nacional to Natividade – a trip of approximately four hours by car (Figure 5-1).

Palmas, the capital city of Tocantins State (population 306,000) has all the major facilities for industrial support as well as State governmental agencies. Palmas supports a regional airport with scheduled commercial service departing several times a day to Brasilia and São Paulo. The principal commercial center in the Almas region is Dianópolis, 45 km east of the Almas town site.

There are commercial flight options from Barreiras airport, with flights to Brasília, Belo Horizonte and Salvador. Barreiras is about four hours by car to Almas.

Barreiras (population of 156,000) and Luiz Eduardo Magalhães (population of 90,000), located east of Almas, with distances of 280 and 190 km respectively., On the BR-242 and TO-040 highways, are cities with good infrastructure, service companies, commerce and industries.

From the city of Almas, the three target areas may be reached by all-weather gravel roads, well maintained by the local government. The 17 km distance from Almas to the Paiol mine is traversed by light vehicle in approximately 20 minutes.

Almas may also be reached by chartered aircraft as the local government maintains a small gravel airstrip south of the town site.

At present, there is no rail service into the Almas area.

5.2 CLIMATE

The climate in the Almas region is characterized by two seasons with relatively constant temperatures but varying degrees of precipitation. The project area is tropical with average monthly temperatures varying from 26°C in the dry season to 22°C in the wet season. The maximum average temperature was recorded in September (28°C), while the minimum average temperature was recorded in July (25.4°C).

The historical average annual rainfall is approximately 1,700 mm, most of which falls in the rainy season, October to March, which is followed by the winter dry season, April to September.

5.3 PHYSIOGRAPHY

The Almas Gold Project area lies wholly within the Cerrado ecoregion, a vast woodland savanna that is best developed in the plateau country of the Central Brazilian Highlands. The Cerrado supports a diverse tropical fauna and flora, which extends over large parts of Goiás, Minas Gerais and Tocantins States. After the Amazonian ecoregion, the Campo Cerrado is the largest of Brazil's major habitats, accounting for approximately 21 percent of the country's land area.

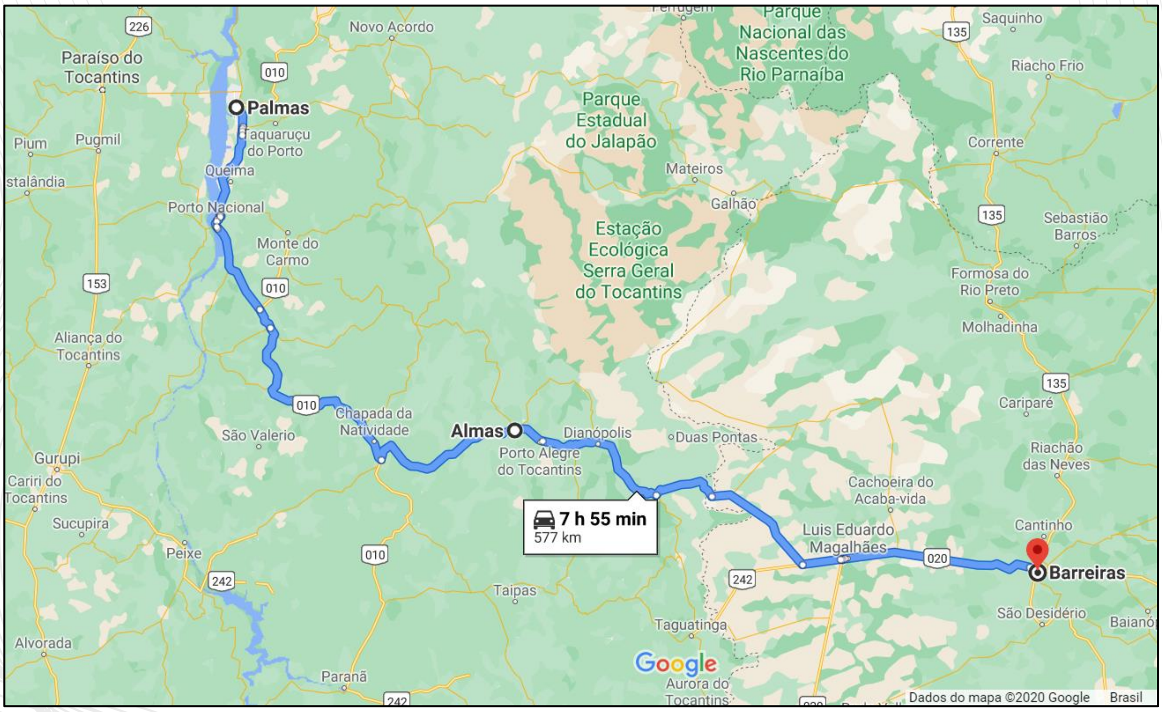
The Almas Gold Project extends over a landscape that is dominated by agricultural activities. Locally the impacts of past mining and ongoing garimpeiro (artisanal mining) activity are evident. Currently the Cerrado savannas in central Brazil are under pressure as more land is converted to agricultural use, by virtue of low land prices and potential for irrigation due to improvements in soil management and irrigation techniques.

The Brazilian Highlands comprise an extensive plateau region which forms the divide between Brazil's largest river systems. Elevation of the plateau varies between 750 m and 900 m above mean sea level. The project area lies within the major Araguaia-

Tocantins river basin which drains portions of Goiás, Tocantins, Maranhão and Pará states by flowing northward into Amazonia before reaching the Atlantic Ocean. The rivers in the region are generally not navigable except for short distances.

Tropical forests occur as "islands" in the Cerrado or as riparian forests in the southern part of the Project area where they border small perennial to intermittent streams.

Figure 5-1 Highways from Google Maps



5.4 LOCAL RESOURCES AND INFRASTRUCTURE

The Almas Gold Project area is sparsely populated largely owing to the undeveloped nature of the area and the presence of dispersed cattle ranching operations for which a few ranch houses occur in the project area.

The city of Almas has few industrial services, primarily small mechanical, machine, and repair shops. Commercial services include small grocery and department stores, as well as restaurants and small hotels. Public services include a clinic, churches, schools, and local government offices. The principal agricultural products of the region include rice, millet, soy, manioc and cattle.

The water supply for the project will be drawn from the Rio Manuel Alves, a westward flowing tributary of the Rio Tocantins and the largest stream in the project area. Water will be drawn at a point south of the tailing's storage facility. River water will be pumped to the facility where it will be combined with reclaimed water and pumped to the reclaimed water pond located adjacent to the processing plant.

Power supply to the project is available from the regional electrical utility company, ENERGISA. Locally power is generated by several hydroelectric plants. A demand in the order of 8.6 MW is estimated at full milling capacity. Power will be supplied by ENERGISA from the Almas substation, located approximately 18 km from Paiol, via a 138 kV overhead power line to a local substation at the plant site, then distributed to the mill and mine facilities by a local network. The current power line was built for the old Paiol Project from VALE and is currently operational, however, the supply line will need to be updated to support the project with provisional energy during construction and is already in progress through a contract signed with ENERGISA.

6 HISTORY

Gold mining in the Almas area began in the 1700s during colonial times when slave labor was used to extract gold from near-surface oxide zones. In more recent times, garimpeiros (artisanal miners) expanded the earlier excavations. In 1977 the exploration arm of VALE identified some potentially prospective volcano-sedimentary sequences of Archean age in the region. Further exploration by VALE in the mid- to late-1980's led to discoveries at Cata Funda and Paiol. In 1996, VALE commenced mining at the Paiol deposit.

6.1 PROJECT OWNERSHIP

Recent project ownership commenced in 1985 with a joint venture between Companhia VALE do Rio Doce (VALE) and Metais de Goiás (METAGO), a mineral exploration company of Goiás state. In 1989, exploration work was interrupted when Tocantins state was formed by dividing Goiás state, which prevented METAGO from continuing as a partner in the exploration venture. Work recommenced after an agreement was signed between VALE, METAGO and Tocantins State. In 2006, VALE transferred the mineral rights, mining license and environmental permits to Mineração Apu., the predecessor to Rio Novo. Table 6-1 summarizes the chronology of ownership.

Table 6-1 Summary of Ownership of Almas Gold Project

OWNERSHIP	PERIOD
VALE S.A. (CVRD)	1985 to 2006
Mineração Apuã Ltda.	2006 to 2010
Rio Novo Ltda.	2010 to 2018
Aura Minerals	2018 to present

6.2 EXPLORATION HISTORY

Gold has been the primary target of exploration in the district. Discoveries thus far, have been made by a combination of mapping and soil sampling, followed by drilling. To date, exploration has primarily targeted near-surface gold anomalies and is therefore still in the early stages. The major exploration milestones are highlighted below:

- 1985: VALE and METAGO, agreed to jointly explore the area.
- 1985 to 1987: Several targets were identified during this phase of exploration: Paiol, Cata Funda, Vira Saia, Morro do Carneiro, Refresco, Vieira, Ijuí, Mateus Lopes and Cemitério.
- 1986: Initial drilling and discovery of the Cata Funda deposit.
- 1987: Discovery of Paiol deposit.
- 1996: VALE reports initial resource estimates for the Paiol deposit.
- 1996 to 2001: VALE conducts mining of the Paiol deposit.
- 2006 – Mineração Apuã commences exploration.
- 2008 to 2010: Rio Novo conducts confirmation drilling, resulting in a resource estimate, reported as an NI 43- 401 Technical Report in February 2010.
- 2010 to 2011: Core drilling initiated by Rio Novo for confirmation and expansion of the Paiol and Cata Funda resource areas as well as exploration of nearby targets.
- 2011: Discovery of the Vira Saia deposit 5 km north of Paiol.

- 2011 to 2012: Infill drilling and resource modeling at Vira Saia brought additional resources and enhanced the overall Almas Gold Project, leading to completion of a Preliminary Economic Assessment (PEA) in March 2012.
- 2013 & 2016: RPM completed two feasibility study level reports (NI 43-101).

6.3 HISTORIC RESOURCE ESTIMATES

Resource estimates have historically been produced by VALE in 1996 and by Rio Novo in 2010 and 2012. These are historical estimates, are not considered 43-101 compliant and cannot be relied on to evaluate the economic viability of the project. Table 6-2 outlines these estimates.

Table 6-2 Summary of Historic Resource Estimates

DEPOSIT OR AREA	CLASS	TONNES (000)	GRADE (g/t Au)	CONTAINED (000 ozs Au)	NOTE
Paiol	N/A	4,300	0.95	132	VALE 1995 (1)
Paiol	N/A	11,600	3.9	1,450	VALE 1996 (2)
Paiol	N/A	872	4.8	252	VALE 2002 (3)
Paiol	IND	5,027	1.89	306	Rio Novo 2010 (5)
Paiol + Cata Funda	M+I	16,518	1.16	614	Rio Novo 2011 (5)
Paiol + Cata Funda	M+I	17,091	0.99	546	Rio Novo 2012 (6)
Paiol + Cata Funda + Vira Saia + Leach Pad	M+I	29,310	0.87	820	RPM (2016)

*Note:
1. N/A = Not Classified M+I = Measured + Indicated IND = Indicated
2. Tonnes and ounces numbers are rounded

VALE reported three resource estimates: (1) in 1995 and, (2) 1996 prior to mining, and (3) in 2002, resource remaining at the end of mining. The estimates used cross-sectional interpretation along with GEMCOM software. The resource estimating procedures did not comply with NI 43-101 requirements and the reliability of the numbers is unknown and therefore cannot be relied upon to evaluate the economic viability of the project.

Rio Novo published a reportedly NI-43-101 compliant resource estimate in 2010 (4) based on work completed by Marston Associates and GeoSim Services (Marston & Marston, 2010). The study created a resource model with mineralized domains based on geological and grade information derived from diamond drill core, reverse circulation drill holes and surface trenches. Grade was interpolated into blocks by Ordinary Kriging. The study estimated 5.0 Mt averaging 1.89 g/t Au (306 Koz. Au) as Indicated Resource with a further 2.4 Mt averaging 3.0 g/t Au classified as Inferred. This resource was for the Paiol deposit only.

Rio Novo completed an interim resource update in May 2011 (5) (Rio Novo Press Release, June 2011). This reportedly NI 43-101-compliant estimate showed Measured + Indicated Resources of 16.5 Mt averaging 1.16 g/t Au (614 Koz. Au) plus Inferred Resources of 3.9 Mt at 1.64 g/t Au. The resources were contained at Paiol (including the leach pad) and Cata Funda.

Rio Novo published a third reportedly NI 43-101-compliant resource estimate in April 2012 (6) based on work completed by GeoSim Services and RPM. The resource estimate included the Paiol, Paiol Leach Pad, Cata Funda, and Vira Saia deposits. The estimates were generated based initially on cross-sections of lithology, alteration, and gold zones, then compiled into 3D solids using Leapfrog software. The final geological and resource models were completed in Surpac software. Block grade estimation used Ordinary Kriging for interpolation. This study estimated a total of 17.1 Mt averaging 0.99 g/t Au (546 Koz. Au) Measured + Indicated, along with an additional 1.2 Mt at 0.86 g/t Au Inferred within the three deposits.

RPM used the same 2012 Rio Novo geological models and published first a feasibility study in 2013 and then an updated feasibility study (both as NI 43-101 reports) in 2016.

6.4 PREVIOUS PRODUCTION

From 1996 to 2001, VALE operated an open-pit and heap leach operation at Paiol. Production was about 86,000 ounces Au from 1.6 Mt of ore. Operations were suspended in 2001 due to the low gold prices. The production history of the Paiol Mine is summarized as follows:

- June 1996 – The Paiol Mine commenced operation, and 418,248 tonnes at 2.42 g/t Au were produced. Gold recovery was 66.41%.
- March 2001 – Operations at Paiol were suspended due to the low gold price of US$279 per ounce and the mine closed down after 4 years and 9 months of operation. During the production period, 4,992 kg of gold were mined, and 2,699 kg of gold were produced for sale. Final production figures are presented in Table 6-3.

Table 6-3 Paiol Mine Historic Production

DESCRIPTION	UNIT	1996	1997	1998	1999	2000	2001	TOTAL
Ore Processed by Heap Leach	t	418,248	455,892	417,240	383,508	344,736	15,027	2,034,651
Au Grade	g/t	2.42	2.21	2.74	2.62	2.28	2.52	2.4
Gold Content	g	1,012,160	1,007,521	1,143,238	1,004,791	785,998	37,868	4,991,576
Recovered Gold	g	672,175	589,268	510,949	497,256	410,551	19,260	2,699,459
Recovered Gold	oz	21,613	18,948	16,429	15,989	13,201	619	86,799
Metallurgical Recovery of Gold	%	66.41	58.49	44.69	49.49	52.23	50.86	54.08
Silver Production	g	45,863	51,060	43,947	37,930	33,917	387	213,101
Gold Left in Heap Leach	g	399,985	418,253	632,289	507,535	375,447	18,608	2,352,117
Grade of Gold in Heap Leach	g/t	0.96	0.92	1.52	1.32	1.09	1.24	1.13
Gold Left in Heap Leach	oz	12,861	13,449	20,331	16,319	12,072	598	75,630

- 2001 – All installations were dismantled and disposed of, and the site was reclaimed in compliance with the requirements of the State environmental authority
- 2001 to 2003 – VALE changed the mining license status with DNPM to one of "indefinite suspension," which allows resumption of operations at short notice.
- 2006 – VALE transferred the mineral rights, mining license and environmental permits to Mineração Apua., the predecessor of Rio Novo.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 TECTONIC PROVINCE

The Tocantins Structural Province (TSP), located in the central region of Brazil, constitutes a system of Brazilian orogens, characterized by folding and thrusting belts called the Brasília, Paraguay and Araguaia Belts, resulting from the convergence and collision of three continental blocks: Amazonian Craton, to the West; Cráton São Francisco, to the East; and Cráton Paranapanema, to the Southwest, covered by the rocks of the Paraná Basin, this one with its limits inferred through gravimetric data. The foundation of the province is composed of Archaean and Paleoproterozoic terrains reactivated during the Brazilian Cycle (Delgado et al., 2003). In the final stages of the Brazilian Cycle, the gravitational collapse, exhumation and / or extrusion of the orogens, which occurred up to the Upper Ordovician have been related as a transition stage between conditions of active convergent tectonics to conditions of intraplate stability, with the beginning of the Paleozoic basins.

These phanerozoic coverings, in almost all Paraná, Parnaíba, Bananal and Pantanal Mato-Grossense basins, hide large parts of this system.

The TSP is compartmentalized in the Cráton São Francisco for small exposures of the basement and the extensive sedimentary coverings, Brasília Belt, built on the west side of the craton, divided into Zones Internal and External, Goiás Massif and Goiás Magmatic Arch.

The Brasília Belt is a well-preserved orogenic belt, consisting of a thick set of passive-edge sedimentary rocks associated with volcanic rocks, a mixture of ophiolitic, calc-alkaline volcanic arch-type rocks and intrusive S-type granites. The degree of metamorphism increases to the West, moving from low-grade non-metamorphic and metamorphic rocks, on the São Francisco Craton border to the East, to high-temperature amphibolite facies and ultra-high-temperature granulites in the metamorphic nucleus, decreasing again for amphibolite and green schist facies in the rocks of Goiás Magmatic Arch.

The tectonic zoning of the Brasília Belt is marked by a foreland transition from the São Francisco Craton to the East, External and Internal allochthonous zones formed in an old passive Neoproterozoic margin of the paleocontinent São Francisco-Congo to the West, for the exotic terrains of the Goiás Massif and the Goiás Magmatic Arch.

In the central part of the Tocantins Province the following Precambrian domains are recognized: Crixás – Goiás Archean Terrain, interpreted as a small allochthonous continental block; Paleoproterozoic sialic base, represented by orthogneisses and metavolcanosedimentary sequences, in the region of Almas – Dianópolis; the Anápolis – Itauçu High-grade Complex; Paleo-Mesoproterozoic Mafic Ultramafic Bedded Complexes (Barro Alto, Niquelândia, Canabrava) and associated metavolcanosedimentary sequences; and Goiás Magmatic Arch, of the Neoproterozoic.

The External Zone of the Brasília Belt is composed of Mesoproterozoic and Neoproterozoic metasedimentary units represented by the Araí and Natividade Groups, Paranoá and Canastra Groups, as well as portions of the Archean-Paleoproterozoic basement that show rejuvenation due to the Brazilian tectonics.

The Northern and Southern portions of the outer zone of the Brasília Belt represent two contrasting styles of proximal continental passive margins. In its Northern part, between Alto Paraíso de Goiás and Natividade / TO, it consists of a large Paleoproterozoic crustal block, partially covered by a gently folded rift sequence of the Arai Group metasedimentary rocks (1.77 Ga and younger). This block is limited to the East for thrust faults that places it over the rocks of Bambuí group. To the South of Alto Paraíso de Goiás, the Araí Group is inconsistently covered by the Paranoá Group, which is also pushed over the Bambuí Group to the East.

The Natividade Group is represented by quartzites, conglomeratic quartzites, phyllites, schist quartz, marbles and the Araí Group, composed of a set of metamorphic and metavolcanic rocks, of low metamorphic degree, superimposed on the Aurumina Suite and the Ticunzal Formation, constitutes a succession deposited in an intracontinental rift basin, whose evolution started before 1.77 Ga in the Paleoproterozoic.

The Paranoá Group corresponds to a psammite-pelite-carbonated succession deposited under platform conditions. Its deposition is interpreted as occurring in the Mesoproterozoic (1,542 to 1,042 Ma). Its age is corroborated by its stratigraphic position (it occurs on sediments from the Araí Group post-rift phase and under the Bambuí Group pelites and carbonates), by the presence of conical stromatolites (conophyton) and isotopic data.

The Canastra Group is considered a lateral equivalent of the Paranoá Group, with more marked metamorphism, and occurring in the south-central part of the Brasília Belt.

The Internal Zone includes allochthonous units of the Araxá Group, as well as portions of the basement heavily involved in the Brazilian tectonics (Goiás Massif, with remnants of greenstone belts). There are also granulitic mafic-ultramafic complexes and Proterozoic volcano-sedimentary sequences.

7.2 REGIONAL GEOLOGY

The Pre-Cambrian lands, which constitute the Tocantins Structural Province, are characterized by folded and metamorphized supracrustal belts, exposed along the edges of the Amazonian Cratons (Paraguay and Araguaia Belts) and São Francisco (Brasília and Uruaçu Belts). Amid the folded belts, there is the so-called Goiás Massif, a massif of crystalline rocks of varied nature and age, whose geotectonic significance is still poorly understood due to the scarcity of geochronological data, and geophysical and structural information.

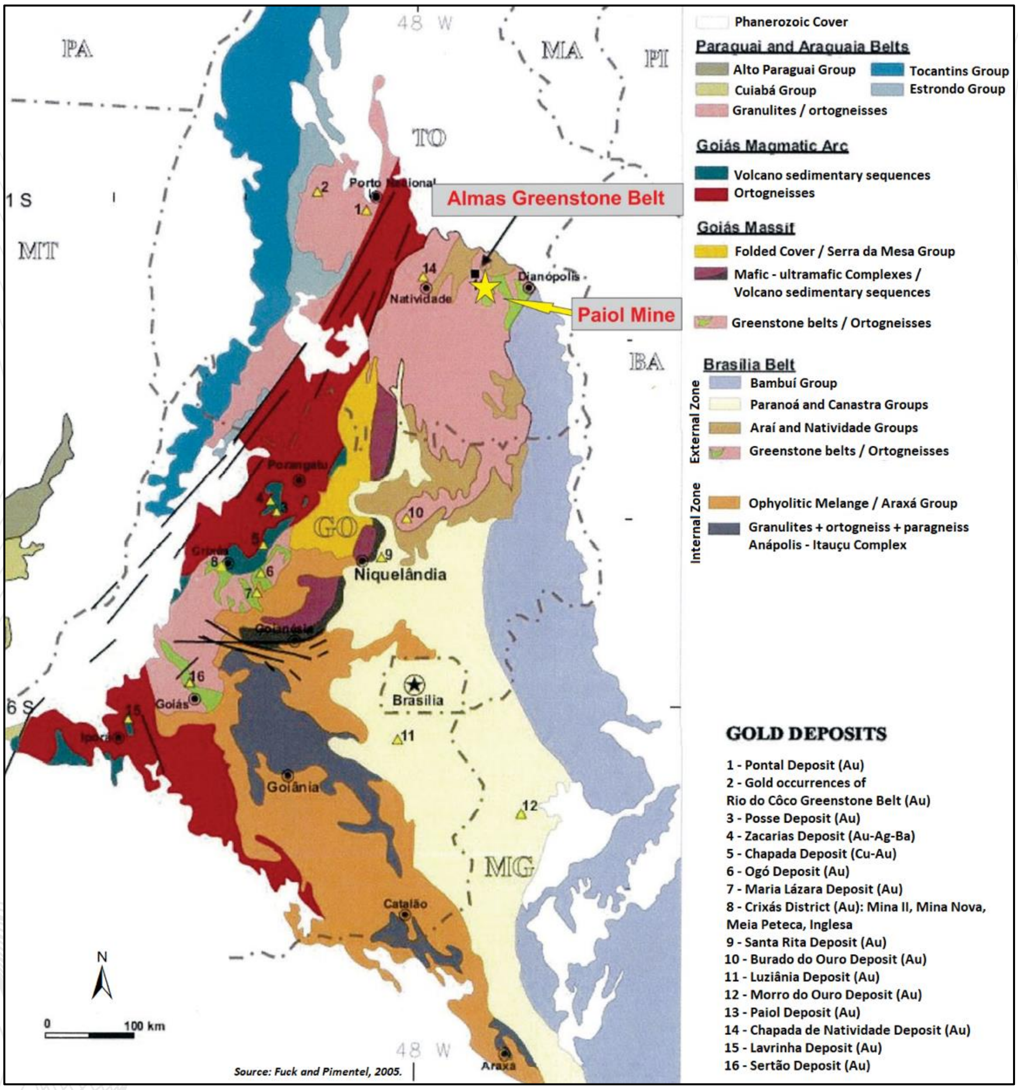
The Massif of Goiás is considered an allochthonous microplate, added to the Western margin of the Brasília Belt during the last stages of evolution of the Neoproterozoic orogenesis. It consists of a set of typical TTG granite-gneiss complexes and narrow bands of greenstone-belt sequences. The granite-gneiss lands are comprised of orthogneisses that, in the Northern part of the land, comprise the Tapir, Caiamar, Moquém and Hidrolina and, in the South, the Caiçara and Uvá complexes.

The Almas Project area occurs within the Goiás Massif: an Archean-to-Paleoproterozoic granite-greenstone terrane. The greenstone belt lies within the Almas-Dianópolis Terrane on the Western block of the Goiás Massif. (Figure 7-1 and Figure 7-2).

7.3 REGIONAL ALTERATION AND MINERAL DEPOSITS

There are two metamorphic events, one, M1, related to Dn, restricted to the greenstones and M2 related to shear zones Dn+1, which affects all units of the Almas-Dianópolis Terrane. Regional metamorphic paragenesis M1 varies from amphibolite to greenschist facies and the main paragenesis in metabasalts is amphibolite + plagioclase ± chlorite ± epidote. M2 paragenesis is characterized by amphibolite + albite + epidote ± white mica ± chlorite, occurring in the greenstones, granite-gneiss complexes and basic-ultrabasic intrusions. The amphibolite composition of M1 paragenesis varies from ferric actinolite to tshermakitic hornblende, while the plagioclase varies from albite to andesine.

Figure 7-1 Tocantins Province Tectonostratigraphic Map

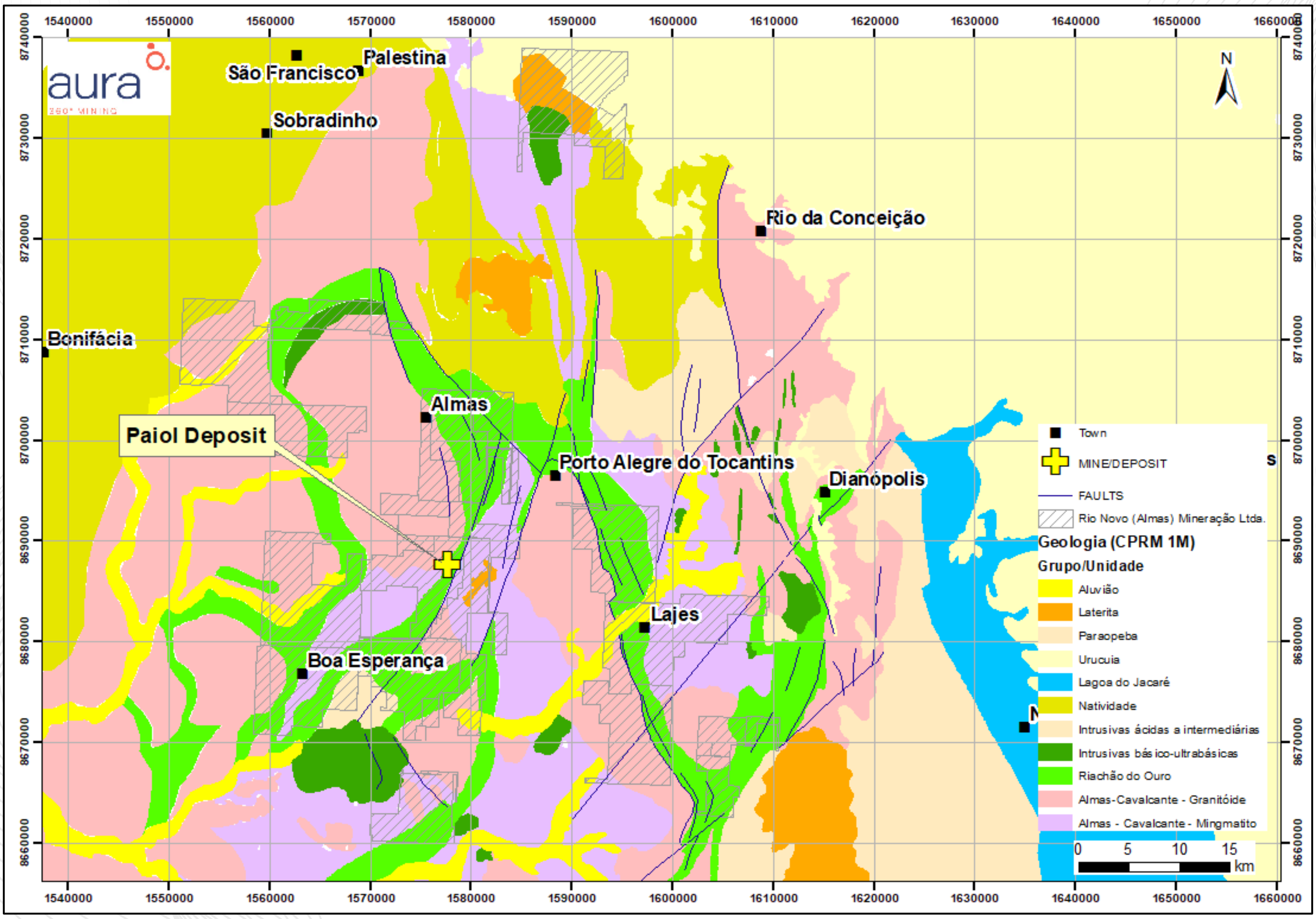


The orogenic gold deposits are characterized by the predominance of quartz veins, containing sulfides (≤3-5%), mainly Fe sulfides and carbonate minerals (≤5-15%). The veins may contain albite, white mica, chlorite, tourmaline in the greenschist domains; or amphibole, diopside, biotite-phlogopite, tourmaline and garnet in the amphibolite facies. The veins evolve a continuous system with an extension of approximately 1-2 km and little change in mineralogy or intensity of mineralization.

Hydrothermal alteration presents more pronounced lateral variation than vertical variation in the plane of mineralization. The hydrothermal mineral assembly and the intensity of mineralization varies according to the type of embedding rock and crustal level of alteration. Regionally, the presence of calcite, dolomite, ankerite, pyrite, chlorite, sericite and fuchsite is recognized in the greenschists facies, and calcite, pyrrhotite, Ca amphiboles, diopside, coarse, biotite and feldspar in higher metamorphic degrees. Sulfidation is more intense in banded iron formations and carbonation is dominant in mafic and ultramafic embedding rocks. Significant enrichment of SiO_2 in the mineralized zones is evidenced by the presence of large amounts of quartz veins.

Always associated with shear zones (Dn+1), the mineralization is essentially hosted in the granite-gneiss complexes and banded iron and greenstone amphibolite formation. In all occurrences, gold is hosted by segregations and / or quartz veins bordered by hydrothermal alteration zones, mainly sericitization and argilization in gneissic granite, chloritization, sericitization and carbonatization in amphibolite and carbonation, sulfidation and tourmalinization in banded iron formation. The main deposits are Vira-Saia, in granite-gneiss and Córrego Paiol, in amphibolite.

Figure 7-2 Tocantins Province Tectonostratigraphic Map



7.4 DISTRICT GEOLOGY

The Almas Gold Project is situated in an historical gold mining area with numerous small artisanal mines (garimpos) and the former Vale Paiol Gold operation. Gold occurs in sheared metavolcanic rocks within the greenstone sequence, as well as in sheared felsic intrusive rocks. The rocks are, in general, Paleo-Proterozoic in age (~2.2 billion years old) and have undergone regional metamorphism ranging in intensity from greenschist- to amphibolite-facies. The metamorphism resulted in deep-seated, shear-hosted, mesothermal, gold deposits which have more recently been referred to as orogenic gold deposits. The gold-mineralized zone occurs in the core of hydrothermal alteration zones, generally associated with variable amounts of quartz, carbonate, albite, sericite and sulphide minerals.

7.5 DISTRICT STRATIGRAPHY

The Almas-Dianópolis terrane, in the Western block of Goiás Massif, comprises several narrow greenstone belts surrounded by granite-gneiss complexes. The volcano-sedimentary sequence of Riachão do Ouro Group is composed at the base by the Córrego do Paiol formation and at the top by the Morro do Carneiro formation. Late granitic intrusions cut the supracrustal sequence.

These rocks were metamorphosed to amphibolite facies during a regional tectonic-metamorphic event (Dn) and then retrogressively altered to greenschist facies assemblages, followed by a subsequent hydrothermal alteration phase linked to late strike-slip shear zone events (Dn+1).

7.5.1 CÓRREGO DO PAIOL FORMATION

Córrego do Paiol Formation is composed of dominant metabasalts, locally pillowed, and rare small occurrences of ultramafic metavolcanic rocks. The metabasalts are essentially massive dark green fine-grained amphibole-plagioclase-chlorite-epidote schists. Moreover, at contacts with granitoids actinolite schists are present, probably as a result of contact metamorphism. Ultramafic rocks are tremolite-chlorite schists that crop out near Dianópolis. These metabasalts correspond to high-Fe tholeiites. The ultramafic volcanic rocks are high-Mg tholeiites with komatiitic affinity. The pillowed metabasalts are indicative of a submarine deposition.

7.5.2 MORRO DO CARNEIRO FORMATION

The upper Morro do Carneiro Formation consists mainly of a monotonous sequence of sericitic phyllites with carbonaceous material rich layers and more rarely chlorite-bearing layers. It also includes beds of variable thickness of more abundant hematite-magnetite banded iron formations; quartzites with magnetite bearing layers; tourmaline quartzites; and metacherts. Iron formations and metacherts are exposed near the contact with the Córrego do Paiol Formation South of Almas.

7.5.3 GRANITE-GNEISS COMPLEXES

The granitoids are classified into two suites differentiated by their predominant mafic mineral. Suite 1 is comprised of hornblende-predominant tonalites, granodiorites, trondhjemites, quartz-monzodiorites and quartz-diorites. Suite 2 is composed of biotite-predominant tonalites, trondhjemites, granodiorites and monzogranites. Both suites have calc-alkaline tonalitic-trondhjemitic chemical affinities.

7.6 DISTRICT STRUCTURAL GEOLOGY

The most notable structural feature of Terreno Almas-Dianópolis is the distribution of greenstone belts and granite-gneiss complexes. Costa et al. (1976) were the first to recognize the Y-shaped distribution of the belts, mainly linear, in the North-South direction with branches to the Northeast and Northwest. They also have curved contacts around the granite-gneissic complexes. The complexes have geometry partially obliterated by directional shear zones that truncate the geological contacts and affect the metasedimentary rock coverings of the Natividade, Paranoá and Araí groups, the oldest structures generated in the Dn event present, in greenstones, a subvertical schistosity with features shear locations, which tend to be parallel to the contour between greenstones and granite-gneissic complexes, tight vertical folds and sub horizontal mineral lineation.

Younger structures include directional shear zones Dn + 1 with distal movement of main direction N20-30E and subsidiary directions N0-10E and N10-20W. These Dn + 1 shear zones have been related to the evolution of the Almas-Dianópolis Terrain. Dn_1 shear zones were not observed in the rocks of the Bambuí and Natividade groups in the region. Two other directions of shear zones are observed N35-50W with sinistral movement and N40-65E with distal movement, forming a conjugated pair resulting from East-West compression. The greenstones are arranged in the directions N10-35E, N10-20W, N45W and NS.

The structural control of the mineralization in the Almas-Dianópolis Gold District is the most important control factor. In most cases gold is concentrated on second and third order structures located near regional deformation zones, especially the transcrustal zones.

Most gold bearing structures present a brittle-ductile nature but can be from different styles:

- Brittle-ductile reverse fault zones, from low to high angle.
- Fracture sequences, stockwork systems or brecciated zones in competent rock.
- Foliation with cleavage zones.
- Fold hinge zones and associated reverse faults in tight folds.

7.7 DISTRICT ALTERATION

Alteration zones are symmetrical with respect to the shear zones, typical of classic greenstone alteration, including: chloritization (distal zone), sericitization (potassic alteration) and silicification, widespread carbonatization and sulphidation (pyrite and/or pyrrhotite) (proximal zone). These alteration zones are developed around shear zones cutting the mafic-to-intermediate volcanic and volcaniclastic rocks previously affected by amphibolite facies metamorphism. The hydrothermal alteration presents more lateral than vertical variation within the mineralization structures.

7.8 DISTRICT MINERALIZATION

Gold mineralization is found in three groups of rocks, in metabasalts of the Córrego Paiol Formation, in metasedimentary rocks of the Morro do Carneiro Formation and in granite-gneiss complexes.

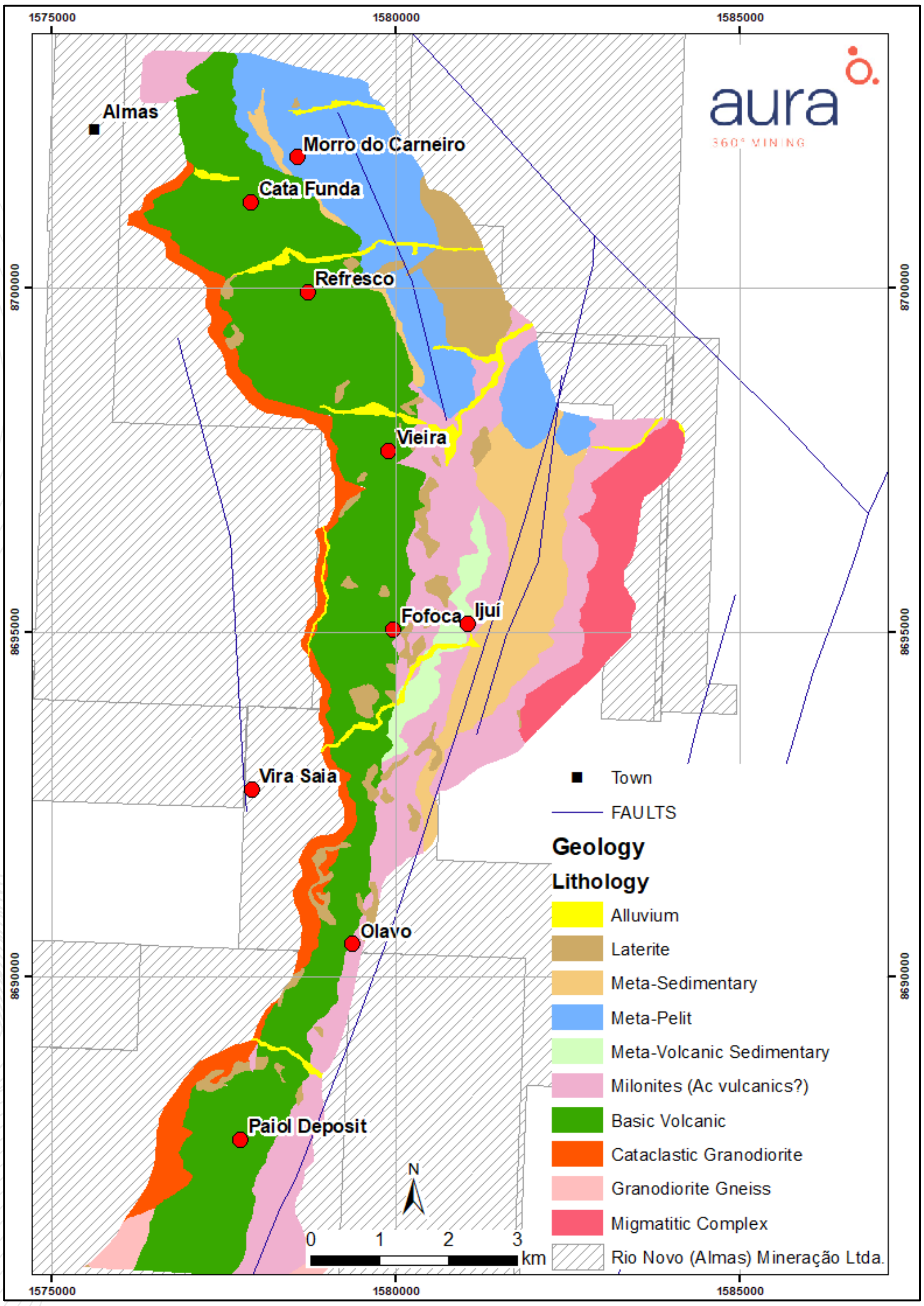
Several occurrences of gold are hosted in the metabasalts, mainly south of Almas, the most important being that of the Córrego Paiol mine. The occurrences hosted in metasedimentary rocks are dominant in the southern portion of the terrain, Conceição do Tocantins region, while the occurrences in granite-gneissic rocks are distributed throughout the granite-greenstone terrain.

Gold in the Almas Greenstone Belt occurs in three different associations:

- Gold associated with hydrothermally-altered shear zones in basic to intermediate volcanic rocks;
- Gold associated with hydrothermally-altered banded iron formation;
- Gold associated with smoky quartz veins in sheared granite gneiss.

Gold mineralization is closely associated with mylonitic banding in shear zones that cut mafic-to-intermediate volcanic rocks, schists and granite-gneiss, the latter being noted at the Vira Saia deposit. Gold occurs as free gold and as gold inclusions within sulfide minerals. The stronger gold mineralization is associated with faults and shear zones.

Figure 7-3 District Geology of the Almas Belt showing Garimpos



7.9 DEPOSIT GEOLOGY

The Cata Funda - Paiol Trend is a structural domain within the Almas volcano-sedimentary sequence that contains the primary and secondary gold targets. The Paiol and Cata-Funda deposits have many similarities with respect to mineralization type, geometry and lithologies, although some distinctive characteristics are noted.

The Paiol and Cata Funda deposits are situated on the same trend that generally strikes N10°-20°E at Paiol but rotates to N10°-20°W at Cata Funda. The dips range from 60° to 80° northwest and 40 to 60 southwest, respectively. Strong gold mineralization is associated with hydrothermal alteration centered on mylonitic bands and sulfide-bearing quartz veins in strongly sheared metavolcanic rocks.

The structural control of mineralization at the Paiol and Cata Funda deposits is related to strike-slip shear zones (Dn+1 event) and later remobilization into radial, brittle-ductile zones (Dn+2 event) (Ferrari & Choudhuri 2000). The ore shoots are believed to have formed where the mylonitic foliation (Sn+1.) is cut by extensional brittle faults of the Dn+2 event (Ferrari and Choudhuri 1999a). These faults possibly served as conduits for the transport of gold-rich hydrothermal fluids.

7.10 PAIOL DEPOSIT

The Paiol deposit is in the middle portion of the Almas Belt, approximately 500 m east of the metabasalt/ intrusive contact. The metabasalts that host mineralization display a primary foliation of 280° azimuth with a 65° dip. The mineralization is a result of hydrothermal alteration processes controlled by shear zones. Based on the drilling data, the altered rocks form a trend at least 1,400 m long, varying from 40 m to 140 m wide from South to North, respectively.

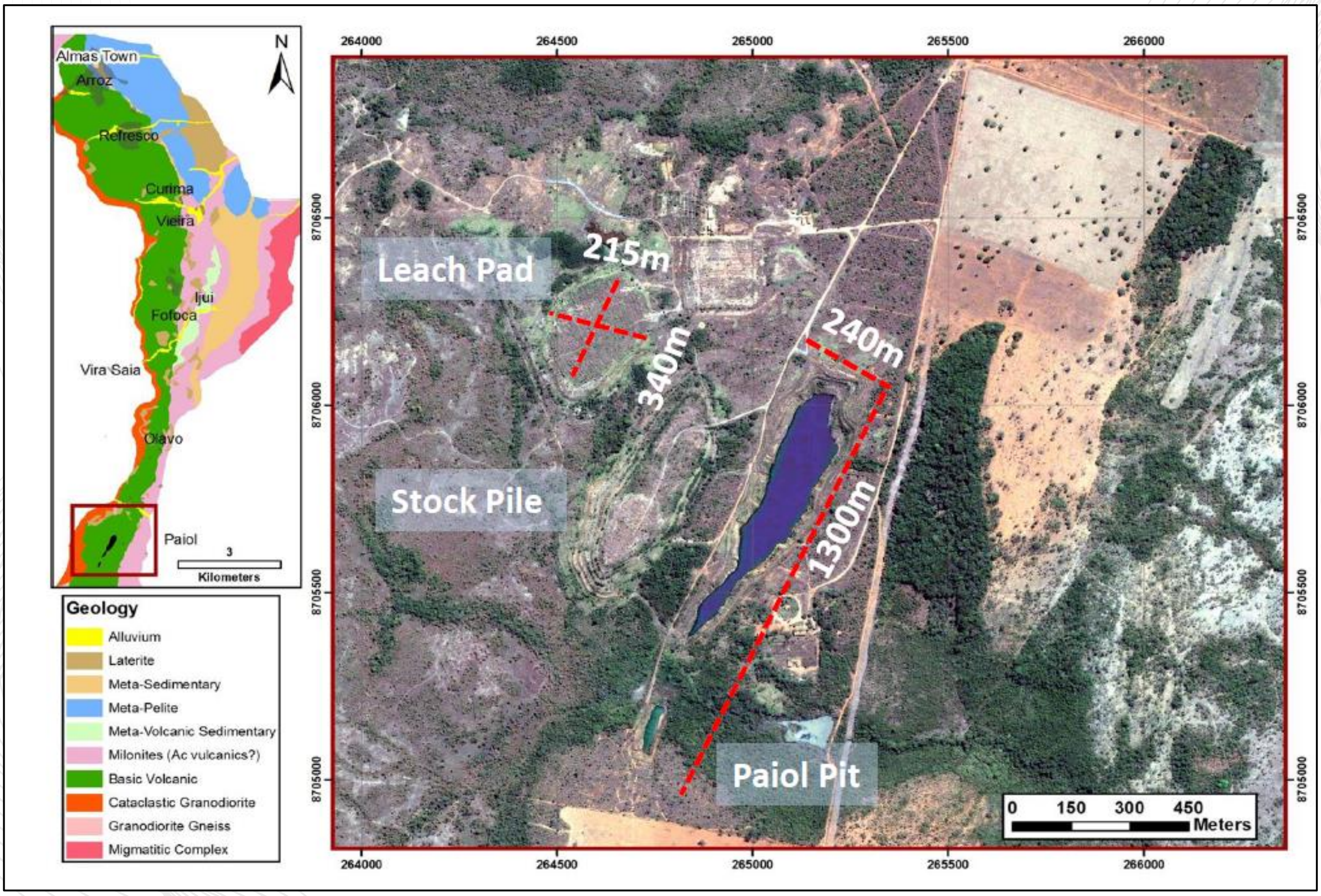
The mineralization is continuous and confined to certain parts of the hydrothermal alteration package. The higher potential of the deposit is in the north-central part of the deposit where an intersection between the main shear zone and a secondary north-south shear zone has been interpreted.

Hydrothermal alteration assemblages at Paiol show a symmetrical zonation inward from the margin to the center of the structure: chloritization, sericitization, carbonatization, albitization and silicification. Sulfide minerals occur as millimetric to sub-millimetric grains, not exceeding more than 5% by volume.

The paragenesis of hydrothermal minerals is represented by chlorite, sericite, ankerite, calcite, albite, quartz, pyrite, (pyrrhotite) and rare tourmaline - epidote. Locally, late stage calcite-pyrite veinlets represent the final stage of alteration. The alteration paragenesis is syn-to-late-orogenic, suggesting that hydrothermal events were concomitant with tectonism.

The main Paiol ore body has overall dimensions of approximately 650 m in the down dip direction, 1,300 m along strike and averages 27 m in thickness (Figure 7-4). Overlying bedrock is 1 to 5 m of brick red, argillaceous soil with sparse, weakly magnetic, pisolites. Beneath the soil horizon is 12 to 35 m of light red, yellow to ocher-colored saprolite, locally mottled and kaolinized.

Figure 7-4 Paiol Deposit Location & Boundaries



Locally relict textures and zones of deeply weathered rock may be preserved in the saprolite zone. The saprolite overlays 2 to 15 m of weathered and oxidized bedrock containing Mn-oxide minerals in fractures and on foliation planes. Locally boxwork texture after sulfide minerals is noted.

The Paiol geological map (Figure 7-5) was compiled by surface projection of drilling data displayed on cross sections of which Figure 7-6 and Figure 7-7 are an example. The map shows the continuity of the mineralization from North to South where the main alteration assemblage of albite-ankerite-quartz schist represents the center of the hydrothermal system, indicated in red. The cross sections show the thickness of the Paiol Deposit, with significant intersections with high-grade at greater depth (below 350 m).

The grade-thickness contour of the Paiol Deposit, shown in Figure 7-8, presents an important ore-shoot plunging at greater depth continuing towards North East.

The lithologies and their rock codes are presented in the Table 7-1.

Table 7-1 Paiol Deposit Lithologies and Rock Codes

ROCK CODE	LITHOLOGIES
ADQX	Albite-Ankerite-Quartz-Schist
CAAX	Chlorite-Albite-Amphibole-Schist
CCQX	Calcite-Chlorite-Quartz-Schist
QSCxb	Quartz-Chlorite-Sericite-Schist-Breccia
SCDX	Sericite-Chlorite-Ankerite-Schist
SDCX	Sericite-Ankerite-Chlorite-Schist
SDQX	Sericite-Ankerite-Quartz-Schist

Figure 7-5 Geological Map – Paiol Deposit

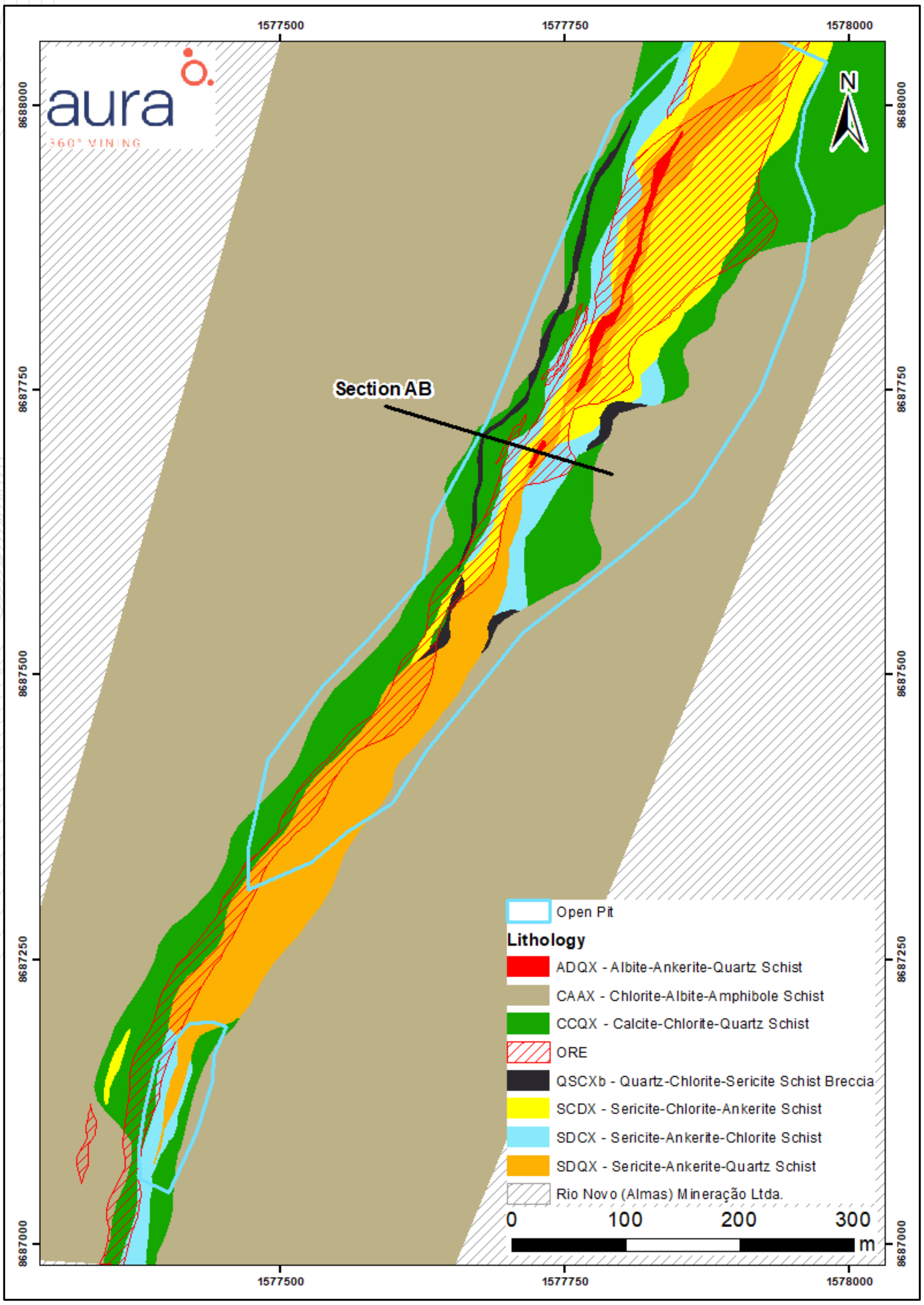


Figure 7-6 Geologic Cross Section A-B of Paiol Deposit

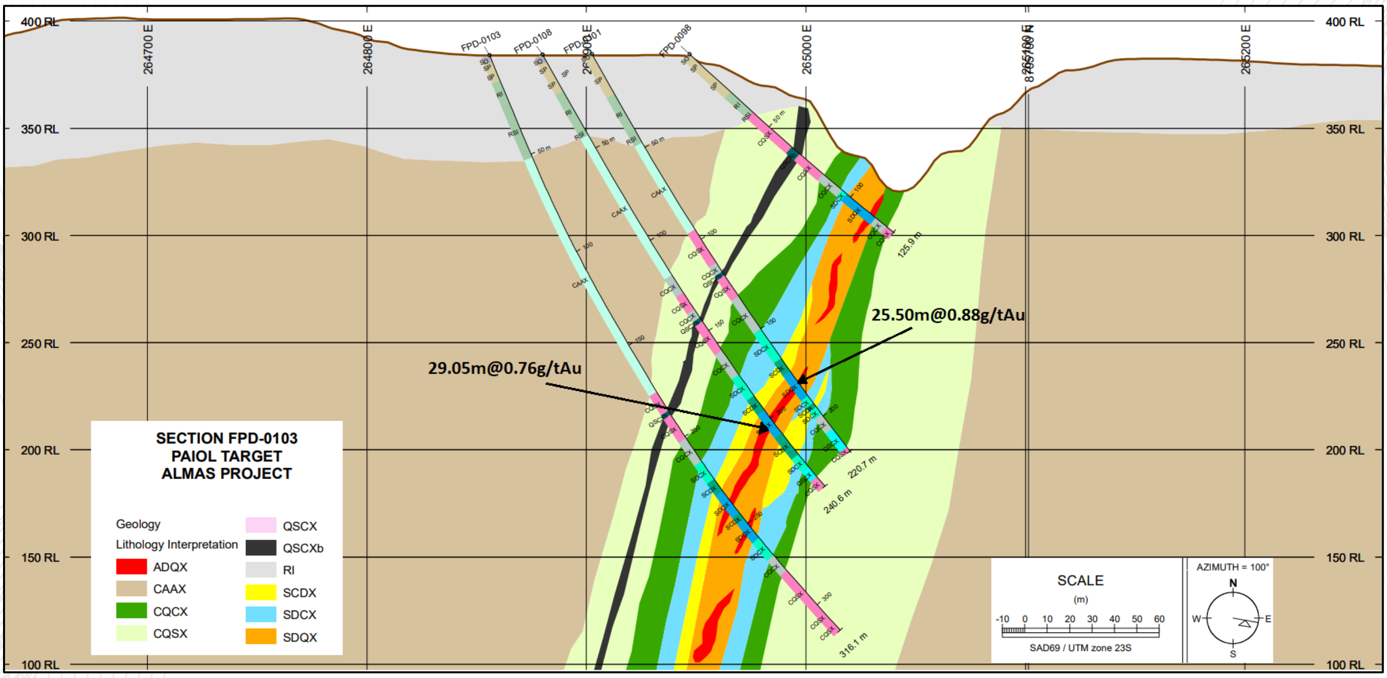


Figure 7-7 Geologic Cross Section of Paiol Deposit

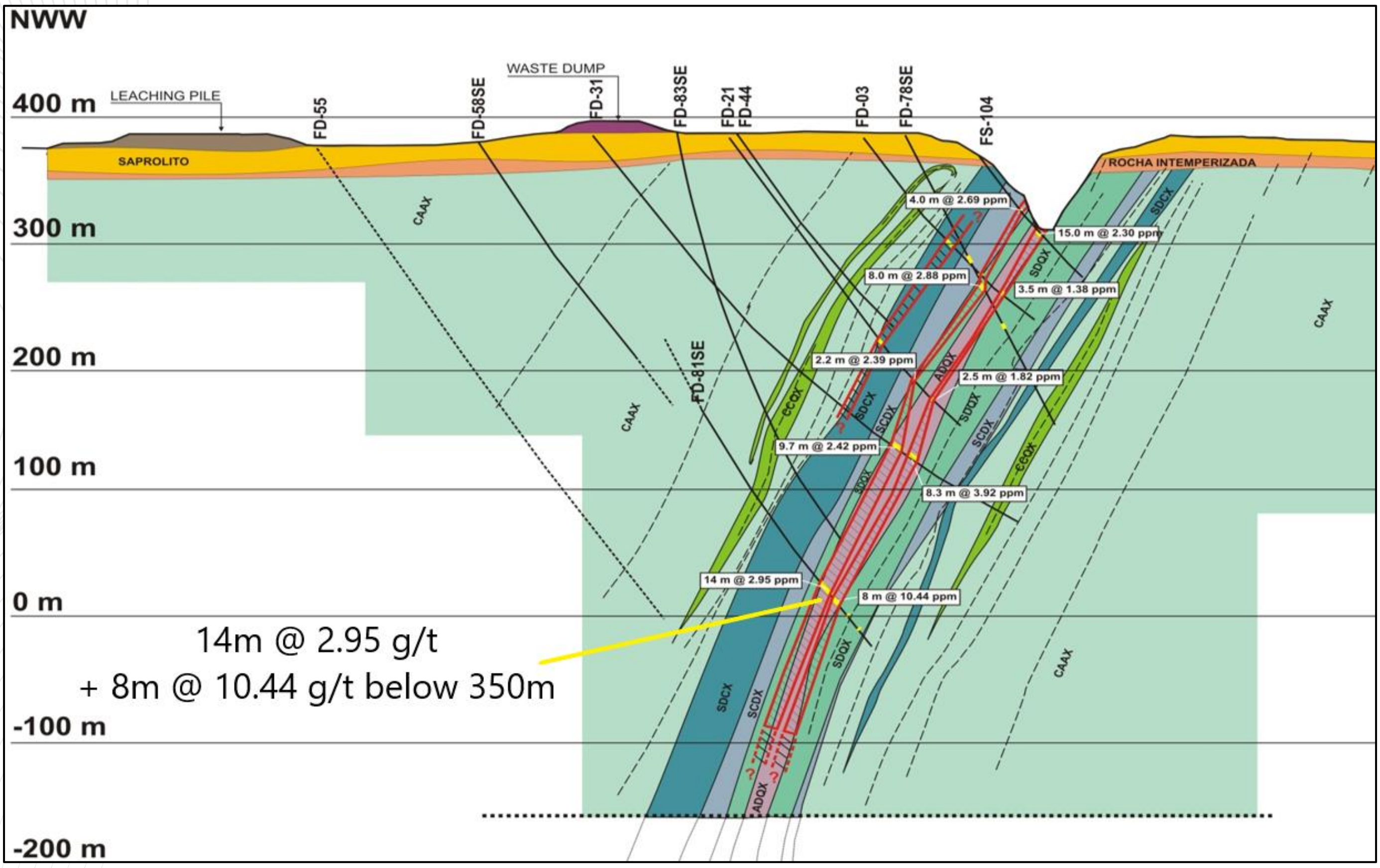
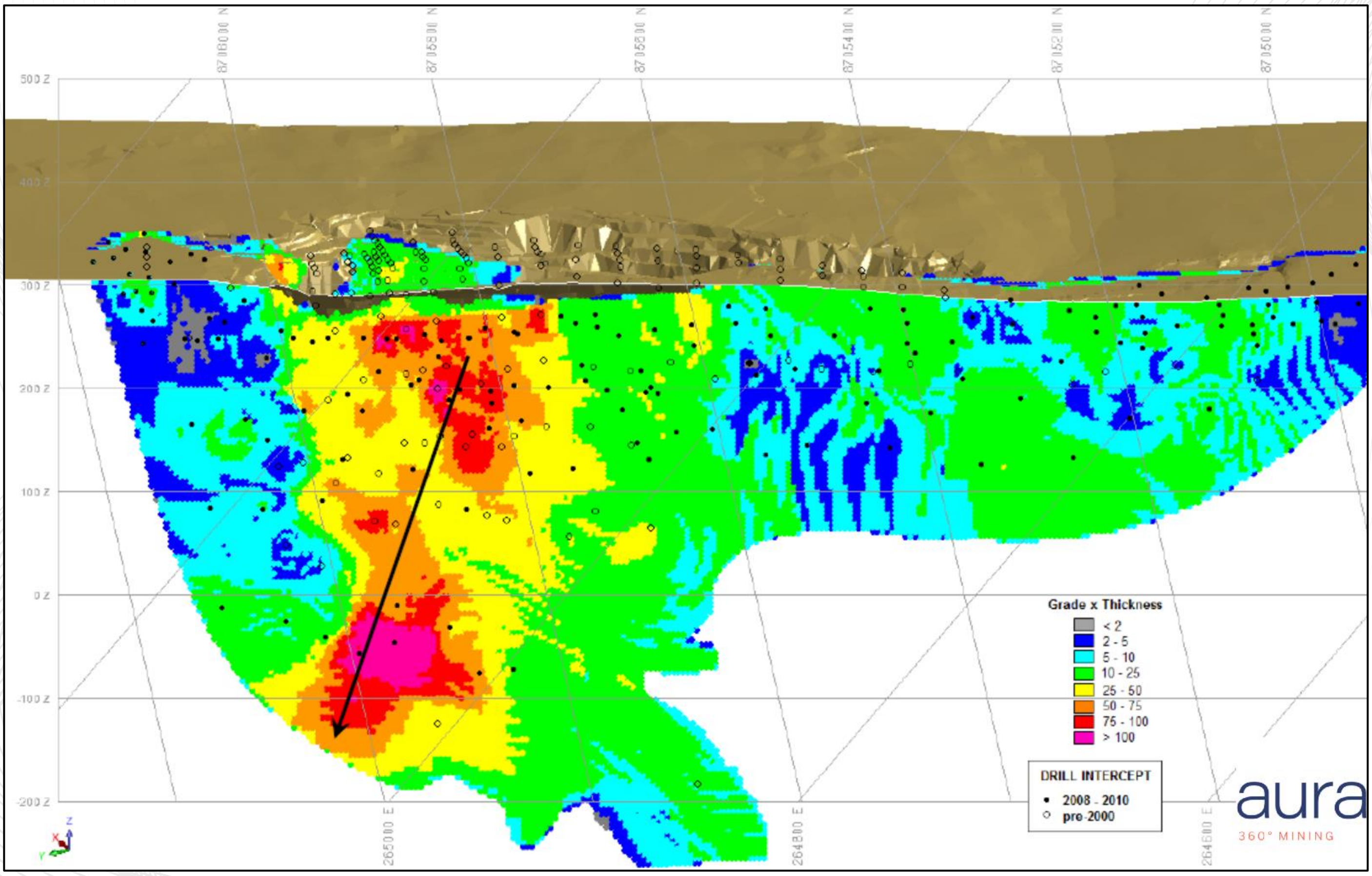


Figure 7-8 Grade-thickness section of Paiol Deposit



7.11 CATA FUNDA DEPOSIT

The Cata Funda deposit is situated in the northern portion of the Almas Greenstone Belt, immediately southeast of the Almas town site. The deposit is hosted in metabasic and metasedimentary rocks that display hydrothermal alteration processes such as sericitization, carbonization, albitization and silicification. Host rocks are in contact with siliceous breccias and quartz-carbonate schists to the west and with tourmaline-bearing quartzites and metapelites of the Morro do Carneiro Formation to the northeast.

The gold mineralization occurs primarily in the central portion of the structure which displays zoned alteration assemblages like that previously described for the Paiol deposit.

The Cata Funda deposit has overall dimensions of approximately 240 m in the down dip direction, 230 m along strike and averages 10 m in thickness (Figure 7-9) .

Overlying bedrock is typically 2 m to 6 m of red, argillaceous soil, weakly magnetic, with low percentages of quartz fragments and pisolites. Beneath the soil horizon is 8 to 30 m of red to yellow saprolite, locally sericitic and mottled containing Fe-Mn-oxides on relict foliations and fractures. The saprolite overlays 2 m to 6 m of weathered and partially decomposed bedrock within which decimeter-sized fragments of fresh rock are preserved.

The Cata Funda geologic map (Figure 7-10) was compiled by surface projection of drilling data displayed on cross sections of which Figure 7-11 and Figure 7-12 are typical examples. The map shows several mineralized bodies displaced by folds and faults. The strongest gold mineralization at Cata Funda is associated with the schistose, sericite-ankerite- quartz (SDQX) alteration assemblage, indicated in orange. The geological cross-sections shows the significant thickness and grade continuity of the ore body at depth.

The lithologies and their rock codes used are presented in the Table 7-2.

Table 7-2 Cata Funda Lithologies and Rock Codes

ROCK CODE	LITHOLOGIES
BQZ	Quartz-Breccia
CAAX	Chlorite-Albite-Amphibole-Schist
MD	Meta Dacite
MV	Meta Volcanic
QCCX	Quartz-Carbonate-Chlorite-Schist
QSX	Quartz-Sericite-Schist
SBX	Sericite-Schist-Carbonaceous
SCDX	Sericite-Chlorite-Ankerite-Schist
SDCX	Sericite-Ankerite-Chlorite-Schist
SDQX	Sericite-Ankerite-Quartz-Schist

Figure 7-9 Cata Funda Deposit Location and old pit outcrop

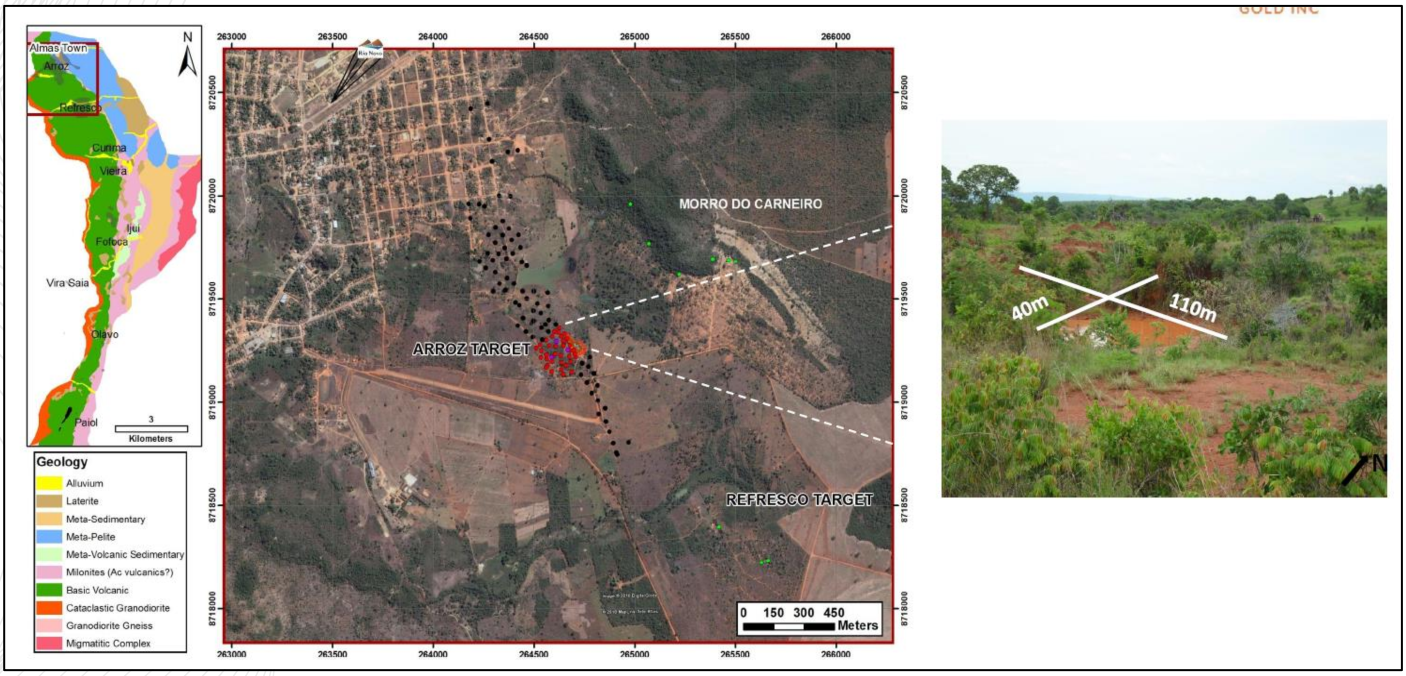


Figure 7-10 Geological Map – Cata Funda Deposit

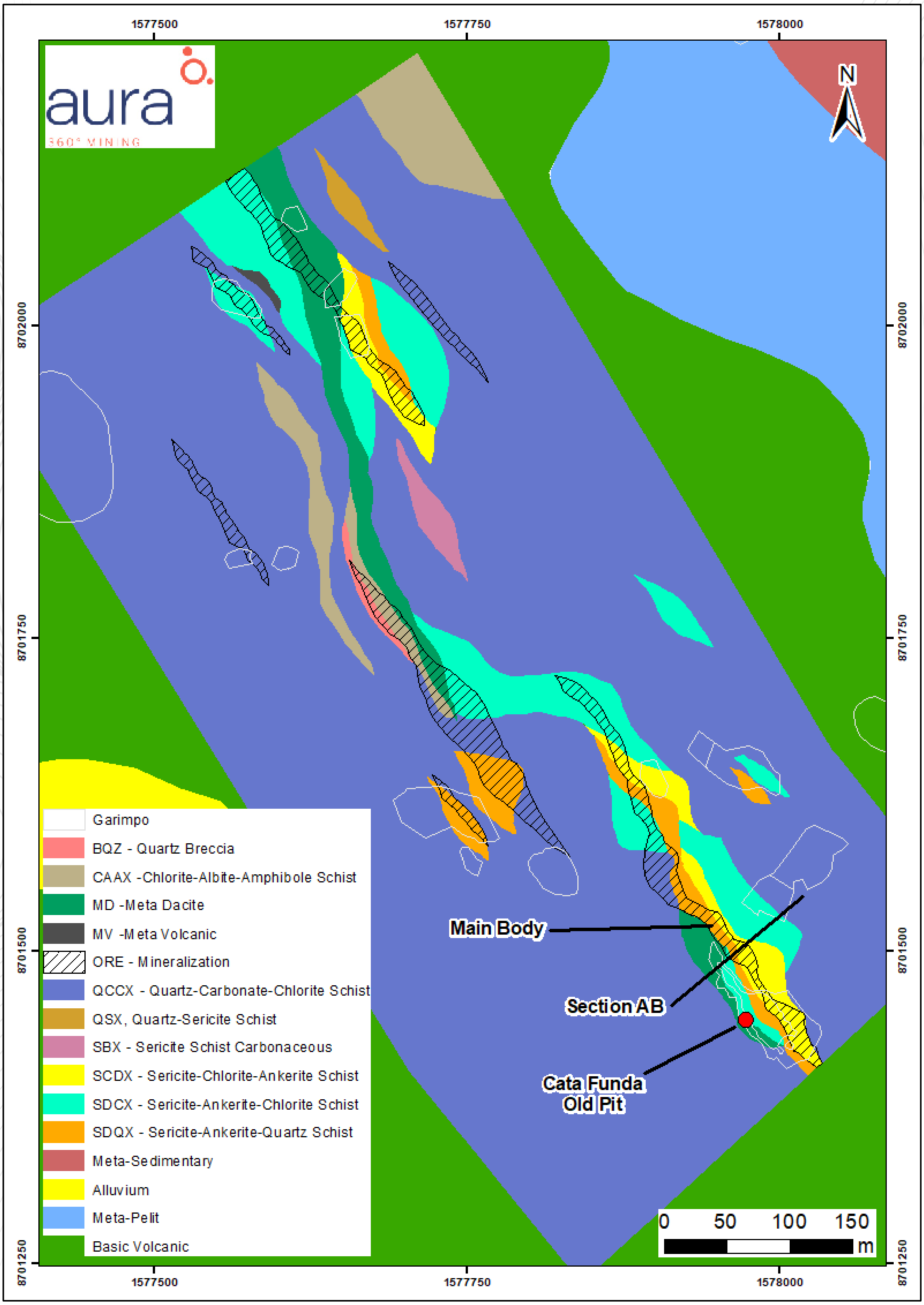


Figure 7-11 Geological Cross Section A-B of Cata Funda Deposit

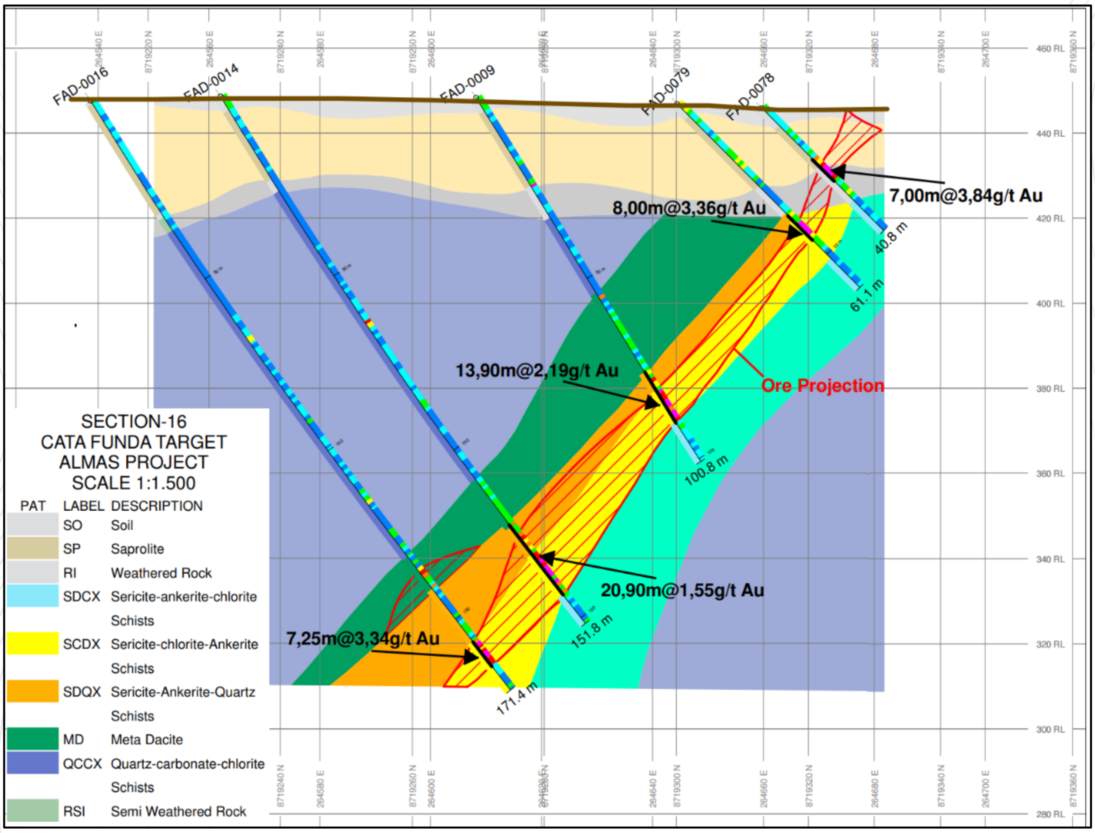
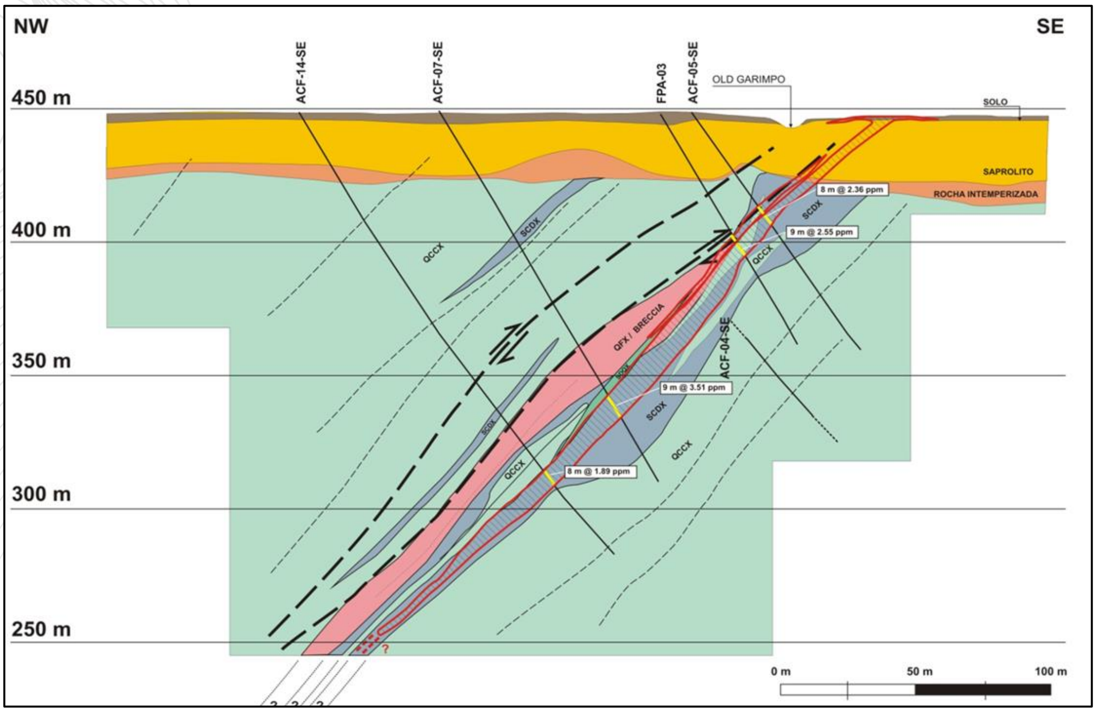


Figure 7-12 Geological Cross Section of Cata Funda Deposit



7.11.1 HYDROTHERMAL ALTERATION UNITS AT PAIOL AND CATA FUNDA

While the contacts between the various zoned, hydrothermal alteration units developed in metabasic rocks at Paiol and Cata Funda are gradational, the main units from these deposits are described and illustrated in Figure 7-13 through Figure 7-15.

Dark green, foliated rock of mafic composition, showing fine to medium grained, grano-nematoblastic texture with minor calcite, epidote and sphene. Lenses of porphyritic metadacite are locally present in the unit. This unit represents the country rock for the mineralized package and contains an average grade of 0.03 g/t Au.

Green, foliated rock with fine-grained grano-lepidoblastic texture, showing millimetric quartz-calcite bands, commonly is broken and boudinaged amid centimetric chlorite-rich bands. The foliation is planar to anastomosing, locally micro folded and transposed with chlorite enrichment in the axial planes. A photomicrograph of this unit is given in the section on Deposit Mineralization. This unit represents the outer-most zone of the hydrothermal package and has an average grade of 0.21 g/t Au.

The next unit is greyish-green in color with fine grano-lepidoblastic texture with notable alternation of millimetric sulfide- bearing, quartz-carbonate bands and centimetric micaceous bands. The micaceous bands are sub-parallel to a planar foliation that locally transposes relict folds. The quartz-carbonate bands have sharp contacts and are surrounded by phyllosilicate minerals in porphyroclasts. This texture is present in all other lithotypes. This unit has an average grade of 0.58 g/t Au.

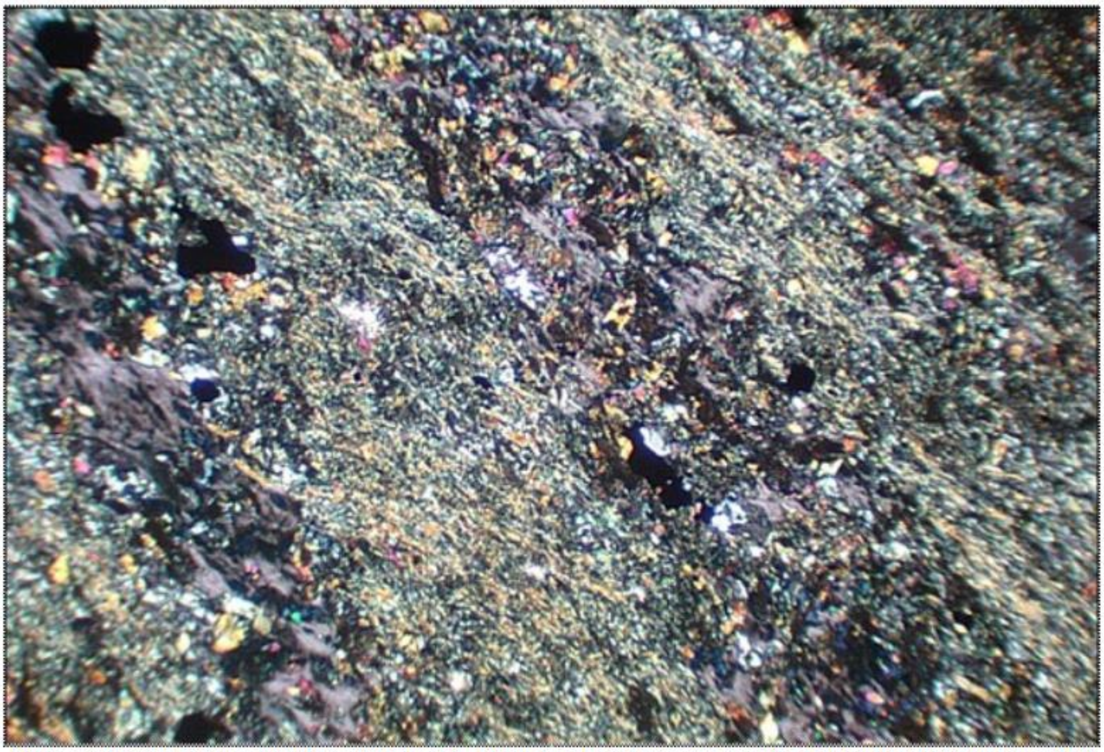
Next is a unit which is grayish green in color with fine granolepidoblastic texture. It shows predominant occurrences of quartz-ankerite bands over the micaceous bands with subordinate amounts of albite and calcite. Pyrite volume varies from 0.3% to 5% and is associated with the quartz-ankerite bands as aggregates and millimetric grains. This unit has an average grade of 1.29 g/t Au.

The next unit is grey in color with fine granolepidoblastic texture, showing quartz-ankerite-albite bands enveloped by thin micaceous films that define an anastomosed foliation. Aggregates of submillimeteric pyrite, 1% to 5% by volume are subparallel to the main foliation. The average grade is 2.31 g/t Au.

The final rock unit is grey in color with fine to medium granoblastic texture. Anastomosed, insipient to nebulitic foliation, locally emphasized phyllosilicates films. Rare microporphyritic quartz grains appear with strong recrystallization of unit. Pyrite is 1% to 5% by volume with subordinate chalcopyrite that is generally disseminated or locally concentrated in millimetric bands. The unit may show well-developed pressure shadows filled with quartz and rarely chlorite. This unit occurs mainly in the north central part of the ore body. The average grade of the unit is 4.29 g/t Au.

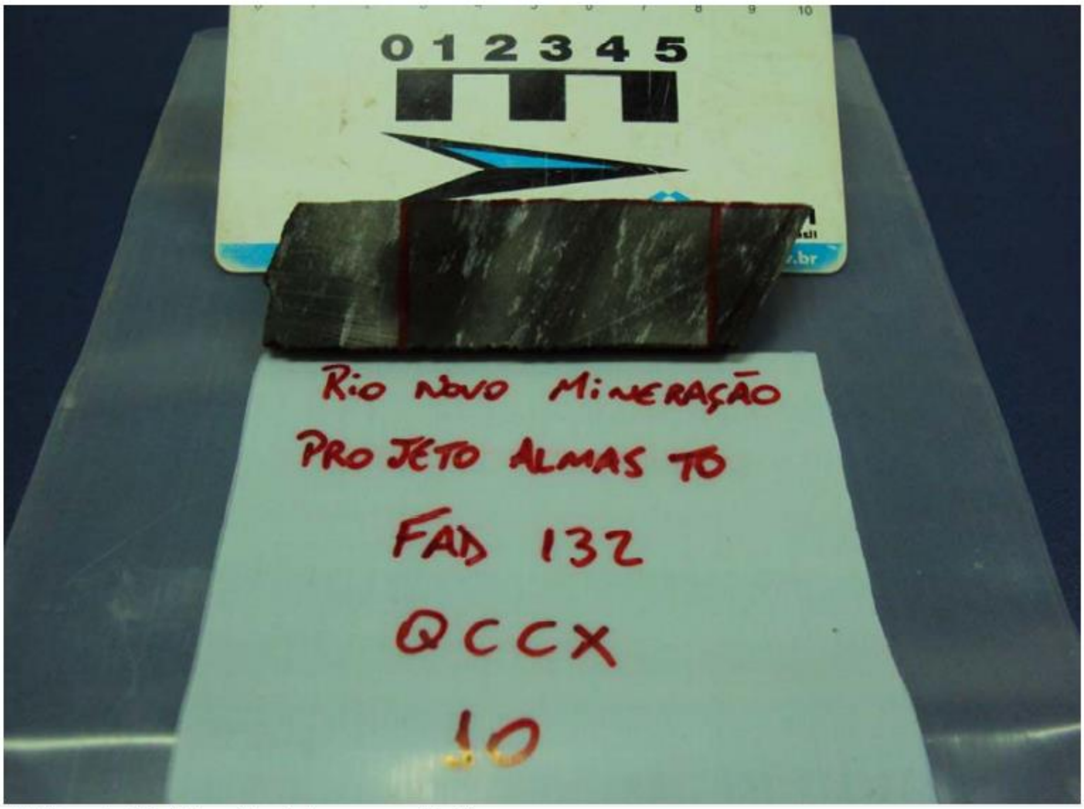
Figure 7-13 Mafic Schists - Paiol - Cata Funda Area

A



Photomicrography of CAAX. Sample A-58168 (LA-12) - Plagioclase-Epidote-Quartz-Chlorite-Hornblende Schist-Metabasalt (?). Epidote-chlorite-rich bands and hornblende-rich bands. 25X, crossed nicols.

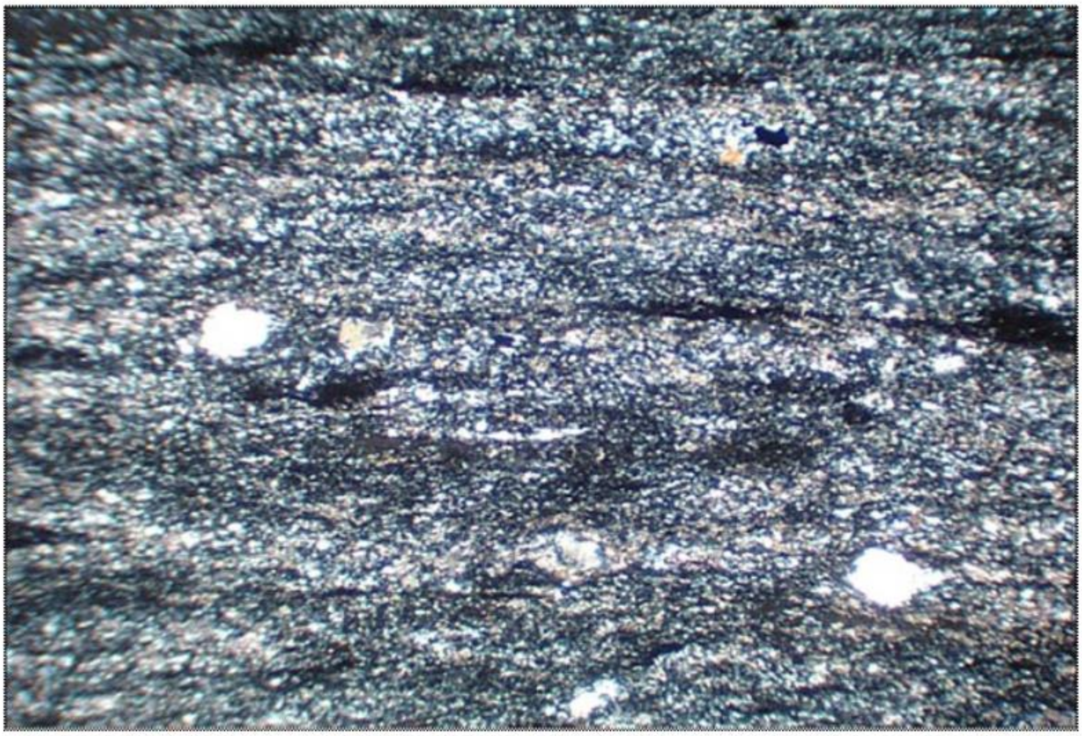
B



CCQX sampled for thin section study purposes (LA-10).

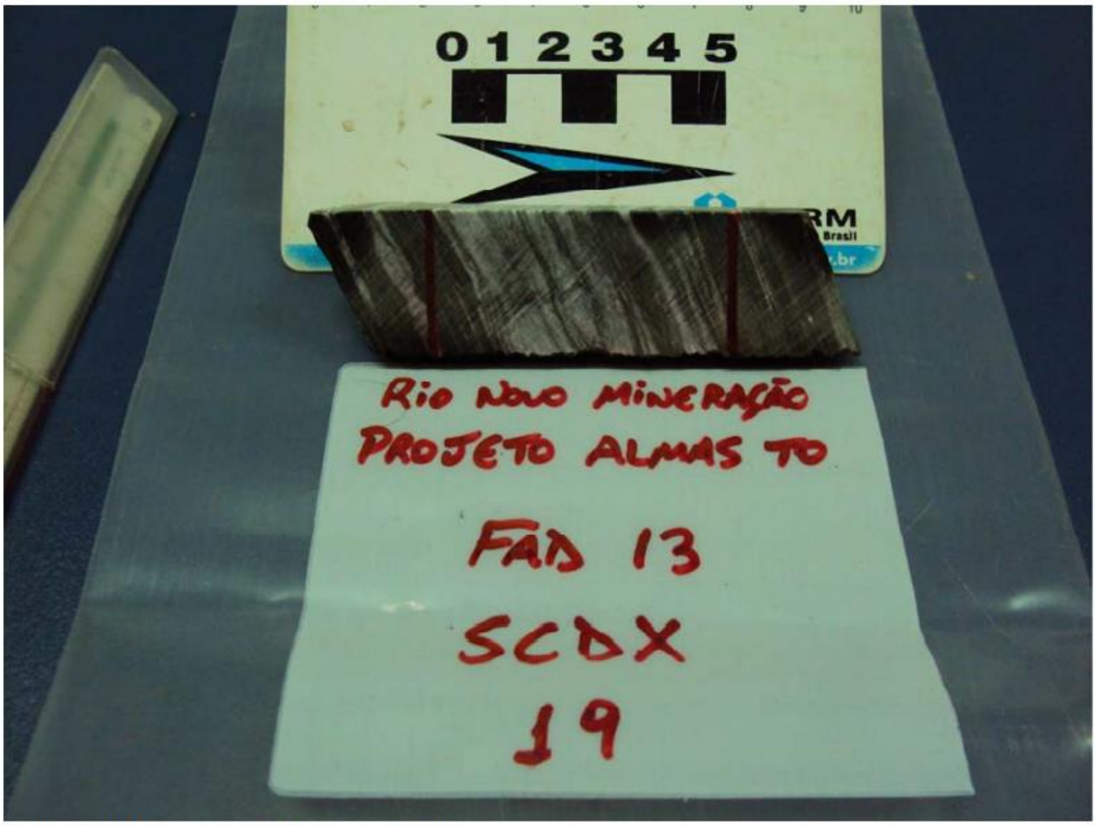
Figure 7-14 Sericite, Chlorite, Carbonate Schist - Paiol, Cata Funda Area

A



Photomicrography of SCDX. Sample A-56663 (LA-24) - Sericite-Chlorite-Carbonate-Quartz Schist. Quartz microporphyroclasts ("quartz eyes"), possibly relicts from the original volcanic porphyritic texture, enclosed by an altered groundmass composed essentially of quartz, chlorite, sericite and carbonate. 25X crossed nicols.

B



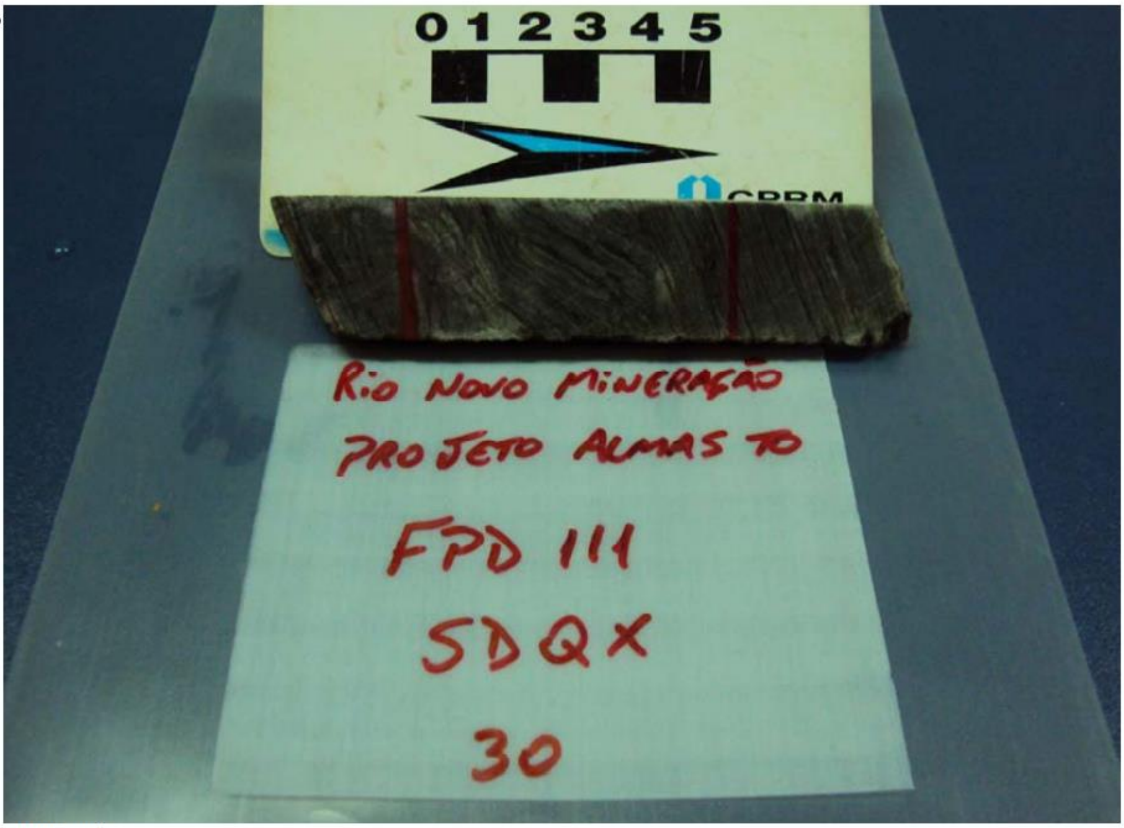
SCQX sampled for thin section study purposes (LA-19)

Figure 7-15 Quartz Eye, Sericite - Ankerite Quartz Schist

A

Photomicrography of SDQX. Sample A-31734 (LA-30). Quartz porphyroclast ("quartz eye"), possibly relict from the original volcanic porphyritic texture, enclosed by an altered and sheared groundmass composed essentially of quartz and carbonate.
Hematite (opaque) along fractures. 25X, crossed nicols.

B



SDQX Sample.

7.11.2 GOLD MINERALIZATION: PAIOL AND CATA FUNDA

Gold mineralization occurs normally in the center of the alteration zone, associated with albite-quartz-ankerite (calcite) and the sulfide minerals, pyrite, chalcopyrite and pyrrhotite, as shown on the following polished sections (Figure 7-17 and Figure 7-18).

At Paiol and Cata Funda individual ore-shoots are shaped as lenses and/or anastomosing bodies within the shear zone. The mineralization shoots typically are steeply dipping and plunging lenses.

Some coarser-grained gold has been observed at Cata Funda (Figure 7-16) where it occurs primarily in quartz-carbonate veins within albite-sericite-pyrrhotite alteration envelopes developed in mafic to intermediate metavolcanic host rocks.

Figure 7-16 Visible Gold in Cata Funda

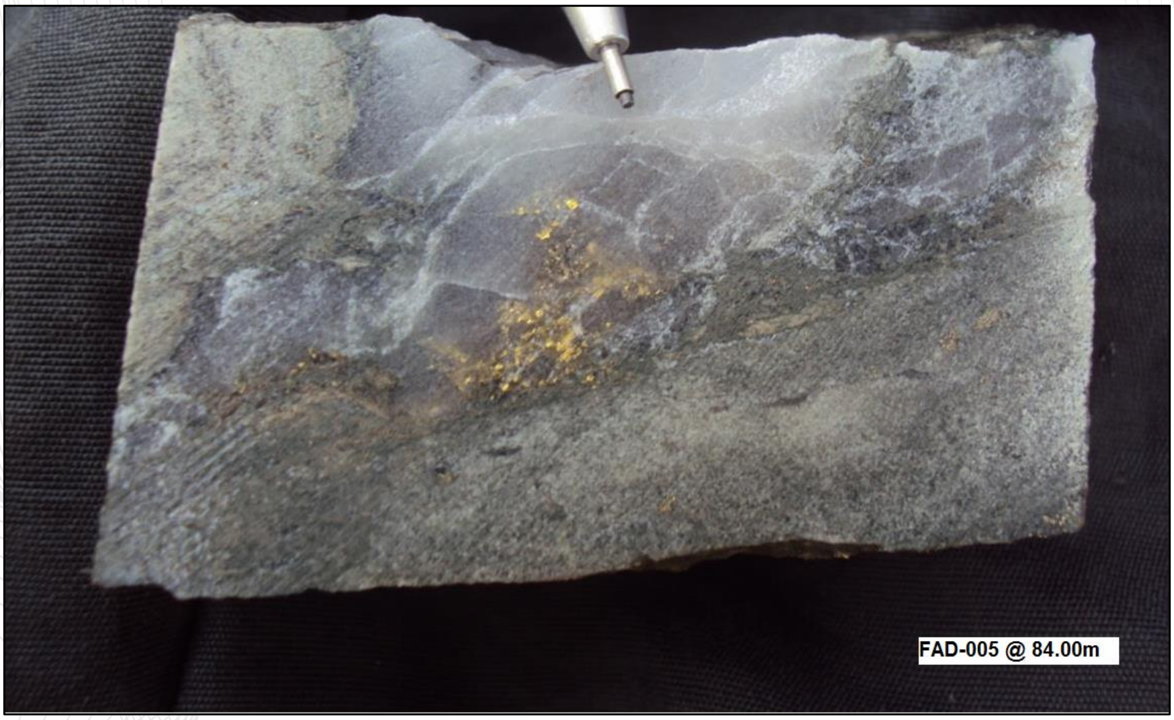
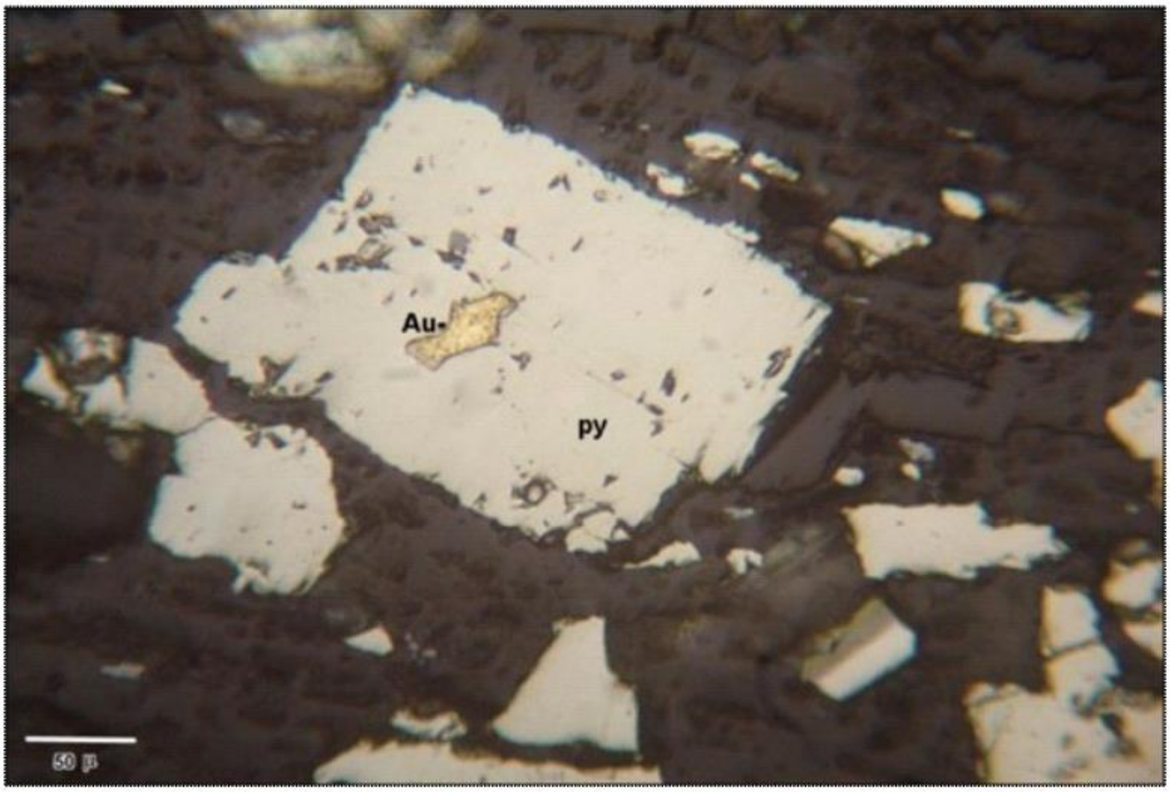


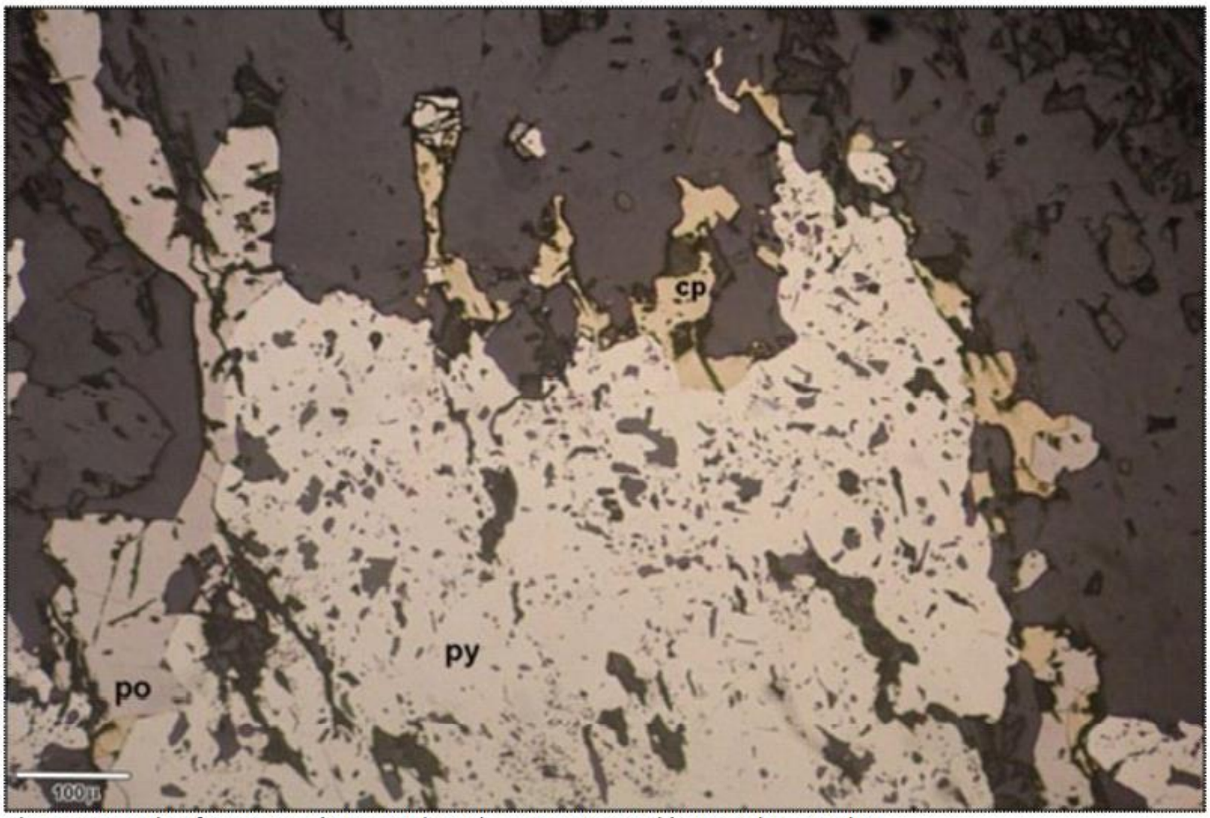
Figure 7-17 Gold Mineralization in Paiol

A



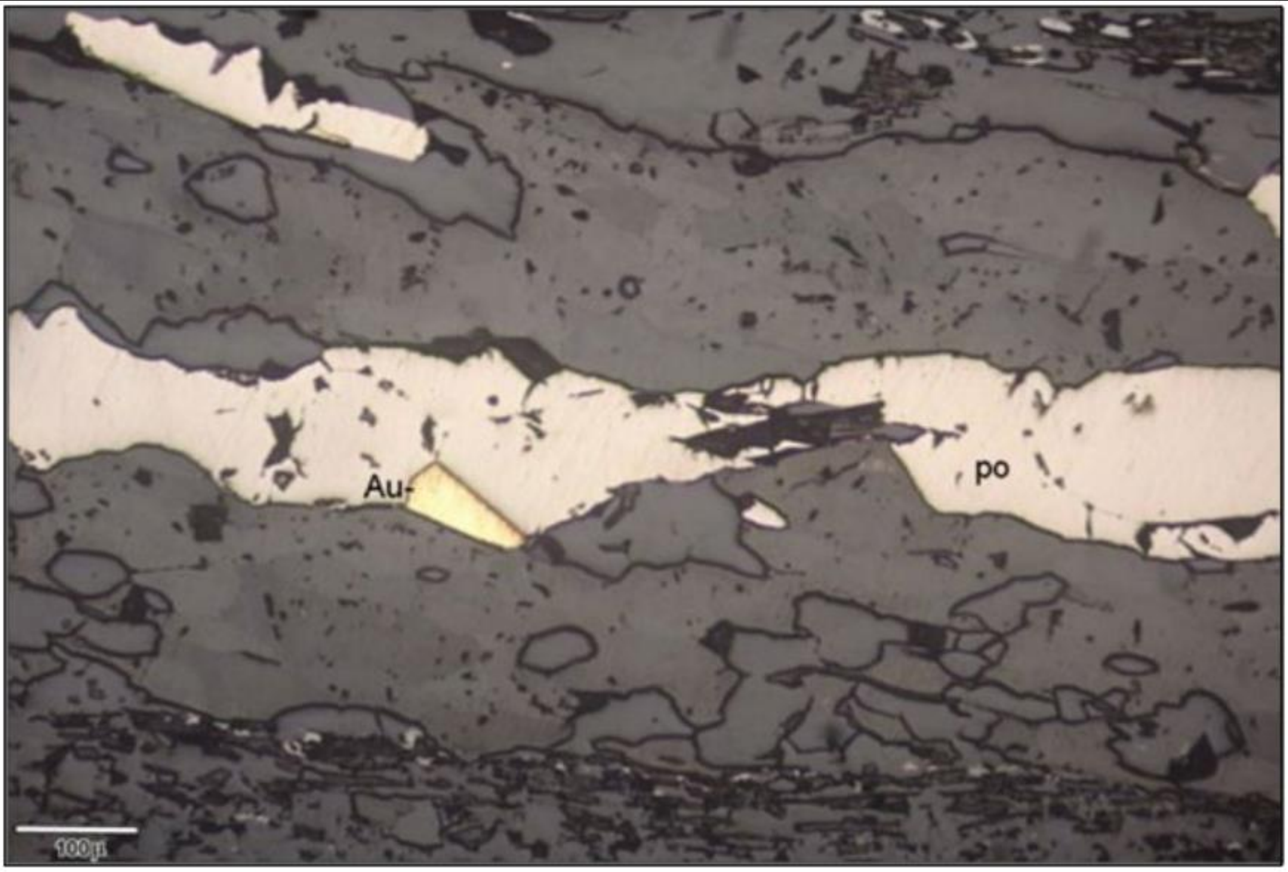
Photomicrography of ADQX. Sample A-31744 (LA-31). Native gold (Au) in pyrite (py). 200X, plane reflected light.

B



Photomicrography of SDQX. Sample A-57755 (LA-05). Quartz-Sericite-Chlorite-Carbonate Schist.
Pyrrotite (po), chalcopyrite (cp) and pyrite (py). 100X, plane reflected light.

Figure 7-18 Gold Mineralization – Cata Funda



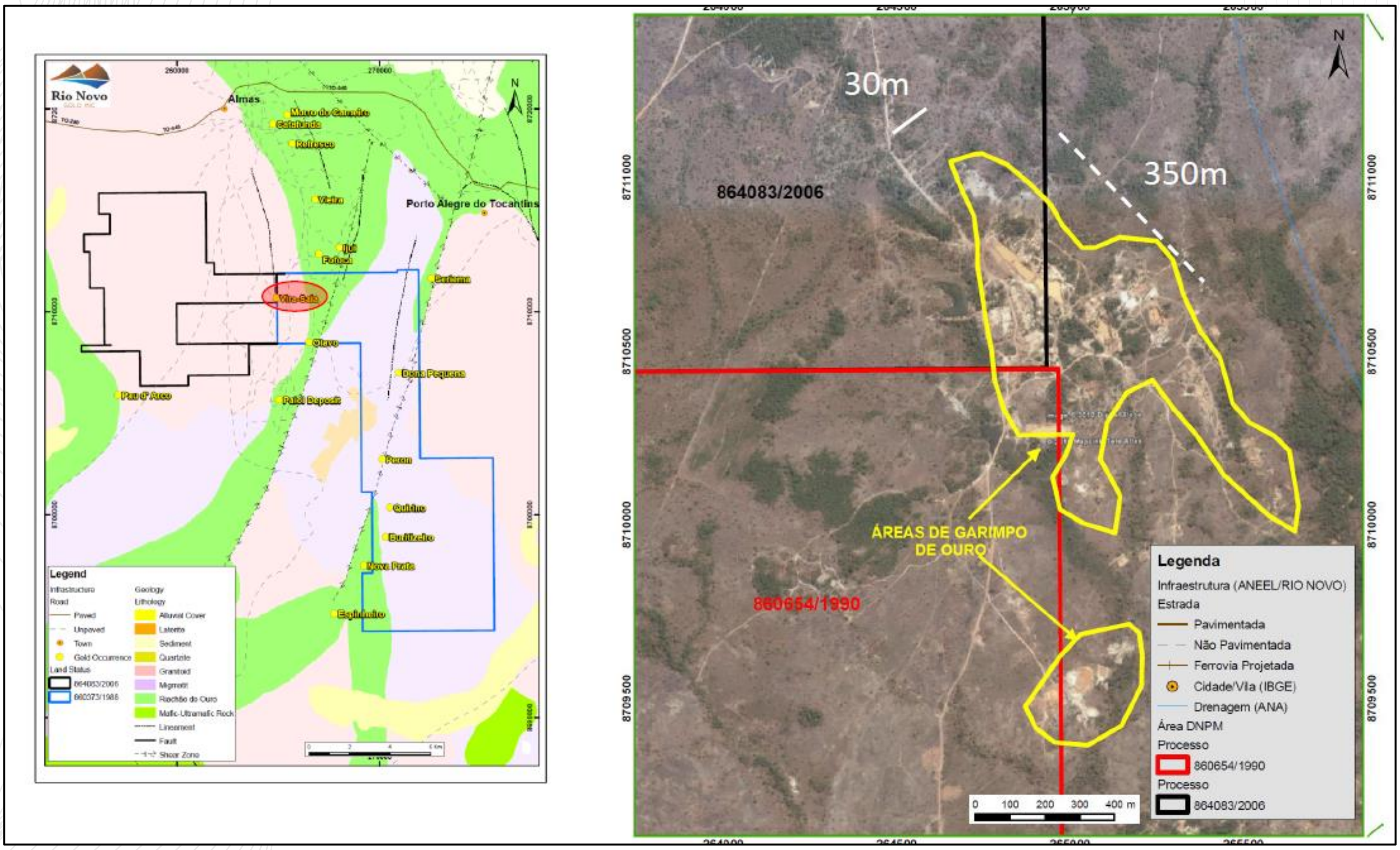
7.12 VIRA SAIA DEPOSIT

The Vira Saia deposit is hosted in the granitic gneiss complex. These complexes are composed of isotropic to weakly-foliated granitoid plutons which have been variably classified as tonalites, trondhjemites, granodiorites, quartz monzonites, amphibole-quartz diorite and monzogranites. A second granitoid suite composed of the same lithologies but containing biotite as the primary mafic mineral, is recognized in the region.

At Vira Saia, a shear zone (N45°W) developed in granodiorite controls brecciation, alteration and gold mineralization.

The main Vira-Saia deposit body has overall dimensions of approximately 200 m in the down-dip direction, 350 m along strike and averages 15 m in thickness (Figure 7-19). Exploration has also identified three smaller zones designated: East Body, NW Body and NW Extension Body.

Figure 7-19 Vira-Saia Deposit Boundaries and Location



Hydrothermal alteration is well developed, the intensity of which is proportional to the intensity of deformation in the granitic host rock. The outer-most alteration zone is foliated and characterized by the appearance of muscovite, albite and epidote. In an intermediate alteration zone, muscovite and albite still occur but are now associated with calcite and sulfide minerals, up to 1% by volume. Interfoliated quartz and recrystalized quartz veins with strong sericite on vein selvages occurs in the core of the alteration zone. Sulfide mineralization is more intense in the central part of the alteration zone where very fine-grained pyrite occurs as inclusions in quartz and muscovite grains.

Overlying bedrock is 0 m to 3 m of sandy soil, orange to grayish brown in color, rich in subangular quartz and feldspar gravel. Beneath the soil horizon is 0 to 7 m of yellowish-white saprolite preserving epidote, sericite and Mn-oxides on relic foliation planes. The saprolite typically overlies 0 to 5 m of deeply weathered and oxidized, fractured granodiorite.

7.12.1 LITHOLOGIC UNITS AT THE VIRA SAIA DEPOSIT

The Vira-Saia geologic map (Figure 7-20) was compiled from surface geologic mapping and surface projection of geology from cross sections of which Figure 7-21 is an example. The map shows several mineralized bodies displaced by faults. The strongest gold mineralization is associated with quartz veins and mylonitic granodiorite.

At Vira-Saia detailed geological mapping identified gradational zones of mylonitization within the granodiorite, which, with increasing levels of deformation, resulted in the progressive formation of protomylonites (GDP), mylonites (GDM) and ultramylonites (QSX).

Dikes of mafic, pegmatitic and aplitic composition intrude the main granodiorite body. A short description of each rock type is presented below.

- DM – Mafic Dikes

Mafic dikes are dark grey, chlorite-rich and typically undeformed to weakly deformed and barren of mineralization. They cut the intrusives at approximately a north-south orientation.

- PGT - Pegmatite

Pegmatite dikes are very common in the region. They are typically orange to white in color with abrupt, irregular contacts with the enclosing granodiorite. Locally quartz, feldspar crystals attain 5 cm in length. These rocks are generally barren.

- GDT – Granodiorite

The most common rock type at Vira Saia is granodiorite. It is typically pink to light grey in color with igneous textures in undeformed to weakly examples. Biotite is present with accessory hornblende and sphene. Granodiorite is typically barren to weakly mineralized.

- GDT – Protomylonite

Protomylonite develops in granodiorite adjacent to shear zones. While this unit displays a mylonitic foliation, some igneous textures are still preserved in a rock whose mineralogical composition remains granodioritic. The foliation generally approaches the attitude of the N45°W shear zone. Protomylonite is typically only weakly mineralized with an average grade of 0.14 g/t Au.

- GDT – Mylonite

With increasing strain in proximity to shear zones, granodiorite is converted to mylonite, a greenish grey rock in which the grain size of mafic minerals is significantly reduced, and intra-foliar calcite and white mica are formed as a result of hydrothermal alteration. The foliation assumes the attitude of the shear zone (N45°W) and tends to dip 50° to 60°. The sulfide content is typically less than 1% by volume. This rock is mineralized having an average grade of 0.38 g/t Au.

- QSX – Ultramylonite - Quartz Sericite Schist

Ultramylonites are found in the center of shear zones where they have been intensively deformed, altered and represent the high-grade core of the Vira Saia gold deposit. They are lime green in color and are composed primarily of greenish white mica and interfoliated fine grained quartz, which is usually boudinaged and brecciated. The sulfide content is typically less than 2% by volume. At Vira Saia, ultramylonites exhibit the highest average grades (1.3 g/t Au) associated with free gold and quartz.

The lithologies and their rock codes are synthetized in the Table 7-3.

Table 7-3 Vira Saia Lithologies and Rock Codes

ROCK CODE	LITHOLOGIES
DM	Mafic Dikes
PGT	Pegmatite
GDT	Granodiorite
GDP	Protomylonite
GDM	Mylonite
QSX	Ultramylonite - Quartz Sericite Schist

Figure 7-20 Vira-Saia Deposit Geological Map

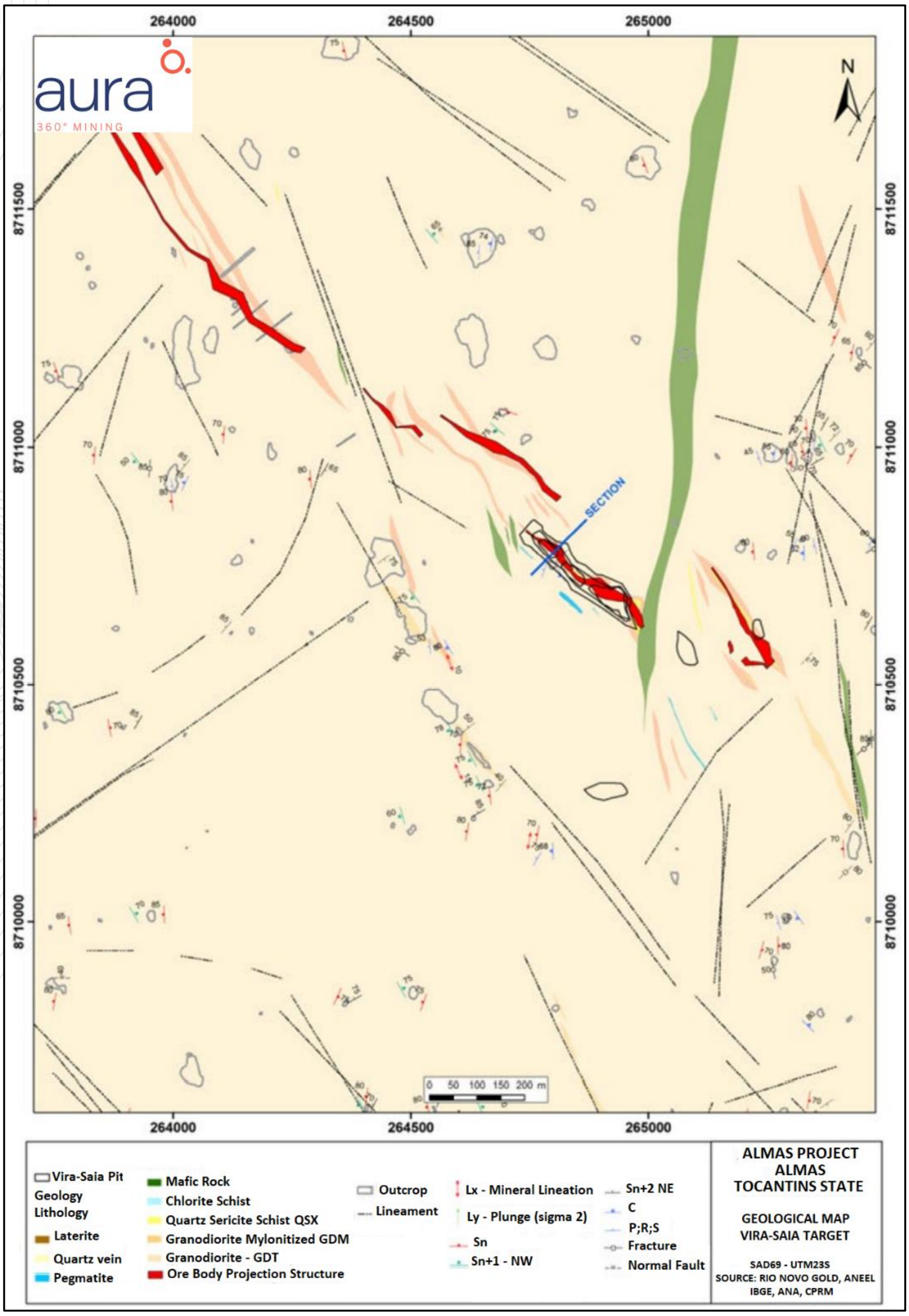
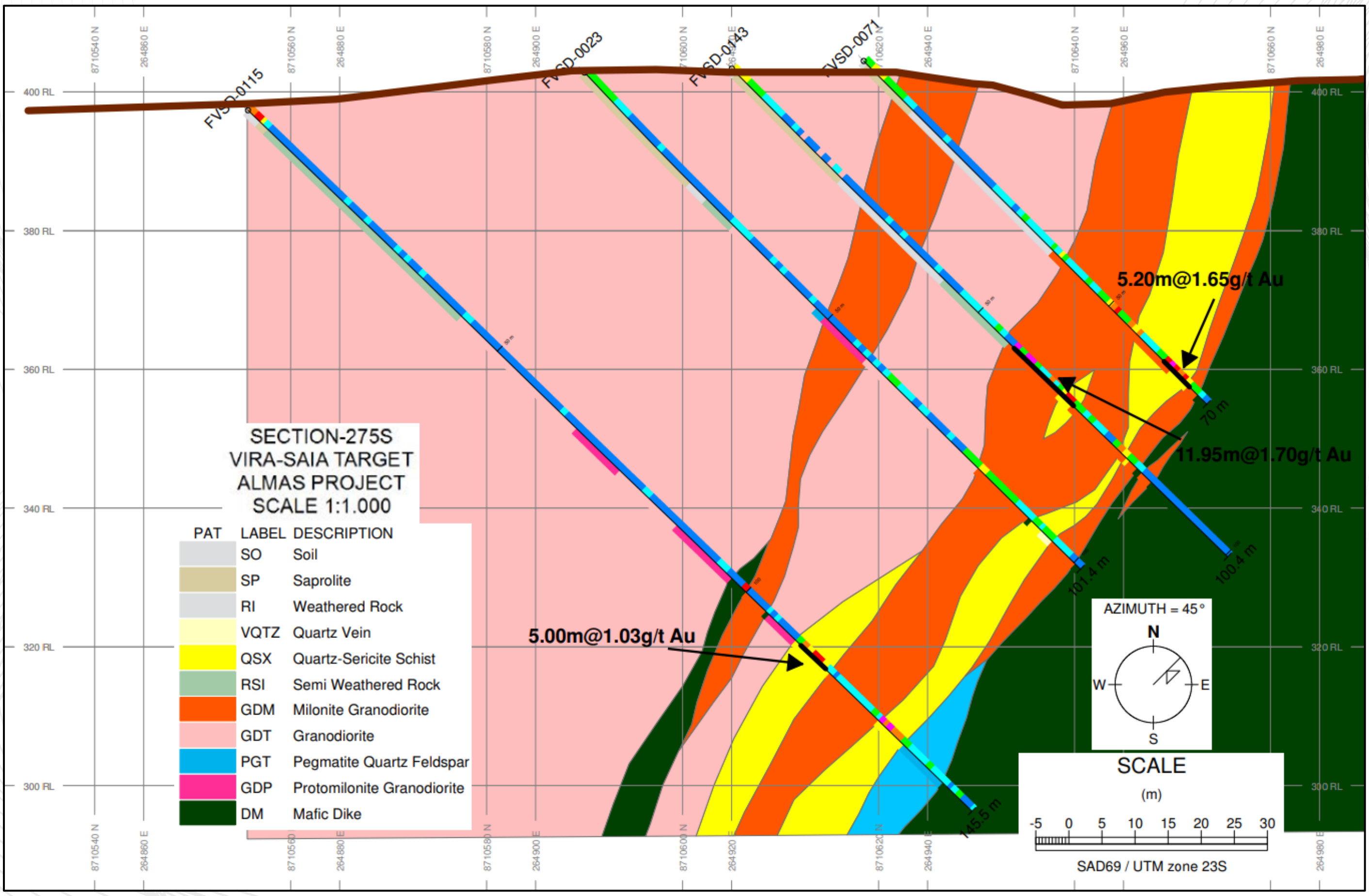


Figure 7-21 Vira Saia Deposit Geological Cross Section



7.12.2 GOLD MINERALIZATION: VIRA SAIA DEPOSIT

At Vira Saia gold is closely associated with sulfide-bearing, quartz-sericite-rich ultramylonites formed in the core of shear zones developed in granodiorite. Chalcopyrite and galena are very rare. The intensity of the hydrothermal alteration is proportional to the progressive deformation in the shear zone. Quartz veins typically have saccharoidal (sugary) textures, believed to have formed by dynamic crystallization in shear zones, suggesting a syntectonic timing of vein formation. The Vira-Saia deposit belongs to the lode gold, orogenic deposit type, with predominant quartz-sericite-carbonate alteration surrounding quartz veins with low iron sulfide content (<2%).

7.13 OTHER EXPLORATION TARGETS

In addition to the presented targets the Almas Gold Project is comprised of other important targets with confirmed gold mineralization, some of which with garimpo activity. All are located in the same district and within the Almas Belt.

7.13.1 OLAVO TARGET

Inserted in the same Belt, the Olavo target is located 2.5 km southeast from the Vira Saia Deposit and 3 km northeast of the Paiol Deposit. It is hosted within the same meta-mafic to meta-ultramafic rocks of Córrego do Paiol Formation, occurring as mylonite or schists with hydrothermal alteration marked by with chlorite and sericite. The gold mineralization is confirmed by soil and rock sampling and by exploration diamond drilling in Quartz Breccia (BQZ), Sericite-Chlorite-Ankerite Schist (SCDX), Sericite-Ankerite-Chlorite-Schist (SDCX), Calcite-Chlorite-Quartz-Schist (CCQX) and Chlorite-Albite-Amphibole-Schist (CAAX). Figure 7-22 presents the geological map of Olavo Target. The lithologies of Olavo target and their rock codes are summarized in Table 7-4.

Table 7-4 Olavo Target Lithologies and rock codes

ROCK CODE	LITHOLOGIES
CAAX	Chlorite-Albite-Amphibole-Schist
CCQX	Calcite-Chlorite-Quartz-Schist
BQZ	Quartz Breccia
QSX	Ultramylonite - Quartz Sericite Schist
SCDX	Sericite-Chlorite-Ankerite Schist
SDCX	Sericite-Ankerite-Chlorite-Schist

7.13.2 FOFOCA AND IJUÍ TARGETS

Continuing the same trend inside Córrego do Paiol Formation, the Fofoca and Ijuí targets are located 4.5 km north of Olavo target and around 3 km northeast from Vira Saia Deposit. They are also hosted within mylonite and schist rocks with chlorite and sericite alteration. Gold mineralization is confirmed by soil and rock sampling and exploration diamond drilling in Quartz-Sericite-Schist (QSX). Both target locations are shown in Figure 7-23. Their lithologies and rock codes are summarized in Table 7-5.

Table 7-5 Fofoca and Ijuí Targets Lithologies and rock codes

ROCK CODE	LITHOLOGIES
CAAX	Chlorite-Albite-Amphibole-Schist
CCQX	Calcite-Chlorite-Quartz-Schist
CQSX	Carbonate-Quartz-Chlorite-Schist
DT	Tonalitic-Dike
GDM	Mylonite
MD	Mafic Dike
MV	Meta-Dacite
QCCX	Quartz-Carbonate-Chlorite-Schist
QSCXb	Quartz-Chlorite-Sericite-Schist breccia
QSX	Quartz-Sericite-Schist
QTZ	Quartzite
QX	Quartz-Schist
SBX	Sericite-Carbonaceous-Schist
SDCX	Sericite-Ankerite-Chlorite-Schist
TUQ	Whitish Quartzite

Figure 7-22 Olavo Target Geological map

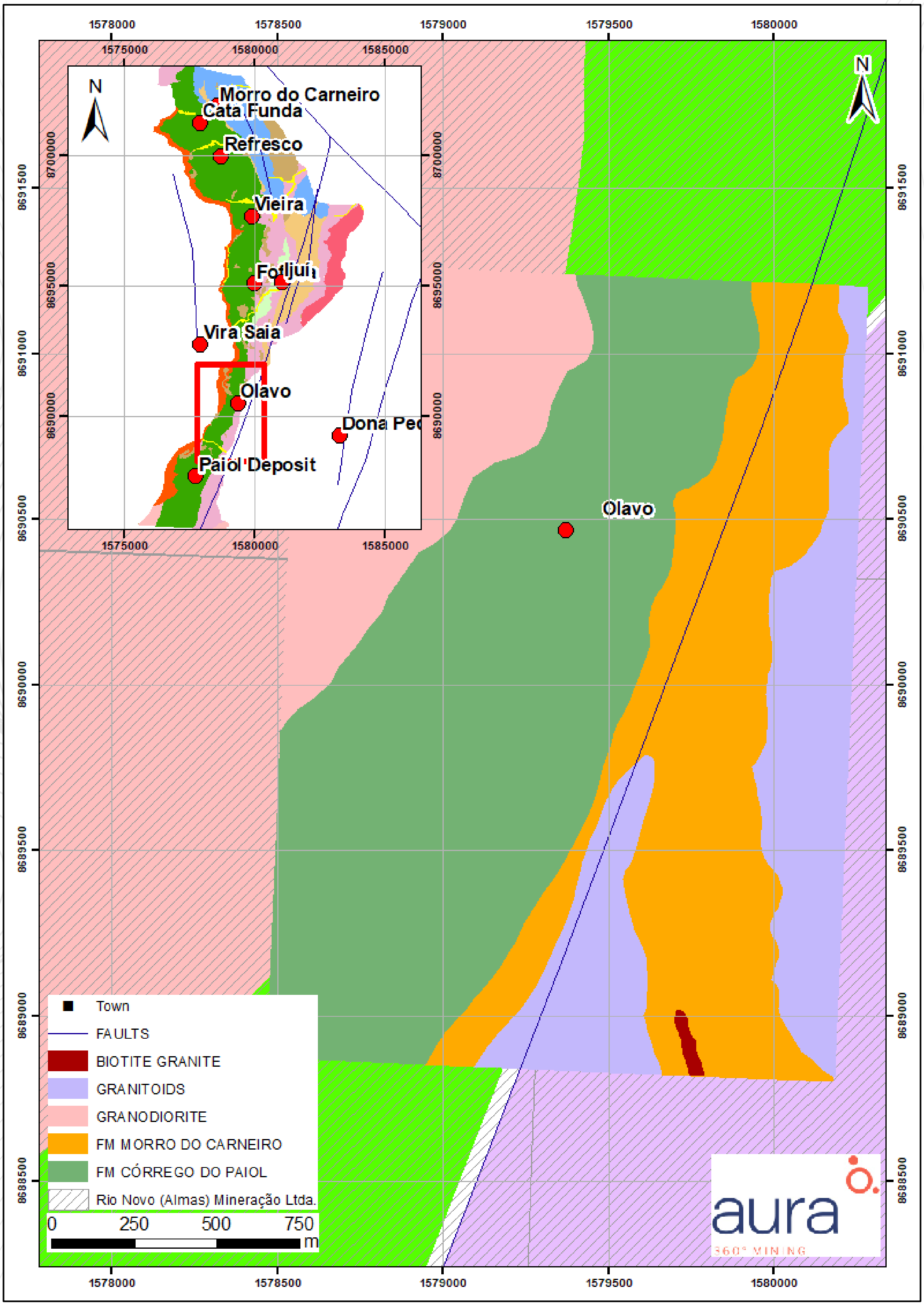
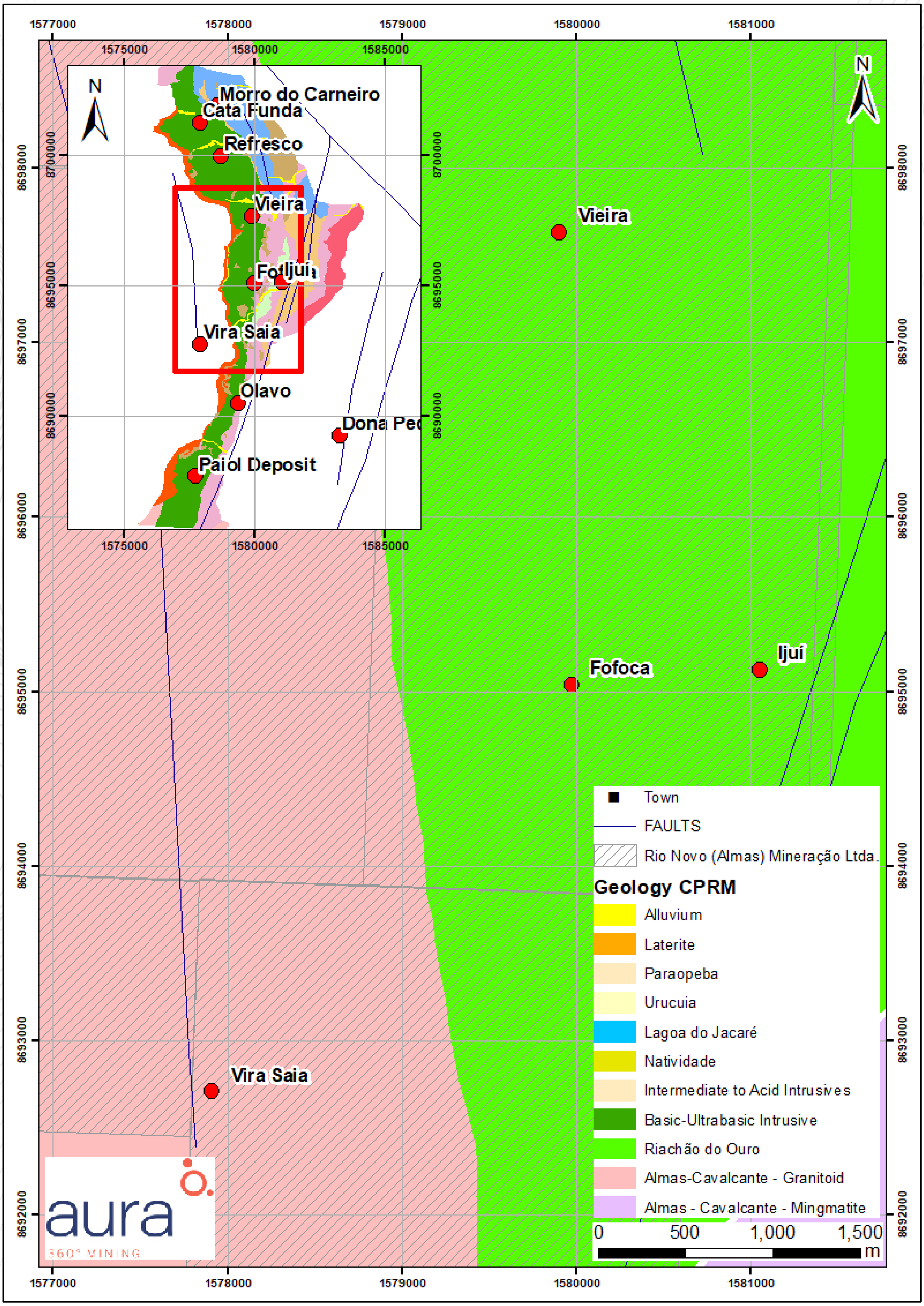


Figure 7-23 Fofoca and Ijuí Targets location and regional geology map



7.13.3 VIEIRA TARGET

The Vieira target is located around 2.5 km north of the Fofoca and Ijuí targets and it is hosted by the same mafic to ultra-mafic rocks of Córrego do Paiol Formation, continuing to confirm the mineralization trend of Almas Belt. Gold mineralization was first seen at the target by soil and rock sampling and further confirmed with exploration diamond drilling in Chlorite-Albite-Amphibole-Schist (CAAX), Chlorite-Quartz-Sericite-Carbonate-Schist (CQSCX), Quartz-Schists (QX) and Quartz-Sericite-Schists (QSX). The Vieira target location is also seen on Figure 7-23.

The lithologies of Vieira target and its rock codes are summarized in Table 7-6.

Table 7-6 Vieira Lithologies and Rock Codes

ROCK CODE	LITHOLOGIES
AFB	Amphibolite
BQZ	Quartz-Breccia
CAAX	Chlorite-Albite-Amphibole-Schist
CCQX	Calcite-Chlorite-Quartz-Schist
CQCX	Carbonate-Quartz-Chlorite-Schist
CQSCX	Chlorite-Quartz-Sericite-Carbonate-Schist
CQSX	Carbonate-Quartz-Chlorite-Schist
CQSXp	Chlorite-Quartz-Sericite-Schist proximal
CQX	Chlorite-Quartz-Schist
CSQX	Chlorite-Sericite-Quartz-Schist
GDM	Mylonite
GQX	Graphite-Quartz-Schist
MD	Mafic Dike
MV	Meta-Dacite
QSCX	Quartz-Sericite-Chlorite-Schist
QSX	Quartz-Sericite-Schist
QX	Quartz-Schist
SBX	Sericite-Carbonaceous-Schist
SCDX	Sericite-Chlorite-Ankerite Schist
SDCX	Sericite-Ankerite-Chlorite-Schist
SQCX	Sericite-Quartz-Chlorite-Schist
SX	Sericite-Schist
VQTZ	Quartz vein

8 DEPOSIT TYPES

The known gold occurrences in the Almas area are classified as orogenic, shear-hosted mesothermal gold deposits. Minor occurrences of lateritic or even placer gold is also found in the area but are typically small and not the target of current exploration.

Mesothermal gold deposits are a distinctive type of gold deposit which are typified by many consistent features in space and time. Perhaps the single most consistent characteristic of the deposits is their consistent association with deformed metamorphic terranes of all ages. Observations from throughout the World's preserved Archaean greenstone belts and most recently active Phanerozoic metamorphic belts indicate a strong association of gold and greenschist facies rocks. However, some significant deposits occur in higher metamorphic grade Archaean terranes or in lower metamorphic grade domains within the metamorphic belts of a variety of geological ages. Pre-metamorphic protoliths for the auriferous Archaean greenstone belts are predominantly volcano-plutonic terranes of oceanic back-arc basalt and felsic to mafic arc rocks. Clastic marine sedimentary rock-dominant terranes that were metamorphosed to graywacke, argillite, schist and phyllite host younger mineralization and are important in some Archaean terranes.

These deposits are typified by quartz-dominant vein systems with ≤3 to 5% sulfide minerals mainly Fe-sulfides and ≤5 to 15% carbonate minerals. Albite, white mica or fuchsite, chlorite, scheelite and tourmaline are also common gangue phases in veins in greenschist-facies host rocks. Vein systems may be continuous along a vertical extent of 1 to 2 km with little change in mineralogy or gold grade. Mineral zoning does occur, however, in some deposits. Au/Ag ratios range from 10 (normal) to 1 (less common) with ore in places being in the veins and elsewhere in sulfurized wallrocks.

Deposits exhibit strong lateral zonation of alteration phases from proximal to distal assemblages on scales of meters. Mineralogical assemblages within the alteration zones and the width of these zones generally vary with wallrock type and crustal level. Most commonly, carbonates include ankerite, dolomite or calcite; sulfides include pyrite, pyrrhotite or arsenopyrite; alkali metasomatism involves sericitization or, less commonly, formation of fuchsite, biotite or K-feldspar and albitization and mafic minerals are highly chloritized. Amphibole or diopside occur at progressively deeper crustal levels and carbonate minerals are less abundant. Sulfidation is extreme in banded iron formation (BIF) and Fe-rich mafic host rocks.

The orogenic gold deposits targeted in current exploration in the Almas Gold Project are hosted in Paleoproterozoic rocks, typically metabasalts and metasediments (commonly called greenstones). Exploration has also identified gold mineralization in granitic intrusives or granitoids, as in the case of the Vira Saia deposit. In all cases, the rocks have been metamorphosed to greenschist or lower amphibolite facies. Mineralization invariably forms along faults or shear zones; typically, the larger mineralized areas correlate with the larger shear zones. As well, flexures and intersection zones, where faults or shears cross, generally correspond to prime sites for these deposits.

The shear zones hosting gold mineralization typically show strong brecciation and mylonitization of the host rocks (Figure 8.2). Alteration of the host rocks is generally localized along the structural zones and is mainly silicification along with widespread carbonatization, potassic alteration, sericite alteration, and pyritization. Gold occurs in association with sulfides in quartz veins and veinlets. Sulfides are primarily pyrite with trace amounts of arsenopyrite, galena, and chalcopyrite. Gold is primarily free gold with an estimated 10 to 40% attached to sulfides, depending on location. Gold is primarily micron-sized, though visible gold is locally present.

Exploration methods in the district typically start with magnetic surveys to identify major structures and magnetic alteration, followed by field mapping and soil sampling. IP surveys are often employed to further identify structures or resistive bodies. Trenching and drilling are used in the final phases. Rio Novo has currently identified over 30 exploration targets in the district and is systematically exploring for additional resources.

Figure 8-1 shows the general schematic cross section through the various deposits of the Almas Gold Project.

Figure 8-1 Schematic Cross Section Showing the Main Deposits of Almas Project

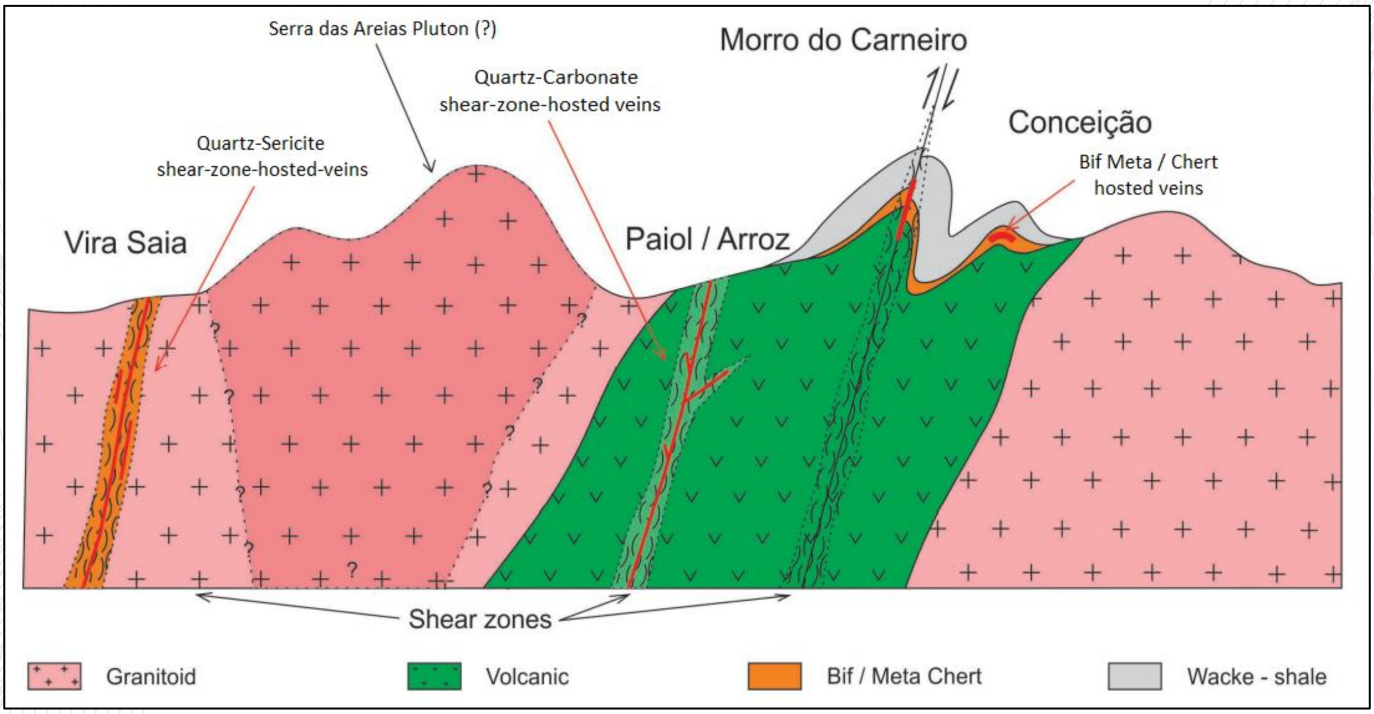
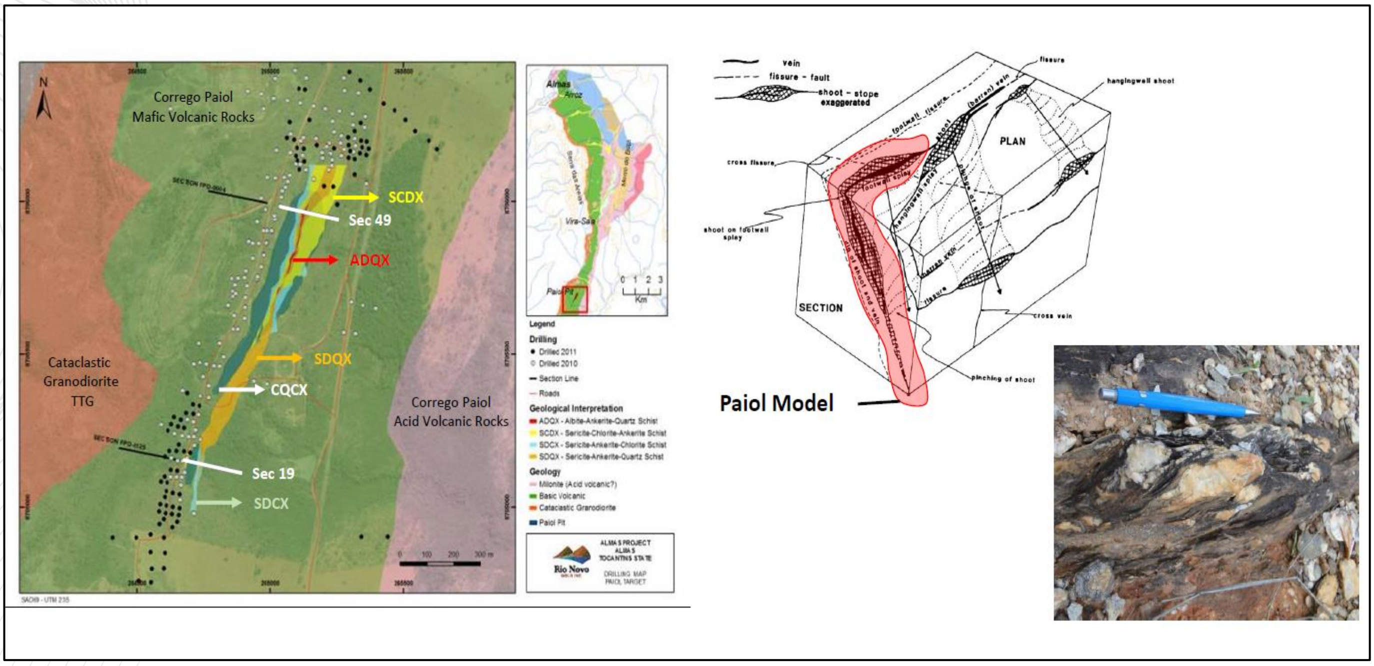


Figure 8-2 Schematic Section Showing the Main Shear Zone at Paiol and the Modelled Hydrothermal Halo Around it



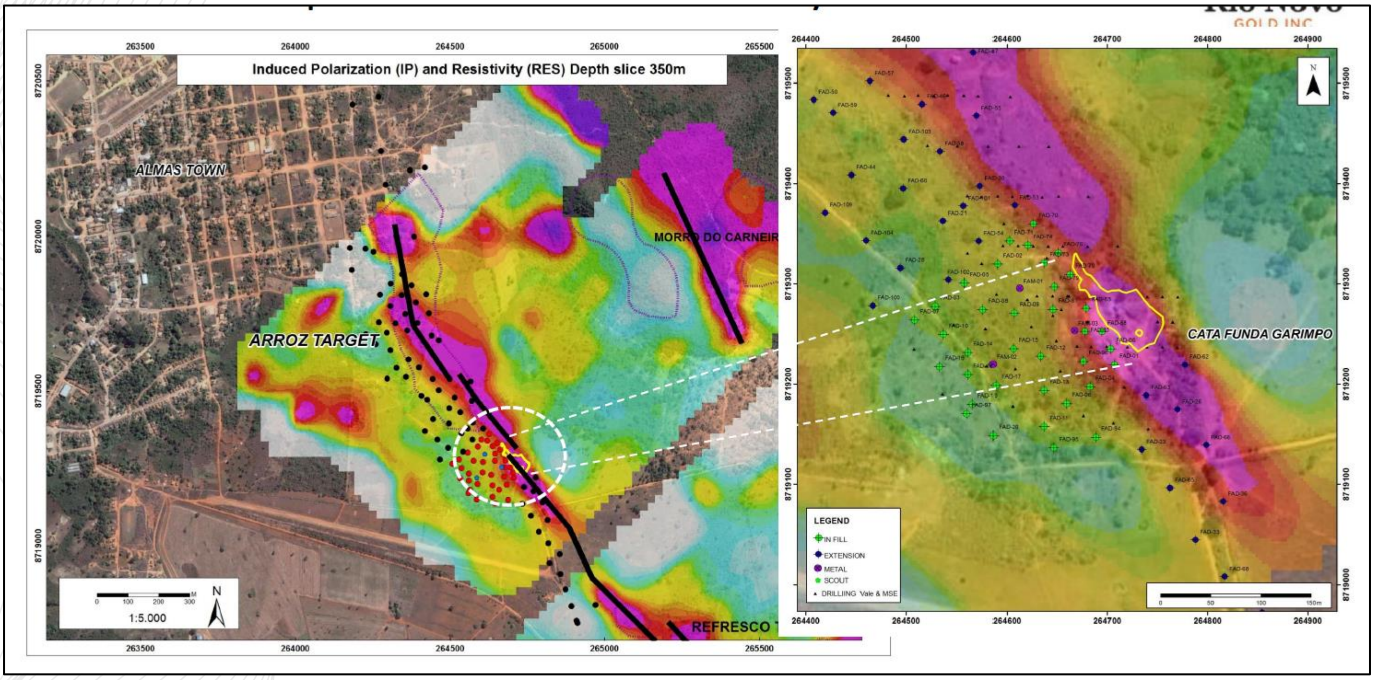
9 EXPLORATION

Exploration within the Almas Gold Project dates to 1977 when VALE identified prospective terrain in the greenstone belts around Almas. A brief history of exploration is presented in Section 6.

Initially, VALE conducted airborne geophysical surveys and ground based geochemical surveys. During the early 1990s, an airborne geophysical HEM, MAG-GAMA survey was performed by Geomag/Fugro using a helicopter with 250 m line spacing and altitudes of 30 m, 45 m and 60 m. The results of this work were helpful in identifying areas underlain by basic volcanic rocks, and radiometry helped define hydrothermal alteration zones.

Shortly thereafter, an IP-Resistivity geophysical survey was carried out in two stages. In the first stage, Geomag used the Gradient IP method to cover the entire Almas Belt. The second stage was carried out by Quantec and consisted of TDIP (Real Section) geophysics covering the Paiol Deposit and part of the Arroz Deposit (Figure 9-1). This technique yielded data from greater depths (between 300 m and 600 m). The results of the geophysical survey show that the mineralized zone is represented by intermediate values of chargeability (10 mV/V and 25 mV/V) and high values of apparent resistivity (>3,000 ohm/m).

Figure 9-1 IP Anomaly in Proximity of the Cata Funda (Arroz) Deposit



VALE conducted geochemical surveys and geological mapping over the bulk of the area now covering the Almas Gold Project. These surveys were conducted at various intervals, depending on prospectively. Generally large-spaced orientation lines were completed on 500 m to 1,000 m intervals, then in-filled. Then most prospective areas were covered at nominally 25 m to 50 m.

The combination of geophysics, geochemistry and geologic mapping led to the discovery of numerous gold anomalies and nine holes were drilled in the Arroz target. The Paiol deposit was discovered in 1987. The Paiol discovery was significant in that the deposit did not crop out, and the discovery was based on a weak soil anomaly and geophysics.

The geological, geochemical, and geophysical surveys conducted by VALE have been passed on to Rio Novo. 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 deposits. These surveys led to the discovery of the Vira Saia deposit in 2011 as well as a few other prospects still in the exploration stage. Rio Novo generally identified prospective areas using a combination of the existing database, plus stream sediment sampling surveys and widely spaced (500+m) orientation lines of geological mapping and sampling. Once identified, a prospective target is mapped in detail (1:500 or 1:1,000 scales) and geochemical soil and rock chip samples are taken. Further exploration will include trenching and possibly drilling.

It is important to note that exploration thus far has been primarily designed to identify near-surface prospects. The deeper, covered areas of the district have yet to be explored. Due to the generally flat terrain and thick soil or saprolite cover, only a small portion of the district has been adequately covered by exploration. Greenstone gold deposits typically have a large vertical extent and the potential for deeper, likely underground targets is good.

10 DRILLING

Drilling on the Almas Gold Project has been completed in various campaigns since 1985 by VALE – Metago, Santa Elina, Mineração Apuã (MA), and Rio Novo. The drilling methods implemented both diamond core and reverse circulation. However, for the purposes of previous studies, Rio Novo elected not to use the reverse circulation drill hole information for the geological models and resource estimates of the main deposits, Paiol, Cata Funda, and Vira Saia. This was done to assure the quality of assays 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. However, in terms of Paiol, RC holes were only used in the estimation phase as data points to have better granularity for estimation, especially close to the surface and in proximity of mined out areas. Reverse circulation and auger drilling were used to evaluate the former Paiol Leach Pad.

10.1 DRILLING METHODS

Core drilling used in this study was a combination of HQ size (63.5mm diameter) and NQ size (47.6mm diameter). Drilling employed standard wireline methods, and generally used split core tubes. Oriented core was taken where possible to allow accurate structural measurements. Drilling angles were in the range of 45 to 70 degrees to intersect the structure and gold 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 MA were also available for their core drilling programs.

Diamond core drilling for Rio Novo was done 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.

The auger 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 is maintained in the core shed.

The RC drilling sampling is done meter by meter; each advanced meter is homogenized and collected. The half of each sample is sent to the laboratory, the other half is maintained in the core shed.

10.2 DRILLING EXTENT AND SPACING

Drilling discussed in this study covers three mineral deposits, Paiol, Vira Saia, and Cata Funda, as well as the former Paiol Leach Pad. The total drilling database includes 830 core holes, 878 reverse circulation holes, and 174 auger holes drilled between 1983 and July 2012. A total of 120,454.58 m comprises the core drilling database.

At Paiol, the known extents of mineralization have been drilled out on nominal 25 m centers. Drilling covers an area of about 2,000 m along strike and 300 m across strike. Additional scout holes have been drilled around the perimeter. The deposit is primarily drilled out to a vertical depth of 250 m to 300 m, although individual drill holes have drilled as deep as 500 m (vertical depth). Most holes were oriented in a range of 100 to 130 degrees azimuth (S80E to S50E), perpendicular to strike. Angle orientations ranged from 50 to 70 degrees SE, for most of the holes, thus cutting the main structural and mineralization trends as near perpendicular as possible.

In total, there have been 459 diamond core holes drilled in the Paiol area, for about 71,191 m. One deep historic Metago core hole was excluded from the database due to a lack of downhole surveys. One Santa Elina hole was also excluded because of location uncertainty. VALE drilled 519 shallow reverse circulation holes. As noted, these were not used in the modeling. Table 10-1 summarizes the Paiol drilling.

At Cata Funda, the deposit has been drilled out at nominal 25 m x 25 m centers. The drilling covers an area of about 700 m along strike and 250 m across strike. Holes are oriented along a 045 azimuth (N45E) for the most part, and with drilling angles of 45 to

60 degrees NE. The deposit is drilled to a vertical depth of about 80 m to 100 m, with an average down hole drilling length of 120 m and the deepest holes reaching vertical depths of 150 m to 170 m.

In addition, 92 reverse circulation holes and 166 auger holes were drilled at the Paiol Leach Pad, to generate its models.

A total of 183 core holes, for 21,408 m, were drilled between 1996 and 2011, and were used to generate the Cata Funda models. Reverse circulation drilling by VALE was not used in the models. Table 10-3 summaries the drilling on Cata Funda.

During 2011 and early 2012, a drilling campaign was completed at the Vira Saia discovery. In total, 194 diamond core holes were completed totaling about 26,513 m. The main drilling was oriented 045 degrees (N45E), perpendicular to the overall strike of the deposit. Holes were angled 45 to 60 degrees to the NE. The deposit has been drilled to a vertical depth of 150 m to 180 m. Drill hole spacing in the resource area is nominally 25 m x 35 m. Table 10-4 details the Vira Saia drilling statistics.

Table 10-1 Total Historical Drilling in the Paiol Area

TYPE	COMPANY	PERIOD	HOLES	AMOUNT (m)	AVERAGE DEPTH	SAMPLES (m)	SERIES
Diamond Drilling	Metago	1987-1989	105	5,544.00	52.8	5,538.65	FS-01 to 105
	Vale	1996-2001	55	17,266.53	313.94	6,776.74	FD-01 to 55
	Santa Elina	2008	48	15,398.90	320.81	13,031.65	FX-XXA or B-SE
	Rio Novo	2010-2012	251	32,982.30	131.4	31,864.05	FPD-0001 to 251
	Subtotal		**459**	**71,191.73**	**155.1**	**57,211.09**	
RC Drilling	Vale	1996-2001	519	26,099.90	50.29	26,096.90	FP-XX
	Rio Novo	2010	33	1,015.00	30.76	1,012.00	FPRC-0060 to 0092
	Subtotal		**552**	**27,114.90**	**49.12**	**27,108.90**	
Metallurgical Core	Rio Novo	2010	8	1,342.25	167,78	1,342.25	FPM-0001 to 0008
	Subtotal		**8**	**1,342.25**	**167.78**	**1,342.25**	
Auger Drill	Rio Novo	2010-2011	8	41.75	5.22	41.75	TRP-00167 to 0174
	Subtotal		**8**	**41.75**	**5.22**	**41.75**	
Total Drilling			**1027**	**99,690.63**	**97.07**	**85,703.99**	

Table 10-2 Total Historical Drilling in the Leach Pad at Paiol

TYPE	COMPANY	PERIOD	HOLES	AMOUNT (m)	AVERAGE DEPTH	SAMPLES (m)	SERIES
RC Drilling	Rio Novo	2010	59	728	12.34	728.00	FPRC-0001 to 0059
	Subtotal		**59**	**728**	**12.34**	**728.00**	
Auger Drill	Rio Novo	2010-2011	166	1,215.15	7.32	1,214.50	TRP-0001 to 0166
	Subtotal		**166**	**1,215.15**	**7.32**	**1,214.50**	
Total Drilling			**225**	**1,943.15**	**8.64**	**1,942.50**	

Table 10-3 Total Historical Drilling in the Cata Funda Area

TYPE	COMPANY	PERIOD	HOLES	AMOUNT (m)	AVERAGE DEPTH	SAMPLES (m)	SERIES
Diamond Drilling	Vale	1996-2001	9	1,072.57	119.17	750.81	FPA-01 to 09
	Santa Elina	2008	23	2,899.33	126.06	2,877.68	ACF-01 to 22-SE
	Rio Novo	2010-2011	148	17,133.60	115.77	17,132.40	FAD-0001 to 148
	Subtotal		**180**	**21,105.50**	**117.25**	**20,760.89**	
RC Drilling	Vale	1996-2001	159	7,976.00	50.16	7,957.00	FP-XX
	Subtotal		**159**	**7,976.00**	**50.16**	**7,957.00**	
Metallurgical Core	Rio Novo	2010	3	302.15	100.72	302.15	FAM-0001 to 0003
	Subtotal		**3**	**302.15**	**100.72**	**302.15**	
Total Drilling			**342**	**29,383.65**	**85.92**	**29,020.04**	

Table 10-4 Vira Saia Historical Drilling

TYPE	COMPANY	PERIOD	HOLES	AMOUNT (m)	AVERAGE DEPTH	SAMPLES (m)	SERIES
Diamond Drilling	Rio Novo	2011-2012	191	26,111.50	136.71	26,090.09	FVSD-0001 to 0189
	Subtotal		**191**	**26,111.50**	**136.71**	**26,090.09**	
RC Drilling	Vale	1996-2001	49	3,214.23	65.6	3,134.61	FVS-01 to 49
	Subtotal		**49**	**3,214.23**	**65.6**	**3,134.61**	
Metallurgical Core	Rio Novo	2011-2012	3	401.45	133.82	401.45	FVSM-0001 to 0003
	Subtotal		**3**	**401.45**	**133.82**	**401.45**	
Total Drilling			**243**	**29,727.18**	**122.33**	**29,626.15**	

10.3 DRILL HOLE LOCATIONS

The location of the historical drill holes and the Rio Novo drill hole locations for Paiol, Cata Funda and Vira Saia are shown in Figure 10-1 to Figure 10-3, respectively. Rio Novo was the only company to drill at Vira Saia.

Figure 10-1 Paiol Drillhole Map

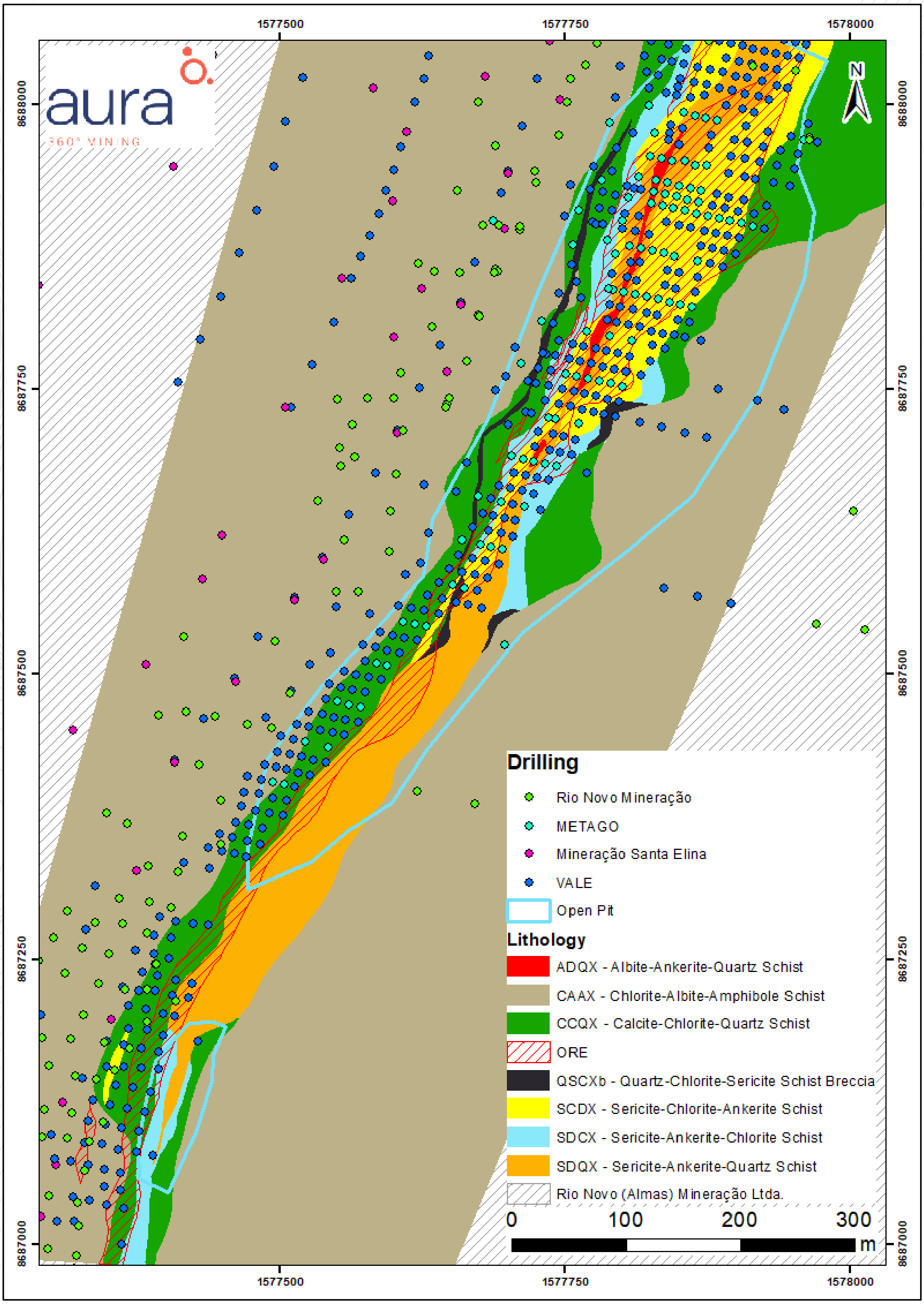


Figure 10-2 Cata Funda Drillhole Map

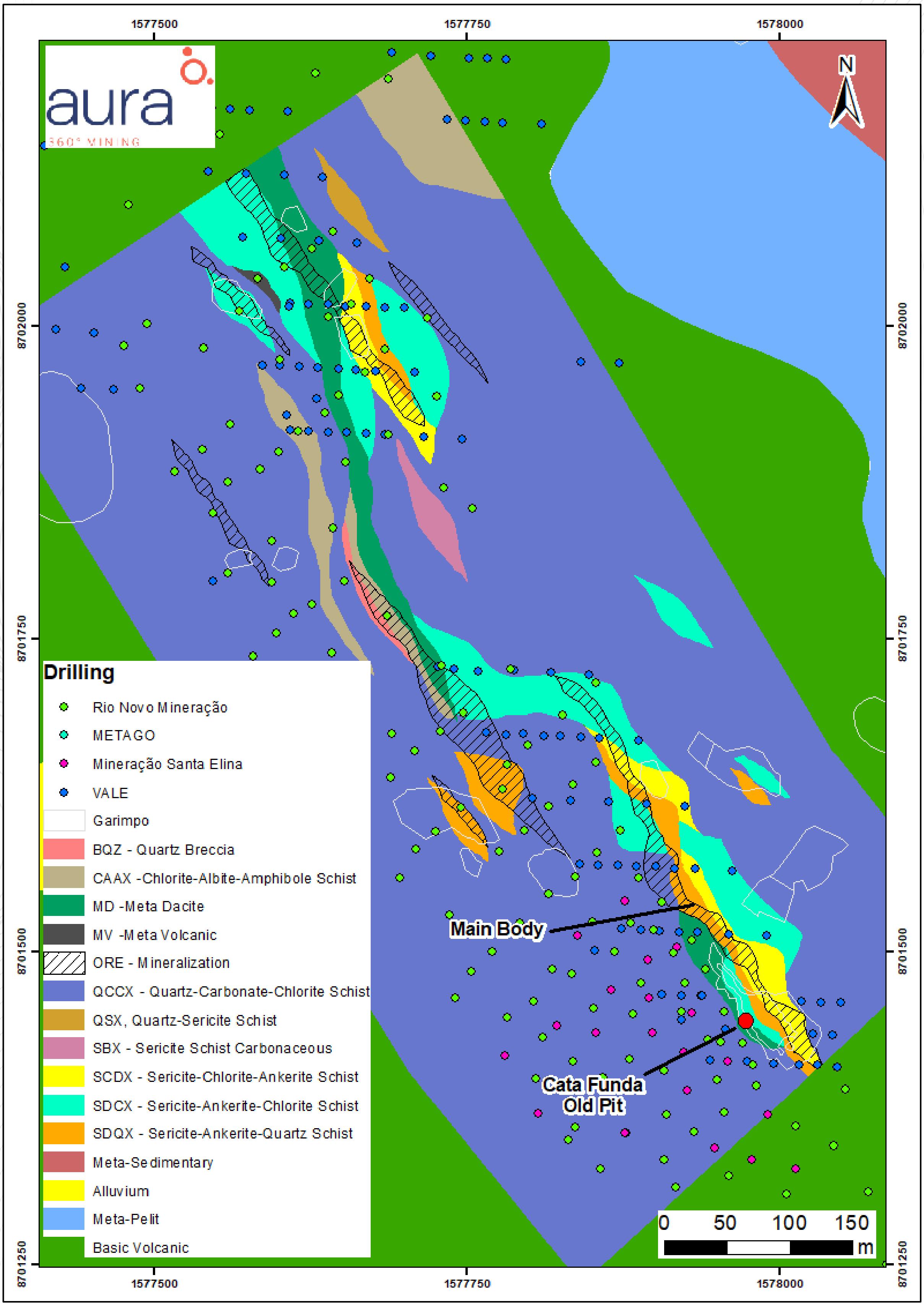
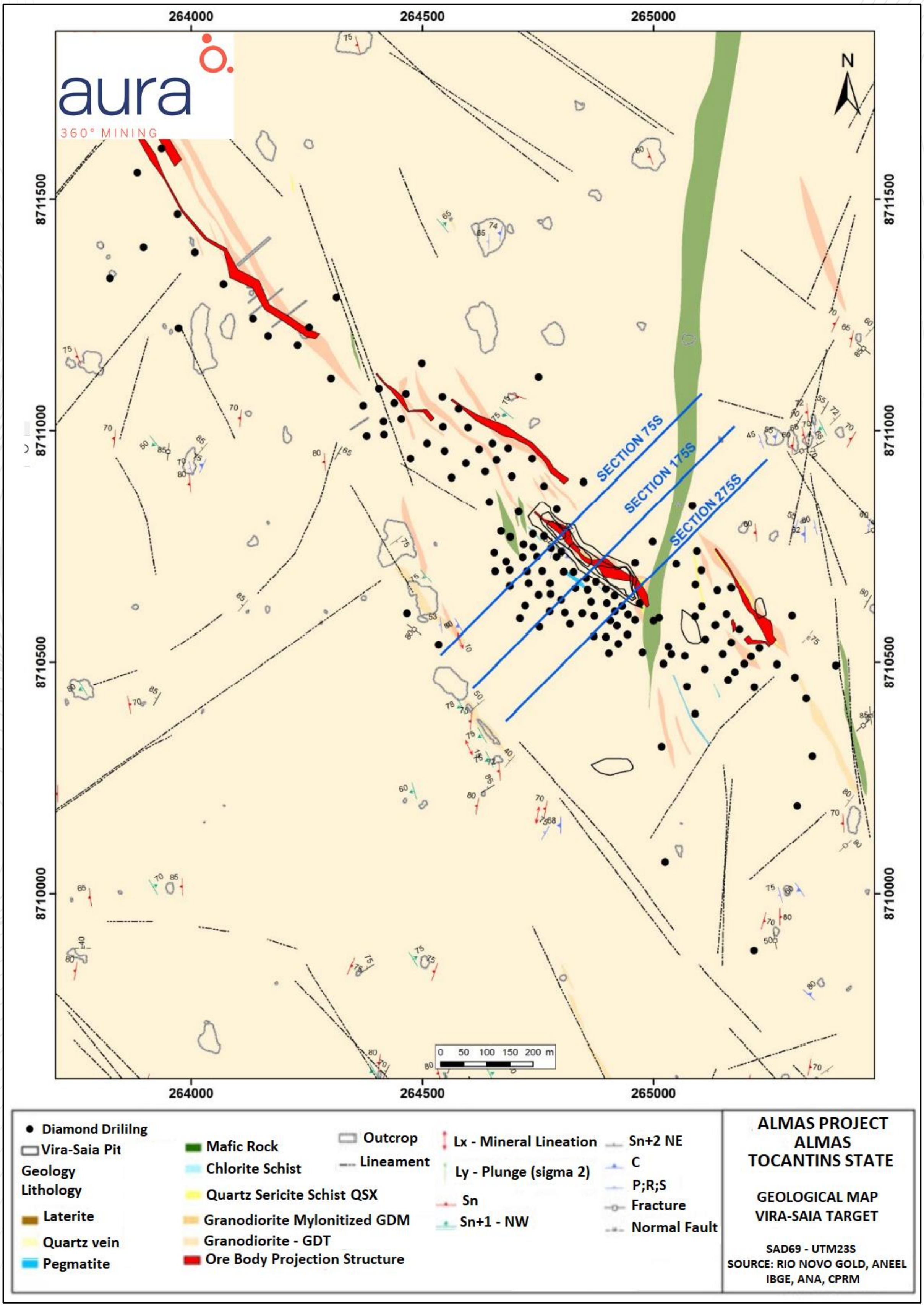


Figure 10-3 Vira Saia Drillhole Map



10.4 DRILLING QUALITY

Paiol, Cata Funda, and Vira Saia have been drilled entirely by diamond core for the purposes of the resource models used herein. The results of reverse circulation drilling at Paiol, which was done by previous operators, is compared to diamond drill hole data and globally did not show any bias (the details of this analysis are presented in Section 12 of this report).

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 50 percent of the core drilled at Paiol, 80 percent of the core drilled at Cata Funda, and 100 percent of the core drilled at Vira Saia, were drilled by Rio Novo. Rio Novo maintained strict, high quality standards for core drilling. The remainder of the core drilling by previous operators was reviewed in detail by Rio Novo and found to be of high quality. Rio Novo removed two core holes from previous operators that did not pass the quality assessment due to recovery or survey issues.

Likewise, the reverse circulation and auger drilling used to determine the grade of the Paiol Leach Pad was conducted by Rio Novo contractors and followed strict quality standards. Rio Novo recognized that there are inherent difficulties in drilling and sampling unconsolidated dump materials. However, the company believes the techniques and quality standards used represent the best practices and provide an accurate assessment of the pad contents.

The drilling programs are consistent with industry standards and the QP opines that it is of sufficient quality to be used in resource estimates and mine planning.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

To date Aura has completed no drilling at the Almas Gold Project. Some resampling of older core and trench sampling of a loaded leach pad has been conducted but the samples were sent for metallurgical test work. The results were not used in the mineral resource estimate.

At the time of the QP's site visit no staff from the previous operators were available for consultation. Section 11 is taken from Rio Novo's 2016 feasibility study by RungePincockMinarco (RPM, RungePincockMinarco, 2016) and summarizes the sample preparation, analyses and security practices of previous operators. The QP has also reviewed Rio Novo's monthly quality assurance/ quality control (QA/QC) reports.

11.1 CORE HANDLING, LOGGING, AND SAMPLING PROTOCOLS

Rio Novo had used only data from diamond core drilling for resource estimation in the 2016 Feasibility Study. Therefore, the following discussion pertains largely to diamond core sampling.

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 processing facility at the former Paiol mine.

On arrival at the core processing facility, the core was laid out, washed and photographed. The core was then logged, and sample intervals 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 were 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. Wherever possible, oriented core samples were taken to give more accurate structural readings, 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 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, core was cut perpendicular to major vein orientations. Geosol then bagged the samples in plastic bags according to Rio Novo procedures. 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 Almas Gold Project are summarized in the flow sheet on Figure 11-1.

11.2 DENSITY DETERMINATIONS

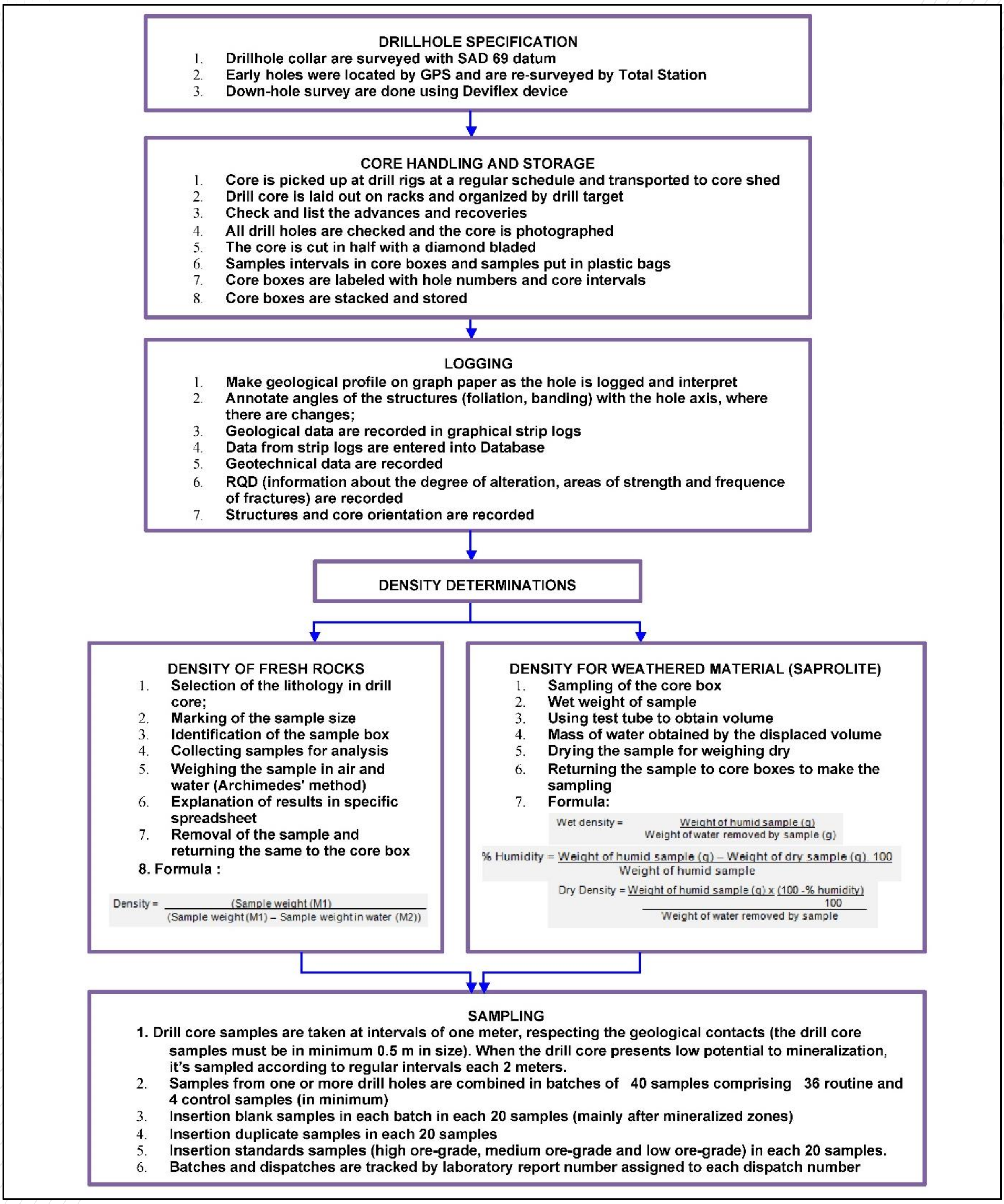
Bulk densities of geological materials encountered in drill core are required to determine mass in mineral resource estimation. Density data must be representative of the lithologies found in the deposit and determined on replicate samples. Rio Novo used different methodologies to determine bulk densities depending 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 three targets in the Almas Gold Project area.

Table 11-2 summarizes the average bulk densities of saprolite and weathered rocks from the three targets in the Almas Gold Project area.

Figure 11-1 Core Handling and Sampling Protocols



Saprolite is a soft, clay-rich material formed by 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 wet weight on an analytical balance. Then a plastic basin was filled with water, a sample was put in the basin until the water flows out. When the water stops flowing from the basin, 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 dry weight was recorded and then used to calculate the bulk density by formula.

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

TARGET	TYPE	ROCK CODE	LEGEND	COUNT	MINIMUM (g/cc)	MAXIMUM (g/cc)	MEAN DENSITY (g/cc)
Cata Funda	Fresh rock	AFB	Amphibolite	3	2.79	2.88	2.84
Cata Funda	Fresh rock	BQZ	Quartz Breccia	3	2.66	2.86	2.73
Cata Funda	Fresh rock	CAAX	Chlorite-Albite-Amphibole Schists	53	2.71	3.03	2.84
Cata Funda	Fresh rock	CCQX	Calcite-Chlorite-Quartz Schist	61	2.70	3.24	2.84
Cata Funda	Fresh rock	CCSX	Carbonate-Chlorite-Sericite Schist	7	2.77	2.86	2.82
Cata Funda	Fresh rock	CQCX	Carbonate-Quartz-Chlorite Schist	12	2.76	2.95	2.85
Cata Funda	Fresh rock	CQSCX	Chlorite-Quartz-Sericite-Carbonate Schist	32	2.67	3.01	2.82
Cata Funda	Fresh rock	CQSX	Chlorite-Quartz-Sericite Schist	8	2.77	2.86	2.82
Cata Funda	Fresh rock	CQSXP	Chlorite-Quartz-Sericite Schist proximal	8	2.79	2.90	2.84
Cata Funda	Fresh rock	CSX	Chlorite-Sericite Schist	23	1.84	2.95	2.77
Cata Funda	Fresh rock	CX	Chlorite Schist	1	2.78	2.78	2.78
Cata Funda	Fresh rock	DT	Tonalite Dike	7	2.50	2.74	2.67
Cata Funda	Fresh rock	GDM	Mylonite Granodiorite	1	2.74	2.74	2.74
Cata Funda	Fresh rock	GQX	Graphite-Quartz Schist	1	2.80	2.80	2.80
Cata Funda	Fresh rock	MD	Meta Dacite	54	1.68	2.94	2.72
Cata Funda	Fresh rock	MP	Metapelite	1	2.95	2.95	2.95
Cata Funda	Fresh rock	MV	Meta-Volcanic	7	2.80	2.97	2.84
Cata Funda	Fresh rock	QCASX	Quartz-Carbonate-Sericite Schist	3	2.71	2.92	2.81
Cata Funda	Fresh rock	QCCX	Quartz-carbonate-chlorite Schists	423	1.07	3.02	2.80
Cata Funda	Fresh rock	QSCAX	Quartz-Sericite-Carbonate Schist	1	2.84	2.84	2.84
Cata Funda	Fresh rock	QSCX	Quartz-Sericite-Chlorite Schist	7	2.30	2.97	2.77
Cata Funda	Fresh rock	QSX	Quartz-Sericite Schist	19	2.56	2.92	2.81
Cata Funda	Fresh rock	QSXn	Quartz-Sericite Schist Nucleus	16	2.70	3.30	2.88
Cata Funda	Fresh rock	QX	Quartz Schist	2	2.74	2.74	2.74
Cata Funda	Fresh rock	SBX	Carbonaceous-Sericite Schist	11	2.48	3.18	2.88
Cata Funda	Fresh rock	SCDX	Sericite-chlorite-Ankerite Schists	139	2.03	3.21	2.84
Cata Funda	Fresh rock	SCX	Sericite-Chlorite Schist	3	2.81	2.86	2.83
Cata Funda	Fresh rock	SDCX	Sericite-ankerite-chlorite Schists	146	1.12	2.92	2.81
Cata Funda	Fresh rock	SDQX	Sericite-Ankerite-Quartz Schists	121	1.08	3.37	2.84
Cata Funda	Fresh rock	SQCX	Sericite-Quartz-Chlorite Schists	1	2.91	2.91	2.91
Cata Funda	Fresh rock	SX	Sericite Schist	46	2.45	3.04	2.81
Cata Funda	Fresh rock	VQTZ	Quartz Vein	2	2.67	2.80	2.73
Paiol	Fresh rock	ADQX	Albite-Ankerite-Quartz Schists	30	2.72	2.92	2.82
Paiol	Fresh rock	AFB	Amphibolite	3	2.95	2.99	2.97
Paiol	Fresh rock	CAAX	Chlorite-Albite-Amphibole Schists	516	1.07	3.70	2.92
Paiol	Fresh rock	CCQX	Calcite-Chlorite-Quartz Schist	627	1.59	3.10	2.86
Paiol	Fresh rock	CQCX	Carbonate-Quartz-Chlorite Schist	31	2.03	2.89	2.81
Paiol	Fresh rock	CQSCX	Chlorite-Quartz-Sericite-Carbonate Schist	20	2.70	2.91	2.83
Paiol	Fresh rock	CQSX	Chlorite-Quartz-Sericite Schist	62	1.94	3.04	2.82

TARGET	TYPE	ROCK CODE	LEGEND	COUNT	MINIMUM (g/cc)	MAXIMUM (g/cc)	MEAN DENSITY (g/cc)
Paiol	Fresh rock	CQSXp	Chlorite-Quartz-Sericite Schist	2	2.82	2.82	2.82
Paiol	Fresh rock	CSX	Chlorite-Sericite Schist	7	2.77	2.96	2.86
Paiol	Fresh rock	DT	Tonalite Dike	16	1.90	2.99	2.67
Paiol	Fresh rock	GD	Granodiorite	6	2.70	3.09	2.84
Paiol	Fresh rock	GDM	Mylonite Granodiorite	11	2.69	2.92	2.75
Paiol	Fresh rock	MD	Meta Dacite	16	2.65	2.85	2.73
Paiol	Fresh rock	MV	Meta-Volcanic	4	2.85	3.00	2.94
Paiol	Fresh rock	QCCX	Quartz-carbonate-chlorite Schists	6	2.79	2.89	2.84
Paiol	Fresh rock	QCX	Quartz-Chlorite Schist	4	2.70	2.86	2.77
Paiol	Fresh rock	QSCX	Quartz-Sericite-chlorite Schist	3	2.81	2.84	2.82
Paiol	Fresh rock	QSCXB	Quartz-Sericite-Carbonate Schist	52	2.67	3.00	2.84
Paiol	Fresh rock	QSX	Quartz-Sericite Schist	43	2.67	3.03	2.91
Paiol	Fresh rock	QX	Quartz Schist	6	2.92	3.07	3.01
Paiol	Fresh rock	RM	Mafic Rock	1	3.03	3.03	3.03
Paiol	Fresh rock	SCDX	Sericite-chlorite-Ankerite Schists	186	1.93	2.96	2.82
Paiol	Fresh rock	SCX	Sericite-Chlorite Schist	11	2.81	3.02	2.90
Paiol	Fresh rock	SDCX	Sericite-ankerite-chlorite Schists	418	-0.16	3.54	2.83
Paiol	Fresh rock	SDQX	Sericite-Ankerite-Quartz Schists	151	2.05	3.87	2.85
Paiol	Fresh rock	SQCX	Sericite-Quartz-chlorite Schists	14	2.76	2.96	2.87
Paiol	Fresh rock	SX	Sericite Schists	6	2.60	2.81	2.68
Paiol	Fresh rock	VQTZ	Quartz Vein	4	2.62	2.70	2.65
Vira Saia	Fresh rock	CAAX	Chlorite-Albite-Amphibole Schists	1	2.97	2.97	2.97
Vira Saia	Fresh rock	DM	Mafic Dyke	139	2.62	3.66	2.89
Vira Saia	Fresh rock	DT	Tonalite Dike	1	2.71	2.71	2.71
Vira Saia	Fresh rock	GDM	Mylonite Granodiorite	429	1.85	3.31	2.71
Vira Saia	Fresh rock	GDP	Protomylonite Granodiorite	335	1.78	3.06	2.71
Vira Saia	Fresh rock	GDT	Granodiorite	1033	1.64	3.69	2.72
Vira Saia	Fresh rock	PGT	Pegmatite Quartz Feldspar	72	2.58	2.75	2.65
Vira Saia	Fresh rock	QSX	Quartz-Sericite Schist	99	2.57	3.02	2.72
Vira Saia	Fresh rock	QX	Quartz Schist	1	2.65	2.65	2.65
Vira Saia	Fresh rock	SAQM	Sericite Ankerite Quartz Schist Mylonite	7	2.65	2.76	2.72
Vira Saia	Fresh rock	VQTZ	Quartz Vein	16	2.58	2.7	2.65

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

TARGET	ROCK CODE	LEGEND	COUNT	MINIMUM SG_WET (g/cc)	MAXIMUM SG_WET (g/cc)	AVERAGE SG_WET (g/cc)	MINIMUM SG_DRY (g/cc)	MAXIMUM SG_DRY (g/cc)	AVERAGE SG_DRY (g/cc)	AVERAGE Moisture (%)
Cata Funda	RI	Weathered Rock	41	1.88	3.05	2.3	1.58	3.04	2.12	8.39
Cata Funda	RSI	Semi Weathered Rock	28	1.93	2.97	2.58	1.8	2.93	2.48	4.04
Cata Funda	SO	Soil	4	1.77	2.19	1.96	1.4	1.88	1.59	18.88
Cata Funda	SP	Saprolite	158	1.4	2.42	1.86	1.07	2.39	1.54	16.96
Paiol	RI	Weathered Rock	113	1.7	3.01	2.29	1.03	3	2.05	11.11
Paiol	RSI	Semi Weathered Rock	87	1.74	3.34	2.68	1.68	3.33	2.58	3.99
Paiol	SO	Soil	67	1.61	2.33	1.98	1.23	2.11	1.62	18.31
Paiol	SP	Saprolite	343	1.18	2.55	1.91	1.1	2.36	1.55	18.61
Vira Saia	RI	Weathered Rock	115	0.24	2.81	2.29	0.21	2.77	2.12	3.48
Vira Saia	RSI	Semi Weathered Rock	24	1.67	2.96	2.48	1.4	2.9	2.34	5.76
Vira Saia	SO	Soil	47	1.67	2.4	2	1.33	2.09	1.72	14.09
Vira Saia	SP	Saprolite	244	1.18	2.76	2.04	1.24	2.89	1.78	13

11.3 SAMPLE PREPARATION – 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 preparation of samples of rock/core and soil/stream sediments as summarized on Figure 11-2 and Figure 11-3, respectively. Before starting sample preparation, proper equipment must be setup, 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.

11.4 SAMPLE ASSAYING

The primary analytical laboratory used by Rio Novo for the Almas Project was the SGS Geosol laboratory, located in Vespasiano, Minas Gerais State, Brazil. The laboratory has ISO 9001 certification and ISO 14001:2004, ISO 17025:2009 certification for environmental chemical analyses.

SGS Geosol employs modern, industry standard techniques and analytical methods. For the purpose of routine gold analysis at the Almas Gold Project, fire assay with atomic absorption (AA) finish was used most frequently. Multielement analyses (34 elements) were determined by ICP subsequent to 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)

Figure 11-2 Sample Preparation Protocol - Rock and Core

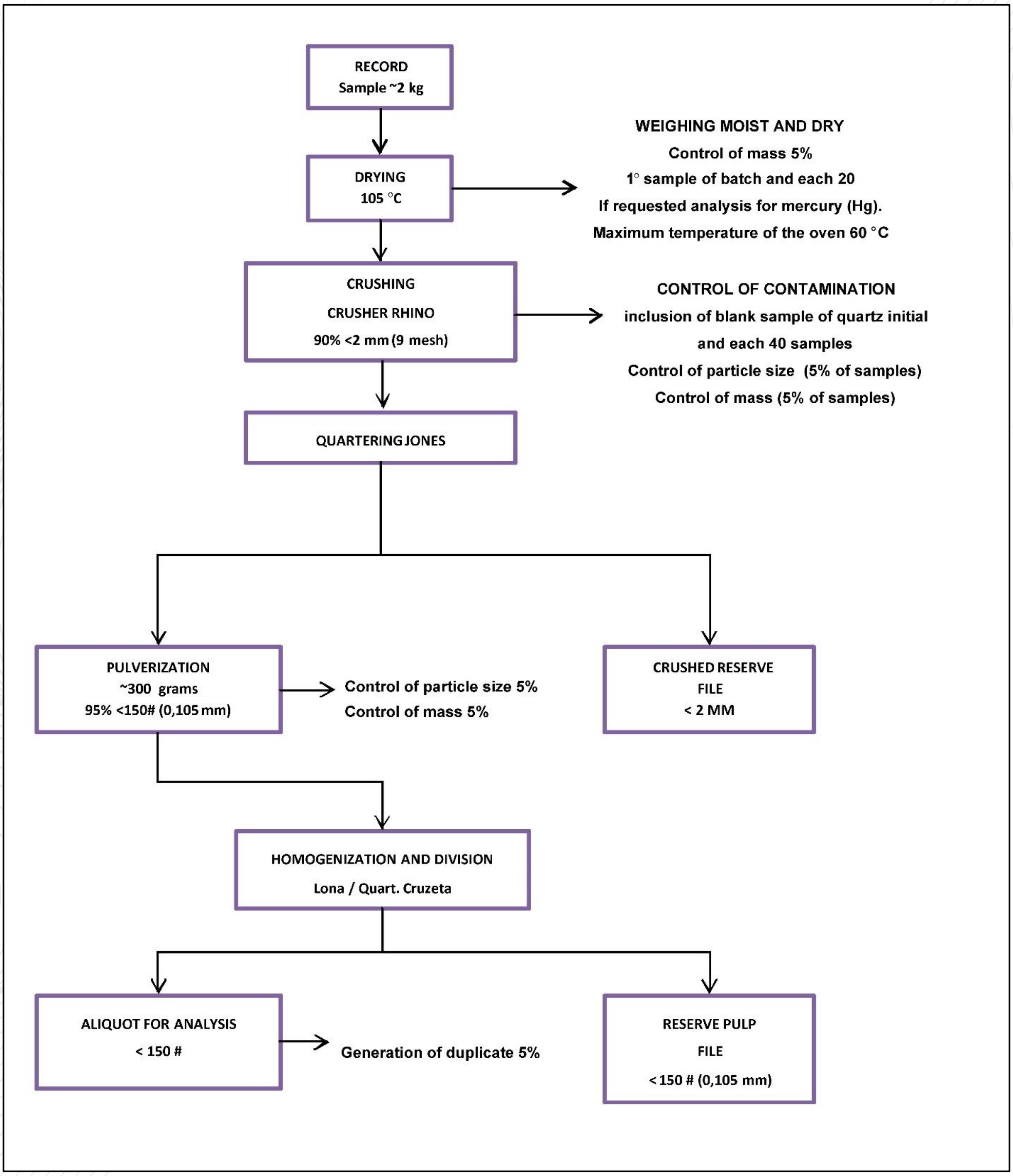
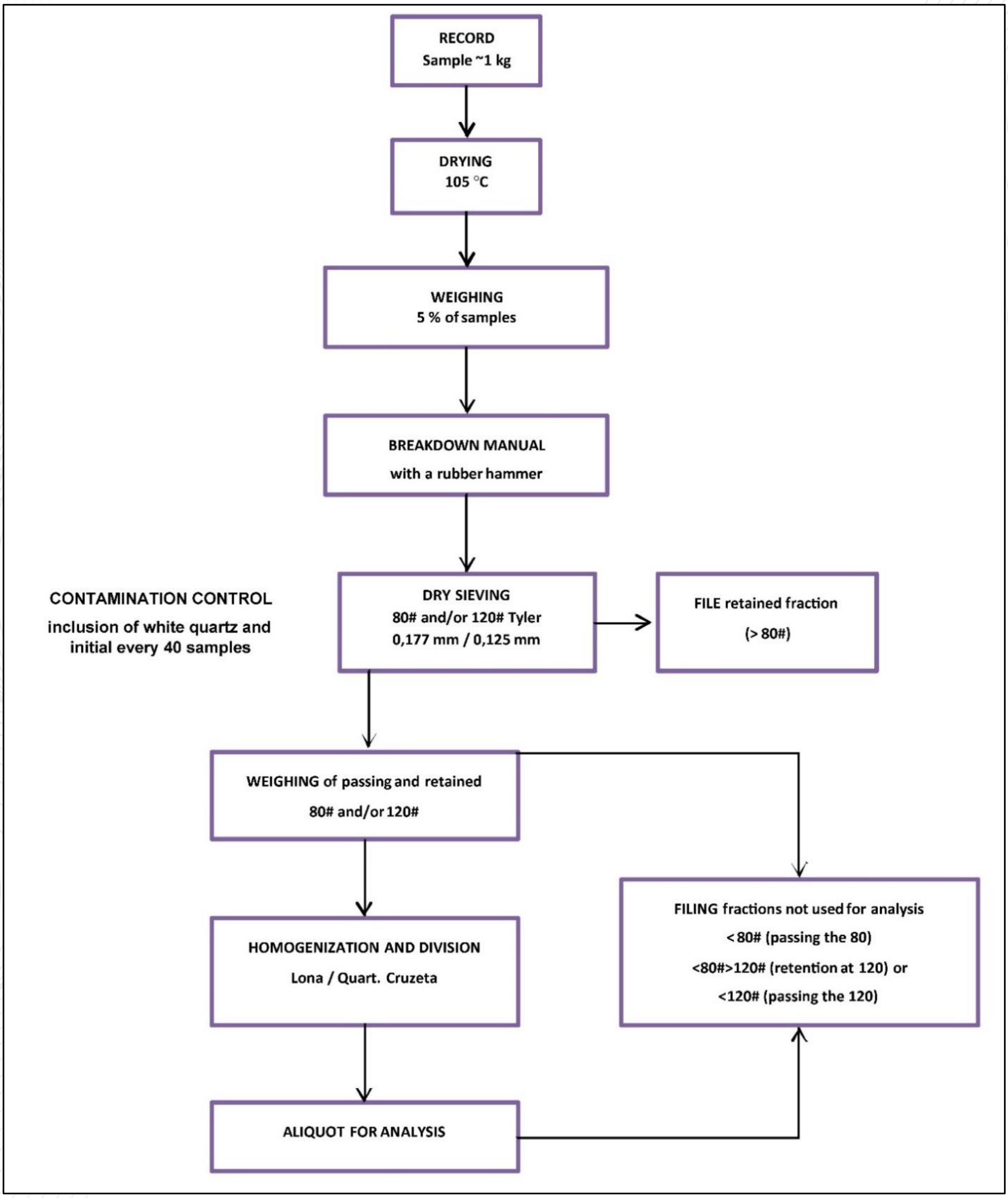


Figure 11-3 – Sample Preparation Protocol - Soils and Stream Sediments



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 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 content.

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). Beginning in November 2011, all samples from the mineralized zone of the Vira Saia target were analyzed using the metallic screen test. 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. 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 Almas Gold Project is presented in Figure 11-4.

The second laboratory used by Rio Novo for check assays was ALS Chemex in Vespasiano, Minas Gerais State and Goiânia, Goiás State, Brazil. The analyses 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.

Table 11-4 Analytical Codes for Gold Analysis: ALS Laboratory

CODE	INTERVAL (ppm)	DESCRIPTION
Au-AA24	0.005-10	Au fire assay with AAS finish, nominal weight of sample of 50g.
Au-AA26	0.01-100	Au fire assay with AAS finish, nominal weight of sample of 50g.
Au-GRA22	0.05-1000	Au fire assay with gravimetric finish, nominal weight of sample 50g.

The third laboratory used by Rio Novo for check assays was Acme Analytical Laboratories in Aparecida de Goiânia, Goiás State, Brazil. The actual analyses were made in Santiago, Chile. The analytical codes and a brief description of the analysis of gold in the Acme laboratory is given on Table 11-5.

Table 11-5 Analytical Codes for Gold Analysis: ACME Laboratory

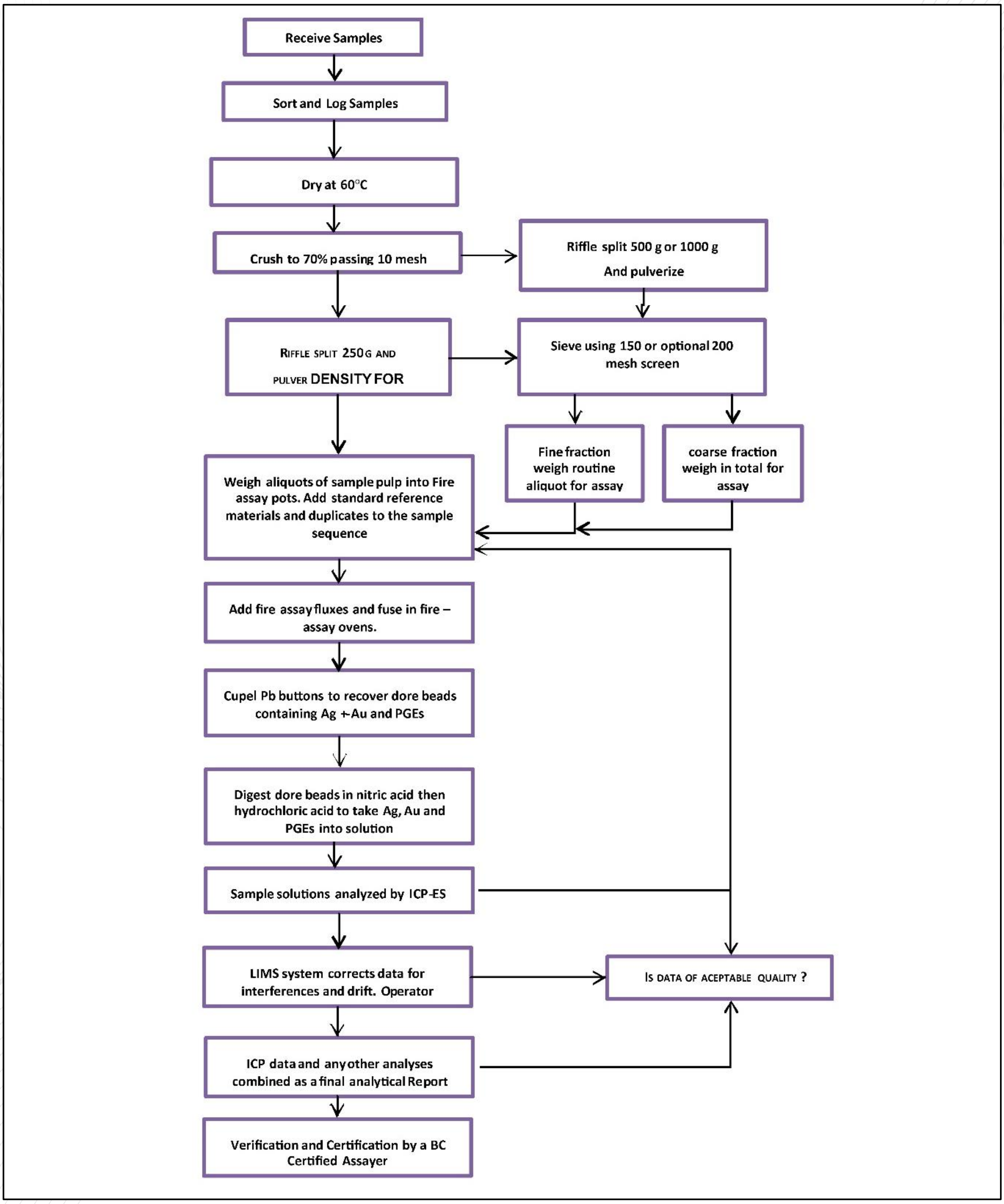
CODE	INTERVAL (ppm)	DESCRIPTION
Au-G610+G610	0.005-10	Determination of Au by Fire Assay on 50g, read by AAS
Au-G612	5-1000	Determination of Au by Fire Assay on a 50g sample with subsequent Dosage by Gravimetry

11.5 QA/QC PROGRAM

11.5.1 RIO NOVO INTERNAL QA/QC PROGRAM

The Rio Novo QA/QC program included submittal of both blind and non-blind control samples into the sample stream being analyzed by the laboratories. Rio Novo maintained internal quality control by inserting blind control samples into the sample stream whilst external quality control was established by each laboratory who insert their own control samples into the sample stream being analyzed. The results of the internal and external QA/QC program are discussed below.

Figure 11-4 Analytical Protocol



The following types of control samples were routinely analyzed as part of Rio Novo's QA/QC program.

- Certified Reference Materials (CRM, "standards") and blanks
- Intra- and inter-laboratory check assays
- Core duplicate assays
- PulP and coarse reject duplicate assays

Additionally, Rio Novo established:

- A secure core, pulp and coarse reject archive system
- Regular contact and site visits to the laboratory
- A protocol for remedial action on analytical batches with poor performance;
- Regular and formal reporting to tract QA/QC history during the project.

11.5.2 LABORATORY QA/QC PROGRAMS

Commercial laboratories contracted for the Almas Gold Project routinely inserted blanks, standards and duplicates into each batch of samples to be analyzed. At SGS Geosol, replicate samples were also inserted. SGS Geosol includes the results of their internal QA/QC analyses with their analytical reports. ALS Chemex issues a separate report for control samples on request. At both laboratories results of control sample analyses were stored in the laboratory's files while a copy was also stored in the Rio Novo Digital Data Base. All analytical results from both laboratories were delivered in digital format to Rio Novo's database manager while the Certificates of Analysis were provided separately. Copies of the digital assay files and certificates are stored in the Rio Novo Digital Data Base.

11.5.3 RIO NOVO QA/QC PROGRAM

Rio Novo's QA/QC program requires that the following minimum number of control samples be inserted into the sample stream being submitted to the laboratory.

- One high ore-grade and one low ore-grade CRM (or medium grade) in each analytical batch of 40 samples (5%);
- A minimum of one blank inserted in each batch mainly after mineralized zones;
- A minimum of two core duplicates in each analytical batch of 40 samples (5%);
- Check assays on pulp samples (greater than 0.3 g/t Au) sent to a secondary laboratory. Approximately 5% to 10% of all pulp samples greater than 0.3 g/t Au were checked in a secondary laboratory.

The control sample assay results of the internal QA/QC program were monitored, including the CRMs, coarse and pulp duplicates and sizing checks during preparation. Additionally, systematic checks of the digital database were conducted against the original signed Certificates of Analysis from the laboratory.

11.5.4 ACCEPTANCE AND REJECTION THRESHOLDS

The following criteria were used to establish acceptance and rejection thresholds for internal control samples analyzed during the Almas Gold Project.

For CRMs:

- Automatic batch failure if the CRM assay result is greater than three standard deviations of the accepted mean value of the CRM, then re-assay the batch;
- Contact laboratory if trends on CRM plots suggest possible bias, work with lab to resolve;

For Blanks:

- If assays on field blanks exceed three times the detection limit of 15 ppb Au, then automatic re-assaying of ten samples surrounding the blank sample in the batch;

High Grade Checks:

- Assays returning greater than 10 ppm Au were automatically re-assayed by the metallic screen method (MET 150) in the same laboratory as the original assays;

Duplicate Samples

- Rio Novo did not routinely submit internal duplicates of pulp and coarse reject material. Rio Novo routinely submitted field duplicates of core; however, assays from field duplicates were not used to determine failed batches;

- Duplicate control samples of pulp and coarse reject materials were analyzed by the laboratory and provide an on-going check on precision. There are two types of control: batch-based and global population. Whereas "A" and "B" are the original and duplicate analyses, respectively, the batch-based rejection thresholds were:
- For pairs in which (A+B)/2 < 15xDL (detection limit)
- Pulp duplicates: I A-B I < 2xDL
- Coarse reject duplicates: I A-B I < 3x DL
- If not, I A-B I * 0,5 / (A+B)
- - < 10% for pulps
- - < 20% for coarse rejects
- For the global duplicate population, the thresholds were:
- Pulp Duplicates - 90% should have relative differences less than 10%
- Coarse Duplicates - 90% should have relative differences less than 20%
- For check assays on a global population basis, in a second laboratory, the maximum acceptable difference was 4%.
- A chart showing the behaviour of duplicate samples was routinely provided in the QA/QC report.
- Sample rejection rates were routinely described in a table in the QA/QC report.

11.5.5 CERTIFIED REFERENCE MATERIALS (STANDARDS)

Certified Reference Materials (CRMs) or standards were used to monitor analytical accuracy and precision of assay results within the primary laboratory or externally, between two or more laboratories. CRM's are well-analyzed, meticulously prepared, ground rock powders, for which the concentration of selected constituent elements are well behaved and vary within low statistical ranges. Once a material containing the desired elements is selected, the CRM producer sends multiple samples of the reference material to a minimum of ten accredited laboratories for analysis by one or more analytical methods. This Round Robin approach is to provide sufficient assay data to determine statistically a representative mean value and standard deviation required for setting acceptance/rejection tolerance limits for the elements of interest.

The Almas Gold Project used ten CRMs; seven provided by the Instituto de Tecnologia November Kekulé (ITAK) and three provided by Geostats Pty Ltd. (Table 11-6).

Table 11-6 Certified Reference Materials (CRM) for Gold used in the Almas Project

CERTIFIED REFERENCE MATERIAL	TYPE	CERTIFIED VALUE (g/t)	STANDARD DEVIATION	NUMBER OF SAMPLES
CRM ITAK 505	Low grade	0.233	0.019	426
CRM ITAK 530	Low grade	0.316	0.03	743
GEOSTAT-G904-6	Low grade	0.36	0.02	170
CRM ITAK 518	Low grade	0.547	0.018	351
CRM ITAK 516	Middle grade	1.01	0.068	272
GEOSTATS-G997-6	Middle grade	1.68	0.08	171
GEOSTATS-G901-1	High grade	2.58	0.13	221
CRM ITAK 531	High grade	2.76	0.18	747

CRM ITAK 509	High grade	3.56	0.13	201
CRM ITAK 506	High grade	8.87	0.27	341

Rio Novo requested that the ITAK Laboratory prepare new CRMs from coarse reject material from Rio Novo's drilling program. This material ranged in concentrations from 0.316 g/t (low grade) to 2.76 g/t Au (high grade). ITAK prepared a sample of approximately 100 kg of material that was dried at 105 °C and then homogenized. After homogenization, the material was split down to aliquots of approximately 60 g which were evaluated on the degree of gold homogeneity. Finally, a group of accredited laboratories were invited to perform the Round Robin assays, the results of which were used to determine the accepted mean value and standard deviations for ITAK530 and ITAK531 which are 0.309 g/t Au (+/- 0.028 g/t) and 2.71 g/t Au (+/- 0.13 g/t), respectively. These CRMs were used from June 2011 onward. The quality of a CRM can be assessed by calculating its coefficient of variation which is the standard deviation divided by the accepted value of the certified element. For ITAK530 and ITAK531, the coefficients of variation are 9% and 5%, respectively, indicating that they are stable and acceptable for use as CRMs.

11.5.6 CRM CHARTS

The aggregate results of Rio Novo's internal QA/QC program with respect to CRMs are presented in graphical form on Figure 11-5, Figure 11-6, Figure 11-7 and Figure 11-8. By inspection, it is clear that for the 10 CRMs used in the Almas Gold Project (Table 11-6), all CRMs returned analyses that are well within the three-standard deviation rejection threshold. Of the 2,800 CRM analyses that were determined in the project (2010 to 2012), only 61 samples failed, comprising less than 3% of the total. As shown in Table 11-7, the most common notation associated with failure was that another analysis of the same CRM was present in the same analytical batch and returned acceptable accuracy. Based on the CRM data presented here, RPM expressed the belief that the accuracy of assays generated for the Almas Gold Project (2010 - 2012) were acceptable by industry standard and would not prevent their use in resource estimation. The QP agrees with this observation.

Figure 11-5 Internal CRM Analyses (1)

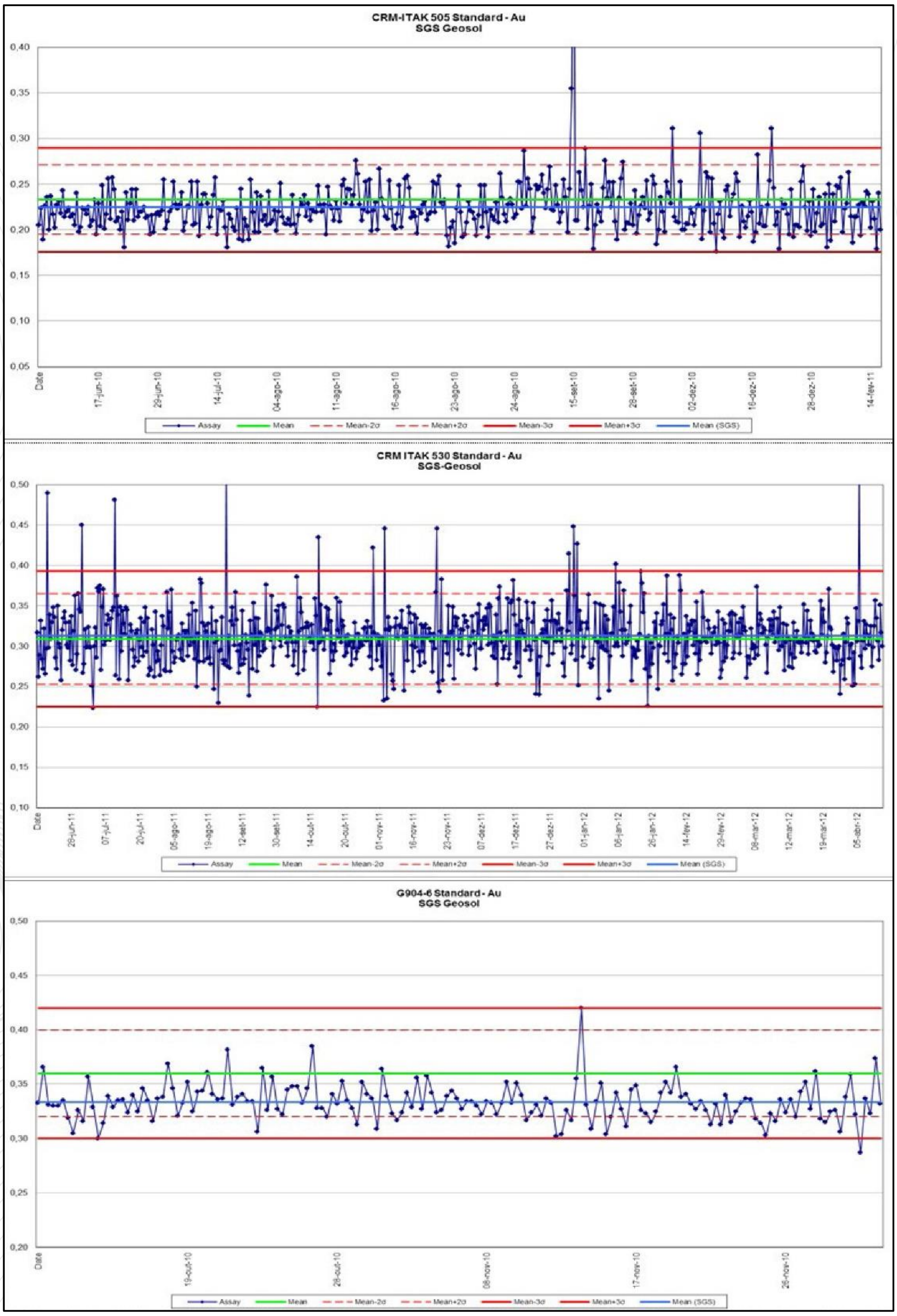


Figure 11-6 Internal CRM Analyses (2)

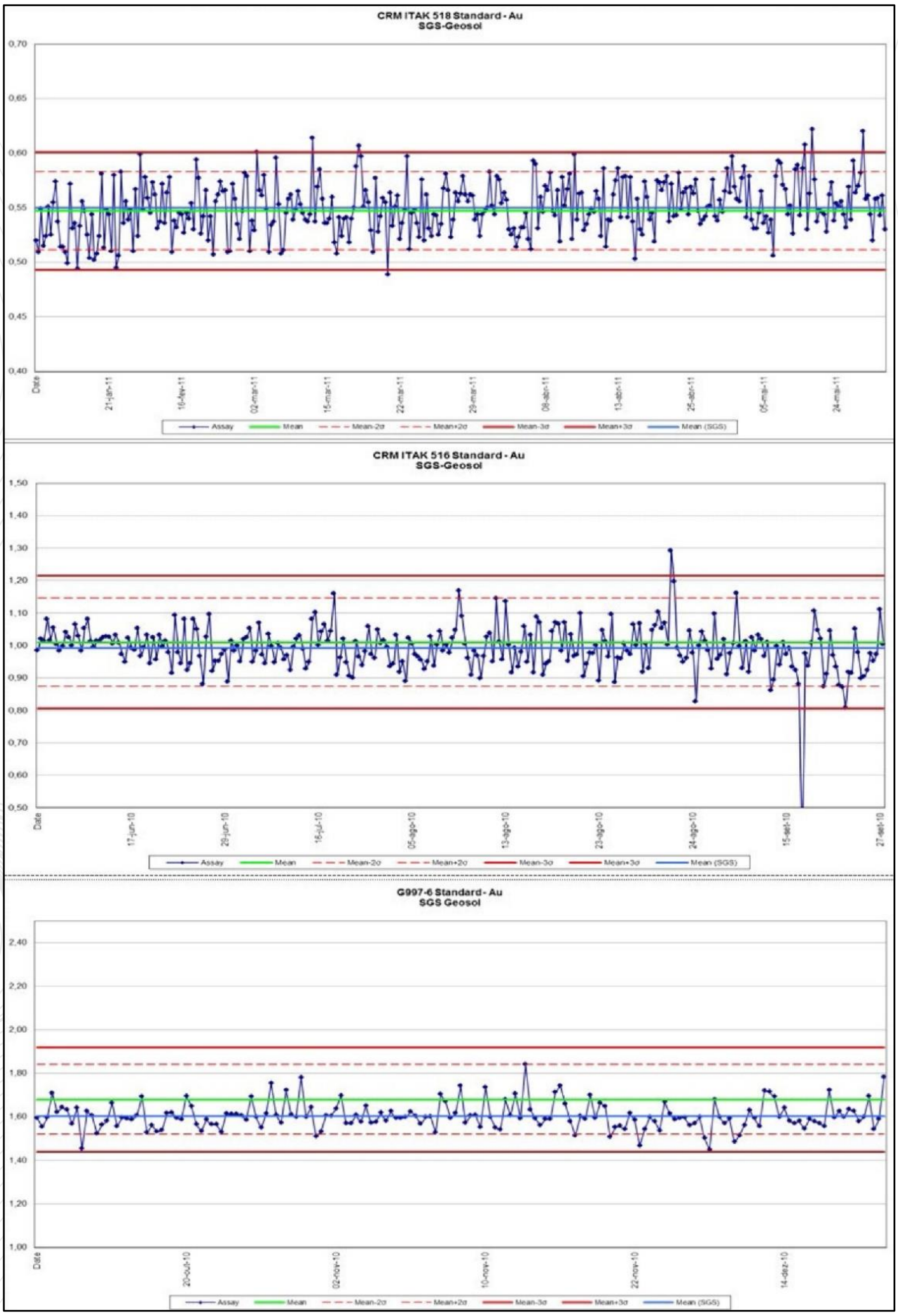


Figure 11-7 Internal CRM Analyses (3)

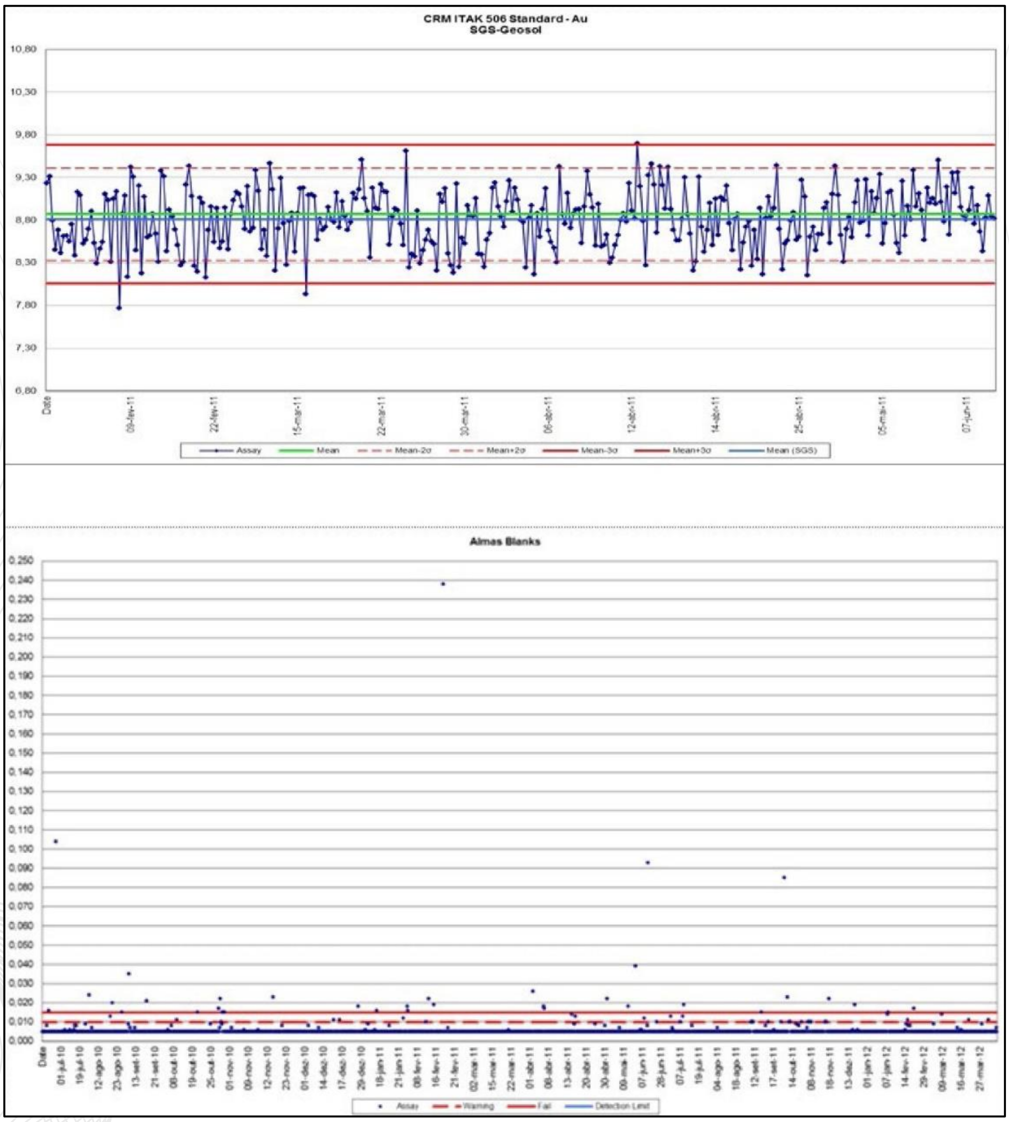


Figure 11-8 Internal CRM and Blank Analyses

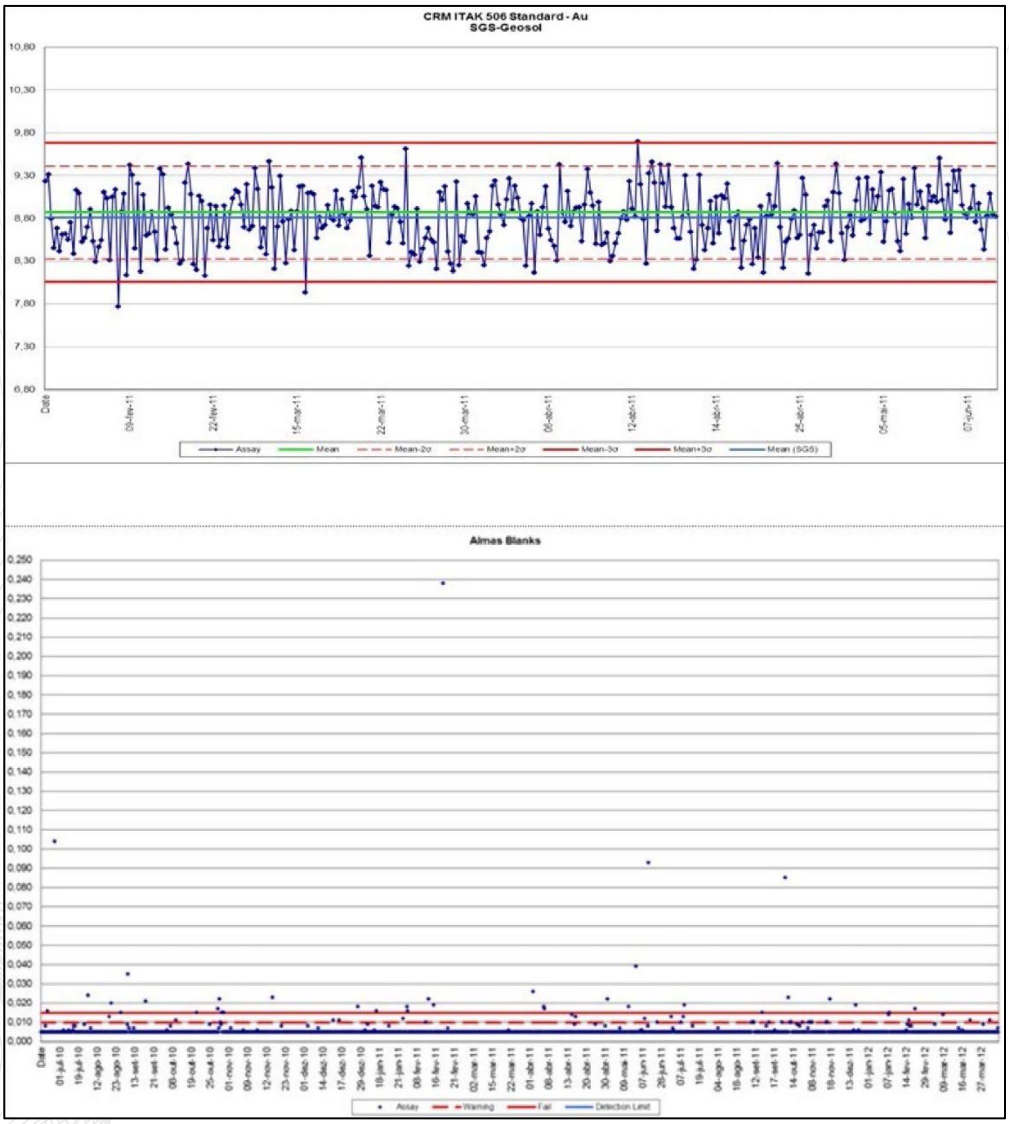


Table 11-7 Certified Reference Materials: Over Limit Assays

CERTIFIED REFERENCE MATERIAL	SAMPLE NO.	AU (ppb)	ACTION
CRM ITAK 505	A-12905	355	The standard failed but in the same batch had other samples of the same standard that validate the batch.
CRM ITAK 505	A-12864	650	The standard failed but in the same batch had other samples of the same standard that validate the batch.
CRM ITAK 505	A-13283	311	The standard failed but in the same batch had other samples of the same standard that validate the batch.
CRM ITAK 505	A-20033	306	The standard failed but in the same batch had other samples of the same standard that validate the batch.
CRM ITAK 505	A-15219	311	The standard failed but in the same batch had other samples of the same standard that validate the batch.
CRM ITAK 530	A-67028	490	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 530	A-90019	450	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 530	A-77857	224	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 530	A-78335	481	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 530	A-84966	523	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 530	A-88707	435	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 530	A-91100	422	The standard failed but in the same batch had other samples of the same standard that validate the batch.
CRM ITAK 530	A-91790	446	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 530	A-93650	446	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 530	A-99701	415	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 530	A-99260	448	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 530	A-99767	427	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 530	A-100134	402	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 530	A-112425	513	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
GEOSTATS-G904-6	A-28540	287	The standard failed but in the same batch had other samples of the same standard that validate the batch.
CRM ITAK 518	A-46675	614	The standard failed but in the same batch have other samples of the same standard that validate the batch.
CRM ITAK 518	A-47649	607	The standard failed but in the same batch have other samples of the same standard that validate the batch.
CRM ITAK 518	A-48443	489	The standard failed but in the same batch have other samples of the same standard that validate the batch.
CRM ITAK 518	A-59314	608	The standard failed but in the same batch have other samples of the same standard that validate the batch.
CRM ITAK 518	A-59446	622	The standard failed but in the same batch have other samples of the same standard that validate the batch.
CRM ITAK 518	A-64261	620	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 516	A-10084	1,293	The standard failed but in the same batch had other samples of the same standard that validate the batch.

CERTIFIED REFERENCE MATERIAL	SAMPLE NO.	AU (ppb)	ACTION
CRM ITAK 516	A-13016	356	The standard failed but in the same batch had other samples of the same standard that validate the batch.
GEOSTATS-G901-1	A-32239	2,047	The standard failed but in the same batch had other samples of the same standard that validate the batch.
GEOSTATS-G901-1	A-22260	2,172	The standard failed but in the same batch had other samples of the same standard that validate the batch.
GEOSTATS-G901-1	A-33046	575	The standard failed but in the same batch had other samples of the same standard that validate the batch.
GEOSTATS-G901-1	A-34371	2,080	The standard failed but in the same batch had other samples of the same standard that validate the batch.
CRM ITAK 531	A-66024	2,265	The standard failed but in the same batch had other samples of the same standard that validate the batch.
CRM ITAK 531	A-67007	3,344	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 531	A-90384	3,127	The standard failed but in the same batch had other samples of the same standard that validate the batch.
CRM ITAK 531	A-78474	2,258	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 531	A-79697	3,249	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 531	A-84298	2,295	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 531	A-85176	2,309	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 531	A-86255	2,236	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 531	A-88866	2,221	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 531	A-88729	3,177	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 531	A-91160	2,193	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 531	A-95290	3,387	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 531	A-96365	3,871	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 531	A-97360	3,113	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 531	A-97313	3,189	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 531	A-99280	2,252	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 531	A-101390	3,114	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 531	A-102055	3,341	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 531	A-107010	3,291	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 531	A-107093	3,136	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 531	A-111779	3,622	The standard failed but in the same batch have other samples of the same standard that validate the batch and it is not mineralized zone.
CRM ITAK 509	A-33893	3,128	The standard failed but in the same batch had other samples of the same standard that validate the batch.
CRM ITAK 509	A-33498	3,099	The standard failed but in the same batch had other samples of the same standard that validate the batch.
CRM ITAK 509	A-44466	3,044	The standard failed but in the same batch had other samples of the same standard that validate the batch.
CRM ITAK 509	A-44242	3,147	The standard failed but in the same batch had other samples of the same standard that validate the batch.
CRM ITAK 509	A-44330	3,142	The standard failed but in the same batch had other samples of the same standard that validate the batch.
CRM ITAK 506	A-39361	7,766	The standard failed but in the same batch have other samples of the same standard that validate the batch.
CRM ITAK 506	A-46911	7,933	The standard failed but was accepted because it is close to the limit. And the same batch had other samples of the same standard that validate it.
CRM ITAK 506	A-55318	9,694	The standard failed but in the same batch have other samples of the same standard that validate the batch.

11.5.7 INTERNAL BLANK SAMPLES

An acceptable blank sample is prepared from rock that is known to contain very low or non-detectable concentrations of the element being sought. A blank is used to monitor cross contamination that may occur during sample preparation, commonly as result of insufficient cleaning of the crushing, grinding or splitting equipment between samples. The blank sample used in the Almas Gold Project was prepared by SGS Geosol from core samples taken from the Hialino Quartz deposit in the Cristalina region of Goiás State, Brazil.

A total of 1,528 blank sample analyses were determined during the Almas Gold Project. The lower detection limit for gold by the fire assay-AA finish method was 0.005 g/t Au. The warning and failure thresholds for blank samples were establish at twice the detection limit (0.010 g/t) and three-times detection limit (0.015 g/t), respectively.

The results of the blank analyses from Rio Novo's drilling campaigns are presented graphically on the bottom of Figure 11-8.

The chart shows that most of the blank samples are below the failure threshold. For those blanks with gold assays above the failure threshold (0.015 g/t), ten samples surrounding the blank sample position in the batch were automatically re-assayed and compared to the original ten assays.

Of the 1,500 blank samples analyzed during the Almas Gold Project, 29 samples failed - a failure rate of less than 2%. As shown on Table 11-8, the most common notation associated with failure was that the blank was not in the mineralized zone and therefore the ten samples surrounding the blank were not re-assayed.

There were 16 blank failures for which the corresponding ten samples surrounding the blank were re-assayed, as a check on the possibility of cross contamination. The results of the re-assaying are shown on a scatter plot which yields a correlation coefficient of 0.999 (Figure 11-9). While the high degree of correlation indicates excellent precision, it does not address the possibility of cross contamination which would have occurred during the sample preparation stage, which is not assessed by re-assaying the original pulps, the majority of which are less than 1 g/t Au.

Based on the overall low failure rate of the blank analyses presented here, RPM expressed the belief that cross contamination during sample preparation is insignificant and, therefore, the blank results are acceptable by industry standard. The QP agrees with this conclusion.

Table 11-8 Analytical Blank Samples: Over Limit Assays

SAMPLE	AU (ppb)	HOLE	ACTION
A-1226	16	FPD-0002	Reanalysis of 10 samples surrounding the blank sample (A-1221 to A-1225 and A-1227 to A-1230 and A-1232)
A-2754	104	FAD-0010	Reanalysis of 10 samples surrounding the blank sample (A-2148 to A-2753 and A-2755 to A-2758)
A-6973	24	FAD-0025	Reanalysis of 10 samples surrounding the blank sample (A-6968 to A-6972, A-6974 and A-6976 to A-6979)
A-9535	20	FPD-0042	Reanalysis of 10 samples surrounding the blank sample (A-9530 to A-9534 and A-9536 to A-9539 and A-9541)
A-11957	35	FPM-0005	Reanalysis of 10 samples surrounding the blank sample (A-11952 to A-11956, A-11958 and A-11960 to A-11963)
A-12977	21	FPM-0006	Reanalysis of 10 samples surrounding the blank sample (A-12972 to A-12976 and A-12978 to A-12982)
A-24609	17	FPD-0083	Reanalysis of 10 samples surrounding the blank sample (A-24604 to A-24608 and A-24610 to A-24614)
A-24734	22	FPD-0091	Reanalysis of 10 samples surrounding the blank sample (A-24728 to A-24731, A-24733 and A-24735 to A-24740)
A-31344	23	FPD-0113	Reanalysis of 10 samples surrounding the blank sample (A-31339 to A-31343, A-31345 to A-31346 and A-31348 to A-31350)
A-29530	18	FRRC-0002	The blank sample failed but it is not in the mineralized zone. Re-assaying of 10 samples surrounding the blank sample was not necessary.
A-38139	16	FAD-0049	The blank sample failed but it is not in the mineralized zone. Re-assaying of 10 samples surrounding the blank sample was not necessary.
A-40318	18	FAD-0045	The blank sample failed but it is not in the mineralized zone. Re-assaying of 10 samples surrounding the blank sample was not necessary.
A-40252	16	FAD-0045	The blank sample failed but it is not in the mineralized zone. Re-assaying of 10 samples surrounding the blank sample was not necessary.
A-41763	22	FAD-0111	The blank sample failed but it is not in the mineralized zone. Re-assaying of 10 samples surrounding the blank sample was not necessary.
A-42149	19	FAD-0060	Reanalysis of 10 samples surrounding the blank sample (A-42144 to A-42148 and A-42150 to A-42154)
A-42853	238	FAD-0109	Reanalysis of 10 samples surrounding the blank sample (A-42848 to A-42852 and A-42854 to A-42858)
A-51977	26	FPD-0160	The blank sample failed but it is not in the mineralized zone. Re-assaying of 10 samples surrounding the blank sample was not necessary.
A-53654	18	FPD-0167	The blank sample failed but it is not in the mineralized zone. Re-assaying of 10 samples surrounding the blank sample was not necessary.
A-53537	17	FAD-0114	The blank sample failed but it is not in the mineralized zone. Re-assaying of 10 samples surrounding the blank sample was not necessary.
A-57999	22	FAD-0134	Reanalysis of 10 samples surrounding the blank sample (A-57994 to A-57998 and A-58000 to A-58005)
A-61753	18	FAD-0141	The blank sample failed but it is not mineralized zone. Re-assaying of 10 samples surrounding the blank sample was not necessary.
A-61943	39	FAD-0143	The blank sample failed but it is not in the mineralized zone. Re-assaying of 10 samples surrounding the blank sample was not necessary.
A-65030	93	FPD-0212	The blank sample failed but it is not in the mineralized zone. Re-assaying of 10 samples surrounding the blank sample was not necessary.
A-78160	19	FVSD-0003	Reanalysis of 09 samples surrounding the blank sample (A-78154 to A-78156 and A-78159; A-78161 to A-78165)
A-88100	85	FVSD-0048	Reanalysis of 10 samples surrounding the blank sample (A-88093 to A-88096, A-88099 and A-88101 to A-88105)
A-87901	23	FVSD-0047	Reanalysis of 09 samples surrounding the blank sample (A-87896 to A-87900 and A-87902 to A-87903; A-87905 to A-87906)
A-93895	22	FVSD-0085	The blank sample failed but it is not in the mineralized zone. Re-assaying of 10 samples surrounding the blank sample was not necessary.
A-96848	19	FVSD-0100	The blank sample failed but it is not in the mineralized zone. Re-assaying of 10 samples surrounding the blank sample was not necessary.
A-105947	17	FVSD-0151	Reanalysis of 10 samples surrounding the blank sample (A- 105942 to A-105946, A-105948 to A-105950 and A-105952 to A-105953)

11.5.8 INTERNAL FIELD DUPLICATE SAMPLES

A field duplicate sample is a second sample collected from the same location as the original sample. Duplicate sample results are used to assess variability associated with the ore type, laboratory analysis and sample collection process. During the Almas Gold Project, field duplicates comprised quartered core samples that were collected simultaneously and treated in an identical manner as the primary samples during storage, transportation, and analysis. In total, 1,760 field duplicate samples of core were made.

When two variables are linked by a statistical relationship, there is a correlation between them. The correlation reflects the degree of relationship between variables, in this case, gold assays from field duplicate samples. The scatter plot is a simple device to check the degree of the correlation for which a high degree of correlation is indicated by the data points plotting close to the trend line.

The poor correlation of field duplicate samples is reflected in the scattered distribution of data points and the low R squared value (0.39) which are indicative of the nugget effect, common in vein-type gold deposits and gold deposits in general (Figure 11-9A). As the nugget effect is a common characteristic of Precambrian mesothermal gold deposits, assays from field duplicates are not used to determine failed analytical batches.

As the nugget effect may be associated with coarse-grained, native gold, Rio Novo routinely used the metallic screen method (MET 150) on assays that return greater than 10 ppm Au. SGS Geosol re- analyzed 604 samples using the metallic screen method. This included 16 samples from Cata Funda, 37 from Paiol (2010 to 2011) and 551 from Vira Saia (2011 to 2012). The results from Cata Funda and Paiol are shown on Figure 11-10B. The correlation evident in an R squared value of 0.9 between the original assays and the metallic screen assays suggests that bias due to coarse gold is not evident in the data from Paiol and Cata Funda. At Vira Saia, 69 of 551 samples (14%) were in the nominal range of 1.0 to 4.4 ppm Au for which the basic statistics are given below.

	Original Assay (Au g/t)	**Metallic Screen Assay (Au g/t)**
• Mean	3.8	3.9
• Std Dev.	5.7	5.8
• Max	4.3	4.4
• Min	1.0	1.1
• R sq	1.0	1.0
• Coef. Variation	1.5	1.5

The strong correlation expressed in the R squared value (1.0) suggests that coarse gold is not evident in the Via Saia data and that the moderate coefficient of variation (1.5) suggests the anisotropic distribution of gold in the coarse reject material from which the metallic samples were made. The relatively low correlation of the field duplicates of core (0.39) is also consistent with this style of mineralization in the project area. The latter is not unexpected in Precambrian mesothermal gold deposits in which the nuggety local distribution of gold is controlled by complex, anastomosing alteration zones and quartz veins that pinch and swell within the mineralized structure.

Figure 11-9 Internal Duplicate and Blank Analyses

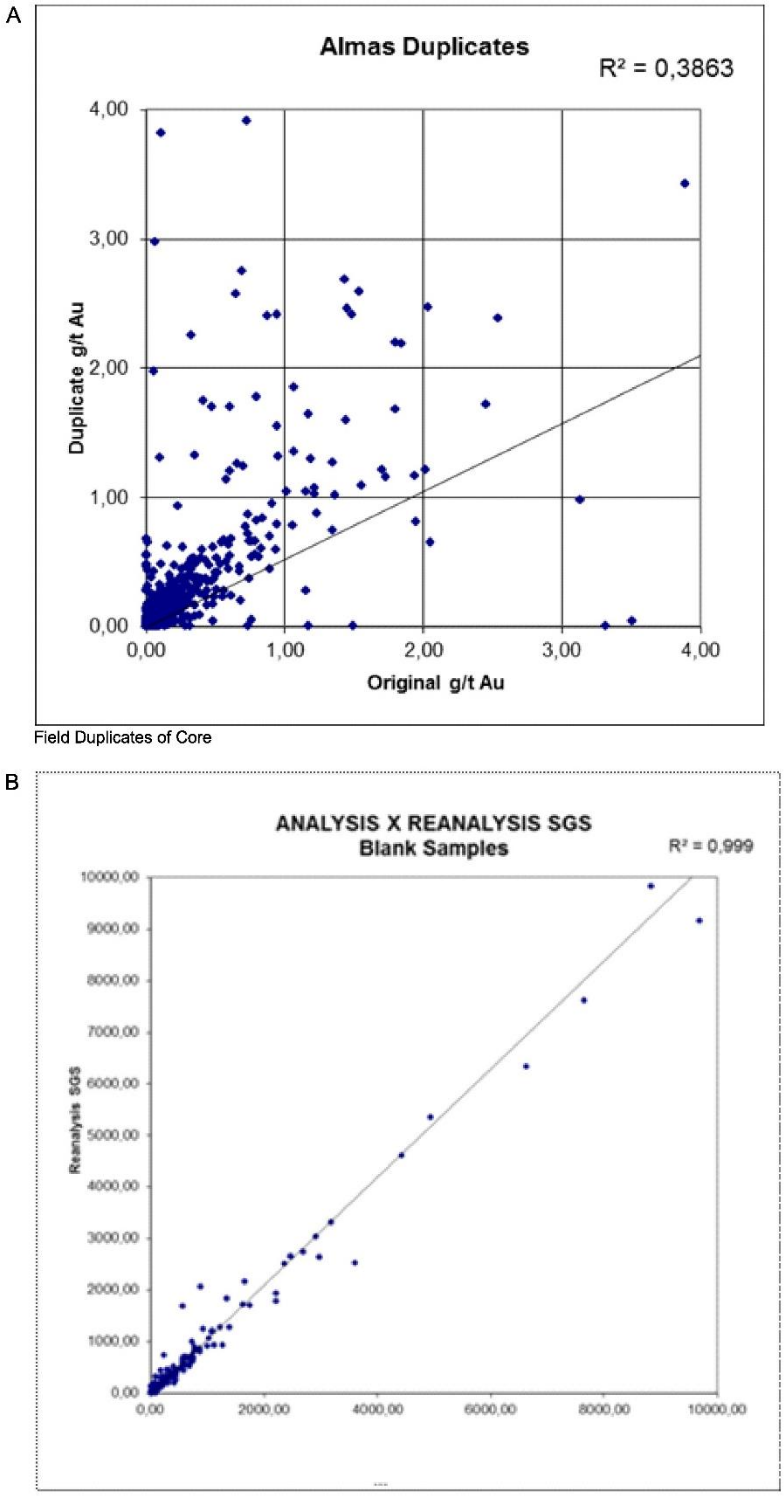
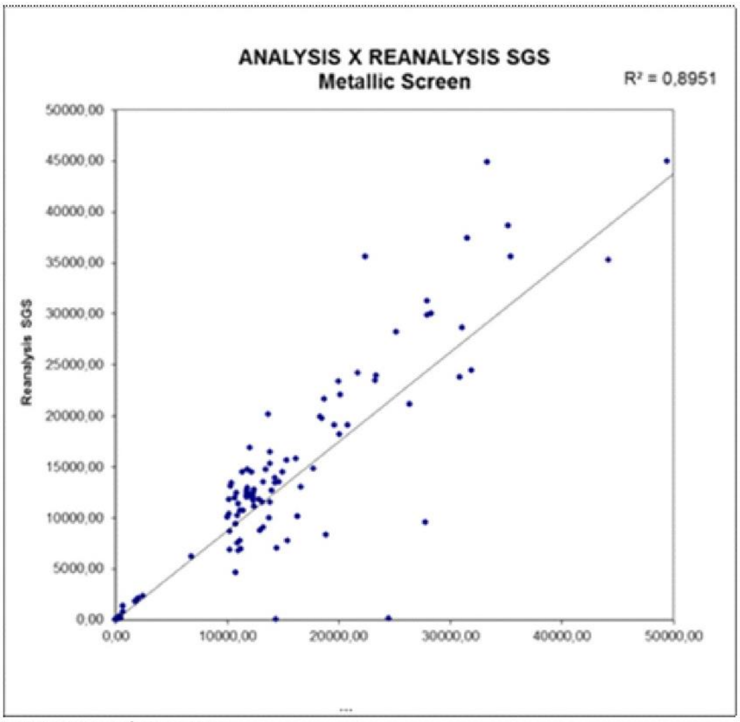


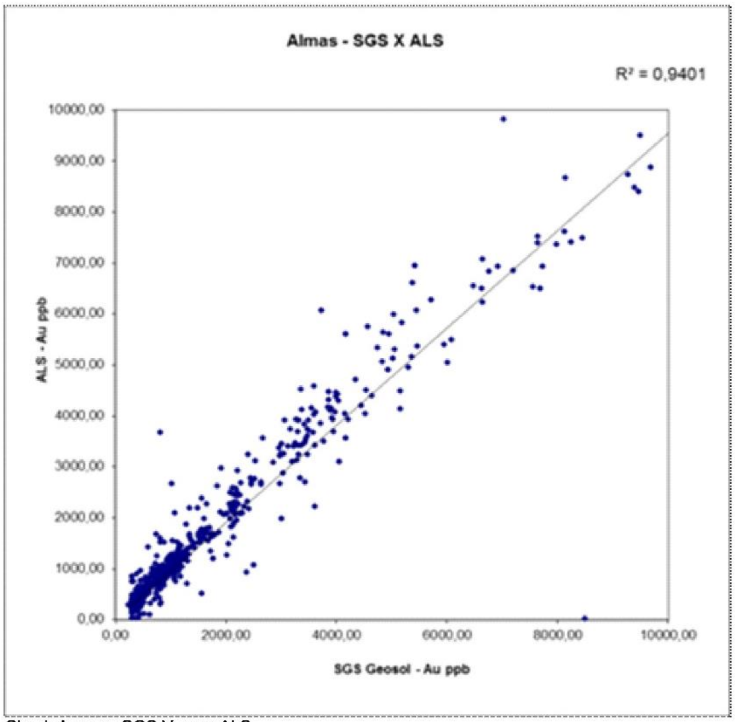
Figure 11-10 Metallic Screen Test and Check Assays

A



Internal Metallic Screen Tests

B



Check Assays: SGS Versus ALS

11.5.9 INTERNAL DUPLICATE CHECK SAMPLES (SGS GEOSOL LAB – ALS LAB)

As part of Rio Novo's internal QA/QC program, approximately 5% to 10% of sample pulps were re-assayed at the ALS Laboratory in Belo Horizonte, which functioned as the Project's secondary lab.

The relative precision between the SGS Geosol and ALS analyses is given on Figure 11-10b for 649 duplicate pulp samples analyzed at both labs from 2010 to 2012. The scatter plot shows good correlation for the duplicate analyses expressed in an R squared value of 0.94.

Based on the overall low failure rate of the blank analyses presented here, RPM expressed the belief that cross contamination during sample preparation was insignificant and therefore the blank results are acceptable by industry standard. The QP agrees with this observation.

11.6 EXTERNAL LABORATORY QA/QC PROGRAMS

The SGS Geosol and ALS laboratories insert blanks, CRMs and duplicate samples into each analytical batch of 40 samples. The SGS Geosol laboratory inserts replicate samples into each batch. The results of the external QA/QC program by SGS Geosol were included with the routine assay results while ALS provides a separate QA/QC report on request. External QA/QC results were archived at the respective laboratories and a copy of each was retained in Rio Novo's digital database.

Both laboratories deliver analytical results in digital format to the database manager. Certificates of Analysis were provided separately. Copies of the digital files and certificates were stored in Rio Novo's digital database for use in data validation.

11.6.1 EXTERNAL QA/QC: BLANK SAMPLES

The acceptance - rejection thresholds for external blank analyses by SGS Geosol were the same as established for Rio Novo's internal blank testing: detection Limit = 0.005 g/t, warning threshold = 0.010 g/t, failure threshold 0.015 g/t. The chart shows that most blank samples were below the warning threshold limit with only three samples above the failure limit in November, 2012 (Figure 11-11). In total 3,455 external blank assays were returned by the laboratories for which no contamination was observed by Rio Novo personnel.

11.6.2 EXTERNAL QA/QC: DUPLICATE AND REPLICATE SAMPLES

Duplicate samples prepared from coarse reject material and replicate pulp samples were used as external control samples at the laboratories. Two methods of control analysis were used: Batch- and Global population-based as discussed previously in Section 11-5.5. As shown on Figure 11-12A, a total of 3,831 accumulated duplicate samples were analyzed by the SGS Geo laboratory from 2010 to 2012. A high degree of correlation is shown between the duplicates and the original assays as defined by an R squared value of 0.99.

As a small number of samples failed, the "batch-based" method was used to control "global population". In Global population, 90% of the sample population should have relative differences below 20%. Of the 3,500 external duplicates analyzed, only 26 samples were rejected, less than 1%, in "batch-based" control, therefore no batch was rejected. Table 11-9 summarizes the samples rejected under the batch-based criterion.

As shown on Figure 11-12B, a total of 2,183 accumulated replicate samples of pulps were analyzed by the SGS Geo laboratory during the drilling project (2010 - 2012). A high degree of correlation is shown between the replicates and the original assays as defined by an R squared value of 0.99.

Since some samples failed to control "batch-based" was used to control "global population". In "global population" 90% should have relative differences below the 10% mark. Over 2,000 Replicate's laboratory samples were analyzed and 85 (< 5% total) samples failed in "Batch-based" control, so no batch was rejected.

As a small number of samples failed, the "batch-based" method was used to control "global population." In global population, 90% of the sample population should have relative differences below 20%. Of the 2,183 external replicates analyzed, only 85 samples were rejected, less than 5%, in "batch-based" control, therefore no batch was rejected.

Figure 11-11 External Blank Analyses - SGS Geosol November 2012

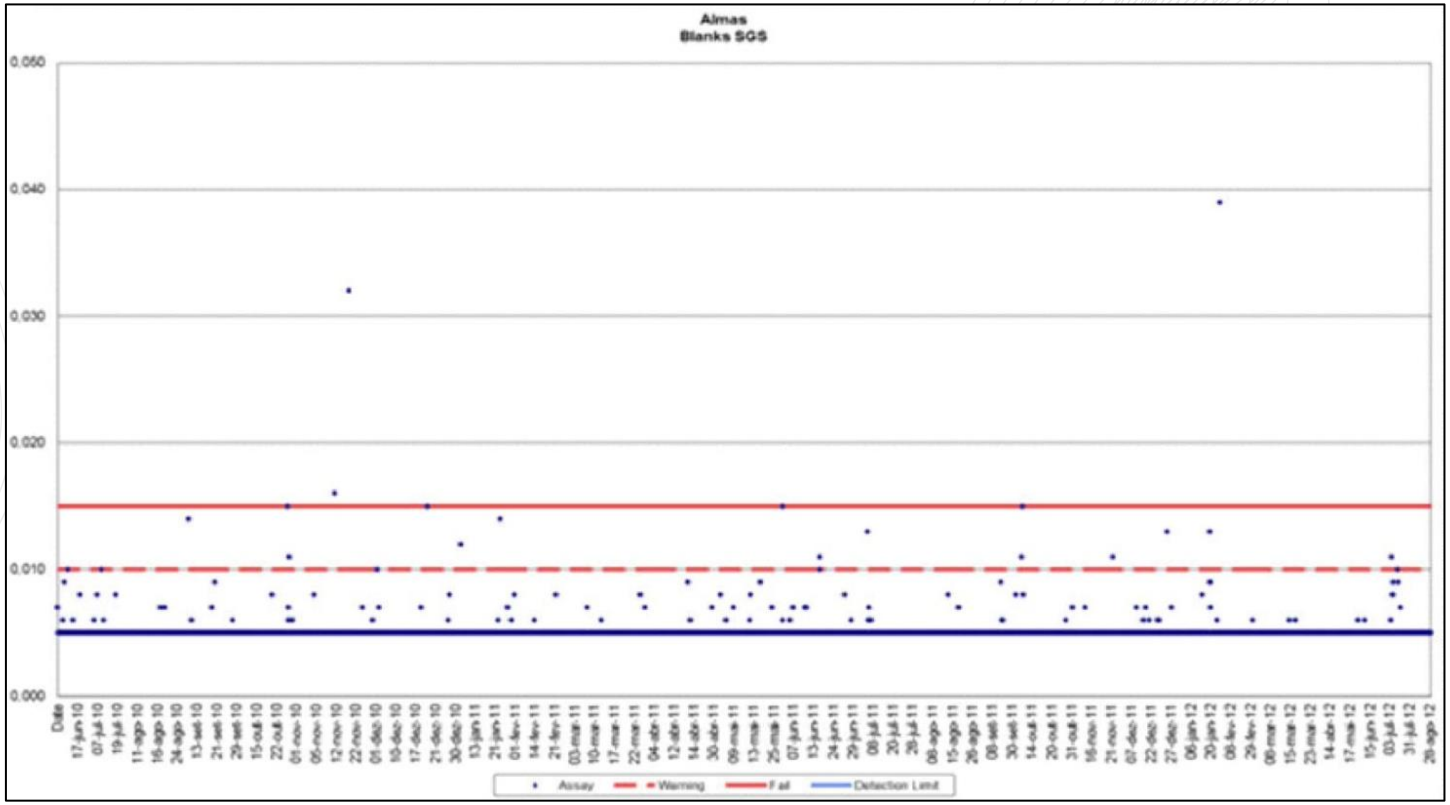
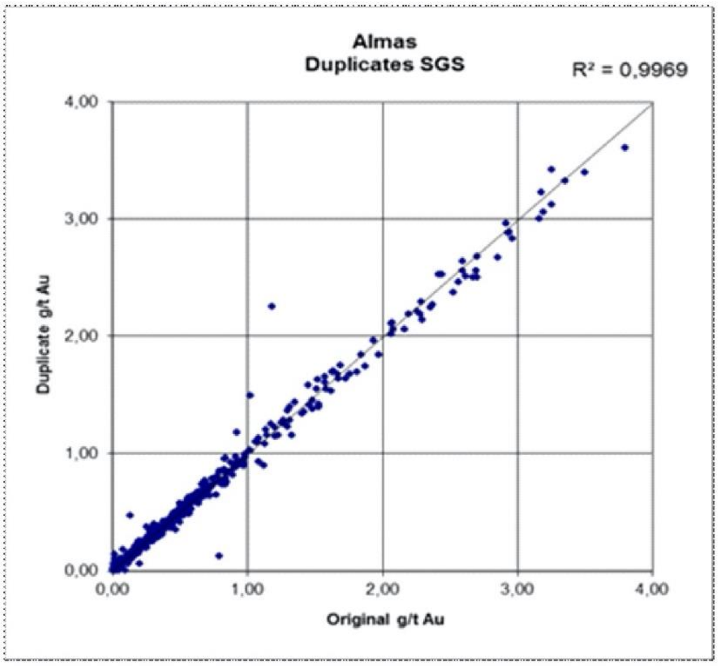


Figure 11-12 External Duplicate and Replicate Analyses - SGS Geosol

A



B

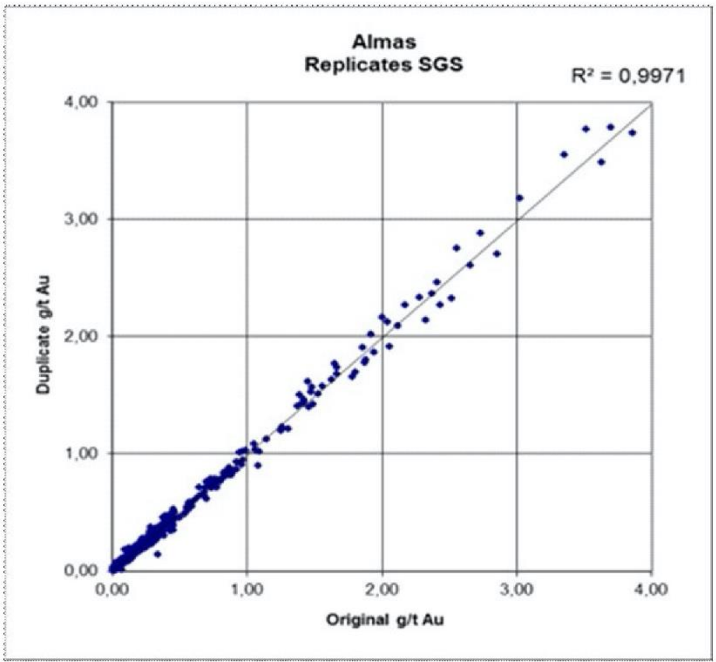


Table 11-9 Laboratory Duplicate Samples: Over Limit Assays

SAMPLE NO.	AU (ppb) ORIGINAL SAMPLE	AU (ppb) DUPLICATE SAMPLE (SGS)	CAUSE OF FAILURE
A-11006	15	102	|A-B| < 3DL
A-111425	4,99	62	|A-B| < 3DL
A-12734	92	4,99	|A-B| < 3DL
A-13450	43	97	|A-B| < 3DL
A-20200	57	79	|A-B| < 3DL
A-20511	10	27	|A-B| < 3DL
A-21693	78	58	|A-B| < 3DL
A-22052	60	76	|A-B| < 3DL
A-26212	55	72	|A-B| < 3DL
A-30294	27	4,99	|A-B| < 3DL
A-30535	50	6	|A-B| < 3DL
A-3581	57	38	|A-B| < 3DL
A-3723	14	42	|A-B| < 3DL
A-49540	46	62	|A-B| < 3DL
A-51305	59	79	|A-B| < 3DL
A-51725	48	32	|A-B| < 3DL
A-55414	64	81	|A-B| < 3DL
A-56036	65	83	|A-B| < 3DL
A-5679	38	21	|A-B| < 3DL
A-64233	31	47	|A-B| < 3DL
A-85979	83	66	|A-B| < 3DL
A-97723	52	68	|A-B| < 3DL
A-2766	19	144	|A-B| * 0,5 / (A+B) < 20%
A-3762	131	470	|A-B| * 0,5 / (A+B) < 20%
A-5539	791	129	|A-B| * 0,5 / (A+B) < 20%
A-6717	199	61	|A-B| * 0,5 / (A+B) < 20%

11.6.3 EXTERNAL CONTROL - CRM SAMPLES

The SGS Geosol Laboratory used 29 CRMs during Rio Novo's drilling programs as listed on Table 11-10.

Table 11-10 Certified Reference Materials for Gold used by SGS Geosol Laboratory

CERTIFIED REFERENCE MATERIAL	TYPE	CERTIFIED VALUE (g/t)	STANDARD DEVIATION	NUMBER OF ANALYSES
CDN_GS_P2	Low grade	0.214	0.020	76
CDN_GS_P2A	Low grade	0.229	0.030	60
CDN_CM_5	Low grade	0.294	0.046	95
CDN_GS_P3A	Low grade	0.338	0.022	171
ITAK523	Low grade	0.399	0.020	33
G311-7	Low grade	0.400	0.030	56
CDN_CM_7	Low grade	0.427	0.042	400
CDN_GS_P4A	Low grade	0.438	0.032	257
G909-10	Low grade	0.520	0.050	39
AUOE-7	Low grade	0.623	0.030	114
G908-4	Low grade	0.960	0.050	27
CDN-GS-1F	Middle grade	1.16	0.130	108
CDN_CM_4	Middle grade	1.18	0.120	76
CDN_CM_6	Middle grade	1.43	0.090	304
G399-2	Middle grade	1.46	0.090	74
CDN-GS-1P5D	Middle grade	1.47	0.150	247
G901-7	Middle grade	1.52	0.060	31
CDN-GS-1P5C	Middle grade	1.56	0.130	385
ITAK522	High grade	1.94	0.073	20
G905-8	High grade	2.55	0.130	19
G909-5	High grade	2.63	0.100	38
CDN_GS_3E	High grade	2.97	0.270	6
CDN_GS_4B	High grade	3.07	0.350	571
AUSK-4	High grade	3.57	0.156	6
G903-6	High grade	4.13	0.170	1
G910-5	High grade	5.23	0.210	3
AUON-4	High grade	5.94	0.262	17
CDN_GS_8A	High grade	8.25	0.600	24
CDN_GS_20A	High grade	21.12	1.54	27

Variation plots showing the behavior of three, representative CRMs analyzed by SGS Geosol is presented in Figure 11-13. The SGS Geosol CRMs are: CDN_CM_7, CDN-GS-1P5C and CDN_GS_4B).

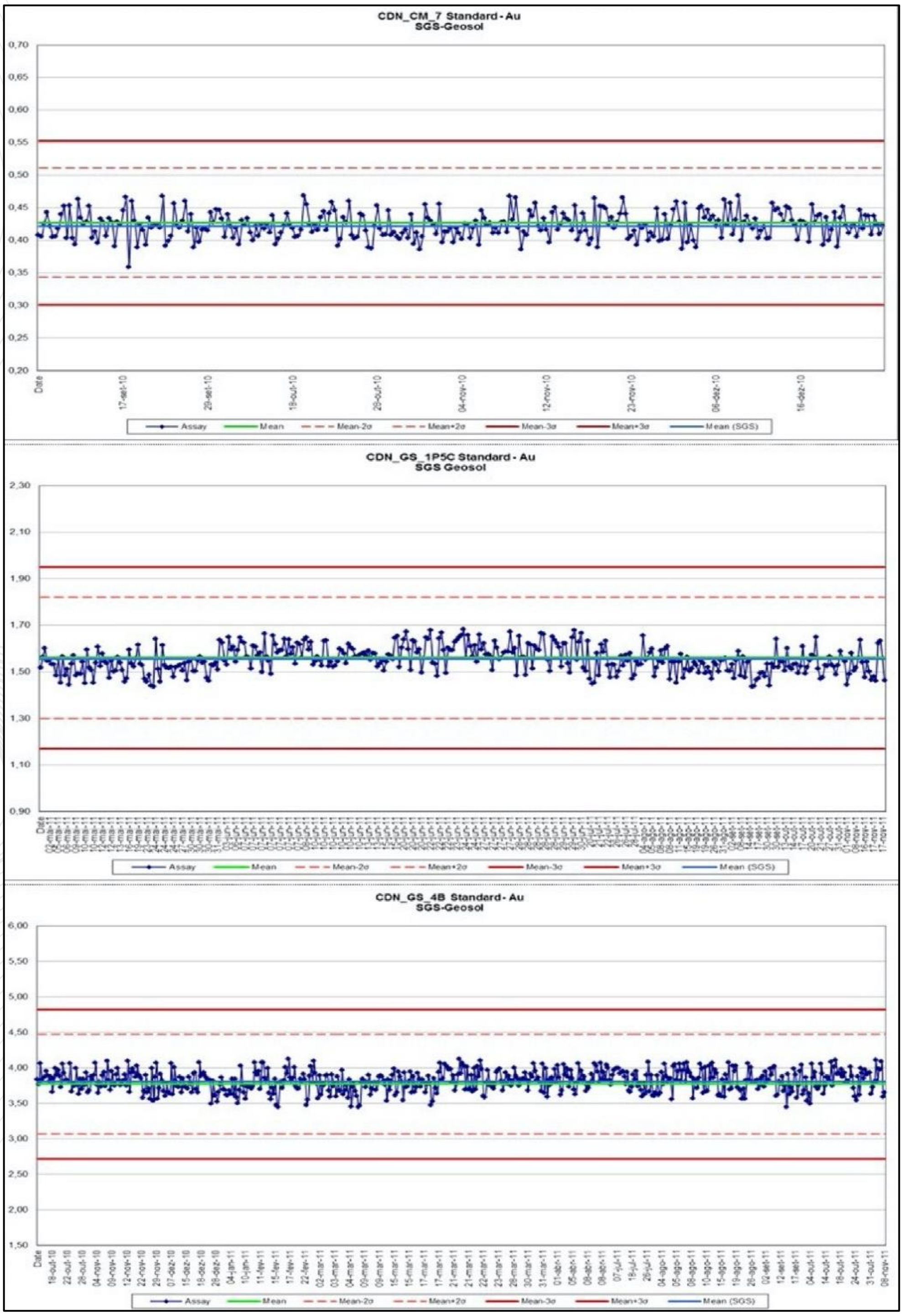
The charts show a high degree of analytical accuracy and precision as all CRM analyses were well within the warning threshold of two standard deviations of the mean. Therefore, no analytical batch was rejected by virtue of external CRM performance.

11.7 INTERNAL QA/QC SUMMARY

Of the 1,500 blank samples analyzed during the Project, 29 exceeded the failure threshold. Of the 29 samples, 16 samples were in the mineralized zone which required re-analysis of 10 surrounding the failed samples according to the Rio Novo QA/QC protocol. Re-analysis of these samples showed no contamination. The other 13 blank samples that failed were not in the mineralized zone; therefore, re-assaying of ten samples surrounding each failed blank was not required.

Over 1,700 field duplicate samples were prepared from quartered core and analyzed by SGS Geosol. A low correlation was found between the original and field duplicate samples expressed with an R squared value of 0.39 calculated from a scatter plot of the data. This low correlation factor suggests a nugget effect in the assay data however, assays from metallic screen testing did not suggest a coarse gold issue at the three targets in the Project area. The wide scatter in the field duplicates precludes using those data for defining failed analytical batches.

Figure 11-13 Variation Plots for CRM Samples



Over 3,600 CRMs (standards) were analyzed during the drilling campaigns of which 61 samples exceeded the failure threshold (<2% total). Most failed CRMs were determined in batches for which other samples of the same CRM returned results below failure thresholds and, therefore, the analytical batches were accepted. Other failed CRMs were outside the mineralized zone and were not considered to be material.

11.8 EXTERNAL QA/QC SUMMARY

Over 3,400 SGS Geosol blank samples were analyzed. Of these analyses only three blanks exceeded the failure threshold.

Duplicate samples prepared from coarse reject material and replicate pulp samples were analyzed by the laboratory. Two types of control methodology were used: batch-based and global population-based. Over 3,500 duplicate samples were analyzed with only 26 samples exceeding the failure threshold in “Batch-based” control. Over 2000 replicate samples were analyzed with only 85 samples exceeding the failure threshold in “Batch-based” control. Since some samples failed control levels, "Batch-based" was used to control "Global population".

In "Global population" 90% should have relative differences below the 10% mark. Therefore, no batch was rejected by virtue of the performance of duplicate and replicate samples.

The SGS Geosol Laboratory used 29 CRMs during the Project. Over 3,200 CRM samples were analyzed by the laboratory for which all determinations were well within the warning threshold of two standard deviations, demonstrating good analytical accuracy and precision.

11.9 ANALYTICAL BIASES

During Vale´s mining operation in the late 1990s extensive research by their consultants, confirmed both the lack of bias due to rock type or alteration, and confirmed that no significant difference exists between the samples collected from reverse circulation drilling with those derived from core samples. Rio Novo had similar confirmation in their extensive drilling campaigns since 2010. Since Vale discarded drill holes with recovery less than 80%, and recovery for the drilling completed by Rio Novo exceeded 95%, RPM expressed the belief that there were no known recovery biases within the assay database. The QP concurs.

For the 2008 drilling campaign, the sampling procedures and quality control procedures were established and implemented under the direction of MultiGeo, a subcontractor to Rio Novo’s predecessor, and were used in earlier resource estimates. RPM did not independently inspect the Vespasiano assay facility near Belo Horizonte that was used for sample preparation and assaying of all Rio Novo samples.

11.10 SAMPLE SECURITY

Rio Novo exploration drilling for the 2008 campaign was completed mainly by contractors, with MultiGeo technical staff providing the technical oversight. Drill core was kept at the drill rig until transferred to the core processing facility at the end of each shift. The core was logged, photographed and sawed and shipped to ALS laboratory in sample sacks. During this time access to the samples was restricted to only drill crews, supervisors and project staff.

RPM considered the analytical protocol and sample security then in use, and that was used during the major drilling campaigns in 2010 and 2011, sufficient to allow the estimation of mineral resource estimates from the data collected, and generally meeting acceptable mining industry standards. The QP concurs.

11.11 SAMPLING METHOD AND APPROACH - FORMER OPERATORS

11.11.1 DIAMOND DRILL CORE SAMPLES

In 1998 MRDI described sample handling by Vale. Samples were prepared by jaw crushing to minus one-fourth inch (in) for either the entire one-half core split or 2 kg for the RC samples. The sample was then homogenized by quartering through a Jones splitter to produce two 200 g to 300 g splits; one of the splits was archived as a coarse reject ,and the other split was ground to 80%, -325 mesh. All samples were assayed for gold by fire assay with AA finish. Until 1995 assays were performed at the Fazenda Brasileiro Mine laboratory; after 1995, samples were assayed at the Paiol Mine site.

Core drilled by METAGO was prepared by crushing to one-fourth inch in a jaw crusher and pulverizing to -450 mesh. Assays were completed using conventional fire assay techniques with a gravimetric finish (MRDI 1998).

For Mineração Apuã’s drilling program in 2008, one-half of the drill core was stored for reference, and the other half was bagged, numbered and submitted to the primary laboratory ALS in Belo Horizonte for preparation and analysis. For each sample consignment, the following forms were completed prior to shipping to ALS - (CORE).

- Sample submittal form.
- Monitoring letter issued by Mineração Apuã Ltda.
- List of samples in shipment.
- Request form for samples preparation and chemical analysis.

Mineração Apuã Ltda. used ALS as the primary analytical laboratory for drill core. ALS is an international analytical service and was corporately accredited to ISO 9001:2000. ALS has a preparation laboratory in Belo Horizonte and an assay laboratory in Lima, Peru. Mineração Apuã Ltda. used SGS Geosol Laboratórios Ltda. (SGS Geosol) in Belo Horizonte as a secondary laboratory. SGS Geosol is an ISO14001 and ISO 9001:2000 accredited international laboratory service.

11.12 DIAMOND DRILL CORE SAMPLE RECOVERY

Drill core recovery measurements were collected by Rio Novo staff on each hole as it was being drilled. This has been done for all diamond drilling. Recovery data were recorded as driller's length and recovered length, with both measurements recorded against driller's depth. In general Rio Novo's drilling during the 2010 and 2011 programs has recovery in the +95% range.

Since Vale discarded drill holes with recovery less than 80% and recovery for the drilling completed by Mineração Apuã Ltda. was greater than 95%, recovery issues were not a problem with pre-Rio Novo data. The exceptions are minor, and RPM opined it will not impact the resource model estimates. The QP concurs.

11.13 CONCLUSIONS

The QP is satisfied with the adequacy of the sample preparation, security, and analytical procedures employed and concludes that they have resulted in data suitable for use in a mineral resource estimate.

12 DATA VERIFICATION

To date Aura has completed no drilling at the Almas Gold Project. Some resampling of older core and trench sampling of a loaded leach pad has been conducted but the samples were sent for metallurgical test work. The results were not used in the mineral resource estimate.

At the time of the QP's site visit no staff from the previous operators were available for consultation. Sections 12-1 to 12-4 are adapted from Rio Novo's 2016 feasibility study by RPM (RungePincockMinarco, 2016). The QP has also reviewed Rio Novo's monthly QA/QC reports. Section 12-5 describes the QP's data verification.

The following sections describe the processes used by RPM to verify the data in the Almas Gold Project study for their feasibility study. This section summarizes three levels of data verification:

- Verification of the original or historic data in the current database from drilling by the various project operators previous to Rio Novo.
- The quality control and verification procedures used by Rio Novo during drilling campaigns from 2010 to present.
- The final verification by RPM for this study.

RPM's summary focuses on the verification of diamond core drilling data. For the purposes of its study, Rio Novo used mainly diamond core drilling results. There are RC drill holes in the Paiol pit area, but they only sample the saprolite and oxidized rock near surface, which has been mined out. Reverse circulation drilling was used in this study to estimate mineral resources on the loaded leach pad. Channel samples were collected from trenches dug on the leach pad but were used for metallurgical testwork. Reverse Circulation drilling samples were assayed by the same methods as the drill core.

12.1 HISTORICAL DATA

Drilling was conducted at the Almas Gold Project between 1987 and 2008 by three companies: Metago, VALE, and Santa Elina. The historical drilling used in relation to this study was conducted at Paiol and Cata Funda. Vira Saia drill results used in this study were generated by Rio Novo-supervised drilling.

In 2010, Rio Novo commissioned Minerotec Consultorio & Servicios to consolidate, review, and validate all historic data on the project (Schumacher, 2010). The Schumacher report addressed:

- Drill hole collar surveys
- Downhole directional surveys
- Drilling and Geological logs
- Analytical results

The Schumacher 2010 report makes several recommendations to Rio Novo that would improve the reliability of the database. These include:

- Survey of coordinates of six old drill holes in the Paiol Mine area.
- Survey of coordinates of 10 additional old drill holes in the Paiol Mine area and eight old drill holes marks in the Arroz (Cata Funda) area. It's necessary to report the procedure used to survey these collar coordinates (equipment type, precision, photos of marks and table with comparison of original coordinates and new ones).
- Request from VALE a formal document and/or original analytical certificates of gold assays not reported by Docegeo.
- Re-sampling and re-assaying of four Santa Elina twin drill holes, using exactly the same procedures and methodologies in current use by Rio Novo Mineração, including sampling procedures, chemical assay method, laboratory and QA/QC standards. The drill holes suggested to be re-assayed are:
- FD-01A-SE or FD-19A-SE

- FD-04A-SE or FD-22A-SE
- FD-16A-SE or FD-18A-SE
- FD-10A-SE or FD-28A-SE

Rio Novo re-assayed 997 Santa Elina samples. The results for all 997 samples are shown in Figure 12-1. Overall, the Rio Novo samples are slightly lower than the Santa Elina samples, and the scatter of samples is significant. A review of the samples with Rio Novo assays ≥ 200 ppb (0.2 g/t) which includes 137 samples out of the original 997 samples is shown in Figure 12-2. The average difference between the Rio Novo samples ≥ 200 ppb and the Santa Elina assays is -2.93% with the Rio Novo samples being slightly lower. But, as can be seen in Figure 12-2, the scatter is significant with an R squared value of 0.75. These results are similar to those seen in later QA/QC results of Rio Novo.

12.1.1 DRILL HOLE COLLAR LOCATION SURVEYS

The validation study selected 21 drill holes at random from the Metago database and compared the survey coordinates with the original data records in the DNPM final exploration report. This was about 20% of the total Metago drill holes and there were no problems detected.

From the VALE database, the review randomly selected 17 drill holes from the Arroz database, or about 10% of the total drill holes, and compared them to information listed in the original VALE report on the Arroz target. No problems were detected. As a secondary check, the plotted locations on a map were compared to the original maps and the collar locations aligned.

At the Paiol area, 45 drill holes were selected from the VALE database and compared to the records in the DNPM final exploration report. No discrepancies were noted.

In 2011, after completion of the Minerotec validation study, Rio Novo completed further validation of the collar surveys. The Rio Novo study selected 37 old drill holes in the Paiol area and 12 old drill holes in the Cata Funda area. A total station survey instrument, with precision to ±5 cm, was used to resurvey the drill hole monuments. There were no significant differences between the original coordinates and the new survey data.

12.1.2 DOWN HOLE DIRECTIONAL SURVEYS

The bearings and inclinations listed for the Metago drill holes were compared to records in the DNPM final exploration report. Of 21 drill holes selected, all the collar bearing, and inclination records checked out against the DNPM report. There was no downhole survey procedure documented for the Metago drill holes; however, in the digital database received from VALE there is downhole survey information. These data were not checked as the original records were not available. The review also verified the depths of the drill holes were correct versus the original data. Thus, for the Metago database, the collar bearing, inclination, and total depth appear to verify against older records; however, there is no confirmation for the downhole survey coordinates.

The diamond core drilling of VALE and Santa Elina was surveyed downhole using a Maxibor survey instrument. However, the database validation study was only able to find original survey raw data for four holes, FD-52 to FD-55. These four were compared to the digital database, and all were confirmed against the original raw data.

Figure 12-1 Comparisons of Reassays

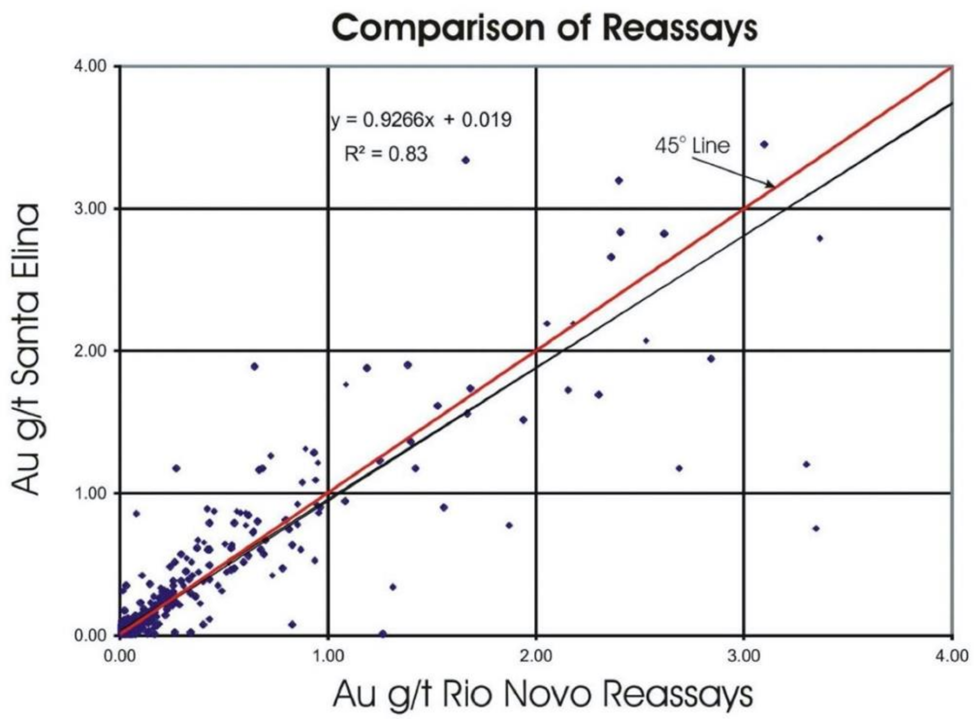
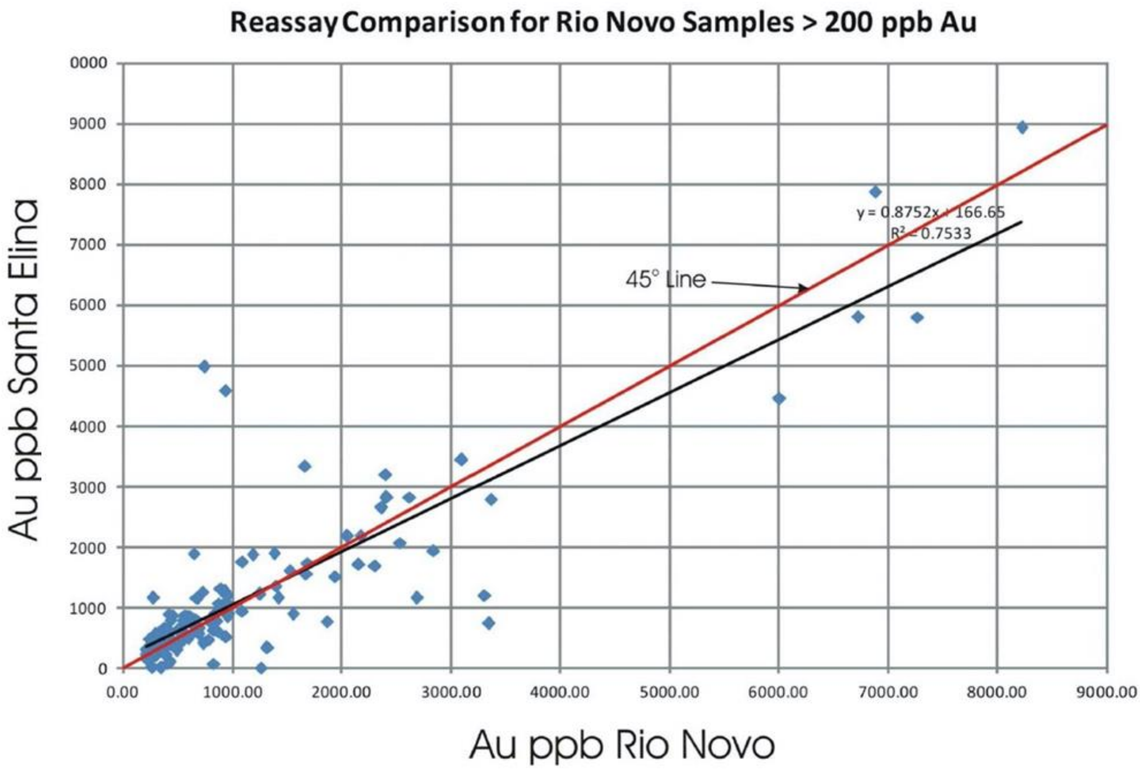


Figure 12-2 Reassay Comparisons For Rio Novo Samples >200 ppb Au



12.1.3 DRILLING AND GEOLOGICAL LOGS

The digital database received from VALE was verified for geology by comparing to the original logs. The validation study selected 11 drill holes from the Metago era and 30 drill holes from the VALE drilling for comparison. The selection process was random and represented about 5% of the data. The digital data checked versus the original logs.

12.1.4 ANALYTICAL RESULTS - ASSAY VALIDATION

The Minerotec review process selected 11 drill holes from the Metago data and 30 drill holes from the VALE data for validation of assays. This represented about 5% of the Metago data and 5% of the VALE drill hole data. First, a logical validation test was completed to check for any gaps or numerical errors in the sampling intervals. No problems were found. Next, the assay information in the database was compared to the original certified assay reports from the lab. All assays checked out. However, it was noted in both the Metago and VALE data, gold assays reported as 0.01 ppm were actually -0.01 ppm on the original certificates, thus meaning below detection limit. These were then corrected in the Rio Novo database.

12.2 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.

Independent consultant (Mr. Ron Simpson) reviewed the procedures of Rio Novo and SGS Laboratory that have been used for sampling, preparing and assaying samples.

12.3 RUNGEPINCOCKMINARCO DATA VERIFICATION

Bart Stone, a QP, C.P.G. and Director of the RPM office in Brazil, visited the project from December 10 to December 14, 2012 to observe the conditions. While there he visited each of the proposed mine sites and reviewed core and core logs along with assay data. The mineralization observed in the core and in the field at Almas was noted to be as modeled, a symmetrical lower grade fringe bordering both sides of a higher grade core.

The structural zone containing the mineralization continues in excess of 15 kilometers, and along this length there are numerous additional indications of gold mineralization, as well as areas of banded iron formation. What was found was a brownfields deposit at Paiol (mined by Vale from 1997 until 2002 producing 86,000 ounces of gold) and two greenfield discoveries at Cata Funda and Vera Saia. Extensive mining was noted at all three sites as evidenced by water-filled linear pits along the mineralized fault line. The timing of the mining, except for Paiol, is unknown and the miners involved are unknown, but certainly there is good evidence that economic values were extracted by VALE at Paiol, and likely garimpeiros at the remaining two proposed mine sites. Typically, garimpeiros seldom chase anything less than 4 opt in their operations.

Observations of the core revealed numerous thin zones of plus 10% pyrite, some pyrrhotite and one small grain of gold. The geology is as logged for the most part.

Based on historical activity and review of VALE reports on Paiol, RPM opined that there is likely gold values present in the amounts and grades estimated in the 2016 resources and the mineralization has the geometry suggested by the geological models. For these reasons RPM had no evidence to suggest additional verification sampling was necessary for confirmation of the presence of gold in the area.

12.4 RPM'S LIMITATIONS TO DATA VERIFICATION

One limitation noted to the verification process by RPM was the lack of original raw data to confirm downhole surveys of the drilling. As noted above, the raw downhole survey data was found from only four drill holes from the Metago and VALE era. This represents about 2% of the data. (There were 217 core drill holes from the era.) In addition, the holes were all drilled in sequence and used the same survey process. Although 2% is less than ideal, the four surveys matched the database versus the raw data.

A second limitation noted is the lack of certified assay reports for the older Metago and VALE drilling. For this reason, the older assays in the database cannot be cross-checked with the original assay reports. Rio Novo did not drill twin holes to verify any of the historical results. Typically, when historical results are used in resource and reserve estimations approximately 10% of the historical drilling is twinned to verify the results and confirm the historical results are consistent with the current standards of the industry. Without direct twin results, the only way to compare the results of the historical drilling and the NI 43-101-compliant drilling is within the context of the resource estimations. RPM found that in the Almas Gold Project geological and resource models, there are no apparent geostatistical discontinuities that result from the combined historical and current data.

While there are limitations within the data verification process, it was the opinion of RPM that the risks due to these limitations are limited. The QP concurs.

12.5 MICON DATA VERIFICATION

Micon's QP travelled to Brazil on May 2, 2019 and visited the project sites and warehouse in the town of Palmas on May 3 and 4.

12.5.1 DRILLING

All of the drilling at the Almas Gold Project has been completed by previous operators. As such, no technical staff intimately familiar with drilling and logging protocols were available for questioning during the site visit. However, the procedures used for diamond drilling by Rio Novo are reasonably well documented in the RPM report.

No description of the reverse circulation (RC) drilling, sampling or assaying procedures is given in the report. RPM also expressed concern over the lack of knowledge of the Quality Assurance/Quality Control (QA/QC) procedures implemented by Vale during RC drilling. For this latter reason, Rio Novo and RPM chose not to use the RC results in mineral resource estimation or describe the procedures used. Given their concerns, the QP concurred with this decision pending further validation (see Section 12.5.5.2).

At Paiol, 459 diamond drill holes were completed of which 259 were drilled by Rio Novo, 105 by Metago, 55 by Vale and 48 by Santa Elina. At Cata Funda, 180 diamond drill holes were completed, of which 9 were by Vale, 23 by Santa Elina and 148 by Rio Novo. At Vira Saia all 189 diamond drill holes were completed by Rio Novo.

At the time of the site visit Aura had created two block models for Paiol, one using only diamond drill holes and the other using diamond and RC holes. Aura reports that there was little difference between the two models and that most of the RC holes were short, mostly in saprolite and weathered rock and are currently mined out. That usage will be confirmed below in Section 12.5.5.2.

The core logging facility used by Rio Novo during drilling, which was reported to have been located at the Paiol mine, had been dismantled and the core boxes moved to a secure rented warehouse in Almas.

The diamond drilling procedures described in the RPM Technical Report indicate that Rio Novo used industry standard practices in its program. While no Rio Novo personnel were available for questioning by Micon, RPM had access to them and was comfortable in validating the procedures used. The report does not describe the procedures used by Metago, Vale and Santa Elina. Further discussion about this and the related sampling and assaying is contained in Section 11.

A review of selected drill holes from all three deposits and field visits to the sites during the QP's site visit confirmed the geological model described by Aura and in the RPM report.

12.5.2 DATA FROM OTHER OPERATORS

The RPM report does not fully document the logging sampling, sample preparation and assaying procedures employed by Metago, Vale or Santa Elina. The report does refer to a study Rio Novo commissioned from Minerotec Consultorio & Servicios to consolidate, review, and validate all historic data on the project (Schumacher, 2010). The Schumacher report is discussed in Section 12.1 above.

The RPM report also summarized Schumacher's results of checks to drill hole collar location surveys, down hole directional surveys, drilling and geological logs and analytical results - assay validation. No serious issues were noted there.

12.5.3 DATA ENTRY

Micon's QP arranged for checks of the entry of a portion of the gold assay data in the database against all of the laboratory certificates provided by Aura. Table 12-1 summarizes the result of the checks.

Table 12-1 Almas Project Assay Entry Checks

STATUS	SAMPLE COUNT
Samples with different entries	369
Samples truncated at 2 decimal places	27,581
Detection limit samples	42,780
Samples not found in certificates	1,265
No sample ID *	73,351
Okay	16,814
Grand Total	162,160

* Note: 14,254 samples have no data and 59,097 have no Sample ID but have an Au value

Of principal concern were the 369 samples with different database entries than those seen in the certificates. The samples with the greatest differences, and of the highest relevance, are summarized in Table 12-2. Further investigation revealed these discrepancies to be due to the use of reassay results in the database.

In addition to those 369 entries, 27,581 were seen to be truncated at 2 decimal places when the assay certificates were reported to three. This is considered to be of lesser significance as it represents an error of only a few parts per billion.

At the time the QP recommended that Aura complete a thorough review of the database, correcting all data entry errors. It is understood that Aura has completed this.

12.5.4 COLLAR SURVEY CHECKS

During the site visit the QP made collar coordinate checks of 11 diamond drill hole collars using a Garmin 60Csx hand-held GPS. Holes at the Paiol, Vira Saia and Cata Funda were checked.

For 10 of the 11 holes checked the agreement was reasonable given the level of accuracy of a hand-held GPS (typically about +/- 3 to 5 m, depending on the number of satellites available). (See Table 12-3)

One hole, FVSD-0002 showed significant errors in the x and y components (-158.8 m and 1,281.62 m, highlighted in yellow on Table 12-3). It was noted, though, that this was the only hole whose concrete monument had a weathered and rusty ID plaque. A review of the database showed the presence of a drill hole, FVSM-0001 which closely matched the location picked up. It is considered likely that this was a transcription error of a hole number from a rusty plaque. However, the QP recommended that Aura complete a more thorough survey check of the drill hole collars used in the resource estimate, prior to the completion of a full feasibility study.

This work is understood to be on hold pending resolution of the Covid 19 pandemic.

12.5.5 USE OF AUGER AND REVERSE CIRCULATION DRILL HOLE DATA

12.5.5.1 PAIOL LEACH PAD ASSAY VERIFICATION

Surface channel sampling by Aura, and drilling by Rio Novo, indicates that the material unloaded from the leach pad and dumped may have sufficient gold remaining that milling it may be justified. Some material remains on the pad.

The drilling of the dump completed by Rio Novo is of two types, reverse circulation (RC) and auger drilling. RC drilling uses a double tube drill string which isolates the returning rock chips inside of the tube. It therefore is not strongly subject to smearing.

Auger drilling uses a simple auger device to return rock chips to the surface and those chips are exposed to the wall of the hole as they come up. The technique is therefore prone to smearing and contamination by already sampled material.

However, the leach pad dump is not very high and is covered by a thin layer of reddish soil (see Figure 12-3). Sampled channels are visible near the geologist holding the hammer. The mineralized layer is usually only a few meters thick.

Figure 12-3 Leach Pad Dump Profile in Trench Showing Thin Red Soil Cover

Figure 12-4 shows a 3D isometric view of the digital terrain model of the dump and the location and distribution of the RC and auger holes in it. Many of the holes went through the dump and into soil and/or rock beneath.

Table 12-2 Sample Entries with the Greatest Difference

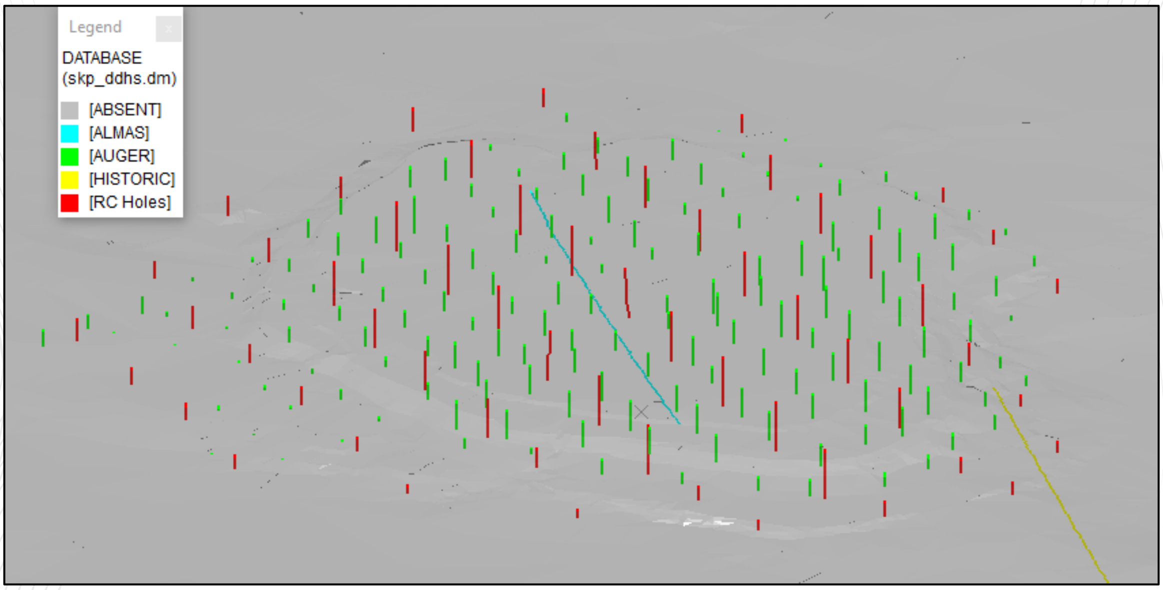
FROM-TO (m)					ASSAY CERTIFICATES					DIFFERENCE VALUE
HOLE	SAMPLE_ID	FROM	TO	Au ppm	SAMPLE MATCH	CERTIFICATE #	Au ppb	Au ppm	CHECK	
FPD-0068	A-36095	121	122	0.05	A-36095	CP1100004	3,500	3.5	DIFF	3.45
FPD-0080	A-25113	167	168	1.65	A-25113	GY1000564	1,857	1.857	DIFF	0.207
FPD-0080	A-25130	183	184	4.71	A-25130	GY1000564	4,922	4.922	DIFF	0.212
FPD-0080	A-25134	187	188	9.2	A-25134	GY1000564	9,731	9.731	DIFF	0.531
FAD-0060	A-42150	21	22	0.35	A-42150	GY1100041	2,443	2.443	DIFF	2.093
FVSD-0031	A-84854	50	51	0.01	A-84854	GY1100960	4,375	4.375	DIFF	4.365
FVSD-0031	A-84855	51	52	0.02	A-84855	GY1100960	4,250	4.25	DIFF	4.23
FVSD-0031	A-84856	52	52.5	0.02	A-84856	GY1100960	1,635	1.635	DIFF	1.615
FVSD-0031	A-84857	52.5	53.4	0.00999	A-84857	GY1100960	3,560	3.56	DIFF	3.55001
FVSD-0032	A-84950	31.5	32	0.00999	A-84950	GY1100960	1,840	1.84	DIFF	1.83001
FVSD-0032	A-84951	32	33	0.00999	A-84951	GY1100960	4,005	4.005	DIFF	3.99501
FVSD-0032	A-84952	33	34	0.00999	A-84952	GY1100960	3,635	3.635	DIFF	3.62501
FVSD-0032	A-84953	34	35.35	0.00999	A-84953	GY1100960	4,730	4.73	DIFF	4.72001
FVSD-0032	A-84954	35.35	36	0.00999	A-84954	GY1100960	2,555	2.555	DIFF	2.54501
FVSD-0032	A-84967	45.95	47	1.29	A-84967	GY1100960	2,520	2.52	DIFF	1.23
FVSD-0032	A-84968	47	48	0.91	A-84968	GY1100960	2,390	2.39	DIFF	1.48
FVSD-0032	A-84969	48	49	1.09	A-84969	GY1100960	2,485	2.485	DIFF	1.395
FVSD-0032	A-84970	49	50	0.33	A-84970	GY1100960	2,345	2.345	DIFF	2.015
FVSD-0032	A-84971	50	51	0.78	A-84971	GY1100960	2,460	2.46	DIFF	1.68
FVSD-0032	A-84972	51	51.5	0.84	A-84972	GY1100960	1,330	1.33	DIFF	0.49

FROM-TO (m)					ASSAY CERTIFICATES					DIFFERENCE VALUE
HOLE	SAMPLE_ID	FROM	TO	Au ppm	SAMPLE MATCH	CERTIFICATE #	Au ppb	Au ppm	CHECK	
FVSD-0032	A-84973	51.5	52	0.36	A-84973	GY1100960	1,070	1.07	DIFF	0.71
FVSD-0032	A-84974	52	53	0.88	A-84974	GY1100960	2,415	2.415	DIFF	1.535
FVSD-0032	A-84975	53	54	0.17	A-84975	GY1100960	2,315	2.315	DIFF	2.145
FVSD-0032	A-84976	54	55	0.32	A-84976	GY1100960	2,590	2.59	DIFF	2.27
FVSD-0032	A-84977	55	56.25	2.41	A-84977	GY1100960	2,905	2.905	DIFF	0.495
FVSD-0032	A-84979	57	58	0.34	A-84979	GY1100960	2,250	2.25	DIFF	1.91
FVSD-0032	A-84980	58	59	0.42	A-84980	GY1100960	2,430	2.43	DIFF	2.01
FAD-0072	A-20499	14	15	0.00499	A-20499	GY1000638	242	0.242	DIFF	0.23701
FAD-0072	A-20531	44	45	0.19	A-20531	GY1000638	421	0.421	DIFF	0.231
FPD-0059	A-22209	6	6.8	0.006	A-22209	GY1000640	260	0.26	DIFF	0.254
FPD-0059	A-22210	6.8	8	0.00499	A-22210	GY1000640	368	0.368	DIFF	0.36301
FPD-0059	A-22211	8	9	0.00499	A-22211	GY1000640	303	0.303	DIFF	0.29801
FPD-0059	A-22219	16	17	0.00499	A-22219	GY1000640	298	0.298	DIFF	0.29301
FPD-0059	A-22221	17	18	0.00499	A-22221	GY1000640	317	0.317	DIFF	0.31201
FPD-0059	A-22222	18	19	0.01	A-22222	GY1000640	372	0.372	DIFF	0.362
FPD-0059	A-22224	20	21	0.007	A-22224	GY1000640	305	0.305	DIFF	0.298
FCD-0001	A-36302	0	0.6	1.55	A-36302	GY1000688	1,837	1.837	DIFF	0.287
FCD-0005	A-36883	1.95	2.6	0.04	A-36883	GY1000692	337	0.337	DIFF	0.297
FCD-0005	A-36930	44.8	45.85	0.55	A-36930	GY1000692	937	0.937	DIFF	0.387
FCD-0005	A-36944	57.6	59	0.11	A-36944	GY1000692	327	0.327	DIFF	0.217
FCD-0005	A-36945	59	59.65	0.55	A-36945	GY1000692	2,378	2.378	DIFF	1.828

Table 12-3 Collar Coordinate Field Checks

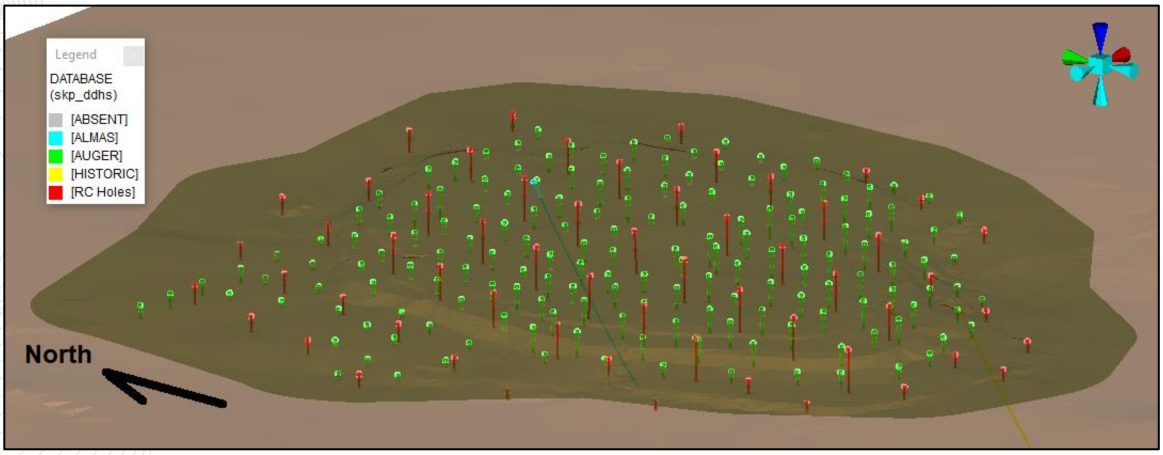
DATABASE RECORDS								SITE VISIT CHECKS			
Hole	TYPE	PROGRAM	TARGET	UTM_EAST	UTM_NORTH	ELEVATION	DEPTH	FIELD X	FIELD Y	DELTA X	DELTA Y
FAD-0085	Diamond	Infill	Arroz	264677.61	8719252.67	447.53	62.25	264673	8719254	-4.61	1.33
FAD-0114	Diamond	Extension	Arroz	264465.21	8719428.30	442.98	101.25	264459	8719429	-6.21	0.69
FPD-0019	Diamond	Infill	Paiol	265265.90	8706187.35	387.07	100.0	265262	8706188	-3.9	0.65
FPD-0021	Diamond	Infill	Paiol	265320.65	8706178.85	386.02	65.65	265317	8706182	-3.65	3.15
FPD-0161	Diamond	Infill	Paiol	265336.80	8706155.24	385.42	100.1	265330	8706156	-6.8	0.76
FPD-0162	Diamond	Infill	Paiol	265293.48	8706206.87	387.53	94.55	265292	8706207	-1.48	0.13
FPD-0165	Diamond	Extension	Paiol	265364.79	8706193.14	385.20	85.75	265358	8706193	-6.79	-0.14
FPD-0169	Diamond	Extension	Paiol	265340.56	8706197.15	385.88	90.0	265335	8706197	-5.56	-0.15
FPD-0174	Diamond	Extension	Paiol	265372.17	8706169.51	384.28	65.6	265364	8706167	-8.17	-2.51
FVSD-0152	Diamond	Infill	Vira Saia	264729.66	8710668.87	407.28	180.35	264723	8710667	-6.66	-1.86
FVSD-0002	Diamond	Scout	Vira Saia	264948.80	8709452.38	382.95	94.9	264790	8710734	-158.8	1,281.62
FVSM-0001	Metal	Metal	Vira Saia	264789.86	8710728.31	404.24	110.65	264790	8710734	0.14	5.68

Figure 12-4 3D Isometric View of Drill Holes in and Around the Leach Pad Dump



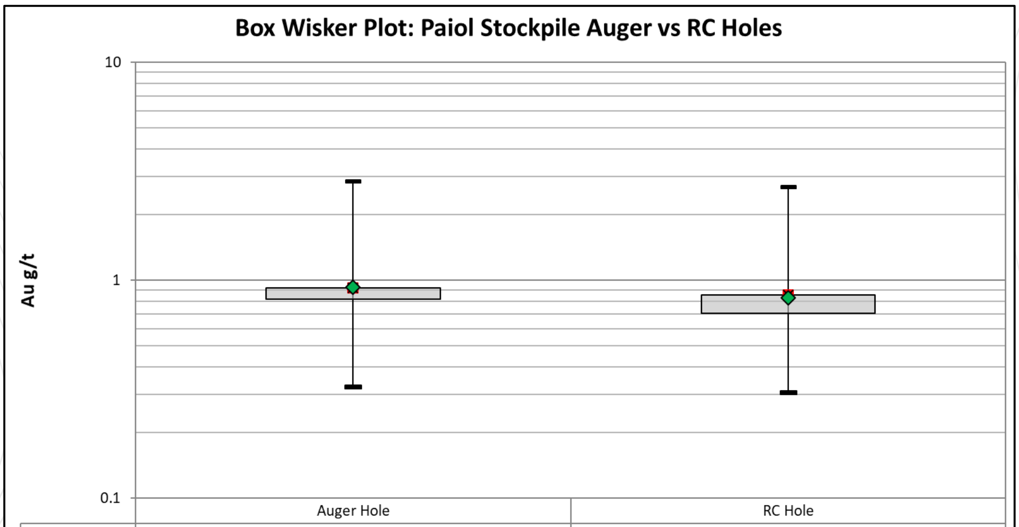
The QP chose to interpolate the bottom contact of the leach pad ore dump using the assays from the holes. There was an obvious cut-off in assays as each hole left the dump and entered the underlying material (see green "sheet" under the holes in Figure 12-5).

Figure 12-5 Interpreted Bottom Contact of Leach Pad Dump



Since all of the holes entered the mineralized pad material shortly after the collar, most of them are short. The pad material would therefore be well mixed, and the pad would have little to no waste within it. It was decided to consider whether there was a serious bias problem between the two data sets. The data above the interpreted bottom contact were sorted by hole type and examined separately using box and whisker plots, histograms and log probability plots (see Figure 12-6 to Figure 12-11).

Figure 12-6 Box and Whisker Plot Comparing Auger and RC Hole Assays



	Auger Hole	RC Hole
Mean	0.926	0.855
Max	2.833	2.68
Q3	1.013	0.886
Median	0.927	0.83
Min	0.325	0.307
Q1	0.818	0.706
CV	0.20	0.37
Nº of Samples	963	342

Figure 12-7 Histogram of Auger Hole Assays

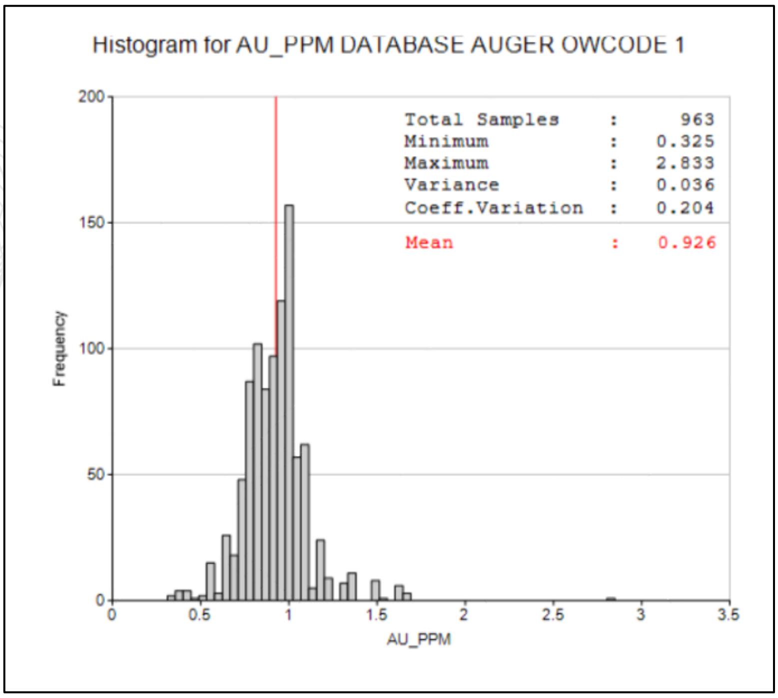


Figure 12-8 Histogram of RC Hole Assays

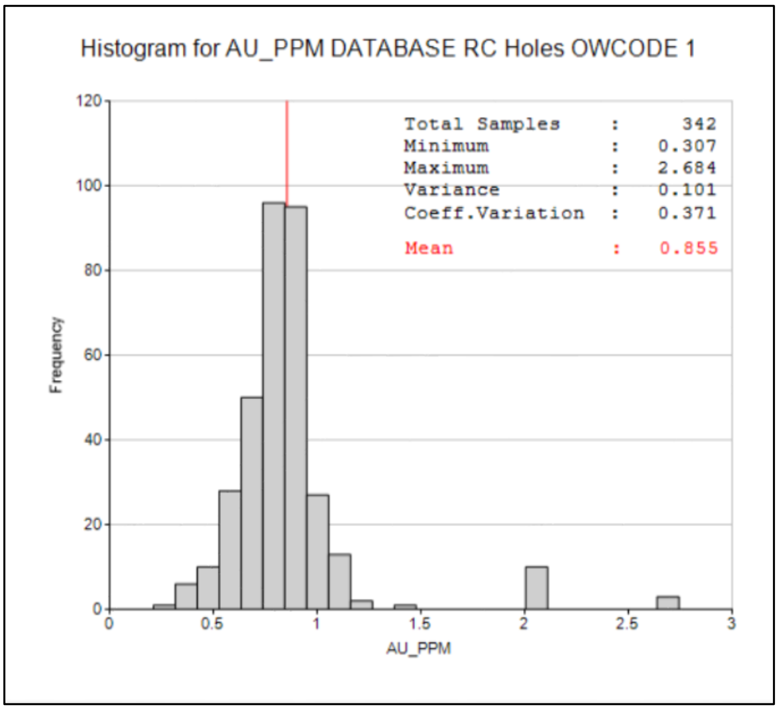


Figure 12-9 Log Probability Plot of All Assays

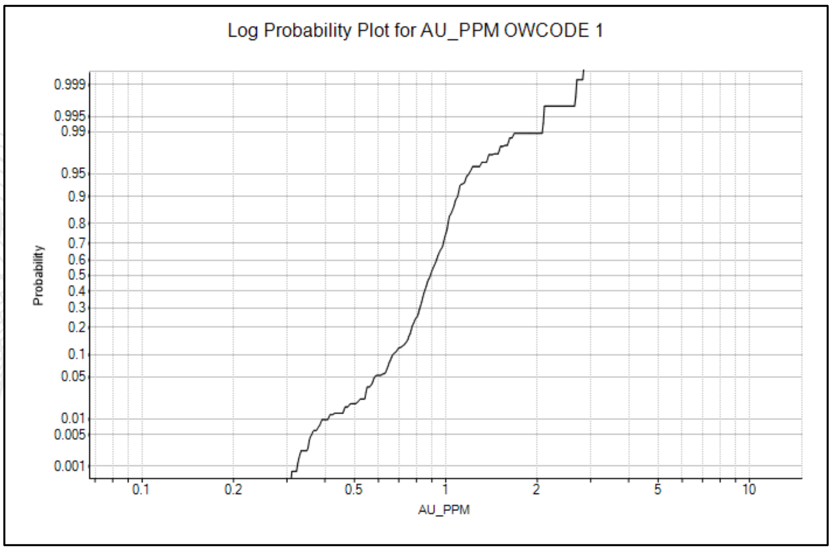


Figure 12-10 Log Probability Plot of Auger Assays

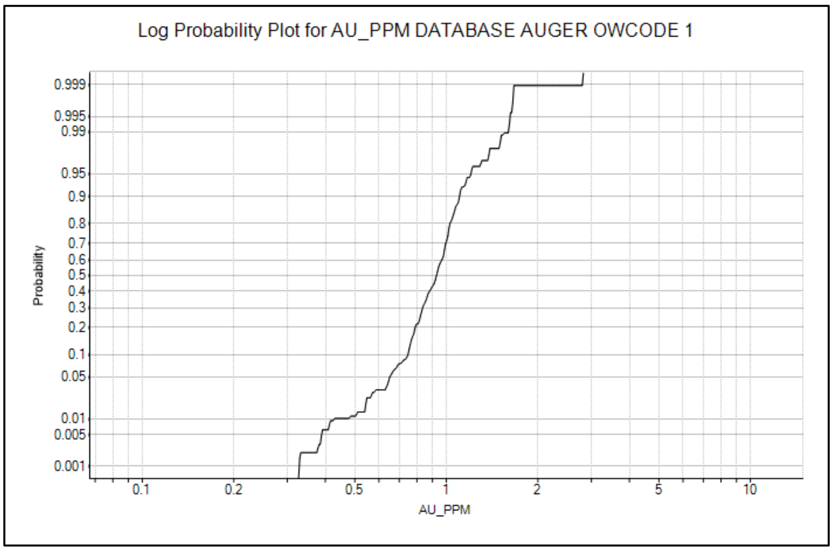
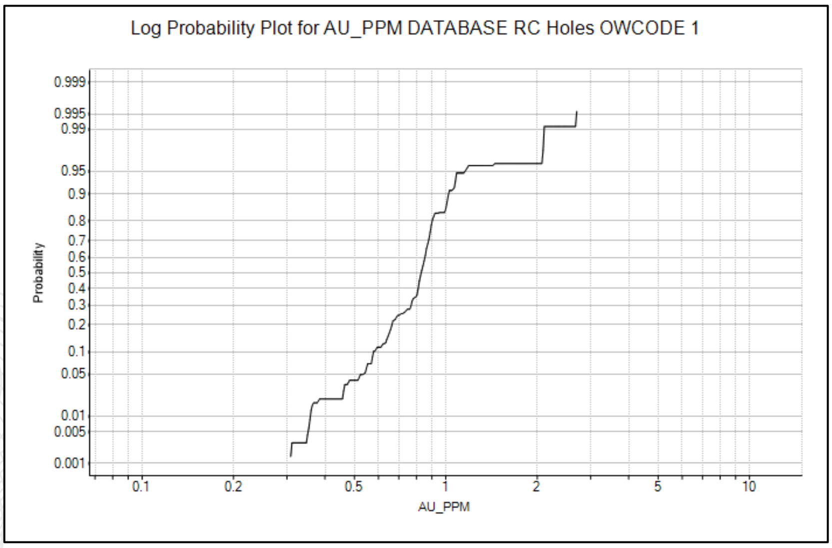


Figure 12-11 Log Probability Plot of RC Assays



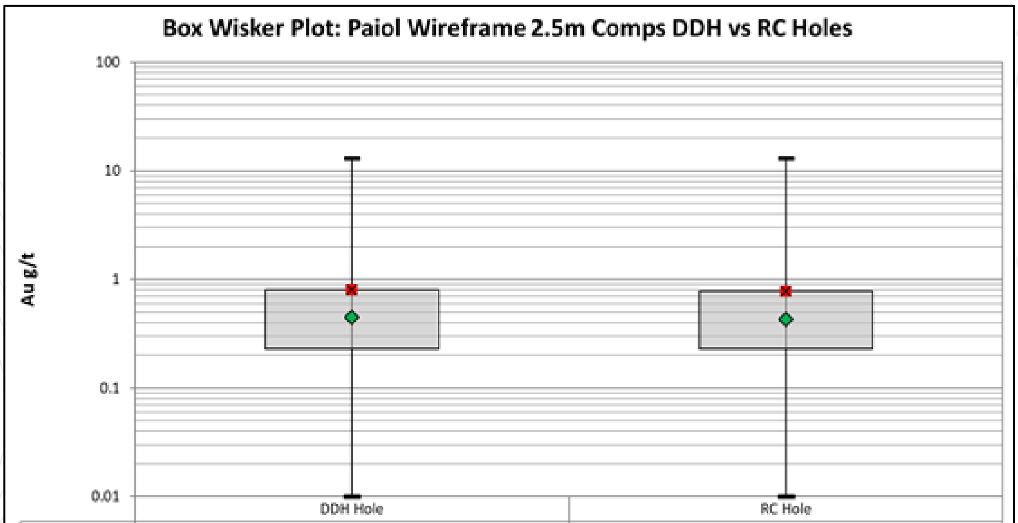
The statistical and graphical comparisons above show that, on average, the RC data are slightly lower grade than the auger data (mean of 0.855 g/t Au vs 0.926 g/t Au). On average the auger holes are 8.3% higher. There is a slight bias but not as large as was feared it might be. From the histograms and log probability plots the likely top-cut would be similar for both data sets, about 1.3 g/t.

The QP was of the opinion that it is acceptable to use the auger data for leach pad dump resource estimation, pending confirmation from the test pit channel samples that Aura has taken while on the site visit.

12.5.5.2 PAIOL REVERSE CIRCULATION VERSUS DIAMOND DRILL HOLE VERIFICATION

For the upper portion of Paiol deposit, where most of the RC drilling is located, a statistical comparison was made to check for bias between the RC and diamond drilling (DDH) methods and sampling. For this analysis, diamond drill holes in locations where RC drilling was absent were removed for better spatial comparison of the data. Figure 12-12 shows box and whisker plots for the two data sets as well as the basic statistics.

Figure 12-12 Paiol Wireframe 2.5 m Composites, DDH vs. RC Holes



	DDH Hole	RC Hole
Mean	0.801	0.784
Max	13.000	13.00
Q3	0.843	0.810
Median	0.450	0.43
Min	0.010	0.010
Q1	0.230	0.230
CV	1.55	1.57
Nº of Samples	1,648	2,516

It can be seen that the data distribution and statistics are very similar, with the RC data having a slightly lower mean grade. This corroborates Aura's observation that there was little difference between the resource models with, or without, RC drill data. The use of the RC data at this location is considered to be justified. However, it is likely to have little effect on the mineral resource presented in the report as the saprolite and weathered rock intersected by the RC drilling has already been mined.

12.6 CONCLUSIONS

The presence of gold at Paiol is supported by Vale's previous mining experience from 1996 to 2001 when approximately 86,000 ounces were produced. As well there are small scale open pit workings at the other two deposits Cata Funda (previously called Arroz) and Vira Saia.

The QP is satisfied that the exploration, sampling, security and QA/QC procedures employed at the Almas 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 AND HISTORICAL BACKGROUND

This section of the report contains the metallurgical test results for the Almas Gold Project conducted during two testwork campaigns. All the previous testwork campaigns conducted on this project were reported and summarized in a document entitled "Updated Feasibility Study Technical Report for the Almas Gold Project Almas Municipality, Tocantins State, Brazil", dated August 9, 2016 issued by Runge Pincock Minarco.

The initial testwork program reported in this document was conducted at the SGS Geosol laboratory in Belo Horizonte, Brazil. The testwork program was conducted during September - December 2018. The mineralogical study on the project deposit samples was conducted at the SGS Lakefield faculty in Lakefield, Canada in 2018. In addition, a senior metallurgist from SGS Lakefield Gold metallurgy group has visited the SGS Geosol laboratory from September 25 to October 3, 2018. During this time period he observed the initial tests and reviewed several SGS standard operating procedures to be applied in the test program.

The SGS testwork reports titles were as follows:

- SGS Minerals Services, Lakefield – Trip Report Summary, SGS Geosol (on-site) – Project 17029- 01A, October 17, 2018
- SGS Geosol - Metallurgical study report- Project 3965- 1801- Final Report- Gravity Separation, Flotation and Leaching Testwork on Gold Ore Samples from the Almas Deposit, September 20, 2019
- SGS Minerals Services, Lakefield - Mineralogy study report - Project 17013-01, MI5030-OCT18 – Final Report – An Investigation by High Definition Mineralogy into the Mineralogical Characteristics of Nine Composite Samples from the Almas and Matupa Gold Projects, Brazil, February 7, 2019

The main objective of this testwork program was to evaluate the potential process flowsheets for a subsequent trade-off study by an engineering company (Ausenco Engineering). The process flowsheets evaluated were as follows:

- Flowsheet 1 - Gravity Separation followed by Flotation and Concentrate Cyanide Leaching. The main emphasis was placed on the development of this flowsheet, specifically evaluating flotation.
- Flowsheet 2 - Gravity Separation followed by Cyanidation- preliminary testing
- Amenability to heap leaching has also been briefly evaluated

The second testwork program reported in this document was conducted by the metallurgical laboratory "TESTWORK Desenvolvimento de Processo Ltda" (TESTWORK Process Development") in Brazil and the chemical analyses were conducted at the SGS Geosol laboratory. Additional settling and rheology tests were conducted by FLSmidth in Brazil and the gravity separation circuit was evaluated and modelled by FLSmidth in Canada. Also, additional breakage testwork was conducted at the Metso:Outotec laboratory in Sorocaba, Brazil. The testwork programs were conducted during March- November 2020.

The following results and reports were issued during this program:

- TESTWORK Process Development Laboratory - testwork results and test details
- SGS Geosol - certificates of chemical analysis
- Coteprom Mineral Consultancy and Advisory Services Ltda – testwork summary tables
- FLSmidth - Solid/Liquid Separation Report – Report Number RTE522/20, Aura Minerals Almas Project, Settling and Rheology of Ore Samples, Brazil, July 8, 2020
- FLSmidth – Gravity Separation Report – Report Number 200903-CA- 1600, Gravity Audit Modelling Report, Aura Minerals, Almas Project, September 3, 2020
- MinPro Solutions - Comminution Process Simulation – Report Aura 01-20, Rev 0 03, September 2020
- Metso:Outotec – Comminution tests report, October 26, 2020

The main objective of the second campaign was to confirm the gravity separation – cyanidation flowsheet configuration and to optimize the process variables for a feasibility level study. This program included further testing of the gravity separation circuit, confirmation of the cyanide leach parameters, cyanide destruction and solid-liquid separation testing.

The additional breakage tests conducted at Metso:Outotec included three SMC Test®, one on each major ore type from the Almas deposit. The results of these tests were published by Metso:Outotec, as they are the only licensed laboratories that can conduct the SMC Test® in Brazil.

The following sub-sections contain the results for both campaigns in chronological order.

13.2 SAMPLE PREPARATION AND HEAD ASSAYS

For the first campaign, six ore type samples from the Almas Gold Project deposits were submitted for testing as individual holes core samples. The ore types for each deposit were identified as follows:

Paiol Ore Deposit (these composites contained saprolite and two lithologies samples)

- Paiol Saprolite: submitted weight - 114 kg. This is oxide material, representing approximately 5-10% of the deposit, which is similar to the other two deposits of saprolite ore
- Paiol SDCX (sericite-chlorite-ankerite schist): submitted weight - 58 kg. This is sulphide material representing approximately 40-45% of the deposit
- Paiol SDQS (sericite-ankerite-quartz schist): submitted weight - 53 kg. This is sulphide material representing approximately 40-45% of the deposit

Vira Saia Ore Deposit (these composites contained two lithologies samples)

- Vira Saia QSX (quartz-serecite shist): submitted weight - 90 kg. Identified as the sulphide material representing approximately 20-25% of the deposit
- Vira Saia GDM (mylonitic granodiorite): submitted weight - 89 kg. Identified as the sulphide material representing approximately 70-75% of the deposit

Heap Leach Pad Material (identified as Trench Composite)

- Trench Composite*:* submitted weight - 61 kg. This is oxidized material from the old heap leach operation by the VALE mine

For the second campaign, a composite representing the first three years of operation was selected by the Aura technical team. The 3-Year composite identified as “Blend 3-Y” contained a blend of Saprolite and Fresh Rock. The average blend composition representing a 3- year period was as follows:

- Paiol: submitted weight 158 kg, representing approximately 75.4.% of the deposit period
- Vira Saia and Vira Saia Saprolite: submitted weight 17.8 kg and 23.2 kg, representing 8.5% and 11.1%, respectively
- Cata Funda: submitted weight 10.6 kg representing 5.1% of the deposit period

The available sample weights and the composite distribution by typology for the first three years of operation are shown in Table 13-1.

Table 13-1 Composite Weights

SAMPLE	WEIGHT, kg	AVERAGE % WEIGHT DISTRIBUTION			
		YEAR 1	YEA 2	YEA 3	AVERAGE*
INDIVIDUAL COMPOSITES					-
PAIOL SAPROLITE	114				5-10
PAIOL SDCX	58				40-45
PAIOL SDQX	53				40-45
VIRA SAIA QSX	90				20-25
VIRA SAIA GDM	89				70-75
TRENCH	61				-
BLEND 3-YEAR COMPOSITE	210	-	-	-	-
PAIOL	158	100	58	57	73
VIRA SAIA	41	0	42	23	20
CATA FUNDA	11	0	0	20	6

*Average composition for each deposit

For the additional SMC Tests®, three ore type samples from the Almas Gold Project deposits were submitted for testing as blended holes core samples. The ore types for each deposit were identified as follows:

- Vira Saia: GDM-QSX-VS001
- Paiol: SDQX-ADQX-PA-003 and SDQX-SCDX-PA01
- Cata Funda: SCDX CAT-001

The sample preparation and the sample handling protocols for low grade gold ores were followed during the testwork programs, to ensure that the QA/QC guidelines and the standard operating procedures were executed throughout the project.

In addition to the sample preparation protocols, the low detection fire assay methodology (especially for tailings and residue analysis) has been reviewed with the testing laboratories. SGS Lakefield has provided an explanatory note and a "precision curve" graph, indicating that if the sample concentration is slightly above or near the detection limit of 0.01-0.02 g/t Au, the analysis will have a significantly large uncertainty at that level. Therefore, this level of uncertainty should be taken into consideration for results evaluation and comparison.

The individual core samples of each ore type were combined and crushed to minus 6 mesh. Each composite was well blended and split into representative test charges. Representative head samples were removed from each composite for assays. Each sample was analyzed for gold by direct fire assay (using nine individual subsamples from each ore type) and by 'screened metallic' protocol, as shown in Table 13-2 and Table 13-3. Calculated head grades from metallurgical balances of gravity and leach tests are also shown for cross reference with the assayed head grades.

Table 13-2 Comparative Gold Head Assays

SAMPLE	WEIGHT, kg	Au g/t		
		AVE FROM NINE ALIQUOTS	SCREENED METALLIC ASSAY	AVE CALC HEAD FROM GRAV SEPARATION TESTS
AIOL SAPROLITE	114	0.65	0.65	0.65
PAIOL SDCX	58	0.89	1.01	0.98
PAIOL SDQX	53	1.20	1.42	1.29
VIRA SAIA QSX	90	1.46	1.59	1.53
VIRA SAIA GDM	89	0.89	0.94	0.91
TRENCH	61	0.89	0.98	0.94
BLEND 3-YEAR	210	-	1.28^/1.34^^	1.86/1.31*

^triplicate screened metallic assay
^^ size fraction analysis head grade assay
* average calculated head grade from gravity tests/whole ore leach tests

Table 13-3 Individual Samples Gold Head Assays

SAMPLE	Au g/t ALIQUOT	Au g/t SUB-SAMPLE	Au g/t AVERAGE	SAMPLE	Au g/t ALIQUOT	Au g/t SUB-SAMPLE	Au g/t AVERAGE
PAIOL SAPROLITE	0.69 0.75 0.67	0.71	0.65	VIRA SAIA QSX	1.53 1.32 1.41	1.42	1.46
	0.59 0.61 0.65	0.62			1.35 1.21 1.29	1.28	
	0.57 0.68 0.64	0.63			1.68 1.53 1.83	1.68	
PAIOL SDCX	0.86 0.88 0.86	0.86	0.89	VIRA SAIA GDM	0.86 0.80 0.82	0.83	0.89
	0.83 1.06 1.01	0.96			0.86 0.88 0.85	0.86	
	0.90 0.84 0.82	0.85			1.12 0.91 0.88	0.97	
PAIOL SDQX	1.15 1.13 1.11	1.13	1.20	TRENCH	0.83 0.85 1.01	0.90	0.89
	1.27 1.19 1.40	1.28			0.77 0.88 0.86	0.83	
	1.24 1.14 1.15	1.18			0.89 0.99 0.97	0.95	

The “Blend 3-Year” composite was prepared following the same standard operating procedures, as applied to the individual composites. Representative test charges were riffled out for testing and the head samples were removed for analysis. The head samples were submitted for the gold assays by the screened metallic method, conducted at 150 mesh and by size fraction analysis, shown in Table 13-4 and Table 13-5, respectively, and illustrated in Figure 13-1.

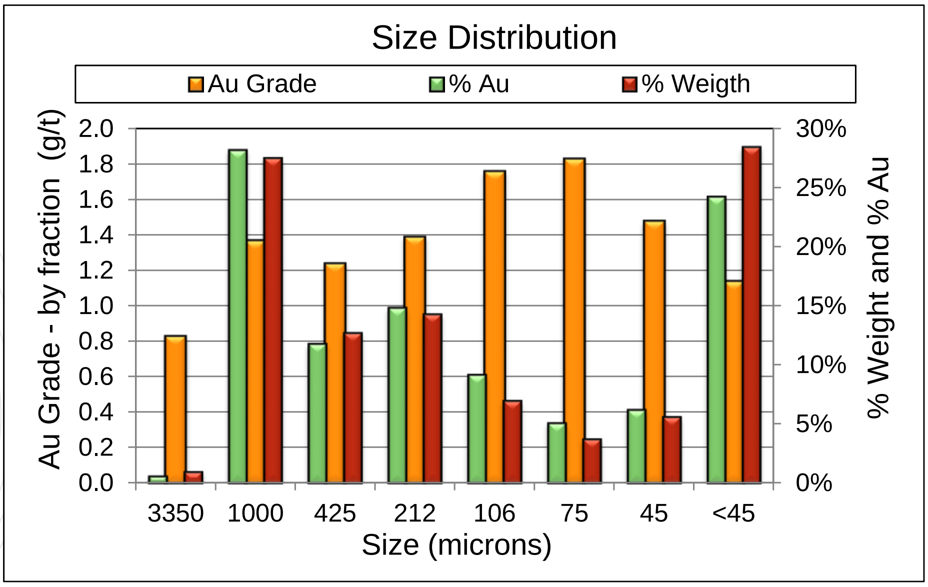
Table 13-4 Blend 3-Year Composite Screened Metallic Assays

FRACTION, MESH	WEIGHT, %	Au, g/t	% Au DISTR'N
+150 mesh	2.88	2.43	5.2
-150 mesh	97.12	1.32	94.8
Head (calc)	100.00	**1.35**	100.0
+150 mesh	3.77	1.69	5.0
-150 mesh	96.23	1.26	95.0
Head (calc)	100.00	**1.28**	100.0
+150 mesh	4.41	2.14	7.7
-150 mesh	95.59	1.18	92.3
Head (calc)	100.00	**1.22**	100.0
Head (average)		**1.28**	

Table 13-5 Blend 3-Year Composite Size Fraction Analysis

SIZE FRACTION, MESH	SIZE FRACTION, µm	Au g/t	% Au RETAINED	% Au RETAINED CUM.	% Au PASSING CUM.
6	3350	0.83	0.6	0.6	99.4
16	1000	1.37	28.2	28.8	71.3
35	425	1.24	11.8	4.5	59.5
65	212	1.39	14.8	55.4	44.6
150	106	1.76	9.2	64.5	35.5
200	75	1.83	5.1	69.6	30.4
325	45	1.48	6.2	75.8	24.2
<325	<45	1.14	24.2	100.0	0.0
Head (calc)		**1.34**	100.0		

Figure 13-1 Blend 3-Year Composite Size Distribution Analysis



The Blend 3-Y comp screened metallic triplicate head assays showed the calculated head grade of 1.28 g/t Au with 5-7% of the gold reporting into the coarse fraction. The size fraction analyses showed the calculated head grade of 1.34 g/t Au. Calculated head grades from metallurgical balances of gravity and leach tests are also shown in Table 13-1 for cross reference with the assayed head grades.

The samples were also submitted for SG determination, sulphur, carbon speciation analysis, and the multi-element ICP scan, as presented in Table 13-6 for the individual and the 3-Year composites. The sulphur grades were approximately 0.5% for the Paiol, Trench, and the Blend 3-Y composite samples. The other samples contained 0.02-0.04% S. The graphitic and organic carbon concentration in all the samples was <0.05-0.1%, indicating that there is no preg-robbing potential, unless the clay minerals present in the ore exhibit such capacity. The silver analyses were included in the ICP scan and reported as <3 g/t Ag for all the samples. The copper and zinc concentrations were low for all the samples tested. The copper speciation conducted in the Blend 3-Y Comp showed very low concentration of cyanide soluble copper of <0.002%. The mercury concentration was also low (0.02-0.07 ppm).

In addition, the whole rock analysis was conducted by SGS Lakefield as a part of the mineralogy program, as shown in Table 13-7.

Table 13-6 Head Assays – Sulphur, Carbon, ICP Scan, and Hg

SAMPLE	SG g/cm³	S %	C %	C org %	C g %	Ag g/t	Al %	As g/t	Ba g/t	Be g/t	Bi g/t
PAIOL SAPROLITE	2.83	0.04	0.13	0.08	<0.05	<3	7.23	11	222	<3	<20
PAIOL SDCX	2.85	0.43	2.45	<0.05	0.09	<3	5.87	17	221	<3	<20
PAIOL SDQX	2.89	0.48	3.37	<0.05	0.10	<3	5.39	37	109	<3	<20
VIRA SAIA QSX	2.70	0.04	0.26	<0.05	<0.05	<3	7.22	<10	717	<3	<20
VIRA SAIA GDM	2.68	0.02	0.47	<0.05	<0.05	<3	7.34	<10	770	<3	<20
TRENCH	2.81	0.49	2.36	<0.05	<0.05	<3	5.33	27	134	<3	<20
BLEND 3-YEAR	-	0.48	2.49	<0.05	-	<3	5.49	63	177	<3	<20

SAMPLE	Ca %	Cd g/t	Co g/t	Cr g/t	Cu g/t	Fe %	K %	La g/t	Li g/t	Mg %	Mn %
PAIOL SAPROLITE	0.11	<3	82	58	91	10.0	1.13	<20	29	0.7	0.14
PAIOL SDCX	5.70	<3	34	11	47	10.0	0.89	<20	19	1.9	0.15
PAIOL SDQX	5.81	<3	39	35	64	7.8	1.46	<20	14	2.4	0.14
VIRA SAIA QSX	0.90	<3	<8	10	12	1.6	3.35	<20	10	0.3	0.02
VIRA SAIA GDM	1.68	<3	<8	4	7	1.8	2.79	22	11	0.4	0.03
TRENCH	4.44	<3	31	35	49	8.0	0.95	<20	13	1.7	0.12
3-YEAR COMP	4.49	<3	33	30	46	7.8	1.04	<20	<3	1.7	0.12

SAMPLE	Mo g/t	Na %	Ni g/t	P %	Pb g/t	S %	Sb g/t	Sc g/t	Se g/t	Sn g/t	Sr g/t
PAIOL SAPROLITE	<3	0.27	64	0.03	<8	<0,01	<10	39	<20	<20	15
PAIOL SDCX	<3	1.55	18	0.08	<8	0.41	<10	28	<20	<20	112
PAIOL SDQX	<3	1.18	51	0.03	<8	0.44	<10	30	<20	<20	92
VIRA SAIA QSX	<3	1.15	5	0.02	13	0.03	<10	<5	<20	<20	118
VIRA SAIA GDM	<3	1.76	<3	0.03	<8	0.03	<10	<5	<20	<20	238
TRENCH	<3	1.4	30	0.05	<8	0.46	<10	26	<20	<20	86
3-YEAR COMP	<3	1.53	30	0.06	<8	0.53	<10	25	<20	<20	120

SAMPLE	Th g/t	Ti %	Tl g/t	U g/t	V g/t	W g/t	Y g/t	Zn g/t	Zr g/t	Hg g/t
PAIOL SAPROLITE	<20	0.34	<20	<20	267	28	8	126	49	0.07
PAIOL SDCX	<20	0.92	<20	<20	236	48	5	146	124	0.07
PAIOL SDQX	<20	0.24	<20	<20	210	37	<3	91	33	0.05
VIRA SAIA QSX	<20	0.11	<20	<20	23	<20	<3	61	37	0.02
VIRA SAIA GDM	<20	0.14	<20	<20	24	<20	<3	60	40	0.02
TRENCH	<20	0.43	<20	<20	184	31	4	111	66	0.02
3-YEAR COMP	<20	0.57	<20	<20	183	33	6	87	87	<0.05

SAMPLE	Cu_S %	Cu_{CN} %	Cu_{Res} %	S* %	S=* %	SO_4 ** %
PAIOL SAPROLITE	-	-	-	< 0.01	< 0.05	-
PAIOL SDCX	-	-	-	0.38	0.33	-
PAIOL SDQX	-	-	-	0.57	0.44	-
VIRA SAIA QSX	-	-	-	0.04	<0.05	-
VIRA SAIA GDM	-	-	-	0.02	< 0.05	-
TRENCH	-	-	-	0.4	0.35	-
3-YEAR COMP	<0.002	<0.002	0.004	-	-	0.07

* SGS Lakefield Assays

** SGS Geosol Assays

Table 13-7 Head Assays – Whole Rock Analysis (SGS Lakefield)

SAMPLE	SiO_2 %	Al_2O_3 %	Fe_2O_3 %	MgO %	CaO %	Na_2O %	K_2O %
PAIOL SAPROLITE	57.9	15.0	14.5	1.15	0.17	0.43	1.44
PAIOL SDCX	47.1	11.0	13.5	3.08	8.41	2.08	1.04
PAIOL SDQX	48.3	9.9	10.6	3.92	8.36	1.6	1.64
VIRA SAIA QSX	77.3	11.7	1.52	0.41	1.16	1.6	3.36
VIRA SAIA GDM	69.4	14.5	2.17	0.61	2.48	2.54	3.74
TRENCH	52.4	10.6	10.9	2.94	6.69	2.15	1.32
SAMPLE	**TiO_2 %**	**P_2O_5 %**	**MnO %**	**Cr_2O_3 %**	**V_2O_5 %**	**LOI %**	**Sum %**
PAIOL SAPROLITE	1.23	0.08	0.21	0.02	0.06	7.54	99.7
PAIOL SDCX	1.72	0.21	0.21	< 0.01	0.04	10.5	98.9
PAIOL SDQX	0.78	0.07	0.18	0.02	0.04	13.3	98.7
VIRA SAIA QSX	0.18	0.05	0.02	0.02	< 0.01	2.13	99.5
VIRA SAIA GDM	0.24	0.07	0.04	< 0.01	< 0.01	3.37	99.2
TRENCH	1.12	0.11	0.17	0.02	0.03	10.3	98.8

13.3 MINERALOGY

Each composite sample from the first campaign was submitted to the Advanced Mineralogy Facility at the SGS Lakefield site for a mineralogical examination by QEMSCAN and XRD. The results were presented in the SGS report identified as 'Project 17013-01, MI5030-OCT18 – Final Report: An Investigation by High-Definition Mineralogy into the Mineralogical Characteristics of Nine Composite Samples from the Almas and Matupa Gold Projects, Brazil'.

The objectives of this investigation were to determine the overall mineral assemblage of each sample and liberation of minerals of interest such as sulphides. A summary of the results obtained from the SGS report is presented below.

Results from the XRD Analysis

The bulk and clay XRD analysis indicated that the three samples consisted of quartz, muscovite, albite, ankerite, chlorite (chamosite and clinochlore), pyrite, phlogopite, goethite, microcline (K-feldspar), and trace amounts of other minerals (<2%).

The clay minerals included kaolinite, nontronite, illite, and illite-montmorillonite. The total clay content ranged from 5% in the Trench to 9% in the Oxide and 33% in the Paiol Saprolite.

<u>Results from QEMSCAN Analysis- Modal Mineralogy</u>

All minerals varied widely within the ore types, as shown in Table 13-8. However, the Paiol Saprolite was characterized by elevated amounts of clays (~ 18%) compared to the other samples (1%-9%) and goethite compared to approximately <1% for the rest of the samples. Elevated ankerite was shown in Paiol SDQX (32%), Trench (21%), Paiol SDCX (14%).

Table 13-8 also illustrates a summary of the mineral mass in each sample as shown in the SGS report. The highlighted minerals can affect the ore processing stages in different ways (in crushing/grinding, flotation, leaching, and material handling).

Table 13-8 Head Assays – Mineral Mass in Each Sample (SGS Lakefield Report)

SAMPLE	PAIOL SAPROLITE	PAIOL SDCX	PAIOL SDQX	VIRA SAIA QSX	VIRA SAIA GDM	TRENCH
Pyrite	0.05	3.3	0.71	0.20	0.03	1.05
Other Sulphides	0.01	0.01	0.03	0.01	0.00	0.01
Quartz/Feldspars	48.7	46.3	47.1	62.3	66.7	56.3
Sericite/Muscovite	16.7	7.3	9.7	32.3	26.6	7.2
Clays	17.9	9.4	4.1	1.7	1.7	6.4
Chl/Biot	5.0	6.2	4.1	0.3	0.2	2.9
Fe Ox/Oxy	10.9	4.6	1.0	0.3	0.4	2.3
Carbonates	0.70	21.8	32.8	2.02	3.6	23.3
Other	0.10	0.98	0.43	0.80	0.70	0.69

Pyrite Liberation

Pyrite was the predominant sulphide mineral ranging from traces to 3% in the samples examined. Free and liberated pyrite accounted for 70% Paiol SDCX to 90%-100% in the rest of the samples. Most of the middling occurred as complex particles (ternary and quaternary composite particles) in the Paiol SDCX.

Gold Deportment

The mineralogical gold deportment study has not been conducted at this stage. It has been noted that gold can be associated with a number of minerals (quartz, sulphides). Gold in some of the oxidized samples can be associated with iron oxides and oxyhydroxides. Both the Fe-oxyhydroxides and pyrite can also contain chemically bound (submicroscopic) gold.

13.4 COMMINUTION TESTING

Each sample from the first testwork campaign conducted at SGS Geosol was submitted for Bond Ball Work index determination, as shown in Table 13-9. The results suggested that the Saprolite sample showed a very low BWI of 4.4 kWh/ton. The other samples showed average Bond indices (8.5-11.9 kWh/ metric ton) for low grade sulphide and oxide ores. These results are in line with previous testwork conducted for the Almas Gold Project, which exhibited values between 6.8 and 11.2 kWh/ metric ton.

Table 13-9 Bond Ball Work Index Summary

SAMPLE	F_{80} microns	CONTROL SCREEN, microns	P_{80}, microns	Wi, kWh/metric ton
PAIOL SAPROLITE	397	106	57	4.4
PAIOL SDCX	2,053	106	75	10.1
PAIOL SDQX	1,975	106	75	9.7
VIRA SAIA QSX	2,066	106	76	11.2
VIRA SAIA GDM	1,978	106	76	11.9
TRENCH	1,180	106	70	8.5

The results of the SMC Tests® indicated that the Cata Funda ore is more competent than Vira Saia and Paiol ores, as shown by the Axb values in Table 13-10 (the lower the Axb value, the more competent is the ore). Previous Axb values for the project were obtained from tests following the MinPro SOLUTIONS methodology, which is different from that patented for the SMC Tests®, and it suggested that the Almas ores were more competent.

Table 13-10 SMC Test® Summary

SAMPLE	DWI kWh/m^3	A x b	SPECIFIC GRAVITY	ABRASION INDEX, ta
VIRA SAIA	4.13	66	2.73	0.63
PAIOL	4.74	60	2.86	0.54
CATA FUNDA	5.87	49	2.86	0.44

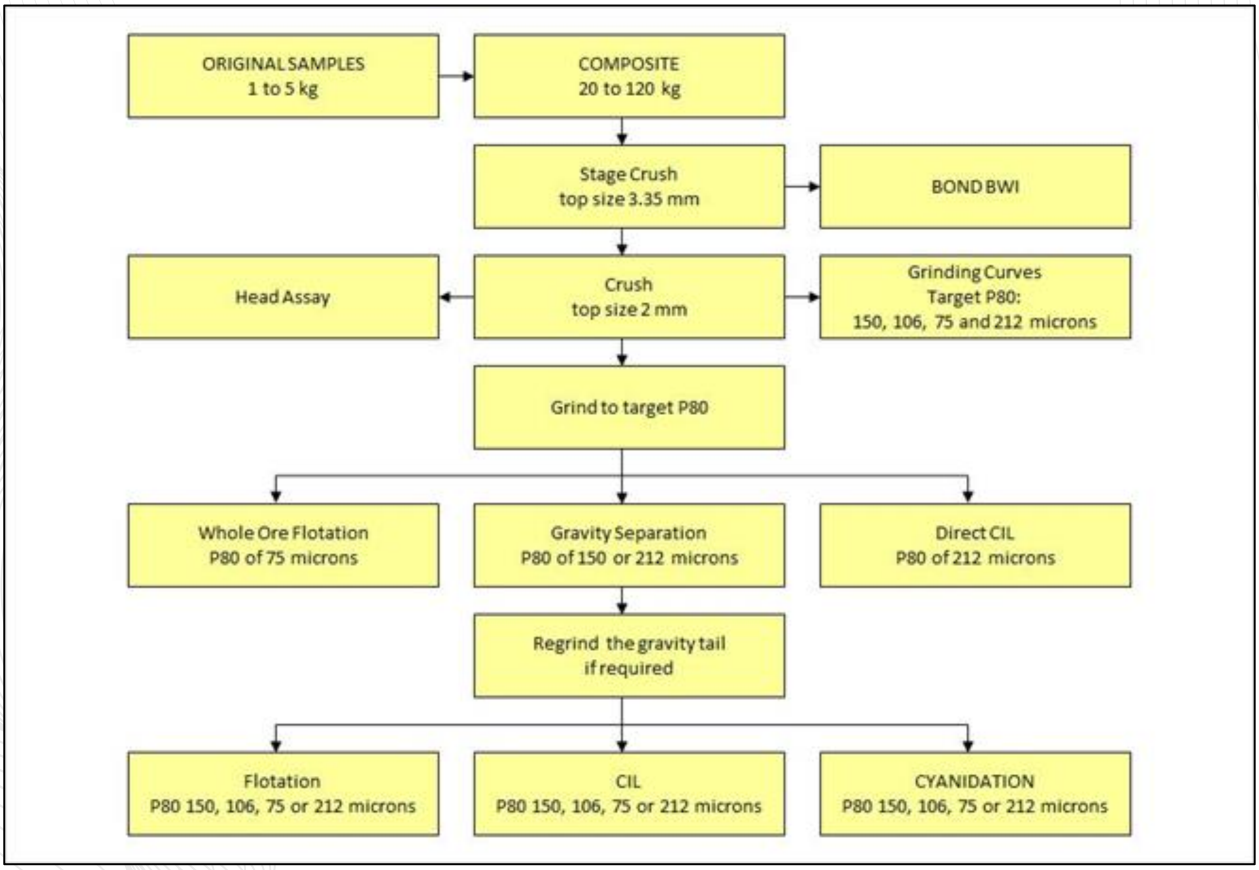
The recommended comminution process design parameters include:

- Axb = 60
- Bond crushing work index = 11.7 kWh/t
- Bond rod mill work index = 11.6 kWh/t
- Bond ball mill work index – 10.1 kWh/t
- Abrasion index = 0.069

13.5 INDIVIDUAL COMPOSITES TESTWORK PROGRAM

The testwork program for the individual composites is presented in the flowsheet below in Figure 13-2.

Figure 13-2 Testwork Program Flowsheet



13.5.1 INDIVIDUAL COMPOSITES FLOWSHEET 1: GRAVITY SEPARATION - FLOTATION

A series of exploratory grinding tests was conducted in a laboratory rod mill to establish the grinding time to reach the particle size K_{80} of 150, 106, and 75 microns. All the samples tested required fairly short grinding time in a laboratory rod mill.

Each ore type was subjected to gravity separation and flotation testing. Flotation concentrate cyanide leaching has not been conducted at this phase of testing.

13.5.1.1 GRAVITY SEPARATION

Ten kilograms of each ore type was ground to a K_{80} of 150 microns and subjected to a gravity separation test using a laboratory Knelson concentrator. The Knelson concentrate was further upgraded by hand panning. The test products were submitted for gold assays and the Pan tailings and the Knelson tailings were combined for subsequent testing. The results are presented in Table 13-11. The results indicated that 0.2-0.6% of the mass was recovered into the gravity concentrate with the grade ranging from 58 to 234 g/t Au. The arithmetic average calculated recovery for all of the samples tested (assuming equal weight ratios from each composite) was 25% with the concentrate grade of 109 g/t Au and the tailings grade of 0.7 g/t Au, as shown in Table 13-12. It should also be noted here that SGS Geosol calculated the metallurgical balances for the gravity circuits using the BILMAT software and the experimental results. This report shows the recoveries from the experimental results.

Additional standard GRG testing has been recommended to confirm the requirements and the design criteria for a gravity circuit.

Table 13-11 Individual Composites Gravity Separation Test Results

SAMPLE	PRODUCT	WEIGHT		ASSAYS	% DIST'N
		g	%	Au (g/t)	Au
PAIOL SAPROLITE	Hand Panning Concentrate	20.0	0.20	57.9	**17.9**
	Hand Panning Tailing	46.0	0.46	7.22	5.1
	Knelson Tailing	9,934	99.3	0.50	76.9
	Comb Hand & Knelson Tailing (calc)	9,980	99.8	0.53	***82.1***
	Head (calc)	10,000	100.0	0.65	100.0
	Head (direct)	10,000		0.65/0.65	
PAIOL SDCX	Hand Panning Concentrate	35.4	0.35	70.4	**25.5**
	Hand Panning Tailing	31.6	0.32	4.00	1.3
	Knelson Tailing	9,933	99.3	0.72	73.2
	Comb Hand & Knelson Tailing (calc)	9,965	99.6	0.73	***74.5***
	Head (calc)	10,000	100.0	0.98	100.0
	Head (direct)	10,000		0.89/1.01	
PAIOL SDQX	Hand Panning Concentrate	55.9	0.56	77.6	**33.6**
	Hand Panning Tailing	20.1	0.20	16.29	2.5
	Knelson Tailing	9,924	99.2	0.83	63.8
	Comb Hand & Knelson Tailing (calc)	9,944	99.4	0.86	***66.4***
	Head (calc)	10,000	100.0	1.29	100.0
	Head (direct)	10,000		1.20/1.42	
VIRA SAIA QSX	Hand Panning Concentrate	14.0	0.14	129	**11.8**
	Hand Panning Tailing	35.0	0.35	154	35.3
	Knelson Tailing	9,951	99.5	0.81	52.8
	Comb Hand & Knelson Tailing (calc)	9,986	99.9	1.35	***88.2***
	Head (calc)	10,000	100.0	1.53	100.0
	Head (direct)	10,000		1.46/1.59	
VIRA SAIA GDM	Hand Panning Concentrate	11.0	0.11	234	**28.3**
	Hand Panning Tailing	37.0	0.37	33.6	13.7
	Knelson Tailing	9,952	99.5	0.53	58.0
	Comb Hand & Knelson Tailing (calc)	9,989	99.9	0.65	***71.7***
	Head (calc)	10,000	100.0	0.91	100.0
	Head (direct)	10,000		0.89/0.94	
TRENCH	Hand Panning Concentrate	35.0	0.35	85.3	**31.7**
	Hand Panning Tailing	45.0	0.45	10.5	5.0
	Knelson Tailing	9,920	99.2	0.60	63.3
	Comb Hand & Knelson Tailing (calc)	9,965	99.7	0.64	***68.3***
	Head (calc)	10,000	100.0	0.94	100.0
	Head (direct)	10,000		0.89/0.98	
AVE GRAVITY CONCENTRATE			**0.29**	**109**	**24.8**

Table 13-12 Individual Composites Gravity Separation Summary

SAMPLE	GRAVITY CONCENTRATE			COMB GRAV TAILING
	WEIGHT, %	GRADE, Au, g/t	RECOVERY, Au %	Au, g/t (calc)
PAIOL SAPROLITE	0.20	58	17.9	0.53
PAIOL SDCX	0.35	70	25.5	0.73
PAIOL SDQX	0.56	78	33.6	0.86
VIRA SAIA QSX	0.14	129	11.8	0.81
VIRA SAIA GDM	0.11	234	28.3	0.65
TRENCH	0.35	85	31.7	0.64
ALMAS AVE	**0.29**	**109**	**24.8**	**0.70**

13.5.1.2 FLOTATION

Samples of the whole ore and gravity tailings were subjected to flotation in order to evaluate the recovery of gold into a flotation concentrate. The main objective of this test program was to maximize the recovery of gold into a flotation concentrate for subsequent cyanide leaching. The effects of fineness of grind and various reagent schemes were evaluated in a series of bulk rougher tests. There were three rougher flotation tests conducted on each ore type and six exploratory rougher tests conducted on the whole ore. No other flotation flowsheet configurations were evaluated at this stage, due to the limited scope of the program.

The sample gravity tailings were reground to the specified grind size and subjected to flotation. The effect of fineness of grind (K_{80} = 150, 106, and 75 microns) was evaluated in this test series. The reagents applied were PAX, as a sulphide collector, copper sulphate as a promoter, and MIBC as a frother. Four stages of rougher concentrates were collected separately over a period of 17 minutes and submitted for gold assays. The rougher tailings were analysed for gold and sulphur. Visually, the flotation appeared to be sluggish with a non-stable froth. It has been noted that very high collector (PAX) dosages of 120-240 g/t were applied in this test series in an attempt to recover all the residual sulphides and gold. Also 40 g/t $CuSO_4$ was added into the last rougher stage. The test results are presented in Table 13-13. The best results were achieved at a finer grind of 75 microns, as illustrated in Figure 13-3 showing cumulative gold grades after each rougher stage versus recovery. The results indicated that 82-87% of the gold was recovered into a flotation concentrate for the Paiol and Vira Saia composites and 71% gold recovery for the Trench composite. The flotation tailings contained 0.11- 0. 24 g/t Au and 0.03% S, (indicating that all the sulphides were recovered into the flotation concentrate). It is more likely that the residual gold present in the tailings is associated with the iron oxides and/or silicates. A diagnostic gold deportment study will be required to confirm the gold associations. The overall recovery by gravity separation and flotation was 86-92 % for the Paiol and Vira Saia composites and 80% for the Trench composite. A very good correlation between the calculated head grade and the assayed gravity tailings grade (flotation feed) was shown for all the composites.

In addition, the PAIOL SDQX whole ore sample, without a gravity separation circuit, was also subjected to exploratory evaluation of the sulphide flotation in order to evaluate the effect of pH and various collectors. These tests were conducted at a grind size of K_{80}= 75 microns. The test results are presented in Table 13-14. Despite applying strong collectors combinations, the results for all the tests were fairly similar. The best test results were achieved with the PAX or SIBX collectors, copper sulphate, and Dowfroth-250 additions at a natural pH. The recovery of gold in these tests was 91-92%. The flotation tailings contained 0.10-0.12 g/t Au and 0.03% S. These results from the whole ore flotation were comparable with the results obtained from gravity- flotation circuit.

Only exploratory scoping flotation testwork has been conducted on the Almas Gold Project composite samples at this stage of testing. Standard bulk sulphide flotation conditions were applied without further optimization including gangue depressing reagents evaluation and possibly different flotation configuration. However, due to relatedly high gold losses into the flotation tailings, the flotation process option has not been further investigated.

Table 13-13 Gravity Tailings Flotation Results – Effect of Grind

SAMPLE	CONDITIONS	GRIND K_{80}, µm	PRODUCTS	WEIGHT, %	Au, g/t	Au DIST'N %	Au O'LL RECOVERY %*
PAIOL SDCX GRAV TAIL	pH- 8.4 120-240 g/t PAX CuSO4- 40 g/t Forther - MIBC Flot time- 17 min	150	Rougher Concentrate	2.6	23.4	73.7	**80.4**
			Rougher Tailings	97.4	0.23	26.3	
			Head (calc)	100.0	0.84	100.0	
			Head (direct)		0.73		
		106	Rougher Concentrate	2.8	22.2	78.4	**83.9**
			Rougher Tailings	97.2	0.18	21.6	
			Head (calc)	100.0	0.79	100.0	
			Head (direct)		0.73		
		75	Rougher Concentrate	3.6	19.0	84.5	**88.4**
			Rougher Tailings	96.4	0.13	16.1	
			Head (calc)	100.0	0.82	100.5	
			Head (direct)		0.73		
PAIOL SDQX GRAV TAIL	pH- 8.4 Collector- 120g/t PAX CuSO4- 40 g/t Forther - MIBC Flot time- 17 min	150	Rougher Concentrate	3.6	18.4	84.8	**89.9**
			Rougher Tailings	96.4	0.12	15.2	
			Head (calc)	100.0	0.77	100.0	
			Head (direct)		0.86		
		106	Rougher Concentrate	3.5	19.2	80.9	**87.3**
			Rougher Tailings	96.5	0.17	19.1	
			Head (calc)	100.0	0.84	100.0	
			Head (direct)		0.86		
		75	Rougher Concentrate	3.9	17.8	87.3	**91.6**
			Rougher Tailings	96.1	0.11	12.7	
			Head (calc)	100.0	0.80	100.0	
			Head (direct)		0.86		
VIRA SAIA QSX GRAV TAIL	pH- 8.1 Collector- 120 g/t PAX CuSO4- 40 g/t Forther - MIBC Flot time -17 min	150	Rougher Concentrate	5.5	10.1	70.8	**74.3**
			Rougher Tailings	94.5	0.24	29.2	
			Head (calc)		0.78	100.0	
			Head (direct)		1.35		
		106	Rougher Concentrate	5.7	11.1	76.4	**79.2**
			Rougher Tailings	94.3	0.21	23.6	
			Head (calc)	100.0	0.84	100.0	
			Head (direct)		1.35		
		75	Rougher Concentrate	7.4	9.2	84.0	**85.9**
			Rougher Tailings	92.6	0.14	16.0	
			Head (calc)	100.0	0.81	100.0	
			Head (direct)		1.35		
VIRA SAIA GDM GRAV TAIL	pH- 8.1 Collector- 120 g/t PAX CuSO4- 40 g/t Forther - MIBC Flot time -17 min	150	Rougher Concentrate	5.5	7.28	72.7	**80.4**
			Rougher Tailings	94.5	0.16	27.3	
			Head (calc)	100.0	0.55	100.0	
			Head (direct)		0.65		
		106	Rougher Concentrate	5.9	6.65	74.0	**81.4**
			Rougher Tailings	94.1	0.15	26.0	
			Head (calc)	100.0	0.63	100.0	
			Head (direct)		0.65		
		75	Rougher Concentrate	9.0	4.72	82.4	**87.4**
			Rougher Tailings	91.0	0.10	17.6	
			Head (calc)	100.0	0.62	100.0	
			Head (direct)		0.65		
TRENCH GRAV TAIL	pH- 8.1 Collector- 120 g/t PAX CuSO4- 40 g/t Forther - MIBC Flot time -17 min	150	Rougher Concentrate	9.0	4.20	66.5	**77.1**
			Rougher Tailings	91.0	0.21	33.5	
			Head (calc)	100.0	0.57	100.0	
			Head (direct)		0.64		
		106	Rougher Concentrate	10.3	3.00	68.5	**78.5**
			Rougher Tailings	89.7	0.16	31.5	
			Head (calc)	100.0	0.46	100.0	
			Head (direct)		0.64		
		75	Rougher Concentrate	10.2	3.30	71.3	**80.4**
			Rougher Tailings	89.8	0.15	28.7	
			Head (calc)	100.0	0.47	100.0	
			Head (direct)		0.64		

* Overall Gold Recovery by Gravity Separation and Flotation

Figure 13-3 Effect of Grind: Au Grade vs. Recovery

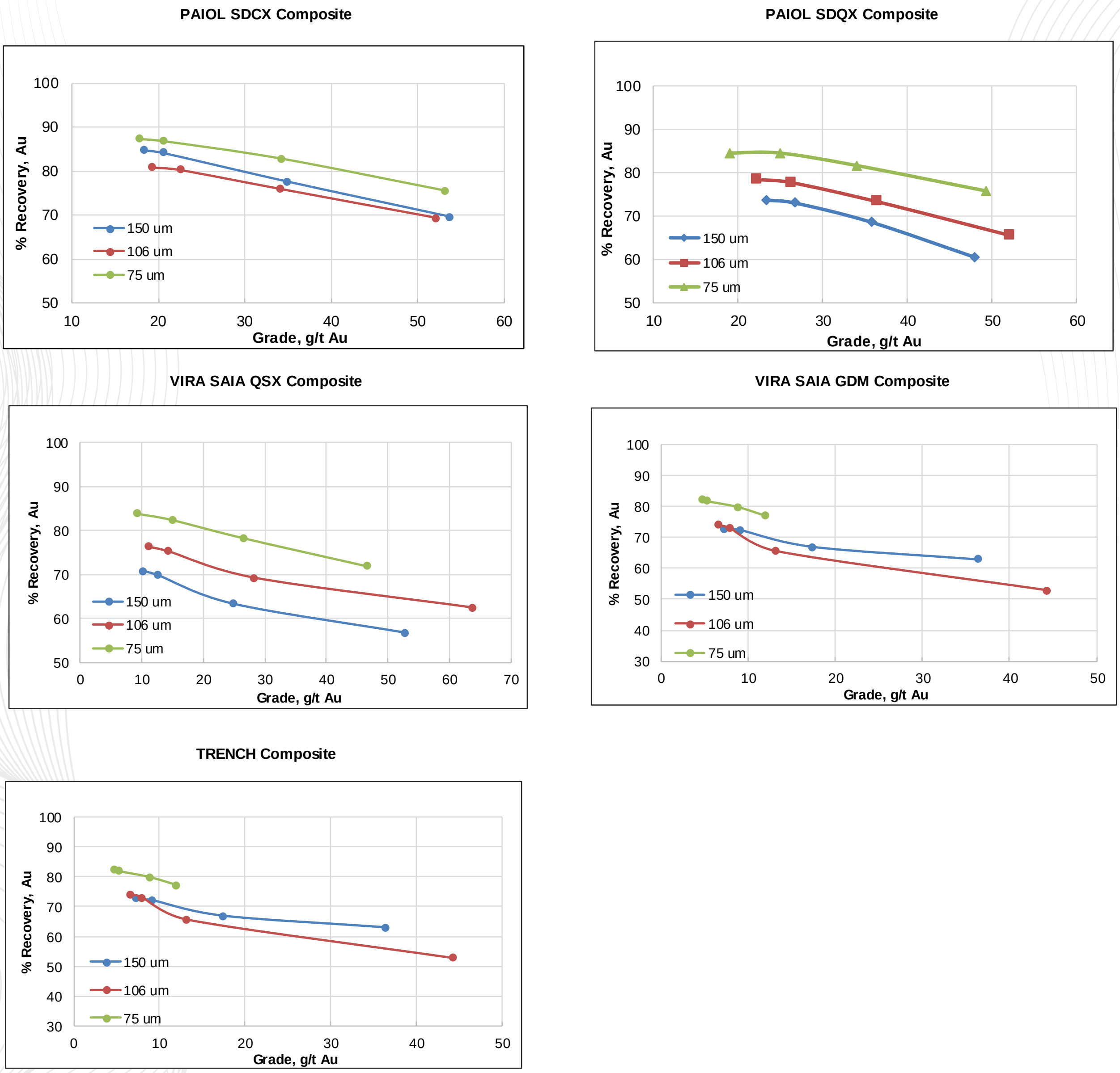


Table 13-14 PAIOL SDQX Whole Ore Flotation Results – Effect of Reagents and pH

CONDITIONS	GRIND K_{80}, µm	PRODUCTS	WEIGHT, %	Au g/t	S %	Au DIST'N %	S DIST'N %
PAIOL SDQX - FLOTATION OF ORIGINAL SAMPLE - P_{80} 75 MICRONS - PAX							
pH-8.6	75	Rougher Concentrate	6.9	15.7	-	**90.8**	94.2
Collector- PAX- 40 g/t		Rougher Tailings	93.1	0.12	0.03	9.2	5.8
CuSO4- 40 g/t, DF 250		Head (calc)	100.0	*1.19*	0.48	100.0	100.0
		Head (direct)		*1.20*			
PAIOL SDQX - FLOTATION OF ORIGINAL SAMPLE - P_{80} 75 MICRONS - MX980							
pH-5.5 with H2SO4	75	Rougher Concentrate	8.3	12.3	-	**88.6**	90.5
Aero MX980- 40 g/t		Rougher Tailings	91.7	0.14	0.05	11.4	9.5
CuSO4- 40 g/t, DF 250		Head (calc)	100.0	*1.16*	0.48	100.0	100.0
PAIOL SDQX - FLOTATION OF ORIGINAL SAMPLE - P_{80} 75 MICRONS - A3418							
pH-5.5 with H2SO4	75	Rougher Concentrate	7.0	14.3	-	**86.8**	92.2
Collector- A3418- 40 g/t		Rougher Tailings	93.0	0.16	0.04	13.2	7.8
CuSO4- 40 g/t, DF 250		Head (calc)	100.0	*1.15*	0.48	100.0	100.0
PAIOL SDQX - FLOTATION OF ORIGINAL SAMPLE - P_{80} 75 MICRONS - A412							
pH-9.0 with lime	75	Rougher Concentrate	6.5	15.5	-	**87.6**	86.4
Collector- A412- 40 g/t		Rougher Tailings	93.5	0.15	0.07	12.4	13.6
CuSO4- 40 g/t, DF 250		Head (calc)	100.0	*1.16*	0.48	100.0	100.0
PAIOL SDQX - FLOTATION OF ORIGINAL SAMPLE - P_{80} 75 MICRONS - OX100							
pH-9.0 with lime	75	Rougher Concentrate	7.8	12.0	-	**81.0**	42.4
Collector- OX100- 40 g/t		Rougher Tailings	92.2	0.24	0.30	19.0	57.6
CuSO4- 40 g/t, DF 250		Head (calc)	100.0	*1.15*	0.48	100.0	100.0
PAIOL SDQX - FLOTATION OF ORIGINAL SAMPLE - P_{80} 75 MICRONS - SIBX							
pH-8.5	75	Rougher Concentrate	8.1	13.2	-	**92.3**	94.3
Collector- SIBX- 40 g/t		Rougher Tailings	91.9	0.10	0.03	7.7	5.7
CuSO4- 40 g/t, DF 250		Head (calc)	100.0	*1.15*	0.48	100.0	100.0

13.5.2 FLOWSHEET 2 – CYANIDATION OF GRAVITY TAILINGS

During the earlier testwork programs, the emphasis was placed on the flowsheet configuration that included either whole ore leaching, or gravity separation followed by cyanidation. This segment of testing included the gravity separation circuit prior to cyanidation. The recovery of gold from the coarser 'as is' and reground gravity tailings of all of the samples from the Almas deposit was evaluated by direct cyanidation and CIL. Three cyanidation and three CIL tests were conducted on each ore type. The tests were conducted in bottles-on-rolls under the conditions presented below. The effect of grind (K_{80}= 150, 106, and 75 microns) was evaluated for each sample.

Cyanidation/CIL test conditions:

- 500 grams ground ore leached at 40% solids
- Grind size K_{80}-150, 106, and 75 µm
- Target pH -10.5-11 adjusted with lime additions
- Target NaCN concentration maintained at ~ 1 g/L
- 10 g/L Carbon (pre-attritioned) for CIL
- 48 hours retention time (with pregnant solution subsample at 24 hours)
- Dissolved Oxygen concentration measured throughout the test period

The final products were submitted for gold assays. The residues were assayed in triplicate and the average value was reported.

The cyanidation/CIL test results are summarized in Table 13-15 and illustrated in Figure 13-4.

Table 13-15 Gravity Tailings Cyanidation/CIL Test Results

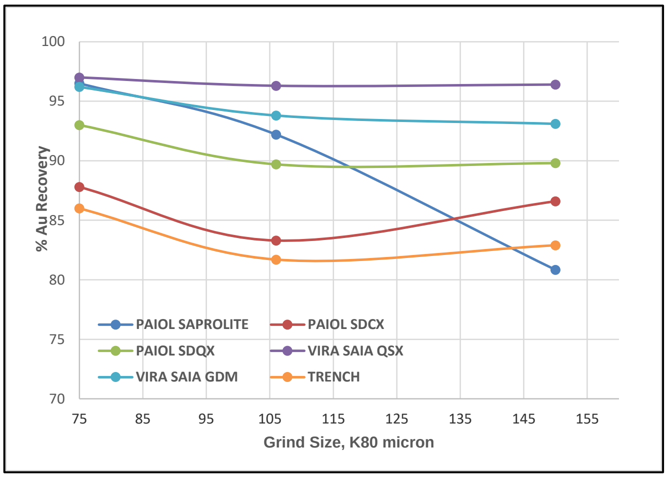
			CN RESULTS								CIL RESULTS					
Sample	Grind Size K_{80} µm	Comb Grav Tails, Au g/t*	24 hr Liquor Assay Au mg/L	48 hr Liquor Assay Au mg/L	Residue Assay Au g/t	48 hr Au Extraction %	Au Extraction % Normalized^	Calc Head Au g/t**	Estimated NaCN Cons kg/t	Estimated Lime Add'n kg/t	Residue Assay Au g/t	Au Extraction %	Au Extraction % Normalized^	Calc Head Au g/t**	Estimated NaCN Cons kg/t	Estimated Lime Add'n kg/t
PAIOL SAPROLITE	150	0.53	0.29	0.33	0.10	80.8	80.9	0.53	1.1	1.3	0.13	80.1	75.6	0.62	1.2	2.4
	106		0.36	0.41	0.05	92.2	90.6	0.58	1.1	1.5	0.05	94.8	90.6	0.96	1.2	2.7
	75		0.34	0.35	0.02	96.5	96.2	0.52	0.9	1.6	0.02	96.8	96.2	0.60	1.1	2.8
PAIOL SDCX	150	0.73	0.42	0.43	0.10	86.6	86.3	0.75	0.9	0.2	0.12	86.7	83.6	0.84	1.0	0.5
	106		0.48	0.46	0.12	83.3	83.6	0.71	0.7	0.2	0.11	87.7	84.9	0.88	0.9	0.3
	75		0.42	0.43	0.07	87.8	90.4	0.61	1.0	0.2	0.07	92.2	90.4	0.81	0.9	0.3
PAIOL SDQX	150	0.86	0.59	0.57	0.08	89.8	90.7	0.79	0.7	0.1	0.09	91.2	89.5	0.93	0.8	0.3
	106		0.45	0.45	0.07	89.7	91.9	0.71	0.9	0.2	0.08	91.9	90.7	0.88	0.7	0.3
	75		0.49	0.49	0.06	93.0	93.0	0.79	0.8	0.2	0.05	94.5	94.2	0.93	0.9	0.3
VIRA SAIA QSX	150	0.88	0.47	0.45	0.03	96.4	96.6	0.79	0.7	0.2	0.04	96.1	95.5	0.89	0.6	0.3
	106		0.51	0.49	0.03	96.3	96.6	0.73	0.6	0.1	0.02	97.5	97.7	0.88	0.8	0.2
	75		0.46	0.46	0.02	97.0	97.7	0.68	0.7	0.2	0.02	98.3	97.7	0.85	0.9	0.2
VIRA SAIA GDM	150	0.64	0.33	0.36	0.04	93.1	93.8	0.54	0.7	0.1	0.03	95.3	95.3	0.59	0.7	0.2
	106		0.31	0.29	0.03	93.8	95.3	0.46	0.6	0.2	0.02	97.0	96.9	0.58	0.8	0.2
	75		0.32	0.28	0.02	96.2	96.9	0.41	0.7	0.2	0.01	97.9	98.4	0.57	0.7	0.2
TRENCH	150	0.64	0.36	0.36	0.09	82.9	85.9	0.54	0.7	0.6	0.11	82.8	82.8	0.63	0.9	0.8
	106		0.31	0.31	0.10	81.7	84.4	0.56	0.7	0.6	0.09	86.0	85.9	0.59	0.9	0.8
	75		0.35	0.33	0.08	86.0	87.5	0.55	1.0	0.6	0.08	88.5	87.5	0.64	0.8	0.8

* calc from gravity separation test products
** calc from cyn test products
^ calc from direct head and residue grades

The following information was obtained from the test results:

- The initial pulp pH was in the range of 7.2-7.5 for the Saprolite material and 8.2-8.5 for the other composites, indicating that a pre-aeration stage maybe required. The pH was stable after the initial lime additions.
- The dissolved oxygen concentration was relatively high and averaged at 7-8 mg/L throughout the test.
- Table 13-14 includes the comparison between the calculated gravity tailings grade (feed to cyanidation) and the calculated head grade obtained from the cyanidation test metallurgical balance. The grades were relatively comparable for all the samples.
- The gold extraction presented in Table 13-15 shows the comparative calculated extractions obtained from the test products metallurgical balance and the 'normalized' extraction calculated as a difference between the gravity tailings grade and the cyanidation residue grade (in order to account for the variations in the calculated head grades).
- All the samples tested were amenable to cyanide leaching and showed excellent gold extraction after 48 hours of leaching averaging above 90% with the average residue assay of 0.06 g/t Au.
- The intermediate 24 hour pregnant solution assays have indicated that the leaching has probably been completed after 24 hours of leaching. Additional kinetic testing will be required to confirm the retention time.
- The CIL results were similar to the cyanidation results showing the average residue assay of 0.06 g/t Au.
- The effect of grind has been somewhat demonstrated. This series of tests showed a trend of the residual gold grade reduction at a finer grind. However, because of fairly low reported residue grades (0.02-0.1g/t Au), slight fluctuations in the calculated gold grades and the allowed procedural and analytical detection limits, the results can be difficult to compare. Additional testwork will be required to confirm the optimum grind size.
- The estimated sodium cyanide consumption averaged at 0.8 kg/t. The lime addition was higher for the Saprolite and Trench composites, between 0.6-2.8 kg/t and approximately 0.2 kg/t for the PAIOL and VIRA SAIA composites. A pre-aeration stage with lime conditioning may be required prior to cyanide addition.

Figure 13-4 Cyanidation Gold Recovery vs. Grind Size



13.5.3 OVERALL RESULTS

The overall results achieved by gravity separation followed by cyanidation/CIL are presented in Table 13-15. The results indicated that all the samples leached with consistent kinetics with the arithmetic average gravity recovery of 25%, and the overall gold recovery achieved by gravity separation followed by cyanidation/CIL for the three grind sizes tested was 93-94% leaving a residue assaying 0.06 g/t Au after 48 hours of leaching.

The overall test results comparing the flowsheets tested - gravity separation/cyanidation and gravity separation/ flotation are also shown in Table 13-16. The overall results showed that the arithmetic average overall gold recovery by gravity separation/ CIL at grind sizes of K_{80} 106 and 75 microns was 94.5% and 96.2% leaving a residue assaying 0.06 and 0.04 g/t Au, respectively.

The estimated average overall recovery by gravity separation/flotation followed by the concentrate leaching can be approximately 81% with the rougher tailings assaying 0.12 g/t Au and the estimated final overall residue (calculated value of flotation tailings plus the estimated leach residue) of ~ 0.3 g/t Au.

These results indicated that the direct cyanide leaching without the flotation circuit resulted in a significantly higher overall gold recovery under the test conditions tested. A trade-off study has been conducted by Ausenco to compare both flowsheets and to confirm the economic viability of each processing flowsheet. The whole ore leaching or the gravity separation followed by leaching flowsheets have been selected for further evaluation in the subsequent stage of testing.

13.5.4 HEAP LEACH AMENABILITY RESULTS

The Almas Gold Project Paiol Sulphide and Saprolite material was submitted for heap leach amenability testing in bottles-on-rolls. The main objective of this testing was to conduct a preliminary assessment of gold recovery by simulating heap leach conditions and to determine the reagent requirements for subsequent column testing, if required. There was no agglomeration or permeability testing conducted at this phase.

Each composite was prepared by blending and crushing to -1/2-inch. The samples were split into the test charges and further crushed to the required sizes. The representative head samples were riffled out and submitted for gold assays.

The reported average head assays were as follows:

- Paiol Saprolite - 0.52 g/t Au
- Paiol Sulphide – 1.06 g/t Au

The crush sizes evaluated were -1/2 inch for the Saprolite material and -1/2, -1/4, and -1/8 inch for the Sulphide material. The tests were conducted in bottles-on-rolls at 45% solids on a 2 kg samples. The bottles were rolled intermittently, rolling for one minute every hour in order to minimize attrition and simulate the heap leach/column testing.

The pH was maintained at the 10.0-10.5 level with lime additions and the cyanide concentration was maintained at 0.5 g/L NaCN throughout the test period. The tests were conducted for a period of 14-20 days with intermittent removal of pregnant solution subsamples for gold assays. The final pulp was filtered and washed, and the products submitted for analyses. The test results are summarized in Table 13-17.

The results indicated that the recovery of gold for the Saprolite ore was approximately 88% and for the Sulphide ore was in the range of 40-68%, increasing with the crush size reduction. The cyanide and lime consumption were below 1 kg/t and 2 kg/t, respectively. However, the lime consumption reported for the Saprolite material was high at 10 kg/t. The reason for such high lime consumption has not been determined.

Further confirmatory testing will be required to evaluate the ore amenability to heap leaching, if this process route will be considered.

Table 13-16 Overall Test Results

SAMPLE	GRIND SIZE K_{80} µm	AVERAGE ORE HEAD GRADE Au g/t	GRAVITY	GRAVITY/CN*		GRAVITY/CIL*				GRAVITY/FLOTATION **	
			Recovery Au %	Overall Recovery Au %	Residue Au g/t	Overall Recovery Au %	Residue Au g/t	Estimated NaCN Cons kg/t	Estimated Lime add'n kg/t	Overall Recovery Au %	Ro Tailings Au g/t
PAIOL SAPROLITE	150	0.65	17.9	84.3	0.10	83.7	0.13	1.2	2.4	-	-
	106			93.6	0.05	95.7	0.05	1.2	2.7	-	-
	75			97.1	0.02	97.4	0.02	1.1	2.8	-	-
PAIOL SDCX	150	0.95	25.5	90.0	0.10	90.1	0.12	1.0	0.5	80.4	0.23
	106			87.6	0.12	90.8	0.11	0.9	0.3	83.9	0.18
	75			90.9	0.07	94.2	0.07	0.9	0.3	88.4	0.13
PAIOL SDQX	150	1.31	33.6	93.2	0.08	94.2	0.09	0.8	0.3	89.9	0.12
	106			93.2	0.07	94.6	0.08	0.7	0.3	87.3	0.17
	75			95.4	0.06	96.3	0.05	0.9	0.3	91.6	0.11
VIRA SAIA QSX	150	1.53	11.8	96.8	0.03	96.6	0.04	0.6	0.3	74.3	0.24
	106			96.7	0.03	97.8	0.02	0.8	0.2	79.2	0.21
	75			97.4	0.02	98.5	0.02	0.9	0.2	85.9	0.14
VIRA SAIA GDM	150	0.91	28.3	95.1	0.04	96.6	0.03	0.7	0.2	80.4	0.16
	106			95.6	0.03	97.8	0.02	0.8	0.2	81.4	0.15
	75			97.3	0.02	98.5	0.01	0.7	0.2	87.4	0.10
TRENCH	150	0.94	31.7	88.3	0.09	88.3	0.11	0.9	0.8	77.1	0.21
	106			87.5	0.10	90.4	0.09	0.9	0.8	78.5	0.16
	75			90.4	0.08	92.1	0.08	0.8	0.8	80.4	0.14
AVERAGE at K_{80}-106um		**1.05**	**24.8**	**92.4**	**0.07**	**94.5**	**0.06**	**0.9**	**0.8**	**82.1**	**0.17**
AVERAGE at K_{80}-75um		**1.05**	**24.8**	**94.7**	**0.04**	**96.2**	**0.04**	**0.9**	**0.8**	**86.7**	**0.12**

*Gravity concentrate leach recovery has not been accounted for

**Gravity and flotation concentrates leach recovery has not been accounted for

Table 13-17 Heap Leach Amenability Test Results

SAMPLE	CRUSH SIZE inch	LEACH TIME days	RESIDUE ASSAY Au g/t	Au EXTRACTION %	CALC HEAD Au g/t	DIRECT HEAD Au g/t	ESTIMATED AAD'N NaCN kg/t	ESTIMATED ADD'N LIME kg/t
PAIOL SAPROLITE	1/2"	20	0.07	88.2	0.58	0.52	0.6	10.3
PAIOL SULPHIDE	1/2"	14	0.66	39.5	1.09	1.06	0.9	1.6
	1/4"	14	0.67	49.8	1.33	1.06	0.9	1.7
	1/8"	14	0.35	67.8	1.07	1.06	0.8	1.6

13.6 BLEND 3-YEAR COMPOSITE TESTWORK PROGRAM

A confirmatory testwork program has been conducted during the 2020 program campaign at the feasibility study level. The tests were conducted at the "Testwork Process Development" metallurgical laboratory in Brazil. The testing included the following investigations:

- Evaluation of gravity circuit inclusion prior to cyanide leaching
- Comparison of whole ore leaching vs. gravity separation followed by gravity tailings leaching
- Cyanide leaching conditions optimization, leaching mode pre-leach/CIL vs. direct CIL evaluation
- Cyanide destruction
- Solids/liquid separation characterization

13.6.1 GRAVITY SEPARATION GRG TESTWORK

Two standard gravity recoverable gold (E-GRG) tests were conducted on the Blend 3-Y composite sample. The standard three-stage protocol has been applied with the final targeted grind size of P_{80} of 75 microns. Each stage test products were submitted for size fraction analysis for gold. The results were submitted to FLSmidth for further evaluation and modeling. The results indicated that the cumulative 3-stage GRG recoveries varied between 31% and 39% with the respective calculated head grades of 1.9 and 1.7 g/t Au, as shown in Table 13-18. The size classification of the GRG has been determined as course to moderate using the AMIRA size classification scale by FLS.

It has been concluded that the ore is amenable to gravity recovery and the GRG particle distribution is fairly coarse. As such, a moderate gravity circuit with concentrate intensive cyanidation has been suggested for inclusion in the flowsheet. Modelling has been undertaken and several options for a gravity circuit installation were suggested. The gravity equipment suggested by FLS was as follows:

- One KC-QS40 Knelson concentrator installed at cyclone underflow
- One Consep Acacia CS2000 unit for intensive cyanidation system to treat the Knelson concentrate. This unit is sized to treat 24 hour production of Knelson concentrate.

For process design purposes, gravity gold recovery of 17.5% is used treating 25% of the cyclone underflow stream.

Table 13-18 GRG Test Summary

GRIND SIZE			PRODUCT	WEIGHT		ASSAY	DIST'N
				g	%	Au g/t	Au %
P_{80} =	845	µm	Stage 1 Concentrate	79.0	0.8	26.9	11.8
P_{80} = -75mm=	169 20.1	µm %	Stage 2 Concentrate	83.0	0.9	26.7	12.3
P_{80} =	68	µm	Stage 3 Concentrate	55.4	0.6	24.1	7.4
			Final Tailings	9287	97.7	1.33	68.5
			Calc Head	9505	100.0	1.89	100.0
			Knelson Concentrate	217.4	2.3	26.1	31.5
			GRG Value				**31.5**
P_{80} =	796	µm	Stage 1 Concentrate	86.2	0.9	25.0	13.3
P_{80} = -75mm=	197 21.8	µm %	Stage 2 Concentrate	79.2	0.8	33.1	16.2
P_{80} =	62	µm	Stage 3 Concentrate	55.2	0.6	29.9	10.2
			Final Tailings	9154	97.6	1.06	60.2
			Calc Head	9375	100.0	1.72	100.0
			Knelson Concentrate	220.6	2.4	29.1	39.8
			GRG Value				**39.8**

13.6.2 WHOLE ORE CYANIDE LEACHING

The first series of tests was conducted without gravity separation, as direct cyanide leaching. The tests were conducted in bottles-on-rolls under the conditions presented below.

Cyanidation/CIL test conditions:

- 1,000-2,000 grams ground ore leached at 45% solids
- Grind size K_{80} - 106 and 75 µm
- Target pH -10.5-11 adjusted with lime additions
- Initial NaCN concentration at 1 g/L
- 20 g/L Carbon (pre-attritioned) for CIL
- 24-48 hours retention time (with intermediate kinetic subsamples)
- Dissolved Oxygen concentration measured throughout the test period at > 4 mg/L

The intermediate and final test products were submitted for gold assays. The residues were assayed in triplicate and the average value was reported.

The effects of leach kinetics, particle grind size of K_{80} of 106 and 75 microns, retention time, and cyanidation versus CIL or pre-leach/CIL flowsheet configurations have been evaluated as shown in Table 13-19. The gold extraction presented in Table 13-19 shows the comparative calculated extractions obtained from the test products metallurgical balance and the ‘normalized’ extraction calculated as the difference between the head grade and the cyanidation residue grade (in order to account for the variations in the calculated head grades).

The final leach residues and barren solution ICP scans from the two selected CIL tests conducted for 24 hours at 106- and 75-microns grind sizes are presented in Table 13-20 and 13-21 respectively.

Table 13-19 Whole Ore Leach Results

SAMPLE	LEACH TIME hours	CN TEST MODE	GRIND SIZE P80, µm	RESIDUE ASSAY Au g/t	EXTRACTION Au %*	NORMALIZED EXTRACTION Au %^	CALC HEAD Au g/t	NaCN CONS kg/t	LIME CONS kg/t
Blend 3 Y	24	CN	106	0.11	92.2	91.4	1.38	0.17	0.96
	24	CIL	75	0.05	95.7	96.1	1.23	0.31	0.93
Blend 3 Y	24	CN	75	0.08	94.3	93.8	1.42	0.18	0.94
	24	CIL	75	0.08	93.6	93.8	1.31	0.34	1.00
Blend 3 Y^^	24	CN	106	0.11	93.1	91.4	1.54	0.25	0.93
	24	CIL	75	0.12	91.8	90.6	1.40	0.27	0.96
Blend 3 Y	4	CIL	106	0.30	79.4	76.6	1.43	0.12	0.82
	4	CIL	75	0.16	87.4	87.5	1.24	0.11	0.78
Blend 3 Y	12	CIL	106	0.13	90.9	89.8	1.40	0.21	1.38
	12	CIL	75	0.13	90.8	89.8	1.42	0.22	1.54
Blend 3 Y	24	CIL	106	0.10	92.3	92.2	1.24	0.34	1.53
	24	CIL	75	0.09	92.8	93.0	1.14	0.27	1.68
Blend 3 Y	48	CIL	106	0.08	93.3	93.8	1.23	0.49	1.94
	48	CIL	75	0.08	93.9	93.8	1.25	0.49	1.32
Blend 3 Y	12+12	CN-CIL	106	0.10	92.0	92.2	1.28	0.33	1.60
	12+12	CN-CIL	75	0.09	92.6	93.0	1.26	0.33	1.64
Blend 3 Y	24	CIL	106	0.09	92.5	93.0	1.21	0.18	0.90
	24	CIL	75	0.08	93.3	93.8	1.12	0.26	0.90
	24	CIL	75	0.11	92.1	91.4	1.35	0.17	1.60
	24	CIL	75	0.11	91.7	91.4	1.37	0.19	1.60
Saprolite	24	CIL	106	0.02	96.4	97.5	0.55	0.39	1.50
	24	CIL	75	0.02	97.2	97.5	0.53	0.40	1.50
AVERAGE at 24 hours 106 µm grind size				**0.10**	**92.4**	**92.0**	**1.33**	**0.25**	**1.18**
AVERAGE at 24 hours 75 µm grind size				**0.09**	**93.1**	**93.0**	**1.29**	**0.26**	**1.25**

*Au Extraction - based on the difference between met balance calculated head grade and residue grades
^ Normalized Au Extraction- based on the difference between direct head grade (1.28 g/t Au) and residue grades for Blend 3 Y
^ Normalized Au Extraction- based on the difference between direct head grade (0.79 g/t Au) and residue grades for Saprolite
^^ Residue washed with NaOH solution

Table 13-20 CIL Residue Analysis

ELEMENTS	g/t, %	24 hour CIL TESTS		ELEMENTS	g/t, %	24 hour CIL TESTS	
		106 µm	75 µm			106 µm	75 µm
Ag	g/t	<3	<3	Ni	g/t	24	25
Al	%	5.56	5.27	P	%	0.06	0.06
As	g/t	40	46	Pb	g/t	<8	<8
Ba	g/t	180	168	S	%	0.53	0.55
Be	g/t	<3	<3	Sb	g/t	<10	<10
Bi	g/t	<20	<20	Sc	g/t	26	25
Ca	%	4.56	4.51	Se	g/t	<20	<20
Cd	g/t	<3	<3	Sn	g/t	<20	<20
Co	g/t	35	34	Sr	g/t	117	114
Cr	g/t	29	27	Th	g/t	<20	<20
Cu	g/t	41	36	Ti	%	0.62	0.63
Fe	%	7.72	7.84	Tl	g/t	<20	<20
K	%	1.16	1.08	U	g/t	<20	<20
La	g/t	<20	<20	V	g/t	197	189
Li	g/t	28	28	W	g/t	30	34
Mg	%	1.72	1.69	Y	g/t	6	7
Mn	%	0.13	0.12	Zn	g/t	107	105
Mo	g/t	<3	<3	Zr	g/t	97	93
Na	%	1.52	1.45				

Table 13-21 CIL Barren Solution Analysis

ELEMENTS mg/L	24 hour CIL TESTS		ELEMENTS mg/L	24 hour CIL TESTS	
	106 µm	75 µm		106 µm	75 µm
Ag	<0,08	<0,08	Mn	0.05	0.05
Al	8.0	7.3	Mo	<0,6	<0,6
As	<3	<3	Na	423	430
Ba	<0,007	0.013	Ni	<0,6	<0,6
Be	<0,002	<0,002	P	<5	<5
Bi	<1	<1	Pb	<2	<2
Ca	2.6	2.3	Sb	<1	<1
Cd	<0,09	<0,09	Se	<3	<3
Co	<0,3	<0,3	Sn	<2	<2
Cr	<0,1	0.3	Sr	0.068	0.064
Cu	2.8	4.3	Ti	0.06	0.04
Fe	5.1	6.1	Tl	<3	<3
K	52	50	V	<0,2	<0,2
Li	<2	<2	Y	<0,02	<0,02
Mg	0.15	0.35	Zn	<0,7	<0,7

The following information was obtained from the test results:

- The first six tests listed in Table 13-19 were the exploratory tests to compare direct cyanidation versus the CIL process (in order to rule out the preg-robbing potential) and to examine the effect of grind size. The next six tests examined the leach kinetics (4-48 hours) and the effect of grind size. The subsequent eight tests evaluated the pre-leach/CIL versus the CIL configurations at two grind sizes.
- The results indicated that the leach kinetics reached a plateau after 24 hours of leaching. The fineness of grind examined (P_{80} of 106 and 75 micron) and the mode of CIL versus pre-leach/CIL did not affect the results. The gold recovery was in the range of 92-93% with the residual gold grade of 0.09-0.10 g/t Au after 24 hours of leaching. The calculated head grade compared well with the direct head grade in this test series and the 'normalised' gold extractions were close in values to the calculated extractions. The NaCN and lime consumptions averaged at 0.3 kg/t and 1.2-1.3 kg/t, respectively. The low cyanide consumption reflected the lack of cyanicides and other cyanide consumers present in the ore.
- The leach products analysis indicated that metal dissolution during cyanidation was low, and there were no obvious concerns with any deleterious elements.

13.6.3 GRAVITY SEPARATION-CYANIDE LEACHING

The second series of tests was conducted with the inclusion of the gravity separation circuit. The gravity separation circuit simulation was conducted in two stages following the flowsheet presented in Figure 13-5. The first stage included gravity separation of the ore crushed to 16 mesh on the laboratory Knelson concentrator. The first stage Knelson concentrate was subjected to an intensive cyanide leach. The leach residue was combined with the gravity tailings and forwarded to the second stage conducted at grind size of 106 and 75 microns. The second stage concentrate was also subjected to intensive leaching. The combined final gravity tailings were subjected to cyanide leaching under the test conditions and results shown in Table 13-22. The effect of leach time, grind size, and the process mode (CIL vs. pre-leach/CIL) were evaluated in this series of tests.

Figure 13-5 Gravity Circuit Flowsheet

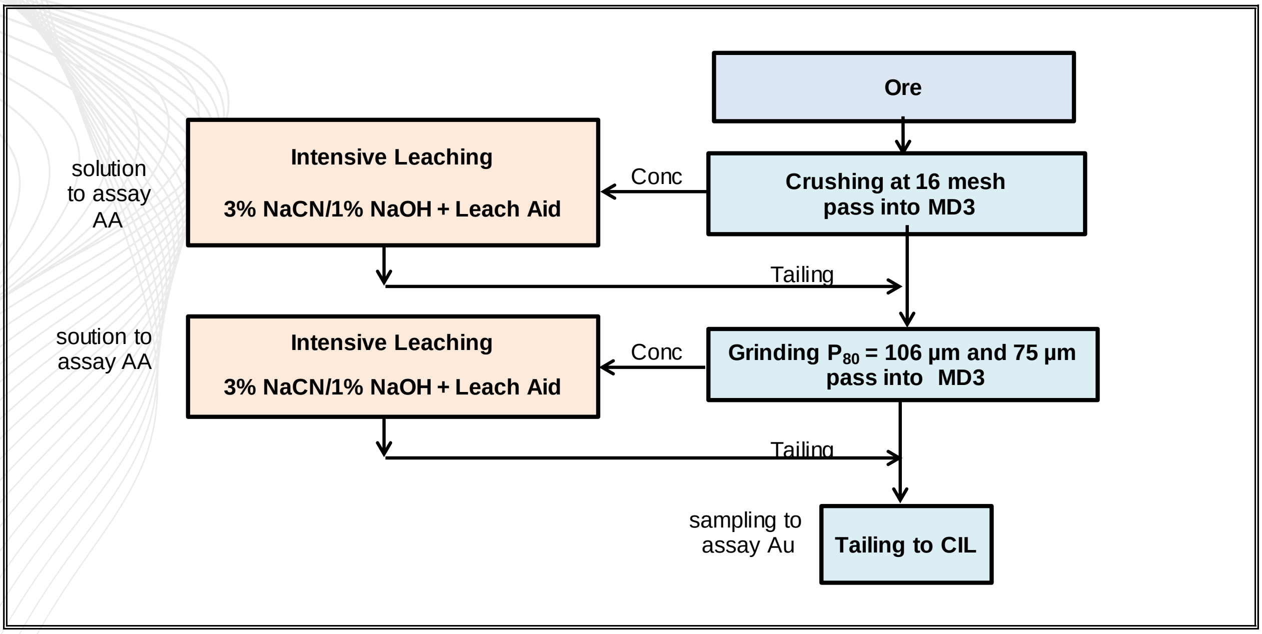


Table 13-22 Gravity Separation-Cyanidation Results

2-STAGE GRAVITY SEPARATION				GRAVITY TAILINGS CYANIDATION- CIL/CIP									
Grav Recovery Au %	Grav Con Weight Recovery %	Grav Conc Grade Au g/t	Calc Head Grade Au g/t	CN test mode	Grind Size P_{80}, µm	Leach Time hours	Residue Assay Au g/t	Individual Cyn Recovery % Au*	O'll Recovery (grav+cyn) Au %**	Normalized Recovery^ Au %	Calc CN Feed Grade Au g/t^^	NaCN Cons kg/t	Lime Cons kg/t
52.4	4.7	16.1	1.43	CIL	106	24	0.09	87.2	93.9	93.0	0.66	0.23	0.85
				CIL	106	48	0.10	85.7	93.2	92.2	0.70	0.45	1.29
				CN-CIL	106	12+12	0.08	89.3	94.9	93.8	0.78	0.31	1.29
56.4	4.6	17.8	1.43	CIL	75	24	0.07	88.6	95.0	94.5	0.65	0.20	0.87
				CIL	75	48	0.07	88.6	95.0	94.5	0.63	0.44	1.28
				CN-CIL	75	12+12	0.07	89.1	95.2	94.5	0.68	0.27	1.02
56.9	6.9	18.2	2.13	CN-CIL	106	12+12	0.10	88.8	95.2	92.2	0.92	0.19	1.61
				CN-CIL^^^	106	12+12	0.20	81.0	91.8	84.4	1.05	0.18	1.62
31.8	6.4	10.5	2.06	CN-CIL	75	12+12	0.09	93.4	95.5	93.0	1.42	0.21	1.63
				CN-CIL^^^	75	12+12	0.23	85.5	90.1	82.0	1.59	0.21	1.62
21.1	6.5	39.7	3.06	CN-CIL	75	3+21	0.10	89.2	91.5	92.2	0.90	0.22	1.69
52.7	7.0	13.9	1.82	CN-CIL	106	12+12	0.11	88.1	94.4	91.4	0.92	0.21	1.41
49.2	6.4	11.2	1.43	CN-CIL	75	12+12	0.10	87.5	93.6	92.2	0.77	0.25	1.32
CIL AVERAGE			1.91	CIL	106	24-48	0.10	86.5	93.5	92.6	0.68	0.34	1.07
				CIL	75	24-48	0.07	88.6	95.0	94.5	0.64	0.32	1.08
CN-CIL AVERAGE			1.91	CN-CIL	106	24	0.12	86.8	94.1	90.4	0.92	0.22	1.48
					75	24	0.12	88.9	93.2	90.8	1.07	0.23	1.46
AVERAGE			**1.91**	**CIL/CN-CIL**	**106**	**24**	**0.11**	**86.6**	**93.8**	**91.5**	**0.80**	**0.28**	**1.28**
					75	**24**	**0.09**	**88.8**	**94.1**	**92.7**	**0.86**	**0.28**	**1.27**

*Individual cyanidation Au recovery- based on the difference between met balance calculated head grade and residue grades

**Overall recovery includes gravity plus cyanidation leach recoveries, assuming gravity concentrate leach recovery=100%

^ Normalized overall recovery - based on the difference between direct head grade (1.28 g/t Au) and residue grades for Blend 3-Y comp

^^ Calculated grav tailings/ CN feed grade

^^^ Confirmed residue assays

The following information was obtained from the test results:

- The results indicated that the two-stage gravity separation tests were not consistent and resulted in the estimated gravity recovery varying from 32 to 57%. It should be noted here that due to the tested flowsheet configuration without the intermediate recycled leach residue assays and the assumption of the intensive cyanidation recovery of 100% after each gravity stage, the gravity circuit recovery should be viewed as an estimated recovery only. The presence of coarse free gold in the ore could have also contributed to the variance in the results. The gravity concentrate weight recovery of 5-7% was significantly higher than in industrial operation. The calculated head grades averaged at 1.9 g/t, which is higher than the direct screened metallic head grade.

  In general, at coarser grinds, gravity separation ahead of leaching can contribute to higher gold recovery by removing coarse gold and therefore reducing the required leach retention time.

- CIL versus Pre-leach/CIL configuration has been examined. The CIL process in general has some inherent disadvantages compared with CIP (such as larger carbon inventory, gold lockup is higher, carbon attrition and the associated gold losses are typically higher, carbon loading is lower and the operating costs are typically higher). However, the CIL process can be more effective if applied in conjunction with a pre-leach step providing a higher gold grade to the first stage of CIL/CIP. Therefore this configuration, including the pre-leach stage has been selected for the process flowsheet.
- The effects of the grind size and the retention time in the CIL and pre-leach/CIL flowsheet configuration have also been examined. The results indicated that the fineness of grind examined (P_{80} of 106 and 75 microns) appeared to have a very minor impact on the overall results and the leach mode configuration of CIL versus pre-leach/CIL did not affect the results as expected.
- Calculated overall recovery of 93.8-94.1% was achieved after 24 hours of leaching, leaving the average residue assay of 0.09-0.11 g/t Au. The NaCN and lime consumptions were 0.2-0.3 kg/t and 1.1-1.5 kg/t respectively. The low cyanide consumption has indicated that there were no significant cyanide consuming species present in the composite sample.

13.6.4 CYANIDE DESTRUCTION

The objective of the cyanide destruction testwork was to investigate the amenability of the Blend 3-Y Composite to detoxification using SO_2/air and to produce treated product containing <2 mg/L residual CN_{WAD} targeting the design criteria parameters of SO_2 additions of 5.5-6.0 g SO_2/g CN_{WAD} and 50 mg/L Cu^{+2} additions at pH 8.5-9.0 with 2 hours retention time.

A series of preliminary batch tests was conducted evaluating the amenability of the sample to treatment using SO_2/air and providing some indication of reagent requirement and the retention time.

It should be noted that batch tests are inefficient and should only be used for determining the amenability of the sample to treatment using SO_2/air and providing some indication of reagent requirement. Continuous testing is required for optimization of parameters such as retention time and reagent requirement.

The slurry used for the cyanide destruction (CND) testwork was the cyanidation slurry at a grind size K_{80} of 75 microns at 45% solids. The final NaCN concentration of the feed pulps to the cyanide destruction testing was allowed to decrease to approximately 100-140 mg /L.

Seven exploratory batch tests were carried out at various sodium metabisulphite ($Na_2S_2O_5$) and/or copper sulphate ($CuSO_4$) dosages. The pH was adjusted with lime additions and oxidation reduction potential of the pulp was monitored. The test condition and results are shown in Table 13-23. The results indicated that the residual CN_{WAD} target of <2 mg/L was achieved under the design criteria conditions at 5.5 g SO_2/1 g CN_{WAD} and 50 mg/L Cu^{+2} additions and that reducing the copper additions below 50 mg/L resulted in higher residual CN_{WAD} concentration.

Table 13-23 Batch Cyanide Destruction Test Conditions and Results

RETENTION TIME	REAGENT ADDIION g / g CN_{WAD}		pH	EMF	PRODUCT SOL'N CN_{WAD}	TREATMENT EFFICIENCY %
hours	SO_2 Equiv.	Cu		mv	mg/L	
Feed	**-**	**-**	**10.5**	**-**	**94**	**-**
2	5.5	50	8.6	86	1.3	98.6
2	5.5	50	8.6	62	1.8	98.1
2	6.0	50	8.6	59	3.5	96.3
2	6.0	50	8.5	55	3.5	96.3
2	5.5	25	8.6	138	10.6	88.8
2	5.5	25	8.7	114	11.9	87.4
2	5.5	119	8.5	144	0.90	99.0

A single continuous test was carried out under the optimum conditions developed in the batch tests. A batch test was completed initially to produce treated pulp with low residual cyanide for use as a starting material for the continuous test. The continuous test examined standard operating conditions for SO_2/air oxidation of the leached pulp at pH 8.5- 9.0 with a retention time of 120 minutes using 5.5-5.7-gram SO_2 per gram CN_{WAD} and 50 mg/L Cu. The leached pulp and the reagents were pumped continuously into a reactor vessel. Air was also applied to the reactor at a continuous flowrate. The target pH was maintained by pumping a lime slurry into the reactor. The oxidation reduction potential (EMF) was monitored and reported. The continuous test was run for four displacement periods to ensure that the steady-state conditions were achieved. The reactor overflowed into a collection vessel which was sub-sampled every 30 minutes in order to monitor the residual CN_{WAD} concentration in the solution phase throughout the destruction test. The collected samples were filtered, preserved and submitted to SGS Geosol for analysis. The test conditions and results are presented in Table 13-24. The test results indicated that the average residual CN_{WAD}, achieved under the conditions tested with a retention time of 2 hours, using the ratio of 5.1-5.7 gram SO_2 equivalent per gram CN_{WAD} and 50 mg/L Cu during the four displacement periods, was 1.2 mg/L, which is below the targeted residual CN_{WAD} concentration of <2 mg/L.

Table 13-24 Continuous Cyanide Destruction Test Conditions and Results

TEST CONDITIONS/PULP					PRODUCT/SOL'N	TREATMENT EFFICIENCY	REAGENT ADDITION, kg/t solids			REAGENT ADDITION, g / g CN_{WAD}		
Contact Time hour	Retention Time, hour	pH	EMF mV	DO mg/L	mg/L	%	SO_2 Equiv.	Cu	$Ca(OH)_2$	SO_2	Cu	$Ca(OH)_2$
Feed	**-**	**9.0**	**-**	**5.5**	**136**	-	-	-	-	-	-	-
0.5	2.0	9.0	85	5.2	0.88	99.4	0.97	0.067	3.1	5.8	0.40	18.8
1.0	2.0	9.1	96	5.4	1.32	99.0	0.88	0.054	1.0	5.3	0.33	5.7
1.5	2.0	9.1	110	5.5	1.32	99.0	1.03	0.069	1.2	6.2	0.42	7.1
2.0	2.0	9.1	103	5.4	0.88	99.4	1.03	0.067	1.3	6.2	0.40	8.0
2.5	2.0	8.9	107	5.2	1.76	98.7	0.99	0.066	2.3	5.9	0.40	13.9
3.0	2.0	9.1	107	5.3	0.88	99.4	0.90	0.058	0.9	5.4	0.35	5.3
3.5	2.0	9.1	113	5.2	0.88	99.4	1.03	0.067	1.3	6.2	0.40	8.0
4.0	2.0	8.9	124	5.5	0.88	99.4	0.96	0.062	1.1	5.8	0.37	6.7
4.5	2.0	9.1	145	5.3	0.88	99.4	1.03	0.069	1.5	6.2	0.42	9.3
5.0	2.0	9.0	164	5.5	1.32	99.0	1.02	0.064	1.1	6.1	0.39	6.5
5.5	2.0	9.1	189	5.5	1.32	99.0	0.99	0.066	1.1	5.9	0.40	6.8
6.0	2.0	9.0	190	5.4	1.32	99.0	1.03	0.067	1.0	6.2	0.40	6.2
6.5	2.0	8.9	198	5.7	2.20	98.4	1.03	0.067	1.3	6.2	0.40	8.0
7.0	2.0	9.0	206	5.1	1.32	99.0	0.99	0.068	1.2	6.0	0.41	7.3
7.5	2.0	9.1	213	5.3	1.32	99.0	0.99	0.066	1.2	5.9	0.40	7.1
8.0	2.0	9.1	210	5.4	1.32	99.0	0.90	0.060	1.2	5.4	0.36	7.1
Average		**9.0**	_	**5.4**	**1.24**	**99.1**	**0.98**	**0.065**	**1.4**	**5.9**	**0.39**	**8.2**
Maximum		9.1	213	5.7	2.20	99.4	1.03	0.069	3.1	6.2	0.42	18.8
Minimum		8.9	85	5.1	0.88	98.4	0.88	0.054	0.9	5.3	0.33	5.3

13.6.5 SOLID/LIQUID SEPARATION TESTING

FLSmidth Brazil (FLS) conducted flocculant screening, static settling, and vane-rheology tests on the Blend 3-Year pre-leaching sample. The testwork was conducted in June 2020 at FLS' laboratory located in Sao Paulo, Brazil.

The test sample was submitted as two ground sub-sample pulps differentiated by their particle sizes. The P_{80} values were 75 microns and 106 microns. Each subsample contained about 33% wt. solids. The pH of the as-received subsamples was 8.5. The pH was adjusted to 9.5 using lime prior to being subjected to testing.

Reagent screening results indicated that BASF Magnafloc 10, an anionic polyacrylamide flocculant with high molecular weight and low charge density, produced good clarification and sedimentation rate.

The best screening results were produced at 10% wt. solids content of the autodiluted thickener feed, for both sub-samples (particle sizes) tested. Autodilution was done using overflow, therefore it will not require addition of fresh water into the feedwell of the industrial thickener.

The static setting tests established a common design criterion for both grind sizes. This allowed in turn for the preliminary sizing of the high-rate thickener that can handle each stream, or any blend thereof. Accordingly, the flocculant dosage ranged from 17 to 25 g/t and from 5 to 10 g/t for the finer (P_{80} ~75 microns) and coarser (P_{80} ~106 microns) samples, respectively. Under these conditions, and, at 33% wt. feed autodiluted to 10% wt., the testwork-predicted solids load was 1.4-tph/m^2 for both samples. This is equivalent to a specific unit area of 0.05 m^2/ tpd.

The corresponding underflows predicted solids concentration was approximately 50.0 wt% for both samples. This solids content was realized during a residence time of approximately 60 minutes. Vane-rheology measurements on these 50% wt. underflows determined that the yield-stress values ranged from 5 - 10-Pa for both samples.

The results allowed establishing a common preliminary sizing criterion for both grind sizes, and for a dry-solid throughput of 195 t/ h being fed as 33% wt. pulp. The underflow removal method consisted of a centrifugal pump, valid for solids content of maximum 50% wt. The recommended process parameters for a Hi-Rate type thickener and for an E-Cat type thickener are shown in Table 13-25 and Table 13-26, as was presented in the FLS report.

Table 13-25 FLS Sedimentation and Rheology Summary for Thickener Type Hi-Rate

PROCESS PARAMETERS	GRIND SIZE 75 µm	GRIND SIZE 106 µm
Solid Conc (wt.%)	10	
Underflow		
Solids in Underflow (wt.%)	50	
Yield Stress (Pa)	5-10	
Required Residence Time (min)	1.0	
Overflow		
Turbidity	<100	
Particulate (ppm)	<200	
Flocculant		
Recommend Flocculant (type)	BASF Magnafloc 10	
Recommended Concentration (g/L)	0.1-0.3	
Recommended Dosage (g/t)	17-25	5-10
Parameters*		
Unit Area (m^2/tpd)	0.05	
Flux Rate (tph/m^2)	1.4	
Max Recommended Rise Rate (m/h)	28.2	
Recommended Sizing**		
Quantity	1	
Diameter/Depth (m)	15/4	
Suggested Unit (model)	LL-130	

Table 13-26 FLS Recommendations and Sizing Summary for Thickener Type E-Cat

PROCESS PARAMETERS	GRIND SIZE 75 µm	GRIND SIZE 106 µm
Solid Conc (wt.%)	10	
Underflow		
Solids in Underflow (wt.%)	50	
Required Residence Time (min)	1.0	
Overflow		
Turbidity	<100	
Particulate (ppm)	<200	
Flocculant		
Recommend Flocculant (type)	BASF Magnafloc 10	
Recommended Concentration (g/L)	0.1-0.3	
Recommended Dosage (g/t)	17-25	5-10
Parameters*		
Unit Area (m^2/tpd)	0.05	
Flux Rate (tpd/m^2)	34.4	
Maximum Rise Rate (m/h)	25.0	
Recommended Sizing**		
Quantity	1	
Diameter (m)	St. 12	

* The reported values were calculated based on the results obtained. For the detailed sizing of industrial equipment, FLS may use the conversion factor of its test apparatus

** The recommended sizing is based on the information provided by the client such as: flow of solids (dry basis) of: 195-t / h @ 33% -wt for both cases.

13.6.6 METALLURICAL RECOVERY ESTIMATE

The Almas Gold Project samples selected for metallurgical testing represented various ore types and lithologies within the different ore types and deposits. In addition, an overall composite representing the first three years of operation has been tested. Sufficient sample mass has been submitted for testing, so that tests were performed on a sufficient amount of material. The samples tested were not refractory and the mineralization was clean with no cyanicides present, except for low concentration of sulphur and iron, suggesting that there will be no obvious environmental concerns.

The ore was amenable to gravity separation followed by cyanide leaching. The suggested flowsheet included gravity separation followed by gravity concentrate intensive leaching and electrowinning. The gravity tailings were subjected to the cyanide leaching circuit following the pre-leach/ CIL circuit configuration. Subsequently, the gold would be recovered from the loaded carbon by elution and electrowinning.

The overall gold recovery is shown in Table 13-27, based on the metallurgical gold recoveries achieved in the testwork program. In addition, the estimated recoveries corrected for economic analyses purposes were derived by reducing the overall gold extraction by 1% to allow for potential gold losses. These losses can include the gravity concentrate intensive leach recovery of 98-99%, the carbon fines, soluble and refining losses. The average estimated overall recovery for the individual ore types/lithologies was estimated at 93- 95% leaving a residue assay of 0.04-0.06 g/t Au, showing minor trends for grind sensitivity. The sodium cyanide and lime consumptions were below 1 kg/t.

The average overall recovery for the 3 -Year Blend composite was estimated at 93% with residue assays of 0.09-0.11g/t Au. The fineness of grind did not affect the results. The sodium cyanide and lime consumptions were 0.2 and 1.5 kg/t respectively. The cyanide consumption was quite low in comparison with typical consumption used in the industry. This low consumption reflects the lack of cyanicides and other cyanide consuming species. Metal dissolution during cyanidation was low, and there were no obvious concerns with any deleterious elements.

The process design criteria include overall gold recovery of 92.5% at a grind of k_{80} = 75 microns.

Table 13-27 Gold Recovery Estimate

SAMPLE	GRIND SIZE K_{80} µm	AVE DIRECT HEAD GRADE Au g/t*	AVE CALC HEAD GRADE Au g/t**	OVERALL RECOVERY BY GRAVITY/CIL Au %	RESIDUE ASSAY Au g/t	REAGENT CONS		ESTIMATED RECOVERY for ECONOMIC ANALYSIS Au %
						NaCN kg/t	Lime kg/t	
AVERAGE INDIVIDUAL COMPOSITS	106 75	1.10	1.05	94.5 96.2	0.06 0.04	0.9 0.9	0.8 0.8	93.5 95.2
BLEND 3-YEAR COMPOSITE	106 75	1.28	1.91/1.31	93.8 94.1	0.11 0.09	0.2 0.2	1.5 1.5	92.8 93.1

* Average direct head grade determined by screen metallic assay
** Average calculated head grade from gravity-CIL/or Whole Ore leach metallurgical balance

13.7 DISCLAIMERS

This report has been prepared based on the results submitted by SGS Geosol and by "TESTWORK Process Development metallurgical laboratories" located in Brazil. The testwork program was not observed by the metallurgist responsible for this section preparation. As such the results were not independently verified but were found to be of a reasonably sound quality.

14 MINERAL RESOURCE ESTIMATES

The Almas Gold Project mineral resources contain three mineral deposits Paiol, Cata Funda, and Vira Saia, and one heap leach pad (from historical production of Paiol deposit). The resource estimate updates were performed for all three deposits plus historical leach pad materials.

A total of four models were constructed which are the basis for the mineral resource estimates discussed herein. For Paiol , Vira Saia and the Paiol Heap Leach pad. 3D updated models were constructed in the Gemcom-Surpac software platform (version 6.3) and mineral resources estimated in same platform by Farshid Ghazanfari, P.Geo. and QP for Aura Minerals.

For Cata Funda, the original 3D model from Rio Novo was used and mineral resources estimated in the Datamine software platform by Adam Wheeler, C.Eng and independent QP. All Models were peer reviewed by Micon International Limited in 2019.

Both QPs for this section of report believe the resource estimates meet industry standards and the general guidelines for NI 43-101 compliant resources for Measured, Indicated, and Inferred confidence levels as discussed herein.

14.1 PREVIOUS ESTIMATES

The summary of previous and historical resource estimates is discussed in section 6 of this report (Table 6.2).

The recent year’s resource models for the three mineral deposits were built originally by Rio Novo geologists along with GeoSim Resource Consultants. All three resource models were built in a similar fashion. In the first step, a structural model was completed by Rio Novo geologists based on core logging, surface geological mapping, and interpretation of cross-sections. Next, lithology and alteration in the database were used as a guide and in the final step, a gold zone model was built using the above models as guidance along with the assay data from drill holes.

The above models were initially created on paper cross sections, perpendicular to the main strike, on generally 25 m centers (in places 35 m or 50 m). This work was then digitized in ACAD and imported into Leapfrog. Using Leapfrog, three-dimensional solids were generated, then verified visually against the original data. The solids were then imported into Surpac software where the block models were constructed, and original resource estimates made.

The resource model for the Paiol Leach Pad is based on assays from reverse circulation and auger drilling. The original models by Rio Novo and GeoSim were completed during 2010 and 2011 and used in the initial Almas 43-101 reports. In 2012, the modeling was turned over to the Belo Horizonte office of RungePincockMinarco (RPM).

RPM initially completed an audit of the Rio Novo models and resource estimates for the Paiol and Cata Funda deposits, as well as the Paiol Leach Pad (PEA, April 2012). For the Vira Saia deposit, RPM completed the statistical analysis, block modeling, and mineral resource estimate. These resource estimates were used in the Preliminary Economic Assessment, published April, 2012 (PEA, April 2012).

RPM produced an updated resource model for Vira Saia in mid-2012 based on new information acquired through infill drilling. In September of 2012, RPM in Denver completed an updated model and resource estimate for the Paiol deposit, based on the latest costs and mining parameters.

14.2 TOPOGRAPHIC AND SAPROLITE SURFACES

For the Cata Funda, and Vira Saia deposits 1 m resolution topographical surveys and for Paiol 10 m topo surveys are available. The latter file includes larger areas around the Paiol existing pit and historical heap leach pad. This data was checked against the drill hole collar surveys and showed good adherence to the topographical surface. The Rio Novo original topographical surfaces in the data room were used to represent topography during the modeling process.

All three deposits are covered by a saprolite layer and a horizon of weathered rocks with variable thicknesses. In Paiol, due to previous mining activities, most of saprolite layer was removed in the central and northern portions of ore the deposit. In 2011-2012, Rio Novo generated a set of surfaces for the bottom parts of the saprolite layer and weathered rock layer horizons which were constructed based on lithological descriptions of historical and recent bore holes for each deposit. These surfaces were used to code different rock types in for resource estimation purposes accordingly.

14.3 ALMAS PROJECT DATABASE

The drilling data from the Paiol, Cata Funda and Vira Saia deposits, included in this report, come from two Microsoft Access databases made available to Aura by Rio Novo.

In one of the databases ("DB_Historical") is the data generated between 1987 and 2008, by different companies including Metago, Vale and Santa Elina. In the second database ("DB_Rio_Novo"), is data generated by Rio Novo between 2010 and 2012. The two databases include information about diamond drilling, reverse circulation drilling, metallurgical holes and auger holes, which can be filtered by different attributes including type of drilling, year, deposit and company. In the tables 10-1 to 10-4 of (section 10) of this report a summary of the drilling and sampling all drilling campaigns is provided.

The current Almas's drillhole database consists of 1,837 drillholes (160,744.61 m) drilled from 1987 to 2012 at the targets Paiol, Vira Saia and Cata Funda, which includes 830 (118,408.73 m) core holes, 14 holes to collect material for metallurgical test work, 819 reverse circulation holes and 174 auger holes. The total assayed and sampled is 146,292.68 m.

All coordinates used in the database are in the Universal Transverse Mercator (UTM) coordinate system, zone 23S, South America 1969. Validations of hole coordinates, as well as downhole survey, lithological intervals, and sampling, were performed by companies contracted in previous years, as described in section 12.

The database DB_Rio_Novo was migrated to an SQL Server database in 2020 by the Aura team. All the data have been validated and imported, including assay certificates that were imported, to keep the metadata available and to ensure authenticity of the information provided by the analysis laboratory.

The laboratory responsible for the samples' preparation and analysis was SGS Geosol Laboratório Ltda, a Brazilian company that provides geochemical analysis services of soils, rocks and ores, concentrates and metallurgical tests. The methods of analysis used were Au_FAA505_ppb and Au_FAASCR_ppb, whose detection limits are <5 ppb and <10 ppb, respectively. For results below the detection limit, Aura uses, as a standard, the value reported by the laboratory divided by 2 (2.5 ppb for <5 ppb and 5 ppb for <10 ppb). Unlike the criterion used by Aura, the values adopted in the Rio Novo database were 4.99 ppb and 9.99 ppb, respectively.

The DB_Historical database was not migrated to the SQL database. No validation of the assay results was performed, comparing the assay results to the certificates issued by the laboratory. Within the verified certificates, it was possible to observe integrity in the gold grades available by the laboratory and the standard adopted for results below the detection limit. The results below the detection limit, <0.01 ppm, were replaced by the value 0 in the database.

Two points of attention to the data provided in the Assay table of DB_Historical, are:

- The original intervals of the samples were split into more than one interval, so that it corresponds to only one geological unit. In this situation, the two or more intervals created received the same gold result as the original interval.
- The sample code, used to identify the material sent to the laboratory and returning in the assay certificates, is not filled out in the Assay table. Comparing the results of some holes, available in the assay table, with the laboratory results, we can observe the existence of integrity in the results loaded in the database.

The two situations reported above need to be addressed and the data updated in the database to ensure reliability, traceability of information.

The April, 2012 PEA report mentioned 21 twin holes drilled during 2008 to validate historical drill holes. The report indicated the total bias (-12 % in historical holes) as an issue inherent to shear-hosted gold deposits. One of the recommendations is to verify the impact this bias on the resources by previous authors.

In summary, the database was considered sufficiently robust to support resource estimation at a feasibility level

14.4 GEOLOGICAL AND DOMAIN MODELING

In all previous estimates, a nominal cut-off that was deemed to be economic was not introduced into 3D modelling of the deposits. This is a key, important component in resource modelling before applying any resource and reserve cut-off grades. Rio Novo geologists created a broad scale litho-alteration model in Leapfrog software which was not adjusted for any specific grade threshold. Although it was claimed in a previous report that the model was adjusted based on shear zones. The outcome models are not resembling any shear structures. They look like a computer-generated model without applying geology. This leads to another problem regarding exploratory data analysis , where the broad litho-alteration models produced too many assay points (low grade and no grade) to analyze which often need to be ignored for this type of analysis.

Consulting geologists who were hired to do technical reports including Geosim (2011-2012) and RPM (2012-2016), did not try to create a grade shell model using a nominal cut-off grade, probably due to a lack of time in order to rush and prepare a model for technical reports.

For all of the above reasons, the authors of this report decided to abandon the previous 3D models and approach at least for the Paiol and Vira Saia deposits. A nominal cut of grade 0.3 g/t was introduced to constrain deposit models within structural and altered corridors for each respective deposit. The Rio Novo original Cata Funda 3D model more or less is following a 0.3 g/t Au assay cutoff grade and, therefore, was used for estimation purposes.

All updated 3D models were clipped to current surface topography to be used in resource estimations.

14.4.1 PAIOL

The Paiol deposit is a steeply dipping, narrow shear-hosted vein deposit with an outer envelope containing splays and discontinuous lenses of gold mineralization. Veins strike at an azimuth of 015° to 025° and typically dip 60° northwest.

Toward the center of deposit and existing pit, several shear zones join and provide a broader steep plunging ore body. Towards the north several shearing corridors tend to distance from each other, having different strikes and provide a larger footprint for the deposit (Figure 14-1) . Most of the past production came from the central and northern part of ore body.

A new 3D model was made at Paiol, considering all these changes in the nature of shearing, plus lithology and alteration logging that was done by previous geologists from Rio Novo. In addition, a nominal cut-grade of 0.3 g/t was introduced to digitize mineralization polygons on 25 m cross-section intervals. Rio Novo geologists created a broad lithological/alteration model which was bounded by calcite-albite-adularia schist (CAAX) and calcite-chlorite-quartz schist (CCQX) units. This model represents the main corridor for shear zones and gold mineralization and was used in the current 3D modeling as a general guide to constrain shear zones. However, some peripheral shear zones were identified and modeled outside this lithological/alteration unit (Figure 14-3).

The Paiol 3D model was constructed in the Gemcom-Surpac software platform using all informing drill holes including RC and diamond drill holes. The model was extended to the pre-mine surface with the goal to use all the samples within the mineralized domain before any production. The pre-mine surface was defined using the drill hole collars. The 3D model was then clipped against the existing Paiol topographic surface

Figure 14-1 shows the general view of Paiol new updated model and figure 14-2 shows a representative cross-section.

Figure 14-1 Paiol 3D Model and Drillhole Data

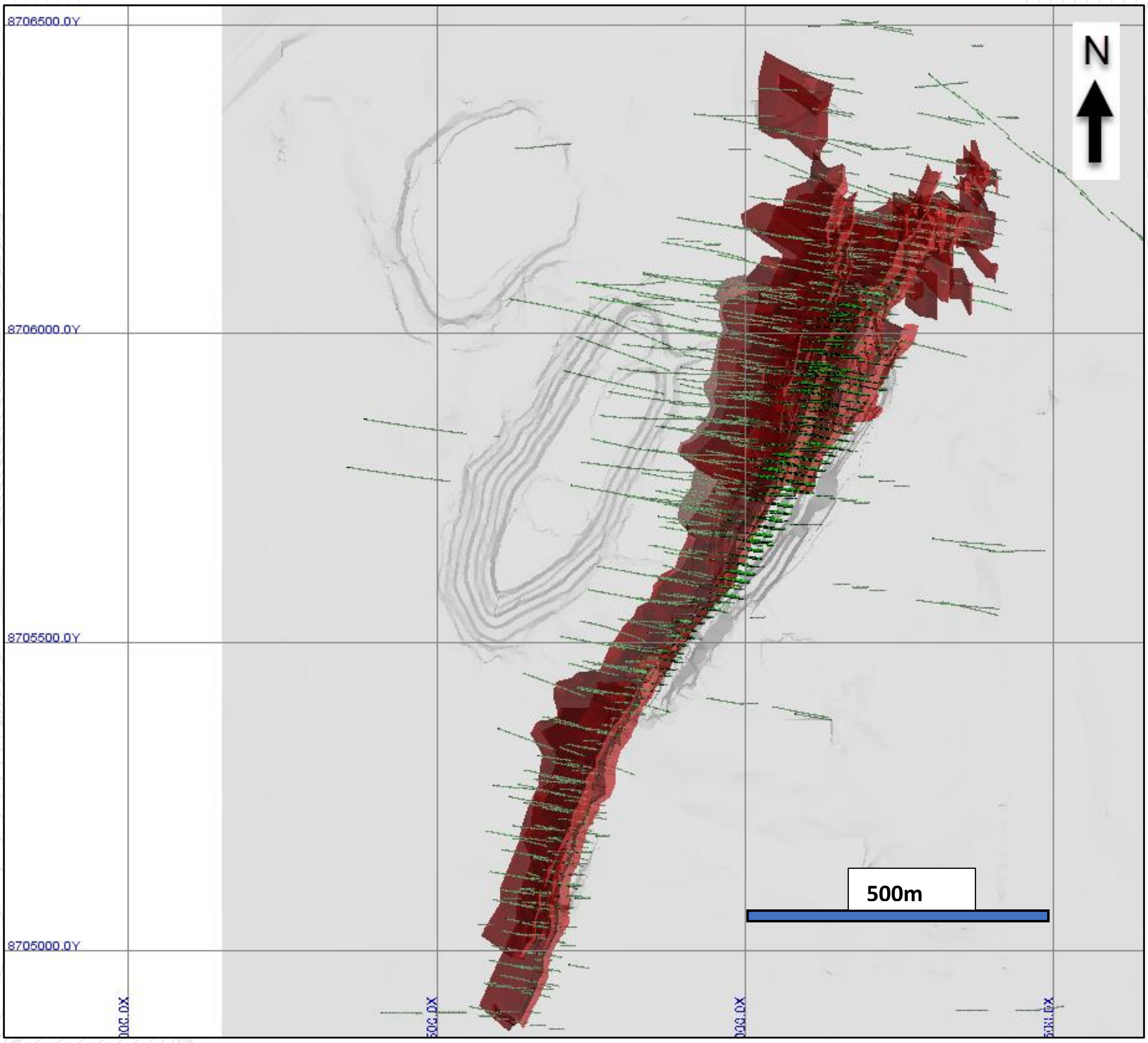


Figure 14-2 A Representative Cross-section in the center of the Paiol Deposit Showing all Assays Above a 0.3 g/t Cut-off Grade vs. New Updated 3D Model

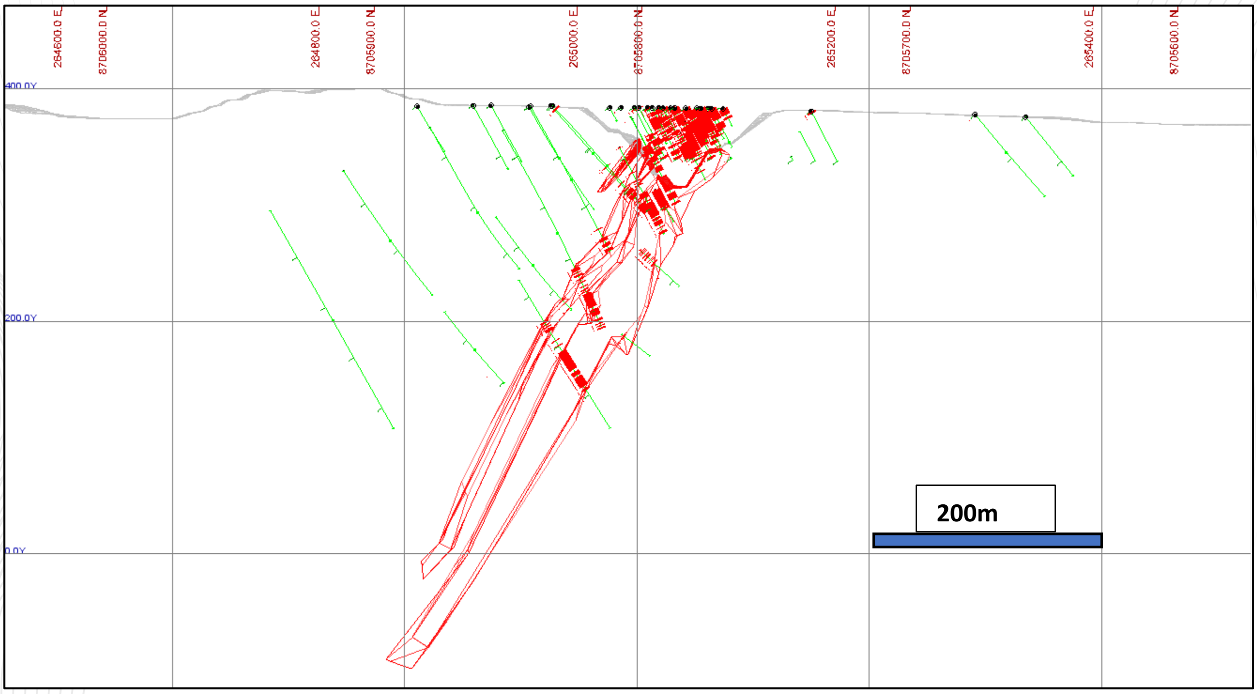
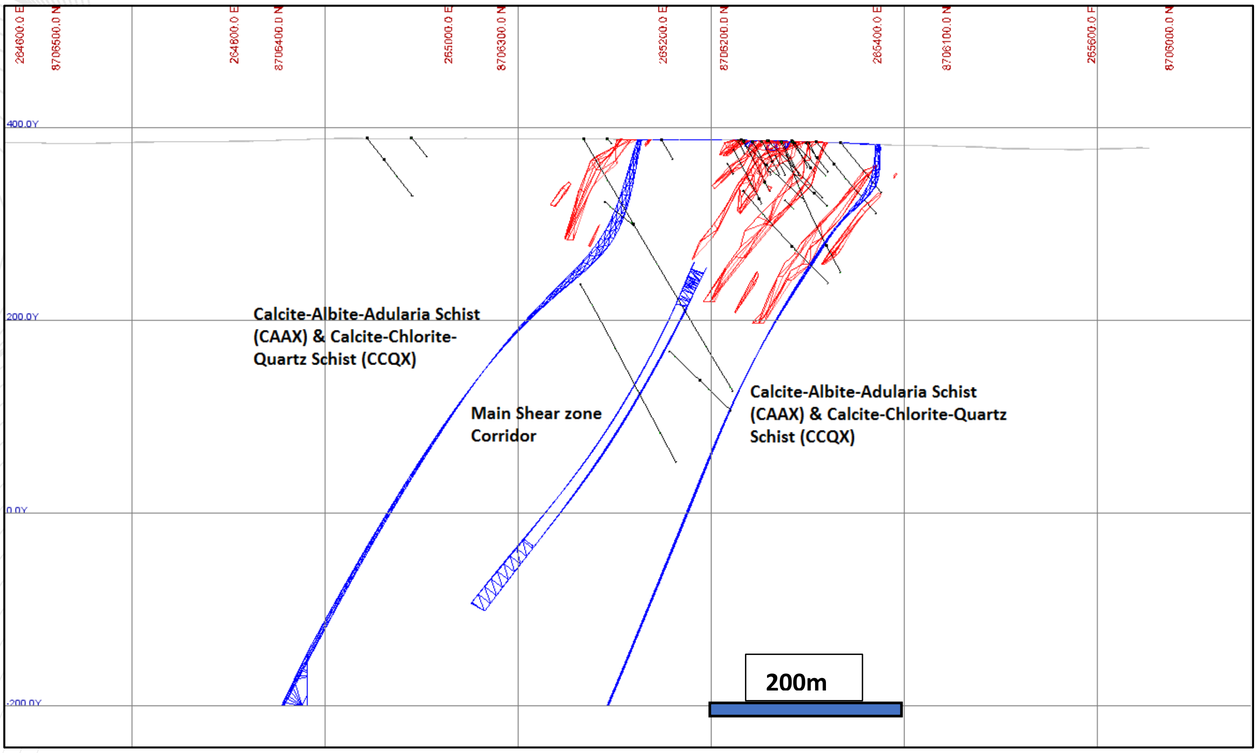


Figure 14-3 A Cross-section, New Updated 3D Model vs. Alteration-Lithological Corridor for Shearing and Gold Mineralization (North End of Paiol Deposit)



14.4.2 VIRA SAIA

Vira Saia was discovered in 2011 by Rio Novo geologists and drilled out during 2011 and early 2012. The deposit is somewhat different from other discoveries in the region in that it is hosted along a structural zone in intrusive rocks (granodiorite) as opposed to other deposits in the region hosted in meta-volcanic and metasedimentary rocks. The original Vira Saia geologic model was completed by Rio Novo geologists in 2012 and was used in previous estimates by RPM. The model was constructed by using two main lithology-alteration units of quartz-sericite schist (ultramylonite) and sheared granodiorite or mylonitized granodiorite (see section 7 of this report). The 3D model was constrained using orientation of shear zones and lithological units.

The geological model of the Vira Saia deposit contains one main north-trending structure, the Vira Saia fault. This main fault is offset by three modelled northeast trending faults.

The Vira Saia model differentiates five rock or lithological units. Near surface are two units: (1) a weathered zone, and (2) a saprolite zone. For bedrock, there are three units modelled: (3) granodiorite or related intermediate intrusive rock, (4) mafic dike, (5) other meta-volcanic or meta-sedimentary rock. The model then breaks out three alteration/mineralized zones: (1) quartz-sericite schist, (2) mylonitized granodiorite, and (3) proto-mylonitized granodiorite. Where the mafic dike cuts the zone, it is excluded from the resource model.

The gold zone, as created in three dimensions, typically exhibits an elongated, lenticular shape. The main axis strikes about N45W following the general trend of the Vira Saia Fault. The dip varies from about 55 to 85 degrees to the southwest, depending on location within the deposit. The zone ranges from about 30 m thick at the maximum, to a pinch-out thickness of less than a meter.

Typical sections for the geologic model are demonstrated in Section 7 of this report.

The Rio Novo drilling database for Vira Saia showed that the shear zone extended to the northwest with same bearing and mineralization continues within favorable granodioritic rocks with the same inclination (Figure 14-4). In the recent updated model, the extension of shear zones toward the northwest is modelled using 0.3 g/t cutoff grade and added to the Vira Saia models. In addition, a hanging wall (HW) splay of the shear zone, wherever it meets the requirements of the wireframing cut-off (0.3 g/t) was added to the model (Figure 14-5). The HW shear is completely hosted within the mylonite unit (GDM).

Figure 14-4 Vira Saia 3D model, Drillhole Data and Extension Shear Hosted Mineralization (Mylonite) Toward the NW

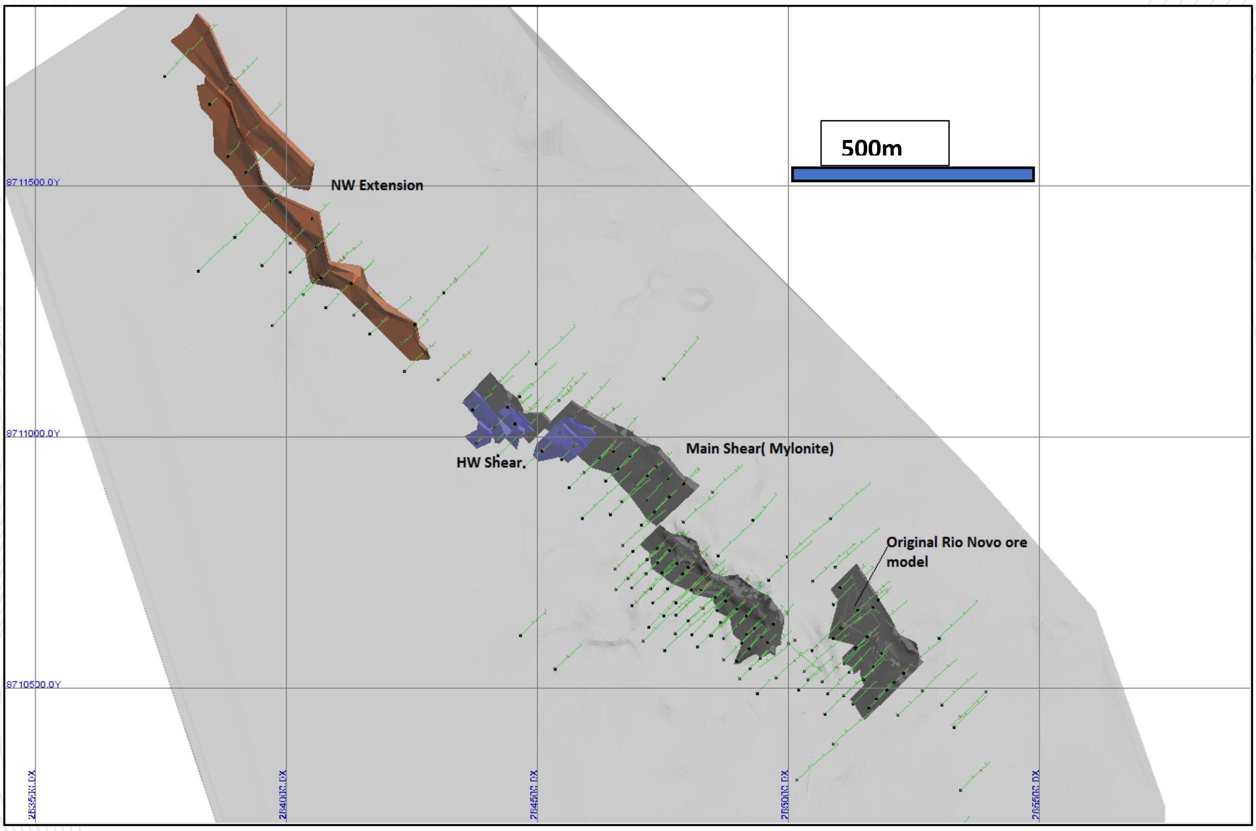
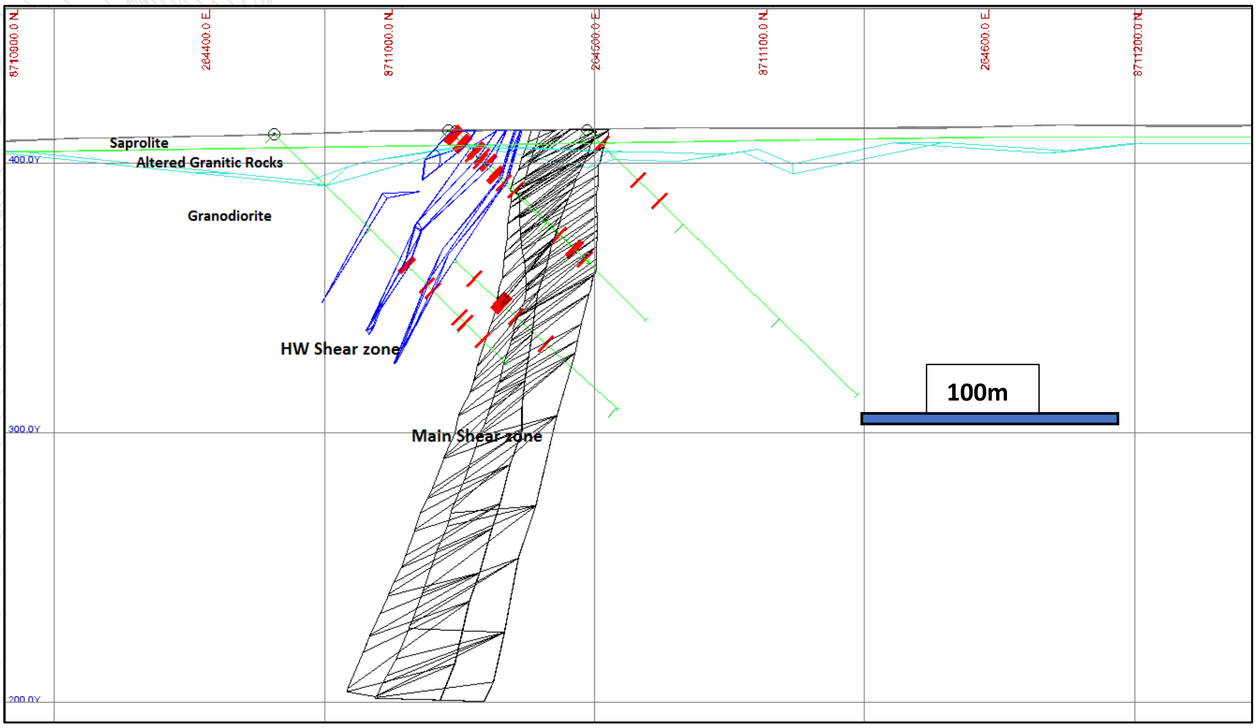


Figure 14-5 A Representative Cross-section in the Center of the Vira Saia Deposit Showing all Assays Above a 0.3 g/t Cut-off Grade vs. Main Shear Zone Updated 3D Model of HW Shear Structure



14.4.3 CATA FUNDA

Cata Funda was discovered around 1988 and was drilled out during various campaigns between 1996 and 2011 by Vale and Rio Novo. The Cata Funda deposit is similar to Paiol in that it is hosted in metavolcanics and metasedimentary rocks. The main control on mineralization is a major northwest-trending shear zone. Quartz veins and silicified breccias have formed along this shear zone and host the main gold mineralization.

The Cata Funda geological model was completed by Rio Novo geologists in 2011 and the same model was used for the current resource block model prepared by A.Wheeler, C.Eng.

The Cata Funda deposit is comprised of metavolcanic and metasedimentary rocks which have been hydrothermally altered to various chlorite-sericite-quartz-carbonate schists. For geological logging and the basic geological model, five individual schist units were broken out. Two units, the sericite-chlorite-ankerite schist (SCDX) and sericite-ankerite-quartz schist (SDQX) contain most of the gold mineralization. Near surface, the schists have been overprinted by various degrees of weathering, namely soil, saprolite, and semi-weathered bedrock or “saprock”.

Mafic Dykes are present in the Cata Funda deposit with variable thicknesses and they appear usually in hanging wall (HW) of mineralized units. They are not modeled, and they do not interfere with current ore model for Cata Funda.

For modelling purposes, three units were defined in the upper weathered region: soil, saprolite, and weathered rock. A fourth unit, termed bedrock, comprises all the un-weathered bedrock schists.

The gold zone, as created in three dimensions, typically exhibits an elongated, lenticular shape. The main axis strikes about N45W following the general trend of the Cata Funda Fault. The dip averaging about 55 degrees to the southwest. The zone averages about 20 m in thickness.

The Cata Funda mineralization model in general follows the sericite-chlorite-ankerite schist and is constrained within a 0.3 g/t Au grade shell. All blocks were restricted to this single domain and a hard boundary was imposed at the grade shell limit.

Historical Cata Funda drilling was a combination of RC and DDH in the database but, for the purpose of modelling, both used only the DDH dataset for estimation of mineral resources.

The Cata Funda deposit is open toward the northwest but it was not modelled there, partly because the extension of deposit towards the northwest is underlying the town of Almas.

Figure 14-6 to Figure 14-10 shows a set of plans, cross-sections and long sections for the Cata Funda area showing the entire drilled area and deposit model area.

Figure 14-6 Cata Funda Overall Plan of Drillhole Data

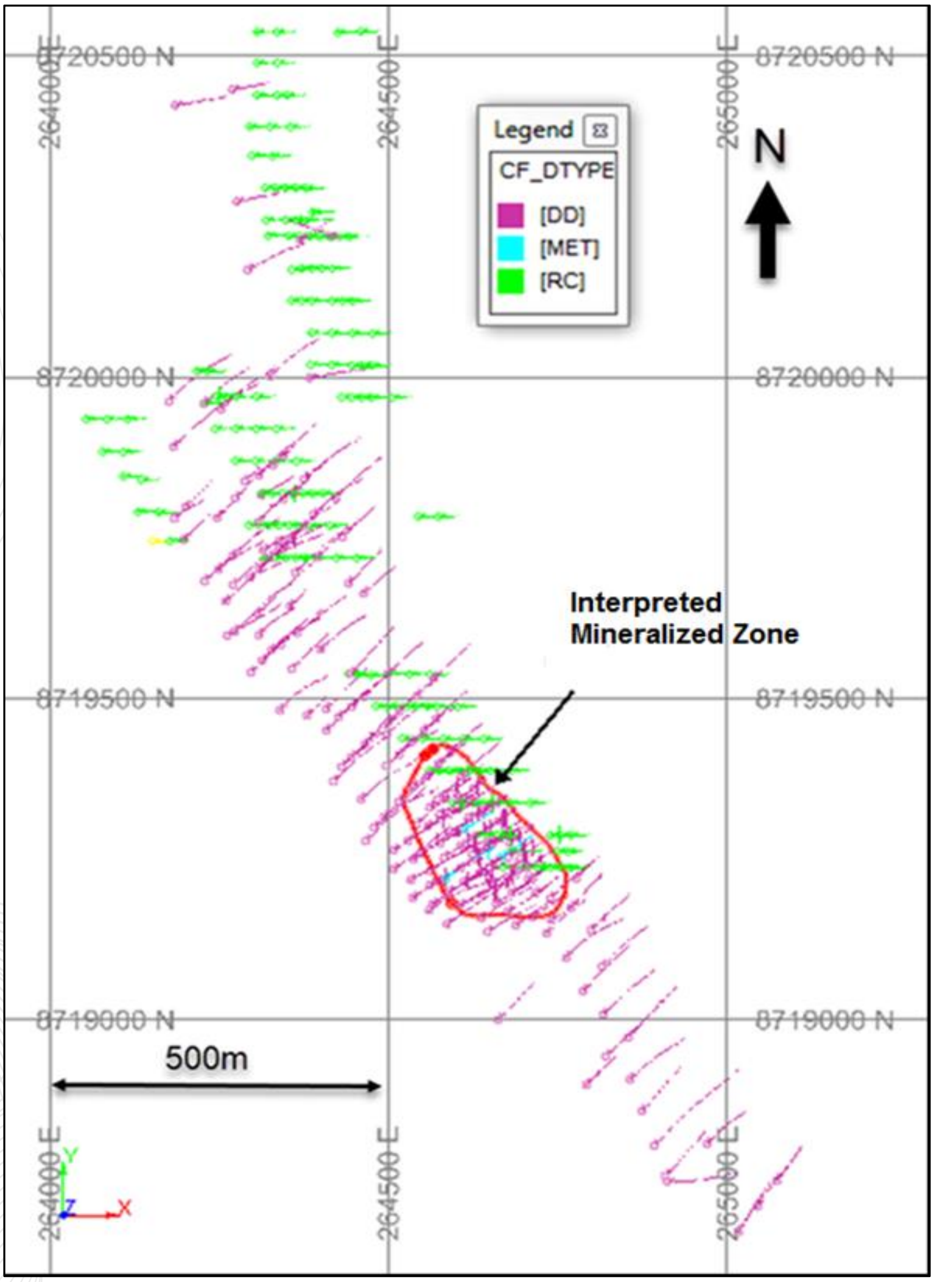


Figure 14-7 Plan of Drillhole Data in the Cata Funda Deposit Area

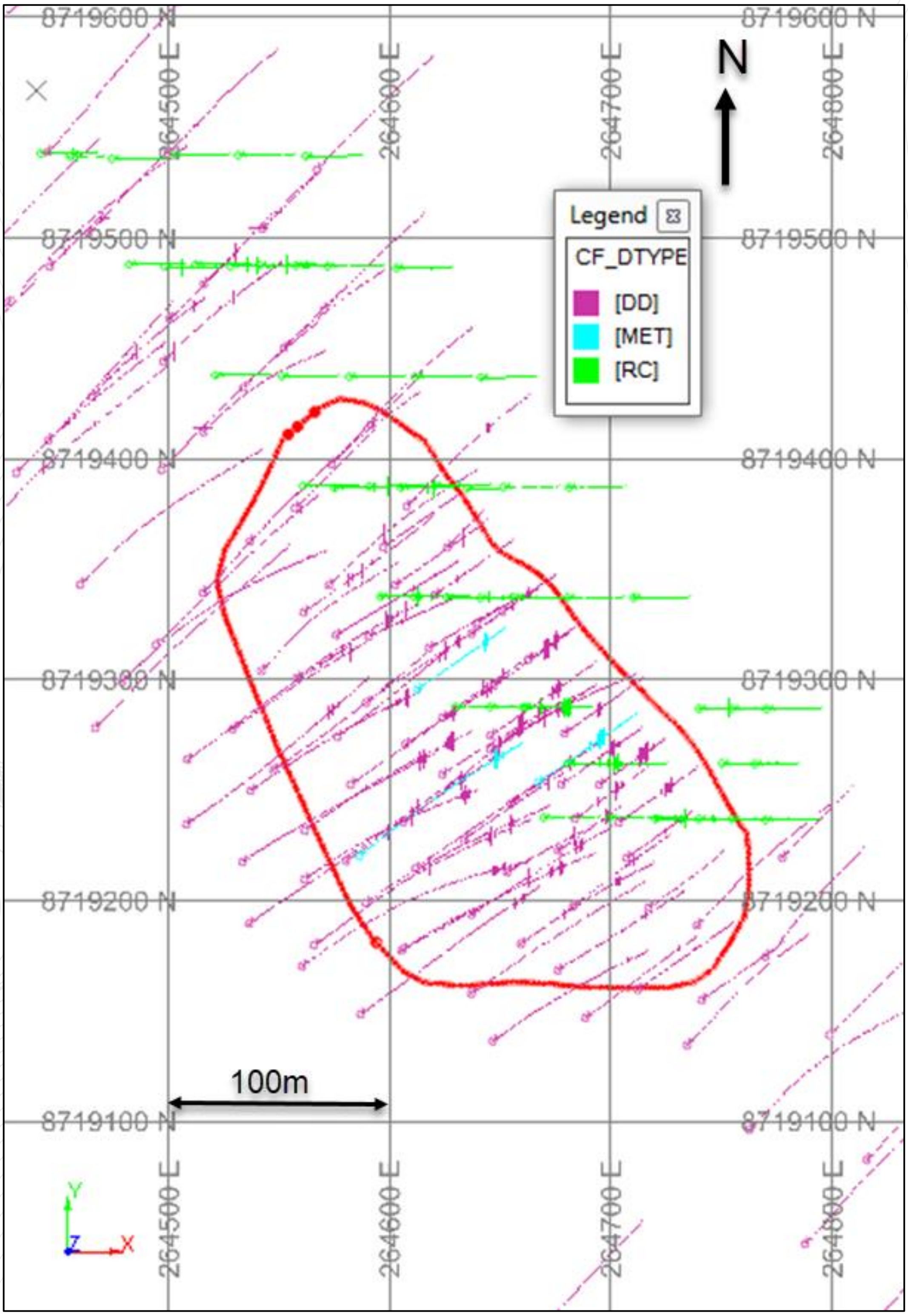


Figure 14-8 Overall Long-Section of Drillhole Data

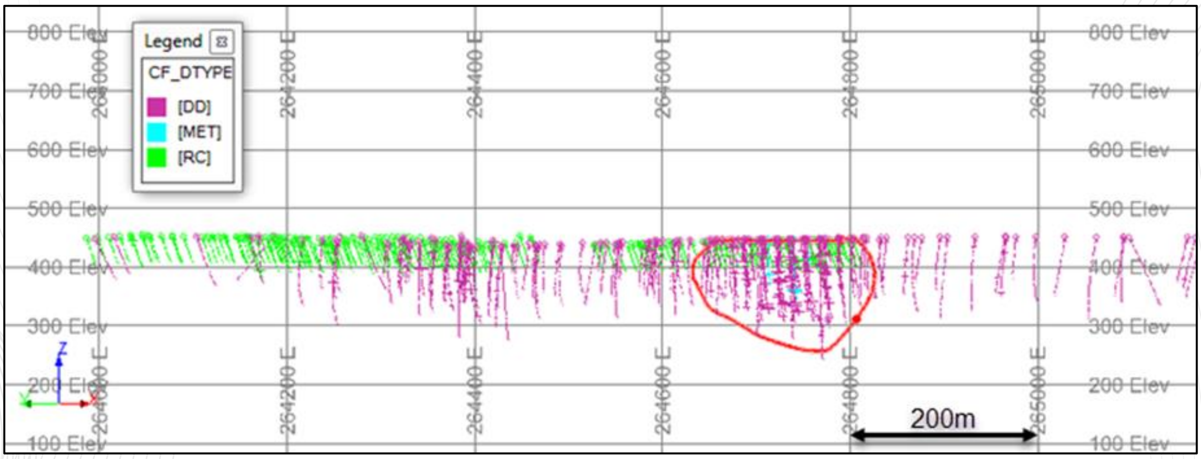


Figure 14-9 Long-Section of Drillhole Data in the Cata Funda Deposit Area

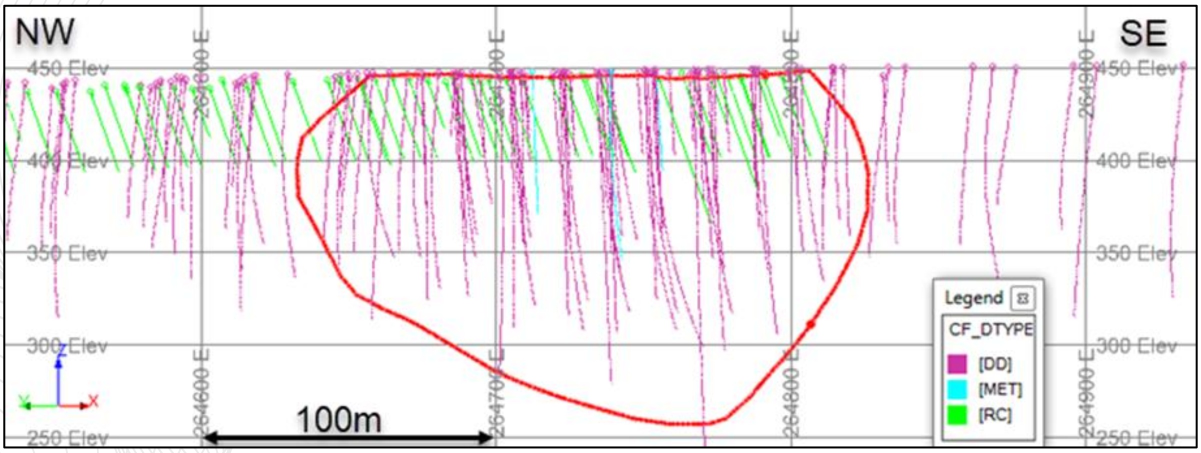
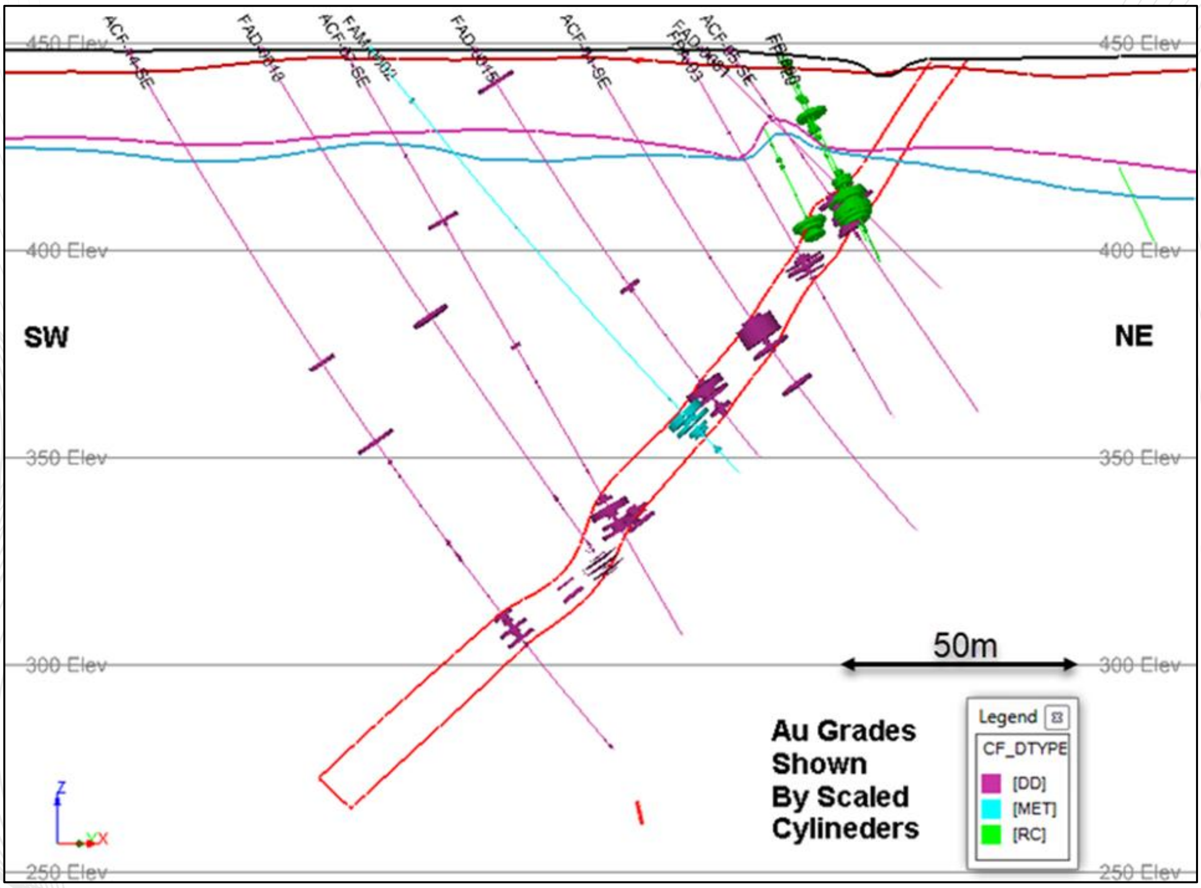


Figure 14-10 Representative Drillhole Cross-Section in the Cata Funda Deposit



14.5 DENSITY DEFAULT MODEL

Details of density measurements, sampling and testing procedures are discussed in section 11.2 of this report. Here the density values which were selected and assigned to the 3D model to convert volume to tonnes will be discussed.

14.5.1 PAIOL

The Paiol density database is representing different lithologies, mineralization types, and degree of weathering. The database has 2,382 density measurements for fresh rock on drill core, including 940 mineralized samples and 1,514 for waste materials. In addition, the density database includes 655 samples for weathered, soil and saprolite samples . The water immersion method was used for weathered, saprolite and soil samples, and the porous samples were sealed in plastic film. The data were analyzed statistically by lithology and outliers removed.

The Paiol model uses three density values, based on rock (lithology) type. All fresh rock below the weathered zone, was assigned an average density of 2.78. Only measurements in mineralization were used for fresh rock. Density values related to the waste were excluded, then the value was assigned as default to the 3D model. All rocks in the weathered zone, located above the fresh rock and below the saprolite zone, were given an average density value of 2.35 tonnes per cubic meter. This value was derived from dry density measurements from total 237 density samples taken from this zone. Note that since this is effectively a transition zone, there is a large range in density values.

In the saprolite zone, essentially all rock between surface and above the weathered zone, an average density of 1.54 tonnes per cubic meter was assigned. This value was derived based on the average of 392 density samples taken from the saprolite and soils. Table 14-1 summarizes the density values used in the Paiol model.

Table 14-1 Average of Density for Different Type of Rocks (Paiol)

LITHOLOGY	DENSITY t/m³
Saprolite & Soil	1.54
Weathered	2.35
Fresh Rock	2.78

14.5.2 VIRA SAIA

Rio Novo carried out 2,586 density measurements on drill core from Cata Funda representing different lithologies, mineralization types, and degrees of weathering. The water immersion method was used for weathered, saprolite and soil samples, and porous samples were sealed in plastic film. The data were analyzed statistically by lithology and outliers removed.

The Vira Saia model uses three density values, based on rock (lithology) type. All fresh rock below the weathered zone, was assigned an average density of 2.72. All rock in the weathered zone, located above the fresh rock and below the saprolite zone, was given a density value of 2.12 tonnes per cubic meter. The assigned density value was based on 148 density samples taken from this zone. Note that since this is effectively a transition zone, there is a large range in density values.

In the saprolite zone, essentially all rock between surface and above the weathered zone, an average density of 1.78 tonnes per cubic meter was assigned. This value was derived based on the average of 294 density samples taken from the saprolite zone. Table 14-2 summarizes the density values used in the Vira Saia model.

Table 14-2 Average of Density for Different Type of Rocks (Vira Saia)

LITHOLOGY	DENSITY t/m³
Saprolite & Soil	1.78
Weathered	2.12
Fresh Rock	2.72

14.5.3 CATA FUNDA

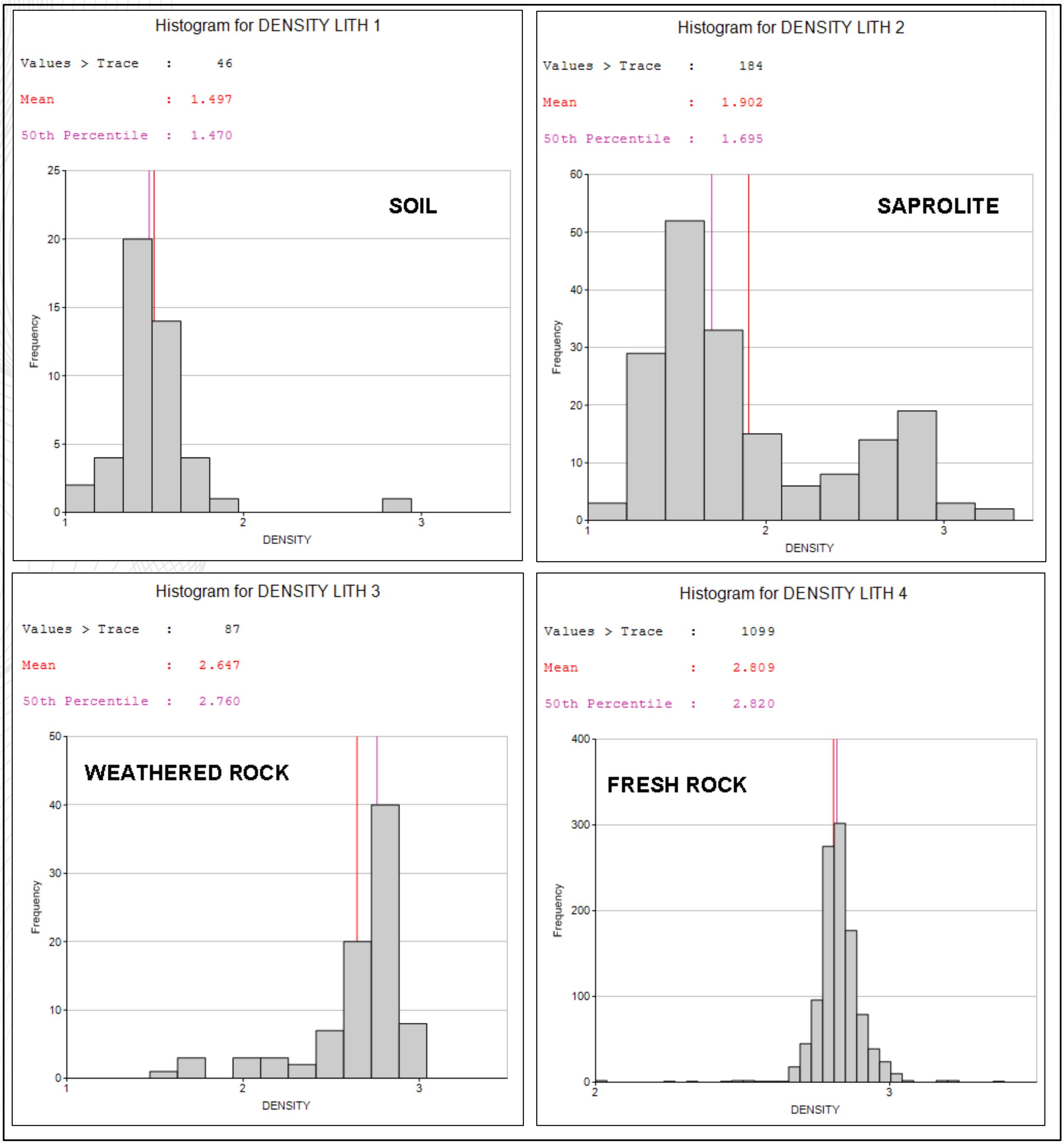
Rio Novo carried out 1,568 density measurements on drill core from Cata Funda representing different lithologies, mineralization types, and degree of weathering. The water immersion method was used for weathered, saprolite and soil samples, and porous samples were sealed in plastic film. The data were analyzed statistically by lithology and outliers removed. In the bedrock samples, no significant differences were seen between different rock types and alteration types. The median of the density dataset was used for converting volume to tonnes. A median of 2.82 g/cm^3 was used for all fresh rock. Weathered rock was assigned a specific gravity of 2.76 g/cm^3. For Soil a median value of 1.47 and for saprolite a median specific gravity of 1.70 g/cm^3 was assigned.

Histograms of density measurements, split by main rock type, are shown in Figure 14-11. The median lithology density values were used to set density values in the corresponding blocks, as summarized in Table 14-3.

Table 14-3 Median of Density for Different Type of Rocks (Cata Funda)

LITHOLOGY	DENSITY t/m^3
Soil	1.47
Saprolite	1.70
Weathered	2.76
Fresh Rock	2.82

Figure 14-11 Density Histograms



14.6 EXPLORATORY DATA ANALYSIS (PAIOL)

The Paiol database contains sufficient data to support a well-informed mineral resource estimate. In order to have a point data set to do statistical analysis, drill holes intersected against mineralized wireframes were selected and all assays extracted with their corresponding lengths. For this analysis only intersecting diamond drilling holes (DDH) were used. Table 14-4 shows a summary of selected intersecting holes.

Table 14-4 Summary of Selected Drillhole Data (Paiol)

TYPE	LITHOLOGY	HOLES	LENGTH (m)	AVG. LENGTH (m)	SAMPLES
Diamond Drilling	Saprolite	276	5,554.81	20.13	4,429
	Weathered	259	8,265.47	31.91	2,665
	Fresh Rock	190	18,870.55	99.32	18,116
	Subtotal	284	32,690.83	115.11	25,210
Metallurgical Core	Saprolite	8	208.75	26.09	206
	Weathered	8	97.55	12.19	98
	Fresh Rock	8	949.90	118.73	957
	Subtotal	8	1,256.20	157.02	1,261
Total Drilling		**292**	**33,947.03**	**116.25**	**26,471**

The database contains 18,927 samples with Au values equal to or greater than zero. Sample lengths are quite variable, from one centimeter to tens of meters. Short sample intervals, which can represent a mineralized vein, are geologically important and they are more probable than high values in longer samples.

Figure 14-12 shows a log histogram of all gold assays inside the Paiol mineralized wireframe.

Figure 14-12 Log Histogram for gold assay values (Paiol)

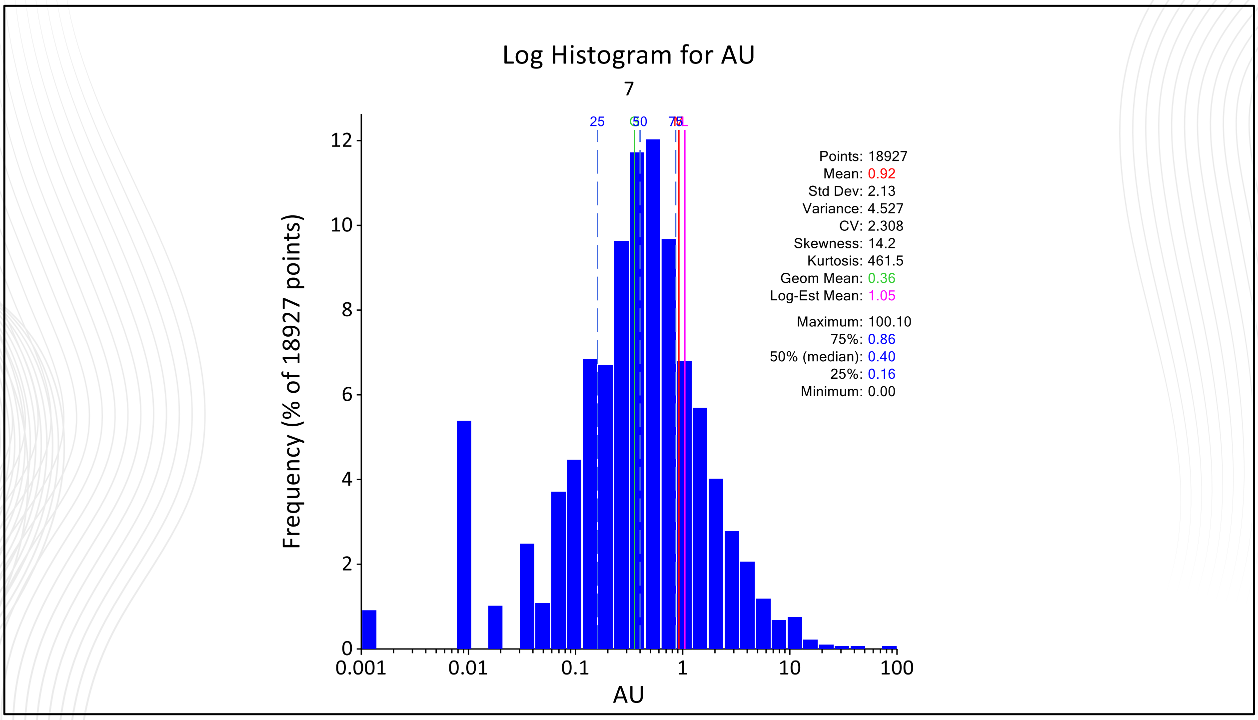


Figure 14-13 shows log probability plot for the same dataset and suggests that the use of a capping grade, on raw data, as high as 35.0 g/t Au could be justified.

The Paiol wireframes have been constructed using both diamond and RC drilling. Samples within the envelope were processed into 2.5 m composites and capped, after compositing, at 13.0 g/t Au. Figure 14-14 and Figure 14-15 show the histogram, univariate statistics and Log of probability plot respectively for the Paiol composites before capping .

The Log probability plot (figure 14-15) shows that capping at 13.0 g/t Au is appropriate for the Paiol dataset. The capping value corresponds to the 99.5 percentile or 0.2% of data (10 samples).

The composites file contains samples with lengths down to 0.25 m which is 10% of composite length. The author believes that short composite intervals are important part of the dataset and help to have better geostatistical analysis.

Figure 14-13 Log Probability Plot for Gold Assay Values (Paiol)

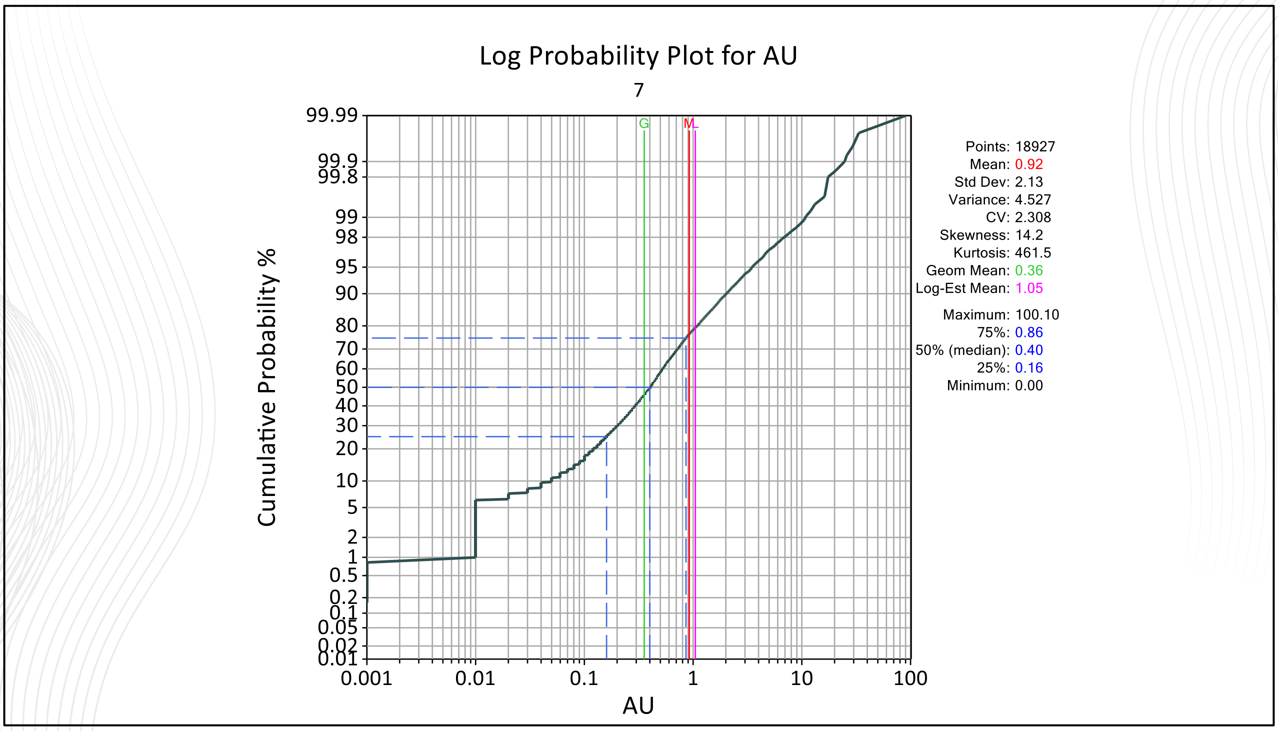


Figure 14-14 Log Histogram for Composited Gold Values(Paiol)

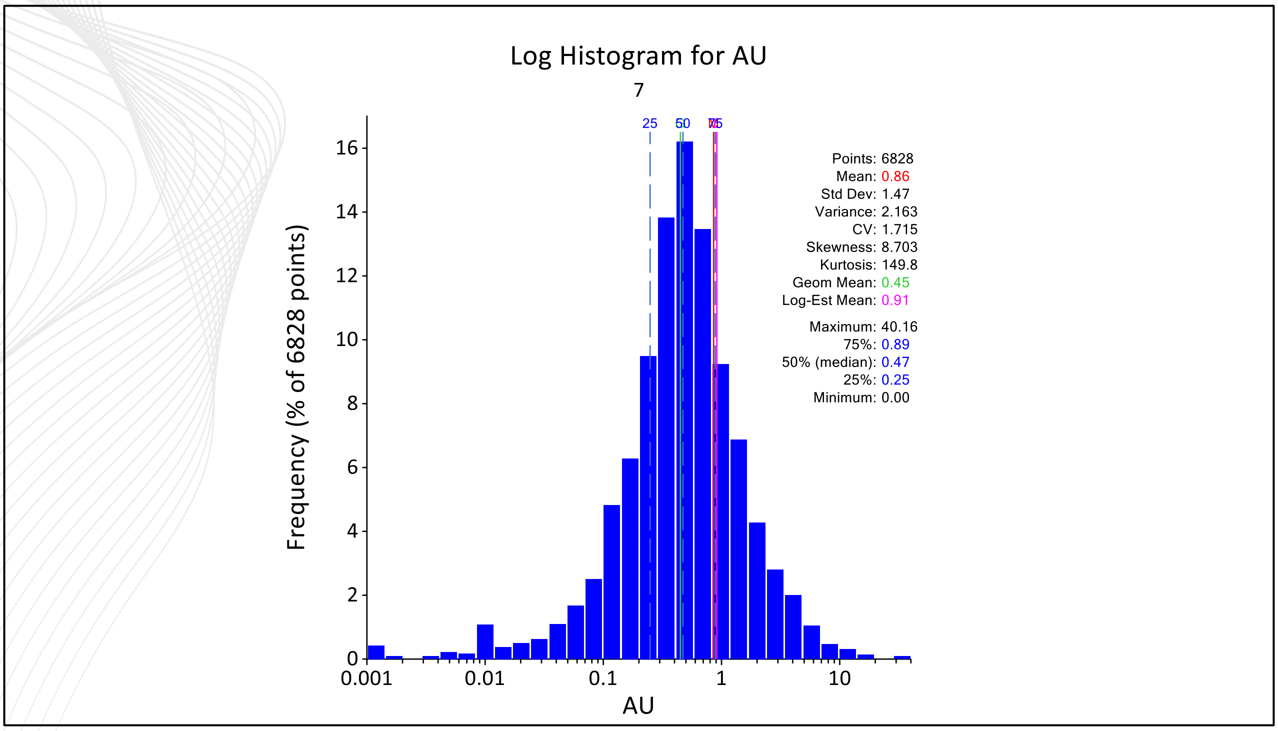
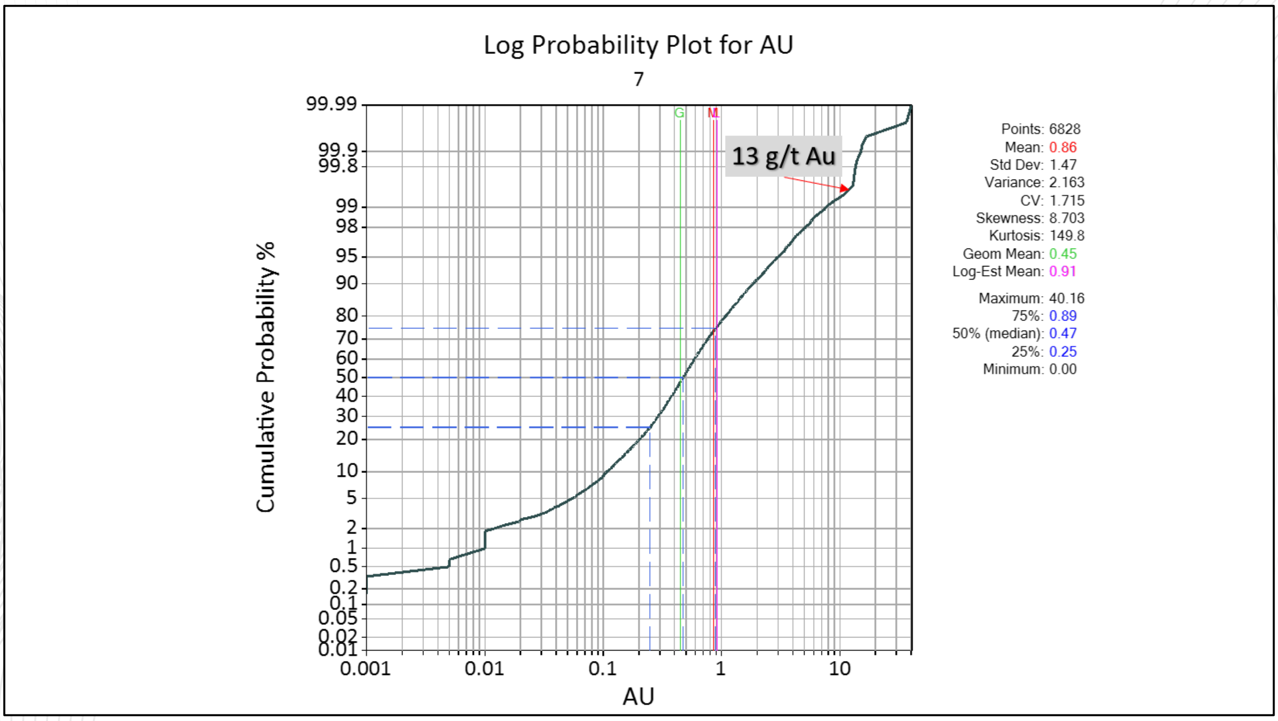


Figure 14-15 Log Probability Plot for Composited Gold Values (Paiol)



For the upper portion of the Paiol deposit, where most of the RC drilling is located, a statistical comparison was made to check for bias between the RC and diamond drilling (DDH) methods and sampling. The results of this comparison were discussed in section 12.5.2.

14.7 EXPLORATORY DATA ANALYSIS (VIRA SAIA)

The Vira Saia database contains sufficient data to support a well-informed mineral resource estimate. In order to have a point data set to do the statistical analysis, drill holes intersected against mineralization wireframes were selected and all assays extracted with their corresponding lengths. For this analysis only DDH drilled by Rio Novo were used. None of the Vale historical RC holes were used in the statistical analysis and consequently in the resource estimation. A summary of the selected drill hole data is shown in Table 14-5.

Table 14-5 Summary of Selected Drillhole Data (Vira Saia)

TYPE	LITHOLOGY	HOLES	LENGTH (m)	AVG. LENGTH (m)	SAMPLES
Diamond Drilling	Saprolite	122	825.9	6.77	749
	Weathered	116	835.35	7.2	787
	Fresh Rock	134	16,239.50	121.19	15,823
	Subtotal	**137**	**17,900.75**	**130.66**	**17,359**
Metallurgical Core	Saprolite	2	10.1	5.05	8
	Weathered	3	14.8	4.93	10
	Fresh Rock	3	346.8	115.6	323
	Subtotal	**3**	**371.7**	**123.9**	**341**
Total Drilling		**140**	**18,272.45**	**130.51**	**17,700**

Figure 14-16 shows a log histogram of all gold assays inside the Vira Saia mineralization wireframe.

Figure 14-16 Log Histogram for Gold Assay Values (Vira Saia)

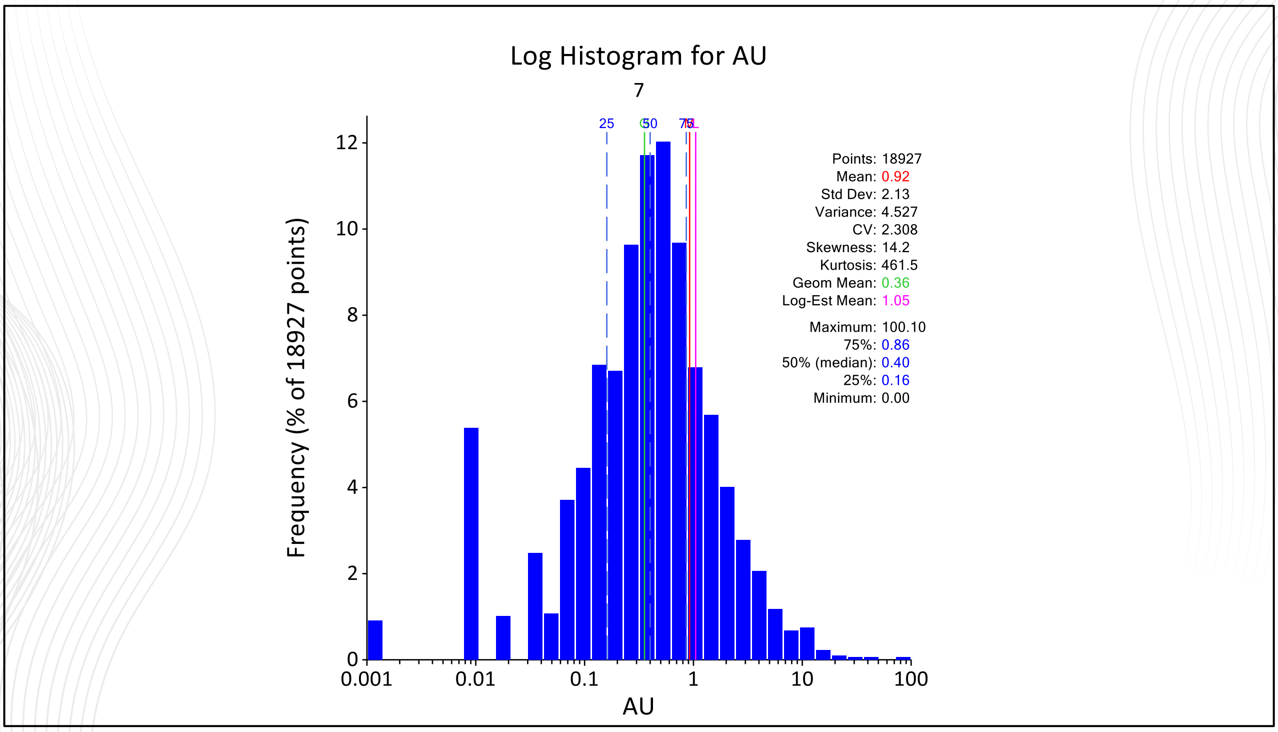


Figure 14-17 shows a log probability plot for the same dataset and suggests that the use of a capping grade, on raw data, as high as 28.0 g/t Au could be justified. This is interpreted from the end of the straight portion of the curve at high grades. However, the coefficient of variation is relatively high and much lower capping is needed to reduce the influence of outliers.

Samples within the envelope were processed into 2.0 m composites and capped, after compositing, at 10.0 g/t Au. Figure 14-18 and Figure 14-19 show the histogram, univariate statistics and log probability plot respectively for the Vira Saia composites before capping.

The composites file contains samples with lengths down to 0.20 m which is 10% of composite length. The author believes that short composite intervals are an important part of the dataset and help to have better geostatistical analysis.

Figure 14-17 Log probability Plot for Gold Assay Values (Vira Saia)

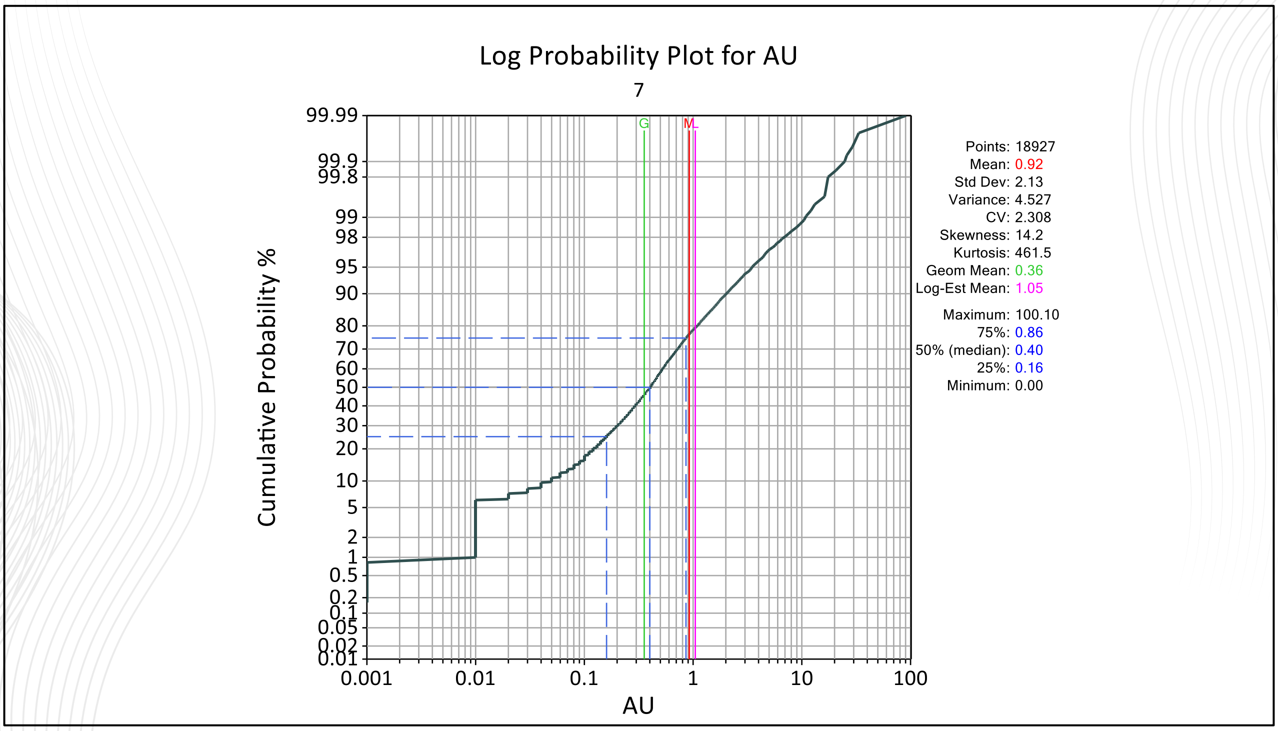


Figure 14-18 Log Histogram for Composited Gold Values(Vira Saia)

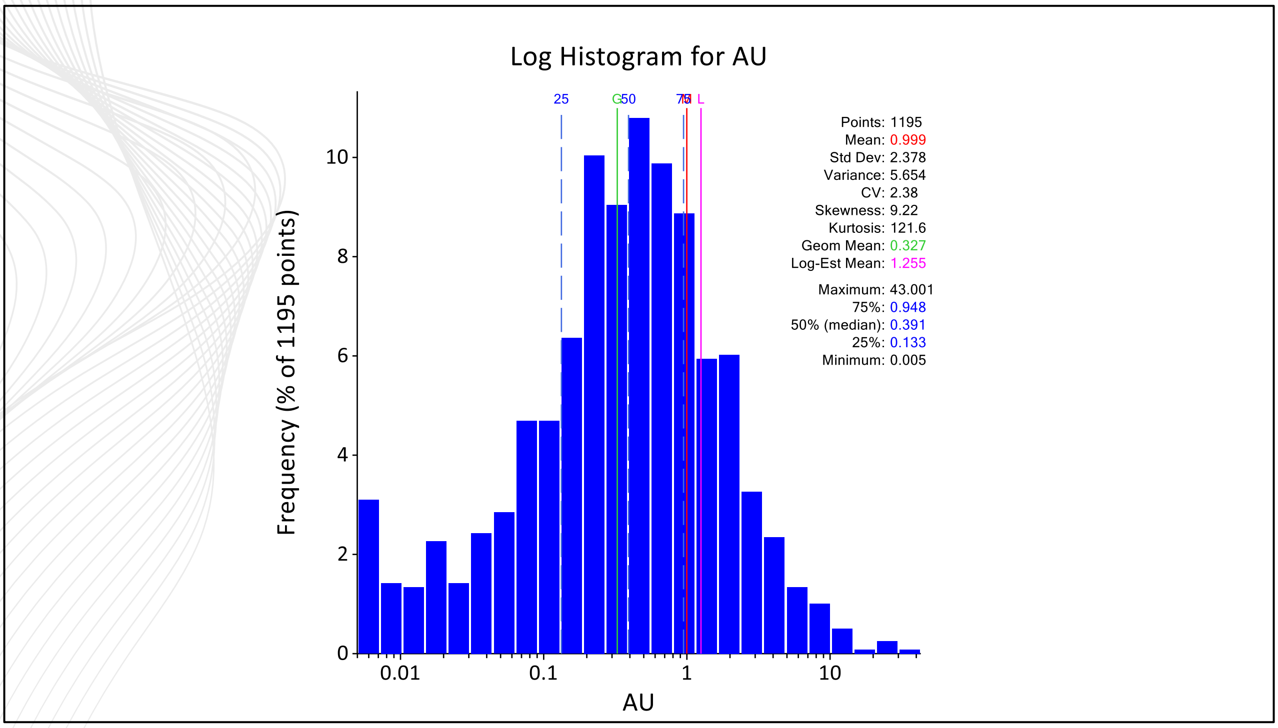
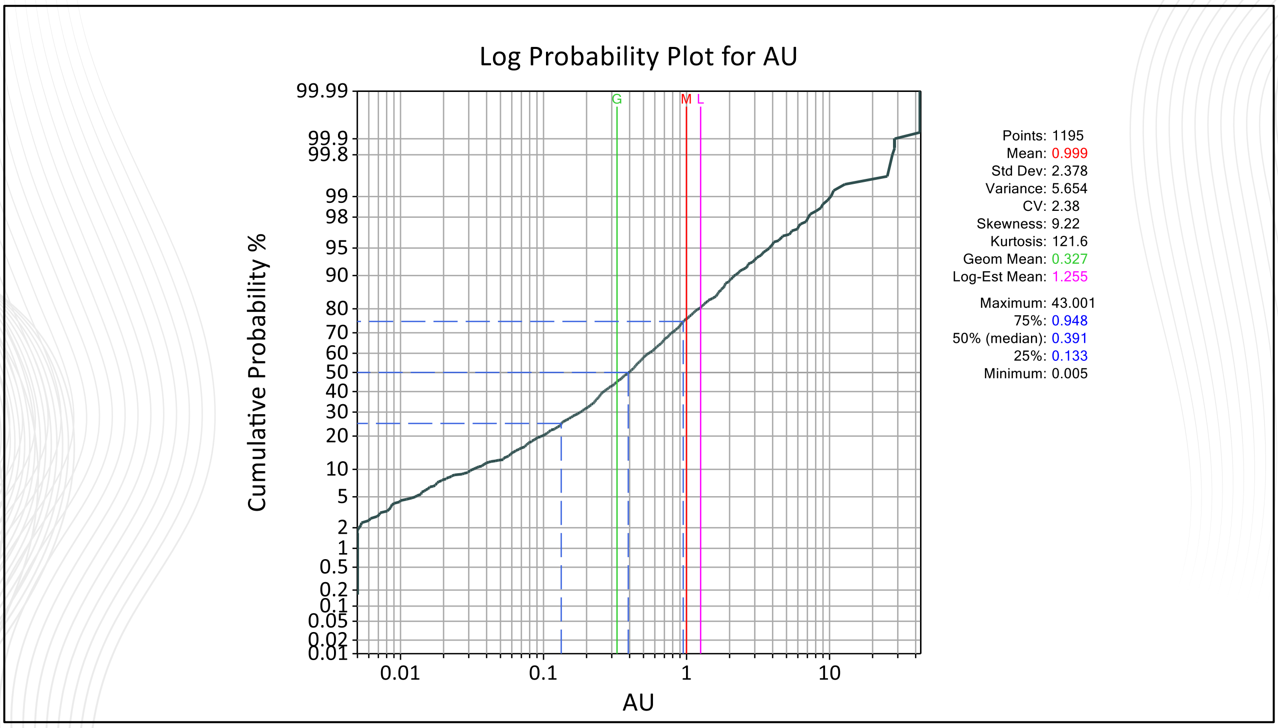


Figure 14-19 Log probability Plot for Composited Gold Values (Vira Saia)



14.8 EXPLORATORY DATA ANALYSIS (CATA FUNDA)

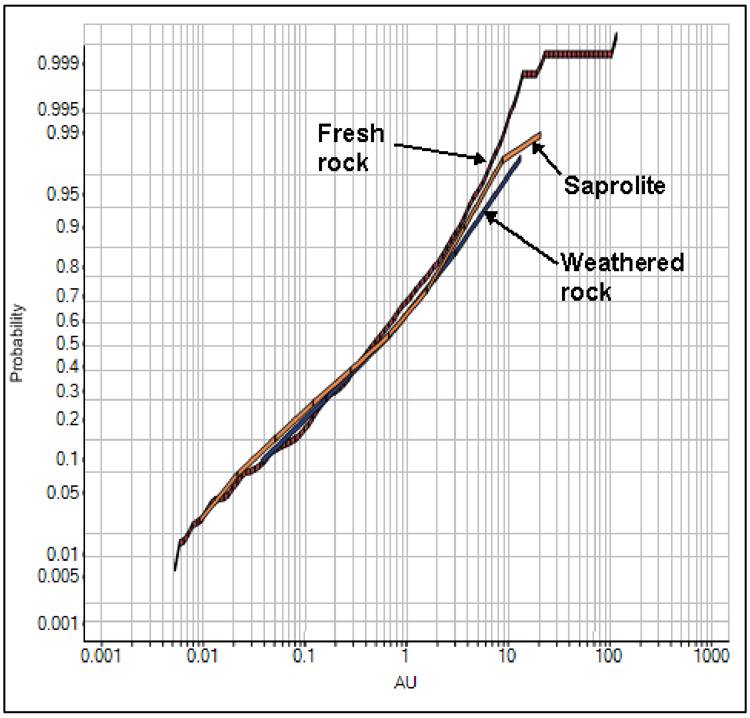
Assay samples were selected according to the interpreted 3D models for Cata Funda. A summary of the selected data is shown in Table 14-6. RC data have been excluded from this and all subsequent stages of resource estimation.

Table 14-6 Summary of Selected Drillhole Data (Cata Funda)

TYPE	LITHOLOGY	HOLES	LENGTH (m)	AVG. LENGTH (m)	SAMPLES
Diamond Drilling	Saprolite	7	39	6	41
	Weathered	4	18	4	19
	Fresh Rock	64	692	11	731
	Subtotal	**71**	**749**	**21**	**791**
Metallurgical Core	Saprolite	3	62.05	20.68	63
	Weathered	3	13.85	4.62	14
	Fresh Rock	3	192.40	64.13	197
	Subtotal	**3**	**268.30**	**89.43**	**274**
Total Drilling		**78**	**1,017.30**	**9.96**	**821**

A log-probability plot of the captured samples, showing the different populations by lithology, is shown in figure 14-20. This shows very similar grade populations for each lithology, and for that reason the estimation has used all samples within the interpreted mineralized zone, without specific lithology controls.

Figure 14-20 Log-Probability Plot of Selected Assays (Cata Funda)



As well as the log-probability plot, outlier grades were also investigated using decile analysis (Table 14-7) and a coefficient-of-variation analysis (Figure 14-21). From these results a top-cut level of 11 g/t Au was used prior to the generation of 2.5 m composites. In the generation of composites, modal control was used to make composite lengths of approximately 2.5 m in length, such that the composites were of equal length across each intersection.

Figure 14-21 Coefficient of Variation Graph

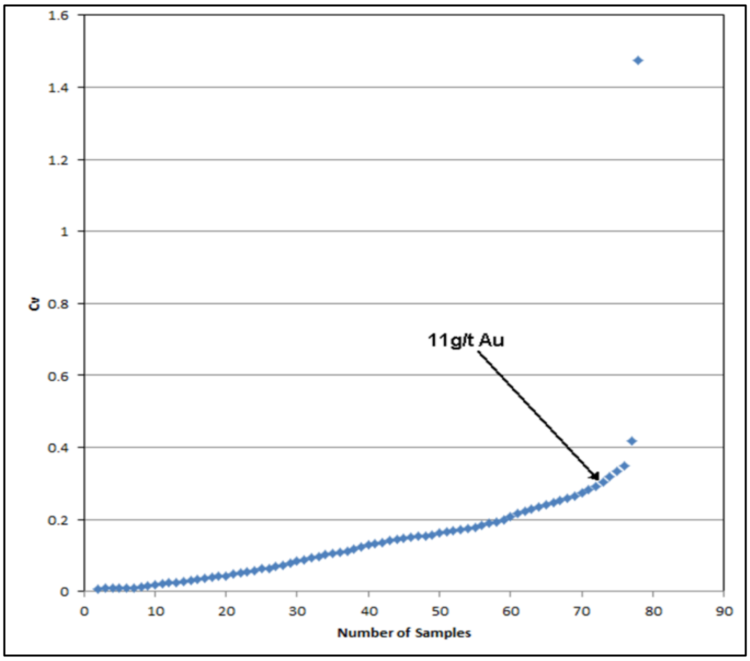


Table 14-7 Decile Analysis Summary

Q%_FROM	Q%_TO	NUMBER	MEAN	MINIMUM	MAXIMUM	METAL	METAL%
0	10	77	0.05	0.02	0.10	3.66	0.30
10	20	86	0.13	0.10	0.18	9.68	0.79
20	30	81	0.26	0.18	0.32	18.96	1.56
30	40	79	0.39	0.33	0.47	28.98	2.38
40	50	81	0.58	0.47	0.71	42.71	3.51
50	60	81	0.86	0.71	1.07	62.96	5.17
60	70	80	1.32	1.08	1.68	96.81	7.95
70	80	77	2.06	1.68	2.49	148.88	12.23
80	90	78	3.22	2.50	4.03	237.20	19.49
90	100	78	7.69	4.04	109.46	567.39	46.61
90	91	7	4.13	4.04	4.18	28.93	2.38
91	92	8	4.36	4.21	4.49	33.81	2.78
92	93	8	4.70	4.56	4.85	34.30	2.82
93	94	6	5.10	4.95	5.31	30.62	2.52
94	95	7	5.57	5.43	5.74	41.47	3.41
95	96	8	6.13	5.75	6.31	51.52	4.23
96	97	9	6.64	6.33	6.88	50.44	4.14
97	98	8	7.77	7.10	8.73	58.27	4.79
98	99	8	9.06	8.76	9.76	59.36	4.88
99	100	9	21.79	9.97	109.46	178.66	14.68
0	100	798	1.66	0.02	109.46	1,217.22	100.00

A statistical summary of the selected samples is shown in Table 14-8. Statistics for the generated 2.5 m composites are shown in Table 14-9. A log-probability plot of the composite data is shown Figure 14-22.

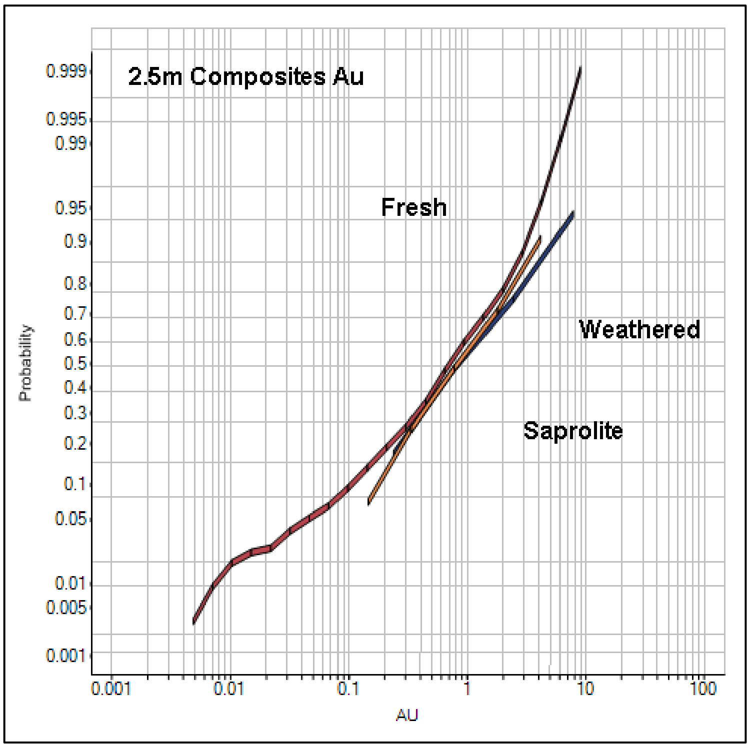
Table 14-8 Summary Statistics – Selected Gold Assay Samples

LITH	NUMBER	MINIMUM	MAXIMUM	MEAN	VARIANCE	STANDDEV	LOGESTMN	COVARTN%
Saprolite	43	0.006	13.47	1.48	5.21	2.28	2.49	154
Weathered	19	0.018	6.30	1.52	3.56	1.89	2.08	124
Fresh	767	0	109.46	1.58	19.33	4.40	2.09	278
All	**829**	**0**	**109.46**	**1.58**	**18.25**	**4.27**	**2.11**	**271**

Table 14-9 Summary Statistics – 2.5 m Composites

LITH	NUMBER	MINIMUM	MAXIMUM	MEAN	VARIANCE	STANDDEV	LOGESTMN	COVARTN%
Saprolite	15	0.0954	6.31	1.42	2.63	1.62	1.48	114
Weathered	7	0.133	4.35	1.50	1.90	1.38	1.66	92
Fresh	280	0	10.80	1.46	2.27	1.51	1.87	103
All	**302**	**0**	**10.80**	**1.46**	**2.28**	**1.51**	**1.84**	**103**

Figure 14-22 Log Probability Plot from 2.5 m Composites



14.9 GEOSTATISTICAL ANALYSIS (VARIOGRAMS)

14.9.1 PAIOL

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

The anisotropy directions are coincident with the deposit shape (geological model). The strike of the deposit was adopted to be azimuth of major axis. The azimuth of major axis was selected to be 25° NE with a plunge of -65°. All data reported are the results from the back-transform of the normal scores. Semi-variograms and correlograms (Figures 14-23) were calculated to analyze geometric and zonal anisotropy. Figure 14-24 is showing back-transformed results and summarizes the semi-variogram parameters for Paiol dataset.

Figure 14-23 Paiol Model Semi-variograms

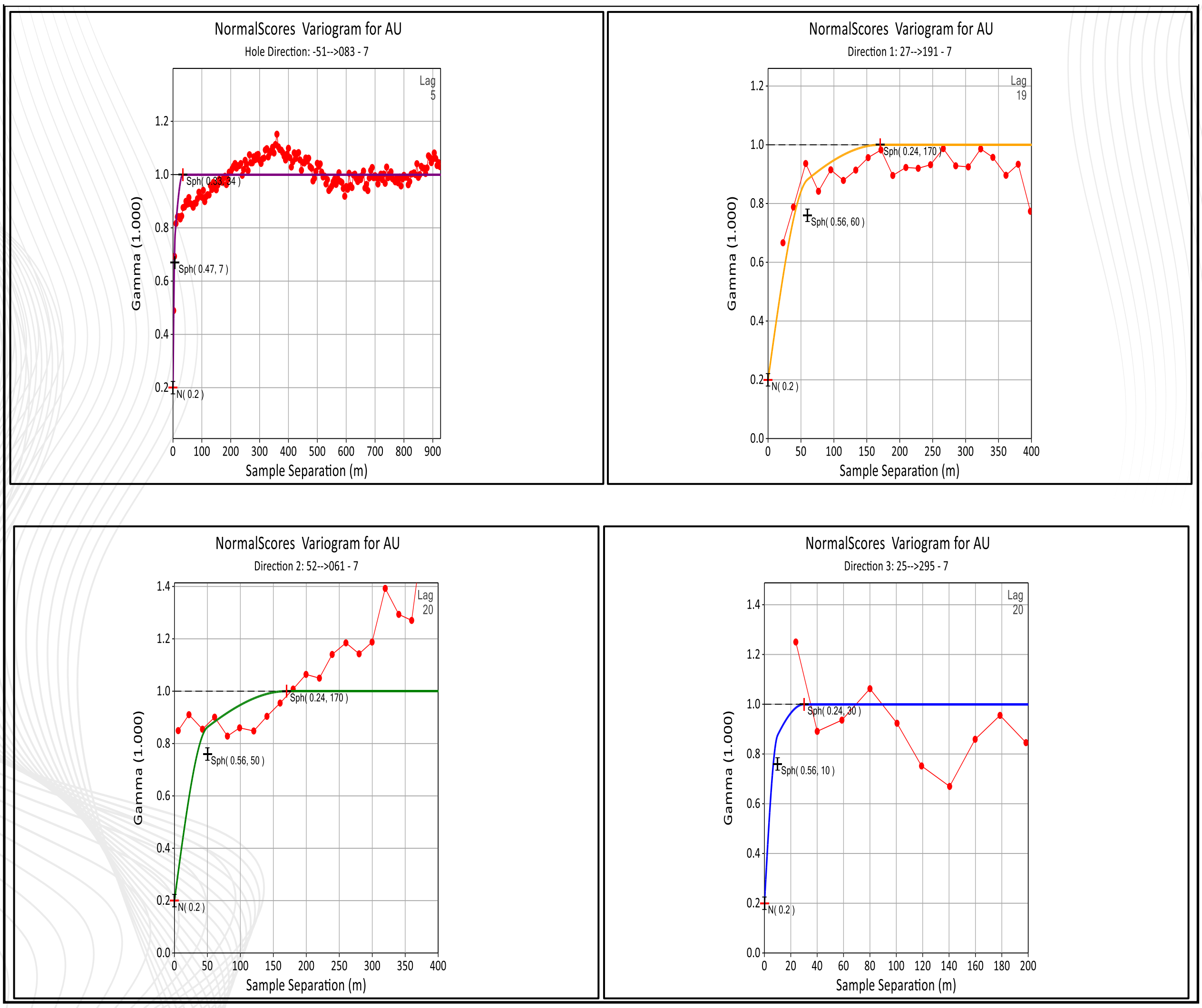
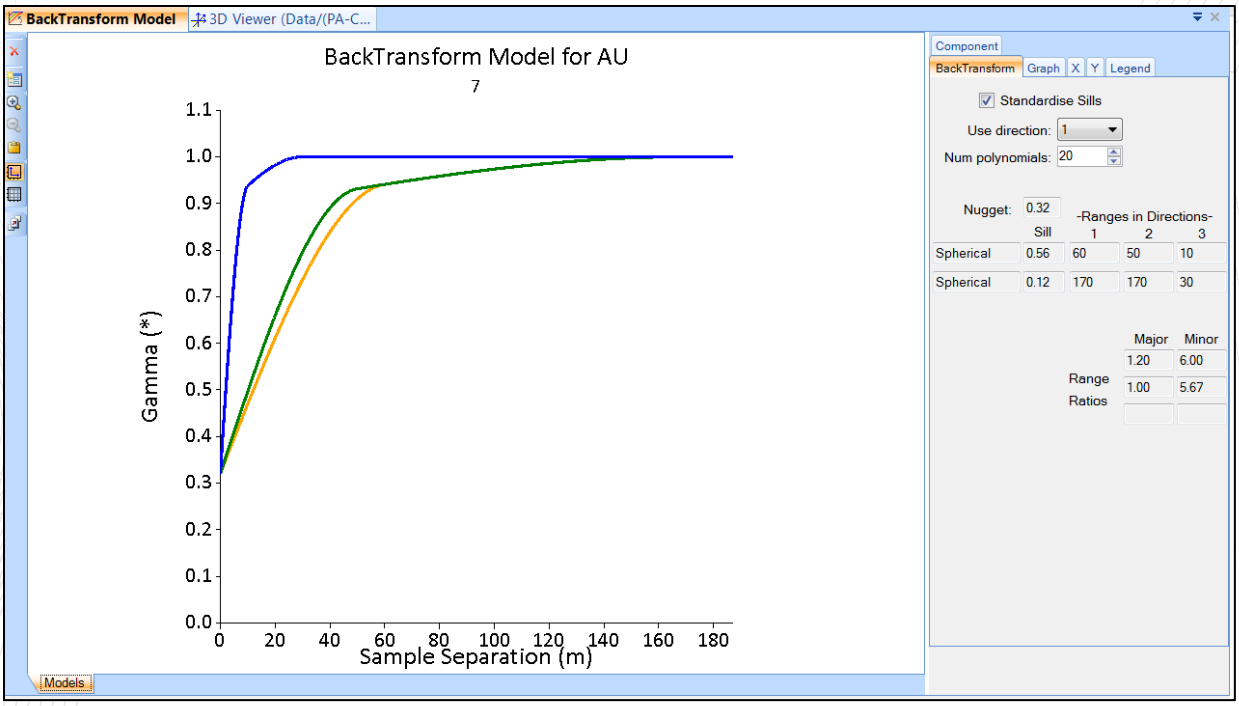


Figure 14-24 Paiol BackTransform Model

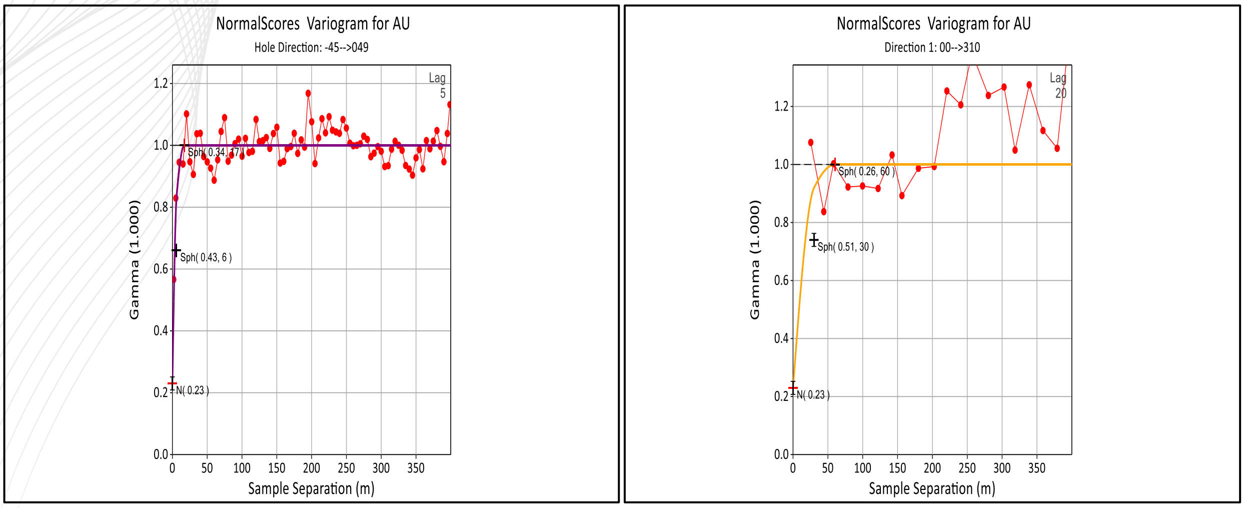


14.9.2 VIRA SAIA

Variography for composited samples was completed using Snowden's Supervisor software .A variography model was fitted for the composited data within Vira Saia deposit model. For continuity modelling with a normal scores transform was used.

The anisotropy directions are coincident with deposit shape (geological model). The azimuth for the major axis is 310° (strike of deposit) with a plunge of -70°. All data reported are the results from the back-transform of the normal scores. Semi-variograms and correlograms (Figures 14-25) were calculated to analyze geometric and zonal anisotropy. Figure 14-26 shows back-transformed results and summarizes the semi-variogram parameters for the Vira Saia dataset.

Figure 14-25 Vira Saia Model Semi-variograms



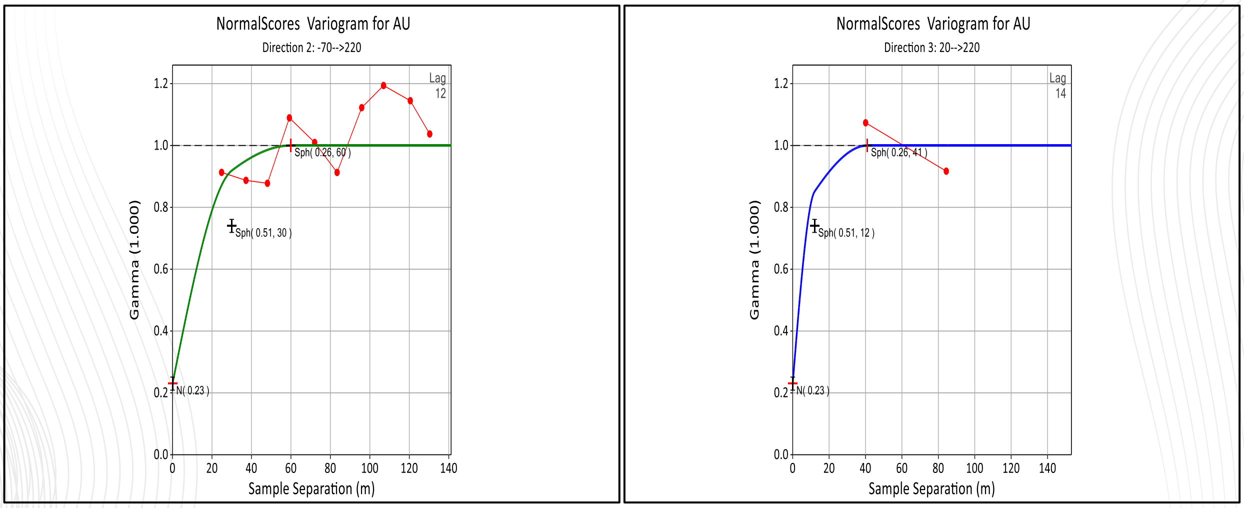
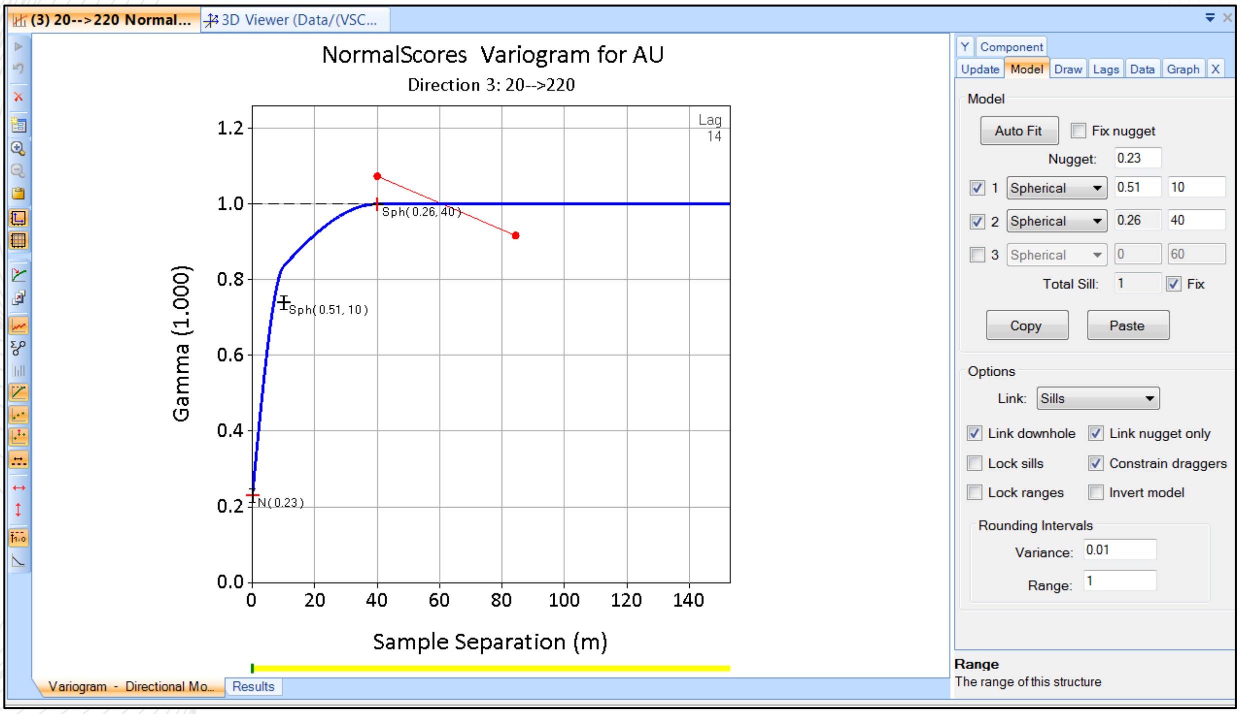


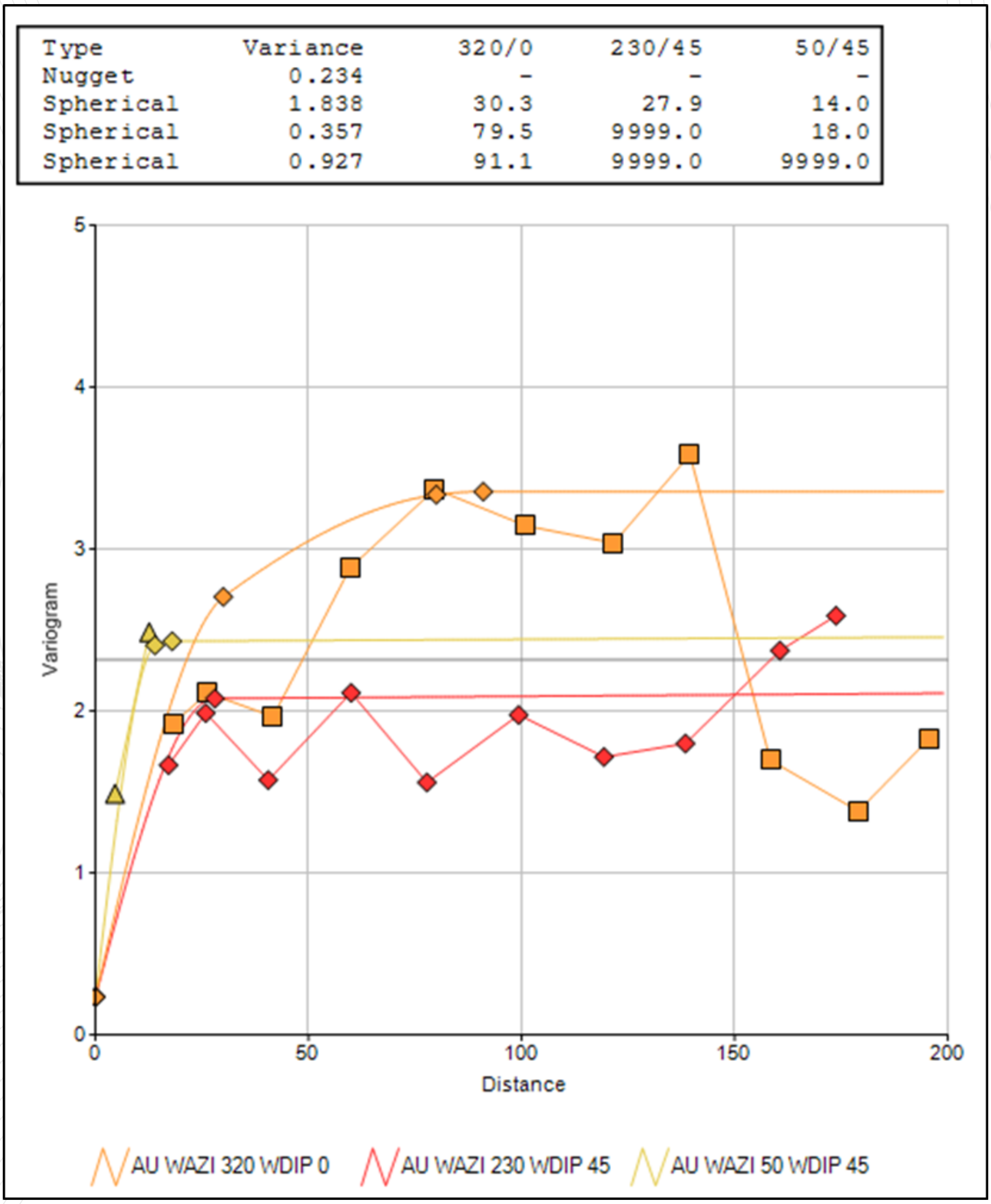
Figure 14-26 Vira Saia BackTransformed Model



14.9.3 CATA FUNDA

Experimental variograms were generated from the composites, and models were fitted, as shown Figure 14-27. It can be seen from this that the down-dip range is approximately 30 m, while the along-strike range is approximately 75 m.

Figure 14-27 Cata Funda Model Variograms



14.10 BLOCK MODELS SET UP

14.10.1 PAIOL

The block model limits were defined using UTM coordinates and the block size selected for the model was 10 x 10 x 5 m. The model was rotated so that the north axis of the model is approximately parallel with the strike of the mineralization, 25°. For Gemcom® software, this is a rotation of -25°. The block model definition is given in Table 14-10.

Table 14-10 Block Model Definition (Paiol)

	ORIGIN	BLOCK SIZE (m)	NO. BLOCKS
Easting (X)	263,610.239	10	170
Northing (Y)	8,704,547.252	10	265
Elevation	450	5	115

14.10.2 VIRA SAIA

The block model limits were defined using UTM coordinates and the block size selected for the model was 5 x 5x 2.5 m. The model was rotated so that the north axis of the model is approximately parallel with the strike of the mineralization, 310°. For Gemcom® software, this is a rotation of -310°. The block model definition is given in Table 14-11.

Table 14-11 Block Model Definition (Vira Saia)

	ORIGIN	BLOCK SIZE (m)	NO. BLOCKS
Easting (X)	265,100	5	120
Northing (Y)	8,710,200	5	430
Elevation	450	2.5	135

14.10.3 CATA FUNDA

The block model limits were defined using UTM coordinates and the block size selected for the model was 10 x 5 x 5 m. The model was rotated so that the north axis of the model is approximately parallel with the strike of the mineralization, 320°. For Datamine® software, this is a rotation of 50° about z axis. The block model definition is given in Table 14-12.

Table 14-12 Block Model Definition (Cata Funda)

	ORIGIN	BLOCK SIZE (m)	NO. BLOCKS
Easting (X)	264,234	10	66
Northing (Y)	8,719,327	5	98
Elevation	225	5	52

14.11 GRADE INTERPOLATION

14.11.1 PAIOL

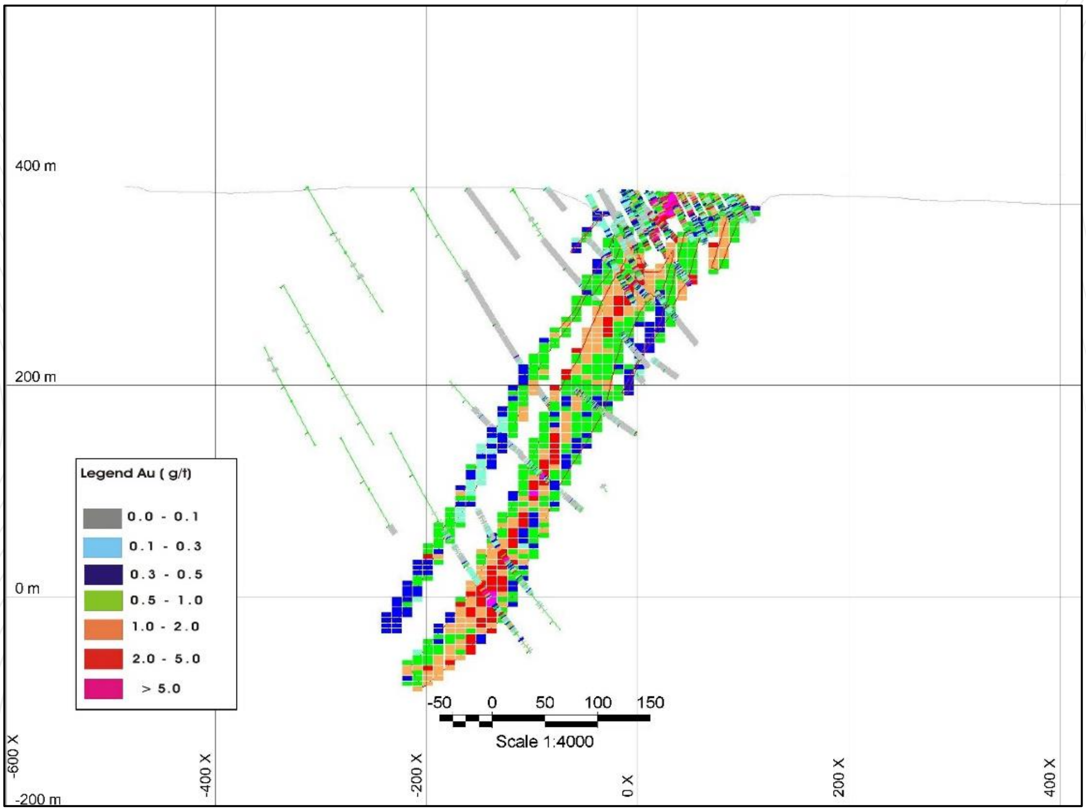
The grade interpolation used Ordinary Kriging with variography as set out in Section 14.9. The updated 3D model coded in the block model , was interpolated, using just the data points from inside that zone as the data source. The grade shell domain was modelled using 4 interpolation runs. Given the current average grid spacing of about 25 m by 25 m in the upper zone of the domains, the short 1st structure in the variograms was adopted for the first and second passes. From this structure in the correlogram model, ellipsoid search distances were obtained using half of the range of the first structure (first pass) and approximately 75% of correlogram model (second pass) (Figure 14-23). The third pass adopted the full range of the correlogram model, and the fourth pass was two times the range of correlogram model (Figures 14-23 and 14-24). More than 90% of the model was populated by gold grade using the first 3 passes. At least two holes were used to estimate the blocks (defining seven composites as maximum by hole).

The search parameters for each interpolation run are listed in Table 14-13. A typical section through the estimated block model is shown in Figure 14-28.

Table 14-13 Grade Interpolation Parameters (Paiol)

SEARCH REFERENCE	SEARCH DISTANCES (m)			MINIMUM Nº COMPOSITES	MAXIMUM Nº COMPOSITES	MAXIMUM Nº SAMPLES PER DRILL HOLES	OCTANT (MINIMUM)	MAXIMUM SAMPLES PER OCTANT
	X	Y	Z					
1	30	25	5	8	32	7	4	6
2	60	50	10	8	32	7	4	6
3	170	170	30	4	16	3	2	4
4	340	340	60	2	12	1	2	2

Figure 14-28 Typical Cross-Section Through the Estimated Paiol Block Model (Looking NE)



14.11.2 VIRA SAIA

The grade interpolation used Ordinary Kriging with variography as set out in Section 14.9. An Updated 3D model, coded in the block model, was interpolated, using just the data points from inside that zone as the data source. The grade shell domain was modelled using 3 interpolation runs. The short 1st structure in the variograms was adopted for first pass which was approximately 75% of the correlogram model and the ellipsoid search distance for the second pass was at the full range of the correlogram model (Figures 14-25 and 14-26). The third pass was set at two times the range of the correlogram model. At least three holes were used to estimate the blocks for the first and second passes.

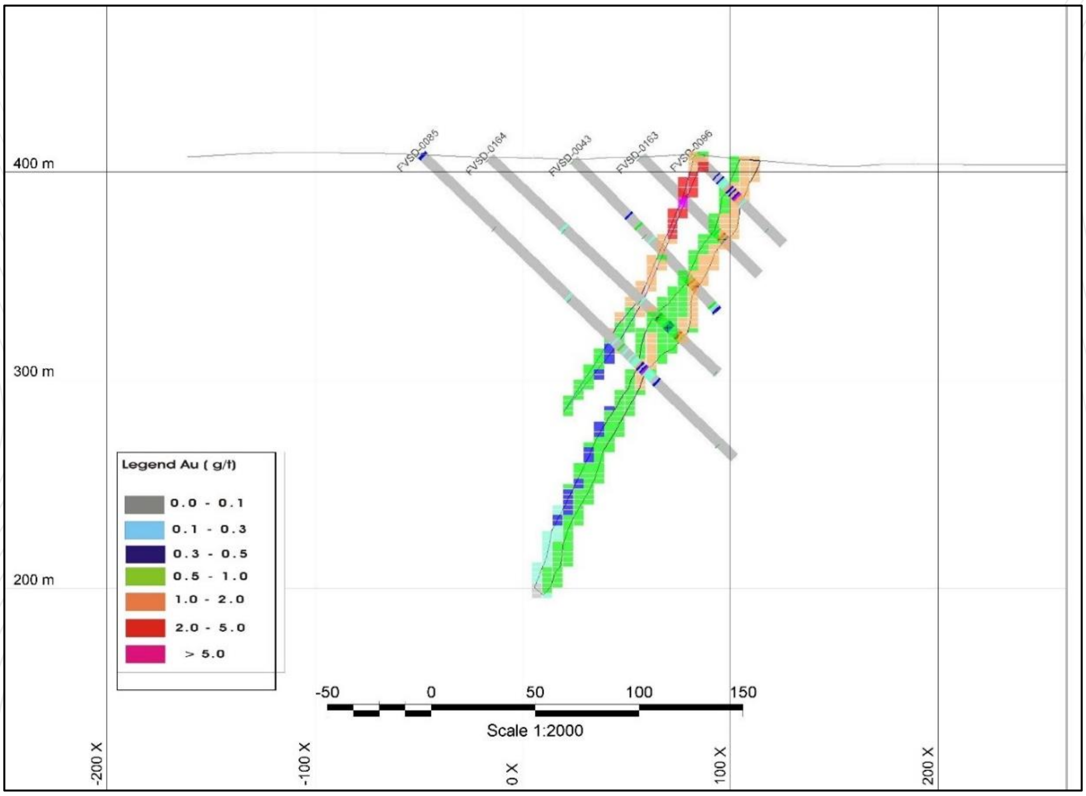
Dynamic anisotropy was applied, such that the search ellipse was oriented parallel to the mineralized zone, considering local variations in strike and dip.

The search parameters for each interpolation run are listed in Table 14-14. A typical section through the estimated block model is shown in Figure 14.29.

Table 14-14 Grade Interpolation Parameters (Vira Saia)

SEARCH REFERENCE	SEARCH DISTANCES (m)			MINIMUM Nº COMPOSITES	MAXIMUM Nº COMPOSITES	MINIMUM DRILLHOLES	MAXIMUM COMPOSITES PER HOLE
	X	Y	Z				
1	30	30	5	9	16	3	4
2	60	60	15	9	16	3	4
3	120	120	30	1	16	1	2

Figure 14-29 Typical Cross-Section Through the Estimated Vira Saia Block Model (Looking NW)



14.11.3 CATA FUNDA

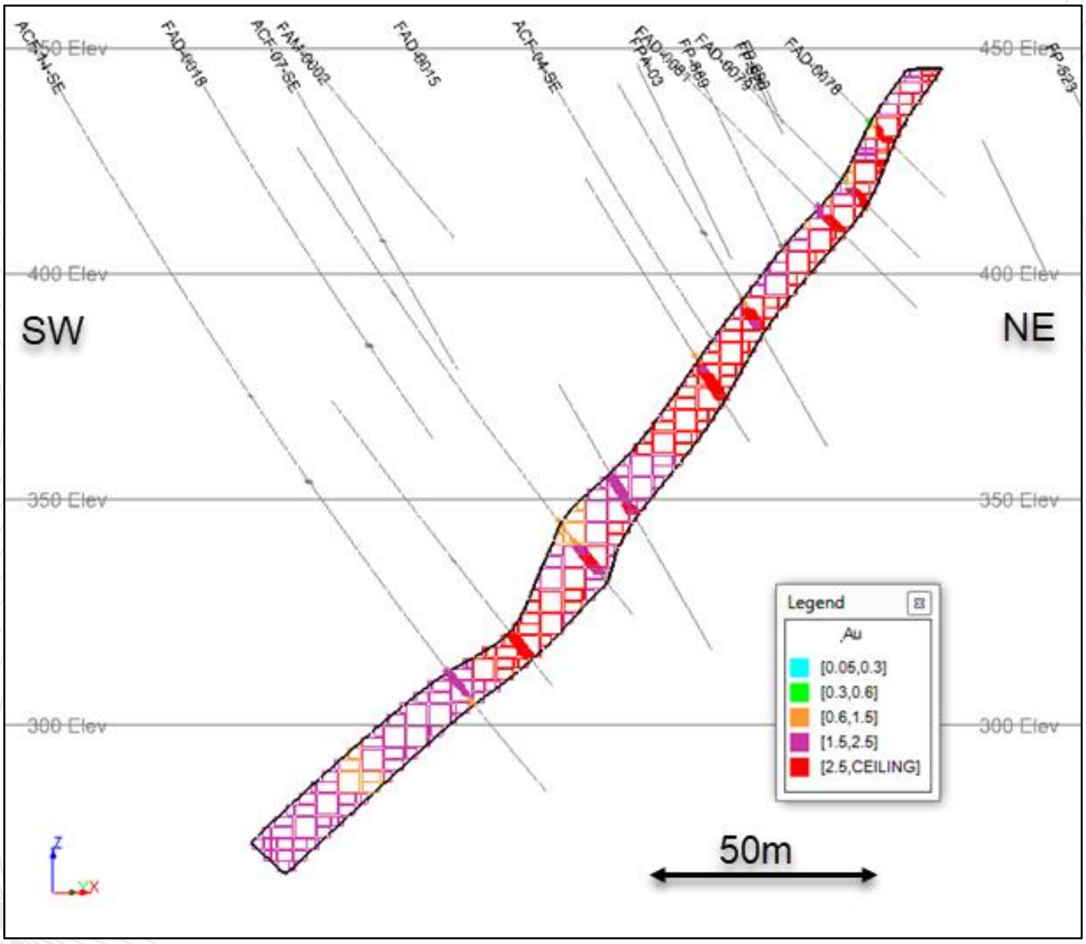
Ordinary kriging was used as the primary means of grade estimation. Alternative Au grades were also determined by the nearest neighbor and inverse-distance weighting methods, for validation purposes. The estimation parameters applied are shown in Table 14-15.

Dynamic anisotropy was applied, such that the search ellipse was oriented parallel to the mineralized zone, considering local variations in strike and dip. A typical section through the estimated block model is shown in Figure 14-30.

Table 14-15 Grade Interpolation Parameters (Cata Funda)

SEARCH REFERENCE	SEARCH DISTANCES (m)			MINIMUM Nº COMPOSITES	MAXIMUM Nº COMPOSITES	MINIMUM DRILLHOLES	MAXIMUM COMPOSITES PER HOLE
	X	Y	Z				
1	50	30	10	9	16	3	4
2	75	45	15	9	16	3	4
3	150	90	30	1	16	1	4

Figure 14-30 Typical Cross-Section Through the Estimated Cata Funda Block Model



14.12 BLOCK MODEL VALIDATION

For all three models, grade estimation was done using Ordinary Kriging (OK), Inverse Distance Squared (ID^2) and Nearest Neighborhood (NN) methods and the results compared against each other for validation purposes. A visual comparison between composites and blocks was done section by section and level by level to investigate local bias.

A global comparison also was made between raw assays, composites which were used for the models and block model average grades.

Finally, swath plots were used to compare input and output gold grades across the mineralized domain in the north and south, directions.

14.12.1 PAIOL

Table 14-16 shows the univariate statistics comparing the unweighted gold composite statistics and the block model cells.

Table 14-16 Paiol Composites vs. Block Model Comparison

	UNWEIGHTED SAMPLES	MODEL CELLS
Nº Records	6,590	44,349
Nº Samples	6590	44,349
Minimum	-	0.02
Q1	0.24	0.47
Median	0.47	0.68
Q3	0.88	1.04
Maximum	13	8.01
Mean	0.84	0.88
Mean Diff vs Model	0.04	-
%Mean Diff vs Model	4.53	-
Std.Dev	1.29	0.66
Variance	1.68	0.44
% Coeff. Variation	153.87	75.18
MAD	0.6	0.41
Model Tonnes		59,656,950

Note: In statistics, the median absolute deviation (MAD) is a robust measure of the variability of a univariate sample of quantitative data. It can also refer to the population parameter that is estimated by the MAD calculated from a sample.

Figure 14-31 shows a graphical representation (box and whisker plots) of the univariate statistics comparing the unweighted gold composite statistics and the block model cells. The lower tail of the composites plot is missing due to a minimum value of zero in the informing data. It is not practical to show zero on a log scale.

Figure 14-32 shows the informing composites vs. block grade swath plot for the Paiol deposit. The agreement is relatively good at the south end of the deposit but at the north (the right-hand side of the graph), where grades are higher, there is more smoothing and smearing happening. This is shown by the greater deviation of the green and red curves in this area. This smearing may lead to a slightly higher estimate of grade in these blocks. This can be seen in Table 14.16 where the mean, quartiles and median of the blocks is higher than the informing samples.

Figure 14-31 Paiol Box and Whisker Plats, Composites vs. Block Model

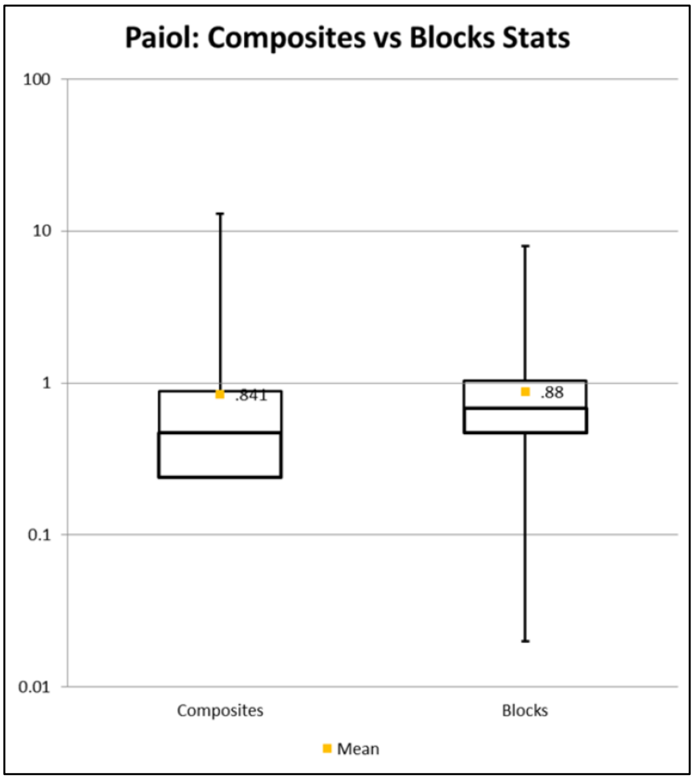
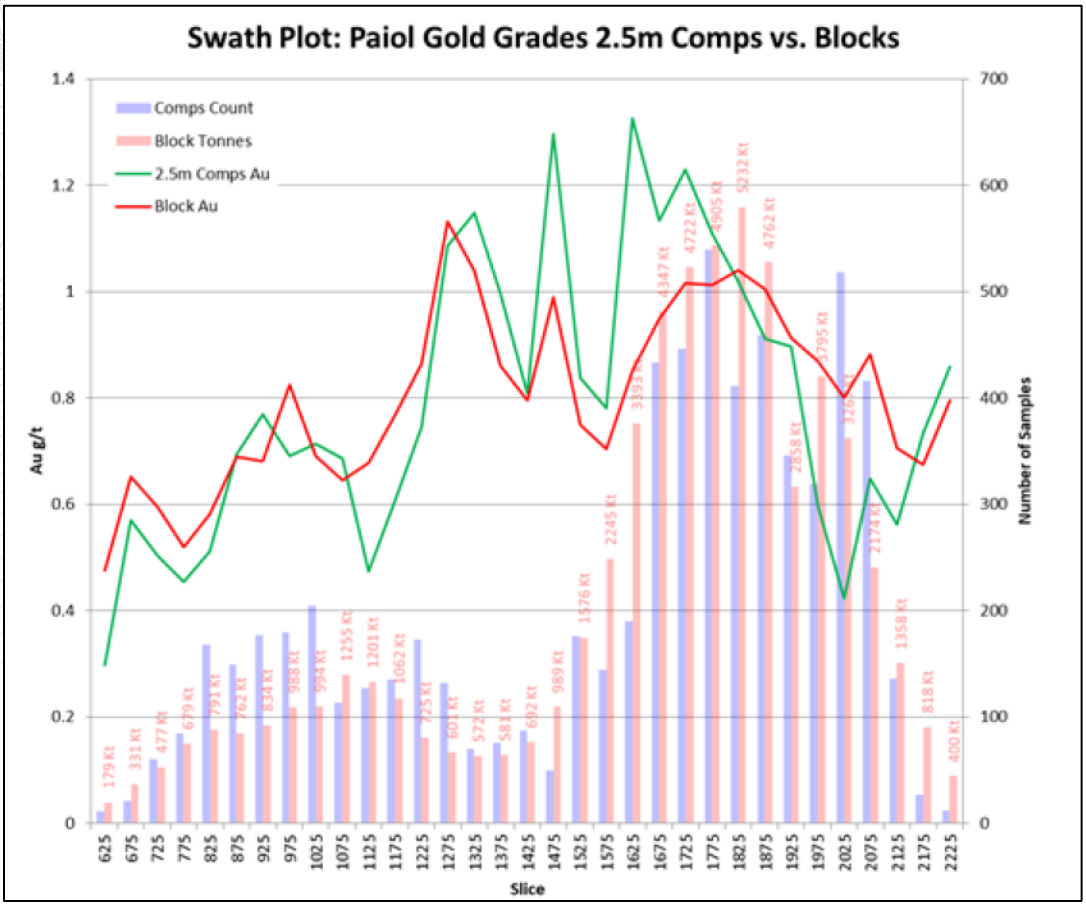


Figure 14-32 Paiol Swath Plot



14.12.2 VIRA SAIA

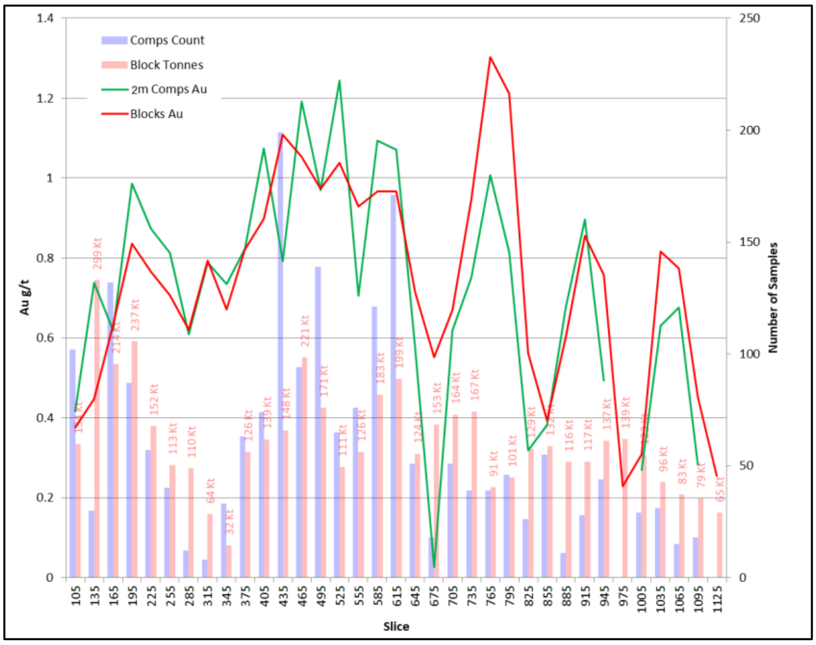
Table 14-17 shows univariate statistics comparing the unweighted gold composite statistics and the block model cells.

Table 14-17 Vira Saia composites vs. Block Model Comparison

	UNWEIGHTED SAMPLES	MODEL CELLS
Nº Records	1,195	111,263
Nº Samples	1195	111,263
Minimum	0.01	0.01
Q1	0.13	0.46
Median	0.39	0.67
Q3	0.95	0.98
Maximum	43.00	6.4
Mean	1.00	0.84
Mean Diff vs Model	-0.16	-
%Mean Diff vs Model	-16	-
Std.Dev	2.38	0.7
Variance	5.65	0.5
% Coeff. Variation	238	85.3
Model Tonnes		11,407,553

Once constructed the model was tested, using Swath plots (Figure 14-33). Swath plots show a suitable level of smoothing during the estimation together with a generally good correlation between the input and output grades. This provides confidence that the grade estimation process is robust.

Figure 14-33 Vira Saia Swath Plot



14.12.3 CATA FUNDA

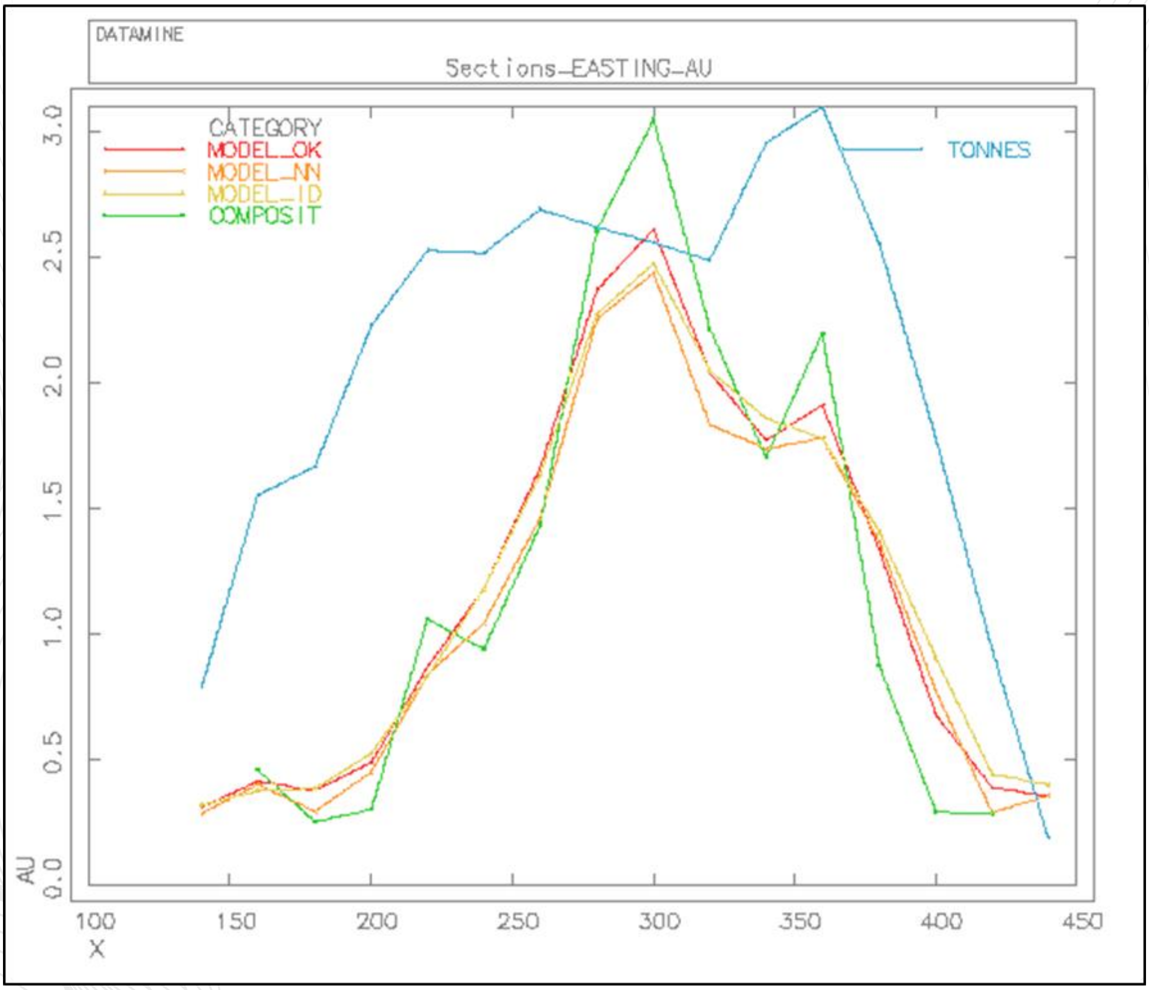
As well as examining the block model sections against composite grades, validation steps included the comparison of global average grades, as shown in Table 14-18; as well as making swath plots of average grades along cross-strike model slices, as shown in Figure 14-34.

Table 14-18 Comparison of Global Average Au Grades

ASSAYS		BLOCK MODEL AVERAGES*		
Raw Samples	Composites	NN	ID	OK
1.57	1.45	1.45	1.49	1.46

Note*:. NN = nearest neighbor
. ID = inverse distance (^2)
. OK - ordinary kriging

Figure 14-34 Swath Plots, Au Grades on 20 m Wide Sections



14.13 RESOURCE CLASSIFICATION

The Mineral Resources for the Paiol, Vira Saia and Cata Funda 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.

14.13.1 PAIOL

The Paiol model used the first and second passes to assign the measured and indicated categories, respectively. The passes resulted from the variography which was discussed in section 14.9. 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 resource classification criteria applied in the current study are those shown in Table 14-19.

Table 14-19 Classification Criteria (Paiol)

CATEGORY	PASS NO. (OKPASS)	APPROXIMATE DISTANCE (m)	MEAN DISTANCE OF COMPOSITES	MIN NO. COMPS	MAX. NO. COMPS	NO. OF DH	NO. OCTANT
Measured	1	≤ 30m	≤ 60	8	32	2	4
Indicated	2	>30 and ≤ 60	≤ 60	8	32	2	4
Inferred	3	No limit	>60	4	16	1	-

In order to avoid "spotted dogs" in classification, a polyline was constructed section by section for all Measured and Indicated blocks using the above criteria. Then a 3D model was constructed and all blocks outside this model assigned to the inferred category. The bottom part of this polyline extended up to 60 m beyond last drill hole in each section to preserve continuity of blocks while taking into consideration the criteria above. Figures 14.35 and 14-36 show the classification scheme at Paiol.

Figure 14-35 Classification Scheme in 3D (Paiol)

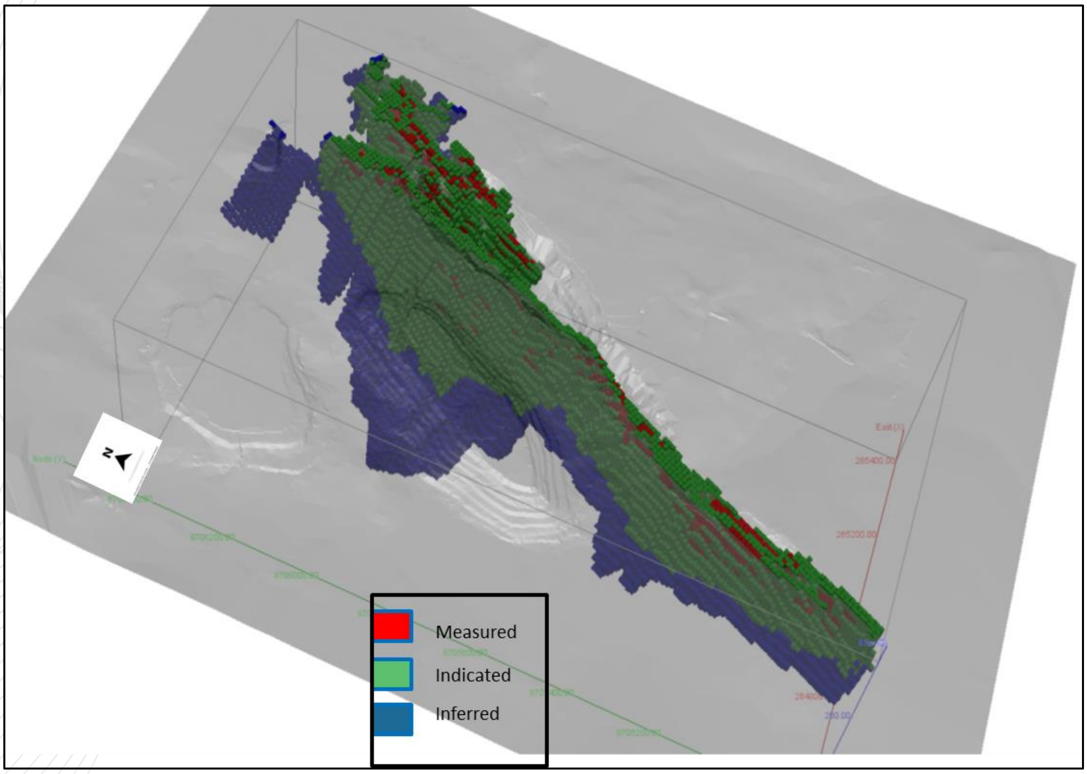
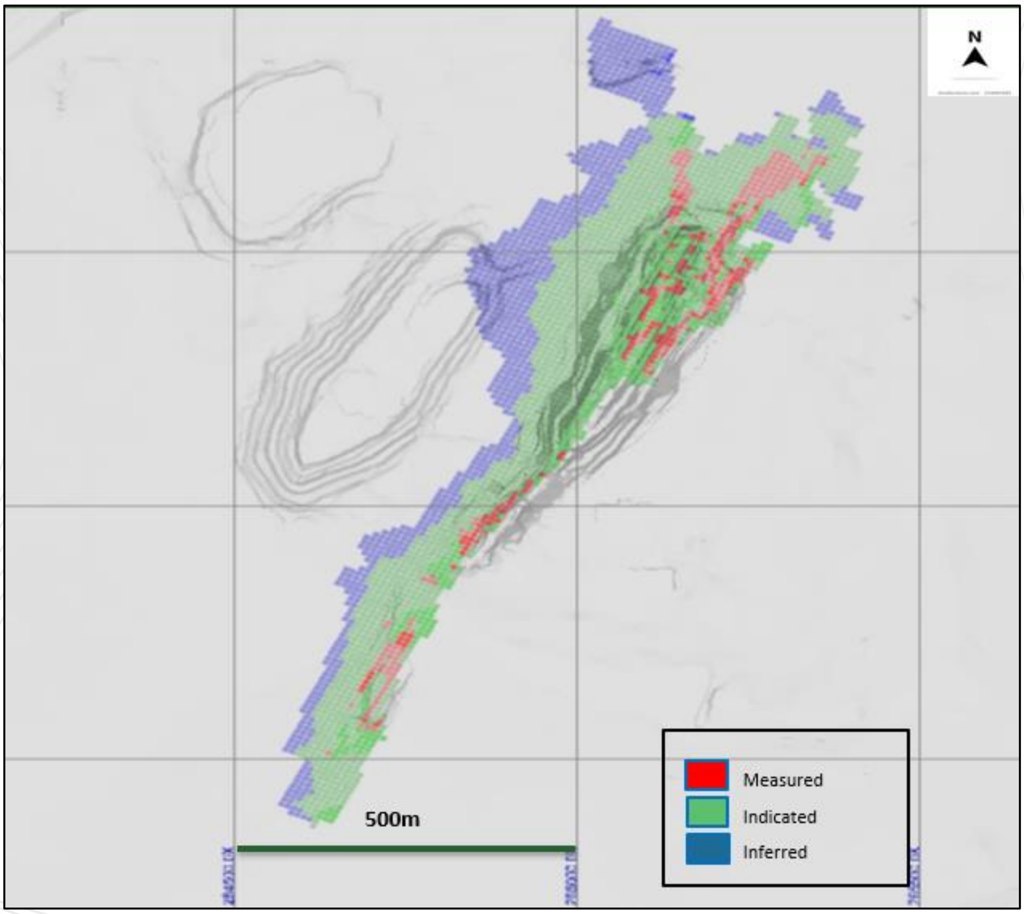


Figure 14-36 Classification Scheme in Plan View (Paiol)



14.13.2 VIRA SAIA

The Vira Saia 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 14.9. 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 resource classification criteria applied in the current study are those shown in Table 14-20.

Table 14-20 Classification Criteria (Vira Saia)

CATEGORY	PASS NO. (OKPASS)	APPROXIMATE DISTANCE (m)	MEAN DISTANCE OF COMPOSITES	MIN NO. COMPS	MAX. NO. COMPS	NO. OF DH
Measured	1	≤ 30m	≤ 60	9	16	4
Indicated	2	>30 and ≤ 60	≤ 60	9	16	4
Inferred	3	No limit	>60	1	16	2

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 3D model was constructed and all blocks outside this model were assigned to the inferred category.

All blocks within the Vira Saia HW Shear and the NW extension of Vira Saia ore model (Figure 14-4) are assigned to inferred resources without consideration of the criteria above. Figure 14-37 and figure 14-38 show the classification scheme in 3D and plan view for Vira Saia deposit respectively.

Figure 14-37 Classification Scheme in 3D (Vira Saia)-Looking NE

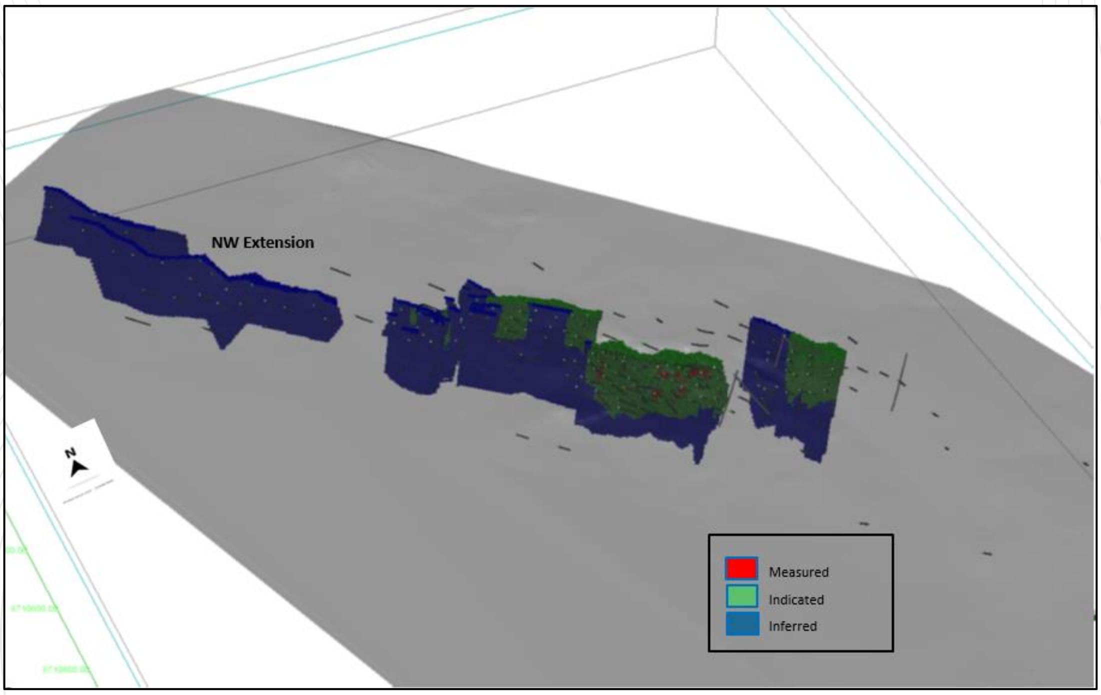
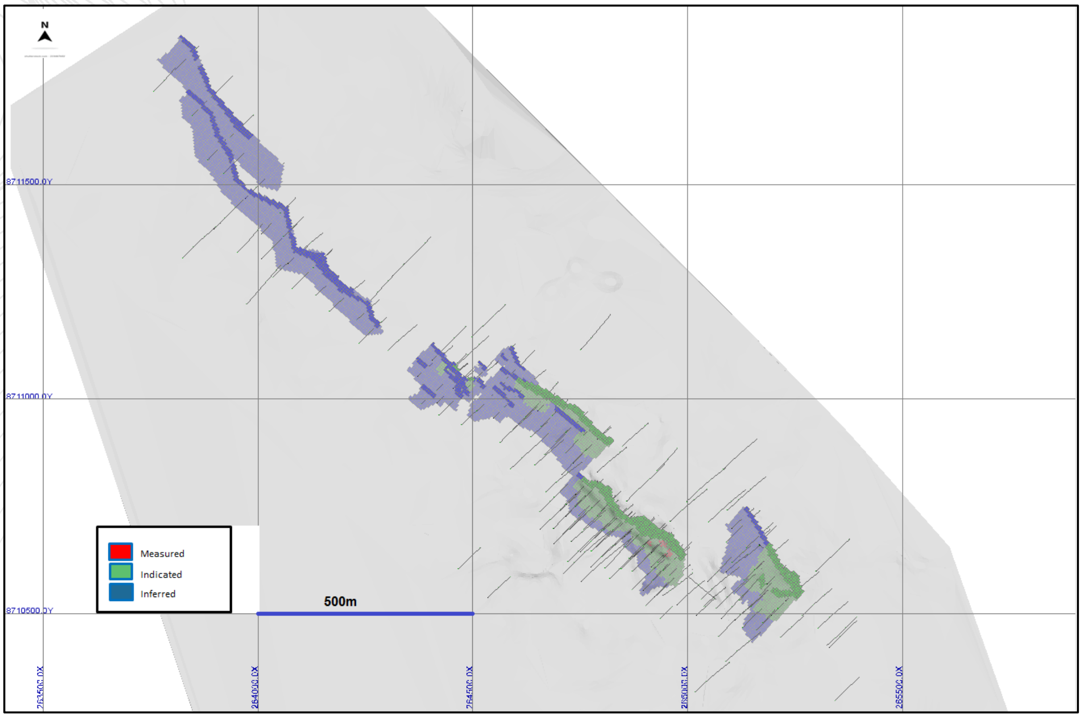


Figure 14-38 Classification Scheme in Plan View (Vira Saia)



14.13.3 CATA FUNDA

The Cata Funda model used the first and second passes to assign the measured and indicated categories, respectively. The passes were a result of the updated variography completed in the current study. The resource classification criteria applied in the current study are those shown in Table 14-21

Table 14-21 Classification Criteria (Cata Funda)

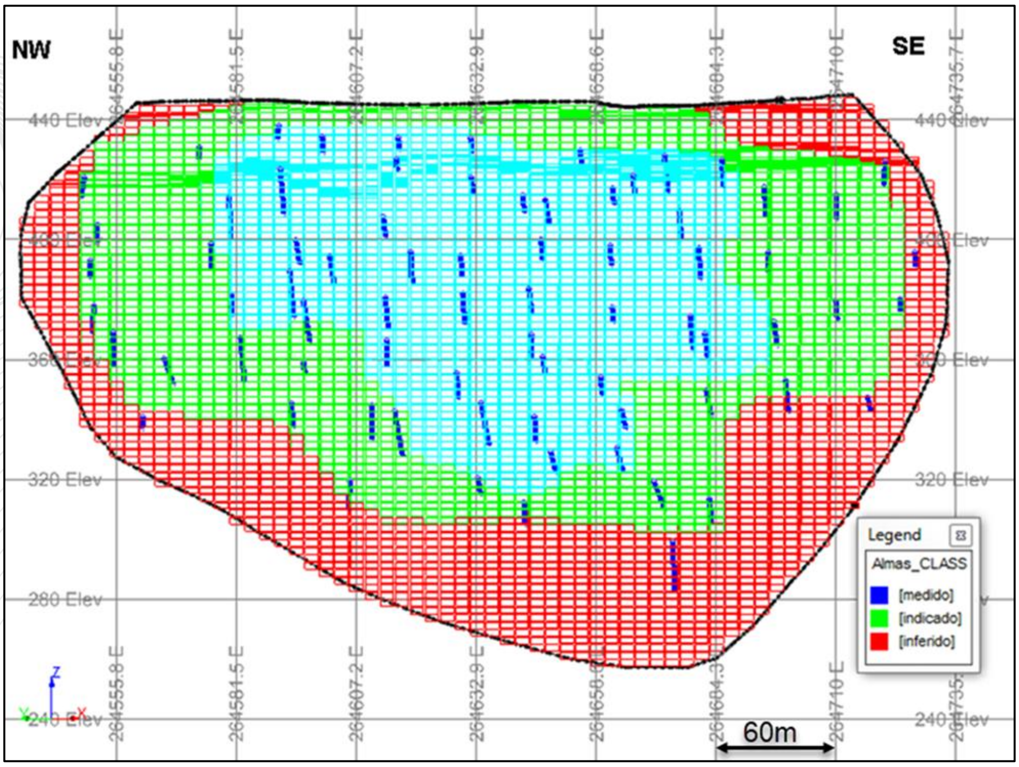
CATEGORY	DESCRIPTION
Measured	At least 4 holes within a search distance of 25m, within at least 4 octants
Indicated	At least 4 holes within a search distance of 40m, within at least 4 octants
Inferred	Grade estimated from drillhole data, but no greater than 80m from sample data

In the current study, these criteria were applied by using the following steps:

- Generation of search volumes based on ellipsoids and octant controls from the criteria above.
- Refinement of these volumes into clear resource class zones, using perimeters defined in long-section.

A long section of the resultant resource categories is shown in Figure 14-39.

Figure 14-39 Long Section of Resource Classification



14.14 MINERAL RESOURCE STATEMENTS

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 resource is reported should also meet this criterion. It should not include material which does not have reasonable potential to be mined and processed.

The definition of a Mineral Reserve on the other hand applies a specific set of economic parameters to a mineral resource to determine which portions of the Resource can be mined under those economic conditions.

In the case of the Paiol, Vira Saia and Cata Funda deposits economic modelling of the blocks in the model has indicated that the lowest grade block to be mined as ore has a grade of 0.29 g/t, 0.31 g/t and 0.34 g/t respectively. Details of these break-even cutoff grade for each deposit are summarized in Table 14-22.

Table 14-22 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
PAIOL	9.5	3.5	0.00	13.00	48.23	92%	0.29
VIRA SAIA	9.5	3.5	0.60	13.60	48.23	92%	0.31
CATA FUNDA	9.5	3.5	2.00	15.00	48.23	92%	0.34

On this basis the cut-off grades for the mineral resources has been set at 0.29 g/t, 0.31 g/t and 0.34 g/t Au respectively. The Mineral Resources above these cut-off grades are declared and summarized in Table 14-23, Table 14-24 and 14-25, while a grade tonnage curve for the deposit is shown in Figure 14-40, 14-41 and 14-42 and Table 14-26, 14-27 and 14-28. Figures 14-43 and 14-44 shows Paiol and Vira Saia against the optimized pit shells @ 1,800 $ gold price.

Table 14-23 Paiol Mineral Resources

RESOURCES CATEGORY	TONNES (t)	Au (g/t)	Oz
Measured	4,366,950	1.03	144,870
Indicated	13,181,190	0.96	407,590
Measured + Indicated	**17,548,140**	**0.98**	**552,460**
Inferred	3,504,330	1.23	138,810

Note:
1. The Mineral Resource estimates were prepared in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by the CIM Council on May 10, 2014, and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, adopted by CIM Council on November 29, 2019 using geostatistical and/or classical methods, plus economic and mining parameters appropriate to the deposit.
2. The Mineral Resource Estimate is based on an updated optimized pit shell using US$1,800/oz gold and at a cut-off grade of 0.29 g/t gold.
3. A density default model based on rock type was used for volume to tonnes conversion with averaging 2.66 tonnes/m^3.
4. Contained metal figures may not add due to rounding.
5. Mineral Resources are inclusive of Mineral Reserves.
6. Surface topography based on December 31st, 2016.
7. The Mineral Resource estimate for the Paiol deposit was prepared by Farshid Ghazanfari, P.Geo. a Qualified Person as that term is defined in NI 43-101

Table 14-24 Vira Saia Mineral Resources

RESOURCES CATEGORY	TONNES (t)	Au (g/t)	Oz
Measured	566,910	1.24	22,600
Indicated	2,787,780	0.91	81,245
Measured + Indicated	**3,354,690**	**0.96**	**103,845**
Inferred	1,516,230	1.05	51,070

Note:

1. The Mineral Resource estimates were prepared in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by the CIM Council on May 10, 2014, and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, adopted by CIM Counci on November 29, 2019, using geostatistical and/or classical methods, plus economic and mining parameters appropriate to the deposits
2. Mineral Resource Estimate is based on a designed pit optimized @ 1800$ gold and cut-off grade of 0.31 g/t gold.
3. A density model based on rock type was used for volume to tonnes conversion with averaging 2.60 tonnes/m^3.
4. Contained metal figures may not add due to rounding.
5. Surface topography based on December 31st, 2016
6. Mineral Resources are inclusive of Mineral Reserves.
7. The Mineral Resource estimate for the Vira Saia deposit was prepared by Farshid Ghazanfari, P.Geo. a Qualified Person as that term is defined in NI 43-101.

Table 14-25 Cata Funda Mineral Resources

RESOURCES CATEGORY	TONNES (t)	Au (g/t)	Oz
Measured	482,000	1.97	30,540
Indicated	356,000	1.39	15,920
Measured + Indicated	**838,000**	**1.72**	**46,460**
Inferred	330,000	1.48	15,735

Note:

1. The Mineral Resource estimates were prepared in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by the CIM Council on May 10, 2014, and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, adopted by CIM Council on November 29, 2019, using geostatistical and/or classical methods, plus economic and mining parameters appropriate to the deposit.
2. The Mineral Resource Estimate is based on an updated optimized shell using 1,800 $/oz and a cut-off grade of 0.34 g/t gold.
3. A density model based on rock type was used for volume to tonnes conversion with averaging 2.72 tonnes/m^3.
4. Contained metal figures may not add due to rounding.
5. Surface topography based on December 31st, 2016.
6. Mineral Resources are inclusive of Mineral Reserves.
7. The Mineral Resource estimate for the Cata Funda deposit was prepared by Adam Wheeler, C.Eng. a Qualified Person as that term is defined in NI 43-101.

Table 14-26 Paiol Grade-Tonnage for Measured & Indicated Resources

Au Cut-Off	TONNES	Au	Au
g/t	Kt	g/t	Oz
0.1	18,458	0.94	559,171
0.2	18,222	0.95	557,901
0.29	17,548	0.98	552,460
0.4	16,026	1.04	535,362
0.5	14,050	1.12	506,715
0.6	11,943	1.22	469,476
0.7	10,074	1.33	430,393
0.8	8,485	1.44	392,175
0.9	7,148	1.55	355,723
1	6,004	1.66	320,819
2	1,322	2.74	116,565

Figure 14-40 Paiol Grade-Tonnage Curve for Measured and Indicated Resources

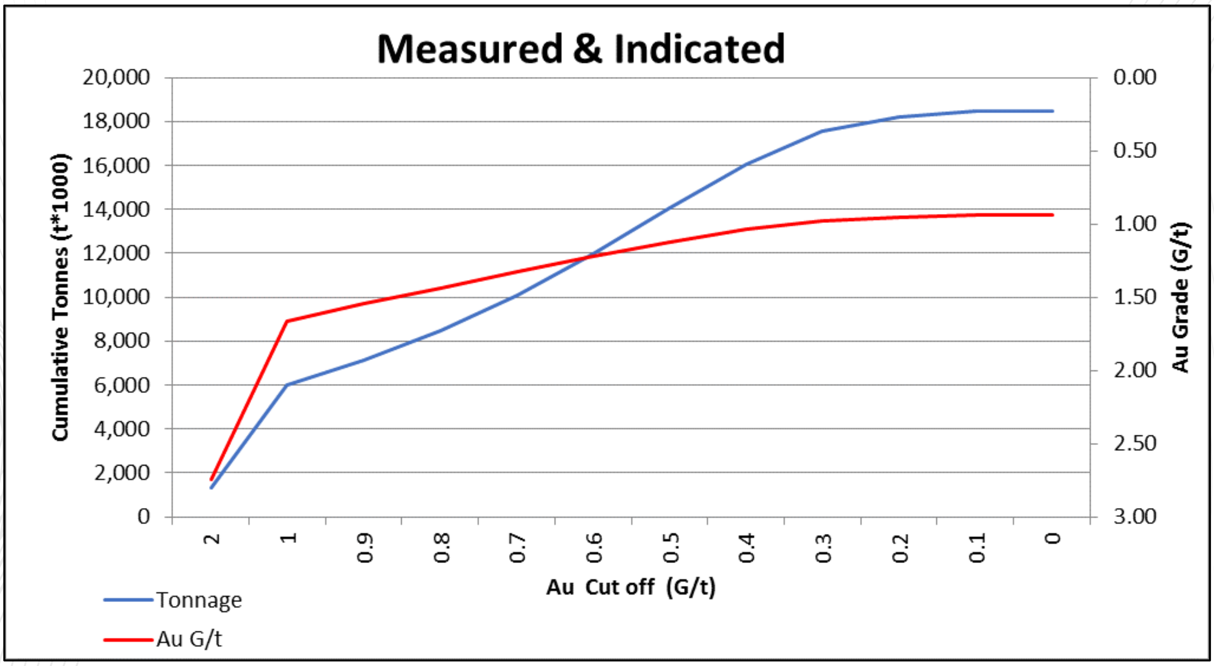


Table 14-27 Vira Saia Grade-Tonnage for Measured & Indicated Resources

Au Cut-Off	TONNES	Au	Au
g/t	Kt	g/t	Oz
0.1	3,575	0.92	105,565
0.2	3,530	0.93	105,321
0.31	3,355	0.96	103,833
0.4	3,037	1.03	100,181
0.5	2,667	1.11	94,818
0.6	2,325	1.19	88,766
0.7	1,979	1.28	81,547
0.8	1,673	1.38	74,175
0.9	1,416	1.48	67,168
1	1,187	1.58	60,156
2	219	2.53	17,838

Figure 14-41 Vira Saia Grade-Tonnage Curve for Measured and Indicated Resources

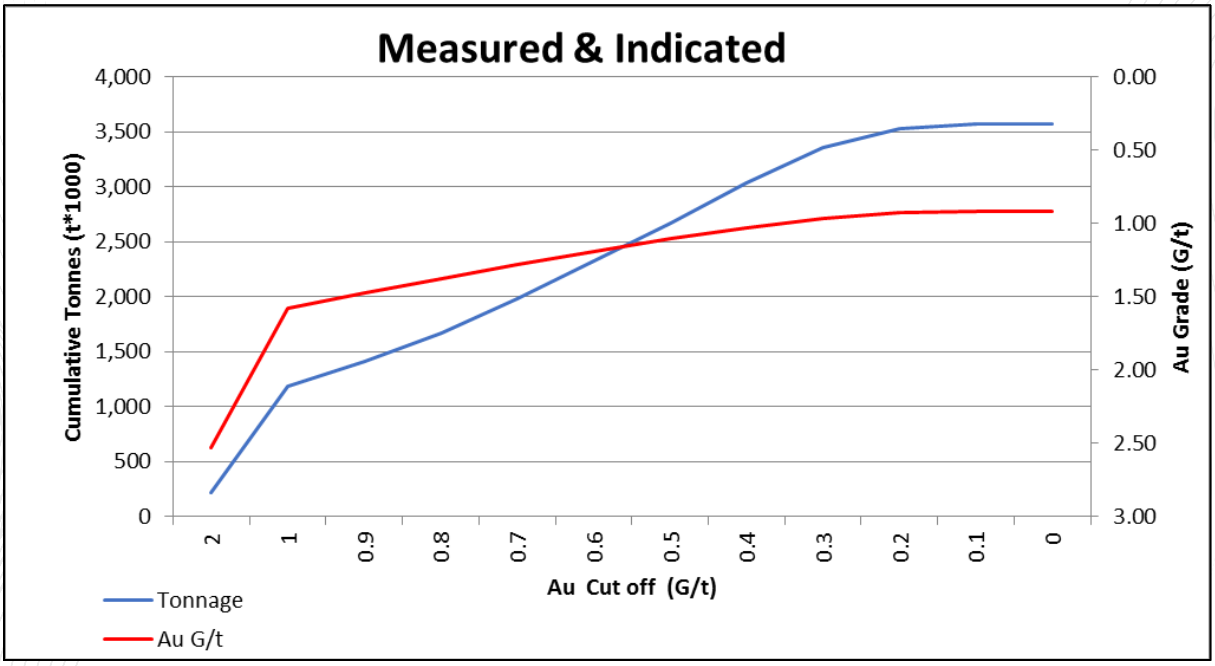


Table 14-28 Cata Funda Grade-Tonnage for Measured & Indicated Resources

Au Cut-Off	TONNES	Au	Au
g/t	Kt	g/t	Oz
0.10	893	1.63	46,846
0.15	886	1.64	46,818
0.20	868	1.67	46,715
0.25	856	1.69	46,629
0.30	846	1.71	46,538
0.35	837	1.73	46,440
0.40	824	1.75	46,285
0.45	814	1.76	46,147
0.50	799	1.79	45,922
0.55	773	1.83	45,478
0.60	751	1.87	45,073

Figure 14-42 Cata Funda Grade-Tonnage Curve for Measured and Indicated Resources

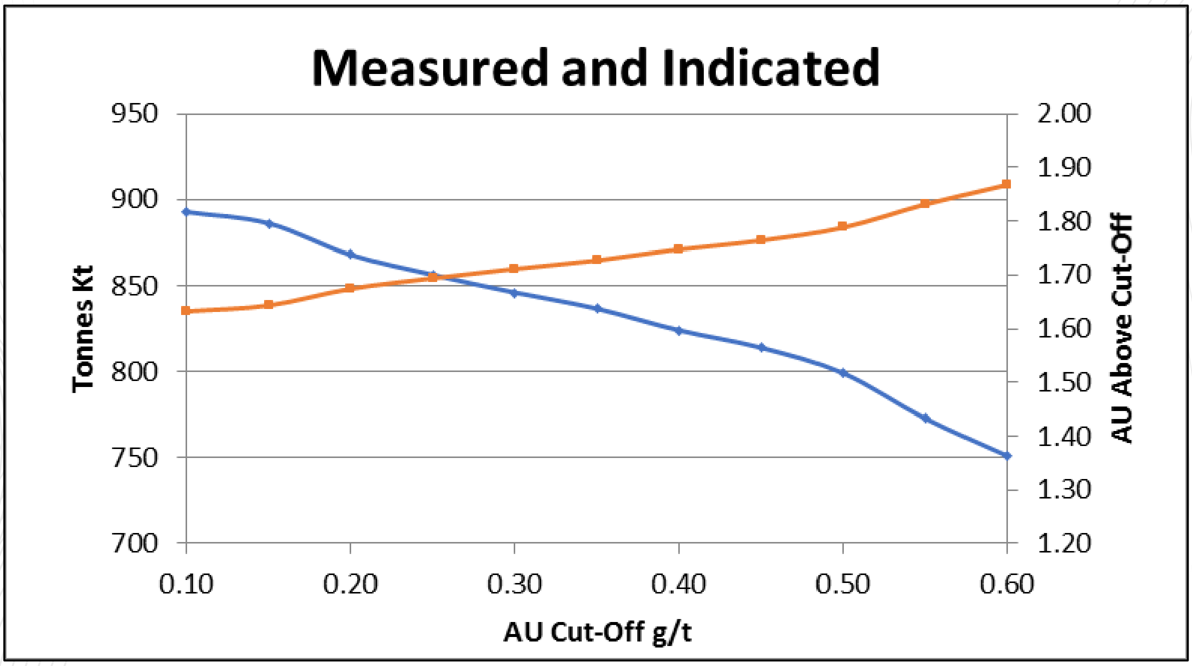


Figure 14-43 Typical Cross-Section (Paiol)-Block Model vs. 1,800 $ Pit shell

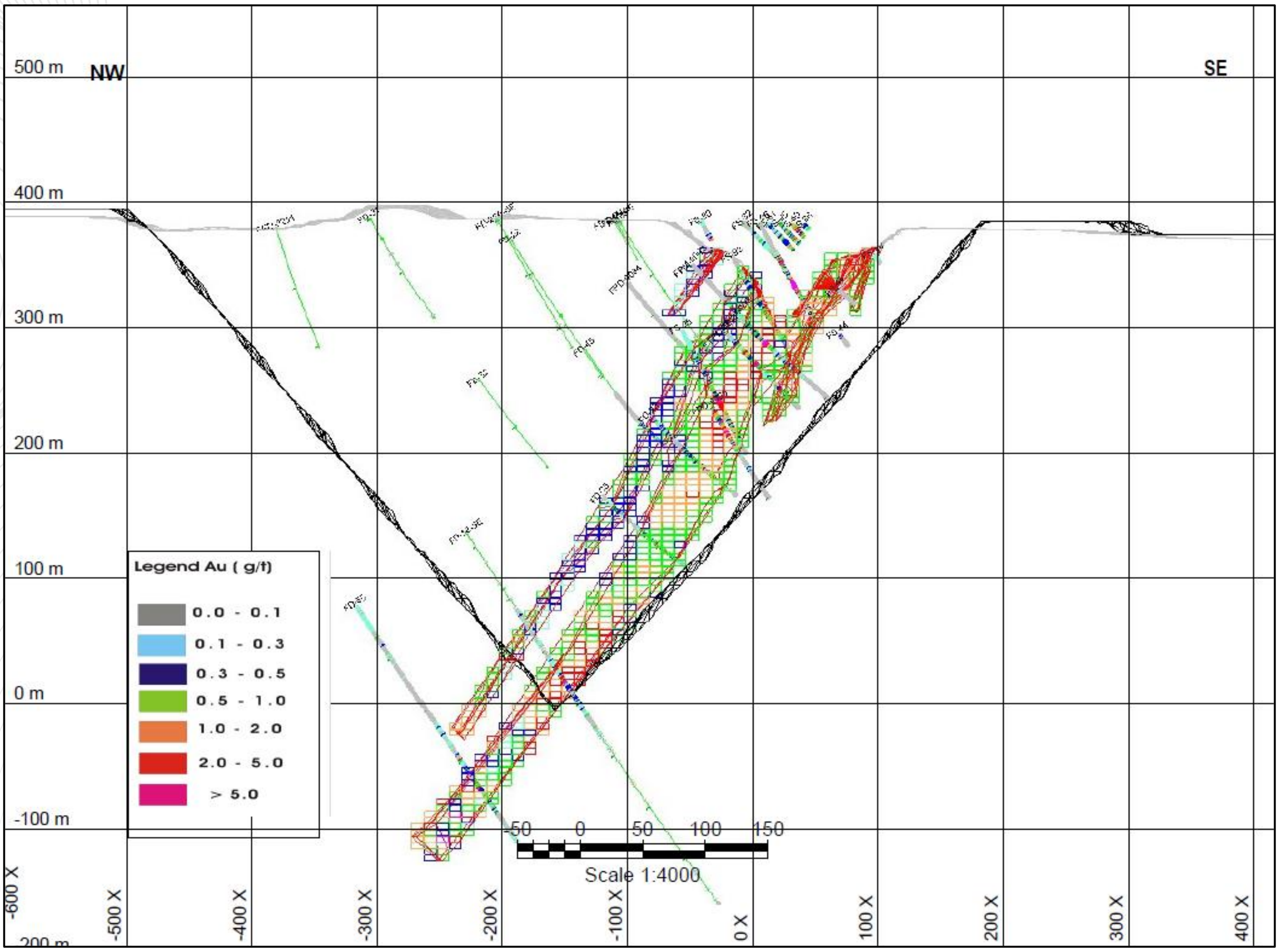
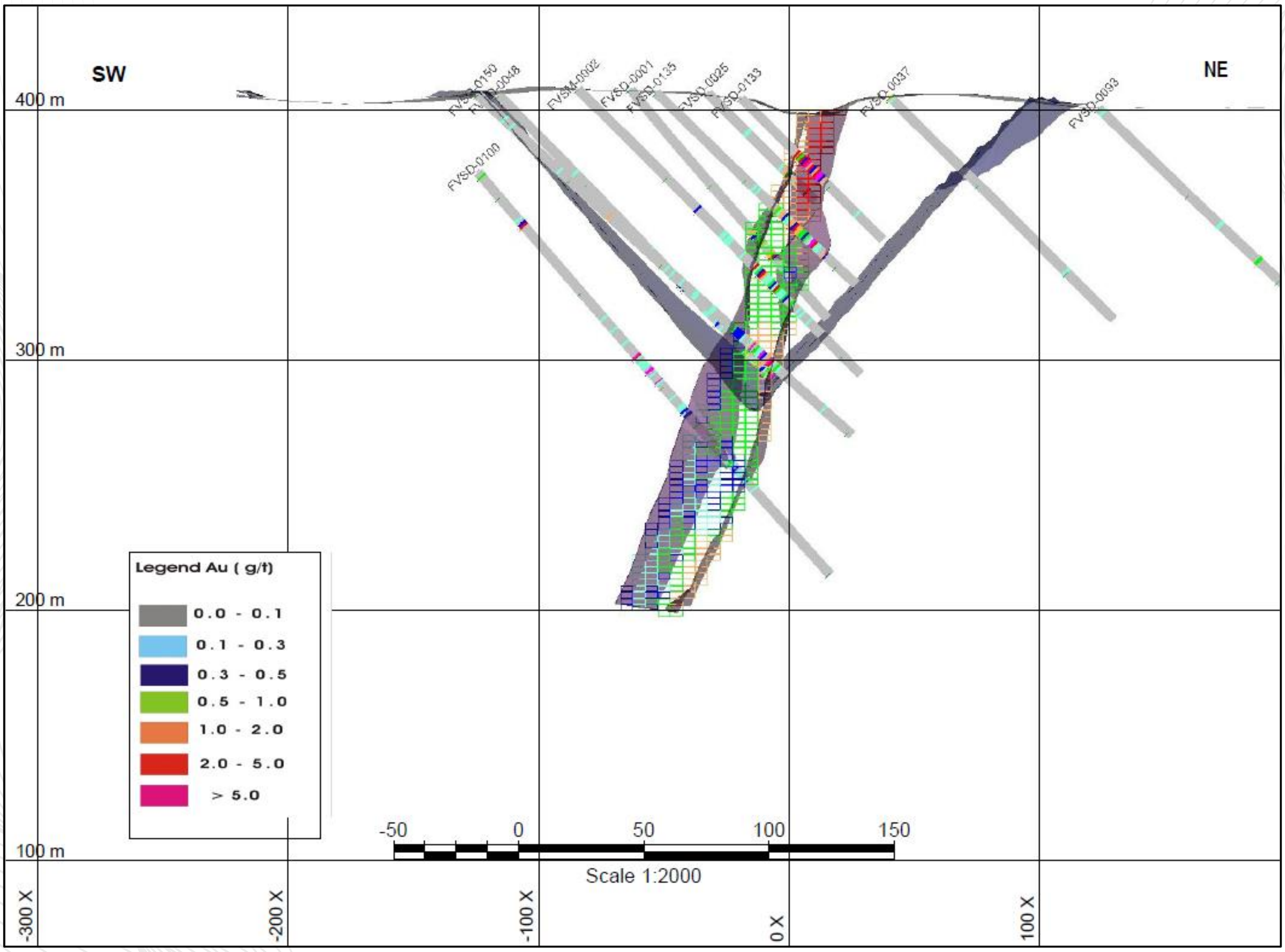


Figure 14-44 Typical Cross-Section (Vira Saia)-Block Model vs. 1,800 $ Pit shell

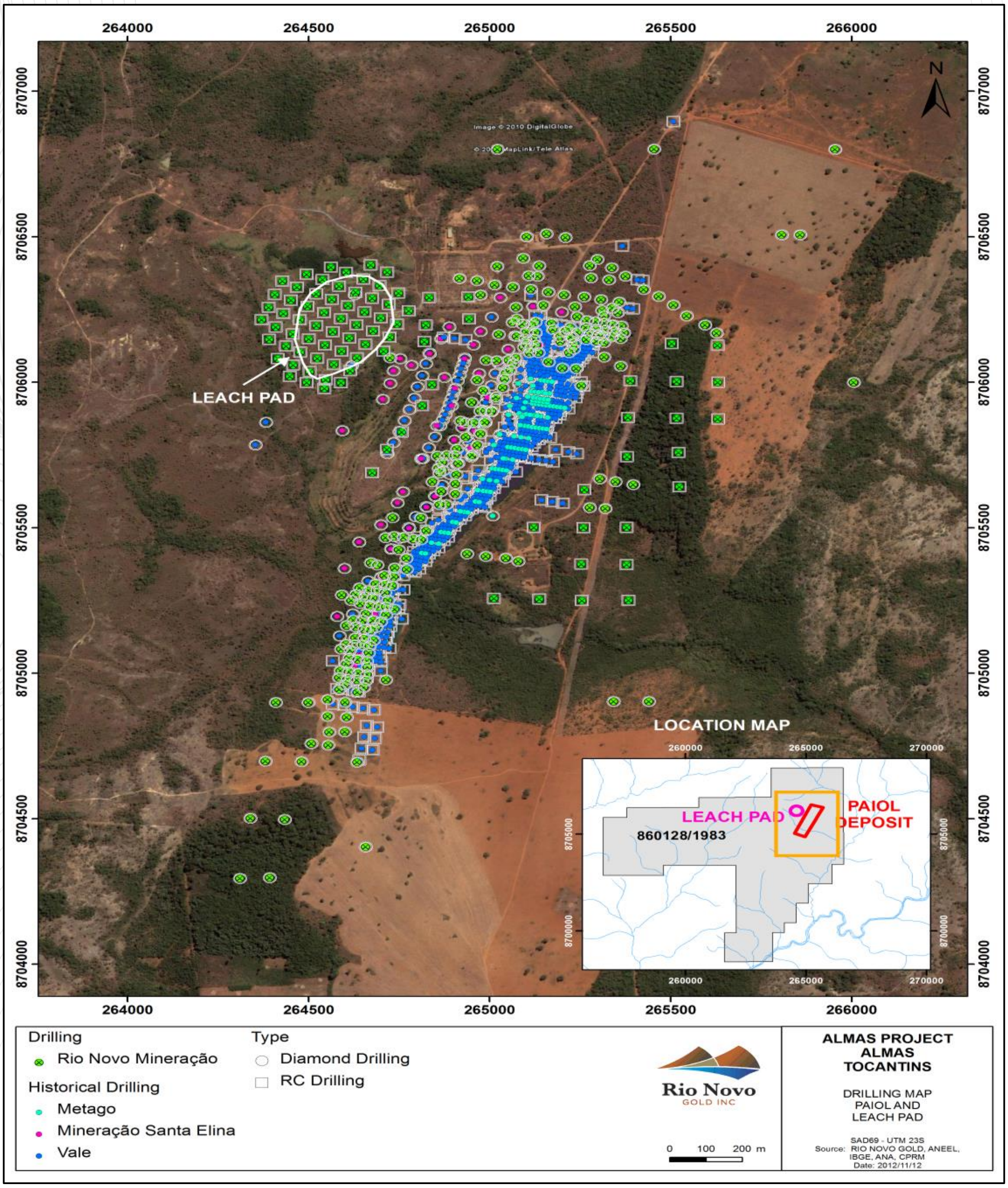


14.15 PAIOL HISTORICAL HEAP LEACH PAD (HLP) MINERAL RESOURCE ESTIMATE

The Paiol Historical Leach Pad (HLP) is a remnant of the Vale operation in the 90's when ore was first leached, and then low-grade tails dumped to a site location at east side of the old Paiol waste dump (Figure 14-45). During 2010, the HLP was drilled by Rio Novo for the purpose of resource estimation and a PEA report. The first resource estimate was prepared by Geosim Consultants from Vancouver, Canada (2011).

The details of the drill hole information were discussed in sections 10 and 11 of this report. In addition, Micon did a comparative study between Auger and RC holes to verify use of different perforation techniques and the results of study are shown in section of 12.5.5 of this report.

Figure 14-45 Paiol Leach Pad Area Location and boundaries & all Paiol drill collar locations



The bottom surface of the Leach pad model was created (Micon, 2019) after identifying a barren white silty layer in bottom of all drilled RC holes by Aura (Figure 14-46). Using current topo surface and the limiting bottom surface, Heap Leach Pad material was constrained between two surfaces to prepare for an updated resource estimate.

Figure 14-46 A RC Hole in Heap Leach Pad Showing Barren Layer at Bottom of Hole

Results from 92 RC holes completed in 2010 along with 166 auger holes were used to estimate the gold content of the historical Heap Leach Pad. Figures 14-47 and 14-48 illustrate the extent of the area and the drill hole coverage which was on approximately 25-m. Figure 14-19 shows a cross-section through the Paiol Heap Leach Pad showing different depths of perforations.

Figure 14-47 Auger and RC drilling – Paiol Leach Pad Area

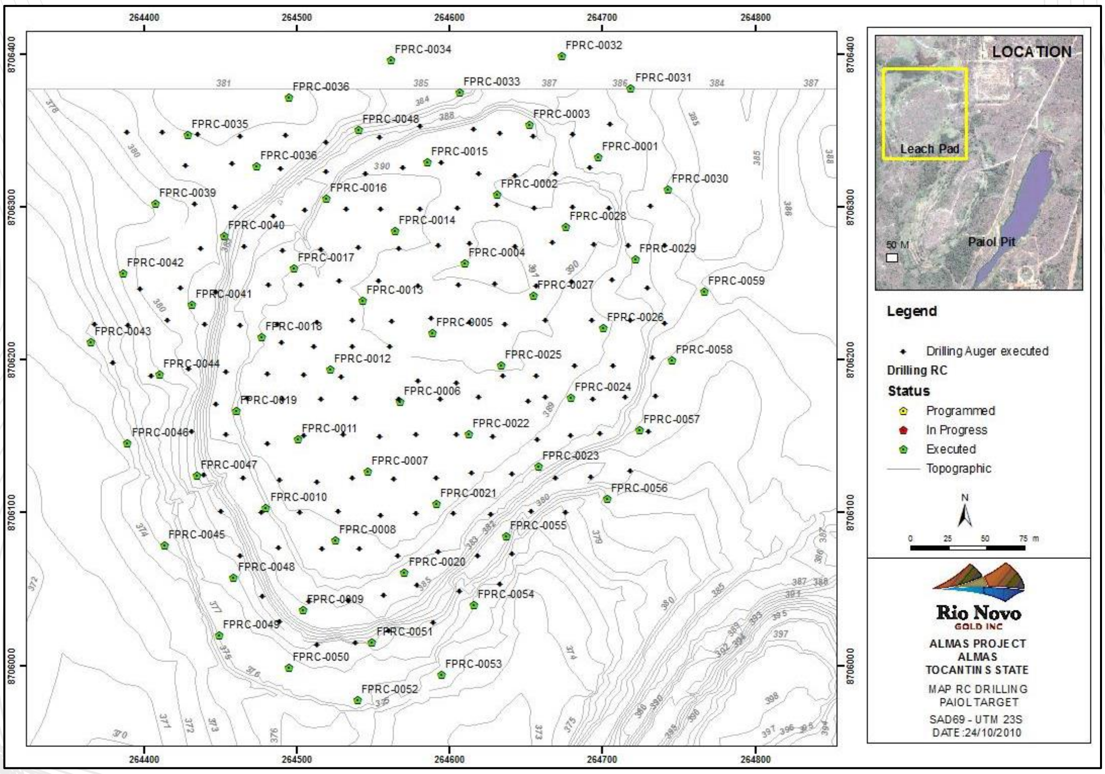
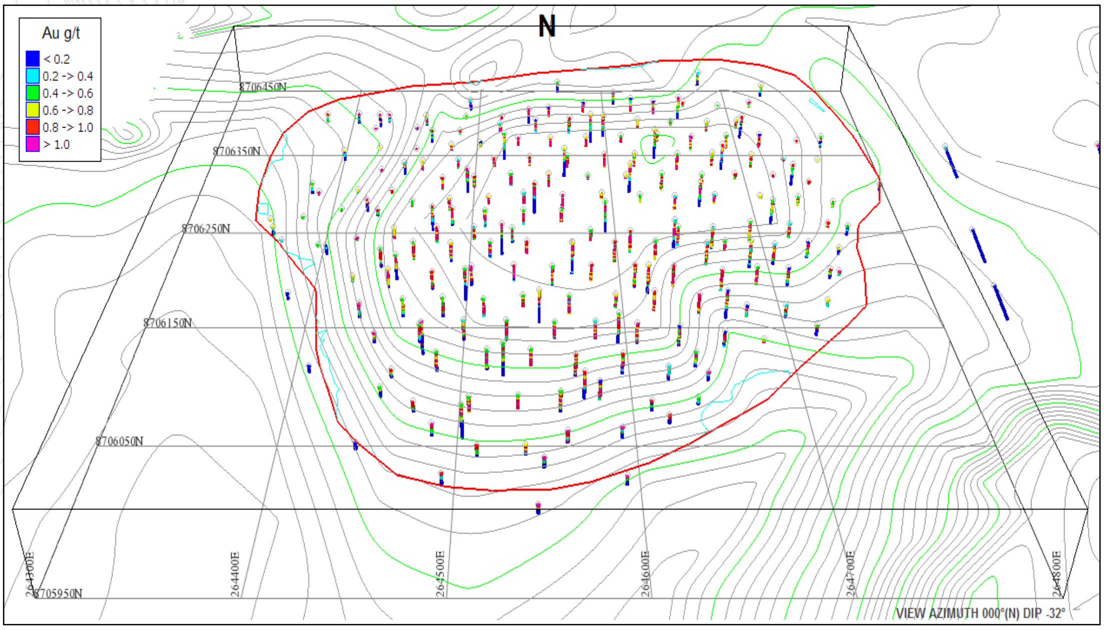


Figure 14-48 Auger and RC drilling – Paiol Leach Pad Area (R.Simpson, 2011)



Bulk density measurements were carried out in February, 2011 by excavating 5 pits within the pad area with dimensions of 2 x 2 x 1.5 m. The material was weighed using a HORAUS precision balance with a capacity of 2,500 g. A total of 59.7 tons of material were excavated from a volume of 30 m^3 giving an average dry density of 1.78 t/m^3. The moisture content was also determined using humidity cells at 10.84%. The density of the wet material averaged 1.99 t/m^3.

A total of 1,414 assay samples for Au on 1 m intervals was used for estimation. Au statistics from samples taken above the base of the dump are shown in figure 14-48.

A top cut value of 2.00 g/t was selected for capping of high-grade outliers (approximately 98% Percentile) to limit the influence of outliers (Figure 14-49).

Figure 14-49 Auger and RC drilling in Cross Section From Paiol Heap Leach Pad (HLP)

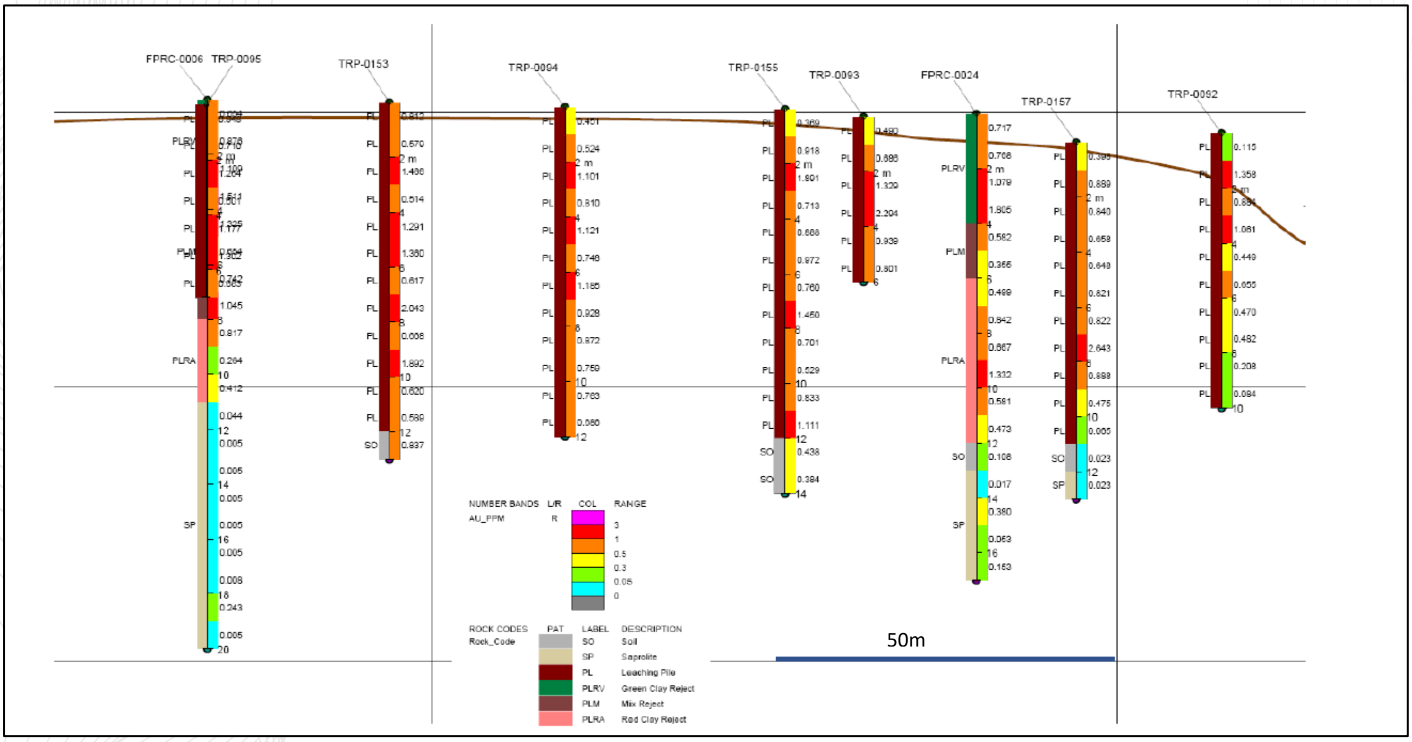


Figure 14-50 Auger and RC assays Histogram

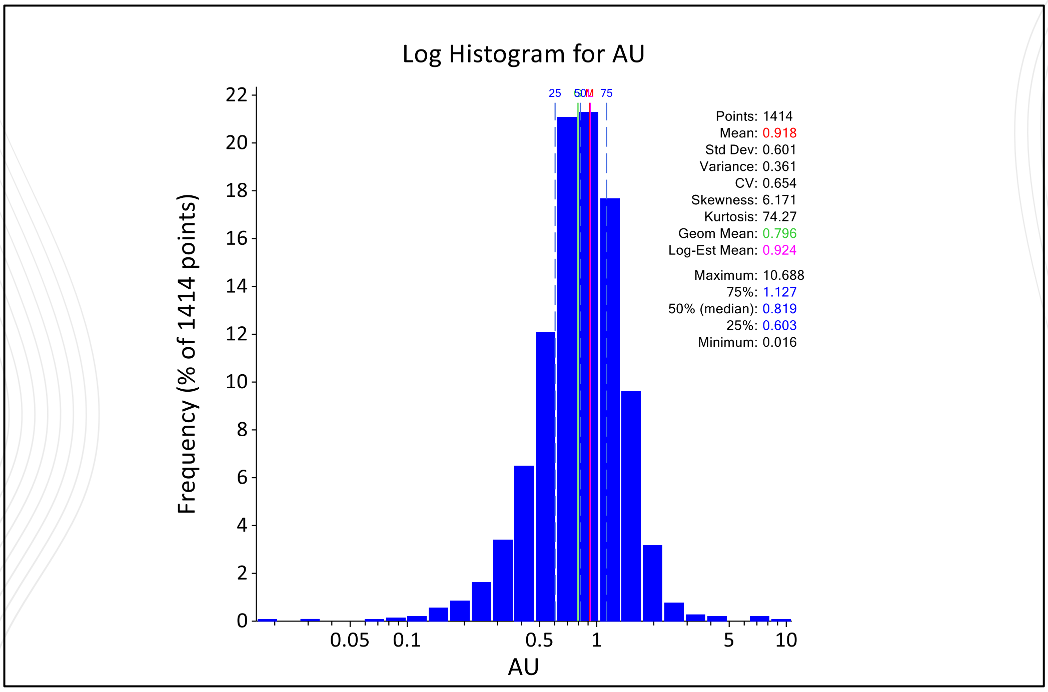
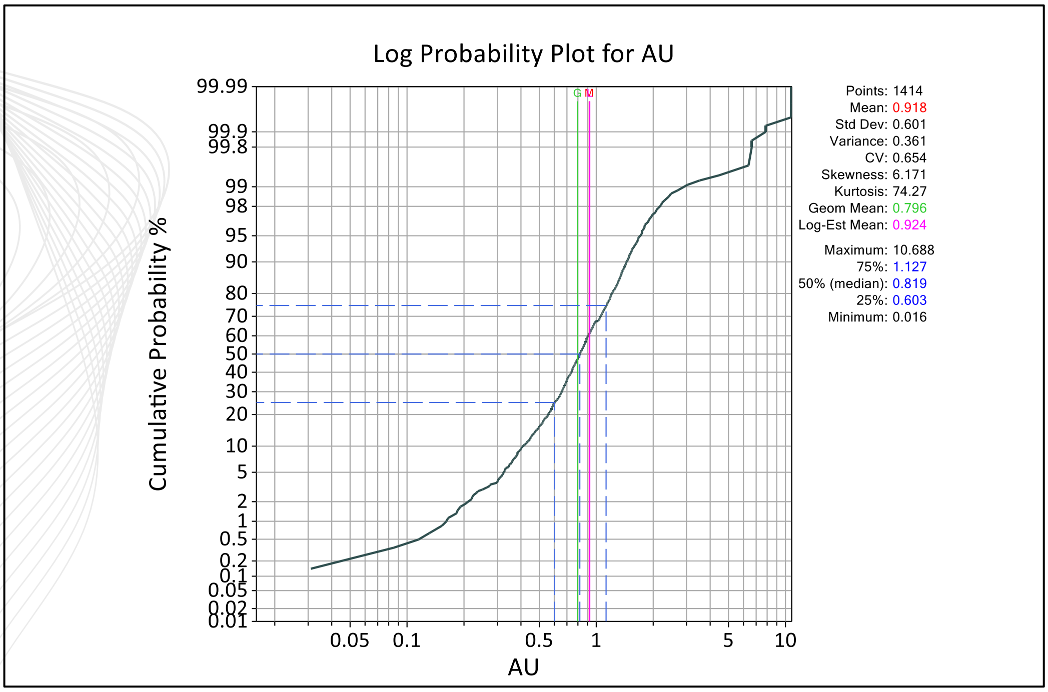


Figure 14-51 Auger and RC Assays Log Probability Plot



A non-rotated block model was created in Gemcom software using a parent block size of 5 x 5 x 2 m. The block model set up is summarized in Table 14.27.

Table 14-29 Block Model Definition (Paiol Heap Leach Pad)

	EASTING (X)	NORTHING(Y)	ELEVATION (Z)
Min	264300	8705900	370
Max	264900	8706500	400
Dimensions	600	600	30
Block Size	5	5	2

Block grades were estimated by the ID^2 method with a minimum of 3 and maximum of 12 composites and maximum of 2 composites per hole required. The maximum horizontal search distance was 50 m, but the vertical search was limited to 2 m in order to simulate the build-up of the dump. The block grade distribution for the ID^2 grade estimation frequency distribution is shown in figure 14-50. Block grades ranged from 0.069 to 1.99 g/t Au and averaged 0.84 g/t.

Blocks within Pad area classified to Indicated based on a horizontal maximum search of 50 m distance. The areas outside of the 50m search zone were also interpolated but not accounted for in any resource category (inventory).

Figure 14-52 Block Model Au Grade Distribution in Paiol Heap Leach (HLP)

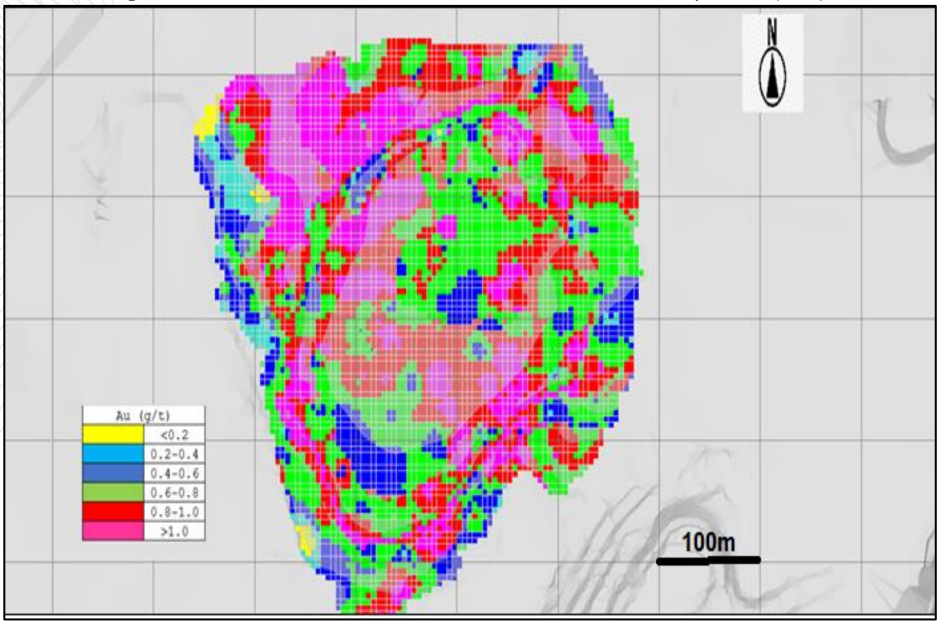


Table 14-30 shows estimated tonnes and grade for the Paiol Heap Leach Pad using both wet (1.99 t/ m^3) and dry (1.78 t/ m^3) densities excluding inventory blocks.

Table 14-30 Paiol Heap Leach Pad Mineral Resources Update (Indicated)

HLP WET DENSITY				HLP DRY DENSITY		
PAIOL	TONNES	Au (g/t)	Au (ounces)	TONNES	Au (g/t)	Au (Oz)
Leach Pad	1,688,245	0.88	47,717	1,510,090	0.88	42,680

Note:
1. The Mineral Resource estimates were prepared in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by the CIM Council on May 10, 2014, and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, adopted by CIM Council on November 29, 2019, using geostatistical and/or classical methods, plus economic and mining parameters appropriate to the deposit.
2. Contained metal figures may not add due to rounding.
3. Surface topography based on December 31st, 2016.

Using the wet bulk density determination of 1.99 t/ m^3 the dump is estimated to contain an indicated stockpile resource of 1.69 million tonnes averaging 0.88 g/t Au. The grades were calculated on dried material, so the final average grade is diluted by the moisture content of 10.5%.

14.16 COMBINED MINERAL RESOURCE STATEMENT

For purpose of Feasibility study only Measured and Indicated resources (M&I) can be considered for pit optimization, reserve disclosure and detail mine planning . Table 14-31 is showing combined Measured and Indicated resources for Almas Project as of December 31, 2020.

*Table 14-31 Almas Gold Project Mineral Resources (M&I)**

ALMAS GOLD PROJECT M&I RESOURCES				
DEPOSIT	CUT-OFF	TONNES	Au (g/t)	Au (Oz)
Paiol	0.29	17,548,140	0.98	552,460
Cata Funda	0.34	838,000	1.72	46,460
Vira Saia	0.31	3,354,690	0.96	103,845
Heap Leach Pad (HLP)	-	1,510,090	0.88	42,680
Total		23,250,920	1.00	745,445

Note:
1. The Mineral Resource estimates were prepared in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by the CIM Council on May 10, 2014, and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, adopted by CIM Council on November 29, 2019, using geostatistical and/or classical methods, plus economic and mining parameters appropriate to the deposit.
2. The Mineral Resource Estimate is based on an updated optimized shell using 1800 $/oz.
3. Contained metal figures may not add due to rounding.
4. Surface topography based on December 31st, 2016.
5. Mineral Resources are inclusive of Mineral Reserves.
6. Mineral Resource estimate for Cata Funda deposit was prepared by Adam Wheeler, C.Eng. a Qualified Person as that term is defined in NI 43-101.
7. The Mineral Resource estimate for the Paiol and Vira Saia deposits and HLP were prepared Farshid Ghazanfari, P.Geo. as a Qualified Person as that term is defined in NI 43-101.

14.17 INTERPRETATION AND CONCLUSIONS

The Almas Gold Project deposits are developed within a mylonitic shear hosted zone in a high greenschist to low amphibolite metamorphic terrain. The Paiol deposit strikes NE-SW and dips typically about 60º to the NW. On average, the zone is about 40 m wide. The Vira Saia deposit strikes NW-SE and dips typically about 75º to the SW. The Cata Funda deposit strikes NW-SE with dips averaging 55º to the SW.

Alteration is dominated by silicification, sericitization, K-feldspar, pyritization and flooding of Fe-carbonates. Gold is positively correlated with the intensity of silicification and Fe-carbonatization. Gold occurs as free gold and as gold inclusions within sulfide minerals. The stronger gold mineralization is controlled by structures and associated with the main shear zones in each deposit.

The dominant alteration features which were identified by Rio Novo geologists are logged as distinctive lithological units in the database. Resource wireframes are constrained by structural controls(shear zone) and a distinctive litho-altered unit. Some secondary shear zones in HW and FW were also modelled using 0.3 g/t cut-off grade.

Both the Paiol and Vira Saia 3D models were updated for the purpose of this report, while the Cata Funda 3D model was kept the same compared to the model used in previous feasibility study (RPM , 2016).

For Paiol, the modelling approach is significantly different compared to previous estimates which included a large envelope of low grade (<0.3 g/t) around the main shear zone structures. The previous consultant (RPM) classified this zone to inventory and estimated the grade within the large envelope with a hard boundary called a main zone which was not constructed by any specific grade shell. The drawback of this approach was creating lots of composites to deal with in data exploratory analysis which basically created a false continuity for the deposit model controlled by a shear zone. The only advantage of this approach was to have a grade shell for waste materials that can help to adjust the dilution.

Despite this fact the continuity of the main mineralized zones in all three deposits is significantly better than other shear hosted zone in similar belts (deposits in same belt and in the Goias Greenstone Belt) in Brazil. Dominance of sericite and carbonate is main key indicator for this continuity which in most cases is associated with grade. In Paiol and Cata Funda, mineralization and waste can easily be identified within the greenschist unit with an orange-brown color (mineralization) that resulted from ankerite compared to the green color in greenschist.

In addition, within the FW and HW of current model, there are some minor shear zones that can be identified and modelled in future. These minor shear zones are usually developed within sericite-chlorite-ankerite schist (SCDX) and sericite- ankerite-chlorite schist (SDCX) but are lacking continuity along strike and down-dip.

The Almas drill hole database contains different types of drilling in different time periods, possibly with drilling qualities, including diamond drill holes (DDH) , RC holes , RC hole for Production (Paiol-Vale RC) holes and auger holes(for the HLP). Rio Novo did some validation and drill hole twinning for some old historical holes. The RC production holes in Paiol probably were the lowest in terms of quality at the time. Visual inspection of RC production holes shows some degree of gold smearing which reflects lower quality of drilling. For all deposits all exploratory analysis and resource classification was done based on DDH and no RC holes were used. For purpose of grade estimation, composites resulted from RC holes were not used for Cata Funda and Vira Saia was only drilled by DDH.

Micon in 2019 did a comparative study between RC holes and DDH and the results of this study is discussed in section 12 of this report. The study showed there was not much difference between mean grade of the RC holes and DDH and using them does not create bias for the mineral resource estimate globally. In addition, Aura ran the estimate using both historical RC holes, and without them, and did not find much difference in global tonnes and grade. It was concluded that using RC holes does not create bias in grade-tonnage curve and they can be used for purpose of mine planning.

Most of Vale's RC production holes are mined out and this has minimum effect on current model, although in some areas it helps to have better grade estimation locally in the walls of the current pits.

It is the opinion of the author of this section of the report that using a combination of RC and DDH holes for the resource model creates a more robust grade distribution locally. For that reason, the data were included in the final model for mine planning. However, they were not used for variography and resource classification.

Table 14-32 shows the changes in Combined Mineral Resources comparing the previous feasibility study report (RPM, 2016) with the current updated resource estimate for the Measured and Indicated categories, for all three deposits.

Table 14-32 Changes in Mineral Resources (M&I)-2016 vs.2020

DEPOSIT	2016 FS STUDY (Oz)	2020 FS STUDY (Oz)	DIFF (Oz)	DIFF (%)
Paiol	667,391	552,460	-114,931	-20.80%
Cata Funda	43,938	46,460	2,522	5.43%
Vira Saia	87,562	103,845	16,283	15.68%
Paiol Leach Pad (HLP)	46,753	42,680	-4,073	-9.54%
Total	845,643	745,445	-100,198	-13.44%

Note that the cut-off grade difference is negligible for Cata Funda and Vira Saia models in both studies. The decrease in ounces in Paiol comes from overestimation of tonnes in the RPM model due to the broad mineralized envelope without an economic cut-off grade constraint and partly is related to a change in cut-off grade from 0.25 g/t Au in 2018 to 0.29 g/t Au in this report. The decrease in ounces in HLP mainly comes from reported grade in RPM study using wet density.

The opinion of authors of this section of report is that the Almas Project Mineral Resource Estimate reported herein is appropriate to support a current feasibility study.

15 MINERAL RESERVE ESTIMATION

15.1 INTRODUCTION

Aura Minerals Inc. (Aura), through its subsidiary in Brazil, Rio Novo Mineração Ltda. (RNM) retained EDEM Engenharia de Minas (EDEM) to complete the mining component of a Feasibility Mining Study (FS) for the Almas Gold Project, located in the State of Tocantins, Brazil.

The Almas Gold Project is formerly owned by Rio Novo, a mining company that was acquired by Aura in 2018. The Project is located to the south of municipality of Almas, approximately 300 km southeast of Palmas, capital of the State of Tocantins, and 45 km west of Dianápolis, a regional commercial center.

The Project consists of three hydrothermal gold deposits, named Paiol, Vira Saia and Cata Funda, which will be mined by open pit mining method. Between 1996 and 2001, the Paiol pit was operated by Vale S.A. and produced 86,000 ounces of gold.

Rio Novo developed multiple engineering studies targeting 2.0 Mtpa through a CIL gold plant. But, in 2018 Aura decided to downsize the current plant capacity and engaged Ausenco to perform a Preliminary Economic Assessment (PEA) based on 1.3 Mtpa.

In 2019, Aura appointed SRK to evaluate several production alternatives: 1.0 Mtpa, 1.3 Mtpa and 1.6 Mtpa, and develop a detailed mine planning study at a PFS level for the selected option. The alternative selected was 1.3 Mtpa.

The mining component is summarized in Section 15 and Section 16 of this report including:

- Data validation:
- Pit optimization with Lower intermediary slope angle.(45 degree)
- Mine scheduling: detailed as follows:: the mining pit design-by-quarter basis and monthly calculated in the first two years production; after the third year the pit design and calculation by annual basis.
- Mining Method.
- Estimates of the fleet and manpower required to achieve the plan.
- Mining operating and capital costs.
- Paiol existent pit drainage legal authoriztaion water pumping volume to the natural drainage.

15.1.1 MINERAL RESERVES

The bases and procedures for mineral reserves estimation for the Almas Gold Project, including the three mineral deposits: Paiol, Cata Funda and Vira Saia is presented in this chapter. In addition, the reclamation of a Heap Leach pad generated during the Vale's operation time was included. Table 15-1 summarizes this mineral reserve estimate.

*Table 15-1 Almas Gold Project Mineral Reserve**

ALMAS RESERVE		Tonnage (t)	Au (g/t)	Au (Oz)
PAIOL	PROVEN	5,357,974	0.89	152,683
	PROBABLE	10,780,501	0.88	304,446
	TOTAL	16,138,475	0.88	457,129
CATA FUNDA	PROVEN	438,612	1.89	26,711
	PROBABLE	250,163	1.79	14,412
	TOTAL	688,775	1.86	41,123
VIRA SAIA	PROVEN	646,016	0.88	18,363
	PROBABLE	3,134,066	0.91	91,758
	TOTAL	**3,780,082**	**0.91**	**110,121.6**
GRAND TOTAL		**20,607,332**	**0.92**	**608,373.1**
HEAP LEACH STOCKPILE	PROVEN	-	-	-
	PROBABLE	1,275,233	0.90	36,900.3
	TOTAL	**1,275,233**	0.90	36,900.29

*Note:
1. The Mineral Reserve estimates were prepared in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by the CIM Council on May 10, 2014, and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, adopted by CIM Council on November 29, 2019, using economic and mining parameters appropriate to the deposit.
2. The Mineral Reserve Estimate is based on an updated optimized shell using 1,500 $/oz gold price, average dilution of 20% , mining recovery of 100% and break-even cut off grades of 0.29 g/t Au for Paiol, 0.31 g/t Au for Vira Saia and 0.34 g/t Au for Cata Funda.
3. Contained metal figures may not add due to rounding.
4. Surface topography based on December 31st, 2016.
5. Mineral Reserve estimate for Almas Gold Project was prepared under the supervision of Luiz Pignatari, P.Eng. as a Qualified Person, competent to sign as defined by NI 43-101.
6. Heap Leach Pad ore was classified as a probable reserve because of the long-time stockpiled presents uncertainty in the metallurgical recovery

15.2 DOCUMENTS AND INFORMATION PROVIDED

This study is based on information supplied by Aura and included:

- Economic feasibility study of 2013.
- 2016 economic feasibility study.
- 2019 economic pre-feasibility study.
- Geometric stability studies.
- Exploration Drilling database.
- Block model from the last economic pre-feasibility study of 2019.
- Tonnage feed plant plan.
- 3D files referring to the production plan.

- Topographic surface
- 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 unit cost and salaries
- Exchange and discount rates
- Technical reports

15.3 GEOTECHNICAL STUDIES

Several geotechnical studies were done in the Feasibility Study conveyed by Rio Novo Mineração on August 2016 involving Fundação Luis Englert, ACZ Engenharia e Mineração and BVP Engenharia They are the basis for several engineering studies, reviewing the existing geotechnical and hydrogeological studies for mineral deposits. It can be listed:

- Description of the geological survey cores (logging and photographs).
- Results of geotechnical laboratory tests, including uniaxial, tri axial and diametrical compression tests, shear tests and physical index and permeability.
- Slope stability analysis, performed independently
- Hydrogeological studies developed

Considering the parameters presented below:

15.3.1 GEOMETRIC PARAMETERS

A summary of the geometric parameters to be used is presented in table 15-2.

Table 15-2 Geometric Parameters for Mining Simulations

PIT	GROUP	SECTOR	BENCH HEIGHT (m)	BENCH WIDTH (m)	BENCH ANGLE FACE (°)	INTER-RAMP (toe-to-toe)	OVERALL[1] (toe-to-toe)	BASIS
PAIOL	SOIL/SAP	All	10	6	45	32	35	FLE (2012)
	WEATHERED	All	10	6	45	35	35	
	FRESH	HW	10	3	80	64.5	55[2]	
		FW	10	3	70	56.5	50[2]	
CATA FUNDA	SOIL/SAP	All	10	6	45	32	35	FLE (2012)[3]
	WEATHERED	All	10	6	45	35	35	
	FRESH	HW	10	3	80	64.5	55[2]	
		FW	10	3	70	56.5	50[2]	
VIRA SAIA	SOIL/SAP	All	10	6	27	21	24	BVP (2012)
	WEATHERED	All	10	6.5	45	31	35	
	FRESH	HW	10	6.5	85	53.5	52	
		FW	10	6.5	80	50.5	49	

Note:

1. General slope angles to be applied for pit optimization.

2. Ramps or safety shoulders with a width of 15 m were considered for the mean slope angle of the pit.

3. The FLE study suggested a double bench operation (20 m high) if the volumes and the mean pit slope angles are met.

4. The average slope angles of the optimized Vira Saia pit were estimated considering a depth of 100 m inside fresh rock and two ramps 15 m wide each.

15.3.2 PAIOL'S PIT

a. Rock mass fractures degree

The Paiol rock masses have low fracturing intensity, so that no significant problems are expected for mining operations. The volume of estimated groundwater entering in the pit does is not expected to affect the stability.

b. General Pit slope stability

The global slope stability analysis of the Paiol pit considered the geotechnical model for four sections, and the geological model of the diagram block; The Hanging wall (HW) slope inclination angle at 55 ° was initially analyzed for circular rupture and the safety factors for the various vertical sections were calculated as shown in figure 15-1 and presented in table 15-3, follow:

Figure 15-1 The Final Pit Perpendicular Plans To Determine Slope Angle Safety Factors, With Slide 5.0 Software Analytical Methods (Circular Rupture)

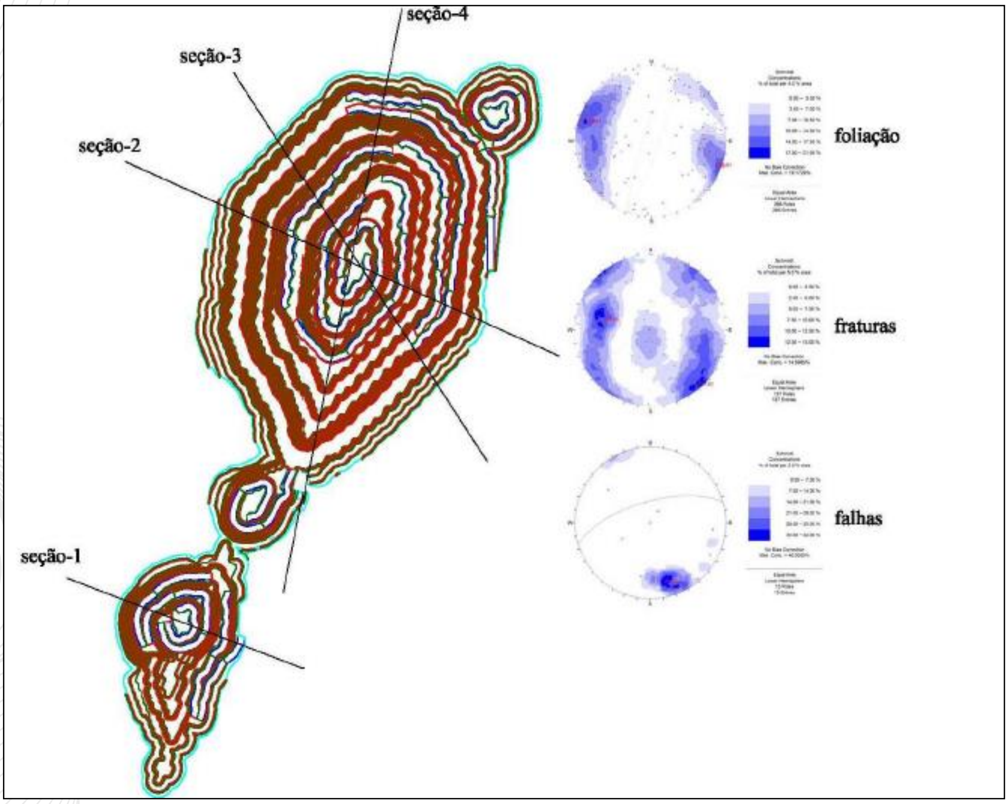


Table 15-3 The Final Pit Perpendicular Plans To Determine Slope Angle Safety Factors, With Slide 5.0 Software Analytical Methods (Circular Rupture)

SECTION	SLOPE HANGINGWALL	SLOPE ANGLE (°)	SAFETY FACTOR (SF)	FOOTWALL ANGLE	SLOPE ANGLE	FS
1	Soil	45 °	1.3	Soil	45 °	1.1
	Fresh Rock	55 °	1.7	Fresh Rock	60 °	2.2
2	Soil	45 °	1.1	Soil	45 °	1.1
	Fresh Rock	55 °	1.6	Fresh Rock	52 ° - 61 °	1.6
3	Soil	45 °	1.1	Soil	45 °	1.1
	Fresh Rock	55 °	2.9	Fresh Rock	55 °	2.2
4	Soil	37 °	1.1	Soil	37 °	2.5
	Fresh Rock	55 °	2.79	Fresh Rock	55 °	1.9

c. Safety Factors

The safety factors, in EDEM's opinion, are suitable for the slope, mainly for waste rock and competent rock, even for a height of 300 m.

Soil has a Safety Factor slightly lower than recommended (SF ≥1,2),but normally the soil there doesn't have a thick layer and for this reason we considered possible to control during the operation time..

d. Dip angle of the mineralized zone

The approximate dip of the mineralized zone is 50 °, according to the sections presented below. Based on this geometry, the Footwall slope angle (FW), to the right of the analyzed sections, should follow, as much as possible, the contact between the ore and the waste, ensuring a maximization use of the deposit.

Figures 15-2 to 15-5 adequately illustrate the profiles defined in figure 15-1.

Figure 15-2 Section 1, Hangingwall Numerical Model (HW - left, FS = 1.26) and Footwall (FW - right FS = 1.41), Massive Without Discontinuities.

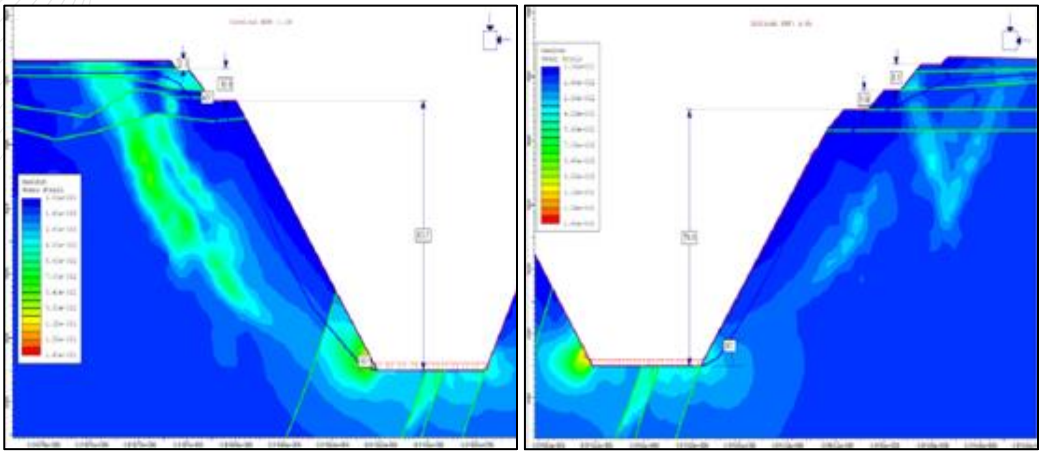


Figure 15-3 Section 2, Hangingwall Numerical Model (HW - left) and Footwall (FW - right), Solid Without Discontinuities, FS = 1.25.

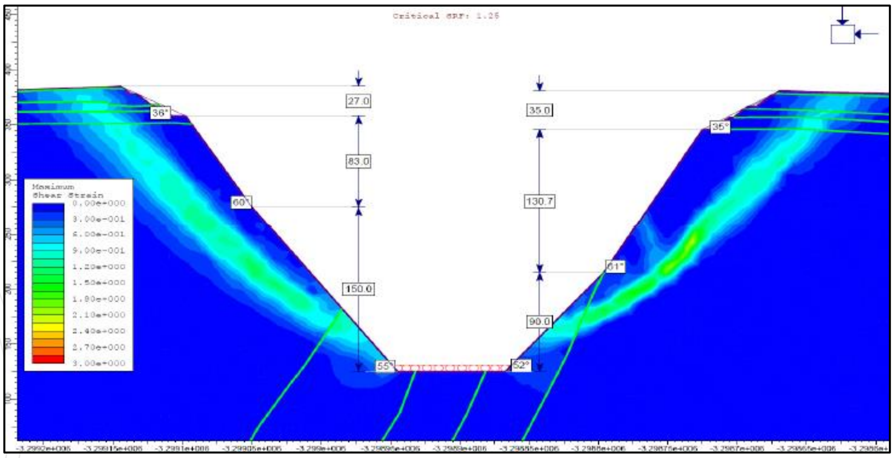


Figure 15-4 Section 3, Hangingwall Numerical Model (HW- left) and Footwall (FW on the right), Massive Without Discontinuities, FS = 1.32.

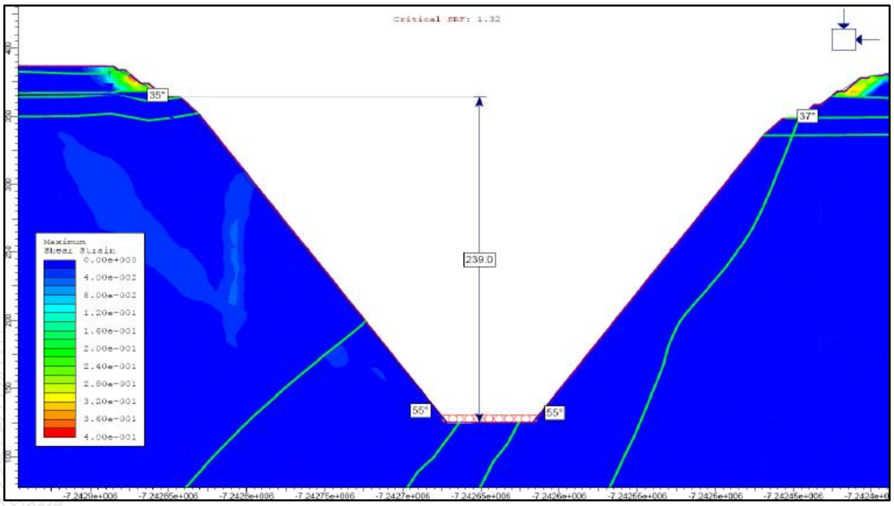
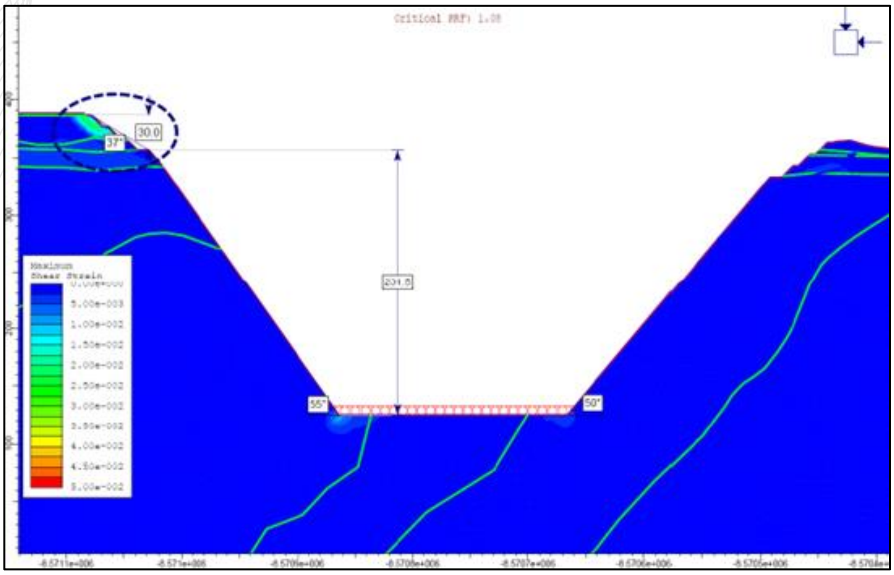


Figure 15-5 Section 4, the Hangingwall Numerical Model (HW on the left) and the Footwall (FW on the right); Rock Mass Without Discontinuities, FS = 1.08 for the Soil And Without Relevant Deformation in the Rock Mass.



By the time of mine development more information assessment will provide more safety subsidies to make a more complete geotechnical work to investigate opportunities to improve and minimize the mining costs. The slope faces will be exposed for mapping and sampling, either for the hangingwall (HW) or for the Footwall (FW). It is important to carry out this assessment to have the information for the Paiol mining operation after the third / fourth mining years.

EDEM understands that the information contained in the existing geotechnical studies are enough to start mining operations. The work already carried out indicates that the slopes will be safe in terms of angle and height. It is expected that an experienced geotechnical team, as part of the technical staff of the mine, will routinely follow the operation, indicating specific areas where slope face reinforcement would be needed, especially in the hangingwall slope (FW) in contact with the orebody. As the wall slopes will be exposed, mapping and sampling the most unstable regions is going to be more favorable.

15.3.3 CONSIDERATIONS FOR VIRA SAIA’S PIT

The structural framework of the Vira Saia Pit region was defined based on the survey of structural measures in two oriented holes, defined as: FVSE-01 and FVSE-02 (Table 15-4 and Figure 15-7).

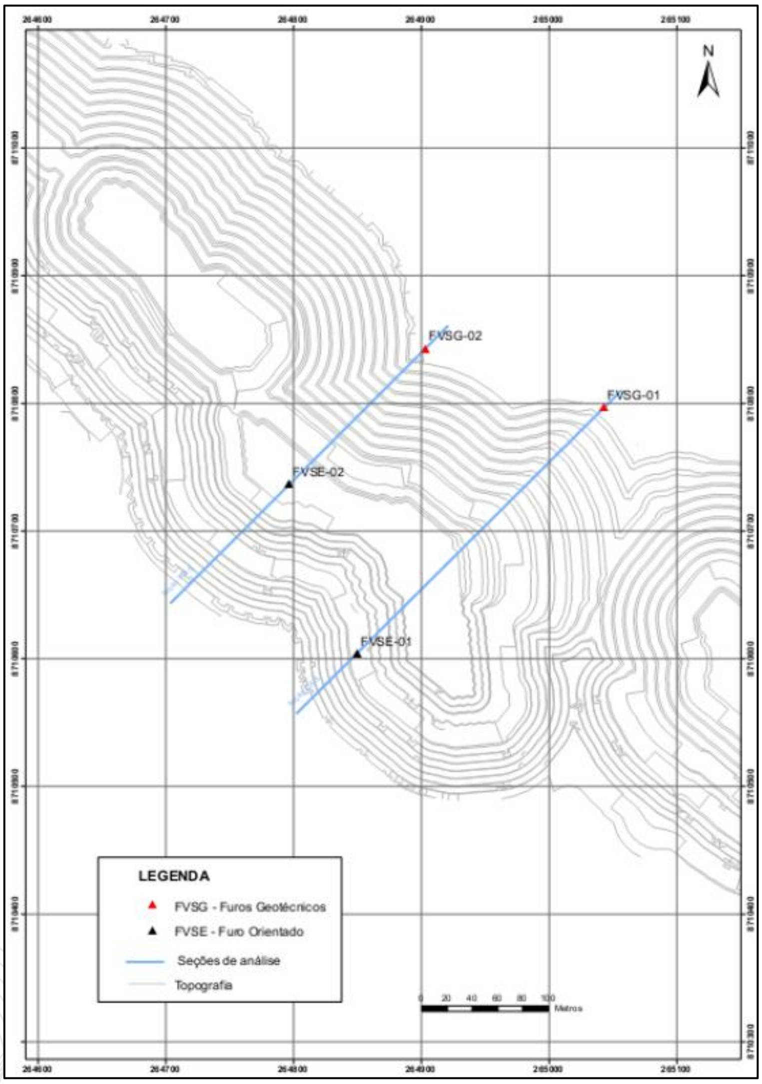
Table 15-4 Described Geotechnical Oriented Drill Holes for Vira Saia

TYPE	COMPANY	HOLES	AMOUNT (m)	SAMPLES	AVERAGE DEPTH	SERIES
Oriented Holes	Rio Novo	2	288.15	0	144.07	FVSE-0001 to 0002
Geotechnical Hole	Rio Novo	2	100.75	0	50.37	FVSG 0001 to 0002
Total Drilling		**4**	**388.90**	0	**97.22**	

The data were treated in Rocscience's Dips program, starting, as already mentioned in the methodology, from the determination of the alpha and beta angles, in addition to the azimuth and inclination of each maneuver drilling hole. It should be noted that, in the treatment of the data, the azimuth used for each hole was the same as measured on the collar, as the device used to orient the cores was not able to measure it. However, it is known that in the given geological context, the azimuth variation is negligible, thus, it does not significantly compromise the results obtained in the statistical treatment.

With the attitudes of discontinuities in mind, pole density stereograms were made to try to define families of fractures, failures, and the variation in foliation attitude.

Figure 15-6 Map of Oriented Drilling Locations in the Vira Saia Pit Region



With the results of the kinematic and stability analysis for the final rock slopes from Vira Saia Pit, the re-confirmation of the pit geometry was done by specific analysis, according to some important aspects:

- The slopes of the final pits were considered stable in terms of stability analysis by limit equilibrium, presenting safety factors well above 1.5 for global ruptures.
- Although there are no thick layers in the analysed sections, the slopes on the ground deserve special attention with maintenance of the angle of the face down, thus ensuring stability in the upper portion of the pit.
- The need to carry out systematic geological mapping simultaneously with the opening of the pit is also emphasized, in order to verify or not the potential of ruptures verified in the kinematic analysis, thus being able to adopt preventive and / or corrective measures.
- The importance of adopting controlled blasting operations is emphasized in order to minimize the damage to the rock mass, thus avoiding its more intense and deep opening according to the foliation plan and, consequently, reducing the potential for breaks at the bench level, mainly controlled by openings and damage to material discontinuities.

Table 15-5 Ultimate Pit Geometry Proposed for Vira Saia

FINAL PIT GEOMETRY PROPOSED BY BVP							
VIRA SAIA PIT							
Direction	**Group**	**Lithologies**	**Sector (1)**	**Maximum height of Slopes (m)**	**Minimum Berms Width (m)**	**Face Angle (°)**	**Inter-ramps Angle (°) 3**
Hangingwall (HW)	Fresh Rock	GD, GDP, DM-GDM	1	10.0	8.0	80 °	46 °
			2		5.0	80 °	56 °
Footwall (FW)	Fresh Rock	GD, GDP, DM-GDM	3	10.0	8.0	85 °	48 °
			4		5.0	85 °	60 °
			5 (2)		8.0	85 °	48 °
FW/HW	Soil - SAPR			10.0	6.0	27°	21°

15.3.4 CONSIDERATIONS FOR CATAFUNDA'S PIT

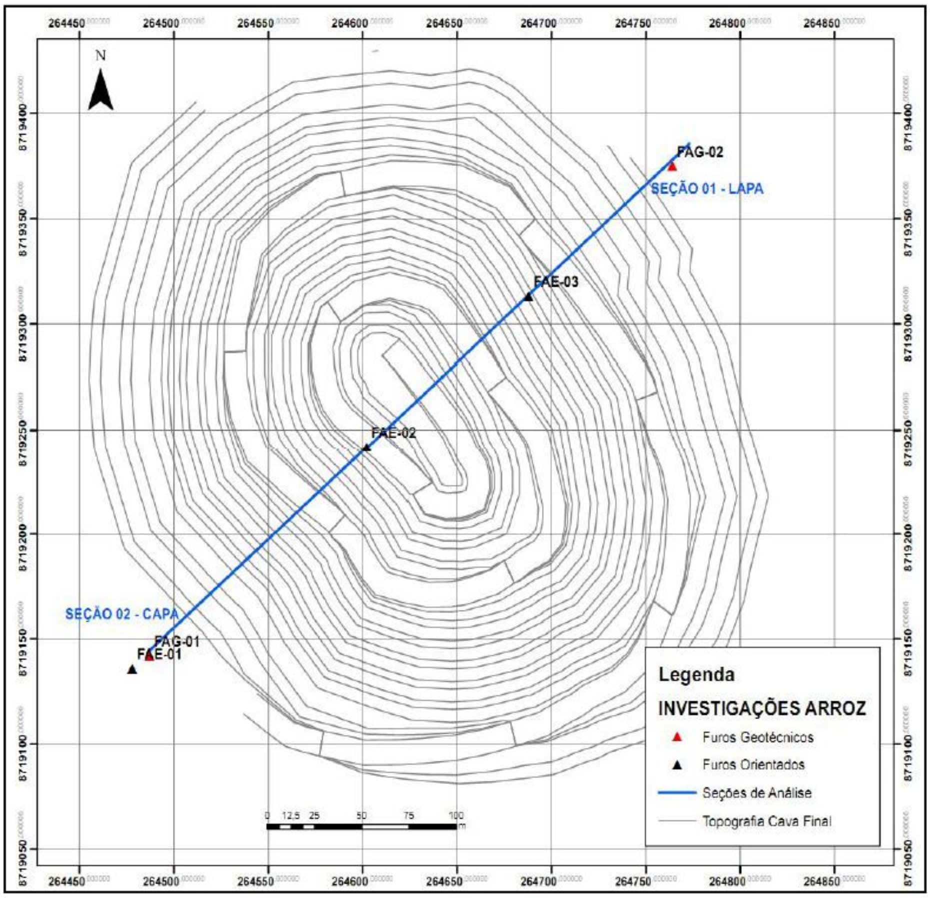
The structural framework of the region of the Cata Funda Pit was defined based on the survey of structural measurements in three oriented drilling holes: FAE-01, FAE-02 and FAE-03 (Table 15-6 and Figure 4.5)

Table 15-6 Described Geotechnical Oriented Drill Holes for Cata Funda

TYPE	COMPANY	HOLES	AMOUNT (m)	SAMPLES	AVERAGE DEPTH	SERIES
Oriented Holes	Rio Novo	3	492.65	0	164.22	FAE-0001 to 0003
Geotechnical Hole	Rio Novo	2	124.25	0	62.12	FAG 0001 to 0002
Total Drilling		**5**	**616.9**	**0**	**123.38**	

The attitude of these data was obtained in Rocscience's Dips program, starting from, as already mentioned in the methodology, the determination of the alpha and beta angles, in azimuth and inclination of each maneuver drilling hole. It should be noted that, in the data processing, the azimuth used for each drill hole, was the same as measured on the collar, since the device that made the orientation of the drilling holes, Reflex, does not measure the azimuth. However, it is known that, in the given geological context, the azimuth variation is negligible, therefore, compromises the results obtained in the statistical treatment.

Figure 15-7 Map of Oriented Drilling Locations in the Cata Funda Pit Region



Global disruptions indicated values of security considerably higher than expected. In view of this, aiming to optimize the safety factor in cuts in soil and consequently the geometry of the hangingwall slope, thus decreasing the excavation volume, the BVP suggests a new geometry for all the Final Slopes of the Cata Funda Pit, in addition to a more stable geometry for slopes in the soil, according to Table 15-7:

Table 15-7 Geometry Proposed for Cata Funda Pit (Arroz)

FINAL PIT GEOMETRY PROPOSED BY BVP						
ARROZ PIT (CATA FUNDA)						
DIRECTION	**GROUP**	**LITHOLOGIES**	**MAXIMUM HEIGH OF SLOPES (m)**	**MINIMUM BERMS WIDTH (m)**	**FACE ANGLE (°)**	**INTER-RAMPS ANGLE (°) [3]**
Hangingwall (HW)	Fresh Rock	ENC2, ZM1/ZM2, MD	10	8	80°	48 °
Footwall (FW)	Fresh Rock	ENC2, ZM1/ZM2	10	8	65° to 70°	45° to 47°
FW/HW	Soil	CO, SSP, SAP, RI	10	6	27°	23°

The probability of rupture occurrences due to tipping, which justifies, therefore, the proposal by BVP Engenharia for larger berms for all slopes at least 8.0m wide. It should be noted that such geometry may be changed according to the interests and mine planning. At least the recommended global angle must be respected, including accesses. Figures 15-8 and 15-9 present the analysis performed for the geometry proposed by BVP, with the water level adopted for the final pit interpreted according to the levels observed in the instruments of holes FAG01 and FAG02.

Figure 15-8 Geometry, Section 01, Hangingwall – Cata Funda Pit Global Rupture (FS = 2.0) and Local Rupture (FS = 1.6).

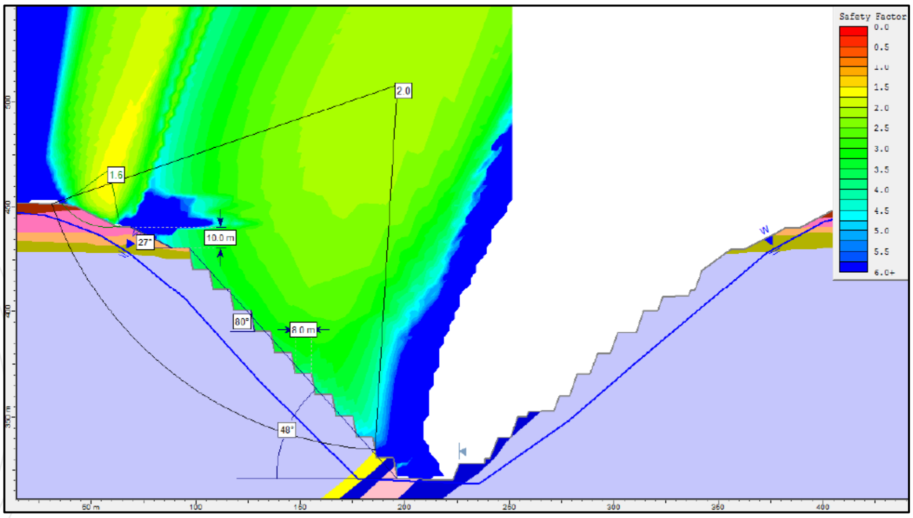
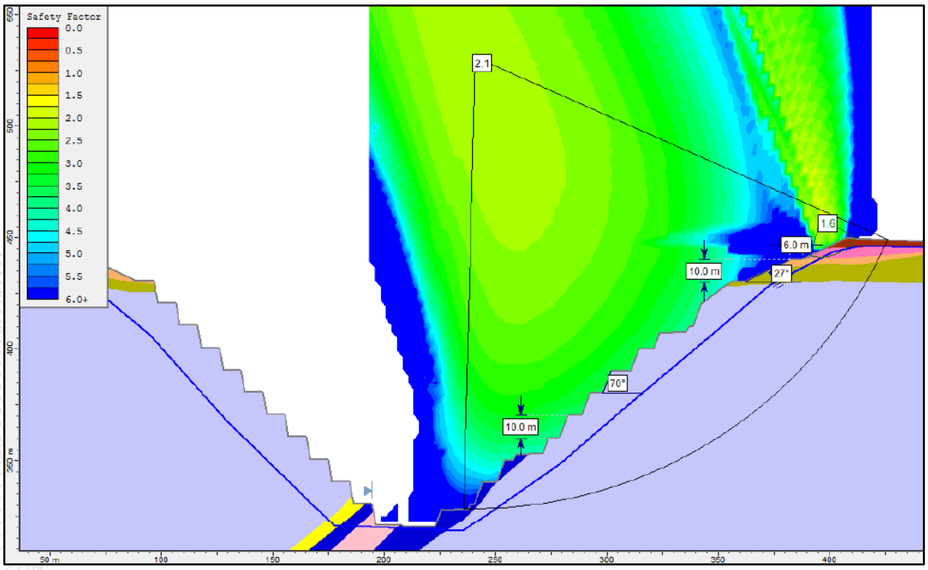


Figure 15-9 Geometry, Section 01, Footwall – Cata Funda. Slope Geometry Adjustment in Solo (FS = 1.6)



The slopes of the final pits, evaluating the geometries we studied, can be considered stable, presenting minimum safety factors ≥1.5 for ruptures globally. To optimize the safety factor and geometry of these slopes, BVP Engenharia suggests slopes with an overall angle of 48 ° (face angles of 80 °) on the Cata Funda Pit Hangingwall.

The global angle for the Footwall slope follows the average angle of the ore, thus it cannot be increased according to the angle of the Hangingwall slope. The new suggested geometry for berms of at least 8.0 m implies favoring safety due to possible falls evidenced in the kinematic analysis. Hole analysis ruled out the possibility of planar ruptures in slopes with a face angle of up to 80 °, so the analysis presented in this report were taken, only for circular breaks, in which resistance parameters were adjusted previously, evaluated and adopted, now, depending on the results of the tests laboratory tests performed.

Another factor to be noted was the need, depending on laboratory results, to slope recovery in soil given the low resistance presented by the soil saprolite, however the global ruptures of the Pit showed factors are well above the minimum allowable.

15.4 HYDROGEOLOGY

A numeric hydrogeological model of the permanent flow regime was prepared for the area around Paiol Pit to estimate the groundwater flow that could enter in the pit at the end of its life. Three scenarios were considered: Baseline - before the pit; Vale's Pit (year 2000); and Final pit.

The current Paiol Pit is filled with water to approximately an elevation of 366 m. Apparently the seasonal variation for the elevation is approximately +/- 1.0m.

Information and Data from Rio Novo Mineração:

In general terms, three types of data were incorporated into the hydrogeological analysis: topography; geological database based on the core descriptions and observations of water filling the Paiol Pit at the end of Vale's operations in 2000.

Due to the similarities of Cata Funda in geology and lithotypes, it is reasonable to expect similar results, with the volume of groundwater increasing proportionally as the pit gets deeper.

15.4.1 HYDROSTRATIGRAPHY

The studied areas contain several types of material in its hydrostratigraphy. It is possible that some of these types belong to the same geological composition, resulting, however, in different flow characteristics, due to their different degrees of alteration or weathering. Several samples of each geological unit were examined, including:

- Red saprolite soils (variable but average thickness of 10-15 m).
- Yellow Saprolite / Weathered Shale (variable, but average thickness 5 m to 8 m).
- Altered Rock (variable, but average thickness of 3 m to 5 m); and
- Fresh Rock.

In general terms, it is estimated that the water table hydraulic gradient is low because of the low and smooth topography in the region, and the groundwater flow is similar around the periphery of the final pit. It is estimated that the conductivity of the fresh rock is approximately three times lower than that of the altered rock zone, and the altered zone is approximately 2 times bigger than in the saprolites. This implies that the altered rock area will contribute with a volume disproportionate to the thickness of the material.

15.4.2 HYDROSTRUCTURE

Examination of the core drilling from exploration work, does not show significant fractures / failures in the fresh rock. Ferrari and Choudhuri's report (2004) indicated that the mapped fractures were filled by quartz and carbonates. There is evidence that the degree of fracturing increases in the proximal halo of the mineralized zone because of the hydrothermal alteration.

15.4.3 HYDROGEOLOGY

MODFLOW Surface was used for the simulation, and three scenarios were created with different geometries. The model domain is 6 km^2 with the current pit centered in the middle of the domain. The Baseline scenario was created to estimate conditions before Vale's Pit excavation. Vale's Pit scenario was created to simulate the conditions at the end of the year of 2000, and it was used to calibrate the model using the results of the pit water filling. The final scenario simulates the conditions of the Final Pit to be excavated at the end of the mine life. The geometry of the final pit was provided by Marston, and the interphase surfaces between layers, by Ron Simpson. After calibration, the following hydraulic conductivity values were estimated by layers: 0.7 m / day for the saprolite layers, 17.5 m / day for the altered zone and 0.0175 m / day for fresh rock.

15.4.4 RESULTS

The simulation's results indicate the water table will contribute about 46 l/s to the scenario of the Final Pit. This figure does not include precipitation contributions. We expect most of the groundwater flowing to be collected in the altered rock zone, but there may be other areas, fresh rock, that contribute because of fractures or local failures. Also, it is possible that in the altered halo around the mineralized zone, there may be a larger recharge area because of the alteration level.

15.5 MINE PLANNING FOR RESERVE ESTIMATION

The Almas Gold Project is planned to include three open pits. Such mining operations suggest 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 for each of the deposits, named Paiol, Cata Funda and Vira Saia. The mine schedule resulted in a 1.3 Mt annual average run-of-mine (ROM) production rate at 0.91 g/t and 137 Mt of waste over the 16 years life of the project at full capacity.

The reserves will be mined from three deposits, i.e., Paiol, which is the project's main deposit, and two satellite deposits Vira Saia and Cata Funda. Vira Saia and Cata Funda are located about 5 km and 15 km away respectively. The reserves include an existing heap leach from the Vale's historic operation.

The pit optimization was performed using Whittle Lerchs-Grossmann shell analysis. Whittle is a software package that uses Lerchs-Grossmann algorithm 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 models include grade, lithology, rock density and resource category (Measured, Indicated, and Inferred) information for all three deposits. The pit optimization involved analysis of candidate gold cut-off grades.

Based on the mine schedule, capital and operating costs were estimated.

15.5.1 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 for each deposit and were applied during the pit optimization and mine planning.

EDEM completed a pit optimization was based on 1.3 Mtpa ore production.

The input optimization parameters were provided by Aura and are listed in Table 15-8. Gol prices are in US Dollars and costs are in Brazilian Reals.

DILUTION

The grade model from Paiol and Vira Saia were reblocked because the original models use ore percentile inside each block. In this way the blocks will have the dilution in the grade model with weighted Gold grade.

The dilution estimated for Paiol and Vira Saia is 20%, and it is not necessary to add a dilution factor in the NPV Schedule simulation.

For Cata Funda it was considered 20% dilution in the NPV Schedule simulation.

Table 15-8 Pit Optimization Parameters

DESCRIPTION	UNITS	PAIOL		CATA FUNDA		VIRA SAIA		HEAP LEACH
		ORE	WASTE	ORE	WASTE	ORE	WASTE	ORE
PARAMETERS FOR NPV CALCULATION								
Gold price by oz	USD/oz	1,500.00	n/a	1,500.00	n/a	1,500.00	n/a	1,500.00
Gold price by g	USD/g	48.23	n/a	48.23	n/a	48.23	n/a	48.23
Ore mining costs	USD/t	2.00	1.80	4.00	1.80	2.60	1.80	n/a
Dilution	%	(*)	n/a	20%	n/a	(*)	n/a	n/a
Mining Recovery	%	100%	n/a	100%	n/a	100%	n/a	100%
Low-Grade Ore Cut-Off Grade	g/t	0.29	n/a	0.34	n/a	0.31	n/a	-
High-Grade Ore Cut-Off Grade	g/t	0.60	n/a	0.60	n/a	0.60	n/a	-
Mine deepening ratio limit by year	m	40	40	40	40	40	40	n/a
Pit Wall Overall angle	(°)	50	50	50	50	50	50	n/a
Processing costs	USD/t	9.50	n/a	9.50	n/a	9.50	n/a	9.50
G & A costs	USD/t	3.50	n/a	3.50	n/a	3.50	n/a	3.50
Total Processing + G&A costs	USD/t	13.00	n/a	13.00	n/a	13.00	n/a	13.00
Discount Rate	%	5.0	5.0	5.0	5.0	5.0	5.0	5.0
Metallurgical Recovery	%	92.5%	n/a	92.5%	n/a	92.5%	n/a	92.5%

Scheduling assumptions and constraints were based on a processing plant start-up on July 1st, 2022 and a ramp-up as follows:

- July: 40%
- August: 60%
- September: 80%
- October: 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, planeed dilution and safety

While the pit optimization used average haul costs, the mine scheduling used variable costs reflecting differential in-pit haul distances to every specific destination (primary crusher, stockpile, or waste dump) (Table 15-7).

15.5.2 PIT OPTIMIZATION RESULTS

Several 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.

Three optimization scenarios are automatically generated by Whittle software:

- Best case. It is based on an increasing pit shell extraction sequence.
- Worst case. It follows a bench by bench mining sequence.
- Specified case. The extraction sequence is created based upon predefined pushback geometries. This scenario is considered as most close to the operational and economic reality.

A summary of the optimization results by deposit are described in the following sections.

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

Simultaneous operation premise: maximum two pit operating simultaneously and operation at a given pit will be continuously mined until the end of its life of mine (LOM).

15.5.2.1 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. It is important to keep the premise that drilling, and blasting will be done in a way to ensure good performance and safety.

The agreed assumptions for the pits and phases designs are listed below:

- Geometric parameters: derived from Table 15-9
- Road and ramp width: 15 m
- Maximum ramp gradient: 10%

- Minimum distance between phases: 40 m
- Minimum bottom width: 20 m

Table 15-9 Geometrical Parameters for the Almas Gold Project

MINE	LITHOLOGY	BENCH HEIGHT (m)	BENCH WIDTH (m)	BENCH FACE ANGLE (°)	INTER RAMP (toe-to-toe) (°)	OVERALL [1] (toe to crest) (°)
Paiol	Soil	10	8	45	32	40
	Saprolite	10	8	65	37	40
	Fresh Rock	20	10	75	60	50
Cata Funda	Soil	10	8	45	32	40
	Saprolite	10	8	65	37	40
	Fresh Rock	20	10	75	60	50
Vira Saia	Soil	10	8	45	32	40
	Saprolite	10	8	65	37	40
	Fresh Rock	20	10	75	60	50

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.

A summary of in pit tonnage and grades by deposit is presented in Table 15-10.

Table 15-10 Mining Inventories by Deposit

ITEM	TOTAL	PAIOL	VIRA SAIA	CATA FUNDA	HEAP LEACH
Ore tonnage (kt)	21,882	16,138	3,780	689	1,275
Au (g/t)	0.92	0.88	0.91	1.89	0.90
Contained Gold (oz)	645,653	456,803	110,122	41,834	36,894
Recovered Gold (oz)	594,000	420,259	101,312	38,488	33,942
Waste tonnage (kt)	117,328	90,735	19,273	7,192	128
Strip Ratio	5.36	5.62	6.10	11.44	0.1
Total tonnage (kt)	139,211	106,873	23,054	7,881	1,403

15.5.4 MINE SCHEDULING

Table 15.11 shows the distribution of total ROM material moved from each mine. As can see in the mining sequencing, after year 2035 the concentration plant is planned to be fed exclusively from stockpiles.

The Pre-Stripping Operation, or only Pre- Operation (PRE-OP), that has plan to happens up to June 2022, has its volumes presented in the table15-14.

Table 15-11 Total Tonnage ROM Moved, by Pit

TONNAGE FROM PITS PER YEAR													
YEAR	*TOTAL MOVED (kt)*				*ORE (kt)*				*WASTE (kt)*				*W:O*
	TOTAL	*PAIOL*	*CATA FUNDA*	*VIRA SAIA*	*TOTAL*	*PAIOL*	*CATA FUNDA*	*VIRA SAIA*	*TOTAL*	*PAIOL*	*CATA FUNDA*	*VIRA SAIA*	*t:t*
2022	6,115	6,115	-	-	1,056	1,056	-	-	5,060	5,060	-	-	4.79
2023	12,184	12,184	-	-	2,271	2,271	-	-	9,913	9,913	-	-	4.37
2024	11,154	11,154	-	-	1,928	1,928	-	-	9,227	9,227	-	-	4.79
2025	15,469	12,828	2,641	-	2,405	2,201	203	-	13,064	10,626	2,438	-	5.43
2026	13,752	11,774	1,978	-	1,579	1,406	174	-	12,473	10,368	1,805	-	7.90
2027	15,431	13,628	1,803	-	1,926	1,773	153	-	13,505	11,855	1,650	-	7.01
2028	17,208	15,749	1,459	-	2,149	1,990	159	-	15,059	13,759	1,300	-	7.01
2029	19,220	15,013	-	4,207	2,251	1,591	-	660	16,969	13,422	-	3,547	7.54
2030	10,881	6,881	-	4,000	1,855	1,195	-	660	9,026	5,686	-	3,340	4.87
2031	6,877	1,549	-	5,328	1,388	728	-	660	5,489	821	-	4,668	3.95
2032	3,715	-	-	3,715	660	-	-	660	3,055	-	-	3,055	4.63
2033	3,788	-	-	3,788	660	-	-	660	3,128	-	-	3,128	4.74
2034	2,015	-	-	2,015	480	-	-	480	1,535	-	-	1,535	3.20
2035	-	-	-	-	-	-	-	-	-	-	-	-	-
2036	-	-	-	-	-	-	-	-	-	-	-	-	-
2037	-	-	-	-	-	-	-	-	-	-	-	-	-
2038	-	-	-	-	-	-	-	-	-	-	-	-	-
2039	-	-	-	-	-	-	-	-	-	-	-	-	-
TOTAL	**137,808**	**106,873**	**7,881**	**23,054**	**20,607**	**16,138**	**689**	**3,780**	**117,501**	**90,735**	**7,192**	**19,273**	**5.70**

Table 15-12 shows the ROM origin to be fed to the concentration plant.

Table 15-12 Concentration Plant Ore Planned to be Fed, by Origin.

	PLANT		ORE BY ORIGIN					
YEAR	ORE FEED	GRADE	PAIOL	CATA FUNDA	VIRA SAIA	HEAP LEACH PAD	HIGH GRADE PILE	LOW GRADE PILE
	kt	*g/t*	*kt*	*kt*	*kt*	*kt*	*kt*	*kt*
2022	529	1.29	329	-	-	-	100	100
2023	1,320	1.36	1,320	-	-	-	-	-
2024	1,320	1.36	1,320	-	-	-	-	-
2025	1,323	1.27	1,112	201	-	10	-	-
2026	1,300	1.19	989	156	-	155	-	-
2027	1,300	1.13	991	146	-	163	-	-
2028	1,300	1.12	989	144	-	167	-	-
2029	1,300	1.22	662	-	495	144	-	-
2030	1,300	1.11	660	-	433	208	-	-
2031	1,300	1.02	484	-	437	80	240	60
2032	1,300	0.79	-	-	383	249	200	468
2033	1,300	0.70	-	-	401	100	-	798
2034	1,300	0.58	-	-	341	-	-	959
2035	1,300	0.45	-	-	-	-	-	1,300
2036	1,300	0.45	-	-	-	-	-	1,300
2037	1,300	0.45	-	-	-	-	-	1,300
2038	1,300	0.45	-	-	-	-	-	1,300
2039	440	0.45	-	-	-	-	-	740
TOTAL	**21,833**	**0,92**	**8,856**	**646**	**2,490**	**1,275**	**540**	**8,226**

15.5.2.2 PAIOL PIT DESIGN

After the Paiol's pit optimization, each quarter period were designed targeting tp achieve, approximately, 330,000 t of high grade ore as presented in Table 15-13. The low grade ore was not constrained. This procedure allowed to have a more accurate approach to accomplish an optimization to the waste removal, and the pre-strip volumes. The volume calculation up to year 2025 was done by quarter presenting ore grades and waste/ore strip ratio; either, for ore, either for waste.

Table 15-13 Paiol - Pit Optimization Results

YEAR	PIT	TONNAGE	REVENUE	PROCESS. COST	MINING COST	CAP. COSTS	NPV	LOW-GRADE ORE	HIGH GRADE ORE	LOW-GRADE Au CONTEND	HIGH-GRADE Au CONTEND	STRIP RATIO W:O
	nr	kt	US$*1000	US$*1000	US$*1000	US$	US$*1000	kt	kt	kg	kg	t : t
PRE-OP	**1**	**2,565**	**27,184**	**9,639**	**4,760**	**-**	**12,040**	**356**	**332**	**158**	**448**	**2.73**
1	**2**	**3,615**	**50,408**	**14,780**	**6,726**	**-**	**25,988**	**394**	**662**	**175**	**953**	**2.42**
	3	**6,642**	**70,919**	**23,549**	**12,302**	**-**	**30,775**	**691**	**991**	**305**	**1,280**	**2.95**
	4	**9,655**	**96,251**	**31,752**	**17,846**	**-**	**38,970**	**948**	**1,320**	**416**	**1,736**	**3.26**
	5	**11,894**	**124,084**	**38,443**	**21,973**	**-**	**49,884**	**1,094**	**1,652**	**478**	**2,297**	**3.33**
2	**6**	**14,699**	**146,233**	**46,570**	**27,144**	**-**	**54,978**	**1,345**	**1,982**	**582**	**2,688**	**3.42**
	7	**16,384**	**169,764**	**53,222**	**30,272**	**-**	**62,179**	**1,490**	**2,311**	**646**	**3,152**	**3.31**
	8	**17,890**	**195,625**	**58,968**	**33,066**	**-**	**70,402**	**1,571**	**2,641**	**683**	**3,696**	**3.25**
	9	**20,828**	**216,175**	**66,717**	**38,468**	**-**	**73,545**	**1,793**	**2,973**	**780**	**4,058**	**3.37**
3	10	22,653	238,449	73,559	41,854	-	78,192	1,952	3,302	853	4,484	3.31
	11	25,939	261,130	82,346	47,897	-	80,989	2,250	3,632	978	4,865	3.41
	12	30,305	280,079	92,248	55,898	-	81,247	2,628	3,961	1,144	5,118	3.60
	13	**33,562**	**297,792**	**99,911**	**61,874**	**-**	**82,399**	**2,845**	**4,292**	**1,243**	**5,415**	**3.70**
4	14	36,899	317,804	107,248	67,990	-	84,108	3,039	4,621	1,324	5,781	3.82
	15	40,713	339,505	115,186	74,973	-	85,742	3,277	4,951	1,427	6,162	3.95
	16	**43,474**	**360,916**	**121,538**	**80,035**	**-**	**87,903**	**3,401**	**5,280**	**1,483**	**6,587**	**4.01**
5	17	49,794	383,356	130,990	91,546	-	88,217	3,744	5,612	1,632	6,936	4.32
	18	54,367	404,701	138,482	99,888	-	89,209	3,950	5,942	1,730	7,316	4.50
	19	**57,702**	**424,875**	**146,362**	**106,006**	**-**	**90,197**	**4,183**	**6,272**	**1,837**	**7,659**	**4.52**
6	20	66,680	449,835	158,276	122,341	-	89,790	4,704	6,602	2,063	7,985	4.90
	21	72,108	471,085	166,865	132,236	-	90,153	4,988	6,931	2,189	8,332	5.05
	22	**76,051**	**492,796**	**174,218**	**139,440**	**-**	**91,022**	**5,183**	**7,261**	**2,277**	**8,730**	**5.11**
7	23	95,131	520,697	189,329	174,014	-	88,638	5,933	7,590	2,586	9,036	6.03

YEAR	PIT	TONNAGE	REVENUE	PROCESS. COST	MINING COST	CAP. COSTS	NPV	LOW-GRADE ORE	HIGH GRADE ORE	LOW-GRADE Au CONTEND	HIGH-GRADE Au CONTEND	STRIP RATIO W:O
	nr	kt	US$*1000	US$*1000	US$*1000	US$	US$*1000	kt	kt	kg	kg	t : t
	24	**97,964**	**542,427**	**196,493**	**179,219**	**-**	**89,540**	**6,113**	**7,923**	**2,667**	**9,442**	**5.98**
8	25	104,147	566,734	207,131	190,507	-	89,724	6,543	8,252	2,853	9,796	6.04
	26	**104,845**	**587,887**	**213,224**	**191,852**	**-**	**90,797**	**6,648**	**8,582**	**2,899**	**10,224**	**5.88**
9	27	105,600	611,605	219,335	193,300	-	91,938	6,757	8,910	2,950	10,705	5.74
	28	106,568	632,791	225,861	195,138	-	92,760	6,893	9,240	3,010	11,120	5.61
	29	**106,574**	**633,107**	**225,939**	**195,149**	**-**	**92,774**	**6,893**	**9,246**	**3,010**	**11,127**	**5.60**

Figure 15-10 shows the designed pit for Paiol.

Figure 15-10 Paiol's Mine Final Pit Already Operationalized

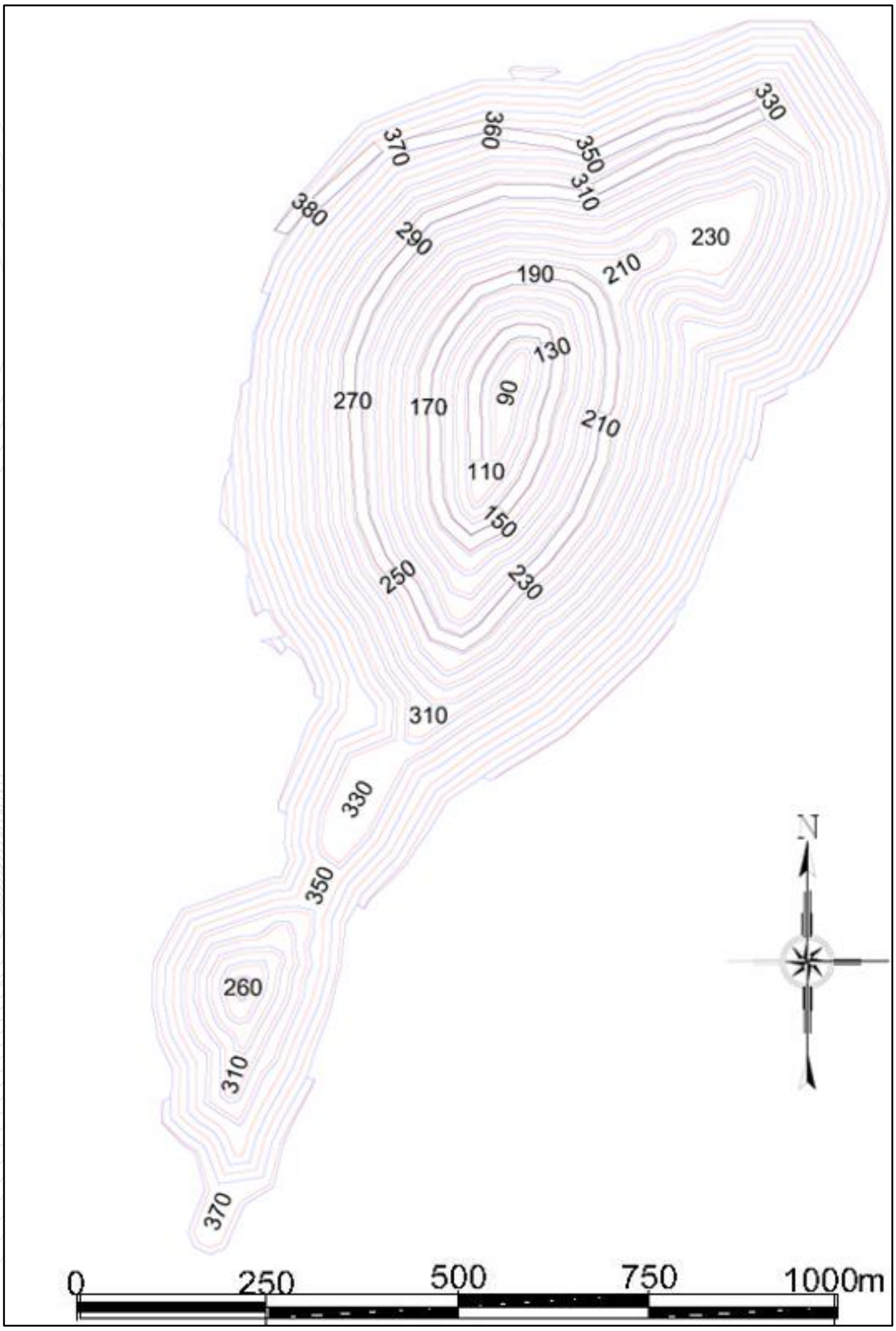
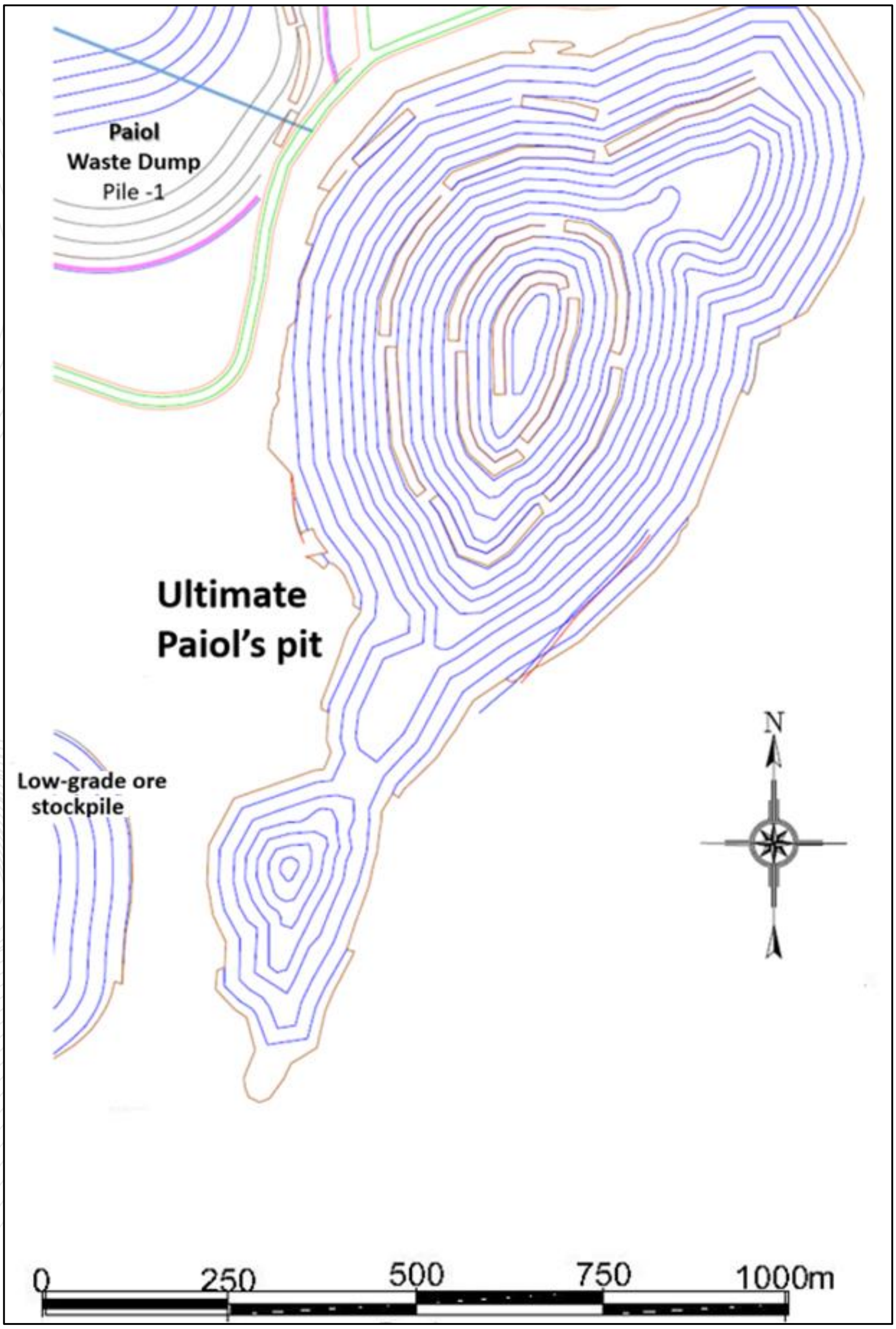


Figure 15-11 shows the Paiol pit design in relation to the neighboring infrastructures including two waste dumps specifically for Paiol and a low grade stockpile for the three open pit operations.

Figure 15-11 Paiol Ultimate Pit View with Neighboring Infrastructure



The pits for the Paiol mine were generated by quarter; Figure 15-12 and figure 15-13 show several pits design: Pre-Operation up to the end of Q2/2022, ramp-up up to the end of Q3/2022 up to the end of Q4/2022 and up to the end Q4/224; the other pits are the AutoCAD files delivered.

Figure 15-12 Paiol Pit Design for Pre-Operation and to Ramp-up on Q3/2022.

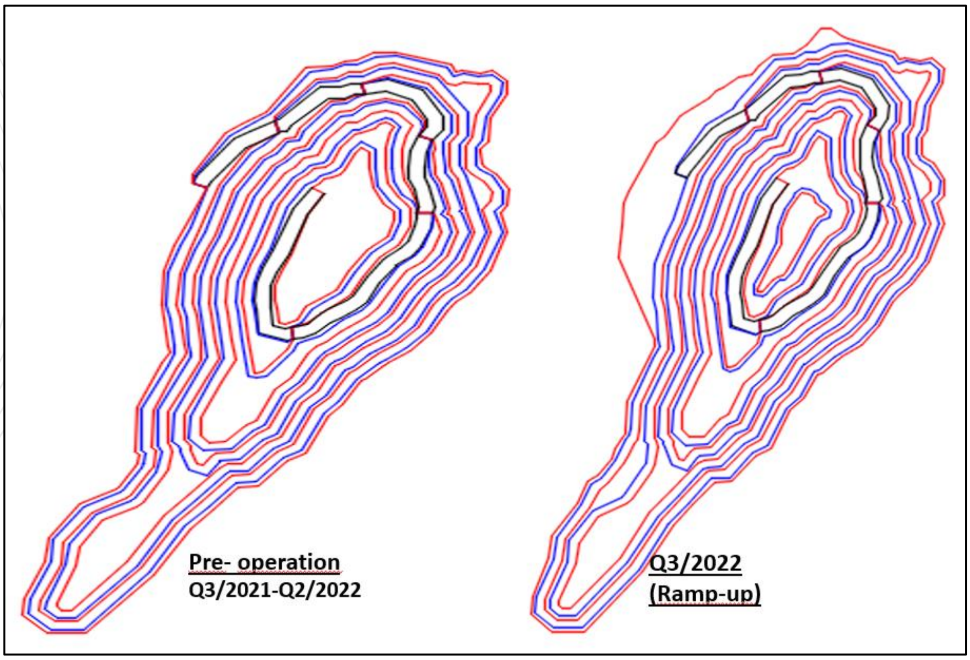
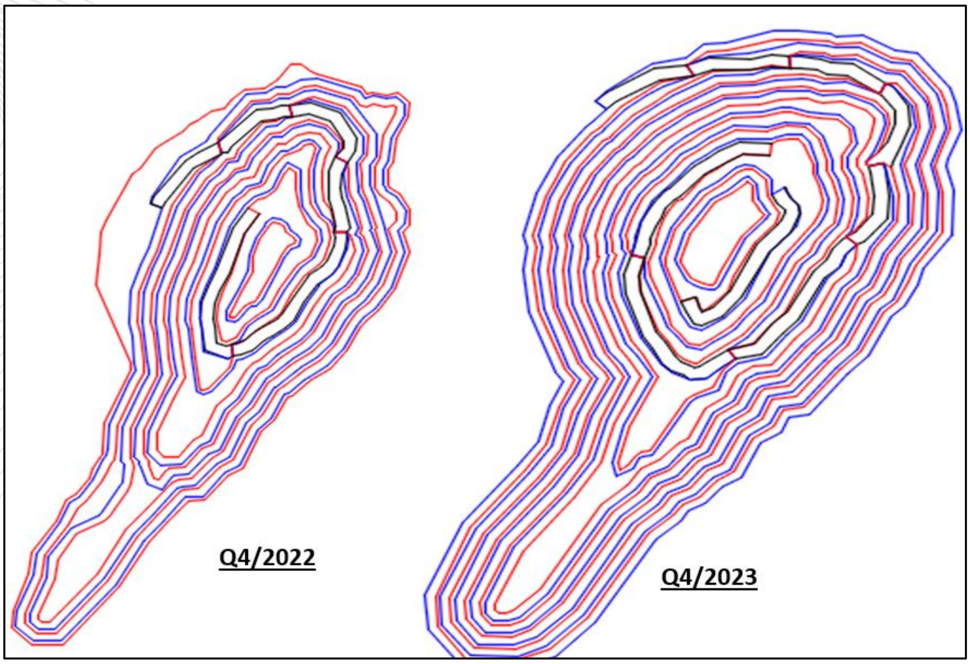


Figure 15-13 Paiol Pit Design to Operation Up to the end2022 and, up to the End of 2023.



A cross section of the Paiol's pit against the orebody position is illustrated in Figure 15-14.

Figure 15-14 Mineral Paiol Orebody Position in Relation to the Ultimate Pit

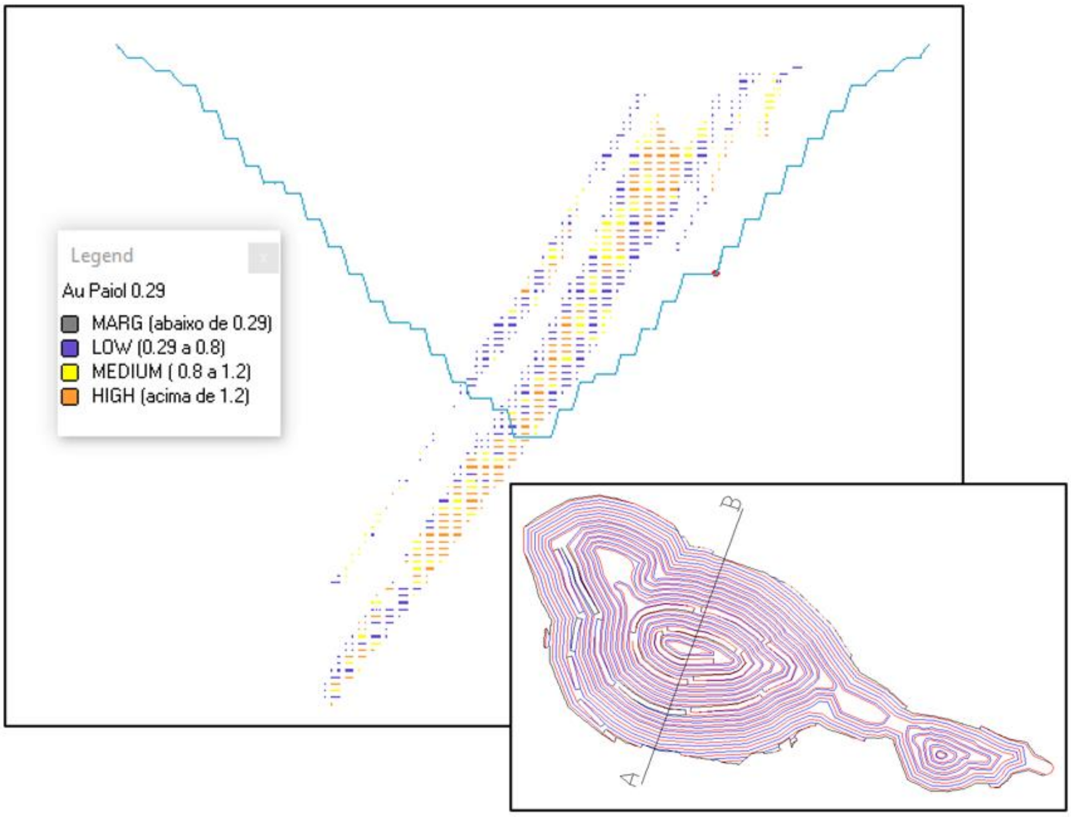


Table 15-14 presents the waste and ore tonnages of calculated tonnages) and related grade and waste/ore strip ratio calculated.

Table 15-14 Paiol's Selected Pit Ore and Waste Tonnages, Ore Grade, Waste/Ore Strip Ratio

PAIOL - TONNAGE MOVED PER YEAR										
PERIOD		TOTAL MOVED	TOTAL ORE	HIGH GRADE ORE			LOW GRADE ORE		WASTE	W:O
YEAR	*QUARTER*	*kt*	*kt*	*total (kt)*	*Grade g/t*	*stock (kt)*	*kt*	*g/t*	*kt*	*t:t*
2022	P. Oper.	2,865	689	332	1.35	332	356	0.44	2,177	3.16
	Q3-R.-up	1,600	-	-	-	232	-	0.46	1,600	-
	P-op.+Q3	4,465	689	332	1.35	232	356	0.44	3,777	5.49
	Q4	1,650	367	329	1.53	232	38	0.44	1,283	3.49
	Total	6,115	1,056	662	1.44	232	394	0.44	5,060	4.79
2023	Q1	3,027	626	329	1.35	232	297	0.43	2,400	3.83
	Q2	3,113	586	329	1.38	232	257	0.43	2,527	4.31
	Q3	2,938	478	332	1.54	232	146	0.41	2,460	5.15
	Q4	3,105	581	329	1.19	232	251	0.44	2,525	4.35
	Total	12,184	2,271	1,320	1.36	232	950	0.43	9,913	4.37
2024	Q1	2,685	475	329	1.41	232	146	0.44	2,210	4.65
	Q2	2,707	410	329	1.65	232	81	0.46	2,296	5.60
	Q3	2,938	554	332	1.09	232	221	0.44	2,384	4.31
	Q4	2,825	489	329	1.29	232	159	0.46	2,337	4.78
	Total	11,154	1,928	1,320	1.36	232	608	0.45	9,227	4.79
2025	Q1	3,486	628	329	1.16	232	298	0.44	2,858	4.55
	Q2	3,766	707	330	1.18	232	378	0.45	3,059	4.32
	Q3	3,057	547	273	0.90	232	217	0.58	2,510	4.59
	Q4	2,519	319	180	1.19	290	139	0.58	2,200	6.90

PAIOL - TONNAGE MOVED PER YEAR										
PERIOD		TOTAL MOVED	TOTAL ORE	HIGH GRADE ORE			LOW GRADE ORE		WASTE	W:O
YEAR	QUARTER	*kt*	*kt*	*total (kt)*	*Grade g/t*	*stock (kt)*	*kt*	*g/t*	*kt*	*t:t*
	Total	12,828	2,201	1,112	1.10	440	1,032	0.49	10,626	4.83
2026	**all**	11,774	1,406	989	1.13	440	417	0.58	10,368	7.38
2027	**all**	13,628	1,773	991	1.08	440	782	0.45	11,855	6.69
2028	**all**	15,749	1,990	989	1.08	440	1,000	0.44	13,759	6.91
2029	**all**	15,013	1,591	662	1.12	440	930	0.42	13,422	8.44
2030	**all**	6,881	1,195	660	1.13	440	536	0.43	5,686	4.76
2031	**all**	1,549	728	484	1.14	440	245	0.45	821	1.13
2032	**all**	-	-	-	-	200	-	-	-	-
2033	**all**	-	-	-	-	-	-	-	-	-
2034	**all**	-	-	-	-	-	-	-	-	-
2035	**all**	-	-	-	-	-	-	-	-	-
2036	**all**	-	-	-	-	-	-	-	-	-
2037	**all**	-	-	-	-	-	-	-	-	-
2038	**all**	-	-	-	-	-	-	-	-	-
2039	**all**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
TOTAL		**106,873**	**16,138**	**9,188**	**1.21**	**-**	**6,893**	**0.45**	**90,735**	**5.62**

15.5.2.3 CATA FUNDA PIT DESIGN

The pit optimization results for Cata Funda are presented in Table 15-15. The selected pit is the nr 33.

Table 15-15 Cata Funda - Pit Optimization Results

	UNIT	PIT NUMBER					
DESCRIPTION	#	30	31	32	33	34	35
	%	34.00%	35.00%	36.00%	**37.00%**	38.00%	40.00%
ROCK	t	1,900,928	2,060,108	2,065,073	**2,082,841**	2,539,246	2,780,611
REVENUE	US$	27,775,719	28,723,701	28,785,319	**28,916,898**	31,652,271	32,935,011
PROCESSING COSTS	US$	5,077,345	5,253,721	5,295,460	**5,344,221**	5,987,094	6,210,054
MINING COSTS	US$	4,092,549	4,396,771	4,411,173	**4,451,092**	5,360,339	5,823,994
CAPITAL COSTS	US$	-	-	-	**-**	-	-
NPV	US$	18,356,837	18,809,873	18,815,178	**18,856,732**	1,998,787	20,573,300
ROCK 1	t	18,520	18,520	19,254	**20,191**	26,203	26,203
ROCK 2	t	283,703	294,202	295,952	**297,918**	330,172	343,443
ROCK 1-AU_OK	g	9,185	9,185	9,468	**9,935**	13,104	13,104
ROCK 2-AU_OK	g	616,924	638,293	639,399	**641,899**	700,389	729,304
ROCK 1-AU_OK R	g	8,450	8,450	8,711	**9,140**	12,056	12,056
ROCK 2-AU_OK R	g	567,570	587,230	588,247	**590,547**	644,358	670,960
REM	t:t	5.30	5.59	5.55	**5.55**	6.33	6.52

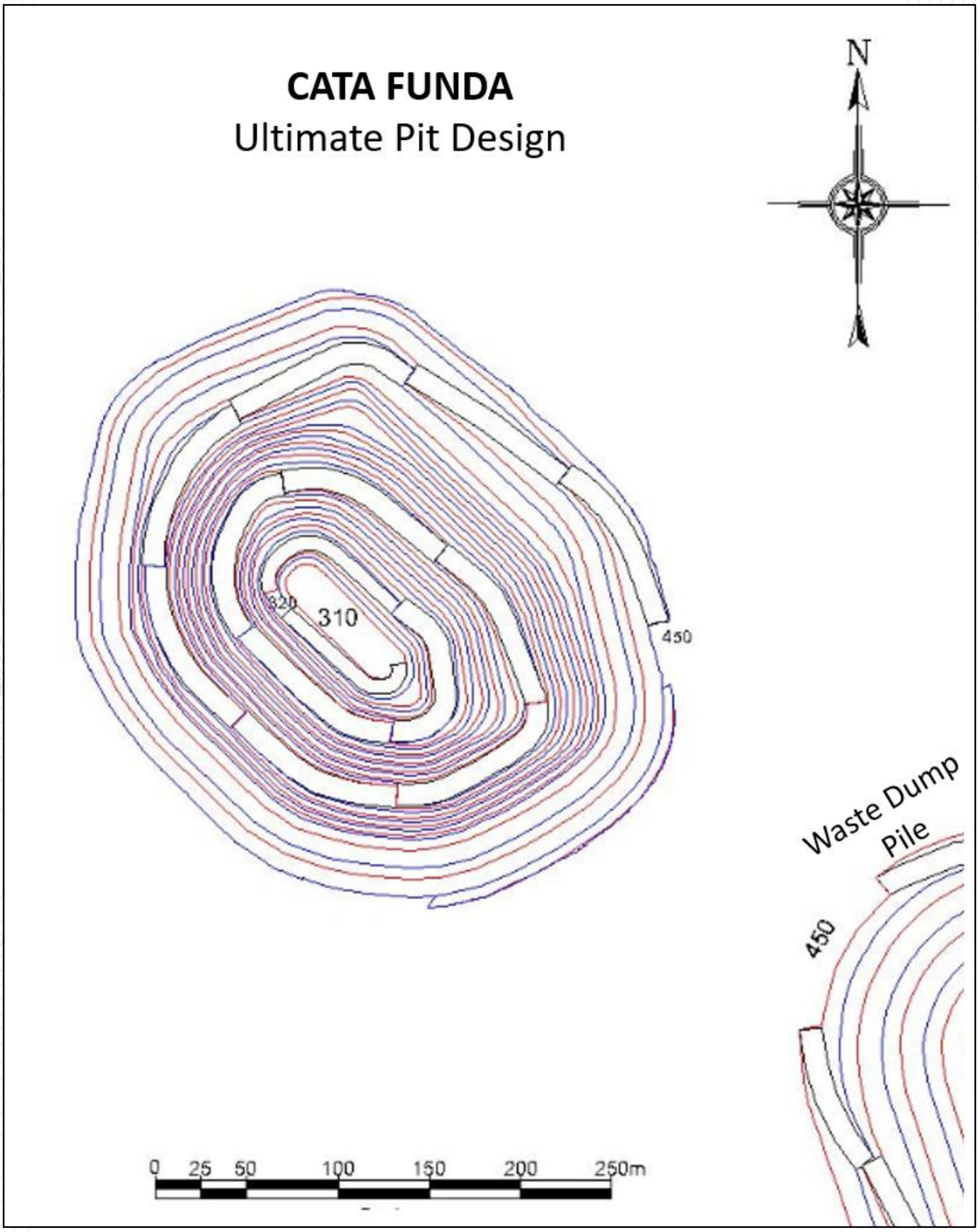
Table 15-16 presents the yearly sequence for Cata Funda including ore, waste tonnage, ore grades and waste/ore strip ratio.

Table 15-16 Mining Sequencing Planned for the Cata Funda Mine

CATA FUNDA - TONNAGE MOVED PER YEAR								
YEAR	*TOTAL MOVED*	*TOTAL ORE*	*HIGH GRADE ORE*		*LOW GRADE ORE*		*WASTE*	*W:O*
	kt	*kt*	*kt*	*g/t*	*kt*	*g/t*	*kt*	*t:t*
2022	-	-	-	-	-	-	-	-
2023	-	-	-	-	-	-	-	-
2024	-	-	-	-	-	-	-	-
2025	2,641	203	201	2.22	2	0.52	2,438	12.0
2026	1,978	174	156	1.94	18	0.50	1,805	10.4
2027	1,803	153	146	1.79	7	0.49	1,650	10.8
2028	1,459	159	144	1.74	15	0.49	1,300	8.2
2029	-	-	-	-	-	-	-	-
2030	-	-	-	-	-	-	-	-
2031	-	-	-	-	-	-	-	-
2032	-	-	-	-	-	-	-	-
2033	-	-	-	-	-	-	-	-
2034	-	-	-	-	-	-	-	-
2035	-	-	-	-	-	-	-	-
2036	-	-	-	-	-	-	-	-
2037	-	-	-	-	-	-	-	-
2038	-	-	-	-	-	-	-	-
2039	-	-	-	-	-	-	-	-
TOTAL	**7,881**	**689**	**646**	**1.95**	**42**	**0.50**	**7,192**	**11.44**

Figure 15-15 shows the designed pit for Cata Funda.

Figure 15-15 Cata Funda Ultimate *Pit Design.*



15.5.2.4 VIRA SAIA PIT DESIGN

Vira-Saia's pit was also run with 330,000 tonnes high-grade pushbacks and the results are set out in Table 15-17.

Table 15-17 Vira Saia's Pit Simulation Results

YEAR	TONNAGE	REVENUE	PROCESS. COST	MINING COST	CAPITAL COST	NPV	LOW-GRADE ORE	HIGH GRADE ORE	LOW-GRADE Au CONTEND	HIGH-GRADE Au CONTEND	STRIP RATIO W:O
	kt	$*1000	$*1000	$*1000	US$	US$*1000	kt	kt	kg	kg	t : t
1	4,207	35,110	9,241	8,079	-	16,874	165	495	76	716	5.37
2	8,207	62,564	18,481	15,787	-	26,052	392	928	179	1,232	5.22
3	12,535	87,509	27,722	24,084	-	31,745	616	1,364	282	1,691	5.33
4	17,250	111,174	36,962	33,074	-	35,666	893	1,747	410	2,096	5.53
5	20,438	134,687	46,200	39,326	-	40,788	1,151	2,149	531	2,505	5.19
6	23,054	151,946	52,921	44,405	-	43,932	1,291	2,490	597	2,828	5.10

Figure 15-17 shows Vira Saia's ultimate and operationalized pit.

Figure 15-16 Ultimate and Operationalized Pit Design for the Vira Saia Mine

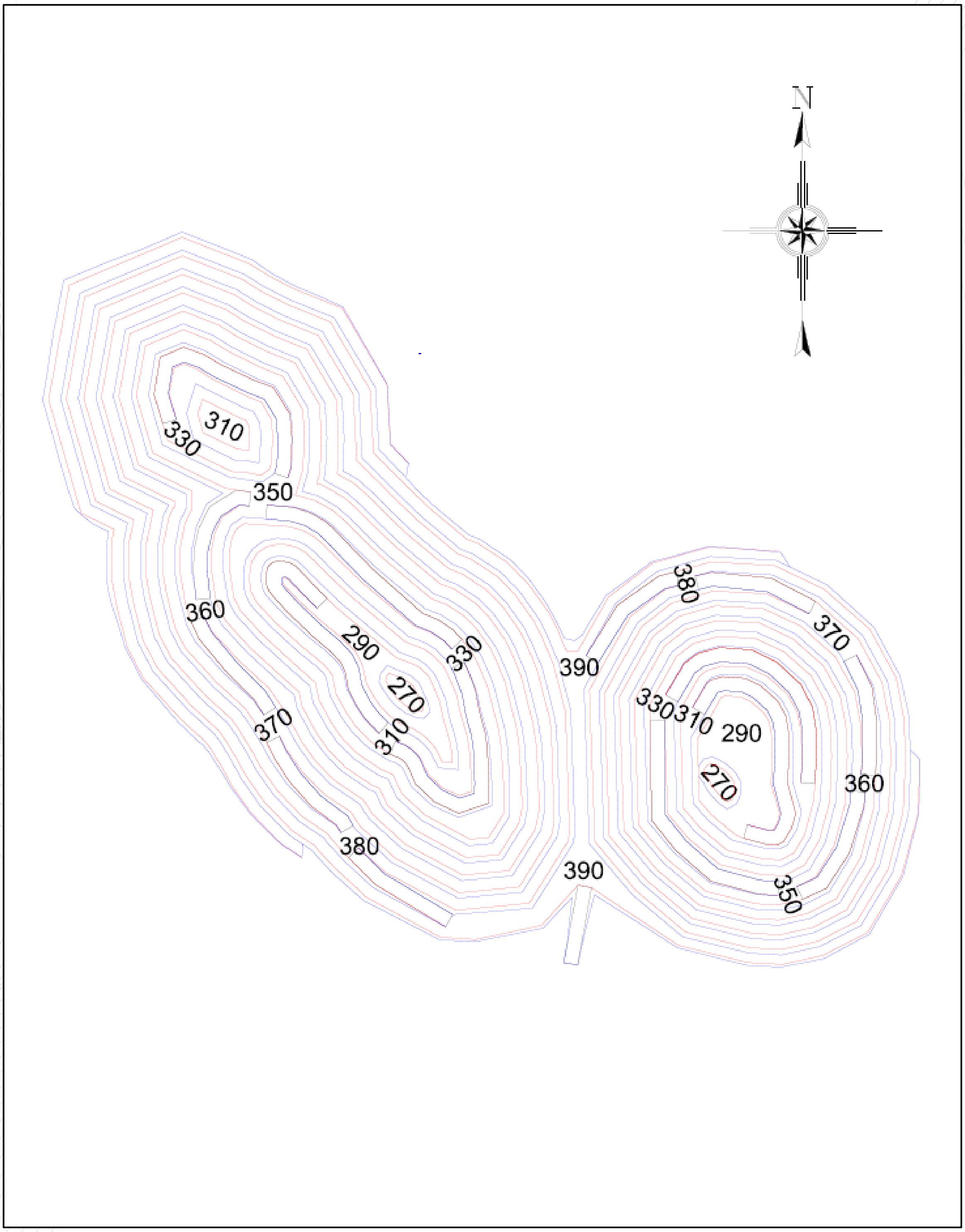


Figure 15-17 shows the ultimate pit design for Vira Saia.

Figure 15-17 Vira Saia's Ultimate Pit View with Neighboring Infrastructure

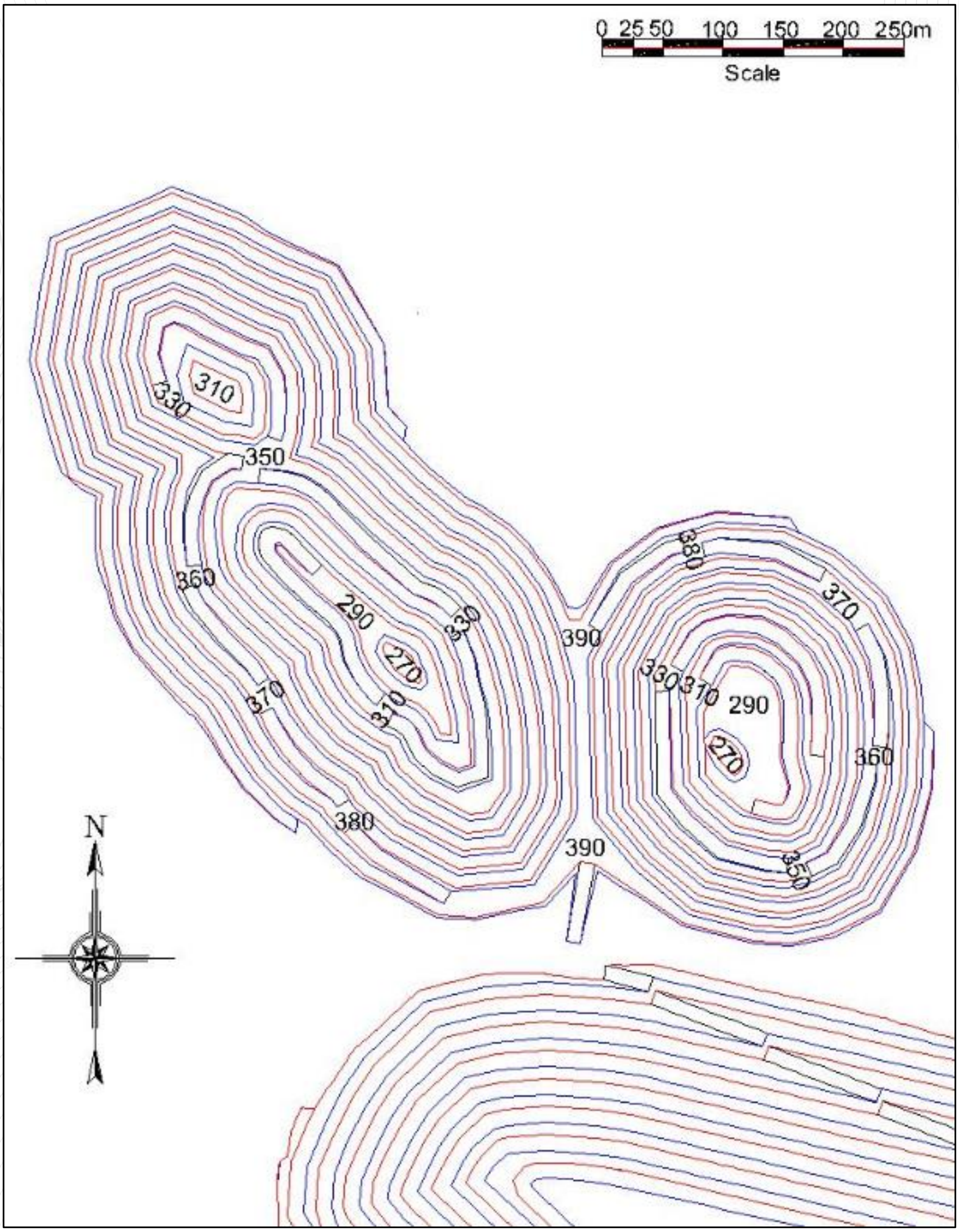


Table 15-18 presents the yearly sequence for the Vira Saia including ore and waste tonnage, ore grades and strip ratio.

Table 15-18 Vira-Saia's Mine Sequencing

VIRA-SAIA – TONNAGE MOVED PER YEAR								
YEAR	TOTAL MOVED	TOTAL ORE	HIGH GRADE ORE		LOW GRADE ORE		WASTE	W:O
	kt	kt	(kt)	g/t	kt	g/t	kt	t:t
2022	-	-	-	-	-	-	-	-
2023	-	-	-	-	-	-	-	-
2024	-	-	-	-	-	-	-	-
2025	-	-	-	-	-	-	-	-
2026	-	-	-	-	-	-	-	-
2027	-	-	-	-	-	-	-	-
2028	-	-	-	-	-	-	-	-
2029	4,207	660	495	1.45	165	0.46	3,547	5.37
2030	4,000	660	433	1.19	227	0.45	3,340	5.06
2031	5,328	660	437	1.05	223	0.46	4,668	7.07
2032	3,715	660	383	1.06	277	0.46	3,055	4.63
2033	3,788	660	401	1.02	258	0.47	3,128	4.74
2034	2,015	480	341	0.95	139	0.47	1,535	3.20
2035	-	-	-	-	-	-	-	-
2036	-	-	-	-	-	-	-	-
2037	-	-	-	-	-	-	-	-
2038	-	-	-	-	-	-	-	-
2039	-	-	-	-	-	-	-	-
TOTAL	23,054	3,780	2,490	1.14	1,291	0.46	19,273	6.10

15.5.2.5 HEAP LEACH PAD (HLP) AND ORE STOCKPILES

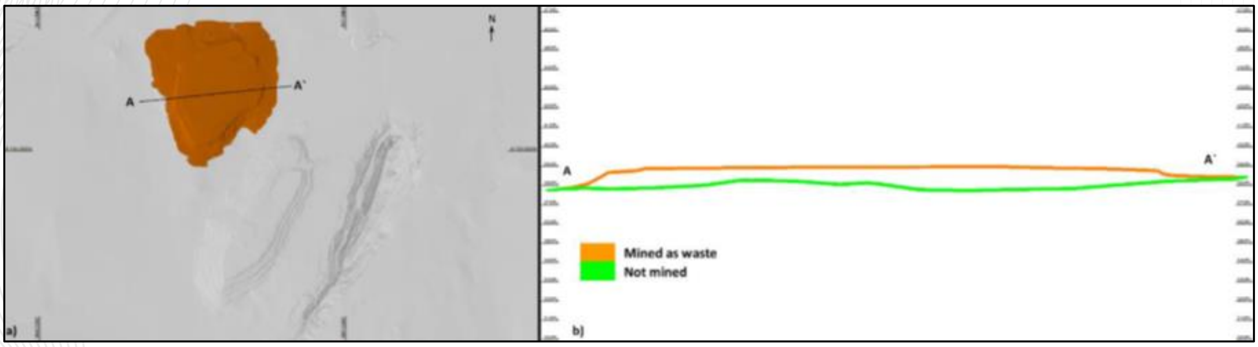
Aura envisages the recovery of the remaining ore located at a historical heap leach pad from VALE's time of operation (Figure 15-18). This material is stockpiled near the Paiol pit. The pile is currently covered by vegetation.

It was assumed that a 0.5 m thickness at the top of the heap leach stockpile will be removed and stored in a topsoil pile. In addition, a thickness of 0.5 m of ore near the bottom will be left in-situ.

As only Indicated resources were considered for HLP ore, and considering the uncertainties in metallurgical recovery due to long time stocked (about twenty years) after to be submitted to a leaching process, the HLP reserves were classified as probable. .

No cut-off grade was applied to the HLP.

Figure 15-18 Heap Leach Pile – a) Plan View – b) Vertical Section



The HLP mine scheduling is presented in Table 15-19.

Considering the HLP ore grades lower than other possibilities directly in the mines, the cash flow is not the most favorable if we start mining in HLP; A more favorable cash flow considered to start digging the Heach leach pad on the year 2025. Anyway, it is an alternative to support any mining problems in the pits.

Low Grade Stockpiles

The Aura Project was configured to operate with a variable cut-off strategy which encompasses the use of low grade stockpiles.

The pit design and mining strategy to stockpile the low-grade ore is aimed to maximize the project's economic return. It will also work as a buffer to offset any issues that may occur in the ROM pit production.

The stockpiles will be reclaimed after year 2031. The low grade ore stockpiles tonnage accumulated from the LOM sequence are:

- Paiol: 6.89 Mt (dry)
- Vira Saia: 1.29 Mt (dry)
- Cata Funda: 0.04 Mt (dry)

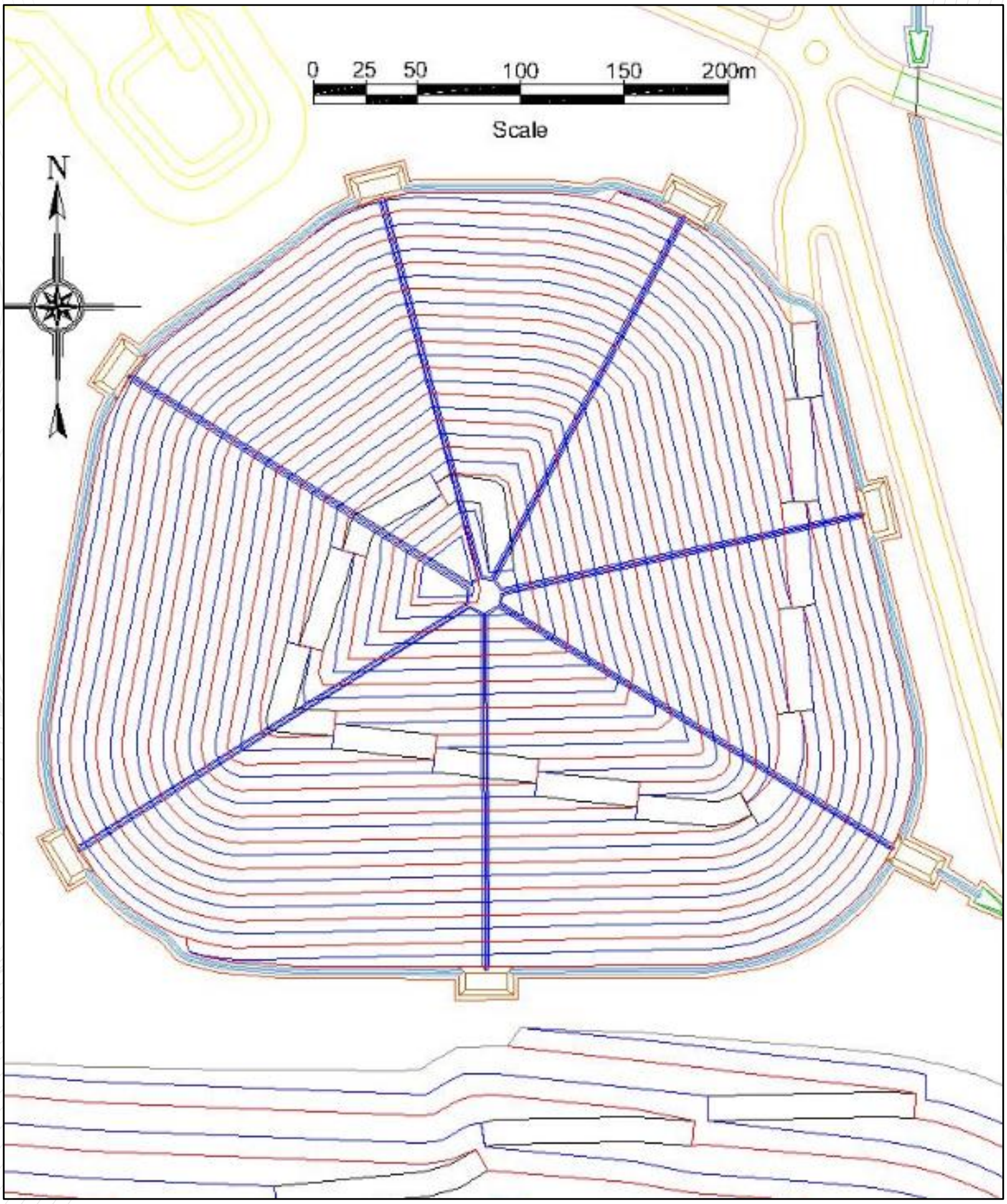
The table 15-19 shows the sequence of rock stockpiled in the low grade pile for later reclaiming.

Table 15-19 Heap Leach, Low-Grade Ore and High-Grade Stockpile Plan to Concentration Plant

ORE FROM STOCKPILES TO CONCENTRATION PLANT									
YEAR	***HEAP LEACH PAD***			***LOW GRADE STOCKPILE***			***HIGH GRADE STOCKPILE***		
	kt	***g/t Au***	***Au (Oz)***	***(kt)***	***g/t Au***	***Au (Oz)***	***kt***	***g/t Au***	***Au (Oz)***
2022	-	-	-	-	-	-	100	1.35	4,340
2023	-	-	-	-	-	-	-	-	-
2024	-	-	-	-	-	-	-	-	-
2025	10	0.90	288	-	-	-	-	-	-
2026	155	0.90	4,498	-	-	-	-	-	-
2027	163	0.90	4,710	-	-	-	-	-	-
2028	167	0.90	4,823	-	-	-	-	-	-
2029	144	0.90	4,153	-	-	-	-	-	-
2030	208	0.90	6,012	-	-	-	-	-	-
2031	80	0.90	2,302	60	0.45	871	240	1.13	8,728
2032	249	0.90	7,213	468	0.45	6,746	200	1.13	7,286
2033	100	0.90	2,894	798	0.45	11,510	-	-	-
2034	-	-	-	959	0.45	13,832	-	-	-
2035	-	-	-	1,300	0.45	18,745	-	-	-
2036	-	-	-	1,300	0.45	18,745	-	-	-
2037	-	-	-	1,300	0.45	18,745	-	-	-
2038	-	-	-	1,300	0.45	18,745	-	-	-
2039	-	-	-	740	-	-	-	-	-
TOTAL	**1,275**	**0.90**	**36,894**	**8,226**	**0.45**	**107,938**	**540**	**1.17**	**20,355**

Figure 15-19 presents the low-grade ore stockpile design.

Figure 15-19 Low-Grade Ore Design Stockpile with Water Fine Solids Separation Boxes and Drainage Channels in the Bottom of the Stockpile



15.5.3 MINING INVENTORY

The table 15-20 shows, by year and origin, the gold content in the ore planned to be fed to the plant.

Table 15-20 Au Contained in Yearly Planned Feed to the Concentration Plant

	PLANT		Au CONTAINED, BY ORIGIN					
YEAR	*TOTAL AU FEED*	*TOTAL AU RECOVERED*	*PAIOL'S PIT*	*CATA FUNDA'S PIT*	*VIRA SAIA"S PIT*	*HEAP LEACH PAD*	*HIGH-GRADE PILE*	*LOW-GRADE PILE*
	t	*oz*	*oz*	*oz*	*oz*	*oz*	*oz*	*oz*
2022	21,992	20,233	16,240	-	-	-	4,337	1,415
2023	57,923	53,289	57,923	-	-	-	-	-
2024	57,726	53,108	57,726	-	-	-	-	-
2025	55,449	51,013	41,148	14,301	-	-	-	-
2026	49,653	45,681	35,914	9,748	-	3,992	-	-
2027	47,357	43,569	34,477	8,384	-	4,496	-	-
2028	46,803	43,058	34,353	8,058	-	4,392	-	-
2029	51,051	46,967	23,886	-	23,012	4,153	-	-
2030	46,533	42,810	23,931	-	16,589	6,012	-	-
2031	44,375	40,825	17,723	-	14,751	2,302	8,728	871
2032	34,280	31,538	-	-	13,035	7,213	7,286	6,746
2033	28,277	26,015	-	-	13,143	4,340	-	10,793
2034	24,221	22,283	-	-	10,388	-	-	13,832
2035	18,745	17,245	-	-	-	-	-	18,745
2036	18,745	17,245	-	-	-	-	-	18,745
2037	18,745	17,245	-	-	-	-	-	18,745
2038	18,745	17,245	-	-	-	-	-	18,745
2039	0	0	-	-	-	-	-	-
TOTAL	**640,619**	**589,369**	**343,321**	**40,491**	**90,920**	**36,901**	**20,351**	**108,634**

A detailed summary of the tonnage and grades by deposit is presented in Table 15-21.

Table 15-21 Detailed Mining Inventories by Deposit

		IN PIT				HEAP LEACH PAD	GRAN TOTAL
	UNIT	PAIOL	VIRA SAIA	CATA FUNDA	TOTAL		
ORE	kt	16,138	3,780	646	20,565	1,275	21,840
AU GRADE	g/t	0.88	0.91	1.86	0.92	0.90	0.91
AU CONT. GOLD	Oz	443,262	110,122	41,166	594,549	36,901	631,450
AU REC.GOLD	Oz	407,801	101,312	37,873	546,985	33,949	580,934
WASTE	kt	90,735	19,273	7,192	117,201	128	117,328
STRIP RATIO W:O	t:t	5.62	5.10	10.44	5.70	0.1	5.37

15.5.4 WASTE DUMP PILES

The assumptions used for the waste dump design is listed below we based in the Feasibility study conveyed by Rio Novo Mineração on August 2016:

- 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%

Table 15-22 presents the yearly, and accumulated, waste moved, by pit, allowing estimation of waste dump volumes.

Table 15-22 Yearly planned Waste and Total Movement for Calculating the Volumes of Waste Dump Stockpiles

	WASTE TONNAGE PER YEAR (kt)								W:O
	MOVED IN PIT		*PAIOL*		*CATA FUNDA*		*VIRA SAIA*		
YEAR	**TOTAL**	**WASTE**	***BY YEAR***	***ACCUMULATED***	***BY YEAR***	***ACCUMULATED***	***BY YEAR***	***ACCUMULATED***	***t:t***
2022	6,115	5,060	5,060	5,060	-	-	-	-	4.79
2023	12,184	9,913	9,913	14,972	-	-	-	-	4.37
2024	11,154	9,227	9,227	24,199	-	-	-	-	4.79
2025	15,469	13,064	10,626	34,825	2,438	2,438	-	-	5.43
2026	13,752	12,473	10,368	45,193	1,805	4,243	-	-	9.75
2027	15,431	13,505	11,855	57,048	1,650	5,892	-	-	7.01
2028	17,208	15,059	13,759	70,807	1,300	7,192	-	-	7.01
2029	19,220	16,969	13,422	84,229	-	7,192	3,547	3,547	7.54
2030	10,881	9,026	5,686	89,914	-	7,192	3,340	6,887	4.87
2031	6,877	5,489	821	90,735	-	7,192	4,668	11,555	3.95
2032	3,715	3,055	-	90,735	-	7,192	3,055	14,610	4.63
2033	3,788	3,128	-	90,735	-	7,192	3,128	17,738	4.74
2034	2,015	1,535	-	90,735	-	7,192	1,535	19,273	3.20
2035	-	-	-	90,735	-	7,192	-	19,273	-
2036	-	-	-	90,735	-	7,192	-	19,273	-
2037	-	-	-	90,735	-	7,192	-	19,273	-
2038	-	-	-	90,735	-	7,192	-	19,273	-
2039	-	-	-	90,735	-	7,192	-	19,273	-
TOTAL	**137,808**	**117,501**	**90,735**		**7,192**		**19,273**		**5.70**

The calculated volumes of the waste dumps are presented in Table 15-23.

Table 15-23 Waste Dumps Piles Required Capacities

	WASTE BY PIT	
	TONNAGE(*1000)	VOLUME(*1000)
PIT	t	m^3
PAIOL	90,735	45,368
VIRA SAIA	19,273	9,637
CATA FUNDA	7,192	3,596

Due to the restriction of lateral limits and the unavailability of larger areas to make a single pile, it was necessary to design two waste dump piles to accommodate the entire volume that is planned to be removed from the final Paiol pit as shown in Figure 15-20. In the same figure there is also an indication of a drain structure on the bottom of the pile and the concrete boxes to separate the fines solids from water to minimize drain clogging. The Paiol waste dump figure gives a localization view with respect to the low grade stockpile and Paiol pit.

Figure 15-20 Paiol Final Waste Dump piles

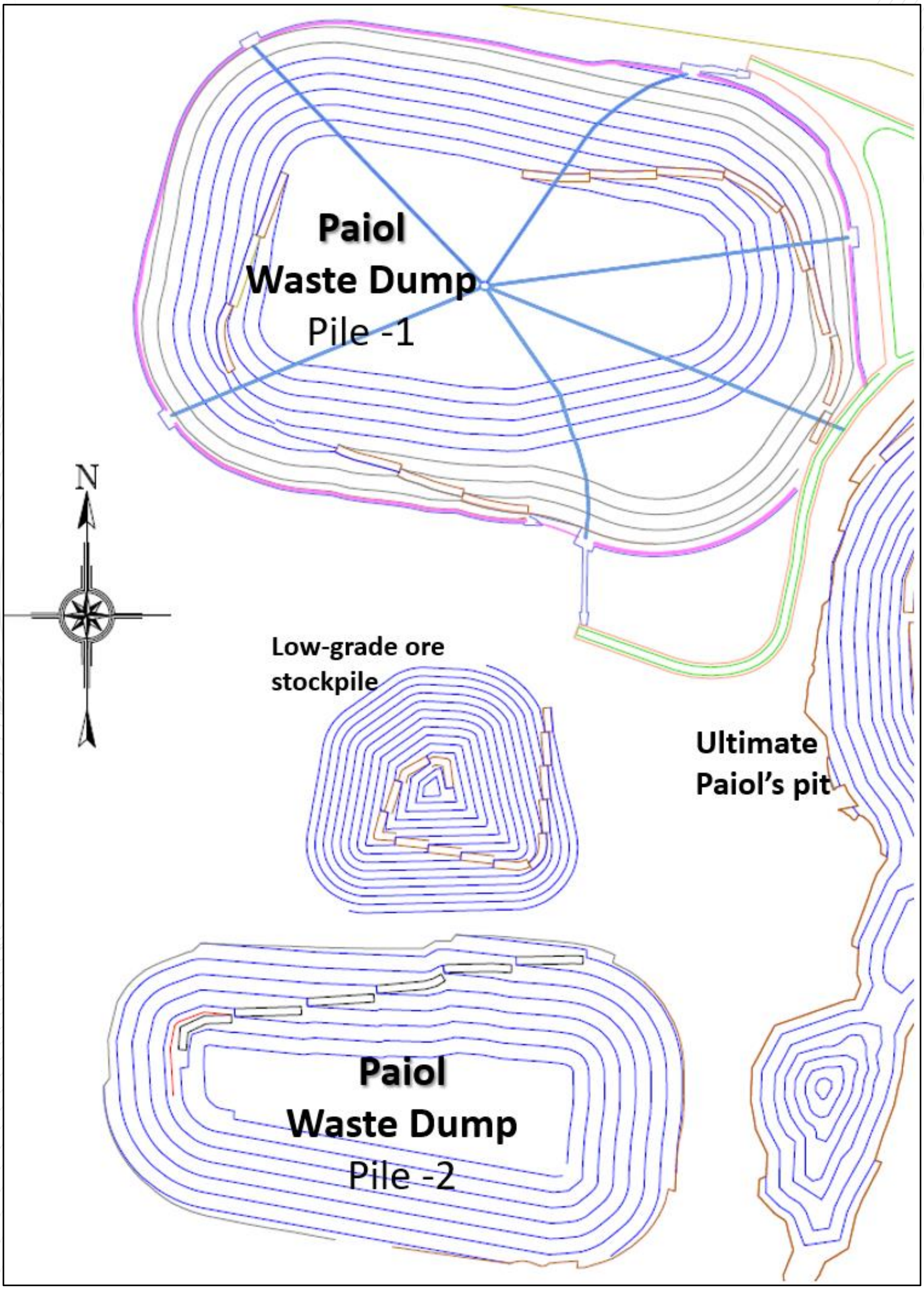


Figure 15-21 shows the waste dump piles for Cata Funda and Figure 15-22 shows the waste dump piles for Vira Saia.

Figure 15-21 Cata Funda Final Waste Dump pile

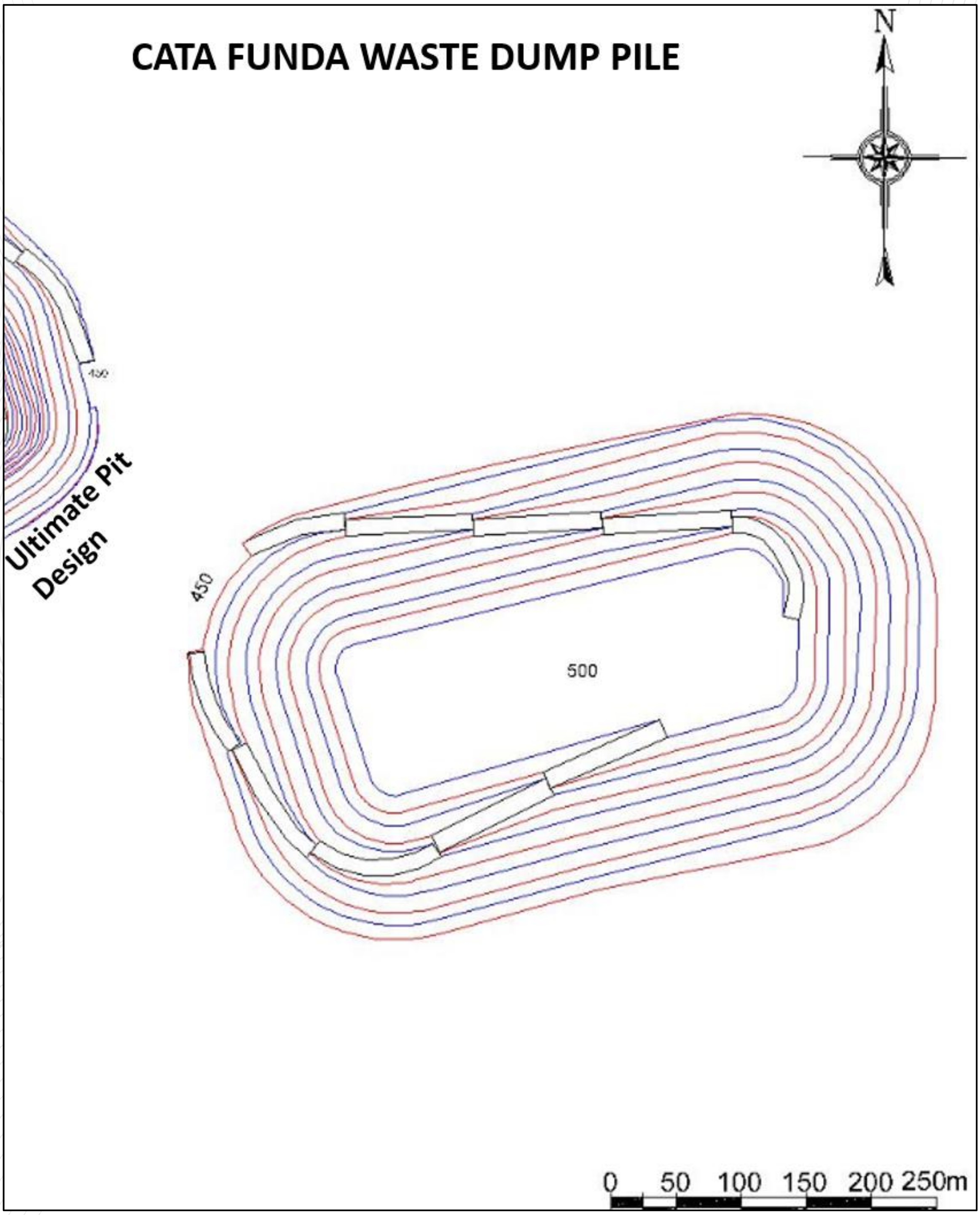


Figure 15-22 Vira Saia Final Waste Dump pile

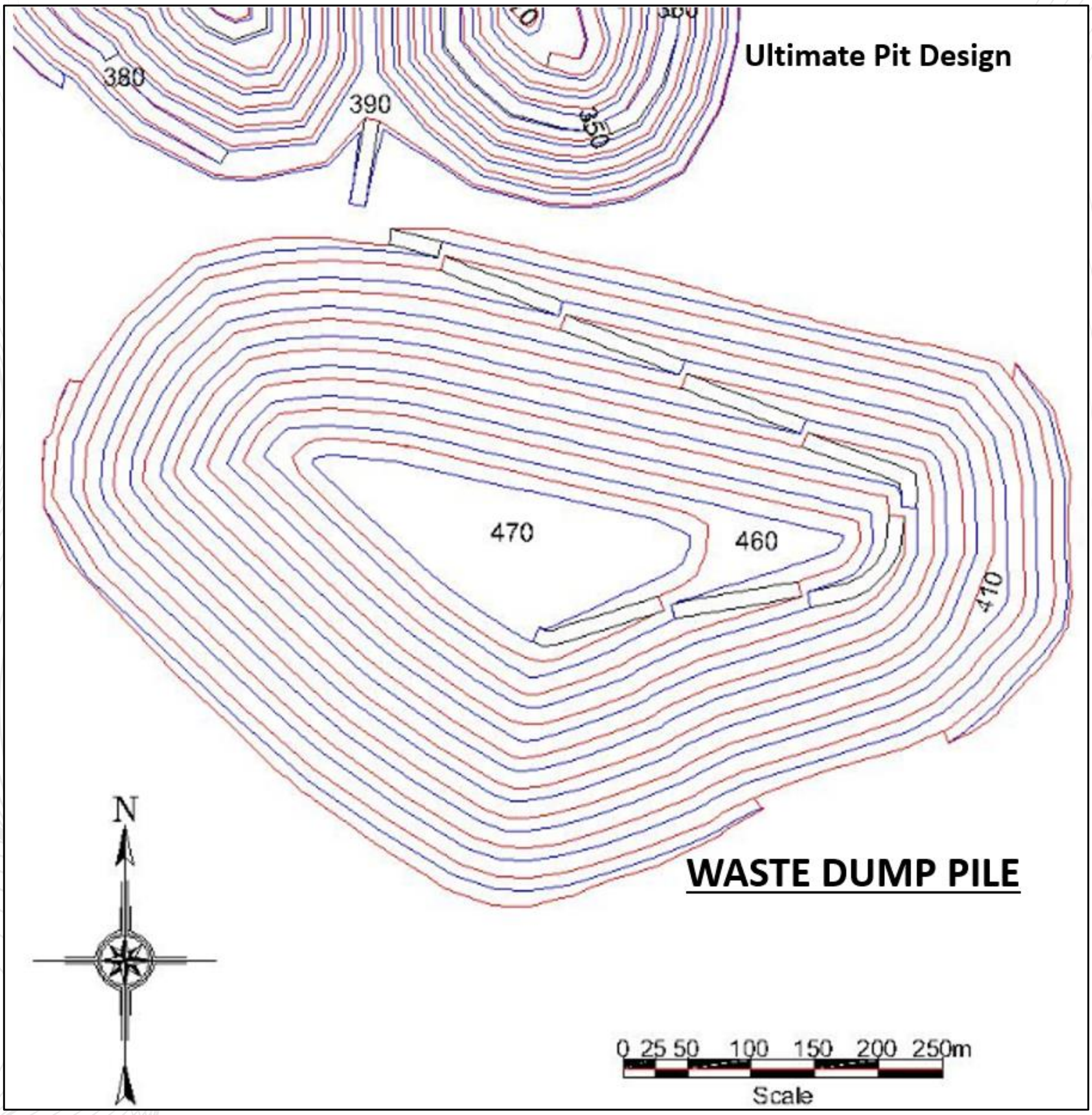
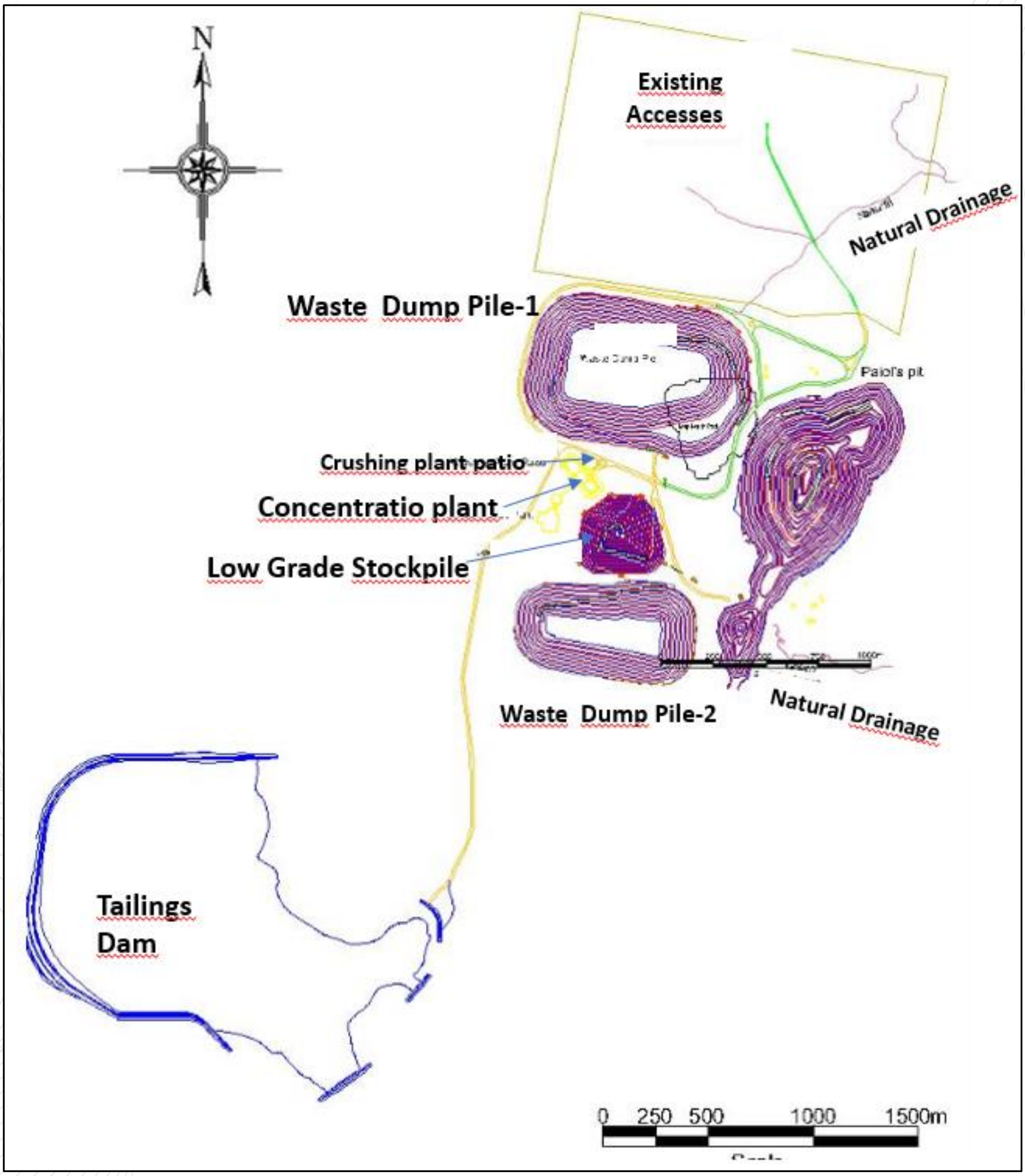


Figure 15-23 shows the general layout for Paiol including the tailings impoundment dam.

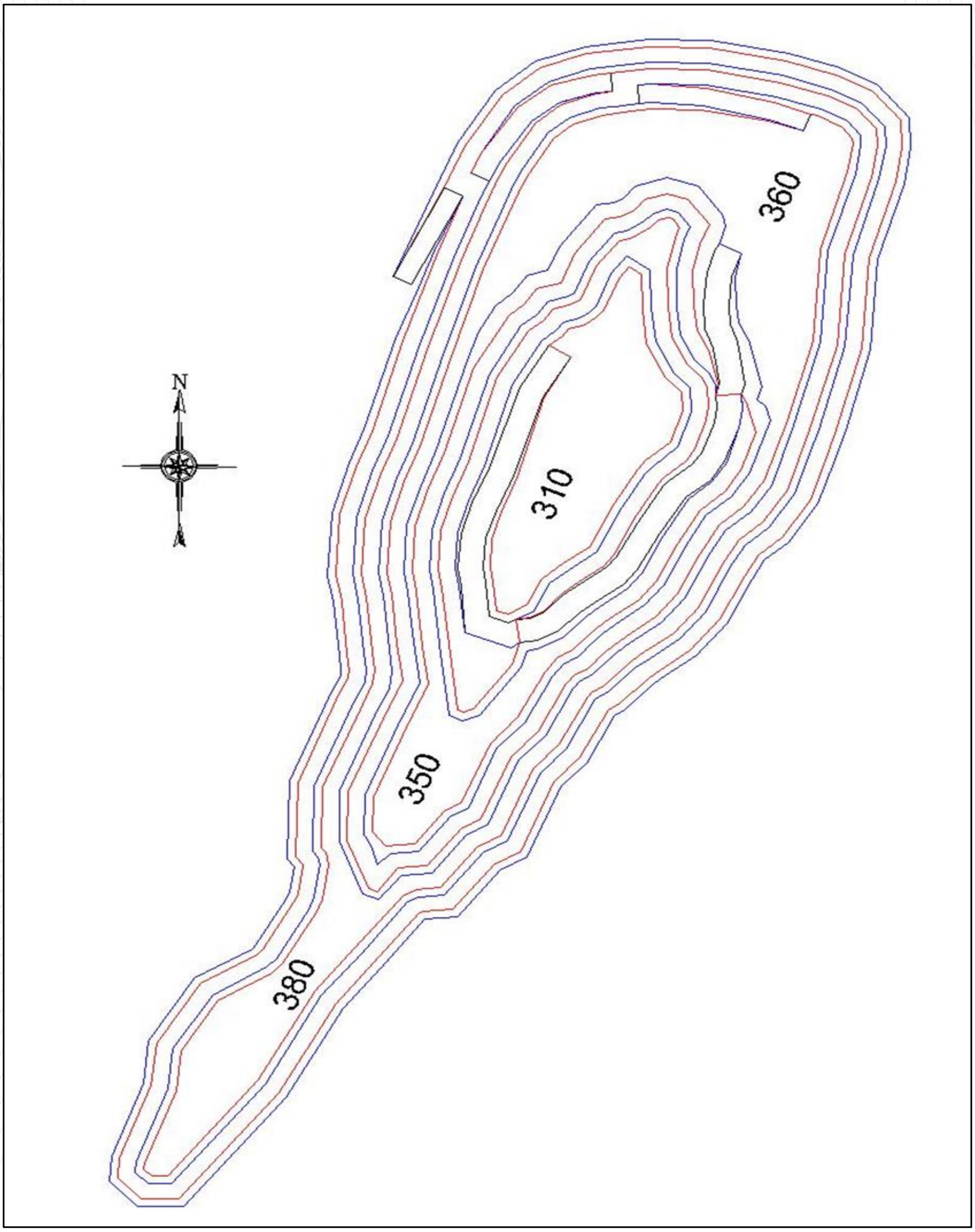
Figure 15-23 General Layout, Paiol Ultimate Pit Design with associated infrastructure: stockpile, waste dumps, drainage.



15.5.5 PRE-STRIPPING AND PLANT PRODUCTION RAMP-UP

A pre-stripping of 2.86 Mt at the Paiol mine (Figure 15-24) is planned to commence in October of 2021, 9 months before the concentration plant operation start-up. The pre-stripping will remove 2.18 Mt to a waste dump, 0.36 Mt will be stockpiled in a low-grade stockpile and 0.33 Mt will be stockpiled in the ROM stockpile.

Figure 15-24 Pre-stripping Paiol pit



The pre-operation is planned to have a ramp-up in the third quarter of 2022 as shown in Table 15-24:

Table 15-24 Plant Monthly Feed Tonnage and Grade Planned to Start Up

	MONTH	PLANT CAPACITY %	TONNAGE PER MONTH (t)	GRADE (g/t)
RAMP-UP (Q3-2022)	JULY	40%	44,000	0.90
	AUGUST	60%	66,000	0.90
	SEPTEMBER	80%	88,000	0.90
Q4 -2022	OCTOBER	100%	110,000	1.53
	NOVEMBER	100%	110,000	1.53
	DECEMBER	100%	110,000	1.53

15.6 MINE DRAINAGE

It is known that the water table within the area is shallow and the existing pits (Paiol and Vira Saia orebodies) have been filled with groundwater over the years after their operation. With respect to surface water, Paiol and Vira Saia are within the Riacho do Ouro watershed while the Cata Funda target is located within the Riberião das Areias watershed, both Manuel Alves River's tributary.

The drains were carefully designed and planned. They should support the drainage strategy at the mine in such a way as to mitigate any environment damage before the beginning of excavation work. A proper design allows Aura to optimize the use of existing excavation equipment in the mine and minimize additional excavations needs in case of an inadequate drainage project. Thus, the course of natural drainage will be respected in the preparation of the piles.

The drains in the base of the pile will drain the natural surface water and the falling rainwater into the pile influence area. A water fines retention box will be built, either to the entrance, or to the exit, in the drain piles: waste dump and low-grade ore. This avoids overcharging the drains under the pile and minimizes the presence of fines in the natural drains. A drainage channel is designed around the waste pile and low-grade ore pile, which will not allow water to escape from the pile's surroundings to the natural drainage.

15.6.1 PAIOL PIT DRAINAGE

This target involves the excavation of two adjacent open pits, a waste dump and a low-grade ore stockpile. The existing lake formed due to groundwater discharge within the pit which needs to be properly dewatered before the commencement of planned mining operations. To design a specific pumping system, a hydrogeological and hydrological study supported the pit water inflow estimation.

In addition to the in-pit system, the mine operation should consider perimeter drainage structures for the pits, waste dump, low grade stockpile and the process plant.

The waste dump footprint will not change over time considering that these facilities will be constructed in a bottom-up sequence. Hence, a perimeter drainage structure should be placed to control runoff and route waters to a reservation facility. It is recommended to use excavated structures instead of concrete channels to reduce costs and allow for the changes over the life of mine to fit the mine plan.

Once the mine approaches its closure, concrete channels should take their place as such structures will last, at least, for 15 years after closure.

Regarding the open pits, the largest pit will increase its footprint over time as the mine operation progresses. The uses of excavated perimeter drainage structures are recommended because the costs of redoing the perimeter drainage structures are minimized compared to the use of concrete. These structures should be designed to control run-in waters. Since the surrounding area is planned to remain over time, it is not expected that any changes of the design for such structures will be needed over the life of mine. It is recommended that those structures be placed reflecting the requirements of each phase of the project.

Given that the area surrounding the pits will remain undisturbed, it is expected to have a low impact over water chemistry. However, like the waste dump, it is expected that a reasonable volume of solids will be in water. It is recommended that all water drained from Paiol be pumped to a sedimentation dike.

Paiol's smaller pit will be mined on an area through which one water stream flows. On the other hand, the remaining topography allowed for the design of a water channel crossing between two pits during the Paiol mine operation. Water management for the operation should consider a detailed study for diverting this water from upstream. This may significantly reduce pumping demand from this pit and avoid potential action imposed by environmental regulators such as a pump and treat system to maintain downstream flows.

PIT DEWATERING

The Paiol mine remains as it was left by the past Vale mining operation, and the pit is flooded. It was calculated that a water volume of 1.2 million m^3 rainwater accumulation. The surface lake water is in the level 365 m. it will be necessary to dewater the pit to allow the mining development works.

For the implementation of the current project, part of this water will be used in civil works and humidification of the roads. Most of it should be discharged into the Paiol stream, which proved to be the best option, not only because it is the closest to the pit, but also for having a dam nearby that will function as a solid sedimentation basin before the final discharge into the watercourse, based on water quality monitoring. In addition, the average drainage flow is very close to the flow of the Paiol stream at the end of the rainy season (150 m^3/h).

We did a study to evaluate the possible conflicts between the land level released as the pumping is lowering the lake water level.

For the simulation the following premises were considered:

- Total Water Volume inside the existent pit calculated: 1.15 million m^3
- Total Water Volume inside the existent pit considered for pumping time schedule simulation: 1.2 million m^3
- Pumping capacity allowed: 150 m^3/h
- Pump working hours per day: 20 hours
- Pumping operation beginning: January 2021
- Annual average rain index : 1,700 mm
- Yearly rain volumes assigned:

	PLUVIOMETRY INDEX ASSIGNED (mm)												
Year	**Jan**	**Feb**	**Mar**	**Apr**	**May**	**Jun**	**Jul**	**Aug**	**Sep**	**Oct**	**Nov**	**Dec**	**Total**
2021	250	250	250	100	100	50	50	50	50	150	200	200	1,700
2022	250	250	250	100	100	50	50	50	50	150	200	200	1,700

- Rainwater contribution area that increases the volume during pumping period: 20 ha
- Level of lake surface in the topography: 365 m

- High-Grade ore stockpile up to June 2022: 100,000 t
- Low-Grade ore stockpile up to June 2022: 100,000 t.

Table 15-25 shows the volume of water stored due to rainwater after Vale's mining operation were interrupted. The ore volumes were calculated, by level, to allow the land surfaces released, as the pumping operation lowers the water level.

Table 15-25 Water Volumes in m³ Storage in the Paiol Pit Since Vale Mining Operation Interruption; High-Grade and Low-Grade Ores, by Level and Accumulated

BENCH LEVEL		HIGH-GRADE ORE (t)			LOW-GRADE ORE (t)			WATER VOLUME STORED	
FLOOR	CREST	PRE OPERATION		FINAL PIT	PRÉ OPERATION		FINAL PIT	BY LEVEL	ACCUMUL.
m	m	By Level	Accumul.	By Level	By Level	Accumul.	By Level	m3	m3
307	320	149,073	400,788	564,089	26,021	224,385	226,071	40,260	1,200,000
320	330	85,383	251,715	458,260	55,544	198,364	239,206	95,608	1,159,740
330	340	66,510	166,332	437,407	68,776	142,821	266,365	163,640	1,064,132
340	350	61,595	99,822	376,070	43,016	74,045	262,950	246,110	900,492
350	365	38,227	38,227	281,829	31,029	31,029	292,736	654,382	654,382

Table 15-26 shows the results of the water surface level simulation and the ore volume liberation from the water.

Table 15-26 Water Level Simulation Considering the Pumping System Starting on January, 2021 Versus the Ore Volume Liberation.

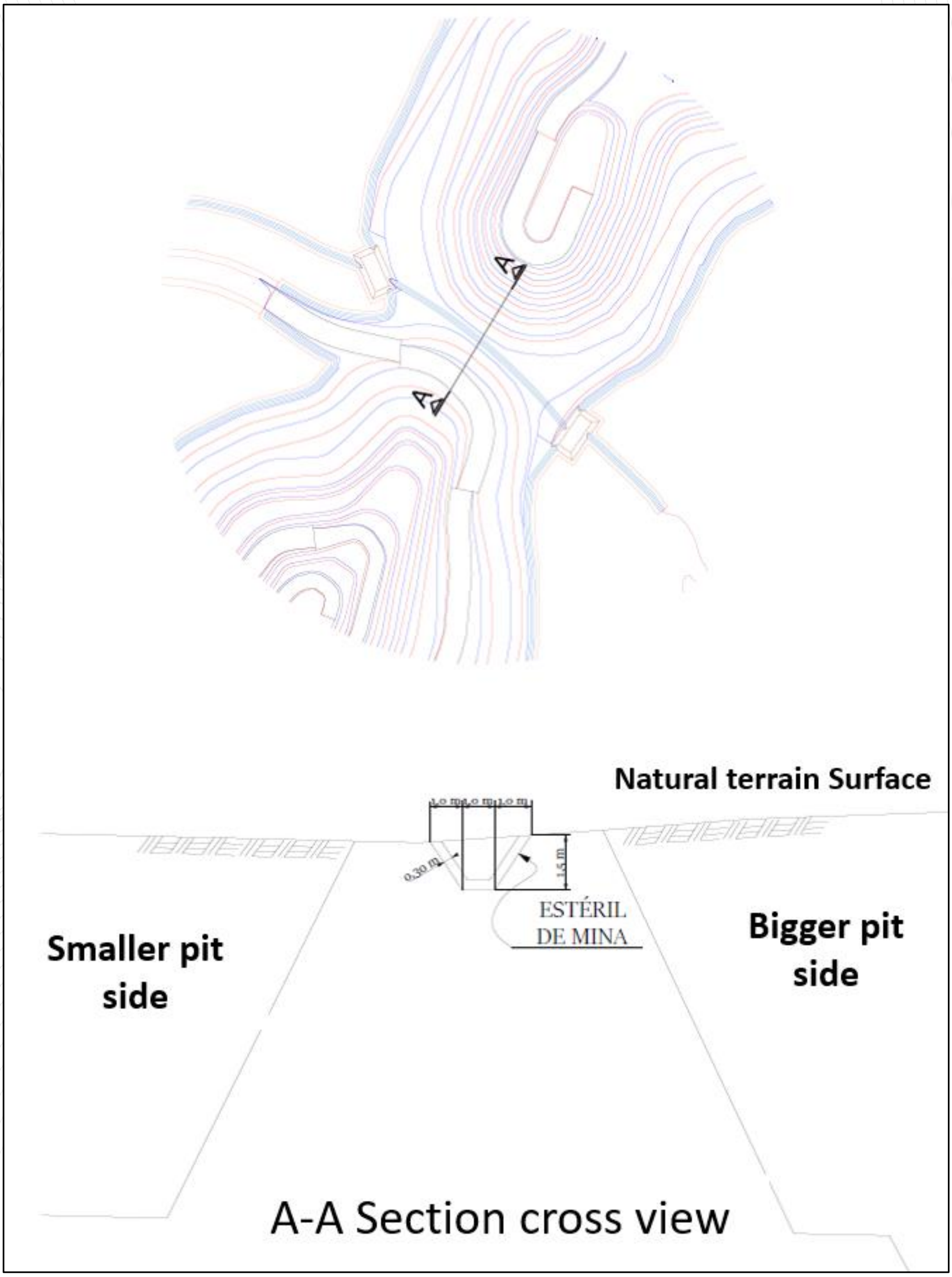
		BENCH LEVEL (m)		PUMPING (1000*m3)		RAIN VOLUME (1000*m3)		REMAINED WATER VOL.	WATER BENCH LEVEL		ACCUM. ORE RELEASED (kt)		
YEAR	MONTH	FLOOR	CREST	BY MONTH	ACCUM.	BY MONTH	ACCUM.	1000*m3	FLOOR	CREST	HIGH GRADE	LOW GRADE	TOTAL
2021	JAN	350	365	90	90	50	50	1,160	350	365	-	-	-
	FEB	350	365	90	180	50	100	1,120	350	365	-	-	-
	MAR	350	365	90	270	50	150	1,080	350	365	-	-	-
	APR	350	365	90	360	20	170	1,010	350	365	-	-	-
	MAY	350	365	90	450	20	190	940	350	365	-	-	-
	JUN	350	365	90	540	10	200	860	350	365	-	-	-
	JUL	340	350	90	630	10	210	780	340	350	38	31	69
	AUG	340	350	90	720	10	220	700	340	350	38	31	69
	SEP	340	350	90	810	10	230	620	340	350	38	31	69
	OCT	340	340	90	900	30	260	560	340	340	38	31	69
	VOV	330	340	90	990	40	300	510	330	340	100	74	174
	DEC	330	340	90	1,080	40	340	460	330	340	100	74	174
2022	JAN	330	340	90	1,170	50	390	420	330	340	100	74	174
	FEB	330	340	90	1,260	50	440	380	330	340	100	74	174
	MAR	330	340	90	1,350	50	490	340	330	340	100	74	174
	APR	330	330	90	1,440	20	510	270	330	330	100	74	174
	MAY	320	330	90	1,530	20	530	200	320	330	166	143	309
	JUN	307	320	90	1,620	10	540	120	307	320	252	198	450
	JUL	307	320	90	1,710	10	550	40	307	320	252	198	450
	AUG	307	307	90	1,800	10	560	40	307	307	401	224	625

Based on the premises and simulation results, it is possible to say: if the 150 m^3/h starts on January 2021, the probability is very high to have 200,000 tons of ore on the stockpile.

15.6.1.2 NATURAL DRAINAGE BETWEEN THE TWO PITS FROM THE PAIOL OPERATION

The Ultimate Paiol design, with the main part of the natural drainage preserved, is shown in Figure 15-25.

Figure 15-25 Drainage Designed Between Two Pits, Paiol Mining Operation.



15.6.2 CATA FUNDA'S PIT

This target is the smallest of the three and has one single open pit and one waste dump. The low-grade ore volume is not significative. The ore production to feed the plant from this pit is scheduled in 2025.

Specific detailed hydrogeological and hydrological studies should be completed to estimate water inflows and runoff for surface drainage structures. Those structures should be based on perimeter design and routing structures to a reservations/containment facility.

Hydrochemistry and geochemistry characterization should also be carried due to the potential presence of sulfides within the ore.

The operation of this target is going to be last for four years, it is recommended that excavated drainage structures be used, and the construction of concrete channels be employed for the closure scenario.

15.6.3 VIRA SAIA PIT

The ore from the Vira Saia pit will feed the plant in the 2029. This target involves two minor open pits, one waste dump, and low-grade ore stockpiles considered to be the same as Paiol's low-grade ore stockpile.

The same recommendations given for the Paiol target are valid here. The pit must be dried out before 2028 and a pumping system should be designed. The system shall not only consider the scenario of drying out the pit but maintaining the water table low enough to allow operations throughout planned years. Perimeter drainage structures must be established for the pits, waste dump and stockpiles considering the final scenarios of each phase to prevent under sizing. The use of excavated structures at first is also recommended along with concrete channels over the LOM to closure.

Specific detailed hydrogeological and hydrological studies should be carried out to estimate water inflows and runoff.

16 MINING METHODS

16.1 MINING MOVED ROCK FOR ALL MINES

Mining costs are based on the mining physicals.

16.1.1 MINING VOLUMES MOVED FROM ALL MINES

The ore to be fed to the concentration plant is presented in the table 16-16. The mining operation is mainly related to the orebodies of Paiol, Cata Funda and Vira-Saia. Table 16-1 presents the ore excavation volumes by year and includes spent heap leach residue stockpiled by Vale during their operating period.

Table 16-1 Yearly Plant Ore Feed Planned and its Origin.

	PLANT		ORE BY ORIGIN					
YEAR	FEED TONNAGE	AU GRADE	PAIOL PIT	CATA FUNDA PIT	VIRA SAIA PIT	HEAP LEACH PAD	HIGH-GRADE PILE	LOW-GRADE PILE
	kt	*g/t*	*kt*	*kt*	*kt*	*kt*	*kt*	*kt*
2022	529	0.90	329	-	-	-	100	100
2023	1,320	1.35	1,320	-	-	-	-	-
2024	1,320	1.38	1,320	-	-	-	-	-
2025	1,323	1.54	1,112	201	-	10	-	-
2026	1,300	1.19	989	156	-	155	-	-
2027	1,300	1.36	991	146	-	163	-	-
2028	1,300	1.41	989	144	-	167	-	-
2029	1,300	1.65	662	-	495	144	-	-
2030	1,300	1.09	660	-	433	208	-	-
2031	1,300	1.29	484	-	437	80	240	60
2032	1,300	1.36	-	-	383	249	200	468
2033	1,300	1.16	-	-	401	100	-	798
2034	1,300	1.18	-	-	341	-	-	959
2035	1,300	1.10	-	-	-	-	-	1,300
2036	1,300	1.65	-	-	-	-	-	1,300
2037	1,300	1.27	-	-	-	-	-	1,300
2038	1,300	1.19	-	-	-	-	-	1,300
2039	440	1.13	-	-	-	-	-	740
TOTAL	**21,833**	**0.91**	**8,856**	**646**	**2,490**	**1,275**	**540**	**8,226**

16.1.2 RUN OF MINE (ROM) DESTINATION.

The mined ROM destinations are:

- High-Grade ore stockpile to crushing plant area: it 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 Au production in the concentration plant.
- Low-grade ore stockpile: following a strategy to maximize the net present value, the low grade value will be destined to the low-grade ore stockpile strategically located close to the concentration plant. The low-grade ore is going to be taken up to feed the plant at the end of the LoM.
- Waste dump piles – the waste dump piles are going to be located close to it its related pit and will be carefully treated as a part of the environment reclamation at the end of LoM.

The ROM by origin and destination is shown in table 16-2.

Table 16-2 Yearly ROM Volumes Mined in the Pits and its Destination.

		FROM IN PIT TO (DESTINATION)					
		PLANT	WASTE DUMP PILES			ORE STOCKPILES	
YEAR	TOTAL PLANT FEED	*HIGH-GRADE*	*PAIOL*	*CATA FUNDA*	*VIRA SAIA*	*HIGH-GRADE*	*LOW-GRADE*
	kt	*kt*	*kt*	*kt*	*kt*	*kt*	*kt*
2022	529	329	5,060	-	-	332	294
2023	1,320	1,320	9,913	-	-	-	950
2024	1,320	1,320	9,227	-	-	-	608
2025	1,323	1,313	10,626	2,438	-	207	1,034
2026	1,300	1,145	10,368	1,805	-	-	435
2027	1,300	1,137	11,855	1,650	-	-	789
2028	1,300	1,133	13,759	1,300	-	-	1,015
2029	1,300	1,156	13,422	-	3,547	-	1,095
2030	1,300	1,092	5,686	-	3,340	-	763
2031	1,300	920	821	-	4,668	-	468
2032	1,300	383	-	-	3,055	-	277
2033	1,300	401	-	-	3,128	-	258
2034	1,300	341	-	-	1,535	-	139
2035	1,300	-	-	-	-	-	-
2036	1,300	-	-	-	-	-	-
2037	1,300	-	-	-	-	-	-
2038	1,300	-	-	-	-	-	-
2039	440	-	-	-	-	-	-
TOTAL	**21,833**	**11,992**	**90,735**	**7,192**	**19,273**	**540**	**8,126**

16.1.3 ALL ROCK VOLUME MOVED, BY DESTINATION AND ORIGIN

Table 16-3 presents the total rock volumes moved, by origin and destination.

Table 16-3 Total Rock Volumes Moved Yearly, by Origin and Destination.

MOVED ROCK PER ORIGIN AND DESTINATION (kt)													
DESTIN →	PLANT					WASTE DUMP PILES				LOW-GRADE STOCKPILE			STOCK PATIO
ORIGIN → / YEAR ↓	*PAIOL PIT*	*CATA FUNDA'S PIT*	*VIRA SAIA'S PIT*	*HEAP LEACH PAD*	*LOW-GRADE PILE*	*PAIOL*	*CATA FUNDA*	*VIRA SAIA*	*HEAP LEACH PAD*	*PAIOL LOW GRADE*	*CATA FUNDA LOW GRADE*	*VIRA SAIA LOW-GRADE*	*PAIOL PIT HIGH- GRADE*
2022	329	-	-	-	100	5,060	-	-	-	394	-	-	332
2023	1,320	-	-	-	-	9,913	-	-	-	950	-	-	-
2024	1,320	-	-	-	-	9,227	-	-	-	608	-	-	-
2025	1,112	201	-	10	-	10,626	2,438	-	26	1,032	2	-	207
2026	989	156	-	155	-	10,368	1,805	-	26	417	18	-	-
2027	991	146	-	163	-	11,855	1,650	-	26	782	7	-	-
2028	989	144	-	167	-	13,759	1,300	-	26	1,000	15	-	-
2029	662	-	495	144	-	13,422	-	3,547	26	930	-	165	-
2030	660	-	433	208	-	5,686	-	3,340	-	536	-	227	-
2031	484	-	437	80	60	821	-	4,668	-	245	-	223	-
2032	-	-	383	249	468	-	-	3,055	-	-	-	277	-
2033	-	-	401	100	798	-	-	3,128	-	-	-	258	-
2034	-	-	341	-	959	-	-	1,535	-	-	-	139	-
2035	-	-	-	-	1,300	-	-	-	-	-	-	-	-
2036	-	-	-	-	1,300	-	-	-	-	-	-	-	-
2037	-	-	-	-	1,300	-	-	-	-	-	-	-	-
2038	-	-	-	-	1,300	-	-	-	-	-	-	-	-
2039	-	-	-	-	740	-	-	-	-	-	-	-	-
TOTAL	**8,856**	**646**	**2,490**	**1,275**	**8,226**	**90,735**	**7,192**	**19,273**	**128**	**6,893**	**42**	**1,291**	**540**

16.2 MINE OPERATION

The mining operation concept for the Almas Gold Project is conventional open pit mining with production schedule that provides an initial 0.6 Mt stock of ROM with 0.92 g/t Au content, equivalent to six months. Commercial operation is scheduled to start up in July, 2022 with a ramping up until October, 2022.

The mine development is planned to allow access to those grade levels to maximize gold production and provide operational flexibility by mining several benches simultaneously. The number of pits mined simultaneously will be no more than two pits.

The waste rock comprises soil, saprolite, weathered and fresh rock. The excavation of these deposits requires the use of drill and blast.30% of saprolite will require explosives. Load and haulage will be performed by a combination of front-end loaders, hydraulic excavators, and on-road trucks.

Benches will be configured as follows:

- A minimum mining width of 30 m on a 10 m-high bench has been maintained for Cata Funda, but Vira Saia and Paiol 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 analysed 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 in rainwater collection contribution areas around the pit allow for the minimization of operational disturbances during heavy rain.

The ultimate Paiol pit will fully overlap the existing pit from the old Vale operation.

The processing plant is located at 0.7 km from the final Paiol pit. The tailings dam already exists from Vale's operation. It is located about 2.0 km from the plant.

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 Centre to the lateral edge of the road, with drainage ditches along the roads. Road conditions must be compatible with good practices for the operation of mining equipment.

The Almas Gold 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 engineering and short-term mine plan. The technology being considered is Down the Hole hammer with reverse circulation.
- Blastholes: the holes are going to be drilled, most probably by an hydraulic Top Hammer drilling rig.
- 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.
- Rock mechanical excavation: must be made by bulldozers or directly by hydraulic excavators.
- Loading operation will be done, preferentially, by retro bucket profile hydraulic excavator, and complemented by front end loaders (FEL).
- Rock transport will be done by conventional on-road trucks, 8 x 4 PBT (Total Gross Weight) bigger than 48 tons.
- 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 be done considering activities on blasted rocks fronts and spreading blasted rock in waste dump piles. That requires tractors of greater weight than soil-cutting tractors only.

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 the complement excavation work.

The destinations of mined materials are:

- RoM stockpile at the primary crusher area,
- Waste dumps
- 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 operating weight hydraulic excavators, which will load 8 x 4 trucks with 22 m^3 dump box size and 48 t capacity.

The ore from the heap leach pad, will be excavated directly by hydraulic excavators.

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 piles will be rehandled and hauled to 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 development and preparation is estimated of six months in advance before the plant start up.

16.3 MINE EQUIPMENT SELECTION

Likewise other Aura's 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 ore loading operation, a hydraulic excavator similar to a Caterpillar CAT374D, equipped with a 3.5 m^3 bucket, is considered. The unit will be used to excavate and load 25m^3 (35 tons) capacity trucks. Also, a front-end loader, similar to a Caterpillar 966H equipped with 3.5 m^3 bucket will be needed to complement the mine loading operation. The model will be similar to the machine will feed the crushing plant.

Similar loading machines are going to be applied to waste rock loading.

The digging depth of these machines allows for top loading vocational trucks in 2.5 layers and the ability to maintain wall 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 & BLASTING

The blasting pattern parameters for this unit operation are based on similar 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, The review was carried out with a view to the fragmentation needs considered for the calculation.

Blastholes of 5.1/2" diameter for a 10-m high waste rock bench and 5- or 10-m high bench for ore depending on the local conditions of mineralization. For the 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/cm³	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	m³	51	79	105	197
SPECIFIC ROCK DENSITY "IN SITU"	t/m³	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/m³	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 for waste and ore and knowing the lithology it is going to be possible to calculate the needs for blasting and allows the selection of the blasthole drilling rigs fleet.

Table 16-5 Annual Rock Tonnage Separated by Main Lithology or Waste.

	TOTAL ROM TONNAGE (kt)					
	ORE			WASTE		
YEAR	SOIL	SAPROLITE	FRESH ROCK	SOIL	SAPROLITE	FRESH ROCK
2022	-	113	942	2,167	1,104	-
2023	-	148	2,122	1,032	4,500	4,381
2024	-	99	1,828	523	4,836	3,868
2025	-	294	2,110	-	6,728	6,336
2026	-	-	1,579	-	-	12,173
2027	-	-	1,926	-	-	13,505
2028	-	-	2,149	-	-	15,059
2029	-	-	2,251	-	2,138	14,831
2030	-	-	1,855	-	763	8,263
2031	-	-	1,388	-	-	5,489
2032	-	-	660	-	-	3,055
2033	-	-	660	-	-	3,128
2034	-	-	480	-	-	1,535
2035	-	-	-	-	-	-
2036	-	-	-	-	-	-
2037	-	-	-	-	-	-
2038	-	-	-	-	-	-
2039	-	-	-	-	-	-
TOTAL	**-**	**655**	**19,952**	**3,343**	**21,132**	**92,726**

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	8,760	8,760	8,760	8,760
PLANNED ANNUAL HOURS	h	8,520	8,520	8,520	8,520
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/m³	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

As we considered that 30% of total Saprolite is going to be drilled and blasted, we assume the drilling and blasting cost from 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.

LITHOLOGY → YEAR↓	ORE TONNAGE (kt)			WASTE TONNAGE (kt)			TOTAL DRILLING RIGS CALCULATED (QUANTITY)					DRILLING RIG QTY ROUNDED
							ORE		WASTE		TOTAL	
	SOIL	SAPROL.	FRESH ROCK	SOIL	SAPROL.	FRESH ROCK	SAPROL.	FRESH ROCK	SAPROL.	FRESH ROCK	ORE + WASTE	
2022	-	113	942	1,789	2,167	1,104	0.02	0.6	0.4	0.4	1.4	2.0
2023	-	148	2,122	1,032	4,500	4,381	0.03	1.3	0.8	1.7	3.8	4.0
2024	-	99	1,828	523	4,836	3,868	0.02	1.1	0.9	1.5	3.5	4.0
2025	-	294	2,110	-	6,728	6,336	0.05	1.3	1.2	2.4	5.0	6.0
2026	-	-	1,579	-	-	12,173	-	1.0	-	4.7	5.6	6.0
2027	-	-	1,926	-	-	13,505	-	1.2	-	5.2	6.4	7.0
2028	-	-	2,149	-	-	15,059	-	1.3	-	5.8	7.1	7.0
2029	-	-	2,251	-	2,138	14,831	-	1.4	0.4	5.7	7.4	8.0
2030	-	-	1,855	-	763	8,263	-	1.1	0.1	3.2	4.4	5.0
2031	-	-	1,388	-	-	5,489	-	0.8	-	2.1	2.9	3.0
2032	-	-	660	-	-	3,055	-	0.4	-	1.2	1.6	2.0
2033	-	-	660	-	-	3,128	-	0.4	-	1.2	1.6	2.0
2034	-	-	480	-	-	1,535	-	0.3	-	0.6	0.9	1.0
2035	-	-	-	-	-	-	-	-	-	-	-	1.0
2036	-	-	-	-	-	-	-	-	-	-	-	1.0
2037	-	-	-	-	-	-	-	-	-	-	-	1.0
2038	-	-	-	-	-	-	-	-	-	-	-	1.0
2039	-	-	-	-	-	-	-	-	-	-	-	1.0
TOTAL	**-**	**655**	**19,952**	**3,343**	**21,132**	**92,726**						

The drilling fleet considered equipment similar to the DP 1500 in size, to drill holes of 5.0" and 5.1/2" diameter.

A utilization of 45% to 50% drilling rigs is what we have experienced from similar applications. Considering a remarkably high utilization for unit operation – drilling, can transform drilling operations into one of the bottlenecks. If we consider a total cost analysis, the highest costs are transport and grinding. It is likely to conclude that it would be important to maintain some extra capacity for drilling equipment with 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.

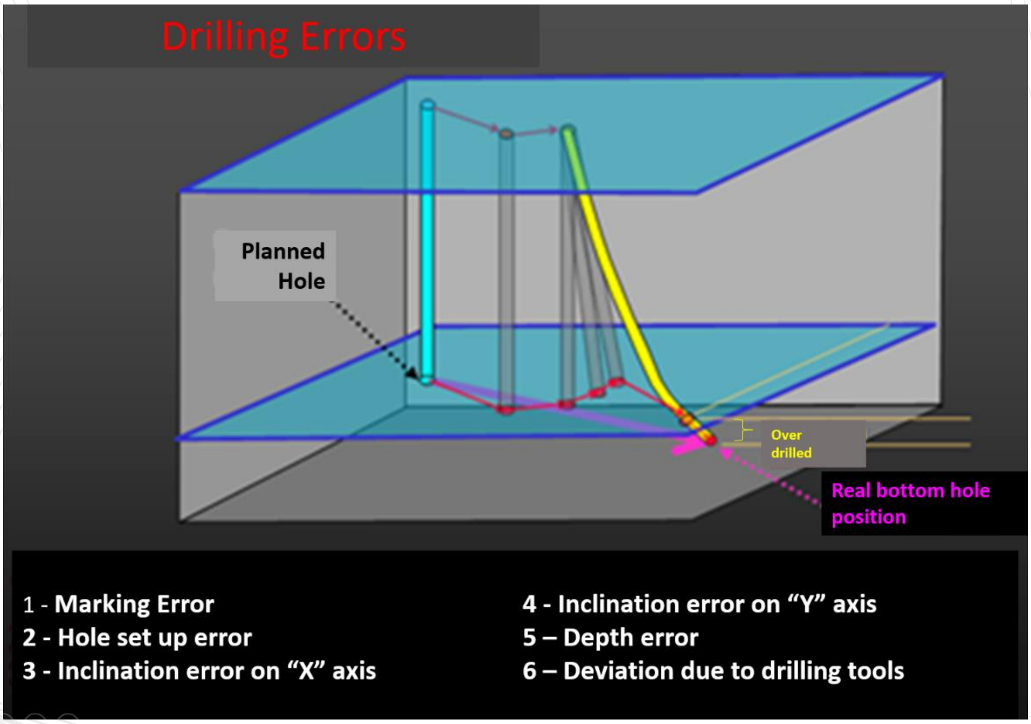
The equipment can help to reduce the dilution of the ore by improving the selectivity of the mine with GPS systems for correct drilling of the holes and safe control of the inclination of the holes, even with the irregularities in surfaces where the drilling rigs usually work.

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 the demand for Bulldozer hours being reduced. 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 16-18.

16.3.2.1 ORE BLASTING FRAGMENTATION FACE TO 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, P80 lower than 250 mm, P50 lower than 80 mm.

The analysis considered the available data of the sericite-quartz-shale rock mass from the Paiol mineral orebody compared with other similar applications.

Considering a drilling diameter of 5 1/2 ", we estimate an explosive charge ratio of 410 g/t.

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 bigger than 600 mm size is around 3%; that could be reduce the size properly by a suitably selected hydraulic breaker, mounted on a hydraulic excavator with a hydraulic breaker greater than 2,400 kg, working in the ore yard.

The hole diameter and the fragmentation analysis in reference (16) is 5 1/2", in 10 m high benches in the waste rock and 5 m high benches in the ore. In the report a 5" hole diameter was considered. The equipment model selected, and the size of the fleet, meet both possibilities. In any case, 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 grinding are significantly higher than drilling and blasting, but it can be bigger if the fragmentation of the blasting operation is not adequate. In waste, 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 can be optimized during operation.

16.3.3 GRADE CONTROL DRILLING RIG

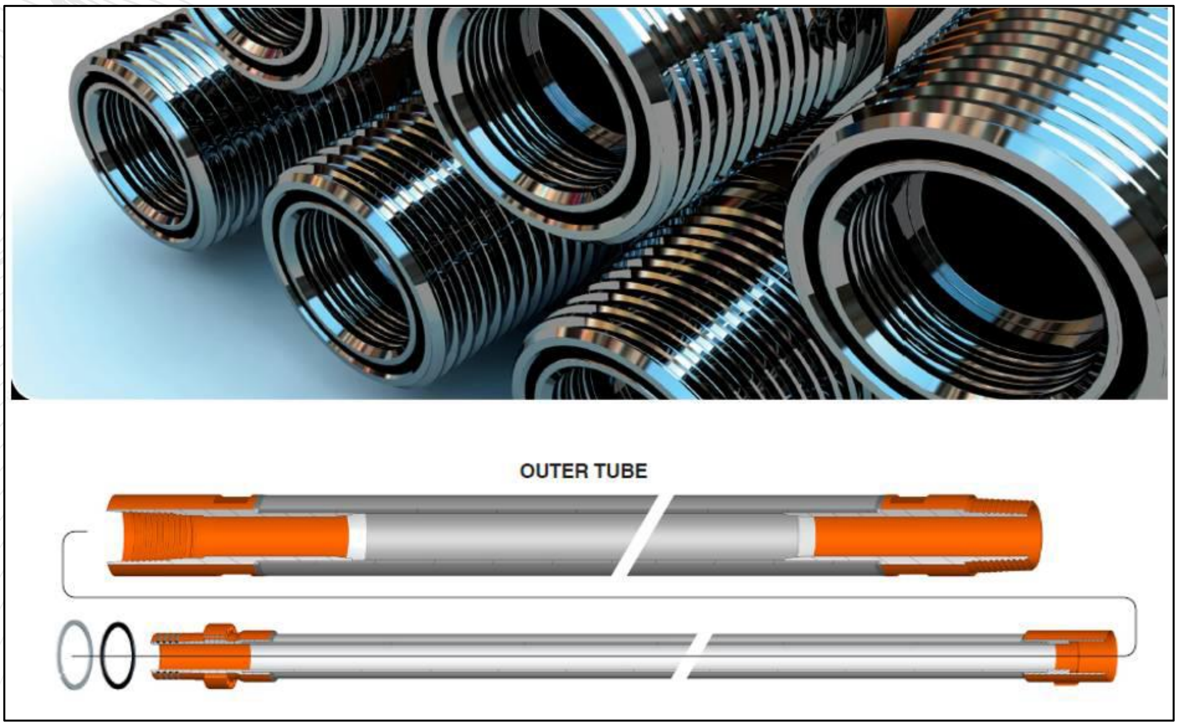
For Grade Control Drilling, used to update the short term mine plan in order to carry out grade control in gold mines, the use of reverse circulation has been shown to be the most effective for supporting mining reconciliation. It is strongly recommended to use this technology.

Reverse circulation (RC) drilling has become standard practice in most mines across the world to make the grade control for mining reconciliation. First developed in Australia in the 1970's, the drilling technique was originally applied as a drilling techniques solution 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 uses 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, similar to the Epirocc 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

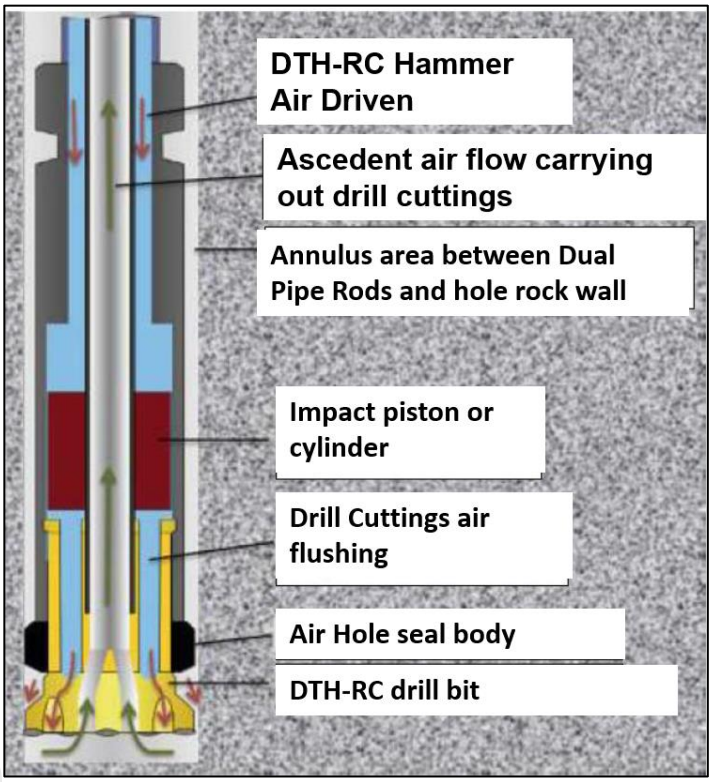
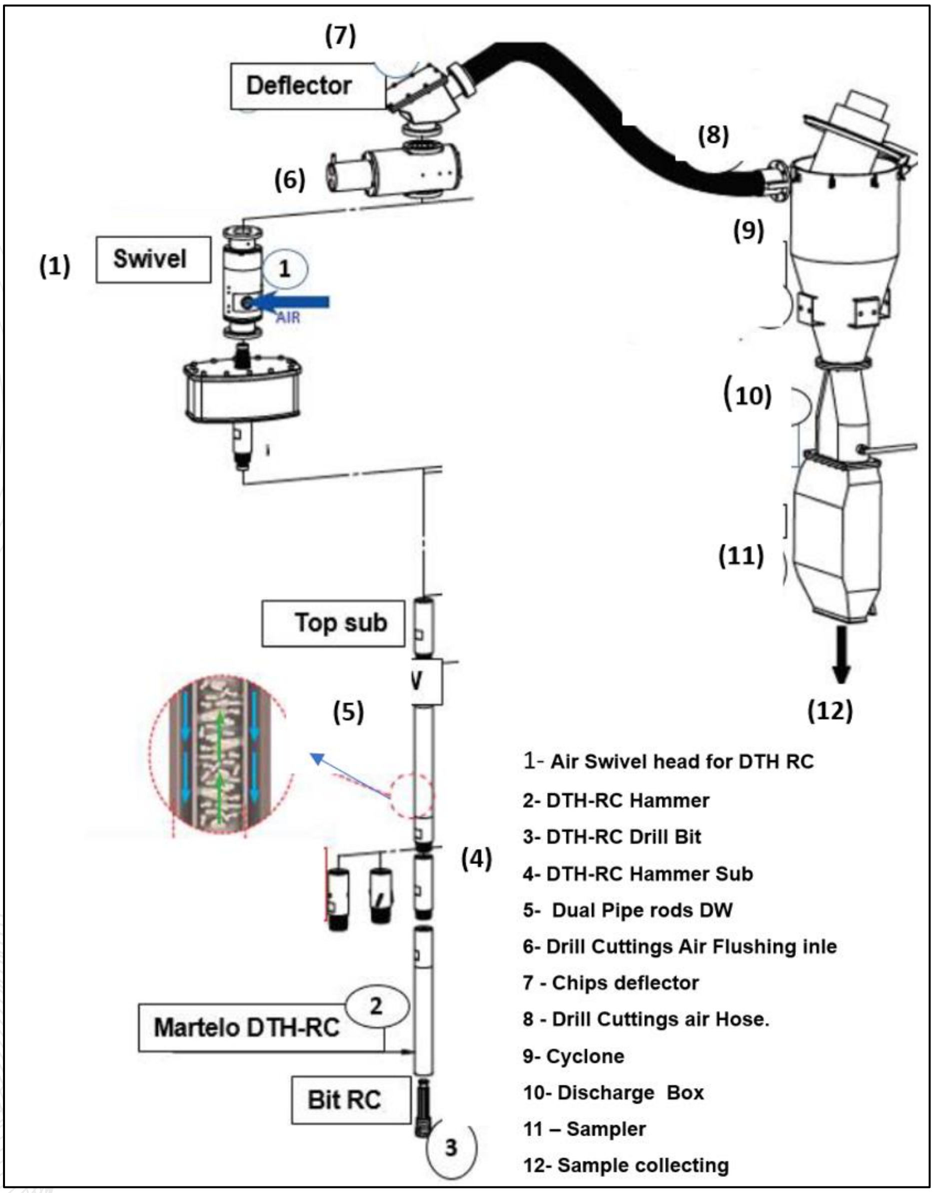


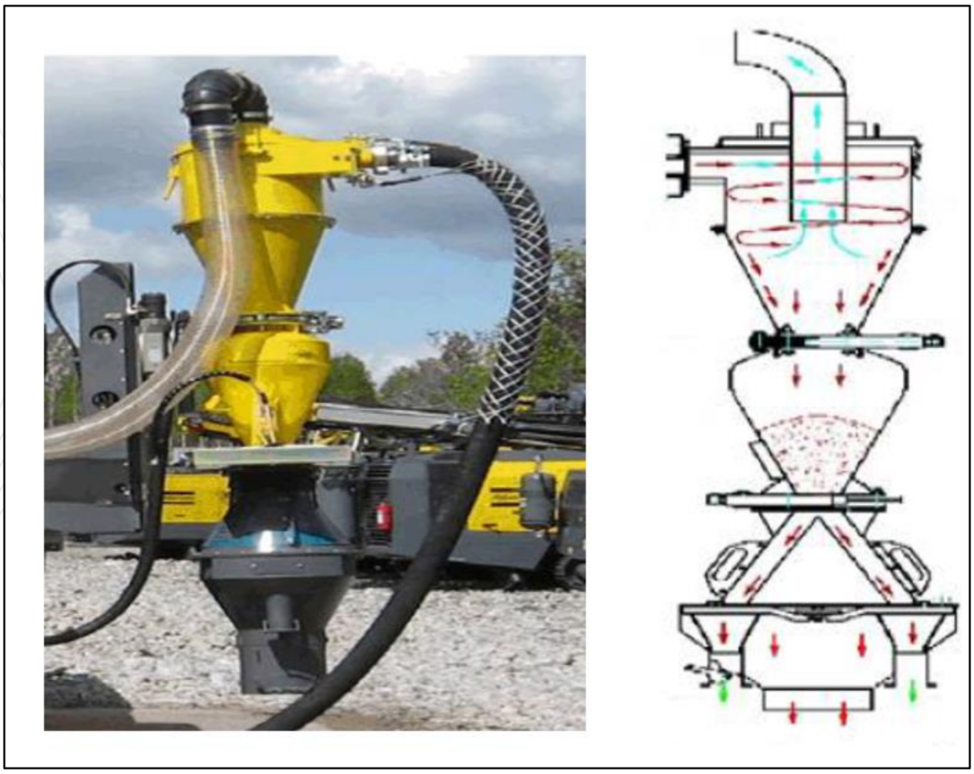
Figure 16-4 shows the main components to complement the regular DTH drill rig and convert it to an RC drilling rig for grade control – Epiroc Product Catalogue.

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 parameter 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: 5.3/4"
- Two shift operation

Table 16-8 presents the yearly drill control fleet calculation needs.

Table 16-8 Drill Control Fleet Needs Per Year

	ORE TONNAGE (KT)			GRADE CONTROL DRILL RIGS	
LITHOLOGY → YEAR ↓	SOIL	SAPROLITE	FRESH ROCK	m/YEAR	QUANTITY
2022	-	113	942	28,504	0.6
2023	-	148	2,122	61,309	1.3
2024	-	99	1,828	52,051	1.1
2025	-	294	2,110	64,922	1.4
2026	-	-	1,579	42,640	0.9
2027	-	-	1,926	52,010	1.1
2028	-	-	2,149	58,013	1.2
2029	-	-	2,251	60,779	1.3
2030	-	-	1,855	50,087	1.0
2031	-	-	1,388	37,481	0.8
2032	-	-	660	17,820	0.4
2033	-	-	660	17,820	0.4
2034	-	-	480	12,960	0.3
2035	-	-	-	-	-
2036	-	-	-	-	-
2037	-	-	-	-	-
2038	-	-	-	-	-
2039	-	-	-	-	-
TOTAL		655	19,952		

It is going to be a challenging situation in the beginning of the operation, once it is expected to have the benches dried by the pumping system in March/April 2022. For the rest of the mining period, we believe that for a well-managed drill control operation, one unit it would be enough, despite the fact that Table 16-8 shows a number slightly over the one unit.

16.3.4 LOADING ROM EQUIPMENT

Table 16-9 Presents the Parameters Used to Select the Loading Fleet for the ROM on the Mining Fronts.

	UNIT	LOADING EQUIPMENT TYPE	
PARAMETERS ↓	#	hydraulic excavator	Front End Loader
Similar Front End Loader model	#	CAT 374	CAT 966L
Truck type	#	Vocational 8x4	Vocational 8x4
Truck Dump box	m^3	22	22
Minimum Total Gross Weight Capacity	t	48	48
Payload capacity	t	35	35
Tonnage per loading cycle	t	6.1	6.4
nr of passes	qty.	5.7	5.5
Rounded cycles quantity (nr)	qty.	6	6
Truck maneuver fixed time	min.	0.5	0.4
Discharged fixed time	min.	1.1	1.1
Waiting time per queue	min.	1.0	1.0
Loading fixed time per cycle	min.	0.6	0.6
Loading total time	min.	3.4	3.9
Total time per truck	min.	6.0	6.0
Loading Capacity	t/h	563	533

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.

LOADER WORKING HOURS NEEDS														
DESTIN →	PLANT					WASTE DUMP PILES				STOCKPILE				TOTAL
ORIGIN → / YEAR ↓	PAIOL'S PIT	CATA FUNDA'S PIT	VIRA SAIA'S PIT	HEAP LEACH PAD	LOW-GRADE PILE	PAIOL	CATA FUNDA	VIRA SAIA	HEAP LEACH PAD	PAIOL LOW-GRADE	CATA FUNDA LOW- GRADE	VIRA SAIA LOW-GRADE	PAIOL HIGH GRADE	
2022	585	-	-	-	178	8,986	-	-	-	700	-	-	590	**11,038**
2023	2,345	-	-	-	-	17,605	-	-	-	1,688	-	-	-	**21,637**
2024	2,345	-	-	-	-	16,386	-	-	-	1,079	-	-	-	**19,810**
2025	1,975	356	-	18	-	18,872	4,330	-	45	1,832	4	-	368	**27,801**
2026	1,756	277	-	276	-	18,413	3,205	-	45	741	31	-	-	**24,744**
2027	1,761	259	-	289	-	21,054	2,930	-	45	1,388	13	-	-	**27,739**
2028	1,757	256	-	296	-	24,435	2,309	-	45	1,777	26	-	-	**30,901**
2029	1,175	-	879	255	-	23,837	-	6,299	45	1,651	-	293	-	**34,434**
2030	1,171	-	768	369	-	10,097	-	5,932	-	951	-	404	-	**19,693**
2031	859	-	776	141	107	1,457	-	8,291	-	434	-	396	-	**12,462**
2032	-	-	680	443	831	-	-	5,425	-	-	-	492	-	**7,871**
2033	-	-	713	178	1,418	-	-	5,556	-	-	-	459	-	**8,323**
2034	-	-	605	-	1,704	-	-	2,726	-	-	-	248	-	**5,283**
2035	-	-	-	-	2,309	-	-	-	-	-	-	-	-	**2,309**
2036	-	-	-	-	2,309	-	-	-	-	-	-	-	-	**2,309**
2037	-	-	-	-	2,309	-	-	-	-	-	-	-	-	**2,309**
2038	-	-	-	-	2,309	-	-	-	-	-	-	-	-	**2,309**
2039	-	-	-	-	1,314	-	-	-	-	-	-	-	-	**1,314**
TOTAL	**15,728**	**1,148**	**4,421**	**2,264**	**14,786**	**161,141**	**12,773**	**34,229**	**226**	**12,241**	**75**	**2,292**	**958**	

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 unit mining operation. It would be recommended that Aura ensures a rational extra capacity to the loading fleet, and all the fleet related to mine infrastructure, to provide a good transport condition.

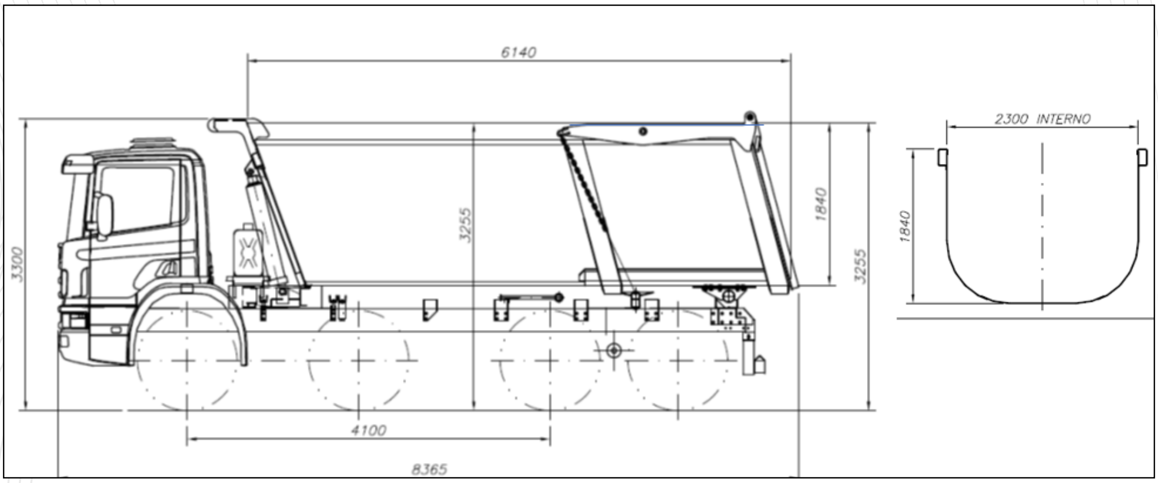
An electronic dispatch system can benefit ROM loading to minimize queues when loading.

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 16-18.

16.3.5 TRANSPORT OF ROM TRUCKS

The conventional 8x4 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 8x4 truck with blasted rock dump box type, manufactured by Rossetti and with a struck capacity of 22 m^3. Unlike Caterpillar off-road trucks, the volume of the bucket is struck and not heaped. This fact means, the dump box heaped load of the truck should be between 24 to 25 m^3.

Figure 16-6 Schematic Drawing of the 8x4 Truck with typical Bucket for Rossetti Mining Operations



If we consider the primary blasted rock density as 1.7 to 1.8 t/m^3 the truck load is going to be around 38 tons; five cycles will be needed for the 4.5 m^3 bucket size excavator to fill it.

In terms of specification, the trucks would be of a PBT (Total Gross Weight) of 48,000 kg. Discounting the tare of 18,000 kg (truck = 11 t + bucket = 5 t) there would be 32 t left for the net load. This overload is a situation that we have already observed in other similar operations.

Safety issues should be the biggest concern and 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 8x4 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 of the LOM period. Table 16-12 shows, for all Paiol's pit sequenced period: average transport distances, calculated travel time and truck productivity with uptime for full time. Table 16-13 and Table 16-14 show the average transport distances, calculated travel time and truck productivity related to all sequenced periods for the pits Cata Funda and Vira Saia.

Table 16-12 Paiol Pit: Average Transport Distance in Meters, Calculated Travel Time in Minutes, and Truck Productivity with Uptime for Full Time (T/H), Year by Year B, Origin and Destination of Each Route.

ROUTE DESTINATION FROM PAIOL'S PIT ORIGIN									
		TRUCK FIXED CYCLE TIME (min.) = 6.0				TONNAGE/TRIP (t) = 35			
DESTINATION→	HIGH-GRADE PATIO			LOW-GRADE PILE			WASTE DUMP PILE		
YEAR ↓	ONE WAY DISTANCE (m)	TRAVEL TIME (MINUTES)	PRODUCTIVITY BY TRUCK (t/h)	ONE WAY DISTANCE (m)	TRAVEL TIME (MINUTES)	PRODUCTIVITY BY TRUCK (t/h)	ONE WAY DISTANCE (m)	TRAVEL TIME (MINUTES)	PRODUCTIVITY BY TRUCK (t/h)
2022	1,850	12.5	114	1,600	10.3	129	2,050	12.1	116
2023	1,900	13.0	111	1,650	10.8	125	2,050	12.1	116
2024	1,950	13.5	108	1,700	11.3	122	2,050	12.1	116
2025	2,000	13.9	106	1,750	11.7	119	2,050	12.1	116
2026	2,250	16.3	94	2,000	14.1	105	2,550	18.8	85
2027	2,450	18.1	87	2,200	15.9	96	2,550	18.8	85
2028	2,750	20.9	78	2,500	18.7	85	3,550	29.7	59
2029	3,250	25.5	67	3,000	23.3	72	4,050	34.1	52
2030	3,750	30.0	58	3,500	27.8	62	4,050	34.1	52
2031	3,850	30.9	57	3,600	28.7	61	4,050	34.1	52
2032	-	-	-	-	-	-	-	-	-
2033	-	-	-	-	-	-	-	-	-
2034	-	-	-	-	-	-	-	-	-
2035	-	-	-	-	-	-	-	-	-
2036	-	-	-	-	-	-	-	-	-
2037	-	-	-	-	-	-	-	-	-
2038	-	-	-	-	-	-	-	-	-
2039	-	-	-	-	-	-	-	-	-

Table 16-13 Cata Funda Pit: Average Transport Distance Meters, Calculated Travel Time in Minutes, and Truck Productivity with Uptime for Full Time (T/H), Year by Year B, Origin and Destination of Each Route

ROUTE DESTINATION FROM CATA FUNDA'S PIT ORIGIN									
		Truck fixed cycle time(min.) = 6.0		Tonnage/trip (t) = 35					
DESTINATION →	HIGH-GRADE PATIO			LOW-GRADE PILE			WASTE DUMP PILE		
YEAR ↓	ONE WAY DISTANCE (M)	TRAVEL TIME (MINUTES)	PRODUCTIVITY BY TRUCK (T/H)	ONE WAY DISTANCE (M)	TRAVEL TIME (MINUTES)	PRODUCTIVITY BY TRUCK (T/H)	ONE WAY DISTANCE (m)	TRAVEL TIME (MINUTES)	PRODUCTIVITY BY TRUCK (T/H)
2022	-		-	-		-	-		-
2023	-		-	-		-	-		-
2024	-		-	-		-	-		-
2025	16,750	66.3	29	16,500	65.8	29	1,750	16.2	95
2026	17,250	67.3	29	17,000	66.8	29	1,750	16.2	95
2027	17,250	67.3	29	17,000	66.8	29	2,250	17.2	91
2028	18,250	75.8	26	18,000	75.3	26	2,250	17.2	91
2029	-	-	-	-	-	-	-	-	-
2030	-	-	-	-	-	-	-	-	-
2031	-	-	-	-	-	-	-	-	-
2032	-	-	-	-	-	-	-	-	-
2033	-	-	-	-	-	-	-	-	-
2034	-	-	-	-	-	-	-	-	-
2035	-	-	-	-	-	-	-	-	-
2036	-	-	-	-	-	-	-	-	-
2037	-	-	-	-	-	-	-	-	-
2038	-	-	-	-	-	-	-	-	-
2039	-	-	-	-	-	-	-	-	-

Table 16-14 Vira Saia's Pit: Average Transport Distance in Meters, Calculated Travel Time in Minutes, and Truck Productivity with Uptime for Full Time (T/H), Year by Year B, Origin and Destination of Each Route

Truck fixed cycle time(min.) = 6.0						Tonnage/trip (t) = 35			
Destination →	**High-Grade Patio**			**Low-Grade Pile**			**Waste Dump Pile**		
YEAR ↓	**One Way Distance (m)**	**Travel Time (minutes)**	**productivity by truck (t/h)**	**One Way Distance (m)**	**Travel Time (minutes)**	**productivity by truck (t/h)**	**One Way Distance (m)**	**Travel Time (minutes)**	**productivity by truck (t/h)**
2022	-	-	-	-	-	-	-	-	-
2023	-	-	-	-	-	-	-	-	-
2024	-	-	-	-	-	-	-	-	-
2025	-	-	-	-	-	-	-	-	-
2026	-	-	-	-	-	-	-	-	-
2027	-	-	-	-	-	-	-	-	-
2028	-	-	-	-	-	-	-	-	-
2029	5,750	22.6	73.4	5,500	22.1	74.7	1,700	15.4	98.1
2030	6,250	25.2	67.3	6,000	24.7	68.4	1,750	15.5	97.7
2031	6,250	25.2	67.3	6,000	24.7	68.4	1,750	15.5	97.7
2032	6,750	30.2	58.0	6,500	29.7	58.8	2,750	17.5	89.4
2033	7,250	31.1	56.6	7,000	30.6	57.4	3,250	18.5	85.7
2034	7,750	32.2	55.0	7,500	31.7	55.7	3,750	19.5	82.4
2035	-	-	-	-	-	-	-	-	-
2036	-	-	-	-	-	-	-	-	-
2037	-	-	-	-	-	-	-	-	-
2038	-	-	-	-	-	-	-	-	-
2039	-	-	-	-	-	-	-	-	-

Table 16-15 presents the planned stockpiles routes taken, including average one way distance transport and related calculated travel time.

Table 16-15 Stockpiles Taken to the Plant and to the Paiol Waste Dump Pile: Average Transport Distance in Meters, Calculated Travel Time in Minutes, and Truck Productivity with Uptime for Full Time (t/h), Yearly by Origin and Destination of Each Route

FROM STOCKPILES TO PLANT AND WASTE DUMP PILE ROUTES									
Truck fixed cycle time(min.) = 6.0					Tonnage/trip (t) = 35				
Origin →	Heap Leach Pad						Low-Grade Pile		
Destination →	Crushing Patio			Waste Dump Pile			Crushing Patio		
YEAR ↓	One Way Distance (m)	Travel Time (minutes)	productivity by truck (t/h)	One Way Distance (m)	Travel Time (minutes)	productivity by truck (t/h)	One Way Distance (m)	Travel Time (minutes)	productivity by truck (t/h)
2022	-	-	-	-		-	750	3	233
2023	-	-	-	-		-	-		-
2024	-	-	-	-		-	-		-
2025	900	3.6	219	500	1.9	266	-		-
2026	900	3.6	219	500	1.9	266	-		-
2027	900	3.6	219	500	1.9	266	-		-
2028	900	3.6	219	500	1.9	266	-		-
2029	900	3.6	219	500	1.9	266	-		-
2030	900	3.6	219	-		-	-		-
2031	900	3.6	219	-		-	1,000	4.0	210
2032	900	3.6	219	-		-	950	3.8	214
2033	900	3.6	219	-		-	950	3.8	214
2034	-	-	-			-	900	3.6	219
2035	-	-	-			-	850	3.4	223
2036	-	-	-			-	800	3.2	228
2037	-	-	-			-	800	3.2	228
2038	-	-	-			-	750	3.0	233
2039	-	-	-			-	750	3.0	233

Table 16-16 presents the necessary truck working hours, by year, to make all the transport related to the total rock moved plan in the LOM.

Table 16-16 Truck Working Hours Considering all Origins and Destinations to Transport all Moved Rock Planned in all LOM Periods.

TRUCK WORKING HOURS NEEDED (h)														
Destin →	PLANT					Waste Dump Piles				Low-Grade Stockpile			Stock	Total
Origin → Year ↓	*Paiol's pit*	*Cata Funda's pit*	*Vira Saia's pit*	*Heap Leach Pad*	*Low-Grade Pile*	*Paiol*	*Cata Funda*	*Vira Saia*	*Heap Leach Pad*	*Paiol Low Grade*	*Cata Funda Low Grade*	*Vira Saia Low Grade*	*Paiol High Grade*	
2022	2,902	-	-	-	429	43,609	-	-	-	3,050	-	-	2,926	52,915
2023	11,946	-	-	-	-	85,439	-	-	-	7,581	-	-	-	104,965
2024	12,260	-	-	-	-	79,524	-	-	-	4,990	-	-	-	96,774
2025	10,539	6,910	-	46	-	91,588	25,774	-	96	8,671	85	-	1,966	145,674
2026	10,497	5,446	-	711	-	122,440	19,077	-	96	3,983	607	-	-	162,857
2027	11,378	5,088	-	744	-	139,999	18,226	-	96	8,133	257	-	-	183,921
2028	12,673	5,609	-	762	-	233,902	14,362	-	96	11,743	577	-	-	279,724
2029	9,923	-	6,740	656	-	256,298	-	36,142	96	12,947	-	2,210	-	325,012
2030	11,306	-	6,429	950	-	108,567	-	34,197	-	8,607	-	3,323	-	173,379
2031	8,498	-	6,490	364	288	15,668	-	47,794	-	4,035	-	3,263	-	86,399
2032	-	-	6,600	1,140	2,183	-	-	34,184	-	-	-	4,711	-	48,818
2033	-	-	7,093	457	3,725	-	-	36,499	-	-	-	4,504	-	52,278
2034	-	-	6,197	-	4,385	-	-	18,642	-	-	-	2,503	-	31,727
2035	-	-	-	-	5,819	-	-	-	-	-	-	-	-	5,819
2036	-	-	-	-	5,695	-	-	-	-	-	-	-	-	5,695
2037	-	-	-	-	5,695	-	-	-	-	-	-	-	-	5,695
2038	-	-	-	-	5,571	-	-	-	-	-	-	-	-	5,571
2039	-	-	-	-	3,171	-	-	-	-	-	-	-	-	3,171
TOTAL	**101,922**	**23,053**	**39,549**	**5,829**	**36,962**	**1,177,035**	**77,439**	**207,457**	**480**	**73,741**	**1,525**	**20,513**	**4,892**	

The estimated fleet for ROM transport starts with a fleet of 8 trucks and reaches a peak in 2028 and 2029. As it is equipment made in Brazil, and quite common in the local market, the most appropriate investments can be made to vary production levels. It is 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. This period is about 90 days.

It is interesting to note that 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 16-18.

16.3.6 MINE DEVELOPMENT AND PREPARATION FLEET (AUXILIARY)

16.3.6.1 MOTOR 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 '(CAT 140). At the beginning of the operation we are selecting 2 units, but it is estimated the need will increase by at least one more unit, when the mine operation would be done in two pits simultaneously. In addition, we would have more than 15 km of production roads to operate in the Cata Funda pit and more than 5 km to operate the Vira Saia pit.

16.3.6.2 CRAWLER TRACTOR

At current production levels, 3 tractors weighing 35 tons (CAT D8) would be required to ensure 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-ton crawler tractor (CAT D6) is lightweight and has great limitations for carrying out work in the mining area. For the spreading activities in the waste dump areas its productivity is also low, but if there are large areas of deposition this can be a palliative. 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, we rarely find a wheel tractor (similar to the CAT 824 model) in mining operations in Brazil. We are considering one CAT D6 per pit on operation and keeping one unit at the end of the operation.

16.3.6.3 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 unit be acquired to meet the mining operation. It can also be used to transport large components of the concentration plant during large scale maintenance.

Its width should consider the ability to transport safely the widest crawler equipment in the mining operation.

16.3.6.4 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. So, we are recommending 2 units when we have only one pit, but certainly is going to be need more units 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 also have a nozzle system to help put out fires during the dry season.

16.3.7 EQUIPMENT WORKING HOURS CALCULATION

Table 16-17 shows the key parameters assumed for equipment fleet sizing, including the main auxiliary equipment selected,

Table 16-17 Equipment Calculated Working Hours

EQUIPMENT DESCRIPTION	REFERENCE MODEL	SHIFTS PER DAY	AVAILABILITY	UTILIZATION	WORKING HOUR BY YEAR
Hydraulic Excavator -backhoe	Caterpillar CAT 374	3	85%	70%	5,141
Front End Loader	Caterpillar CAT 966	3	80%	70%	4,838
Truck 8x4 with 22 m3 Dump Box	Mercedes Bens -Actros 4844	3	90%	70%	5,443
Bulldozer	Caterpillar CAT D8	3	85%	50%	3,672
Grader	Caterpillar CAT 140	2	85%	60%	2,938
Bulldozer	Caterpillar CAT D8	3	85%	60%	4,406
Truck for Explosives	Mercedes Bens Axor 6x4	1	85%	45%	1,102
Water tank Trucks	Mercedes Bens Axor 6x4	3	85%	60%	4,406
Backhoe loader	Caterpillar CAT 316	3	85%	45%	3,305
Blasthole Production Drill rig	Sandvik DPi1 500	3	85%	60%	4,406
Grade Control Drill rig	Epiroc Flexi-Roc D65 RC	2	85%	50%	2,448
Hydraulic Breaker+Excavator	CAT 330	3	85%	50%	3,672
Lube/Fuel truck	Mercedes Bens Axor 6x4	3	85%	45%	3,305
Field Maintenance truck	Mercedes Bens Axor 6x4	3	85%	15%	1,102
Portable Lightning Tower	Terex RL 4000	2	85%	30%	1,469
Light Vehicle	Toyota Hilux	3	85%	40%	2,938
Low bed transport truck	to be specified	3	85%	15%	1,102

16.3.8 SUMMARY OF MINING EQUIPMENT FLEET

For optimized operation 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 face 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 16-18, below:

Table 16-18 Summary of the Fleet Selected for the Mining Operation for 1.3 Mtpy

	LOADING ROM		TRUCK	DRILLING		MINE AUXILIAR AND INFRASTRUCTURE OPERATIONS												
	Hydraulic Excavator	Front End Loader	Vocational Truck 8x4	Production Blasthole	Grade Control	Bulldozer	Grader	Bulldozer	Backhoe Loader	Excavator + Hydr. Breaker	Explosive Truck 6x4	Lube/Fuel Truck	Field Maint Truck	Water Tank Truck	Portable Light.Tower	Low bed Truck	Pick up	TOTAL
REFER. MODEL	CAT 374	CAT 966L	MB ACTROS 4844	SANDVIK DP1500	EPIROC D65 RC	CAT D8	CAT 140	CAT D6	CAT 416	CAT 330	MB AXOR 6x4	MB AXOR 6x4	MB AXOR 6x4	MB AXOR 6x4	TEREX RL4K	MB + Rand.	Hilux	
2022	3	2	10	2	1	3	2	1	2	1	1	2	1	2	8	1	10	**52**
2023	5	2	20	4	1	4	2	1	2	1	1	2	1	2	12	1	10	**71**
2024	4	2	18	4	1	4	3	1	2	1	1	2	1	2	10	1	10	**67**
2025	6	2	27	6	1	4	3	1	2	1	1	3	1	2	14	1	15	**90**
2026	5	2	31	6	1	4	3	2	2	1	1	3	1	4	12	1	15	**94**
2027	6	2	35	7	1	4	3	2	2	1	1	3	1	4	14	1	15	**102**
2028	6	2	52	7	1	4	3	2	2	1	1	3	1	4	14	1	15	**119**
2029	7	2	61	8	1	4	3	2	2	1	1	4	1	4	16	1	20	**138**
2030	4	2	33	5	1	4	3	2	2	1	1	3	1	4	10	1	20	**97**
2031	3	2	16	3	1	3	3	2	2	1	1	3	1	4	8	1	20	**74**
2032	2	2	9	2	1	3	2	2	2	1	1	2	1	4	6	1	15	**56**
2033	2	2	10	2	1	3	2	2	2	1	1	2	1	4	6	1	15	**57**
2034	1	2	6	1	1	2	2	2	2	1	1	2	1	4	4	1	10	**43**
2035	1	2	1	1	0	1	1	0	1	1	0	1	1	1	4	1	5	**22**
2036	1	2	1	1	0	1	1	0	1	1	0	1	1	1	4	1	5	**22**
2037	1	2	1	1	0	1	1	0	1	1	0	1	1	1	4	1	5	**22**
2038	1	2	1	1	0	1	1	0	1	1	0	1	1	1	4	1	5	**22**
2039	1	2	1	1	0	1	1	0	1	1	0	1	1	1	4	1	5	**22**

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, in the Ernesto's Mine – MT. The contract was signed on October 2020 and they involve 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 montly payment to the contractor
- Explosive and accessories supplying. The explosives and accessories is going to be supplied by Aura Minerals and

deducted form monthly payment to the contractor.

- Grade Control Drilling, contracted by meter. The diesel is going to be supplied by Aura Minerals and deducted from monthle payment to the contractor.

The diesel price by liter considered was the price in Almas region, Tocantins state on October 2020

Table 16-19 summarizes the variations in transport costs by distance, for all the pits: Paiol, Cata Funda and Vira Saia.

Table 16-19 Costs in R$/t for Distances Greater than 2,000 m.

COSTS VARIATION BY DISTANCE	
DISTANCE (m)	R$/t
up to 2,000	reference value
2,000 a 2,500	0.125
2,500 a 3,000	0.250
3,000 a 3,500	0.375
3,500 a 4,000	0.500

Tables 16-20 to Table 16-22 present, by pit, the costs in Brazilian reais per ton to the mining operations described above.

Table 16-20 Paiol's Operational Costs in R$/Ton for Blasthole Drilling, Blasting, ROM Loading, ROM Transport, Dozer Spreading Material, Prepare and Maintenance of All Mine Infrastructure

PAIOL	OPERATIONAL COSTS (R$/t)					
	ORE			WASTE		
OPERATION MINING UNIT	SOIL	SAPROLITE	FRESH ROCK	SOIL	SAPROLITE	FRESH ROCK
Blasthole Drilling	0.00	0.37	1.12	0.00	0.26	0.81
Blasting	0.00	0.42	1.26	0.00	0.23	0.72
Loading / Transport / Dozer Spreading material	3.90	4.49	5.08	3.29	3.73	3.95
SUB- TOTAL	3.90	5.29	7.47	3.29	4.21	5.48
Transport to the Plant	0.00	0.00	0.00	0.00	0.00	0.00
TOTAL	**3.90**	**5.29**	**7.47**	**3.29**	**4.21**	**5.48**

Table 16-21 Cata Funda's Operational Costs in R$/Ton for Blasthole Drilling, Blasting, ROM Loading, ROM Transport, Dozer Spreading Material, Preparation and Maintenance of all Mine Infrastructure.

CATA FUNDA	OPERATIONAL COSTS (R$/t)					
	ORE			WASTE		
OPERATION MINING UNIT	SOIL	SAPROLITE	FRESH ROCK	SOIL	SAPROLITE	FRESH ROCK
Blasthole Drilling	0.00	0.37	1.12	0.00	0.26	0.81
Blasting	0.00	0.42	1.26	0.00	0.23	0.72
Loading / Transport / Dozer Spreading material	3.90	4.49	5.08	3.29	3.73	3.95
SUB- TOTAL	3.90	5.29	7.47	3.29	4.21	5.48
Transport to the Plant	3.99	3.99	3.99	0.00	0.00	0.00
TOTAL	**7.89**	**9.28**	**11.46**	**3.29**	**4.21**	**5.48**

Table 16-22 Vira Saia's Operational Costs in R$/Ton for Blasthole Drilling, Blasting, ROM Loading, ROM Transport, Dozer Spreading Material, Preparation and Maintenance of all Mine Infrastructure.

VIRA SAIA	OPERATIONAL COSTS (R$/t)					
	ORE			WASTE		
OPERATION MINING UNIT	SOIL	SAPROLITE	FRESH ROCK	SOIL	SAPROLITE	FRESH ROCK
Blasthole Drilling	0.00	0.37	1.12	0.00	0.26	0.81
Blasting	0.00	0.42	1.26	0.00	0.23	0.72
Loading / Transport / Dozer Spreading material	3.90	4.49	5.08	3.29	3.73	3.95
SUB- TOTAL	3.90	5.29	7.47	3.29	4.21	5.48
Transport to the Plant	1.40	1.40	1.40	0.00	0.00	0.00
TOTAL	**5.30**	**6.69**	**8.87**	**3.29**	**4.21**	**5.48**

Table 16-23 shows the costs to take up material from stockpiles and the Heap Leach pad.

Table 16-23 Costs in R$/t for Stockpile Take Up.

STOCKPILE RESUMPTION	
Complementary stockpile	R$ / t
Heap Leach	3.19
Low-Grade Ore -Paiol	3.59
Low Grade Ore Vira Saia	3.59

Table 16-24 shows the grade control costs considered for grade control engineering.

Table 16-24 Grade Control Costs to be Applied in Ore and Waste.

GRADE ENGINEERING	R$ / t
Geology/Mining Plan/Grade Control	**0.97**

Table 16-25 presents an Almas' Project Mining Costs Summary by quarter up to year 2025 and annually after 2026 up to 2039.

Table 16-25 Almas Gold Project Summary Mining Costs

		ALMAS' PROJECT MINING COSTS - Million BRL (R$*1,000,000.00)														
		PAIOL			CATA FUNDA			VIRA SAIA			HEAP LEACH	LOW GRADE	MINE TOTAL COST		
YEAR	QUARTER	ORE	WASTE	TOTAL	ORE	WASTE	TOTAL	ORE	WASTE	TOTAL			ORE	WASTE	TOTAL
2022	Pre Oper.	5.0	9.2	14.2	-	-	-	-	-	-	-	-	5.0	9.2	14.2
	Q3	-	6.2	6.2	-	-	-	-	-	-	-	-	-	6.2	6.2
	Q4	2.6	5.6	8.3	-	-	-	-	-	-	-	-	2.6	5.6	8.3
	ALL	7.6	21.1	28.7	-	-	-	-	-	-	-	-	7.6	21.1	28.7
2023	Q1	4.6	11.2	15.8	-	-	-	-	-	-	-	-	4.6	11.2	15.8
	Q2	4.3	11.6	15.9	-	-	-	-	-	-	-	-	4.3	11.6	15.9
	Q3	3.5	11.6	15.1	-	-	-	-	-	-	-	-	3.5	11.6	15.1
	Q4	4.3	12.0	16.3	-	-	-	-	-	-	-	-	4.3	12.0	16.3
	ALL	16.6	46.4	63.0	-	-	-	-	-	-	-	-	16.6	46.4	63.0
2023	Q1	3.5	10.5	14.0	-	-	-	-	-	-	-	-	3.5	10.5	14.0
	Q2	3.0	10.7	13.7	-	-	-	-	-	-	-	-	3.0	10.7	13.7
	Q3	4.1	11.2	15.3	-	-	-	-	-	-	-	-	4.1	11.2	15.3
	Q4	3.6	11.0	14.6	-	-	-	-	-	-	-	-	3.6	11.0	14.6
	ALL	14.2	43.3	57.5	-	-	-	-	-	-	-	-	14.2	43.3	57.5
2023	Q1	4.6	13.7	18.3	-	-	-	-	-	-	-	-	4.6	13.7	18.3
	Q2	5.2	14.8	19.9	-	-	-	-	-	-	-	-	5.2	14.8	19.9
	Q3	3.9	12.3	16.2	0.5	4.4	4.9	-	-	-	-	-	4.4	16.7	21.1
	Q4	2.4	11.1	13.5	1.6	6.9	8.5	-	-	-	-	-	3.9	18.0	21.9
	ALL	16.0	51.8	67.9	2.1	11.3	13.4	-	-	-	-	-	18.1	63.1	81.2
2026	ALL	10.7	58.2	68.8	2.0	9.9	11.9	-	-	-	0.4	-	13.1	68.1	81.2
2027	ALL	13.5	66.5	80.0	1.8	9.3	11.0	-	-	-	0.5	-	15.8	75.8	91.5
2028	ALL	15.4	78.9	94.3	1.9	7.5	9.3	-	-	-	0.5	-	17.7	86.4	104.1

		ALMAS' PROJECT MINING COSTS - Million BRL (R$*1,000,000.00)													
		PAIOL			CATA FUNDA			VIRA SAIA			HEAP LEACH	LOW GRADE	MINE TOTAL COST		
YEAR	QUARTER	ORE	WASTE	TOTAL	ORE	WASTE	TOTAL	ORE	WASTE	TOTAL			ORE	WASTE	TOTAL
2029	ALL	12.5	78.6	91.1	-	-	-	5.9	16.7	22.6	0.5	-	18.8	95.4	114.2
2030	ALL	9.5	34.0	43.6	-	-	-	5.9	17.3	23.2	0.7	-	16.0	51.4	67.4
2031	ALL	5.8	4.9	10.7	-	-	-	5.9	25.6	31.5	0.3	0.2	12.2	30.5	42.7
2032	ALL	-	-	-	-	-	-	6.0	17.1	23.2	0.8	1.7	8.5	17.1	25.6
2033	ALL	-	-	-	-	-	-	6.1	17.9	24.0	0.5	2.7	9.3	17.9	27.2
2034	ALL	-	-	-	-	-	-	4.5	9.0	13.5	-	3.4	7.9	9.0	16.9
2035	ALL	-	-	-	-	-	-	-	-	-	-	4.7	4.7	-	4.7
2036	ALL	-	-	-	-	-	-	-	-	-	-	4.7	4.7	-	4.7
2037	ALL	-	-	-	-	-	-	-	-	-	-	4.7	4.7	-	4.7
2038	ALL	-	-	-	-	-	-	-	-	-	-	4.7	4.7	-	4.7
2039	ALL	-	-	-	-	-	-	-	-	-	-	2.3	2.3	-	2.3
TOTAL		**121.9**	**483.7**	**605.6**	**7.8**	**37.9**	**45.6**	**34.3**	**103.8**	**138.0**	**4.1**	**29.0**	**197.0**	**625.4**	**822.4**

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 16-26.

Table 16-26 Workforce People for the Outsource 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

17 RECOVERY METHODS

17.1 OVERALL PROCESS DESIGN

The process design is based on the results of several testwork programs. This includes testwork completed for the feasibility study and historical testing. Historical testing evaluated different flowsheet options. The flowsheet selected for the feasibility study is based on typical industry unit operations for gold processing plants.

The flowsheet includes primary crushing followed by grinding to achieve a particle size distribution of 80% passing 75 µm. Part of the cyclone underflow will be processed in a gravity circuit and the cyclone product (overflow) will feed a pre-leach thickener, with the underflow processed through a leach/carbon in leach (CIL) circuit. CIL tailings will be treated for cyanide destruction. The carbon from CIL will go to elution, regeneration and the final solution will go to electrowinning and the gold room.

Key process design criteria are listed below:

- Nominal throughput of 3,560 t/d or 1.3 Mt/a
- crushing plant availability of 70%
- plant availability of 92% for grinding, gravity concentration, leach plant and gold recovery operations

17.2 MILL PROCESS PLANT DESCRIPTION

The process design is comprised of the following circuits:

- primary crushing of run-of-mine (ROM) material
- surge bin to provide buffer capacity ahead of the grinding circuit
- emergency stockpile fed from the overflow of the surge bin
- low-aspect SAG mill with trommel screen and cyclone classification
- gravity recovery of the cyclone underflow slurry by one semi-batch centrifugal gravity concentrator, followed by intensive cyanidation of the gravity concentrate and electrowinning of the pregnant leach solution in a dedicated cell located in the gold room
- trash screening
- pre-leach thickening
- leach + adsorption (L/CIL hybrid)
- acid washing of loaded carbon and Anglo-American Research Laboratory (AARL) type elution followed by electrowinning and smelting to produce doré
- carbon regeneration
- cyanide destruction of tailings using the SO_2/air process
- carbon safety screening
- tailings management facility

17.2.1 PLANT DESIGN CRITERIA

Key process design criteria are listed in Table 17.1.

17.2.2 PRIMARY CRUSHING & STOCKPILING

The crushing circuit is designed for an annual operating time of 6,130 h/or 70% availability at the capacity of 3,560 t/d.

Material is hauled from the mine or stockpiles and fed by front-end loader into the mobile crushing system. This system is composed of the run-of-mine run-of-mine (ROM) hopper, a vibrating grizzly feeder, a primary crusher and a discharge conveyor, along with some auxiliary equipment. As part of the mobile system, the ROM is dumped into the ROM hopper, equipped with a static grizzly. Provision for dumping on the ROM pad for blending and re-handling into the ROM hopper is provided. Material from the ROM hopper is crushed by a primary jaw crusher. ROM hopper material is reclaimed by a vibrating grizzly at 212 t/h to feed the jaw crusher.

A mobile rock breaker is utilized to break oversize rocks at the feed to the jaw crusher. The crushed material is conveyed to a surge bin that provides approximately 3 hours of live storage at the nominal processing rate. The bin has an overflow system, which forms an emergency stockpile next to the bin. Given the milling operation is designed for an annual operating time of 8,059 h/or 92% availability, this will result in excess crushed material production when the crusher is operational. The excess crushed material will allow routine crusher maintenance to be carried out without interrupting feed to the mill.

The mill feed surge bin is equipped with two vibrating feeders to regulate feed at 161 t/h into the SAG mill. When the surge bin has material, crushed material is drawn from the surge bin by the vibrating feeders and feeds the SAG mill circuit via the SAG mill feed conveyor. When operating using the bin overflow stockpile, front-end loaders (FELs) reclaim the material to a reclaim bin equipped with a vibrating feeder that also feeds the SAG mill circuit. Pebbles from the SAG mill are fed to a recycle circuit via conveyor and discharged on the SAG mill feed conveyor to recycle to the SAG mill. The transfer point of the pebble recycle conveyor has a chute that allows purging of the pebbles as required.

The material handling and crushing circuit includes the following key equipment:

- ROM hopper
- vibrating grizzly
- primary jaw crusher
- surge bin
- mill feed vibrating feeders (equipped with VSDs)
- material handling equipment

Table 17-1 Summary of Key Process Design Criteria

DESIGN PARAMETER	UNITS	VALUE
Plant Throughput	t/d	3,560
Head Grade – Design	g/t Au	1.58
Crushing Plant Availability	%	70
Mill Availability	%	92
Bond Crusher Work Index (CWi)	kWh/t	17.1
Bond Ball Mill Work Index (BWi)	kWh/t	10.1
JK Axb	-	47
Bond Abrasion Index (Ai)	g	0.069
Primary Crusher		Metso C116 or Equivalent
Material Specific Gravity	t/m³	2.79
Angle of Repose	degrees	37
Moisture Content	%	5.0
SAG Mill Dimensions		5.0 m dia. X 9.0 m EGL
SAG Mill Installed Power	MW	3.75
SAG Mill Discharge Density	% w/w	70
SAG Mill Ball Charge	% v/v	21
Primary Grind size (P80)	µm	75
Gravity Circuit Feed Source		Cyclone underflow slurry
Gravity Circuit Feed Rate	% of cyclone underflow	25
Gravity Circuit Recovery	Au (%)	17.5
Pre-leach thickener settling rate	t/d/m²	34.4
Pre-leach thickener diameter	m	12
L-CIL Residence Time	h	24
L-CIL Extraction	Au (%)	92.5
L-CIL Operating Density	% w/w	50
L-CIL Dissolved Oxygen Target	mg/L	5-8
L-CIL pH Target		10.5 – 11.0

DESIGN PARAMETER	UNITS	VALUE
CIL Carbon Concentration	g/L	15
L-CIL Sodium Cyanide Addition	kg/t	0.8
L-CIL Hydrated Lime Addition	kg $Ca(OH)_2$/t	0.26
Leach & CIL Tanks	#	1 + 6
Elution Circuit Capacity	t	3.0
Detox Residence Time	minutes	120
Detox Oxygen Addition Rate (weight)	$O_2:SO_2$	3.0
Detox Feed Cyanide Concentration	mg/L CN_{WAD}	150
Detox Cyanide Discharge Target	mg/L CN_{WAD}	<2.0
Detox Copper Sulphate Addition	mg/L Cu^{+2}	50
Detox SO_2 Addition (weight)	$SO_2:CN_{WAD}$	5.5
Detox Lime Addition (weight)	$CaO:SO_2$	1.0

17.2.3 GRINDING CIRCUIT

The grinding circuit consists of a low-aspect single stage SAG mill in closed circuit with hydrocyclones. The SAG mill is an adaptation of an existing ball mill, already purchased by Aura Minerals. The mill size and design were reviewed, and it is suitable for this new application. The circuit is sized based on a SAG F_{80} of 85 mm and product P_{80} of 75 µm. The SAG mill slurry discharges through a trommel screen where the pebbles are screened and recycled back to the SAG mill via a conveyor, with the ability to purge the pebbles at the conveyor transfer point. Trommel undersize discharges into the cyclone feed pumpbox.

Water is added to the cyclone feed pumpbox to obtain the appropriate density prior to pumping to the cyclones. Cyclone underflow is split and part feeds the gravity circuit scalping screen and the rest recycles back to the SAG mill. Cyclone overflow gravitates to the pre-leach thickener via a trash screen.

The grinding circuit includes the following key equipment:

- 3,750 kW single stage SAG mill
- cyclone feed pumpbox
- cyclone feed pumps
- classification cyclones
- trash screen

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 at +2 mm reports to the gravity tails pumpbox, 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 pumpbox.

The gravity recovery circuit includes the following key equipment:

- gravity feed scalping screen
- gravity concentrator
- gravity tails pumpbox
- gravity tails pump

17.2.5 INTENSIVE LEACH REACTOR

Concentrate from the gravity circuit reports 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 (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 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 cyanidation reactor
- ILR pregnant solution tank
- ILR electrowinning cell

17.2.6 PRE-LEACH THICKENING

Trash screen undersize feeds the pre-leach thickener, which increases the solids concentration to 50% (w/w) prior to the leach-CIL circuit. Flocculant is added 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 pumpbox.

The pre-leach thickening circuit includes the following key equipment:

- pre-leach thickener
- pre-leach thickener underflow pump

17.2.7 LEACH & ADSORPTION CIRCUIT

The leach-adsorption circuit consists of one leach tank and six carbon-in-leach (CIL) tanks. The circuit is fed by the pre-leach thickener. The leach and CIL tanks are identical in size, with a total circuit residence time of 24 hours at 50% w/w density.

Air is sparged to each tank to maintain adequate dissolved oxygen levels for leaching at 5-8 mg/L. Hydrated lime is added to adjust the operating pH to the desired set point of 10.5 - 11. Cyanide solution is added to the first leach tank. Fresh/regenerated carbon from the carbon regeneration circuit is returned to the last tank of the CIL circuit and is advanced counter-currently to the slurry flow by pumping slurry and carbon. Slurry from the last CIL tank gravitates to the cyanide detoxification tanks.

The intertank screen in each CIL tank retains the carbon whilst allowing the slurry to flow by gravity to the downstream tank. This counter-current process is repeated until the loaded carbon reaches the first CIL tank. Recessed impeller pumps are used to transfer slurry between the CIL tanks and from the lead tank to the loaded carbon screen mounted above the acid wash column in the elution circuit.

The leach and carbon adsorption circuit include the following key equipment:

- leach/CIL tanks and agitators
- loaded carbon screen
- intertank carbon screens
- carbon sizing screen

17.2.8 CYANIDE DESTRUCTION

CIL tails at approximately 50% w/w solids flow by gravity to the two cyanide destruction tanks. The water used for acid rinse and carbon transfer is also included in the feed to the detoxification circuit. As a result, the percentage of solids in the feed to the detoxification circuit is estimated to be closer to 47% w/w solids.

Each tank operates with a total residence time of approximately 60 mins to reduce weak acid dissociable cyanide (CNWAD) concentration from 150 mg/L to less than 2.0 mg/L to comply with environmental requirements prior to deposition in the TSF.

Cyanide destruction is undertaken using the SO_2/air method. The reagents required are air, lime, copper sulphate, and sodium metabisulphite (SMBS). The cyanide destruction tank is equipped with compressed air spargers and an agitator to ensure that the oxygen and reagents are thoroughly mixed with the tailing's slurry.

From the detoxification tank, the tailings report to the carbon safety screen. Screen undersize feeds the tailings pumpbox, whilst screen oversize (recovered carbon) is collected in a fine carbon bin for potential return to the CIL circuit.

The main equipment in this area includes:

- cyanide destruction tanks and agitators
- carbon safety screen
- tailings pumpbox
- tailings pump

17.2.9 CARBON ACID WASH, ELUTION & REGENERATION CIRCUIT

17.2.9.1 CARBON ACID WASH

Prior to gold stripping stage, loaded carbon is treated with a weak hydrochloric acid solution to remove calcium, magnesium, and other salt deposits that could render the elution less efficient or become baked on in subsequent steps and ultimately foul the carbon.

Loaded carbon from the loaded carbon recovery screen flows by gravity to the acid wash column. Entrained water is drained from the column and the column is refilled from the bottom up with the hydrochloric acid solution. Once the column is filled with the acid, it is left to soak, after which the spent acid is rinsed from the carbon and discarded to the cyanide destruction tank.

The acid-washed carbon is then hydraulically transferred to the elution column for gold stripping.

The main equipment in this area includes:

- acid wash carbon column – 3 t capacity
- hydrochloric acid feed pump
- spent solution discharge sump pump

17.2.9.2 GOLD STRIPPING (ELUTION)

The gold stripping (elution) circuit uses the AARL process.

The elution sequence commences with the injection of a set volume of water into the bottom of the elution column, along with the simultaneous injection of cyanide and sodium hydroxide solution to achieve a strong NaOH and strong NaCN solution. Once the prescribed volume has been added, the pre-soak period commences. During the pre-soak, the caustic/cyanide solution is circulated through the column and the elution heater until a temperature of 95°C is achieved.

Upon completion of the pre–soak period, additional water is pumped through the trim heat exchanger and elution heater, then through the elution column to the pregnant eluate tank at a rate of 2.0 Bed Volumes (BV)/h. At this stage, the temperature of the strip solution passing through the column is increased to 120°C and the gold is stripped off the loaded carbon.

Strip solution flows up and out of the top of the column, passing through the heat exchanger via the elution discharge strainers and to the pregnant solution tank.

Upon completion of the cool down sequence, the carbon is hydraulically transferred to the carbon regeneration kiln feed hopper via a de-watering screen.

The stripping circuit includes the following key equipment:

- elution carbon column – 3 t capacity
- direct strip solution heater (propane gas)
- heat exchangers
- strip eluate, and pregnant solution tanks

17.2.9.3 CARBON REACTIVATION

Carbon is reactivated in a gas-fired rotary kiln. Dewatered barren carbon from the stripping circuit is held in a 3-t kiln feed hopper. A screw feeder meters the carbon into the reactivation kiln, where it is heated to 650° to 750°C in an atmosphere of superheated steam to restore the activity of the carbon.

Carbon discharging from the kiln is quenched in water and screened on a carbon sizing screen located on top of the CIL tanks to remove undersized carbon fragments. The undersize fine carbon gravitates to the carbon safety screen, whilst carbon screen oversize is directed to the CIL circuit.

As carbon is lost by attrition, new carbon is added to the circuit using the carbon quench tank. The new carbon is then transferred along with the regenerated carbon to feed the carbon sizing screen.

The carbon reactivation circuit includes the following key equipment:

- carbon dewatering screen
- regeneration kiln (electric) including feed hopper and screw feeder

- carbon sizing screen
- carbon quench tank

17.2.9.4 ELECTROWINNING & GOLD ROOM

Gold is recovered from the pregnant solution by electrowinning and smelted to produce doré bars. The pregnant solution from both elution and the intensive cyanidation circuit is pumped through one electrowinning cell with stainless steel mesh cathodes. Gold is deposited on the cathodes and the resulting barren solution is pumped to the CIL circuit for recovery of the remaining dissolved gold.

The gold-rich sludge is washed off the steel cathodes in the electrowinning cell using high-pressure spray water and gravitates to the sludge hopper. The sludge is filtered, dried, mixed with fluxes, and smelted in an electrical induction furnace to produce gold doré. Slag is separated, quenched and any metallic gold is removed. Barren slag is returned to the SAG mill feed. The electrowinning and smelting process takes place within a secure and supervised gold room equipped with access control, intruder detection, and closed-circuit television equipment.

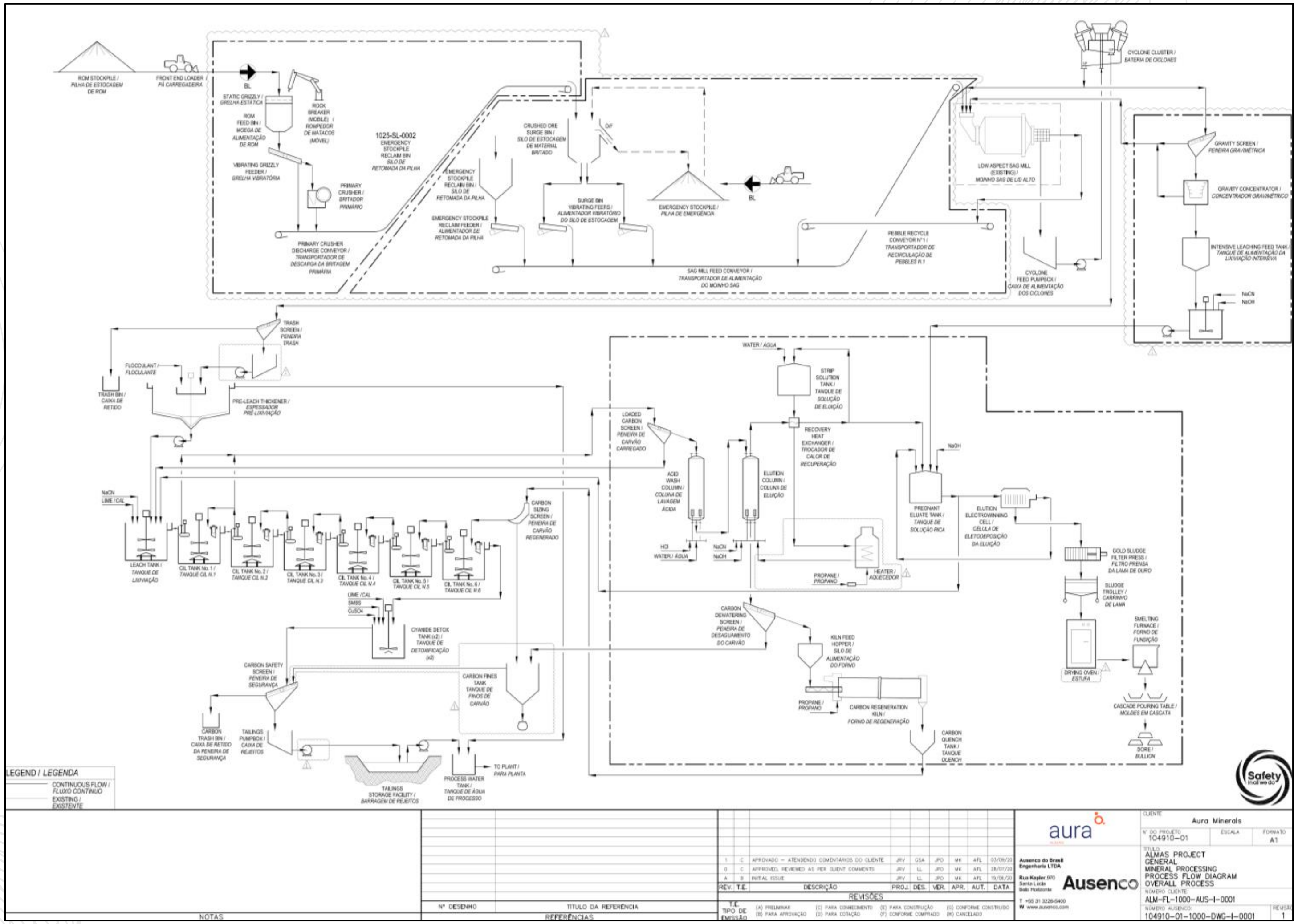
The electrowinning circuit and gold room include the following key equipment:

- electrowinning cell with rectifier
- sludge pressure filter
- drying oven
- flux mixer
- induction smelting furnace with bullion moulds and slag handling system
- bullion vault and safe
- dust and fume collection system
- gold room security system

17.2.10 FLOWSHEET & LAYOUT DRAWINGS

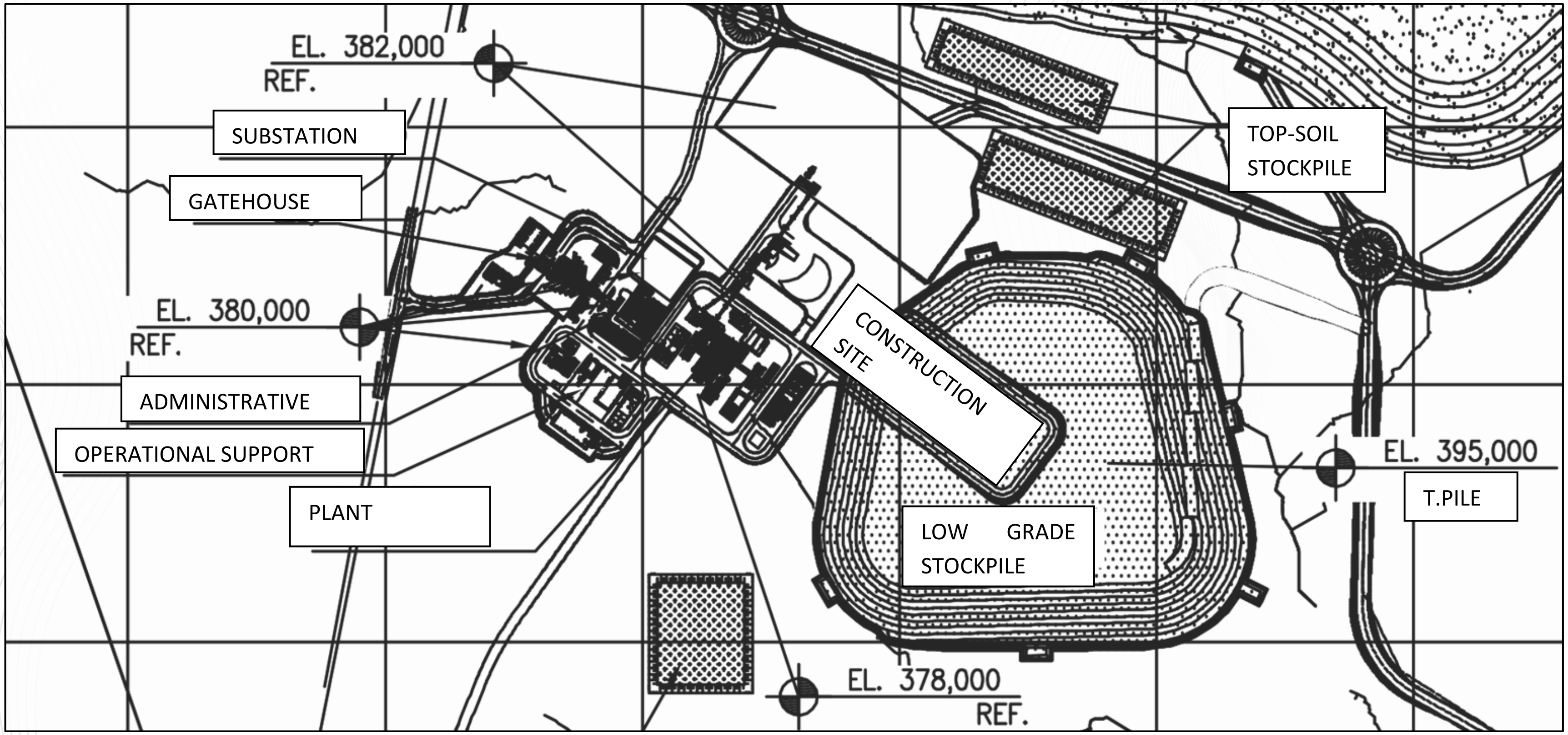
An overall process flow diagram showing the unit operations in the selected process flowsheet is presented in Figure 17-1. Plan and section views of different parts of the process facilities are shown in Figure 17-2 to Figure 17-8.

Figure 17-1 Overall Process Flow Diagram



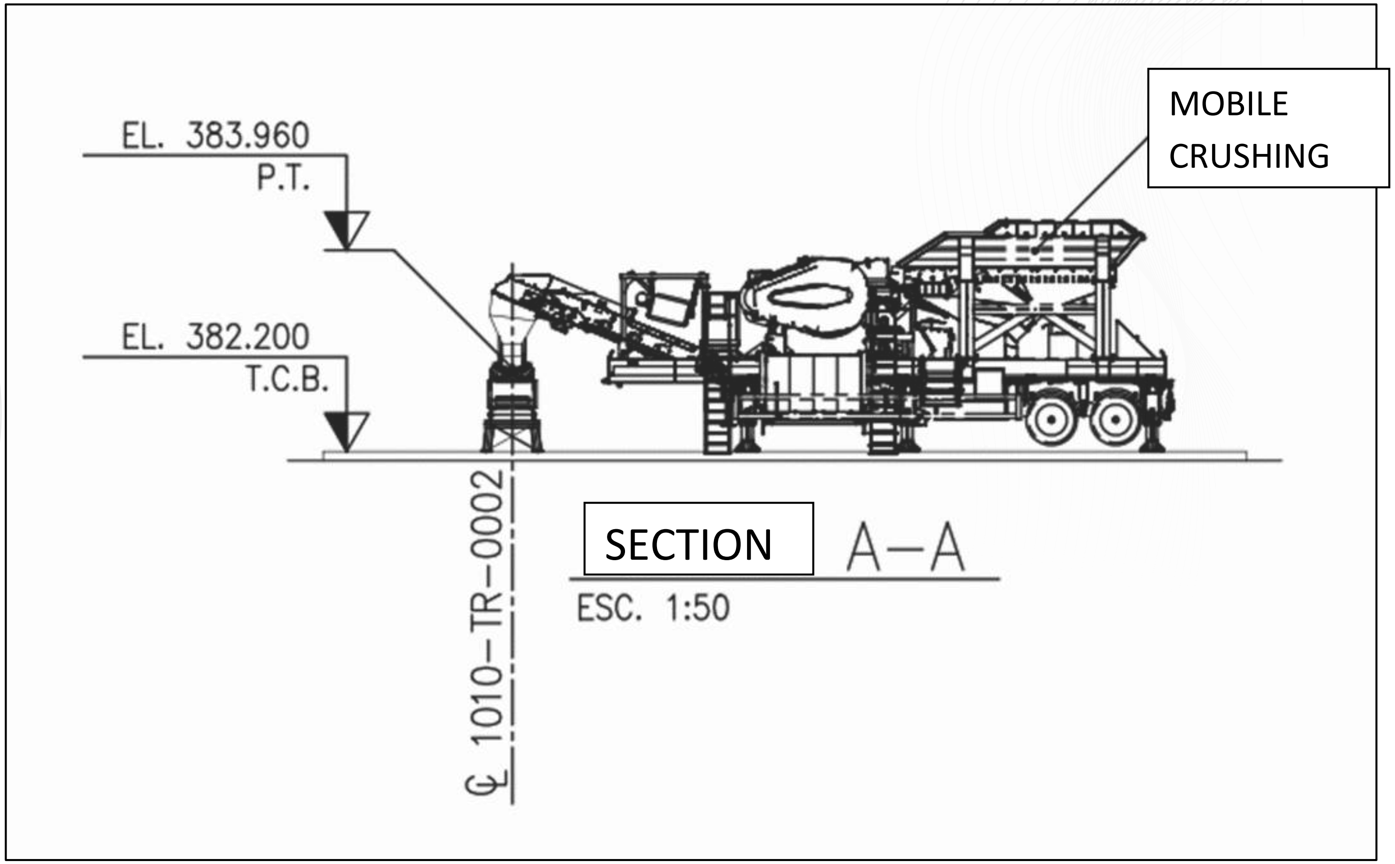
Source: Ausenco, 2020

Figure 17-2 Overall Plant Layout



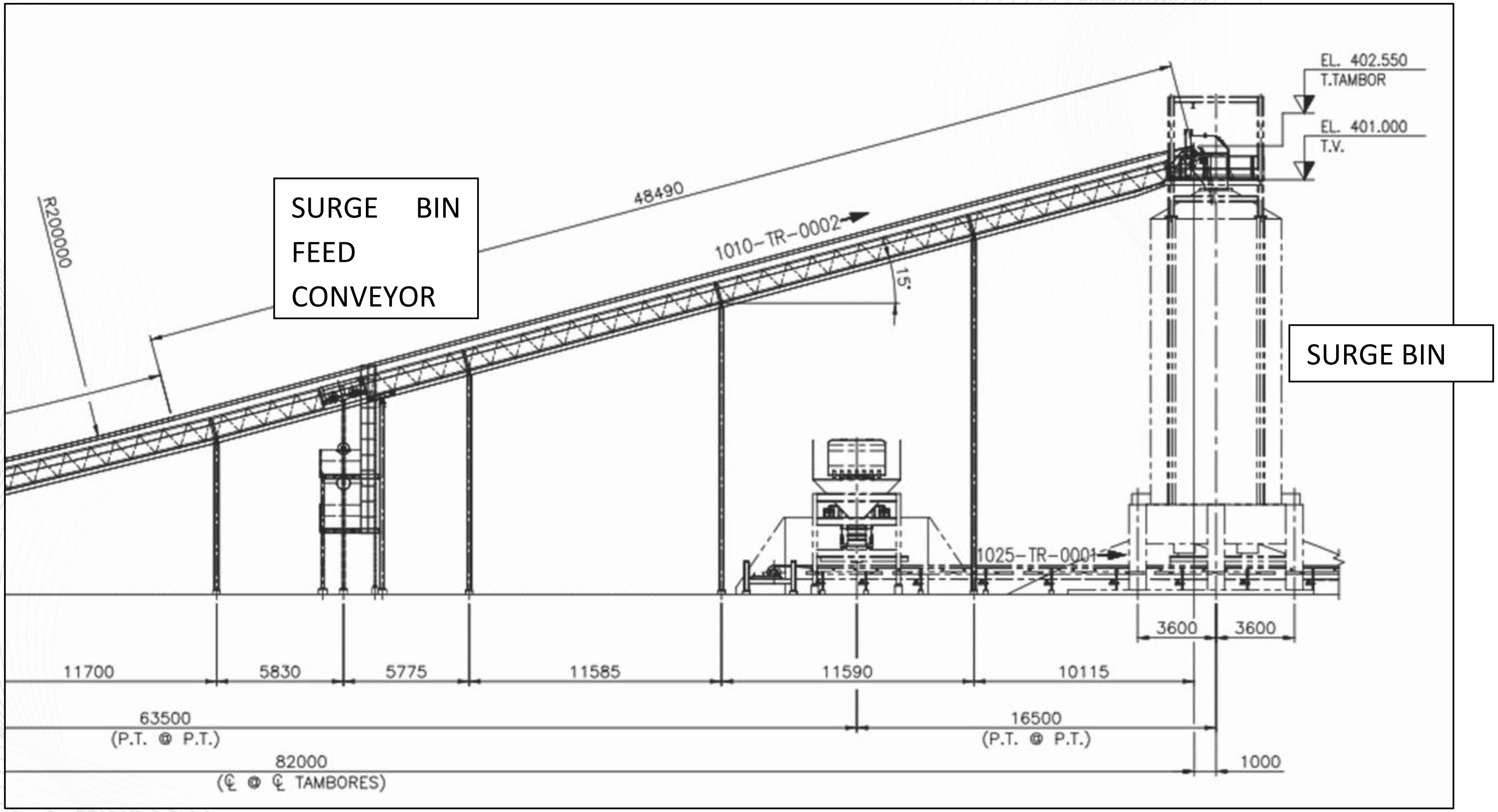
Source: Ausenco, 2020

Figure 17-3 Crushing Area Section



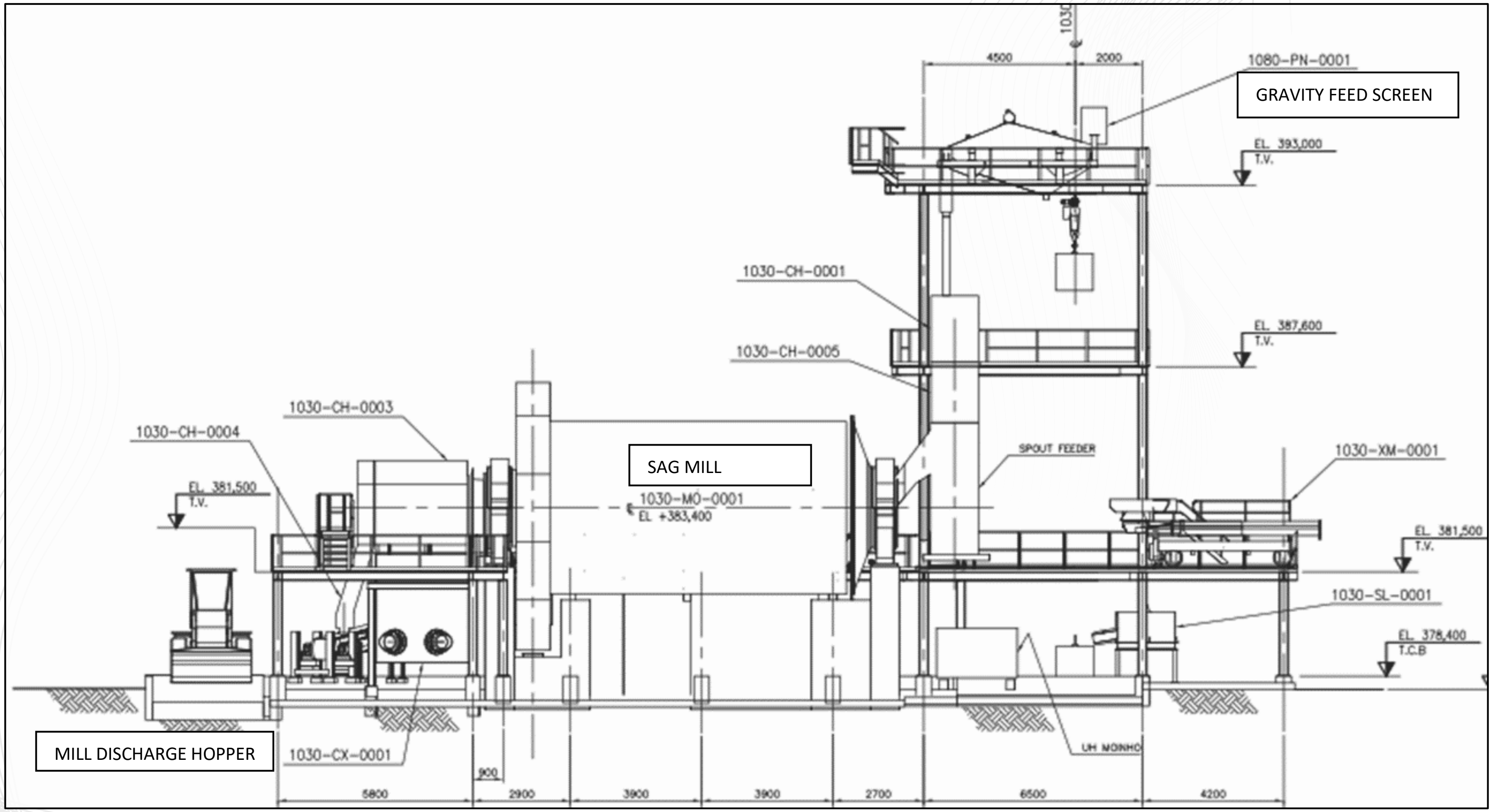
Source: Ausenco, 2020.

Figure 17-4 Surge Bin Area Section



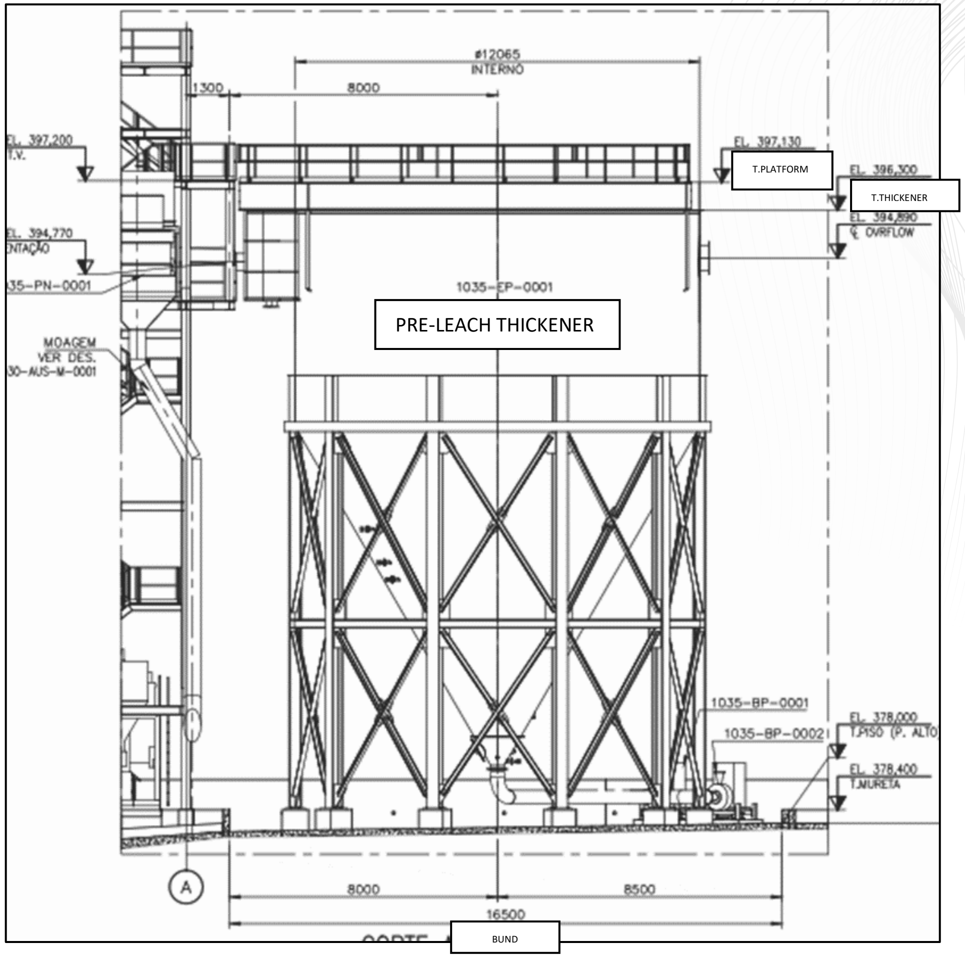
Source: Ausenco, 2020

Figure 17-5 Grinding Area Section



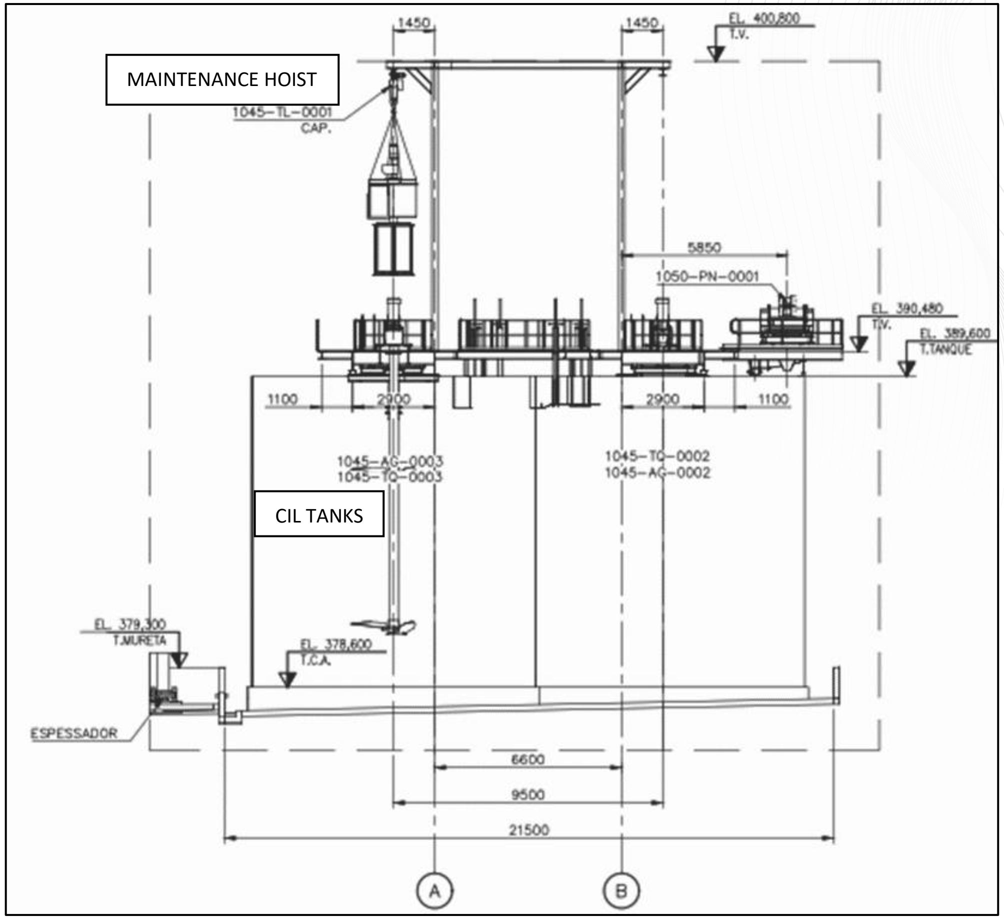
Source: Ausenco, 2020

Figure 17-6 Thickener Area Section



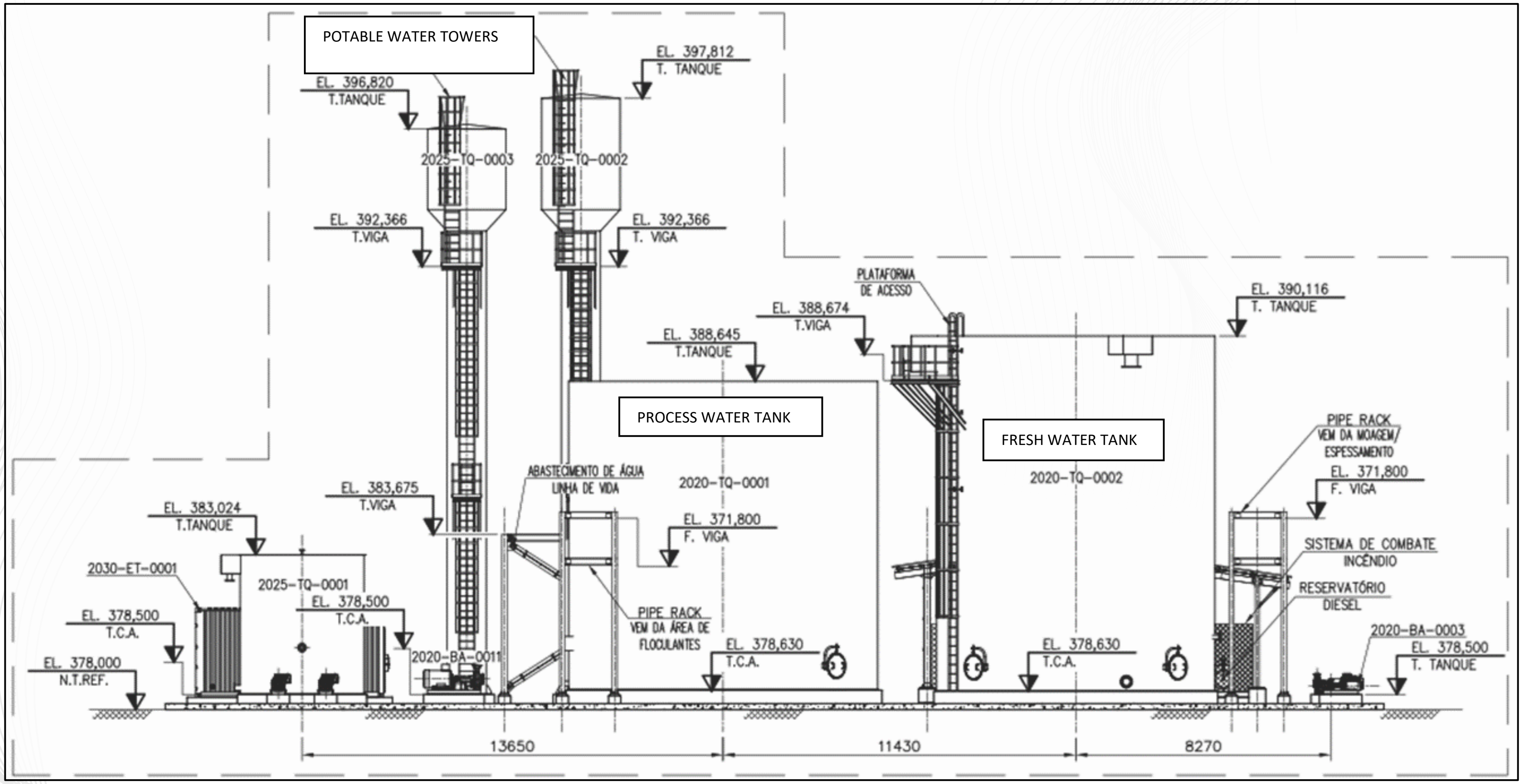
Source: Ausenco, 2020

Figure 17-7 CIL Tank Area Section



Source: Ausenco, 2020

Figure 17-8 Water Distribution Area Section



Source: Ausenco, 2020

17.3 REAGENT HANDLING AND STORAGE

Each set of compatible reagent mixing, and storage systems are located within curbed containment areas to prevent incompatible reagents from mixing. Storage tanks are 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 are located throughout the facilities. Sumps and sump pumps are provided for spillage control.

The following reagent systems are required for the process:

- hydrated lime
- sodium cyanide
- hydrochloric acid
- copper sulphate pentahydrate
- sodium metabisulphite
- sodium hydroxide
- flocculant
- activated carbon
- smelting fluxes

17.3.1 HYDRATED LIME

Hydrated lime ($Ca(OH)_2$) is delivered in 1 t bags, which are lifted using a frame and hoist into the hydrated lime bag breaker on top of the mixing/storage tank. The solid reagent discharges into the tank and is slurried in process water to achieve the required dosing concentration. The slurried hydrated lime is pumped through a ring main with distribution points in leaching and cyanide destruction. An extraction fan is provided over the lime bag breaker/mixing tank to remove reagent dust that may be generated during reagent addition/mixing.

17.3.2 SODIUM CYANIDE

Sodium cyanide (NaCN) is delivered to site in secured boxes containing the 1 t reagent bags. Bags are lifted using a frame and hoist into the sodium cyanide bag breaker on top of the tank. The solid reagent discharges into the tank and is dissolved in water to achieve the required dosing concentration.

After the mixing period is complete, cyanide solution is transferred to the cyanide storage tank using a transfer pump. Sodium cyanide is delivered to the leach circuit, intensive leach circuit and elution circuit with dedicated dosing pumps. An extraction fan is provided over the sodium cyanide bag breaker/mixing tank to remove reagent dust that may be generated during reagent addition/mixing.

17.3.3 COPPER SULPHATE

Copper sulphate pentahydrate ($CuSO_4 \bullet 5H_2O$) is delivered in solid crystal form in small 25 kg bags on pallets and stored in the warehouse. Process water is added to the agitated copper sulphate mixing tank. A pallet of bags is lifted using a frame and hoist, and periodically a single bag is placed on the copper sulphate bag breaker on top of the tank. The solid reagent falls into the tank and is dissolved in water to achieve the required dosing concentration.

Copper sulphate solution is transferred by gravity to the copper sulphate storage tank, which has a stacked arrangement with the mixing tank. Copper sulphate is delivered to cyanide destruction circuits using the copper sulphate dosing pump. An extraction fan is provided over the copper sulphate bag breaker/mixing tank to remove reagent dust that may be generated during reagent addition/mixing.

17.3.4 SODIUM METABISULPHITE

SMBS ($Na_2S_2O_5$) is delivered in the form of solid flakes in 1 t bulk bags and stored in the warehouse. Process water is added to the agitated SMBS mixing tank. Bags are lifted using a frame and hoist into the SMBS bag breaker on top of the tank. The solid reagent falls into the tank and is dissolved in water to achieve the required concentration. After the mixing period is complete, SMBS solution is transferred to the SMBS storage tank using the SMBS transfer pump. SMBS is delivered to the cyanide destruction circuit using the SMBS dosing pump. An extraction fan is provided over the SMBS mixing tank to remove SO_2 gas that may be generated during mixing. The SMBS mixing area is ventilated using the SMBS area roof fan.

17.3.5 SODIUM HYDROXIDE

Sodium hydroxide (NaOH or caustic soda) solution at 35% strength is delivered in 1 m^3 intermediate bulk containers (IBC) as a solution and stored adjacent to the elution circuit until required. Dosing pumps automatically deliver the reagent to the required locations—gravity concentrate leach circuit, elution circuit, and electrowinning—to ensure the dosing requirements are met.

17.3.6 HYDROCHLORIC ACID

Hydrochloric acid (HCl) is delivered in 1 m^3 IBC at 33% solution strength and stored adjacent to the elution circuit until required. Hydrochloric acid is mixed with raw water (inline) to achieve the required 3% w/v concentration. Hydrochloric acid is delivered to the acid wash circuit using the hydrochloric acid dosing pump.

17.3.7 FLOCCULANT

Powdered flocculant is delivered to site in bulk bags and stored in the warehouse. A self-contained mixing and dosing system is installed, including a flocculant storage hopper, flocculant blower, flocculant wetting head, flocculant mixing tank, and flocculant transfer pump. Powdered flocculant is loaded into the flocculant storage hopper using the flocculant hoist. Dry flocculant is pneumatically transferred into the wetting head, where it is contacted with water.

Flocculant solution, at 0.50% w/v, is agitated in the flocculant mixing tank for a pre-set period. After a pre-set time, the flocculant is transferred to the flocculant storage tank using the flocculant transfer pump. Flocculant is dosed to the various high rate thickeners using variable speed helical rotor style pumps. Flocculant is further diluted just prior to the addition point.

17.3.8 ACTIVATED CARBON

Activated carbon is delivered in solid granular form in 0.5 t bulk bags. When required, the fresh carbon is introduced to the carbon quench tank, or directly to the final CIL tank.

17.3.9 ANTI-SCALANT

Anti-scalant is delivered as a solution in IBC and stored in the warehouse until required. Anti-scalant is dosed neat, without dilution. Positive displacement-style dosing pumps deliver the anti-scalant to the strip solution tank as needed.

17.3.10 GOLD ROOM SMELTING FLUXES

Borax, silica sand, sodium nitrate, and soda ash are delivered as solid crystals/pellets in bags or plastic containers and stored in the warehouse until required.

17.4 SERVICES & UTILITIES

17.4.1 PROCESS / INSTRUMENT AIR

High-pressure air at 700 kPag is produced by compressors to meet plant requirements. The high-pressure air supply is dried and used to satisfy both plant air and instrument air demand. Dried air is distributed via the air receivers located throughout the plant.

17.4.2 LOW PRESSURE AIR

Compressed air is injected into the leach/CIL tanks and cyanide detox tanks to meet oxygen requirements.

17.5 WATER SUPPLY

17.5.1 RAW WATER SUPPLY SYSTEM

Raw water is supplied to a raw water storage tank. Raw water is used for all purposes requiring clean water with low dissolved solids and low salt content, primarily as follows:

- gland water for pumps
- reagent make-up
- elution circuit make-up
- raw water is treated and stored in the potable water storage tank for use in safety showers and other similar applications
- fire water for use in the sprinkler and hydrant system

17.5.2 PROCESS WATER SUPPLY SYSTEM

Overflow from the pre-leach thickener and TMF decant water meet the main process water requirements. Raw water provides any additional make-up water requirements.

17.5.3 GLAND WATER

One dedicated gland water pump is fed from the freshwater tank to supply gland water to all slurry pump applications in the plant.

18 PROJECT INFRASTRUCTURE

18.1 OVERALL SITE

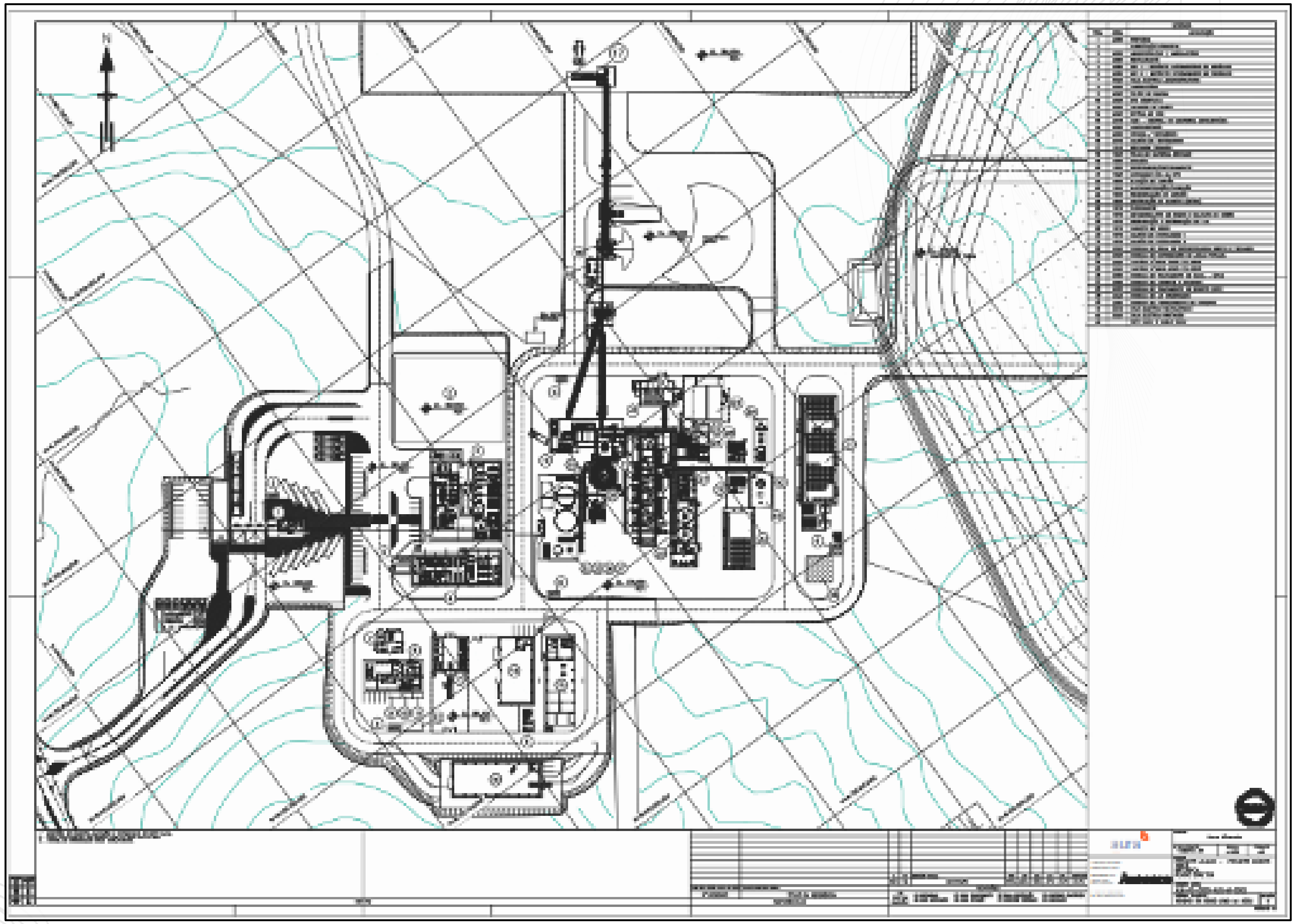
The overall site plan (see Figure 18-1) shows the major project facilities, including the open pit mines, tailings management facility (TMF), waste rock facilities, mine services and access roads. Access to the facility is from the west side of the property from the existing access road. Main access 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 west of the Paiol deposit, with the TMF to the southwest.

Site selection took into consideration the following factors:

- locate the major process equipment foundations on competent bedrock and utilise rock anchors for foundation design
- 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
- locate the ready line close to the mining admin/office area and changehouse

Figure 18-1 Overall Site Plan (Ausenco, 2020)

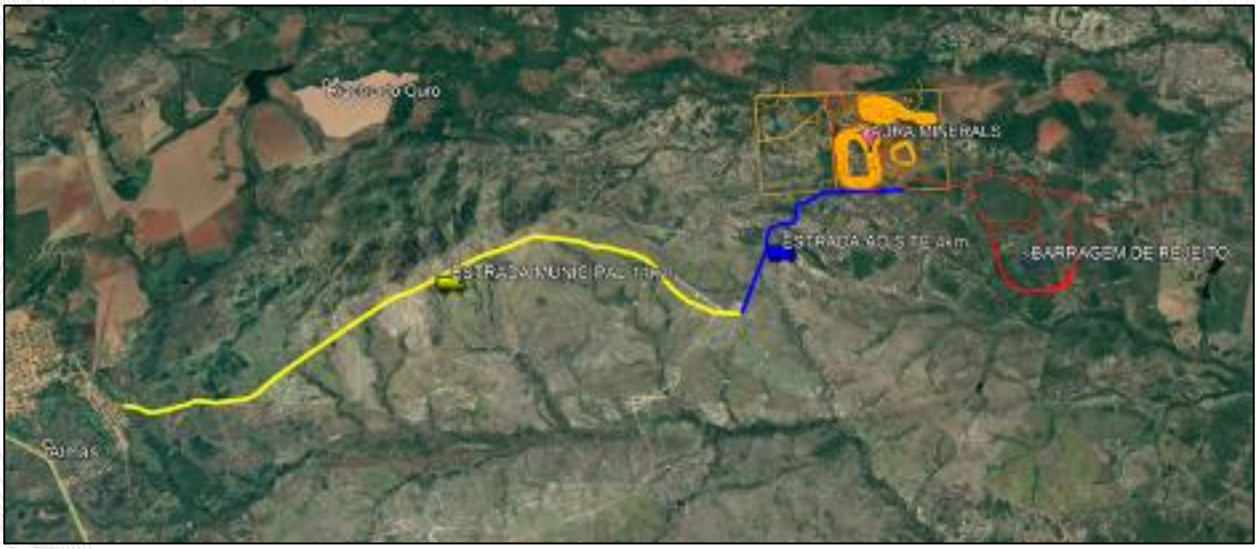


18.2 ROADS

18.2.1 ACCESS TO SITE

From the municipality of Almas (state of Tocantins, Brazil) to site, access is via 15 km of road, as shown in Figure 18-2. The first 11 km is on the municipal road and the remaining 4 km is on a rural road that also provides access to other properties. In the next phases of engineering, Aura should consider improvements on this road, to widen the road, improve the drainage system and follow better safety standards.

Figure 18-2 Site Access (Ausenco, 2020)



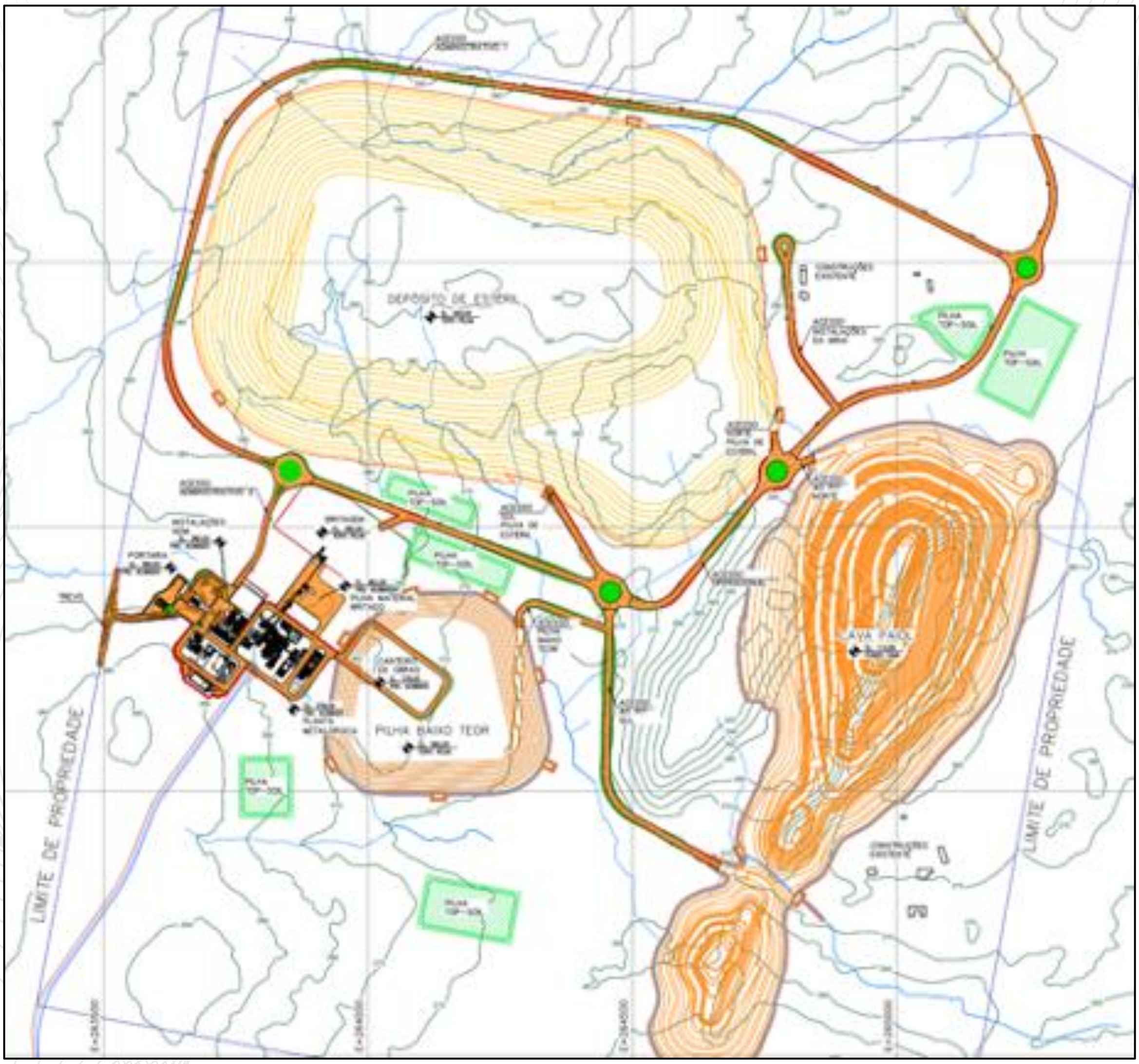
18.2.2 PLANT SITE ROADS

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 (Ausenco, 2020)



18.3 POWER SUPPLY

18.3.1 ELECTRICAL POWER SOURCE

Power will be provided from an existing sub-station at Almas city operated by ENERGISA, the local power utility. A new 18 km, 138kV overhead power line will be constructed to the project site main substation, located to the west of the process plant close to the administrative area. The powerline 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 fenced 138kV substation site will contain an incoming structure and isolation switch, main circuit breaker, provision for utility metering, bus work to deliver 138kV power to a 12/15 MVA stepdown transformer complete with primary circuit breaker, and isolating switches. This transformer will feed associated secondary switchgear and is arranged to provide 13.8kV power to the main processing plant, the crushing plant, the Administration Area, the accommodation area, the mine support area, and the raw water supply and recycle system. Provision is included for automatically switched capacitor banks to assist with site power factor correction. The sub-station will be automated to allow for remote operation.

18.3.2 ELECTRICAL DISTRIBUTION

The primary distribution voltage will be radial, at 6.6 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 MVA ONAN/ONAF transformer from 138 to 6.6 kV.

18.3.4 SECONDARY SUBSTATIONS

Site electrical was selected and designed around the major load centers and are shown in Table 18-1.

Table 18-1 Plant Substations

TAG NUMBER	TYPE	CHARACTERISTICS	POWER DISTRIBUTION FROM MAIN
3015-SE-0001 (Metallurgy)	E-room	Feed: 6.6 kV-25 kA Process loads: 480 V-50 kA Lighting: 380/220 V-50kA	Conduits – 150 m
3020-SE-0001 (Crushing)	E-room	Feed: 6.6 kV-25 kA Process loads: 480 V-50 kA Lighting: 380/220 V-50kA	Conduits – 180 m
3040-SE-0001 (Administrative)	E-room	Feed: 6.6 kV-25 kA Process loads and lighting: 380/220 V-50kA	Conventional aerial network - 110 m Conduits – 160 m
3050-SE-0001 (Raw water capture)	Skid	Feed: 380 kV Process loads and lighting: 380/220 V-50kA	Conduits – 60 m Conventional aerial network: from main substation to derivation – 1,300 m. From derivation to substation – 8,000 m.
3055-SE-0001 (Decant water capture)	Skid	Feed: 380 kV Process loads and lighting: 380/220 V-50kA	Conduits – 60 m Conventional aerial network: from main substation to derivation – 1,300 m. From derivation to substation – 1,400 m.

The substations will feed the following areas:

- 3015-SE-0001: Grinding, thickening, gravity, leach, detox, elution and electrowinning, reagentes, compressed air system, water distribution systems

- 3020-SE-0001: Primary crushing and stockpile/surge bin
- 3040-SE-0001: Administrative buildings, shop and laboratory
- 3050-SE-0001: Raw water capture system
- 3055-SE-0001: Decant return water system

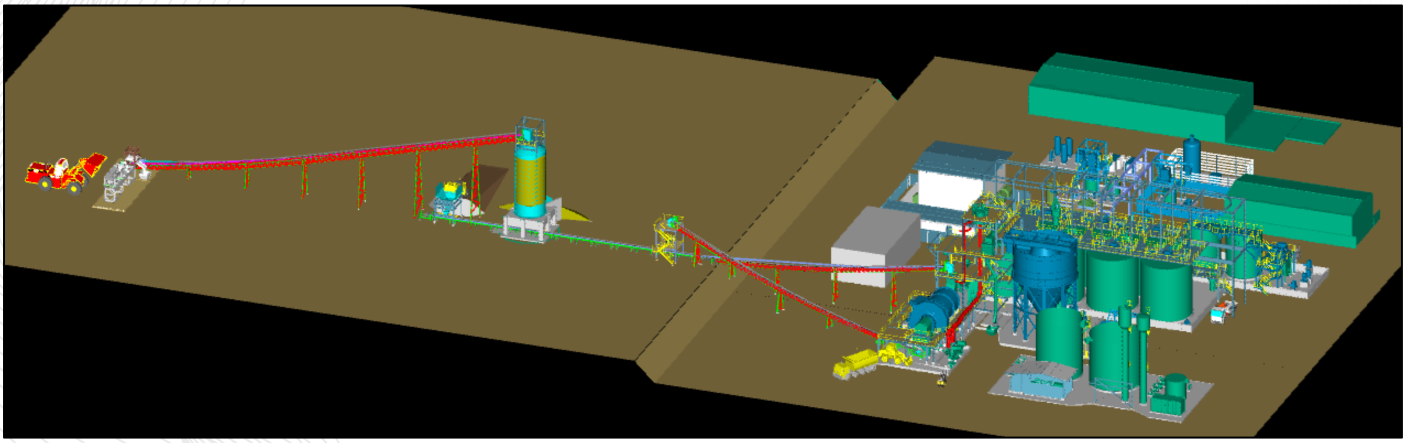
18.3.5 EMERGENCY POWER

Two 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 centre.

18.4 SUPPORT BUILDINGS

Figure 18-4 shows a 3D image of the process plant.

Figure 18-4 Rendered image of the process plant (Ausenco, 2020)



18.4.1 PRIMARY CRUSHING AREA

The primary crushing area will be located northeast 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 a 30 m (long) x 33.7 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 5 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 46 m (long) x 21.5 m (wide) and will include one 10-m diameter leach tank 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 south 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 a 13.5 m (long) x 13.5 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 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 18 m (long) x 7.5 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 5 m (wide). The flocculant is also separate from the other reagents, to be closer to the thickener and minimize piping. The area is 12 m (long) x 9 m (wide). The area for the detox reagents (i.e. sodium metabisulphite and copper sulphate) is next to the detox tanks, to the east of the leach tanks, and the contained area is 12 m (long) x 7.5 m (wide).

Two reagent storage houses are planned, with one exclusive for cyanide being 25 m (long) x 12 m (wide), and for all other reagents being 60 m (long) x 30 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 provided 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.

There are buildings from the old VALE operation that are in a good state of structural conservation, but which will need renovation and adaptations that must be made by the company hired for technical support and maintenance of the equipment and also for the administrative team.

Aura will supply water and electricity at the contracted company's facilities.

18.4.7 WASTE MATERIAL WAREHOUSE

The waste material warehouse will be a steel structure building, with an area of 225 m² to store hazardous and class I and II waste. The warehouse will have access on either side, allowing entrance of a forklift.

18.4.8 WAREHOUSE

The warehouse will be a steel structure building, with an area of 450 m², and an uncovered yard also with an area of 450 m². There will be an office with two workstations.

18.4.9 MAINTENANCE SHOPS & CHANGEROOM

The plant maintenance shop will be a steel structure building, with an area of 360 m², with separate stalls for maintenance, a lubricant storage area, local compressors and an area to wash parts. The changeroom will be adjacent to the maintenance shop, in a two-storey building, with the top floor being office space.

18.4.10 CORE SHED

The core shed will have an area of 600 m² and will be constructed of steel structures with masonry and a metallic roof. An office for geologists and technicians with two workstations is included in this area. The doors on either side are sufficiently big for truck access.

18.4.11 EXPLOSIVES STORAGE & 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 experience carried out at the Apoena Unit, which is also 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 certified containers, which will facilitate a quick and safe installation.

Aura plans to hire companies to set up these facilities in the first quarter of 2021.

The entire area will be security fenced with a guard post at the entrance.

18.4.12 FUEL STATION

As informed in the previous section 18.4.6, the entire mine support infrastructure, including the Fuel Station. The contracted company must also supply the fuel tanks to supply its equipment.

For Aura's mobile equipment, they will be supplied by a mobile supply train with fuel supplied by the gas stations in the city of Almas.

For vehicles the supply will also take place at gas stations in the city of Almas.

18.4.13 PLANT ADMINISTRATION BUILDING

The administration office will have an area of 912 m², in a "U" shape. It will contain the administrative office space and the control room, both totalling 52 workstations, in addition to meeting rooms, a training room and a small lunchroom. The medical station will also be in this building.

18.4.14 MESS HALL

The mess hall will be composed of an industrial kitchen area, the meal area with tables and an area to return the trays with garbage bins for waste. There will be room for 72 people to eat simultaneously.

18.4.15 LABORATORY

The laboratory will be a masonry, single storey building on precast concrete blocks, totalling 300 m² of area. This area considers office space, restrooms, and separate areas for the typical assays for a gold plant, such as sample preparation and fire assays.

18.4.16 SECURITY GATE

The security gatehouse will be a building with 110 m² of area. It will include a waiting / training room, accessible men's and women's restroom installations, a locker area to store the belongings of visitors and employees, a reception with workstations for two guards and one supervisor, a small lunchroom area and a search room. The passage area to enter the site will have turnstiles.

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 were analysed using the most appropriate drill holes as per their location. There was 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 1.5 m for all areas of the plant. The exceptions 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 TAILINGS STORAGE FACILITIES (TSF)

18.6.1 PROJECT WATER BALANCE

In 2020, GEOHYDROTECH ENGENHARIA prepared a water balance study, report ALM-RL-6005-GHT-Y-0001. This report considered the inflows and outflows to the tailings dam reservoir. The tailings from the gold ore beneficiation process will be deposited in the reservoir with a solids content of 47%. Water for process use is recovered from the Tailings Storage Facility (TSF) located 2 km due west of the Paiol open pit. All plant tailings, after cyanide detoxification are pumped to the TSF. Make-up water is drawn from the largest local river, Rio Manuel Alves, and pumped to the process plant to maintain water at all times of the year.

A site water balance was prepared based on the inflows and outflows to the TSF. The water balance takes into the following inflows and outflows (see Figure 18-5);

Inflows:

- Tailings slurry water
- Basin surface run-off
- Direct precipitation on TSF surface and reservoir surface

Outflows:

- Evaporation from active tailings surface and reservoir surface
- Decant ("return flow" to maintain a portion of the original runoff downstream of the dam)
- Water retained in final tailing particle pore space

- Recirculation to process plant
- See page

The meteorological data series closest to the project site that meet these requirements are found in Taguatinga (TO), located about 140 km southeast of Almas (TO), and Porto Nacional (TO), located about 150 km northwest of Almas (TO).The local climate consists of two distinct seasons, wet from November to April and dry from May to October.

The normal ranges of precipitation and evaporation in the region of Almas (TO) are respectively illustrated in figures 18-6 and 18-7. The top of each bar corresponds to the upper tertile observed in that month, while the base of each bar corresponds to the lower tertile.

Based on the water balance of the plant carried out by AUSENCO, the plant will need approximately 41 m^3/h of new water. In the water balance carried out by GEOHYDROTECH ENGENHARIA the maximum need for make-up water in the tailings dam will be in the order of 140 m^3/h, so the maximum need for new water for the project will be in the order of 185 m^3/h and this is the licensing flow (outorga) requested from NATURATINS, the Tocantins State Environmental Agency.

Figure 18-5 Flowsheet Water Balance

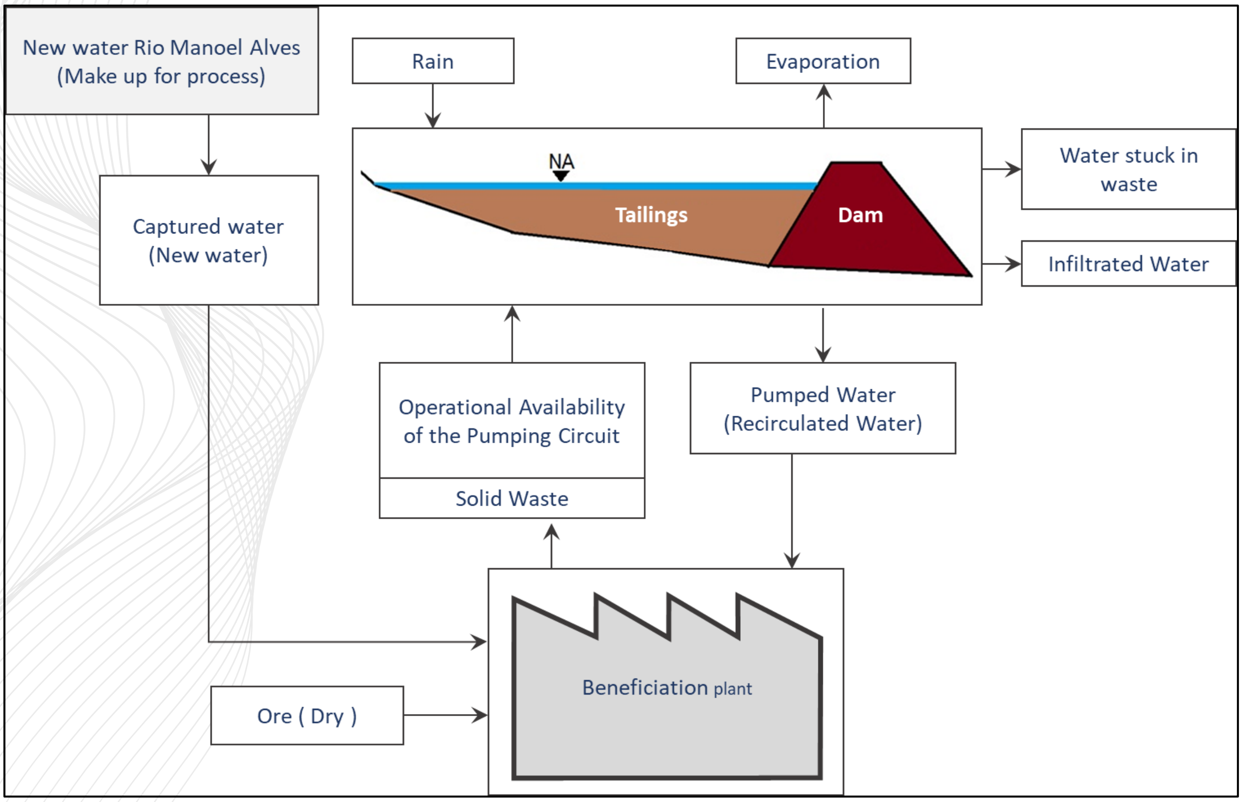


Figure 18-6 Normal Precipitation Range in the Region of Almas.

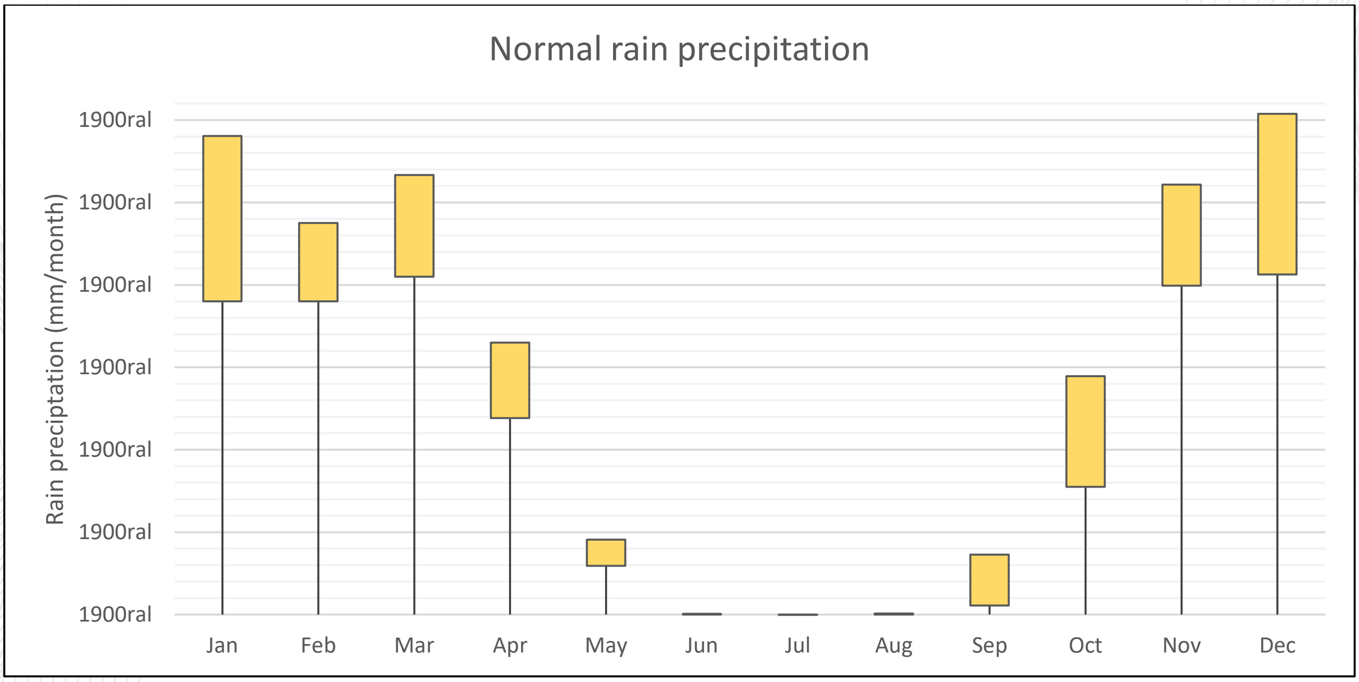
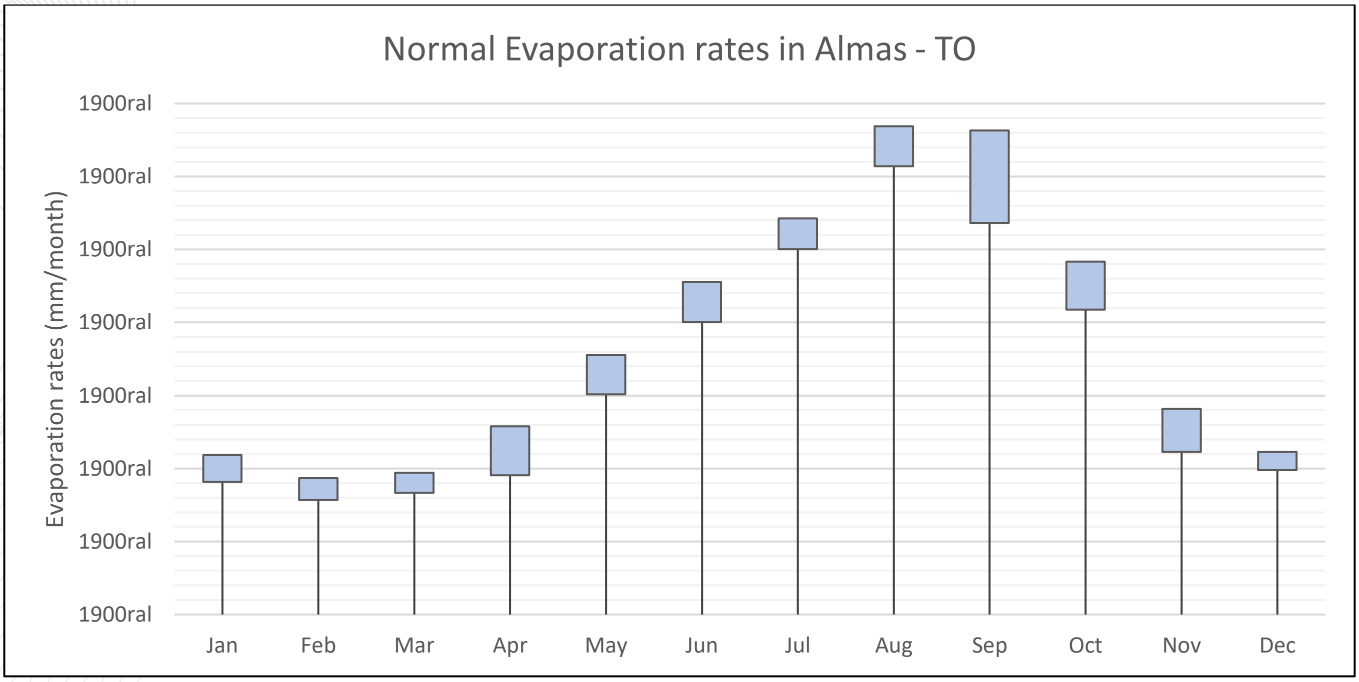


Figure 18-7 Normal Evaporation Range in the Region of Almas



18.6.2 TAILINGS STORAGE FACILITY

The mine will require tailings storage facilities (TSF) for the duration of its life. The company GEOHYDROTECH ENGENHARIA was contracted to carry out a conceptual study for the storage of about 15 million cubic meters of tailings. This study resulted in a tailings dam with up to 5 raising stages as shown in figure 18-8.

If necessary, the Vira Saia pit can also be used to store tailings after exhausting the economic ore. This pit will have a capacity of about 6 million cubic meters.

At the currently assumed tailings density, the mine will create approximately 15 million cubic meters of tailings over its life. The basic strategy for handling tailings is shown in Table 18-2.

It is important to note that the tailings dam should also accumulate water from rain and from the water balance of process water without failing to comply with Brazilian legislation, so the capacity should always be greater than the volume of tailings generation. This greater storage capacity will also allow the circuit to be closed without discharging water to the environment. See the figure 18-9.

All Brazilian laws and regulations in force until the middle of 2020 are being met in the design of this tailings dam.

Figure 18-8 Typical section of TSF

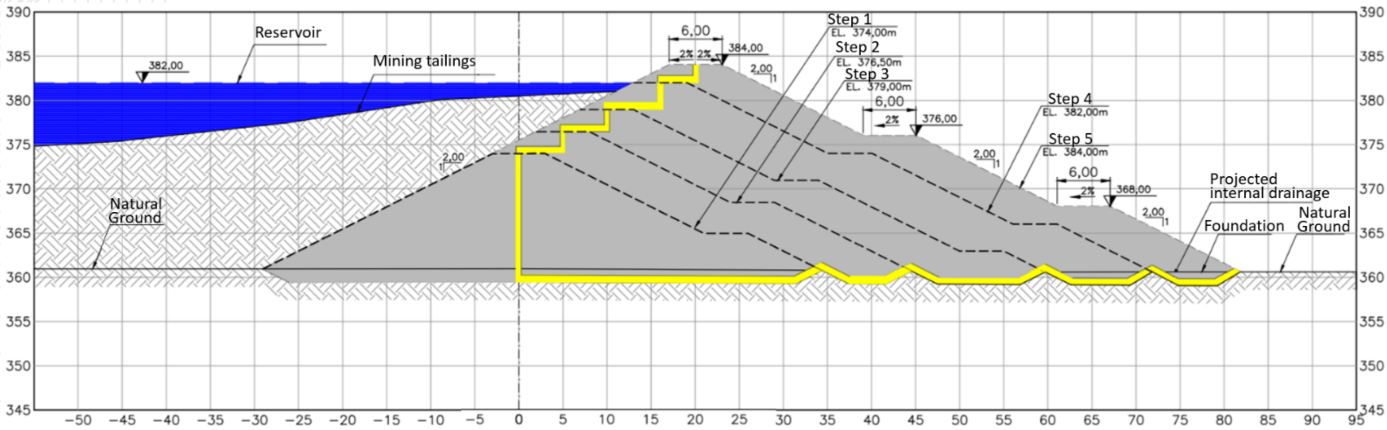


Figure 18-9 Elevation of Crest and Spillway of TSF

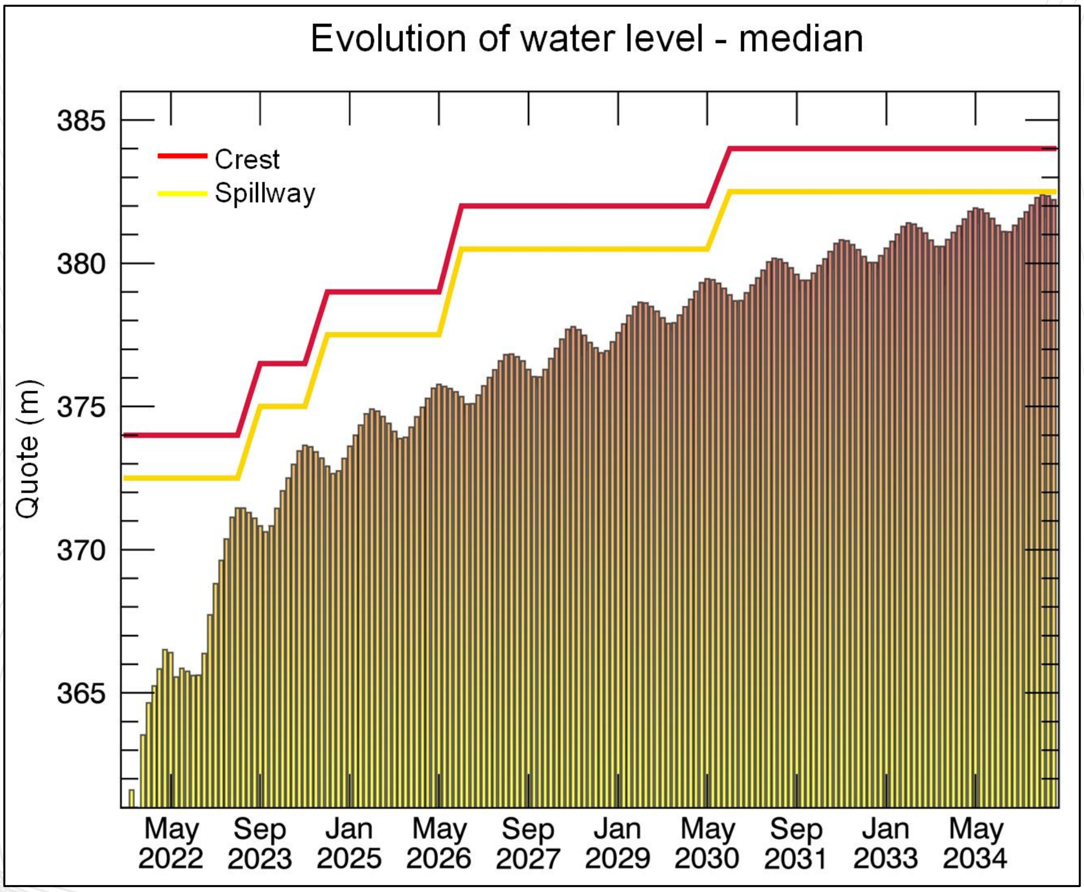


Table 18-2 Tailings Handling

TSF FACILITY	APPROXIMATE YEARS OF USE	APPROXIMATE VOLUME OF TAILINGS
Tailings Dam - Phase 1 (El. 374m)	2 years	2.6 million m^3
Tailings Dam - Phase 2 (El. 376.5m)	1.4 years	1.8 million m^3
Tailings Dam - Phase 3 (El. 379m)	2.1 years	2.7 million m^3
Tailings Dam - Phase 4 (El. 382m)	3.6 years	4.7 million m^3
Tailings Dam - Phase 5 (El. 384m)	2.8 years	3.7 million m^3
Tailings Dam Total	**11.9 years**	**15.5 million m^3**
Vira Saia Pit (if necessary)	4.6 years	6 million m^3
Totals	**16.5 years**	**21.5 million m^3**

The Tailings Dam will have a maximum height of 25.0 m and will be constructed in 5 stages as required for the water level in the reservoir. At its final crest elevation of 384 m, approximately 25 m high, the Tailings Dam will be at its capacity of 15.5 million m^3 and will have served the mine for the first 12 years of its life. Sides slopes of the dam will be at 1V:2H, and a 5 m wide bench for stability is proposed 10 m below the crest at each stage of the dam.

The Tailings Dam is be constructed with a homogeneous type soil compacted in lifts. Clay type soils CL and/or ML, as classified by the Unified System of Soil Classification (USCS) will be suitable. The permeability of the compacted material should be less than 1 x 10-5 cm/s. Investigations were conducted that indicated that CL and ML material to build the dam are on site or within economical haul distance. The material is present in the Paiol pit and may be used for the dam construction.

Construction of the dam will start by clearing, grubbing, and topsoil removal. A 2 m deep “cutoff” trench will be constructed along the length of the dam, underneath the dam. The trench serves to stop water seepage below the dam within the soil layer beneath the topsoil, as this layer may be permeable. A vertical drain system will be constructed within the dam to alleviate saturation of the dam. This vertical drain system requires a horizontal outlet, which is provided by a permeable drainage blanket beneath the dam, from the vertical drain to the toe at the outside of the dam. The inside slope of the dam will be lined with riprap rock to protect it from erosion. A synthetic material liner is not proposed. The protective riprap rock will be abundant on site from mining activities and/or borrow pits. The outside slope of the dam will be protected with plantings of native legumes.

A spillway has been designed to release water in the event of extreme storm events for a decamillenary recurrence time. There will be a spillway designed and constructed for each stage of the dam construction. The spillway is proposed as a trapezoidal channel and will accommodate the storm flows from the reservoir’s entire catchment area. The flow and velocity from the spillway were analysed, and energy dissipation measures are proposed prior to the flow emptying onto existing grade.

The foundation soils at the proposed location of Dam were reviewed. The materials were tested in the lab and reviews deemed the soils suitable for Dam foundation.

18.6.3 TAILINGS STORAGE FACILITY – EXECUTIVE ENGINEERING – FIRST STAGE

Geometric Characteristics of the Tailings Dam

The executive project for the first stage of the tailings dam of the Almas Gold Project was prepared by GEOHYDROTECH and presented in the report ALM-RT-6005-GHT-B-0001. The dam will consist of a compacted landfill and raised by the method called downstream, the geometric characteristics of the first construction stage being shown in Table 18-3 below.

Table 18-3 Tailings Dam Data for First Phase

CHARACTERISTICS	UNIT	QUANTITIES
Maximum height	m	13.4
Maximum crest elevation	m	374
Reservoir volume	$x10^3m^3$	~3.590
Total length of the crest	m	779
Elevation of the spillway	m	372.5
Tilt the downstream slope	-	2.0H:1V
Tilt the upstream slope	-	2.0H:1V
Total area occupied by the reservoir	m^2	510,565

Spillway

The dam was designed so that it does not overflow, that is, it works in a closed circuit, however, an emergency spillway system was designed for a 1,000-year return period design rain, as recommended by the NBR 13.028 / 2017 standard. The overflow will be excavated in natural terrain and covered with Geocell. Its geometric characteristics are shown in Table 18-4 and Figure 18-10 below.

Figure 18-10 Typical Section of spillway.

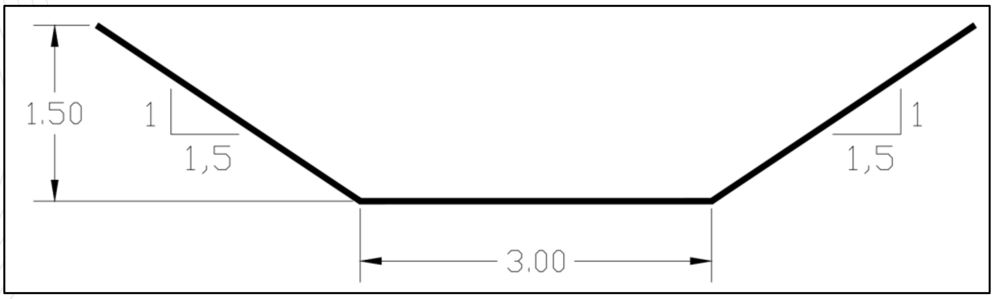


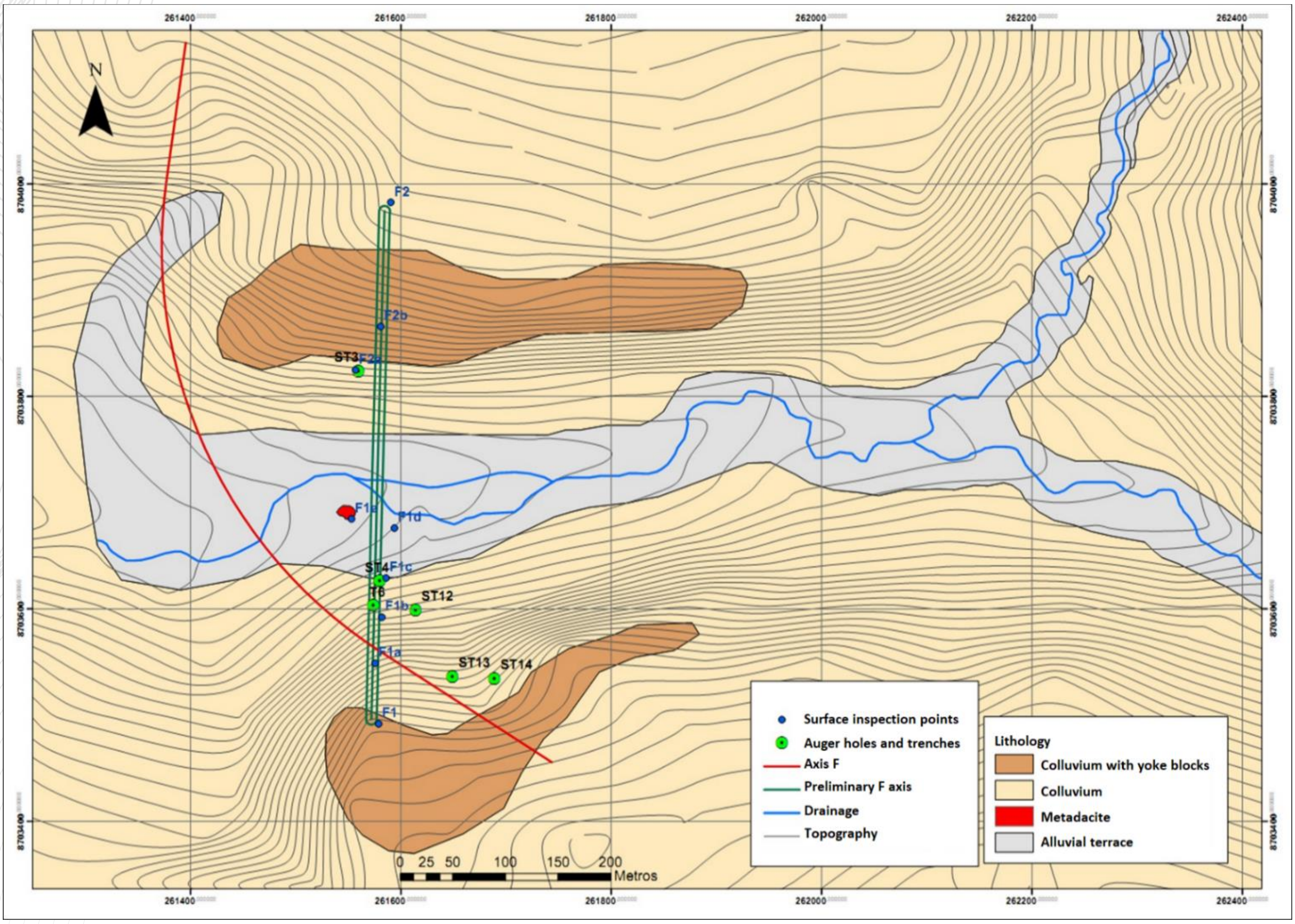
Table 18-4 Geometric Characteristics of Spillway

SECTION	TRAPEZOIDAL
Base [m]	3
Height [m]	1.5
Slopes	1.5 H:1.0 V
Minimum declivity [%]	0.5

Geological-Geotechnical Research and Loan Areas

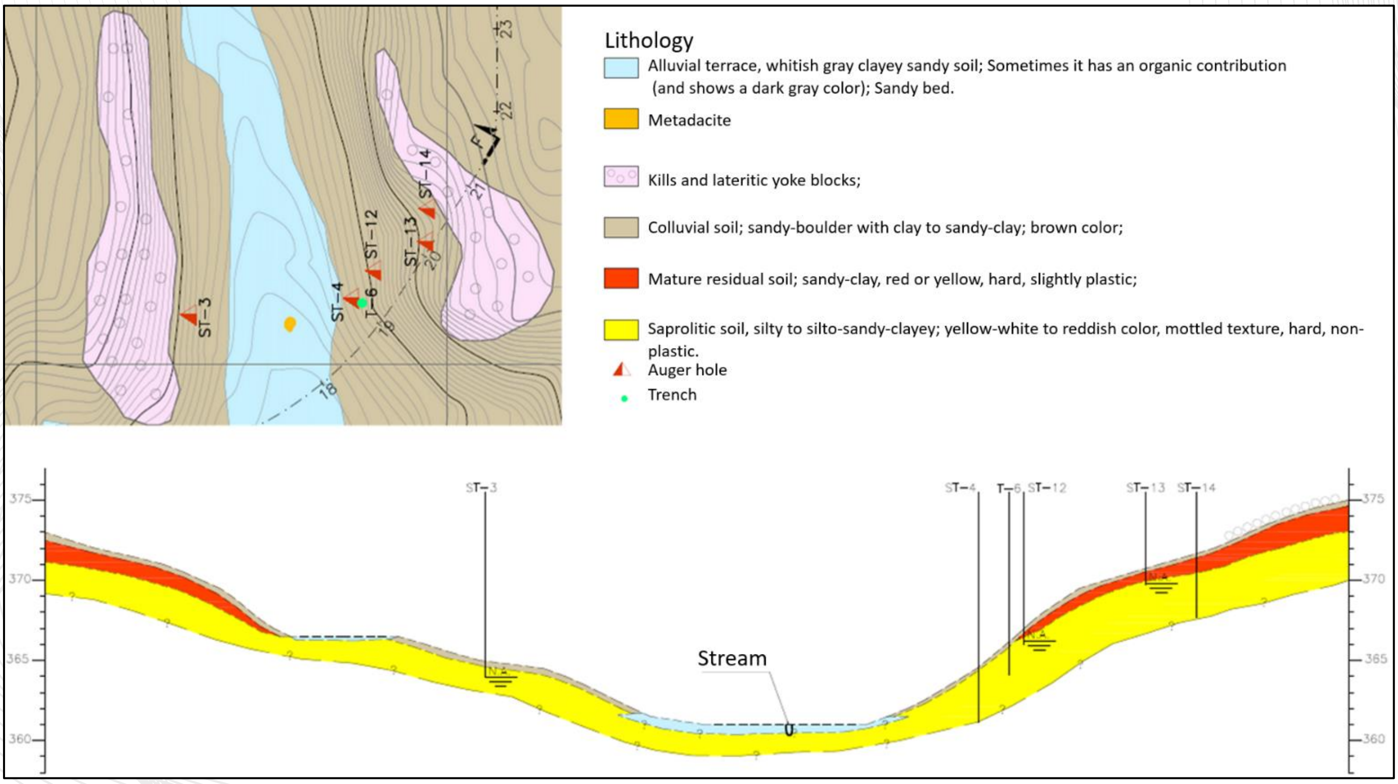
Studies of geological-geotechnical research were carried out in 2011 in the area of implementation of the dam of the Almas Gold Project. In order to know the characteristics of the soil layers existing in the area, drill holes were made, trenching and visual inspections were performed (Figure 18-11).

Figure 18-11 Research location And Lithology of the Dam Implantation Area



From the results of the drilling surveys, located in the area of the central region of the axis of the future dam of the Almas Gold Project, the geological profile was drawn as seen in Figure 18-12.

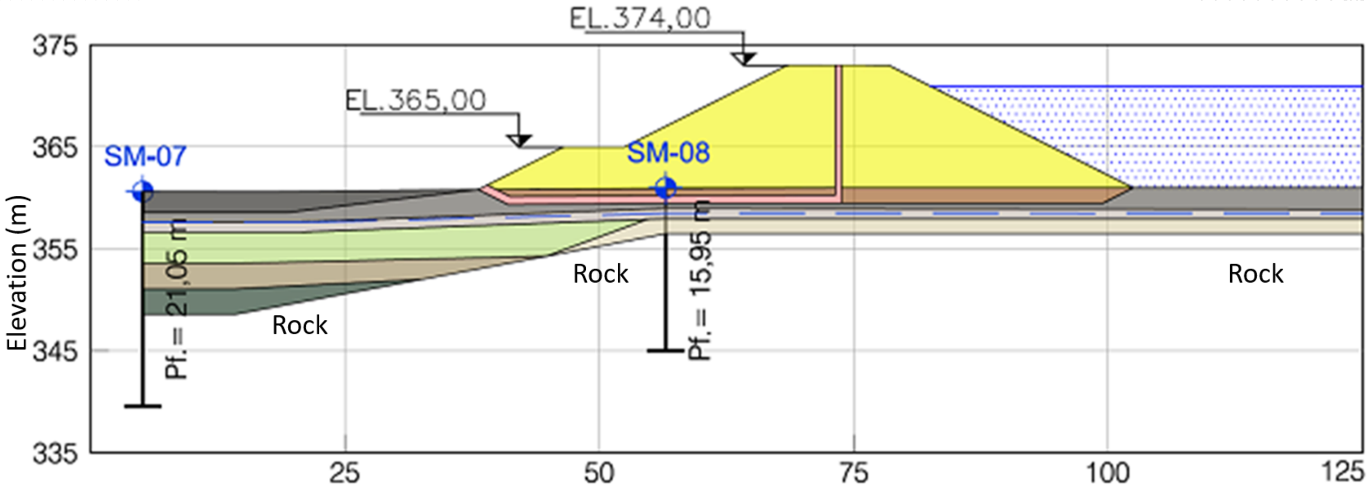
Figure 18-12 Geological Profile of the Main dam Area from Auger Drilling



In Figure 18-12, it can be seen that the profile has a flat to smooth wavy relief, a small intermittent stream crosses the cross section. The approximate dimensions of this stream are between 1.0 and 3.0m wide and a trough of about 0.5 m to 1.0 m deep. It is also observed that the stream has some anastomosed stretches (bifurcated).

Figure 18-13 shows a section made in the central region of the dam, where the SM-7 and SM-8 mixed drilling holes were made. The surface layers are composed of fine to medium silt sand, compacted fine sand with a little silt and a little compacted and fine to medium sand with a little clay. The embankment of the dam is located on fine silt-free and compacted sand.

Figure 18-13 Section of the Central Area of the dam with Mixed Boreholes SM-07 and SM-08.



Color	Name	Model	Unit Weight (kN/m³)	Cohesion (kPa)	Phi (°)
	Compacted landfill	Mohr-Coulomb	15	5	25
	Foundation	Mohr-Coulomb	22	8	27
	Filter	Mohr-Coulomb	19	0	32
	Silky, gray, compacted (fine to medium) sand (Nspt 27)	Mohr-Coulomb	20	10	35
	Fine sand, silty, gray, poorly compacted (Nspt 6-10)	Mohr-Coulomb	19	2	30
	Sand (fine to medium) with little clay, beige and gray (Nspt 19-22)	Mohr-Coulomb	19	5	35
	Sand (fine to medium) with little clay, beige and gray (Nspt 7-9)	Mohr-Coulomb	19	5	30
	Medium loamy sand, beige (Nspt 41-46)	Mohr-Coulomb	20	15	35
	Sand (fine to medium) with little clay, beige and gray (Nspt 7-9)	Mohr-Coulomb	19	5	35
	Medium clayey, greenish gray sand (Nspt 16)	Mohr-Coulomb	19	8	32

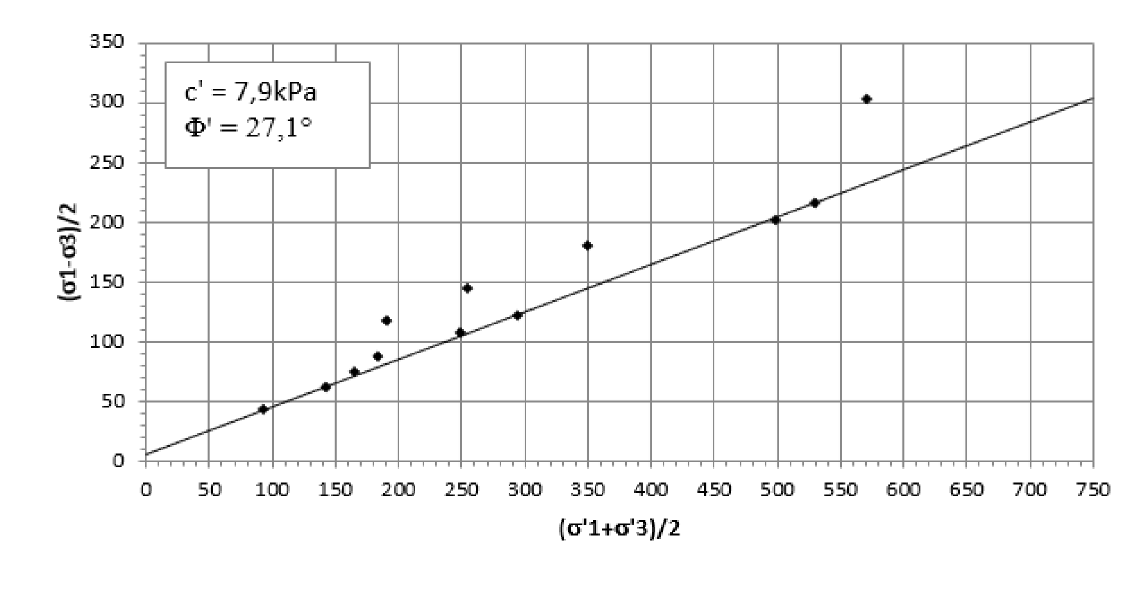
In Report 097/11, issued by EMPRESA SETE - Serviços Técnicos de Engenharia Ltda., on February 23, 2012, results were presented from mixed surveys, drilling surveys and laboratory tests carried out on samples taken from inspection wells. Sampling was carried out along the dam axis (studied in 2012).

Tests were carried out for the conceptual phase with the foundation material to determine the permeability coefficient. Undisturbed samples were taken from the topsoil, which was subjected to permeability tests, with constant and variable loads. During the execution of the mixed surveys, tests were carried out on the loss of water under pressure in the rock mass. The results obtained were:

- Central region: Soil with permeability coefficient ranging from 10-4 to 10-6 cm/s, and rock with permeability from 10-4 to 10-6 cm / s;
- Left dam shoulder: Soil with permeability coefficient ranging from 10E-3 to 10E-6 cm/s, the permeability of the rock mass of the left shoulder was not presented in this report;

CU triaxial tests (densified undrained) were also performed on undisturbed saturated samples. The results were treated to obtain the residual shear strength parameters (at 20% deformation), in terms of effective stresses. The resistance envelope in terms of effective stresses based on the results of the 3 samples is shown in Figure 18-14.

Figure 18-14 Strength envelope in Terms of Effective Stresses for 20% Deformation (Foundation Material)



The values of the permeability coefficients used in the percolation analyses are shown in Table 18.5. These values were adopted based on analysis of the results of tests carried out with material from the loan and foundation area, presented in reports P96, prepared by LTEC, RL097-11, prepared by SETE, and RE 584/12 - Rev.01, prepared by DIEFRA. The values must be reviewed after carrying out the scheduled laboratory tests.

Table 18-5 Coefficient of permeability of the Materials that Make Up the Dam

MATERIAL	PERMEABILITY COEFFICIENT (m/s)
Compacted landfill (unsaturated)	1.3 E-07
Filter	5.0 E-05
Fundation	7.0 E-08
Sand (fine to medium) silty, gray, compact (SPT 27)	1.0 E-08
Fine silty sand, not very compact (SPT 6-10)	1.0 E-07
Sand (fine to medium) with little clay, beige and gray (SPT 19-22)	1.0 E-07
Sand (fine to medium) with little clay, beige and gray (SPT 7-9)	1.0 E-07
Medium clayey sand, beige (SPT 41-46)	1.0 E-07
Medium clayey sand, greenish gray (SPT-16)	1.0 E-07
Rock	1.0 E-07
Tailings	1.0 E-06

18.7 WATER SYSTEMS

18.7.1 RAW WATER SUPPLY SYSTEM

The design considered that raw water will be captured from the Manuel Alves 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.

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 is not considered. The workers will be housed in rented residences in the cities of Almas, Porto Alegre de Tocantins and Dianópolis.

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

GE21 Consultoria Mineral Ltda. ("GE21") was engaged by Aura Minerals Inc. ("Aura") to prepare an Independent Technical Report ("ITR") containing a Market Study on Gold (chapter 19) and an Economic Analysis (chapter 22) for the Almas Gold Project, located in the State of Tocantins, 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, cooper 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 Almas 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 DFS Report for the Almas Project.

19.2 MARKET STUDY

The World Bank forecast indicates an increase in the average price of gold to US$/oz 1,775 in 2020 from an average of US$/oz 1,392 in 2019. In the next ten years, the gold price is expected to reach around US$/oz 1,400 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, the gold price averaged US$/oz 1,560, which was about 6% higher than December, 2019. Throughout 2020 the spot price of gold reached approximately US$/oz 2,000, 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 purposes all over 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 gold returns in periods of low-interest rates are twice as high as their historical average. Moreover, in such an environment gold seems to be more effective in portfolio diversification, mitigation of risk and long-term returns as compared to government bonds. So, in the current conditions of low-to-negative interest rates, demand for gold from investors and the Central Bank is going to continue strengthening, thereby moving prices up.

The value of US$ 1,558/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 Almas Project includes three gold deposits, Paiol, Cata Funda and Vira-Saia which are located around 14 to 5 km from the town of Almas.

The Paiol Mine, which houses most of the mineral resources to be mined, was formerly operated by CVRD, currently Vale, from 1996 to 2001. The Vira-Saia and Cata Funda areas, which have been degraded by illegal artisanal mining ("Garimpo") over the years, will be open pit mines and the extracted ore will be processed at the Paiol Mine mill plant.

The former open pit is currently filled with water and the waste dump and spent heap leach facilities have been reclaimed. Most of the process equipment and other facilities have been removed in conformance with the reclamation requirements of the state environmental authority.

Most of the environmental and social studies for the resumption of the Paiol Mine, the main mineral deposit of the Almas Gold Project, were carried out between 2010 and 2012, among them the Environmental Assessment (EA), 2011, required for the simplified permitting of the Paiol Mine and, according to the instructions of the Instituto Natureza do Tocantins – NATURATINS, which is the state's regulatory environmental authority. Although the EA has contemplated the socioeconomic aspects of the Vira-Saia and Cata Funda mineral deposits, it is worth noting that it was carried out exclusively for the permitting of the Paiol Mine. The study was conducted and prepared by the consulting firm Conestoga-Rovers e Associados (CRA), from São Paulo.

From this permitting, the Paiol Mine obtained the Installation License No. 5437/2011 (Licença de Instalação –LI), which has already expired, and subsequently the Preliminary License No. 286/2017 (Licença Prévia –LP) and Installation License No. 297/2017 (Licença de Instalação – LI), which is undergoing analysis for renewal by the technical staff of NATURATINS.

Currently, as part of the resumption of the Almas Gold Project by Aura Minerals, additional studies, including, but not limited to, Geochemistry Tests, Water Quality Characterization, Forest Inventory, Detox Tests, Updated Plan for Monitoring and Rescue of Fish and Wildlife are being carried out to support both the renewal of the Installation License as well as other required permits to complete the Paiol Mine Permitting process.

For the permitting of the Vira-Saia and Cata Funda deposits, another Environmental Assessment (EA) was recommended by NATURATINS, since the areas are already degraded and the potential for negative impacts is low.

20.2 OVERVIEW

The project is located in the municipality of Almas, in the state of Tocantins, approximately 276 km southeast of the capital Palmas. It is located in the Manuel Alves River Basin, with a humid to semi-humid climate, with winter droughts and an average rainfall of 1,700 mm / year. Average temperatures range from 32 ° C to 22 ° C, with the highest recorded temperature in September (35 ° C) and lowest in July (15 ° C).

The project's area of influence includes, in addition to Almas, the project's headquarters; other municipalities potentially impacted by the project such as Dianópolis, Porto Alegre do Tocantins and Natividade. All these municipalities are connected by State Highway TO-050, which is the main highway for travel from Palmas to Almas. Among these municipalities, Almas has the largest territorial extent, 4,106.4 km^2, closely followed by Dianópolis and Natividade and finally Porto Alegre do Tocantins, with the smallest area (482 km^2). The demographic density of Almas is 2.1 inhabitants/km^2.

The illiteracy rate in the region is high and it is estimated that approximately 31% of people over the age of 25 have not learned to read or write. Only 36% of households in Almas benefit from the public water distribution system, which relies entirely on underground water supply from wells, and energy is available to 65% of residents. Municipal waste collection benefits 27% of all households.

20.3 MAIN ENVIRONMENTAL AND SOCIAL ASPECTS

20.3.1 VEGETATION AND WILDLIFE

The landscape of the project region is characterized by typical vegetation cover of the Cerrado, a kind of Savanna, which is composed of plant communities in different stages of preservation. The surroundings of these environments have been occupied over the last few years by different types of anthropic activities, such as agriculture, livestock and artisanal mining, which have been progressively mischaracterizing the referred landscape.

In the project's area of influence, mammals such as Tapir (Tapirus terrestrial) and Jaguar (Panthera onca) are found, in addition to about 187 species of birds from 52 families.

Both vegetation and wildlife will be subjected to rescue, management and monitoring before, during and after the suppression of vegetation. The Environmental Permit for the Management of Fish and Wildlife has already been issued by NATURATINS (Permit AMAS No. 7188-2020).

20.3.2 PAIOL OPEN PIT DEWATERING

The former open pit remains as it was left by CVRD, current VALE, and is filled with water (1.2 million m^3) by the accumulation of rainwater and the groundwater table, up to the 365 m level. In order to start the mining development, it will be necessary to dewater the pit.

For the implementation of the current project, part of this water will be used in civil works and dampening of roads, though most of it should be discharged into the Paiol stream. This proved to be the best option, not only because it is the closest to the pit, but also due to a nearby dam that will function as a solid sedimentation basin before the final release into the watercourse, based on water quality monitoring. In addition, the average drainage flow is very close to the flow of the Paiol stream at the end of the rainy season (150 m^3/h).

The grant for both the use of water and its discharge, is currently being analyzed by the technicians of the above-mentioned environmental agency.

The Table 20-1 below shows, in summary, the comparison between the project that was presented in 2010 and the current one to obtain the Open Pit Dewatering Grant.

Table 20-1 Open Pit Dewatering Project

DEWATERING PROJECT	PREVIOUS (2010)	CURRENT (2020)
Drainage System	3 Pumps	1 Pump
Average Drainage Flow	900 m^3/h	150 m^3/h
Operation (h/day)	16 h/days	16 h/day
Duration	90 days	500 days

20.3.3 WATER

The supply of drinking water, both to supply the construction and the operation, will be done, initially, through one existing tubular well in the area, with a flow already granted by NATURATINS of 20 m^3/day (Permit DUI No. 7574-2020). This water, after being collected, will undergo treatment until it reaches the necessary quality standards of potability for human consumption.

Current studies indicated that, during the operational phase, the need for raw or new water will be on average 180 m^3/ h. Complete recirculation of water from the dam to the plant is expected, however, as the water balance in the region is negative, that is, evaporation is greater than precipitation, there will be losses and the water in the dam will not meet the needs of the mill. Thus, water will be pumped from the Manuel Alves River, as already forecast in the Environmental Assessment (EA) of 2011. The difference is that the makeup water estimated in EA was 230 m^3/h, which is greater than the current need of the project.

20.3.4 SOCIOECONOMIC ASPECTS

Employment in Almas is based mainly on informal work. The Municipal Government is the main formal employer in the municipality. More than half (55%) of formal jobs are related to public careers, with the majority of positions held by women. Agriculture ranks second in terms of jobs, accounting for 28% of the total number with 90% of these jobs being held by men. In the agricultural sector, expressive crops are maize, cassava, bananas and, more recently, soy.

Almas experienced negative consequences from Vale's unexpected closure of the Paiol Mine in 2001, which led to an outflow of the population and the consequent economic depression. The resumption of the Almas Project, especially the Paiol Mine, will generate strong expectations for job creation and new opportunities. For this reason, it is planned, in 2021, to resume communication activities with the community to clarify the project, both from the environmental point of view and as well as of job creation and local development.

20.3.5 CONSERVATION UNITS AND TRADITIONAL POPULATIONS

The project's area is not located within Conservation Units (Parks, Environmental Protection Areas, and Forest Reserves) or in its buffer zones. Indigenous lands and settlement projects in areas of direct influence of the project were also not identified.

20.3.6 HISTORICAL AND ARCHAEOLOGICAL HERITAGE

No significant archaeological or cultural heritage resources were identified within the area of influence of the project in the surveys carried out in 2012. The final report, concluded and filed with IPHAN (Institute of National Historical and Artistic Heritage) in October 2012, was preceded by a population awareness-raising program with theoretical and practical workshops aimed at sharing the results of the survey with the student community of Almas and promoting the management of the region's historical and cultural resources.

20.4 ENVIRONEMENTAL 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 the CONAMA Resolution No. 237/97 lists the activities and undertakings that use environmental resources, effectively or potentially polluting, that are subject to Environmental Permitting. Federal Law No. 6,938 / 81 and CONAMA Resolution No. 237/97 defines three (3) types of environmental license, namely:

- Preliminary License (Licença Prévia – LP)

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 (Licença de Instalação – LI)

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 (Licença de Operação – LO)

Authorizes the operation of the enterprise, after verifying the effective fulfillment of previous licenses, environmental control measures and conditions determined for the operation.

Federal Law No. 6,938/81 assigned 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 Almas Gold Project, located in the state of Tocantins, the Instituto Natureza do Tocantins - NATURATINS is responsible for Environmental Permitting.

NATURATINS' environmental policy was established through Decree No.1,024/90, which determines that the Environmental State Council (COEMA) is responsible for the elaboration of environmental regulations and procedures (including those related to permitting), monitoring of compliance with environmental policy and settlement of major environmental issues.

Through the COEMA-TO Resolution No. 07/2005, the State of Tocantins instituted a simplified permitting procedure for projects considered to have less potential for negative environmental impact and projects to be resumed. According to this permitting procedure, the three stages (PL, IL and OL) can be merged into two, or only one, at the discretion of the Regulatory Environmental Agency. In this case, the environmental report required to instruct the permitting process and to replace the Environmental Impact Study-Environmental Impact Report (EIS/EIR) is the Environmental Assessment (EA), a shorter type of EIS, which was recommended for the simplified permitting of the Paiol Mine.

In addition to these licenses, others are required during the process, such as Water Use Grant and Vegetation Clearing Permit. Table 20-2 below shows the period of validity of the licenses, according to COEMA-TO Resolution No. 07/2005.

Table 20-2 Permits Terms

PERMITS	PERIOD OF VALIDITY (YEARS)		
	PL	IL	OL
Main Licenses	2	2	4
Vegetation Clearing Permit	2		
Water Use Grant	3- 5		
Environmental Permits for Fish and Wildlife Management and Rescue, temporary works, etc.	1		

The Preliminary and Installation Licenses, Environmental Permits and Vegetation Clearing Permits may have their validity terms extended or be renewed for a shorter or equal period as long as the application with NATURATINS is made at least 30 days before expiration (Article 16, paragraph 3 of COEMA Resolution No. 07/2005)

Water Use Grants can be renewed several times, as long as the renewal application with NATURATINS is made at least 45 days before expiration.

20.4.2 STATUS OF ENVIRONMENTAL PERMITTING

As already mentioned, the Almas Gold Project is comprised three areas at different stages of development: the Paiol Mine, and the Vira-Saia and Cata Funda deposits.

The Paiol Mine, the main mineral deposit, is at a more advanced stage in terms of Environmental Permitting and since 2011, it has obtained several permits. Currently, Aura is requesting the renewal of some those already expired licenses and applying for new ones. Table 20-3 shows the status of Licenses.

Table 20-3 Environmental Permitting Status

PERMITS	VALIDITY	STATUS	EXPECTED ISSUING DEADLINE (**)
Installation License-LI No. 5.437-2011	Expired	Replaced by LI No. 297-2017	
Preliminary License-LP No.286-2017	Expired	No need for renewal	
Installation License-LI No. 297-2017	As there was no manifestation by NATURATINS, the License is considered extended and valid (*)	Undergoing renewal analysis by NATURATINS	1st quarter 2021
Vegetation Clearing Permit (VCP)		New VCP has been required	1st quarter 2021
Management and Rescue Fish and Wildlife – AMAS No.7188-2020 –Issued on 09/04/20	09/04/2021	Can be renewed annually	
Water Use Grant - DUI No.7574-2020 (Human Consumption) - Issued on 09/18/2020	09/18/2025	Can be renewed several times after a period of 5 years	
Water Pit Use Grant	Expired	Grant Renewal requested - under review	1st quarter 2021
Pit Dewatering Grant	New	Grant requested – under review	1st quarter 2021
Pit Dewatering Pumping System Permit	New	Permit requested in December/2020 – under review	1st quarter 2021
Make Up Water Grant (Manuel Alves River)	New	Grant requested in December /2020- under review	1st quarter 2021
Make Up Water pumping system and Water Pipeline Permit		To be requested in February /2020	Between 1st and 2nd quarter/2021
Preliminary License-LP and Installation License-LI – Power Line 138KV (***)	New	To be requested in May 2021	1st quarter 2022

Notes:
*: According to Art. 41, sole paragraph of COEMA Resolution No. 07/2005, once ALL the documentation required by NATURATINS has been filed, within the period of 30 days before expiration, the expired environmental permit will be extended until formal manifestation by the environmental agency
**: Estimated deadlines, may be changed due to new NATURATINS requirements
***: ENERGISA, Energy Concessionaire of the state of TO, is responsible for the design, permitting and implementation of the Power Line

Installation License LI No. 297-2017 includes the pre-stripping of the Paiol Mine, removal and construction of low grade and waste rock piles, earthworks, construction of the mill, ancillary facilities and access. The tailings storage facility project is being analyzed by NATURATINS in conjunction with the renewal of the Installation License to enable the issuance of a Single Installation License.

In the same manner of Paiol Mine, the Vira-Saia and Cata Funda mineral deposits will be subjected to simplified environmental permitting. For that, an Environmental Assessment (EA) will be carried out to request the Preliminary (LP) and Installation (LI) Licenses. Additional Permits such as Water Use Grants, Vegetation Clearing Permit, Fish and wildlife Management and Rescue Permit, among others, will be requested throughout the permitting process. The tentative schedule is set to start the EA at the beginning of 2021, to present the final report and apply for Preliminary and Installation Licenses with NATURATINS in 4th quarter of 2021 and the forecast for obtaining Licenses between the 3rd and 4th quarter of 2022. The expected start of operation of these deposits, according to the mine's production plan, is from 2025 (Cata Funda) and 2028 (Vira-Saia), which can be anticipated if necessary.

The Energy Company of Tocantins – ENERGISA, will be in charge of the design, environmental permitting and implementation of the Power Line. It is anticipated that an Environmental Studies will be prepared, in accordance with NATURATINS' Terms of Reference, for the application of the Preliminary and Installation Licenses. Since ENERGISA is the concessionaire responsible for all energy projects in the state, it is expected that the permitting process will be faster.

20.5 ENVIRONMENTAL AND SOCIAL STUDIES

As already mentioned, most of the environmental and social studies were carried out between 2010 and 2012, nonetheless, other studies and tests were conducted in 2020 for the resumption of the project and will continue in 2021. The following is a summary of the main studies:

20.5.1 ENVIRONMENTAL ASSESSMENT (EA)

Through the Environmental Assessment (EA), carried out in 2011, to start the simplified permitting of the Paiol Mine, the potential environmental and social impacts in the Project's area of influence were identified and evaluated, based on primary and also secondary surveys involving air quality, soil, climate, geology, geomorphology and geochemistry, surface and underground waters, vegetation, wildlife, social, cultural and archaeological aspects. Mitigating measures, plans and programs were proposed for the management of impacts. From the approval of the EA and Environmental and Social Plans, NATURATINS issued Preliminary License No. 286-2017 and Installation License No. 297-2017.

20.5.2 FISH SURVEY IN THE OPEN PIT PAIOL MINE AND THE SURROUNDING STREAMS

In 2010, aiming at the future dewatering of the open pit Paiol Mine, a fish survey was carried out by ICOM Engenharia, from Porto Nacional (TO), in the pit and streams around it. Only the *Astyanax goyacensis*, popularly called “lambari”, was registered in the open pit. This species was captured in all points located in the streams surrounding the pit, being considered mainly responsible for the similarity between these locations (streams and open pit).

According to the study, the similarity in the composition of the Fishes in the pit and the streams in its surroundings, as well as the trophic characteristics of the only species recorded in this environment, may indicate that the emptying of the open pit for one of these nearby water bodies does not tend to represent a relevant impact for the local fishes.

20.5.3 CHARACTERIZATION OF THE WATER QUALITY OF OPEN PIT PAIOL MINE AND SURROUNDING STREAMS

Several campaigns were carried out to characterize the water quality of the open pit Paiol Mine and the adjacent water bodies. Between 2010 and 2013, water samples were collected and analyzed by the laboratories of the Federal University of Tocantins and CONÁGUA AMBIENTAL, located, respectively, in Palmas (TO) and Goiânia (GO). Recently, in April 2020, and under the supervision of the CONAGUA AMBIENTAL Laboratory, water collection was resumed, covering around 20 points. Another campaign is planned for January, 2021. The water samples were analyzed for metals (including mercury and arsenic), Total Cyanide, Free Cyanide and WAD Cyanide, pH, Total Dissolved Solids, Total Sulphate and Sulphides, Total and Termotolerant Coliforms, Alkalinity, etc. Cyanide and Mercury were not detected.

Comparing the results of the analysis of water in the open pit and surrounding streams with the maximum values allowed by the

CONAMA Resolution No. 357/2005 (Brazilian Environmental Norm that Establishes the Classification of Water Bodies and the Water Quality Conditions and Standards within Brazilian Territory), the waters are considered to be of good quality.

20.5.4 HYDROLOGY AND HYDROGEOLOGY

In 2012, CLAM Engenharia was hired to make conceptual and computational hydrogeological models for the development of the Paiol Mine and Cata Funda and Vira-Saia deposits, with the mapping of permanent and ephemeral streams and springs in the three deposits. In 2020, based on the knowledge obtained in 2012, the company was hired once more to continue the studies and monitoring of water resources (surface and groundwater).

20.5.5 GEOCHEMISTRY

According to the 2011 "Technical Visit Report - Hydro Geochemical and Groundwater Review" report, preliminary field analyzes did not indicate the potential for acid mine drainage and metal leaching associated with the Paiol Mine and waste rock material. Seeing that the field analysis does not replace laboratory tests, it was recommended that geochemical studies be carried out.

In order to assess the potential for Acid Mine Drainage, representative composed samples of the types of ore (7 samples from the Paiol Mine, Cata Funda, Vira-Saia and spent heap leach pad), waste rock (18 samples) and tailings (2 samples) were collected to carry out prediction tests of acid mine drainage - MABA – Modify Acid Basic Accounting and NAG – Net Acid Generation.

The tests were conducted by the SGS-Geosol laboratory in Belo Horizonte. The interpretation and evaluation of the results were carried out by an independent consultant and is included in the technical report "Results of the Geochemical Characterization of the Geological Materials of the Almas Gold Project - Almas-TO". The studies were carried out between April and July, 2020.

The results of all ore samples, waste rock and tailings tested, showed NPR = NP / AP> 2 and NAGpH> 4.5. The combined analysis of these results indicates a low probability of occurrence of acid drainage generation, regardless of the type of material evaluated, especially due to the high content of carbonates found in the samples. NPR stands for Neutralization Potential Ratio; NP for Neutralization potential; AP for Acid Potential and NAG for Net Acid Generation.

20.5.6 TAILINGS STORAGE FACILITY - STUDIES

Throughout the environmental permitting process of the Paiol Mine, NATURATINS' technical staff were uncertain about whether it was necessary to line the tailings storage facility reservoir with HDPE. Thus, with the resumption of the Installation License LI No. 297-2017 renewal process, new studies were conducted such as Detox tests (SO_2/air), tailings characterization and acid mine drainage prediction tests to demonstrate that the design of the tailings storage facility reservoir without HDPE liner is safe.

Among the studies, new cyanide neutralization tests (DETOX) stand out, now with the adoption of the SO_2/air method. This method is recommended for both pulp and solution and is more efficient, since it is able to remove WAD cyanide (Weak Acid Dissociable) and free cyanide to concentrations below 1 mg/l. The tests were carried out by the Testwork Desenvolvimento de Produção Ltda Laboratory and the chemical analyzes, conducted by SGS-Geosol Laboratórios Ltda, between May and July, 2020, in Belo Horizonte, MG.

The results indicated high treatment efficiency with final cyanide concentrations (Free and WAD) lower than the Maximum Allowed Value of 0.2 mg/L of free CN, established by CONAMA Resolution No. 430/2011 - Brazilian Norm for Effluent Discharge Standards.

Regarding the assessment of acidity potential of the tailings samples (MABA – Modify Acid-Base Accounting and NAG – Net Acid Generation tests), the results of the two tailings samples tested, presented NPR = NP/ AP> 2 and NAGpH> 4.5. The combined analysis of these results indicates a low probability of occurrence of acid mine drainage generation, mainly due to the high content of carbonates found in the samples, which implies a greater neutralization potential than the acid generation potential.

The tailings samples were also submitted to solid waste characterization tests according to the Brazilian Technical Standard ABNT 10004, 10005 (leaching) and 10006 (solubilization). The results did not show any of the constituents of the samples, leached or solubilized concentrations higher than the Maximum Allowed Value by the Standard and, for this reason, the tailings are classified as Class IIB, Non-Hazardous and Inert.

The joint evaluation of the results of these studies and of the characteristics of the local geology, of limited permeability and infiltration condition (low hydraulic conductivities between 10-5 and 10-6 cm/s), resulted in the adoption of the tailings storage facility reservoir without HDPE liner and a zero discharge design, resulting in zero effluents.

20.5.7 SOCIOECONOMIC ASPECTS

Between 2010 and 2011, socioeconomic diagnoses and assessments, sensitivity analysis, stakeholder mapping and a social communication plan were carried out by the company Integratio Mediação Social e Sustentabilidade, from Belo Horizonte.

In 2011, Conestoga Rovers e Associados (CRA) expanded the database and the assessment of socioeconomic impacts for the Environmental Assessment (EA) of the Paiol Mine aiming at environmental permitting.

To establish dialogue and strengthen the relationship between the company and the local community, 10 public meetings were held with the participation of 260 people from the municipalities of Almas, Porto Alegre do Tocantins and Dianópolis. The target audience were the population and other concerned parties such as representatives of the judiciary branch, government authorities and trade associations. The agenda for these meetings was to present the project and provide information about environmental permitting.

Among the main demands and desires of the population are the creation of jobs and income, qualification of workforce and local suppliers, opportunity for services, environmental and social impacts, as well as a schedule for the implementation of the project and increased traffic in the town of Almas.

The update on the socioeconomic diagnosis, mapping of stakeholders, social communication plan, in addition to a new perception survey and social management plan is projected for the beginning of 2021. Contact with local authorities to define partnerships has already been resumed and new public meetings will be held from January onwards, 2021.

20.5.8 HISTORICAL AND ARCHAEOLOGICAL HERITAGE

The Almas Gold Project is located in a region known for its archaeological and historical resources, since the area has been the object of gold exploration since the 18th century.

However, after a field survey (of an area of around 400 ha) carried out between July-August 2012, in the areas directly affected and directly influenced by the Paiol Mine project, no resources of relevant archaeological and historical interest were found.

All studies, documents and plans for the rescue and management of archaeological and historical heritage, required by NATURATINS and IPHAN (National Institute of Historical and Artistic Heritage), have been completed and filed within these organizations. IPHAN issued a Technical Opinion in favor of issuing Installation License LI No. 297-2017.

20.6 MAIN IMPACTS

The Environmental Assessment (EA) identified and evaluated the potential environmental, social and economic impacts in the area of influence of the Paiol Mine, as well as the auxiliary infrastructure (mill plant, tailings storage facility, pipelines, accesses, etc.) needed for the resumption of the project.

As already mentioned, the Vira-Saia and Cata Funda mineral deposits will be the subject of another Environmental Assessment (EA) for simplified permitting. It is anticipated that the potential impacts for these two areas will be similar and inferior to the Paiol Mine, since they will only have the open pit mine, waste rock and low-grade piles and minimal infrastructure for the operation. The ore from these two mines will be transported by trucks to be processed at the mill at the Paiol Mine.

Among the main potential impacts (negative and positive) identified in the Paiol Mine EA, the following stand out: alteration of the hydrological and hydrogeological dynamics, alteration of wildlife and fish habitats, alteration in the Cerrado vegetation, generation of expectations for employment, pressure on infrastructure and urban equipment and boosting the region's economy.

20.7 ENVIRONMENTAL AND SOCIAL PLANS AND PROGRAMS

Environmental and Social Plans and Programs were proposed in the Environmental Assessment (EA) of the Paiol Mine, in accordance with the Term of Reference issued by NATURATINS. These Plans present strategies for controlling and monitoring the environmental parameters of the areas of influence of the project, aiming at monitoring the actions to be developed in the implementation, operation and decommissioning phases, in addition to assessing the environmental quality of the region, from the implementation of the project. It should be noted that these same Plans and Programs will be extended to Vira-Saia and Cata Funda, in the future, for the management of impacts. Table 20-4 below shows the Environmental and Social Plans and Programs.

Some programs have already been executed, such as the Historical and Archaeological Heritage Management Program, which, with the positive Technical Opinion of IPHAN (National Historical and Cultural Heritage Institute), allowed NATURATINS to issue the Installation License LI No. 297-2017.

Other programs, such as Social Communication and Water Quality Monitoring, were also started, but were interrupted due to the suspension of the Project by Rio Novo Mineração. With the resumption of the Almas Gold Project by Aura, all plans and programs will be executed according to the project's implementation and operation schedule.

Table 20-4 Environmental and Social Plans and Programs

ENVIRONMENTAL AND SOCIAL PLANS AND PROGRAMS			
MANAGEMENT PLANS	**PHYSICAL PROGRAMS CONTROL AND MONITORING**	**BIOTIC PROGRAMS CONTROL, MONITORING AND RESCUE**	**SOCIOECONOMIC PROGRAMS**
Environmental Management	Air Quality	Vegetation Clearing	Social Communication Program
Construction Management	Noise Emissions and Vibrations	Vegetation Rescue and Management (especially epiphytes)	Environmental Education Program
Management of solid waste and liquid effluents	Geotechnical Monitoring of the waste rock pile, tailings storage facility and erosion processes	Fish and Wildlife Rescue and Management	Workforce Qualification and Development of Local Suppliers
Reclamation of Degraded Areas and Mine Closure	Surface and Groundwater Quality	Flora Conservation and Monitoring	Support to the Municipality
Controlled Fire	Hydrological and Hydrogeological Monitoring	Monitoring and Conservation of Wildlife and Aquatic Biota	Socioeconomic Monitoring
Controlled Chemicals Management			Management of Historical and Archaeological Heritage

20.8 RECLAMATION AND CLOSURE

The Reclamation Plan for Degraded Areas (RPDA) and Closure Plan, presented in the Environmental Assessment (EA) of the Paiol Mine, establish guidelines for planning the reclamation and closure of the Paiol Mine. The RPDA establishes general rehabilitation measures to be taken during and after mining, to ensure progressive, and eventually, rehabilitation from the site to conditions similar to those presented prior to mining. The Closure Plan will be updated as the project progresses. In addition to the revegetation efforts, other important rehabilitation measures to be implemented include recontouring the topography of the land, drainage and slope stabilization.

The final reclamation and closure project will include:

- Mill and ancillary facilities: dismantling and removal of infrastructure and foundations, recontouring of the land surface and drainage to allow rainwater runoff; coverage with organic soil and revegetation;
- Waste rock pile: slope reconformation, drainage and revegetation;
- Tailings Storage Facility: neutralization and pumping of the remaining liquid phase; installation of a rainwater drainage system, coverage with inert material, if necessary, and coverage with organic soil and revegetation;
- Access and water pipeline: disassembly, removal and recovery of areas along the water supply lines;

- Pumping system for raw water collection and pumps: removal and demolition;
- Explosive magazines area and ancillary installations: demolition, removal and revegetation;
- Power transmission line: removal of towers, substations and lines, including all foundations, recontouring of the land surface and revegetation;
- Recovery of areas degraded by mining activity within the limits of the mineral exploration 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 reclaimed areas, as well as surface and underground water, in order to identify any possible changes or contamination that may affect the environment.

The costs for reclamation and mine closure, encompassing all of the actions described above, is estimated in US$ 5.5M and concerns the three mineral deposits, Mina Paiol, Vira-Saia and Cata Funda.

20.9 CONCLUSIONS

The Paiol Mine is at an advanced stage of permitting and most of the Permits have already been required and are under analysis by NATURATINS. The Environmental Assessment (EA) and other studies for the permitting are in compliance with Federal and State Standards and Regulations.

The Environmental Assessment (EA) for the permitting of the Cata Funda and Vira-Saia deposits will start in 2021 and, since they will only be implemented in 2025 and 2027, there is time to carry out all the baseline and environmental permitting studies for the expected implementation deadlines.

21 CAPITAL AND OPERATING COSTS

Unless stated otherwise, all costs presented in this section are in United States dollars (USD).

21.1 CAPITAL COSTS

The estimate conforms to Class 3 guidelines for a feasibility study level estimate with an accuracy according to the Association for the Advancement of Cost Engineering International (AACE International). All values presented are in United States dollars (USD).

Table 21-1 provides a summary of the estimate for overall initial capital cost. The estimate includes costs for mining, site preparation, process plant, dams, first fills, buildings, roadworks and off-site infrastructure.

Table 21-1 Summary of Capital Costs (USD)

DESCRIPTION	TOTAL Cost w/taxes (USD)
DIRECT COSTS	**52,619,475**
EQUIPMENT	**9,502,645**
Mechanical Equipment	8,616,388
Electrical Equipment	164,623
Instrumentation and Automation	721,634
PACKAGES	**9,940,141**
Elution Package	3,956,306
Main Substation	1,972,947
Secondary Substation	3,535,441
Telecommunications	475,447
MATERIALS	**5,703,495**
Plate Work	1,306,458
Steel Structure	912,441
Piping	2,248,740
Electrical, Instrument, Automation and Telecom	1,235,855
CONSTRUCTION AND ERECTION	**18,610,725**
EM Erection	8,917,316

DESCRIPTION	TOTAL Cost w/taxes (USD)
Earthworks	1,821,675
Civil Works	4,124,491
Ancillary Facilities	3,747,242
AURA INFORMATION	**8,862,470**
138 kV Power Supply	1,817,302
Mine	3,997,599
Tailings dam	2,446,210
Laboratory	601,358
INDIRECTS COSTS	**15,518,370**
EPCM	4,519,884
Supervision by vendor	344,729
Spare Parts and Special Tools	517,093
Owner Costs	7,392,059
Freight	829,061
First Fill	534,432
Indirect Field Construction	1,068,863
Engineering, Construction and Civil Responsibility risk insurance	312,249
Contingency	4,636,670
TOTAL	72,774,515

21.2 BASIS OF CAPITAL COST ESTIMATE

21.2.1 Mining Capital Cost

Mining will be undertaken by a contract mining company, with the resulting reduction in initial capital costs, Mining capital costs are shown in Table 21-2. The capital costs cover the period 2021 and 2022.

Table 21-2 Mining Capital Costs (USD)

DESCRIPTION	TOTAL Cost w/ taxes (USD)
Soil Removal	391,400
Pre-Stripping - Saprolite	945,200
Pre-Stripping - Fresh Ore	612,400
Pre-Production Ore Mining	1,112,300
Mobilization	116,400
Pumping Stations	46,600
Pit Dewatering	41,900
Explosives Storage	87,300
Emulsion Plant	174,600
Site Preparation	126,100
Pre Stripping Tree Removal	29,100
Waste Stockpile	62,100
Low Grade Stockpile	23,300
TOTAL	3,768,700

21.2.2 PROCESS PLANT COSTS

The estimate is based on quotes both for equipment and for material take-offs for bulk items, such as piping, earthworks and concrete. No allowance is included for contracts based on a cost-plus or accelerated schedule. The erection of tankage, structural, mechanical, piping, electrical, instrumentation, and civil works will be performed by experienced contractors using local labor.

Direct costs are based on quantities and include all permanent equipment, bulk materials, subcontracts, labor, contractor indirect, and growth associated with the physical construction of the facilities.

21.2.2.1 EQUIPMENT COSTS

A minimum of three quotes were sourced per mechanical equipment package, with the exception of some minor equipment that was sourced from Ausenco's database. The budget quotes cover over 85% of the overall mechanical equipment supply cost.

21.2.2.2 COMMODITY TAKE-OFFS

The gold plant and associated facility estimates were prepared on a commodity basis (i.e., divided into earthworks, concrete, structural, piping, etc.) and reported by area (i.e., crushing, milling, etc.). For each commodity, a minimum of three quotes were sourced for pricing.

The majority of quantities were based on a first-principles approach, such as mechanical, platework, overland piping, electrical, earthworks, concrete and steel. Minor costs, such as small-bore piping, piping internal to each building and instrumentation were factored.

21.2.2.3 INSTALLATION COST

Installation and erection were bid in the market as per electromechanical assembly package. This quote included all direct and indirect expenses of the contractor required to perform this service. The cost for instrument assembly/installation was factored.

This cost was bid from three different specialized vendors. They include:

- temporary construction area preparation
- team hiring and mobilization
- work health and safety expenses
- consumable expenses
- meals and lodging for their team
- daily transportation of team members to and from the site
- transportation of resources required to perform the service
- supply of personal protective equipment (PPE) and tools
- rental of mobile cranes, compressor, welding machine, scaffolding and any other equipment required
- rental of specialty cranes, to assemble the mill, crusher, screens, tanks and transformers, where applicable
- insurances of any kind required, including taxes
- BDI (Budget Difference Income), profit
- Demobilization of construction area and team
- Temporary electrical installations to feed the work front under their responsibility from main Substation
- structural, mechanical and piping

Indirect construction costs include all temporary accesses, fences and installations, area signaling, warehousing and materials receipt, temporary power, construction services (surveying, safety, medical assistance, scaffolding, concrete testing, etc.), construction vehicles, consumables, sanitation, medical exams, security).

Indirect expenses were also included in the quotes received for electromechanical assembly.

The electromechanical assembly provider will be responsible for the costs of construction, operation and maintenance of their temporary facilities. The installations will be sized sufficient to the required team for the service.

21.2.2.4 TAXES

The taxes included in vendor proposals will be considered, in accordance with current Brazilian tax laws. For equipment priced from Ausenco's database, appropriate taxes were added. The 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
- II (Imposto de Importação – Importation Taxes) and applicable fees

21.2.2.5 GROWTH ALLOWANCE

Growth allowance is a percentage applied to quantities, rates and costs to allow for design development and specification changes. This addition is a subjective quantity added to the material take-offs based on the level of engineering completed using benchmarking with historical experience of the expected values. The growth allowance factors used on the amounts are shown in Table 21-3.

Table 21-3 Growth Allowance Definitions

Level of Engineering	Growth (%)	Description
90-100% Designed Definition	8,00	All disciplines between 90 and 100% defined
75-90% Engineering Definition	10,00	All disciplines between 75 and 90% defined
60-75% Engineering Definition	12,00	All disciplines between 60 and 75% defined
50-60% Engineering Definition	15,00	All disciplines between 50 and 60% defined
Equipment	0,00	Supply of mechanical and electrical equipment, automation systems and other systems

21.2.2.6 INDIRECT COSTS

Indirect costs include all costs associated with implementation of the plant and incurred by the Owner, engineer or consultants in the project design, procurement, construction, and commissioning. These costs typically include:

- EPCM/commissioning, start-up and pre-operation
- Field assembly supervision (by vendor representatives)
- Reagent first fills (factored)
- Spare parts (factored)
- Freight – factored only when value was not received in item quote
- Engineering risk insurance (factored)
- Owner's costs

21.2.2.7 TAX BENEFITS

Aura Minerals hired Ernst & Young to carry out a general assessment of the tax benefits potentially applicable to the Almas Gold Project.

For CapEx of the Project, only the tax benefits that can be automatically applied and do not depend on the approval of the Brazilian Tax Authorities, such as the "Receita Federal" and "Secretaria de Fazenda do Estado do Tocantins", were considered. For the tax benefits that depend on the approval of the Brazilian Tax Authorities, they are considered as possible opportunities for Upsite and Aura Minerals will request them in order to capture these benefits.

For CapEx, the following benefit does not need to be requested to Tax Authorities and were considered:

- **ICMS Agreement 52/91** (it allows taxpayers to reduce the State VAT “ICMS” tax basis to 8.8% for determined products listed in the Agreement by their HTS Code):

  The potential tax saving represents BRL 544,077THSD for ICMS and BRL 1,534,595M for ICMS Differential Rate “DIFAL”.

On the other hand, as mentioned above, Aura Minerals will file a request to the "Receita Federal" and the "Secretaria de Fazenda do Estado do Tocantins" in order to be granted the following benefits. These tax benefits below were not considered in the Project's CapEx and may be upsides.

- **RECAP** (Special Regime that allows taxpayers to exempt Social Contributions (PIS/COFINS) levied upon the import and internal acquisition of determined fixed assets listed in the Federal Decree): The potential tax saving represents BRL 1,135,175M for PIS and BRL 5,225,422M for COFINS.
- **PRÓ-INDÚSTRIA** (Special Regime that allows taxpayers to exempt ICMS levied upon imports and acquisitions within State of Tocantins of fixed assets, as well as to exempt ICMS DIFAL levied upon interstate acquisitions of fixed assets): The potential tax saving represents BRL 3,978,474M for ICMS and BRL 5,325,837M for ICMS Differential Rate “DIFAL”.
- **TAX SERVICE “ISSQN”** (it depends on negotiation with the Municipality and it was considered a Tax Service “ISSQN” reduction from 5% to 2% levied upon construction services): The potential tax saving represents BRL 2,867,417M for ISSQN.

21.2.2.8 AREA 301 – GENERAL INFRASTRUCTURE

General infrastructure, as described in section 18 of this report, was estimated at the level of basic engineering by Ausenco, based on Q4 2020 unit rates from the Aura operation in the state of Mato Grosso, Brazil, at the Ernesto Pau a Pique Mine, Apoena.

21.2.2.9 AREA 330 – POWER SUPPLY, SITE RETICULATION & COMMUNICATIONS

The 138 kV Transmission line was quoted by Energisa, a local concessionaire that will be responsible for the construction on turnkey basis.

21.2.2.10 TAILINGS MANAGEMENT FACILITY Phase 1 tailings embankment quantities were estimated at the detailed engineering level by Geohydrotech. based on Q4 2020 unit rates from the Aura operation in the state of Mato Grosso, Brazil, at the Ernesto Pau a Pique Mine, Apoena.

21.3 OPERATING COSTS

The operating cost estimate is presented in Q4 2020 United States dollars (USD). The estimate includes mining, processing, general and administration (G&A), and accommodations costs. Mining operating cost is the LOM average annual expenditure. Details of annual expenditure variations are described in section 21.3.2.

The operating cost estimates for the life of mine are provided in Table 21-4.

Table 21-4 Operating Costs

DESCRIPTION	YEARLY COST (M$USD)	YEARLY COST (USD/T)
Mining	8.86	7.31
Process	14.88	11.44
General and Administration	2.50	1.93
Total	26.24	20.68

21.3.1 BASIS OF OPERATING COSTS

Common to all operating cost estimates are the following assumptions:

- Cost estimates are based on Q4 2020 pricing without allowances for inflation
- The estimate was done in Brazilian reais (BRL), then converted to United States dollars (USD) for this report
- For material not sourced in BRL, the exchange rate used was $1 USD = $5.155 BRL
- Fuel costs and taxes were established by Aura
- The annual power costs were calculated adding consumption costs and demand costs. For consumption, unit prices of $0.070/kWh for normal operating hours and $0.112/kWh for peak hours were used. The average split between normal and peak hours considered was 87.5% normal hours and 12.5% peak hours. Aura established this value with Energisa (local electrical provider) and supplied it to Ausenco. Demand costs considered were $2.31/kWh for normal operating hours and $5.69/kWh for peak hours, with the same distribution as consumption.
- Labour is assumed to come mostly from Tocantins, locally, with no need for a camp installation
- An analysis of fiscal benefits was performed, in order to reduce taxes applied to the different opex categories (power and consumables)

21.3.1.1 BASIS OF MINING OPERATING COSTS

The mining sequencing considered to feed 1,3 Mt per year in the concentration plant, maximize the NPV according to the highest safety standards and in accordance with best practices to minimize the Environmental impacts. As the waste / ore strip ratio varies by year, the volume of waste rock will vary by year to assure a constant ore feed tonnage in the concentration plant

The total costs are presented considering the outsourced mining operation concept. The calculation was done to achieve the costs per ton for each lithology, ore or waste. The total cost achieved was a result of summing the separate mining unit operation as follows:

- Blasthole Drilling.
- Explosive and related blasting accessories specifically applied for ore or waste rock,
- Loading and Transport the ROM, Dozer for spreading the material, prepare and maintain all mine infrastructure. Either for ore, or for waste rock, the variations in transport costs by distance longer than 2 km was considered
- Grade Engineering: Drilling: Geology, Mining Plan, Grade Control.
- The stockpiles Low-Grade ore resumption cost.
- The Heap Leach pad ore resumption cost.

21.3.1.2 BASIS OF PROCESS OPERATING COSTS

The following was used to determine the project's LOM process operating costs in agreement with the cost definition and estimate methodologies outlined below. This basis considers the development of a facility capable of processing 3,560 t/d of ore.

Assumptions made in developing the process operating cost estimate are listed below:

- Mill production is set at an average of 1.3 Mt annually
- Process plant operating costs are calculated based on labour, power consumption, and process and maintenance consumables.
- Off-site gold refining, insurance, and transportation costs are excluded

- Operating costs incurred during the pre-production period have been capitalised within Aura.
- General and administration (G&A) costs were defined by Aura
- Consumables costs are based on data from Aura
- Maintenance costs are straight-lined through the life of mine (no reduction in initial years for reduced wear)
- Grinding media consumption rates have been estimated based on the ore characteristics
- Reagent consumption rates have been estimated based on the metallurgical testwork results at a nominal basis
- Mobile equipment cost provides for vehicle lease

21.3.2 MINING OPERATING COSTS

The costs per tonne (USD/t) for ore and waste, by origin, lithology and for distance over than 2.0 km; are presented in Table 21-5. The mining yearly total costs (M$USD) and yearly mining costs per tonne (USD/t) are presented in the table 21-6.

Table 21-5 ROM Operating Costs per ton. by Pit and Lithology

<table>
<tr><th>ORE /WASTE</th><th>ORIGIN</th><th colspan="2">LITHOLOGY</th><th>USD/t</th></tr>
<tr><td rowspan="11">Ore</td><td rowspan="3">Paiol</td><td colspan="2">Soil</td><td>0,76</td></tr>
<tr><td colspan="2">Saprolite</td><td>1,03</td></tr>
<tr><td colspan="2">Fresh Rock</td><td>1,45</td></tr>
<tr><td rowspan="3">Cata Funda</td><td colspan="2">Soil</td><td>1.53</td></tr>
<tr><td colspan="2">Saprolite</td><td>1.80</td></tr>
<tr><td colspan="2">Fresh Rock</td><td>2.22</td></tr>
<tr><td rowspan="3">Vira Saia</td><td colspan="2">Soil</td><td>1,03</td></tr>
<tr><td colspan="2">Saprolite</td><td>1.30</td></tr>
<tr><td colspan="2">Fresh Rock</td><td>1.72</td></tr>
<tr><td>Heap Leach Pad</td><td colspan="2">Fresh Rock (Pile resumption)</td><td>0.62</td></tr>
<tr><td>Low-Grade</td><td colspan="2">Fresh Rock (Pile resumption)</td><td>0.69</td></tr>
<tr><td rowspan="3">Waste</td><td rowspan="3">All Pits</td><td colspan="2">Soil</td><td>0,64</td></tr>
<tr><td colspan="2">Saprolite</td><td>0.82</td></tr>
<tr><td colspan="2">Fresh Rock</td><td>1.06</td></tr>
<tr><td rowspan="6">Ore and Waste</td><td rowspan="6">All Pits</td><td colspan="3">All Lithologies</td></tr>
<tr><td rowspan="5">Costs variation by distance</td><td>Up to 2,000m</td><td>-</td></tr>
<tr><td>2,000 to 2,500m</td><td>0.024</td></tr>
<tr><td>2,500 to 3,000m</td><td>0.048</td></tr>
<tr><td>3,000 to3,500m</td><td>0.073</td></tr>
<tr><td>3,500 to 4,000m</td><td>0.097</td></tr>
</table>

Table 21-6 Annual Mining Operating Costs

YEAR	YEARLY COST (M$USD)	YEARLY COST (USD/t)
2022	2,81	5,3
2023	12.23	9.26
2024	11.15	8.45
2025	15.76	11.91
2026	15.75	12.11
2027	17.75	13.66
2028	20.19	15.53
2029	22.15	17.04
2030	13.08	10.06
2031	8.29	6.38
2032	4.97	3.82
2033	5.28	4.06
2034	3.29	2.53
2035	0.90	0.70
2036	0.90	0.70
2037	0.90	0.70
2038	0.90	0.70
2039	0.45	1.02
Total	**159.53**	**7.31**

21.3.3 PROCESS OPERATING COSTS

The LOM process operating cost is $424 M over 17 years. A breakdown of this value and its unit costs is presented in Table 21-7.

Table 21-7 Average Annual Process Plant Operating Costs

DESCRIPTION	YEARLY COST (M$USD)	YEARLY COST (USD/t)
Labour	3.66	2.81
G&A	2.50	1.93
Laboratory	1.98	1.52
Access Maintenance	0.12	0.09
Mobile Equipment	0.72	0.55
Power	2.00	1.54
Reagents and Consumables	5.35	4.12
Maintenance	1.04	0.80
Water Treatment and Sewage Treatment Plants	0.00	0.00
Total	17.38	13.37

21.3.3.1 CONSUMABLES

Individual reagent consumption rates were estimated based on the metallurgical testwork results, Ausenco's in-house database and experience, industry practice, and peer-reviewed literature. Reagent pricing was provided by Aura. A detailed description of the reagents required for the process is provided in Chapter 17.

Other consumables (e.g., liners for the primary crusher, SAG mill, ball mill, and ball media for the mills) were estimated using:

- metallurgical testing results (abrasion)
- Ausenco's in-house calculation methods, including simulations
- vendor information
- forecast nominal power consumption based on the load list derived from the mechanical equipment list

Reagents and consumables represent approximately 31% of the total process operating cost at $4.12/t milled.

21.3.3.2 MAINTENANCE

Annual maintenance consumable costs were calculated based on a total installed mechanical capital cost by area using an average factor of 3.5%. The factor was applied to mechanical equipment, platework, and piping. This item also includes consumables for maintenance and fuel and lubricants. The total maintenance consumables operating cost is $0.80, which is equivalent to approximately 6% of the total process operating cost.

21.3.3.3 POWER

The processing power draw was based on the average power utilization of each motor on the electrical load list for the process plant and services. Power will be supplied by the Energisa grid to service the facilities at the site.

21.3.3.4 LABORATORY & ASSAYS

Operating costs associated with laboratory and assay activities were estimated according to the anticipated number of assays per day and per year, by Aura. Assay costs include environmental sampling and assaying. Assay costs associated with processing mine grade control samples or exploration samples are included in the mine operating costs. The laboratory and assays comprise approximately 11% of the total process operating cost.

21.3.3.5 MOBILE EQUIPMENT

Vehicle costs are based on rental allowances for the equipment required to operate the process plant.

21.3.3.6 LABOUR

Staffing was estimated by Aura. The labor costs incorporate requirements for plant operation, such as management, metallurgy, operations, maintenance, site services, assay lab, and contractor allowance. The total operational labor averages 94 employees.

Individual personnel were divided into their respective positions and classified as either day shift or 12-hour shift employees. Salaries were provided by Aura. Aura also confirmed the specific benefits and bonuses to be allocated. Thus, the rates were estimated as overall rates, including all burden costs.

An organizational manning plan outlining the labor requirement for the process plant is shown in Table 21-8. The G&A manning plan is summarized in Table 21-9.

Table 21-8 O&M Manning Plan

POSITION	PERSONNEL
Plant administration	12
Crushing	8
Grinding	8
Leach	8
Detox	8
Smelting	1
Laboratory	2
Plant maintenance	32
Cover personnel	5

Table 21-9 G&A Roster

POSITION	NUMBER PER SHIFT
Plant management	1
HSEC	15
Administrative	28

21.3.3.7 GENERAL & ADMINISTRATIVE

General and administrative (G&A) costs are expenses not directly related to the production of gold and include expenses not included in mining, processing, external refining, and transportation costs. These costs were developed with input from Aura, as well as Ausenco's in-house data on existing Brazilian operations.

A bottom-up approach was used to develop estimates for G&A costs over the life of mine. The G&A costs were determined as an average cost of $1.93 /t milled. These costs were assembled according to the following departmental cost reporting structure:

- General expenses (including travel, lodging and consultants)
- HSEC (including recruiting exams, first aid, personal protective equipment, uniforms, safety and signage, environmental monitoring, community costs, environmental license and software development and maintenance)
- Administrative costs (including communication, software licenses, insurance and legal fees)
- Contracts (including site security, cleaning services, meals at site, water, gas and vehicles)
- Bonuses (including educational incentives, training, mobilization and demobilization cost assistance, worker feeding program, transport, Christmas kit and bonuses)

Table 21-10 Annual Average G&A Operating Cost Summary

DESCRIPTION	YEARLY COST (USD)	COST PER TONNE (USD/t milled)
General expenses	144,052	0.11
HSEC	239,154	0.18
Bonuses	645,146	0.50
Administrative costs	473,059	0.36
Contracts	991,116	0.76
Others	12,027	0.01
Total	2,504,555	1.93

21.3.4 TAX BENEFITS

For OpEx, all benefits need to be requested from the Tax Authorities. However, as the operation will start in mid-2022, Aura considers these benefits in the OpEx, as there is enough time for application and approval by the Tax Authorities before the operation begins.

- **DRAWBACK SUSPENSION** (Special Regime that allows taxpayers to exempt Social Contributions (PIS/COFINS), Import Duty, Federal VAT (IPI) and Additional of Freight (AFRMM) levied upon import and internal acquisition of raw material):

  The tax saving considered represents BRL 493,626THSD for PIS, BRL 2,268,329M for COFINS, BRL 2,533,594 for Import Duty and BRL 175,952THSD for AFRMM.

- **PRÓ-INDÚSTRIA** (Special Regime that allows taxpayers to exempt ICMS levied upon imports and acquisitions within State of Tocantins of raw material and electric power):

  The tax saving considered represents BRL 6,432,383M for ICMS levied upon raw material and BRL 4,117,838M for ICMS levied upon electric power.

- **EPE** (Special Regime that allows taxpayers to exempt PIS/COFINS levied upon freight acquisition):

  The tax saving considered represents BRL 19,380THSD for PIS and BRL 89,320THSD for COFINS.

22 ECONOMIC ANALYSIS

22.1 INTRODUCTION

This Section (22.0, Economic Analysis) was compiled by GE21 using capital costs and operating expenditures (Section 21.0) and the production schedule (Section 16.0) with inputs by Aura, EY and Ausenco on the economic model.

This section describes the economic evaluation and financial metric methodologies to establish the financial model for the Almas Gold Project feasibility study.

The economic model has been developed by GE21 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, 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. • Execute the economic model and provide results to the project team to establish the optimal path forward.
Ausenco	• 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.
EY	• Analysis of tax incentives
EDEM	• Responsible for the Mine Plan and mine operating costs

The economic model was developed on an Excel spreadsheet-based on a financial model composed of several worksheets. Contributions from Ausenco, EY and EDEM were developed using independent spreadsheets and interfaced into Aura's economic model.

All currency in this Section is provided in United States Dollars ("US$"), unless otherwise indicated. The exchange rate used is US$1.00 = R$5.155.

Table 22-2 and Table 22-3 present the financial model main indicators and Table 22-4 presents the summary results of the financial model that will be detailed in the following topics of this chapter.

Table 22-2 Main Indicators of the Financial Model

DESCRIPTION	UNIT	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
Exchange rate	**US$/US$**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**
Plant Feed	t x 1,000	0	529	1,320	1,320	1,323	1,300	1,300	1,300	1,300	1,300
Production / Sales Volume	toz	0	20,351	53,579	53,397	49,968	45,929	43,806	43,292	47,222	43,043
Gold Price	US$/toz	0	1,558	1,558	1,558	1,558	1,558	1,558	1,558	1,558	1,558
Net Revenue	US$ x 1,000	0	31,706	83,476	83,192	77,851	71,558	68,249	67,450	73,572	67,061
Cash Cost	US$/toz	0	452	506	488	613	665	742	808	782	625
Cash Cost	US$/t	0	17	21	20	23	23	25	27	28	21
AISC	US$/toz	0	605	679	635	804	822	978	973	940	821
Cash Cost	US$/toz	0	452	506	488	613	665	742	808	782	625
Freight	US$/toz	0	6	6	6	6	6	6	6	6	6
Refining	US$/toz	0	1	1	1	1	1	1	1	1	1
SG&A	US$/toz	0	81	61	62	66	72	75	76	70	51
CFEM	US$/toz	0	23	23	23	23	23	23	23	23	23
Royalties	US$/toz	0	42	42	42	39	40	40	40	41	41
CAPEX Sustaining	US$/toz	0	0	39	14	56	16	91	19	18	74

Table 22-3 Main Indicators of the Financial Model (cont.)

DESCRIPTION	UNIT	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
Exchange rate	**US$/US$**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**
Gross Margin	%	0.0%	59.1%	58.4%	59.0%	50.1%	45.2%	39.4%	33.9%	37.1%	45.9%
SG&A	US$ x 1,000	0	1,644	3,289	3,289	3,289	3,289	3,289	3,289	3,289	2,209
SG&A / Net Revenue	%	0.0%	5.2%	3.9%	4.0%	4.2%	4.6%	4.8%	4.9%	4.5%	3.3%
Net Profit	US$ x 1,000	0	10,404	35,565	35,886	27,647	22,167	17,657	14,268	17,758	21,879
Net Margin	%	0.0%	32.8%	42.6%	43.1%	35.5%	31.0%	25.9%	21.2%	24.1%	32.6%
EBITDA	US$ x 1,000	0	19,395	49,237	50,039	40,464	34,555	29,381	26,176	30,038	34,920
EBITDA margin	%	0.0%	61.2%	59.0%	60.1%	52.0%	48.3%	43.0%	38.8%	40.8%	52.1%
CAPEX	US$ x 1,000	36,099	36,675	2,116	737	2,784	737	3,969	834	834	3,185
CAPEX	US$ x 1,000	36,099	36,675	0	0	0	0	0	0	0	0
Sustaining CAPEX	US$ x 1,000	0	0	2,116	737	2,784	737	3,969	834	834	3,185
Working Capital Need	US$ x 1,000	0	2,718	4,941	4,839	5,245	5,173	5,335	5,575	5,825	4,852
Financial Cycle	days	0	31	21	21	24	26	28	30	29	26
Cash Flow (FCFF)	US$ x 1,000	(36,099)	(25,350)	38,504	42,951	32,304	29,908	22,079	22,540	25,764	28,776
Accumulated Cash Flow (FCFF)	US$ x 1,000	(36,099)	(61,450)	(22,945)	20,006	52,310	82,218	104,296	126,836	152,600	181,376

Table 22-4 Main Indicators of the Financial Model (cont.)

DESCRIPTION	UNIT	2031	2032	2033	2034	2035	2036	2037	2038	2039	TOTAL OR AVERAGE
Exchange rate	**US$/US$**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**
Plant Feed	t x 1,000	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	647	22,040
Production / Sales Volume	toz	39,586	30,489	27,001	22,404	17,339	17,339	17,339	17,397	8,661	598,142
Gold Price	US$/toz	1,558	1,558	1,558	1,558	1,558	1,558	1,558	1,558	1,558	1,558
Net Revenue	US$ x 1,000	61,675	47,502	42,068	34,906	27,014	27,014	27,014	27,105	13,494	931,905
Cash Cost	US$/toz	547	589	677	719	770	770	770	768	608	648
Cash Cost	US$/t	17	14	14	12	10	10	10	10	8	18
AISC	US$/toz	680	804	823	979	948	1,104	936	930	750	828
Cash Cost	US$/toz	547	589	677	719	770	770	770	768	608	648
Freight	US$/toz	6	6	6	6	6	6	6	6	6	6
Refining	US$/toz	1	1	1	1	1	1	1	1	1	1
SG&A	US$/toz	41	40	45	48	57	57	57	57	48	61
CFEM	US$/toz	23	23	23	23	23	23	23	23	23	23
Royalties	US$/toz	41	41	41	41	42	42	42	39	42	41

Table 22-5 Main Indicators of the Financial Model (cont.)

DESCRIPTION	UNIT	2031	2032	2033	2034	2035	2036	2037	2038	2039	TOTAL OR AVERAGE
Exchange rate	**US$/US$**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**	**5.155**
CAPEX Sustaining	US$/toz	21	104	31	142	48	205	37	37	22	49
Gross Margin	%	49.6%	50.0%	51.9%	48.4%	41.8%	43.6%	41.6%	43.6%	47.5%	-179.4%
SG&A	US$ x 1,000	1,631	1,217	1,217	1,066	996	996	996	996	415	36,404
SG&A / Net Revenue	%	2.6%	2.6%	2.9%	3.1%	3.7%	3.7%	3.7%	3.7%	3.1%	3.9%
Net Profit	US$ x 1,000	22,390	17,461	12,482	9,499	6,059	6,369	6,011	6,440	3,584	293,528
Net Margin	%	36.3%	36.8%	29.7%	27.2%	22.4%	23.6%	22.3%	23.8%	26.6%	31.5%
EBITDA	US$ x 1,000	35,603	26,167	20,680	16,159	11,418	11,418	11,418	11,559	7,192	465,819
EBITDA margin	%	57.7%	55.1%	49.2%	46.3%	42.3%	42.3%	42.3%	42.6%	53.3%	50.0%
CAPEX	US$ x 1,000	834	3,185	834	3,185	834	3,546	640	640	192	101,861
CAPEX	US$ x 1,000	0	0	0	0	0	0	0	0	0	72,775
Sustaining CAPEX	US$ x 1,000	834	3,185	834	3,185	834	3,546	640	640	192	29,087
Working Capital Need	US$ x 1,000	4,325	3,796	3,716	3,439	3,099	3,092	3,100	3,097	0	5,825
Financial Cycle	days	25	29	32	35	41	41	41	41	0	28
Cash Flow (FCFF)	US$ x 1,000	31,272	20,375	13,503	8,365	6,710	3,504	6,580	6,510	7,158	285,354
Accumulated Cash Flow (FCFF)	US$ x 1,000	212,649	233,024	246,527	254,892	261,601	265,105	271,685	278,196	285,354	285,354

Table 22-6 Summary Results of the Financial Model

DESCRIPTION	UNIT	VALUE
Project Term	years	19
Start-up	date	July 2022
Revenue / Production Volume	ktoz	598.1
Exchange Rate	R$/US$	5.155
Average Selling Price	US$/toz	1 558
Royalties	%	2.62%
CFEM	%	1.50%
Average Cash Cost	US$/toz	648
Average Cash Cost	US$/t	17.6
Average AISC	US$/toz	828
CAPEX (2021 and 2022)	US$ x 1,000	72,775
Sustaining CAPEX (2023 to 2039)	US$ x 1,000	29,087
EBITDA - Annual Average	US$ x 1,000	25,879
EBITDA Margin - Annual Average	%	50.0%
Maximum Working Capital Need	US$ x 1,000	5,825
Financial Cycle - Average	days	28
Tax Regime	-	Real Profit
Income Tax and Social Contribution	%	34.0%
Discount Rate	%	5.0%
IRR - Internal Rate of Return **	%	43.9%
NPV - Net Present Value **	US$ x 1,000	182,734
Profitability Index **	Index	3.1
Discounted PAYBACK **	years	3.7

* Project return indicators (without leverage).

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, royalties and other general parameters.

22.2.1 PRODUCT

Only gold is considered for production minerals – specified in grade and Troy Ounces (toz).

22.2.2 PRODUCTION

The period of the construction and production plans is based on Project years. The construction period begins in year 2021. The first output of saleable gold (doré) is planned to begin in Year 2022. Mining activity is planned to finish early in year 2039.

The metallurgical recovery for the contained gold is expected to be 92.5%, which results in 598.1 ktoz after processing.

The contract return assumed by refinery is 99.99% which results in 598.1 ktoz of payable gold.

Table 22-7 summarizes the annual feed to the process plant with the tonnes of ROM, mineral grades and gold content recover.

Table 22-7 Summary of Production Plan

DESCRIPTION	UNIT	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039
Plant Feed	t	-	529,400	1,320,300	1,320,300	1,322,850	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	647,205
Grade	g/t	-	1.29	1.36	1.36	1.27	1.19	1.13	1.12	1.22	1.11	1.02	0.79	0.70	0.58	0.45	0.45	0.45	0.45	0.45
Au content	toz	-	22,001	57,923	57,726	54,020	49,653	47,357	46,803	51,051	46,533	42,796	32,961	29,190	24,221	18,745	18,745	18,745	18,808	9,364
Recovery ratio	%	-	92.50%	92.50%	92.50%	92.50%	92.50%	92.50%	92.50%	92.50%	92.50%	92.50%	92.50%	92.50%	92.50%	92.50%	92.50%	92.50%	92.50%	92.50%
Au recovered by period	toz	-	20,351	53,579	53,397	49,968	45,929	43,806	43,292	47,222	43,043	39,586	30,489	27,001	22,404	17,339	17,339	17,339	17,397	8,661

22.2.3 CAPITAL INVESTMENT

The initial capital cost amounts to US$ 73 million, which includes an allowance for contingencies.

Table 22-8 summarizes the initial capital cost expenditure by commodity and disbursement schedule

Table 22-8 Initial Capital Cost Summary and Disbursement Schedule

ITEM	TOTAL	2021	2022
DESCRIPTION	**US$ X 1000**		
Administrative Costs - Owner Cost	1,613	800	813
Construction Management - Owner Cost	637	316	321
General Services - Owner Cost	83	41	42
Tailings Dam	2,446	1,213	1,233
Equipment	229	114	115
Lands - Owner Cost	1,249	620	629
Ausenco Scope	51,883	25,736	26,147
SSMAC - Owner Cost	962	477	485
Pre-Operation - Owner Cost	2,000	992	1,008
Mine	3,769	1,869	1,899
Electric - LT 138kV	1,817	901	916
Electric - Owner Cost	481	239	242
Consultancy and Expediting - Owner Cost	368	183	185
Contingency	4,637	2,300	2,337
Others	601	298	303
Total - CAPEX	**72,775**	**36,099**	**36,675**

The total sustaining capital expenditure during the operation periods amounts to US$ 29 million. Table 22-9 presents the sustaining capital breakdown.

Sustaining capital includes plant feed, tailings dam and other necessary costs to maintain the planned level of activities until the end of the Project life.

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.

According to the above premises, the capital requirement for the project's working capital is around US$ 6 million, equivalent to an average financial cycle of 28 days. Table 22-10 presents the working capital summary

Table 22-9 Sustaining Capital Summary

ITEM	TOTAL	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039
DESCRIPTION	**US$ X 1000**																		
Feed Plant	4,114	0	236	252	252	252	252	252	252	252	252	252	252	252	252	252	252	252	95
Tailings Dam	16,437	0	1,394	0	2,047	0	3,232	0	0	2,351	0	2,351	0	2,351	0	2,712	0	0	0
Maintenance	8,535	0	485	485	485	485	485	582	582	582	582	582	582	582	582	582	388	388	97
Total - Sustaining	29,087	0	2,116	737	2,784	737	3,969	834	834	3,185	834	3,185	834	3,185	834	3,546	640	640	192

Table 22-10 Working Capital Summary

DESCRIPTION	UNIT	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039
Aplicattions	US$ x 1,000	3,967	8,388	8,201	8,823	8,627	8,869	9,253	9,749	7,896	6,872	5,866	5,766	5,205	4,528	4,528	4,528	4,531	-
Accounts Receivable		881	2,319	2,311	2,163	1,988	1,896	1,874	2,044	1,863	1,713	1,320	1,169	970	750	750	750	753	-
Inventories		1,534	4,517	4,339	5,108	5,087	5,421	5,827	6,154	4,482	3,607	2,995	3,045	2,684	2,226	2,226	2,226	2,226	-
Other Inventories		1,552	1,552	1,552	1,552	1,552	1,552	1,552	1,552	1,552	1,552	1,552	1,552	1,552	1,552	1,552	1,552	1,552	-
Sources		1,249	3,447	3,362	3,577	3,454	3,534	3,678	3,924	3,044	2,547	2,070	2,050	1,766	1,429	1,436	1,428	1,433	-
Accounts Payable		1,026	2,853	2,763	3,116	3,084	3,239	3,439	3,628	2,678	2,173	1,778	1,782	1,562	1,300	1,300	1,300	1,296	-
Tax Liabilities		223	594	600	462	370	295	238	297	365	374	292	268	204	130	136	129	138	-
Working Capital	US$ x 1,000	2,718	4,941	4,839	5,245	5,173	5,335	5,575	5,825	4,852	4,325	3,796	3,716	3,439	3,099	3,092	3,100	3,097	-
Working Capital Variation	US$ x 1,000	2,718	2,223	(102)	406	(73)	162	240	250	(972)	(527)	(529)	(80)	(277)	(341)	(7)	8	(3)	(3,097)

22.2.4 OPERATING COSTS

The average pre-tax cash cost for on-site mining, processing, general and administrative operational activities and a 5% contingency over the processing costs (excluding labor) is US$ 709/toz produced. The total operating costs, including non-recoverable taxes and refining and transport, but not including royalties, is US$ 738/toz produced. 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 22-11 presents the operating cost summary.

Table 22-11 Operating Costs Summary

DESCRIPTION	R$/toz	R$/t ROM	Part %
Cash Cost	648	17.58	87.7%
Mining	262	7.11	35.5%
Processing	386	10.47	52.2%
G&A	61	1.65	8.2%
Non-Recoverable Taxes	23	0.63	3.2%
Refining & Transportation	6	0.17	0.9%
TOTAL OPERATING COSTS & EXPENSES	**738**	**20.04**	**100.0%**

The LOM annual operating cost projections are shown in Table 22-12 to Table 22-15.

The mine closure is an additional cost that Aura considered yearly at US$ 1.1 million in the last five years of the LOM Almas Project resulting in the amount of US$ 5.5 million.

22.2.5 REVENUE

Projections of net revenue are based on the quantity of gold to be delivered (598.1 ktoz LOM) at an average long-term gold price of US$ 1,558/toz gold. Third-party services for treatment and refining are fixed at US$ 0.60/toz, while the transportation of the doré from site to refinery has an average unit cost of US$ 5.74/toz.

Payable gold is assumed at 99.99% of the contents sold.

Annual average net revenue is US$ 56 million from year 1 (full run rate production period) to year 16. Annual projections are shown in Table 22-16.

Table 22-12 Mining Costs Summary (without depreciation)

DESCRIPTION	UNIT	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039
Heap Leach + Complementary – ROM	US$ x 1,000	0	0	0	0	0	85	96	94	89	129	91	480	614	668	905	905	905	905	451
Paiol – ROM		0	513	3,229	2,753	3,112	2,072	2,613	2,981	2,422	1,848	1,126	0	0	0	0	0	0	0	0
Paiol – Waste		0	2,295	8,998	8,402	10,057	11,282	12,900	15,305	15,256	6,600	953	0	0	0	0	0	0	0	0
Cata Funda – ROM		0	0	0	0	407	390	348	365	0	0	0	0	0	0	0	0	0	0	0
Cata Funda – Waste		0	0	0	0	2,184	1,920	1,795	1,446	0	0	0	0	0	0	0	0	0	0	0
Vira Saia – ROM		0	0	0	0	0	0	0	0	1,136	1,136	1,152	1,168	1,184	873	0	0	0	0	0
Vira Saia – Waste		0	0	0	0	0	0	0	0	3,246	3,366	4,966	3,324	3,480	1,745	0	0	0	0	0
Mine Costs (without depreciation)	**US$ x 1,000**	0	2,808	12,227	11,154	15,760	15,749	17,753	20,191	22,150	13,079	8,289	4,972	5,278	3,286	905	905	905	905	451

Table 22-13 Processing Costs - Variable Cost

DESCRIPTION	UNIT	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039
Plant Feed	t	0	529,400	1,320,300	1,320,300	1,322,850	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	647 205
Reagents and Consumables	US$ / t	0	4.12	4.12	4.12	4.12	4.12	4.12	4.12	4.12	4.12	4.12	4.12	4.12	4.12	4.12	4.12	4.12	4.12	3.12
Maintenance		0	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.61
Energy		0	1.58	1.51	1.51	1.51	1.54	1.54	1.54	1.54	1.54	1.54	1.54	1.54	1.54	1.54	1.54	1.54	1.54	1.16
ETE / ETA		0	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.001
Processing Costs - Variable Cost	US$ / t	0	6.5	6.44	6.44	6.43	6.46	6.46	6.46	6.46	6.46	6.46	6.46	6.46	6.46	6.46	6.46	6.46	6.46	4.89
Reagents and Consumables	US$ x 1,000	0	2,180	5,438	5,438	5,448	5,354	5,354	5,354	5,354	5,354	5,354	5,354	5,354	5,354	5,354	5,354	5,354	5,354	2,017
Maintenance		0	424	1,056	1,056	1,058	1,040	1,040	1,040	1,040	1,040	1,040	1,040	1,040	1,040	1,040	1,040	1,040	1,040	392
Energy		0	835	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	753
ETE / ETA		0	1	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	1
Processing Costs - Variable Cost	US$ x 1,000	0	3,440	8,497	8,497	8,509	8,397	8,397	8,397	8,397	8,397	8,397	8,397	8,397	8,397	8,397	8,397	8,397	8,397	3,163

Table 22-14 Processing Costs - Fixed Cost

DESCRIPTION	UNIT	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039
Labor	US$ x 1,000	0	1,500	2,999	2,999	2,999	2,999	2,999	2,999	2,999	2,827	2,827	2,780	2,780	2,601	2,236	2,236	2,236	2,236	932
General Management		0	104	208	208	208	208	208	208	208	208	208	208	208	208	208	208	208	208	87
Open Pit Mine		0	325	650	650	650	650	650	650	650	503	503	464	464	374	283	283	283	283	118
Plant / Maintenance		0	895	1,789	1,789	1,789	1,789	1,789	1,789	1,789	1,789	1,789	1,789	1,789	1,701	1,506	1,506	1,506	1,506	627
SSMAC		0	176	352	352	352	352	352	352	352	327	327	319	319	319	239	239	239	239	100
Laboratory		0	806	1,979	1,979	1,979	1,979	1,979	1,979	1,979	1,227	792	495	495	495	495	495	495	495	186
Access Maintenance		0	59	117	117	117	117	117	117	117	117	117	117	117	117	117	117	117	117	49
Equipment Rental		0	360	720	720	720	720	720	720	720	720	720	720	720	720	720	720	720	720	300
Processing Costs - Fixed Cost	**US$ x 1,000**	0	2,724	5,815	5,815	5,815	5,815	5,815	5,815	5,815	4,891	4,456	4,112	4,112	3,933	3,568	3,568	3,568	3,568	1467

Table 22-15 Freight / Refining

DESCRIPTION	UNIT	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039
Freight to Refinery	US$/toz	0	5.74	5.74	5.74	5.74	5.74	5.74	5.74	5.74	5.74	5.74	5.74	5.74	5.74	5.74	5.74	5.74	5.74	5.74
Refining	US$/toz	0	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60
Produced Gold	**toz**	0	20,351	53,579	53,397	49,968	45,929	43,806	43,292	47,222	43,043	39,586	30,489	27,001	22,404	17,339	17,339	17,339	17,397	8,661
Freight / Refining	**US$ x 1,000**	0	129	340	339	317	291	278	274	299	273	251	193	171	142	110	110	110	110	55
Exchange Rate	R$/US$	5.155	5.155	5.155	5.155	5.155	5.155	5.155	5.155	5.155	5.155	5.155	5.155	5.155	5.155	5.155	5.155	5.155	5.155	5.155
Freight / Refining	**R$ x 1,000**	0	665	1,751	1,745	1,633	1,501	1,432	1,415	1,543	1,407	1,294	996	882	732	567	567	567	569	283

Table 22-16 Gross Revenue Projection

ITEM	UNIT	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039
Au recovered by period	toz	20,351	53,579	53,397	49,968	45,929	43,806	43,292	47,222	43,043	39,586	30,489	27,001	22,404	17,339	17,339	17,339	17,397	8,661
Price	US$/toz	1,558	1,558	1,558	1,558	1,558	1,558	1,558	1,558	1,558	1,558	1,558	1,558	1,558	1,558	1,558	1,558	1,558	1,558
GROSS REVENUE	US$ x 1,000	31,706	83,476	83,192	77,851	71,558	68,249	67,450	73,572	67,061	61,675	47,502	42,068	34,906	27,014	27,014	27,014	27,105	13,494
Exchange Rate	R$ / US$	5.1550	5.1550	5.1550	5.1550	5.1550	5.1550	5.1550	5.1550	5.1550	5.1550	5.1550	5.1550	5.1550	5.1550	5.1550	5.1550	5.1550	5.1550
GROSS REVENUE	R$ x 1,000	163,445	430,319	428,856	401,321	368,880	351,825	347,703	379,265	345,698	317,933	244,874	216,859	179,938	139,255	139,255	139,255	139,727	69,563

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 presented in Table 22-17.

Table 22-17 Royalty Rates by Area

AREA	UNIT	VALUE
Paiol	%	2.70%
Cata Funda	%	1.95%
Vira Saia	%	2.50%

22.2.7 TAXATION

Taxes that are due for the Almas 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% is applied to pre-tax profit.

- PIS, COFINS and ICMS

  These taxes were not applied in this analysis since all production is directed for exportation.

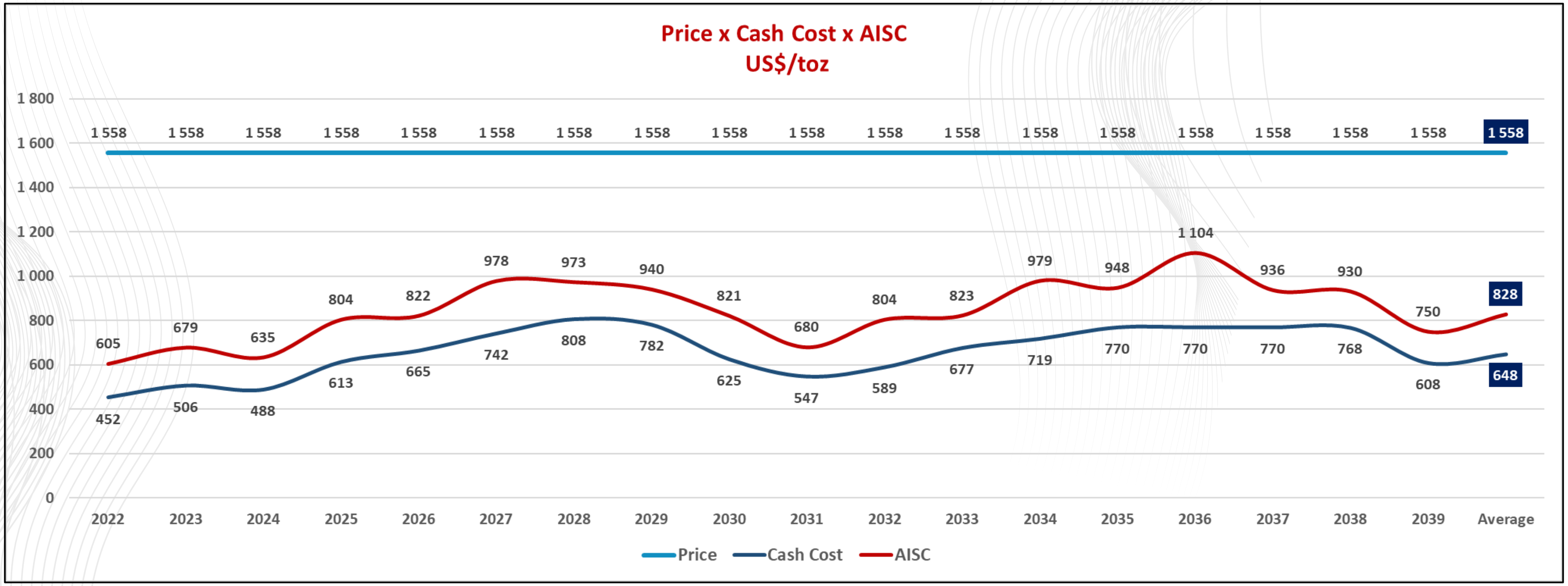
22.2.8 ALL-IN SUSTAINING COSTS

Table 22-18 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 22-18 All-In Sustaining Costs

DESCRIPTION	UNIT	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	TOTAL OR AVERAGE
AISC	US$ / toz	605	679	635	804	822	978	973	940	821	680	804	823	979	948	1 104	936	930	750	828
Cash Cost	US$ / toz	452	506	488	613	665	742	808	782	625	547	589	677	719	770	770	770	768	608	648
Freight	US$ / toz	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Refining	US$ / toz	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
SG&A	US$ / toz	81	61	62	66	72	75	76	70	51	41	40	45	48	57	57	57	57	48	61
CFEM	US$ / toz	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23
Royalties	US$ / toz	42	42	42	39	40	40	40	41	41	41	41	41	41	42	42	42	39	42	41
Sustaining CAPEX	US$ / toz	0	39	14	56	16	91	19	18	74	21	104	31	142	48	205	37	37	22	49

Figure 22-1 Price x Cash Cost x AISC Graph



22.3 FINANCIAL ANALYSIS

The financial model used by GE21 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 Brazilian reais (US$ x 1000) 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$ 183 million, at a Discount Rate of 5.0%. The internal rate of return ("IRR") is 43.9% and the annual average EBITDA (from year 1 to year 16, full run rate production period) is US$ 27 million. Payback after the start-up of operations is 2.0 years.

The results are summarized in Table 22-19 and the operating income statement and the project cash flow are respectively presented in Table 22-20 and Table 22-21.

Table 22-19 Financial Results Summary (Post tax)

ITEM	UNIT	VALUE
DISCOUNT RATE (WACC)	%	5.0%
NET PRESENT VALUE – NPV	US$ million	182.7
CAPEX NPV	US$ million	(87.8)
OPERATIONAL NPV	US$ million	270.5
PROJECT IRR	%	43.9%
PROJECT PROFITABILITY INDEX		3.1
DISCOUNTED PROJECT PAYBACK	Years	3.7
SIMPLE PAYBACK (including start-up)	Years	3.5
SIMPLE PAYBACK (after start-up)	Years	2.0

On analysis of the pre-tax cash flow, NPV rises to US$ 231 million and the IRR to 53.5%.

GE21 understands that Almas Gold Project is economically viable and attractive based on these results

Table 22-20 Operating Income Statement (R$ x 1000)

DESCRIPTION	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039
Gross Operating Revenue	0	31,706	83,476	83,192	77,851	71,558	68,249	67,450	73,572	67,061	61,675	47,502	42,068	34,906	27,014	27,014	27,014	27,105	13,494
Deductions from Operating Revenue	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Net Operating Revenue	0	31,706	83,476	83,192	77,851	71,558	68,249	67,450	73,572	67,061	61,675	47,502	42,068	34,906	27,014	27,014	27,014	27,105	13,494
Cash Cost	0	(9,206)	(27,104)	(26,032)	(30,651)	(30,522)	(32,526)	(34,964)	(36,923)	(26,890)	(21,642)	(17,967)	(18,273)	(16,102)	(13,356)	(13,356)	(13,356)	(13,356)	(5,265)
Freight / Refining	0	(129)	(340)	(339)	(317)	(291)	(278)	(274)	(299)	(273)	(251)	(193)	(171)	(142)	(110)	(110)	(110)	(110)	(55)
Depreciation and Exhaustion	0	(3,639)	(7,277)	(7,701)	(7,848)	(8,405)	(8,552)	(9,346)	(9,090)	(9,109)	(9,189)	(5,570)	(1,774)	(1,774)	(2,244)	(1,774)	(2,317)	(1,808)	(1,769)
Gross Profit	0	18,733	48,754	49,121	39,035	32,340	26,894	22,865	27,260	30,789	30,592	23,772	21,849	16,887	11,303	11,773	11,231	11,831	6,405
Gross margin (without depreciation)	0.0%	59.1%	58.4%	59.0%	50.1%	45.2%	39.4%	33.9%	37.1%	45.9%	49.6%	50.0%	51.9%	48.4%	41.8%	43.6%	41.6%	43.6%	47.5%
SG&A	0	(1,644)	(3,289)	(3,289)	(3,289)	(3,289)	(3,289)	(3,289)	(3,289)	(2,209)	(1,631)	(1,217)	(1,217)	(1,066)	(996)	(996)	(996)	(996)	(415)
SG&A - Depreciation	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
SG & A / Net Revenue	0.0%	5.2%	3.9%	4.0%	4.2%	4.6%	4.8%	4.9%	4.5%	3.3%	2.6%	2.6%	2.9%	3.1%	3.7%	3.7%	3.7%	3.7%	3.1%
CFEM	0	(476)	(1,252)	(1,248)	(1,168)	(1,073)	(1,024)	(1,012)	(1,104)	(1,006)	(925)	(713)	(631)	(524)	(405)	(405)	(405)	(407)	(202)
Royalties	0	(856)	(2,254)	(2,246)	(1,963)	(1,827)	(1,752)	(1,734)	(1,920)	(1,763)	(1,623)	(1,245)	(1,095)	(913)	(729)	(729)	(729)	(678)	(364)
Income before Income Tax / Social Contribution	0	15,757	41,960	42,338	32,616	26,151	20,829	16,830	20,948	25,811	26,414	20,597	18,906	14,385	9,173	9,643	9,101	9,751	5,423
Income Tax	0	(2,364)	(6,294)	(6,351)	(4,892)	(3,923)	(3,124)	(2,525)	(3,142)	(3,872)	(3,962)	(3,090)	(2,836)	(2,158)	(1,376)	(1,446)	(1,365)	(1,463)	(813)

DESCRIPTION	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039
Income Tax (over R$ 60 thousand in the quarter)	0	(1,571)	(4,191)	(4,229)	(3,257)	(2,610)	(2,078)	(1,678)	(2,090)	(2,576)	(2,637)	(2,055)	(1,886)	(1,434)	(913)	(960)	(905)	(970)	(538)
Income Tax - Benefit	0	0	7,867	7,938	6,115	4,903	3,905	3,156	3,928	4,840	4,953	3,862	0	0	0	0	0	0	0
Social Contribution	0	(1,418)	(3,776)	(3,810)	(2,935)	(2,354)	(1,875)	(1,515)	(1,885)	(2,323)	(2,377)	(1,854)	(1,701)	(1,295)	(826)	(868)	(819)	(878)	(488)
Net Income	0	10,404	35,565	35,886	27,647	22,167	17,657	14,268	17,758	21,879	22,390	17,461	12,482	9,499	6,059	6,369	6,011	6,440	3,584
Net Margin	0.0%	32.8%	42.6%	43.1%	35.5%	31.0%	25.9%	21.2%	24.1%	32.6%	36.3%	36.8%	29.7%	27.2%	22.4%	23.6%	22.3%	23.8%	26.6%
EBITDA	0	19,395	49,237	50,039	40,464	34,555	29,381	26,176	30,038	34,920	35,603	26,167	20,680	16,159	11,418	11,418	11,418	11,559	7,192
EBITDA margin	0.0%	61.2%	59.0%	60.1%	52.0%	48.3%	43.0%	38.8%	40.8%	52.1%	57.7%	55.1%	49.2%	46.3%	42.3%	42.3%	42.3%	42.6%	53.3%

Table 22-21 Project Cash Flow (R$ x 1000)

DESCRIPTION	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039
EBIT	**0**	**15,757**	**41,960**	**42,338**	**32,616**	**26,151**	**20,829**	**16,830**	**20,948**	**25,811**	**26,414**	**20,597**	**18,906**	**14,385**	**9,173**	**9,643**	**9,101**	**9,751**	**5,423**
(+) Depreciation	0	3,639	7,277	7,701	7,848	8,405	8,552	9,346	9,090	9,109	9,189	5,570	1,774	1,774	2,244	1,774	2,317	1,808	1,769
(=) EBITDA	**0**	**19,395**	**49,237**	**50,039**	**40,464**	**34,555**	**29,381**	**26,176**	**30,038**	**34,920**	**35,603**	**26,167**	**20,680**	**16,159**	**11,418**	**11,418**	**11,418**	**11,559**	**7,192**
(-) CAPEX	(36,099)	(36,675)	(2,116)	(737)	(2,784)	(737)	(3,969)	(834)	(834)	(3,185)	(834)	(3,185)	(834)	(3,185)	(834)	(3,546)	(640)	(640)	(192)
(+-) Working Capital Variation	0	(2,718)	(2,223)	102	(406)	73	(162)	(240)	(250)	972	527	529	80	277	341	7	(8)	3	3,097
(-) Mine Closure Cost	0	0	0	0	0	0	0	0	0	0	0	0	0	0	(1,100)	(1,100)	(1,100)	(1,100)	(1,100)

DESCRIPTION	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039
(-) Income Tax / Social Contribution	0	(5,353)	(6,394)	(6,452)	(4,969)	(3,983)	(3,172)	(2,562)	(3,190)	(3,932)	(4,023)	(3,136)	(6,423)	(4,886)	(3,114)	(3,274)	(3,090)	(3,311)	(1,839)
(=) Free Cash Flow to Firm (FCFF)	**(36,099)**	**(25,350)**	**38,504**	**42,951**	**32,304**	**29,908**	**22,079**	**22,540**	**25,764**	**28,776**	**31,272**	**20,375**	**13,503**	**8,365**	**6,710**	**3,504**	**6,580**	**6,510**	**7,158**
(=) Accumulated Free Cash Flow to Firm	**(36,099)**	**(61,450)**	**(22,945)**	**20,006**	**52,310**	**82,218**	**104,296**	**126,836**	**152,600**	**181,376**	**212,649**	**233,024**	**246,527**	**254,892**	**261,601**	**265,105**	**271,685**	**278,196**	**285,354**

22.4 SENSITIVITY ANALYSIS

22.4.1 SPIDER GRAPH ANALYSIS

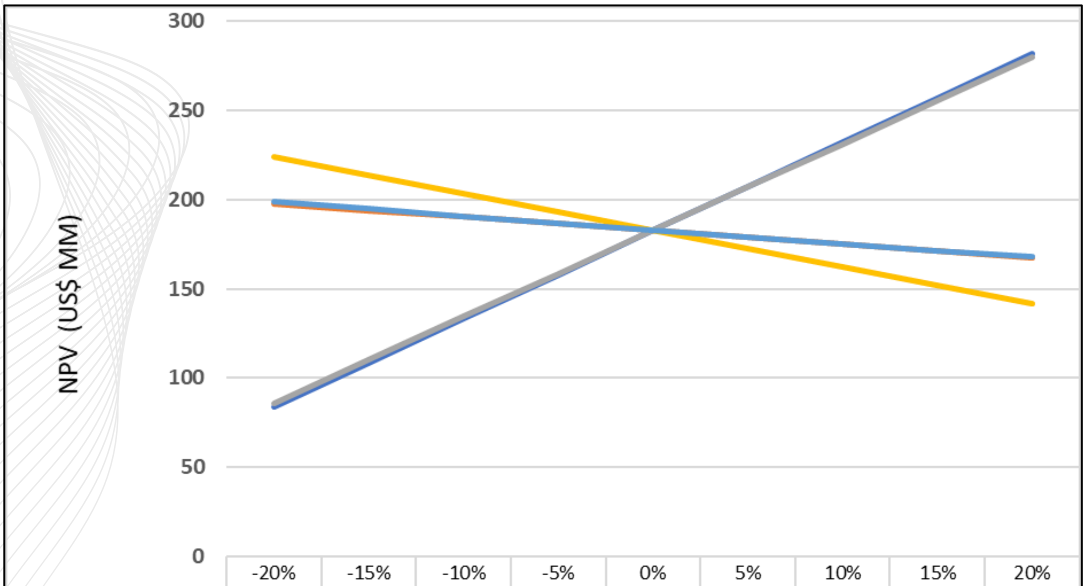
The sensitivity analysis shows the impact of the variation of the gold price, exchange rates, operating and capital costs upon the Project NPV and IRR. The analysis encompasses the following range of variation in the key inputs:

- Gold price: ±20%.
- CAPEX: ±20%.
- Exchange Rate: ±20%.
- Cash Cost: ±20%.
- Discount Rate: ±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.

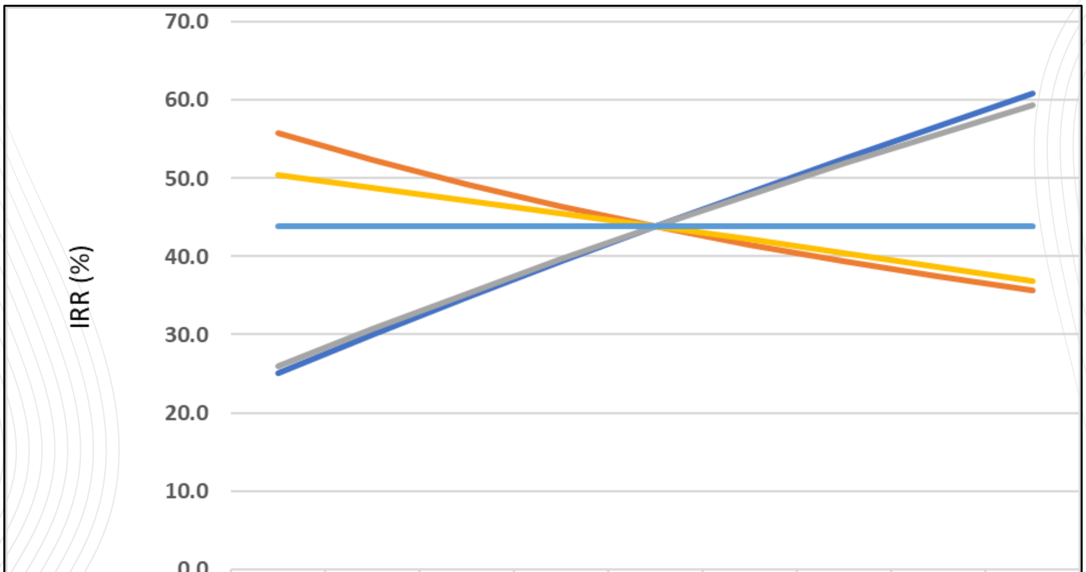
Figure 22-2 illustrates the results of the sensitivity analysis for Project NPV (after tax) and these effects for each of the critical variables and Figure 22-3 presents the same scenario for the IRR. NPV results are reported at a discount rate of 5.0%.

Figure 22-2 Sensitivity Analysis Graph – NPV



	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%
Gold Price	84	108	133	158	183	208	232	257	282
CAPEX	198	194	190	187	183	179	175	171	168
Exchange Rate	86	110	134	159	183	207	231	255	280
Cash Cost	224	214	203	193	183	172	162	152	141
Discount Rate	199	195	191	187	183	179	175	171	168

Figure 22-3 Sensitivity Analysis Graph – IRR



		-20%	-15%	-10%	-5%	0%	5%	10%	15%
Gold Price	25.1	30.1	34.8	39.4	43.9	48.2	52.5	56.7	60.8
CAPEX	55.8	52.3	49.2	46.4	43.9	41.5	39.4	37.4	35.6
Exchange Rate	25.9	30.7	35.3	39.7	43.9	47.9	51.9	55.7	59.4
Cash Cost	50.4	48.8	47.2	45.5	43.9	42.2	40.4	38.7	36.9
Discount Rate	43.9	43.9	43.9	43.9	43.9	43.9	43.9	43.9	43.9

22.4.2 TWO PARAMETERS ANALYSIS

Additionally, secondary sensitivity analyses were made varying two parameters simultaneously to assess the impact on the IRR and NPV:

- Gold Price x CAPEX (Table 22- 22);
- Gold Price x Exchange Rate (Table 22- 23);
- Gold Price x Costs (Table 22- 24);
- Gold Price x Recovery Ratio (Table 22- 25);
- Gold Price x Discount Rate (Table 22- 26);.

Table 22- 27 presents another sensitivity analysis that was made varying Price and Exchange Rate to assess the impact on the AISC.

Table 22-22 Sensitivity Analysis Graph – Price x CAPEX – IRR and NPV

IRR (%)		GOLD PRICE (US$/OZ)								
		1,200	1,400	1,558	1,700	1,800	1,900	2,000	2,100	2,200
Discount Rate	5.00%	77.0 %	89.9 %	100 %	109.1 %	115.5 %	122.0 %	128.4 %	134.8 %	141.2 %
CAPEX (%)	120.0 %	16.4	27.6	35.6	42.4	47.1	51.7	56.1	60.5	64.9
	115.0 %	17.7	29.2	37.4	44.5	49.3	54.0	58.7	63.2	67.7
	110.0 %	19.0	30.9	39.4	46.7	51.7	56.6	61.4	66.1	70.7
	105.0 %	20.4	32.7	41.5	49.1	54.3	59.4	64.3	69.2	74.1
	100.0 %	22.0	34.7	43.9	51.7	57.1	62.4	67.6	72.7	77.7
	97.5%	22.8	35.7	45.1	53.1	58.6	64.0	69.3	74.5	79.6
	95.0%	23.7	36.9	46.4	54.6	60.2	65.7	71.1	76.4	81.7
	92.5%	24.6	38.0	47.8	56.2	61.9	67.5	73.0	78.4	83.8
	90.0%	25.6	39.3	49.2	57.8	63.6	69.4	75.0	80.6	86.0

NPV (US$ MM)		GOLD PRICE (US$/OZ)								
		1,200	1,400	1,558	1,700	1,800	1,900	2,000	2,100	2,200
Discount Rate	5.00 %	77.0 %	89.9 %	100 %	109.1 %	115.5 %	122.0 %	128.4 %	134.8 %	141.2 %
CAPEX (%)	120%	54	117	168	213	245	276	308	340	372
	115%	58	121	171	217	248	280	312	344	376
	110%	61	125	175	220	252	284	316	348	379
	105%	65	129	179	224	256	288	320	351	383
	100%	69	132	183	228	260	292	323	355	387
	98%	71	134	185	230	262	293	325	357	389
	95%	73	136	187	232	263	295	327	359	391
	93%	75	138	188	234	265	297	329	361	393
	90%	76	140	190	235	267	299	331	363	394

Table 22-23 Sensitivity Analysis Graph – Price x Exchange Rate – IRR and NPV

IRR (%)		GOLD PRICE (US$/OZ)								
		1,200	1,400	1,558	1,700	1,800	1,900	2,000	2,100	2,200
Discount Rate	5.00 %	77.0 %	89.9 %	100 %	109.1 %	115.5 %	122.0 %	128.4 %	134.8 %	141.2 %
EXCHANGE RATE (R$/US$)	4.25	5.7	19.5	28.3	35.6	40.5	45.3	50.0	54.6	59.0
	4.50	11.0	24.0	32.8	40.3	45.3	50.2	55.0	59.8	64.4
	4.75	15.5	28.2	37.2	44.8	49.9	55.0	59.9	64.8	69.6
	5.00	19.6	32.3	41.4	49.1	54.4	59.6	64.7	69.7	74.6
	5.155	22.0	34.7	43.9	51.7	57.1	62.4	67.6	72.7	77.7
	5.25	23.4	36.1	45.4	53.3	58.8	64.1	69.3	74.5	79.5
	5.35	24.9	37.6	46.9	55.0	60.5	65.8	71.1	76.3	81.5
	5.50	27.0	39.8	49.3	57.4	63.0	68.4	73.8	79.1	84.4
	5.75	30.4	43.4	53.1	61.4	67.1	72.7	78.2	83.7	89.1

NPV (US$ MM)		GOLD PRICE (US$/OZ)								
		1,200	1,400	1,558	1,700	1,800	1,900	2,000	2,100	2,200
Discount Rate	5.00 %	77.0 %	89.9 %	100 %	109.1 %	115.5 %	122.0 %	128.4 %	134.8 %	141.2 %
EXCHANGE RATE (R$/US$)	4.25	3	68	119	164	195	227	259	291	323
	4.50	24	89	139	184	216	248	279	311	343
	4.75	43	107	157	202	234	266	298	329	361
	5.00	60	123	173	219	250	282	314	346	378
	5.155	69	132	183	228	260	292	323	355	387
	5.25	74	138	188	233	265	297	329	361	392
	5.35	80	143	194	239	271	303	334	366	398
	5.50	88	151	202	247	279	310	342	374	406
	5.75	100	164	214	259	291	323	355	386	418

Table 22-24 Sensitivity Analysis Graph – Price x Costs – IRR and NPV

IRR (%)		GOLD PRICE (US$/OZ)								
		1,200	1,400	1,558	1,700	1,800	1,900	2,000	2,100	2,200
Discount Rate	5.00 %	77.0 %	89.9 %	100 %	109.1 %	115.5 %	122.0 %	128.4 %	134.8 %	141.2 %
COSTS (%)	110.0 %	17.6	31.0	40.4	48.5	54.0	59.3	64.6	69.7	74.8
	107.5 %	18.7	31.9	41.3	49.3	54.8	60.1	65.3	70.5	75.5
	105.0 %	19.8	32.8	42.2	50.1	55.6	60.9	66.1	71.2	76.3
	102.5 %	20.9	33.8	43.0	50.9	56.3	61.6	66.8	71.9	77.0
	100.0 %	22.0	34.7	43.9	51.7	57.1	62.4	67.6	72.7	77.7
	97.5 %	23.0	35.6	44.7	52.5	57.9	63.1	68.3	73.4	78.4
	95.0 %	24.1	36.5	45.5	53.3	58.7	63.9	69.0	74.1	79.1
	92.5 %	25.1	37.4	46.4	54.1	59.4	64.7	69.8	74.8	79.8
	90.0 %	26.1	38.2	47.2	54.9	60.2	65.4	70.5	75.6	80.5

NPV (US$ MM)		GOLD PRICE (US$/OZ)								
		1,200	1,400	1,558	1,700	1,800	1,900	2,000	2,100	2,200
Discount Rate	5.00 %	77.0 %	89.9 %	100 %	109.1 %	115.5 %	122.0 %	128.4 %	134.8 %	141.2 %
COSTS (%)	110.0 %	48	112	162	207	239	271	303	334	366
	107.5 %	53	117	167	212	244	276	308	340	371
	105.0 %	59	122	172	218	249	281	313	345	377
	102.5 %	64	127	178	223	255	286	318	350	382
	100.0 %	69	132	183	228	260	292	323	355	387
	97.5 %	74	138	188	233	265	297	329	360	392
	95.0 %	79	143	193	238	270	302	334	365	397
	92.5 %	84	148	198	243	275	307	339	371	402
	90.0 %	90	153	203	249	280	312	344	376	408

Table 22-25 Sensitivity Analysis Graph – Price x Recovery Ratio – IRR and NPV

IRR (%)		GOLD PRICE (US$/OZ)								
		1,200	1,400	1,558	1,700	1,800	1,900	2,000	2,100	2,200
Discount Rate	5.00 %	77.0 %	89.9 %	100 %	109.1 %	115.5 %	122.0 %	128.4 %	134.8 %	141.2 %
RECOVERY RATIO (%)	90.5 %	20.2	32.9	42.0	49.7	55.0	60.2	65.3	70.4	75.3
	91.0 %	20.7	33.3	42.4	50.2	55.6	60.8	65.9	70.9	75.9
	91.5 %	21.1	33.8	42.9	50.7	56.1	61.3	66.5	71.5	76.5
	92.0 %	21.6	34.2	43.4	51.2	56.6	61.9	67.0	72.1	77.1
	92.5 %	22.0	34.7	43.9	51.7	57.1	62.4	67.6	72.7	77.7
	93.5 %	22.9	35.6	44.8	52.7	58.1	63.5	68.7	73.8	78.9
	94.5 %	23.7	36.5	45.7	53.7	59.2	64.5	69.8	74.9	80.0
	95.5 %	24.6	37.4	46.7	54.7	60.2	65.6	70.9	76.1	81.2
	96.5 %	25.4	38.2	47.6	55.7	61.2	66.6	72.0	77.2	82.4

NPV (US$ MM)		GOLD PRICE (US$/OZ)								
		1,200	1,400	1,558	1,700	1,800	1,900	2,000	2,100	2,200
Discount Rate	5.00 %	77.0 %	89.9 %	100 %	109.1 %	115.5 %	122.0 %	128.4 %	134.8 %	141.2 %
RECOVERY RATIO (%)	90.5 %	61	123	172	216	247	278	310	341	372
	91.0 %	63	125	175	219	250	282	313	344	376
	91.5 %	65	128	177	222	254	285	316	348	379
	92.0 %	67	130	180	225	257	288	320	352	383
	92.5 %	69	132	183	228	260	292	323	355	387
	93.5 %	73	137	188	234	266	298	330	362	394
	94.5 %	77	142	193	240	272	305	337	370	402
	95.5 %	81	147	199	245	278	311	344	377	410
	96.5 %	85	152	204	251	284	318	351	384	417

Table 22-26 Sensitivity Analysis Graph – Price x Discount Rate – IRR and NPV

IRR (%)		GOLD PRICE (US$/OZ)								
		1,200	1,400	1,558	1,700	1,800	1,900	2,000	2,100	2,200
Discount Rate	5.0%	77.0%	89.9%	100 %	109.1%	115.5%	122.0%	128.4%	134.8%	141.2%
DISCOUNT RATE (%)	7.0%	22.0	34.7	43.9	51.7	57.1	62.4	67.6	72.7	77.7
	6.5%	22.0	34.7	43.9	51.7	57.1	62.4	67.6	72.7	77.7
	6.0%	22.0	34.7	43.9	51.7	57.1	62.4	67.6	72.7	77.7
	5.5%	22.0	34.7	43.9	51.7	57.1	62.4	67.6	72.7	77.7
	5.0%	22.0	34.7	43.9	51.7	57.1	62.4	67.6	72.7	77.7
	4.5%	22.0	34.7	43.9	51.7	57.1	62.4	67.6	72.7	77.7
	4.0%	22.0	34.7	43.9	51.7	57.1	62.4	67.6	72.7	77.7
	3.5%	22.0	34.7	43.9	51.7	57.1	62.4	67.6	72.7	77.7
	3.0%	22.0	34.7	43.9	51.7	57.1	62.4	67.6	72.7	77.7

NPV (US$ MM)		GOLD PRICE (US$/OZ)								
		1,200	1,400	1,558	1,700	1,800	1,900	2,000	2,100	2,200
Discount Rate	5.0%	77.0%	89.9%	100 %	109.1%	115.5%	122.0%	128.4%	134.8%	141.2%
DISCOUNT RATE (%)	7.0%	54	110	154	194	222	250	278	305	333
	6.5%	58	115	161	202	231	259	288	317	346
	6.0%	61	121	168	210	240	270	299	329	359
	5.5%	65	126	175	219	250	280	311	342	373
	5.0%	69	132	183	228	260	292	323	355	387
	4.5%	73	139	191	238	270	303	336	369	402
	4.0%	77	145	199	248	282	316	350	384	418
	3.5%	82	152	208	258	294	329	364	399	435
	3.0%	86	160	217	269	306	343	379	416	452

Table 22-27 Sensitivity Analysis Graph – Price x Exchange Rate – AISC

AISC (Average 5 Years) / US$/OZ		GOLD PRICE (US$/OZ)								
		1,200	1,400	1,558	1,700	1,800	1,900	2,000	2,100	2,200
		77.0%	89.9%	100%	109.1%	115.5%	122.0%	128.4%	134.8%	141.2%
DOLLAR (R$/US$)	4.25	830	838	845	850	855	859	863	867	871
	4.50	787	795	802	807	812	816	820	824	828
	4.75	748	757	763	769	773	777	781	786	790
	5.00	714	722	729	734	739	743	747	751	755
	5.155	694	702	709	715	719	723	727	731	735
	5.25	682	691	697	703	707	711	715	720	724
	5.35	671	679	685	691	696	700	704	708	712
	5.50	654	662	669	675	679	683	687	691	695
	5.75	628	636	643	649	653	657	661	665	669

22.4.3 LEVERAGE ANALYSIS

All return indicators of the project were estimated based on the Free Cash Flow to Firm (“FCFF”). Considering that the third party resources have an important effect on the return on equity, a sensitivity analysis of the shareholder return was made in accordance with the following economic assumptions:

- Capital cost: 4.5% per year. + exchange rate variation;
- Grace period for financing repayments: 4 years;
- 6 annual amortization installments;
- Annual interest rate.

Table 22-28 presents the results of the sensitivity analysis for the shareholder return in accordance with the % of Debt x CAPEX and different gold price scenarios.

Table 22-28 Sensitivity Analysis Graph– % Debt x Gold Price – Return on Equity

RETURN ON EQUITY (%)		GOLD PRICE (US$/OZ)								
		1,200	1,400	1,558	1,700	1,800	1,900	2,000	2,100	2,200
		77.0%	89.9%	100%	109.1%	115.5%	122.0%	128.4%	134.8%	141.2%
DEBT / CAPEX (%)	30%	30.4	48.2	61.0	71.8	79.2	86.4	93.5	100.5	107.3
	35%	32.7	51.8	65.4	77.0	84.9	92.6	100.1	107.5	114.8
	40%	35.4	56.0	70.6	83.0	91.4	99.6	107.7	115.6	123.4
	45%	38.7	60.9	76.7	90.0	99.1	108.0	116.6	125.2	133.6
	50%	42.7	66.9	84.0	98.4	108.3	117.8	127.3	136.5	145.6
	55%	47.8	74.3	92.9	108.6	119.3	129.8	140.1	150.2	160.1
	60%	54.2	83.4	103.9	121.2	133.0	144.6	155.9	167.1	178.1
	65%	62.7	95.1	117.9	137.2	150.4	163.3	176.0	188.5	200.8
	70%	74.2	110.6	136.3	158.2	173.2	187.9	202.4	216.6	230.7

22.4.4 CONCLUSION

GE21 has prepared the financial model for the Almas Project using capital costs, operating expenditures and production schedule with inputs provided by Aura, EY, Ausenco and EDEM.

The financial model used by GE21 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 and includes some fiscal benefits.

After the evaluation, the NPV Post-tax, at a WACC rate of 5% per year, resulted in US$ 183 million, with an IRR of 43.9% and a 2.0-year Payback Time. For this scenario, the gold price adopted was US$1,558/toz and the exchange rate used was US$1.00 = R$5.155.

A series of analysis was made varying Gold Price, CAPEX, Exchange Rate, Cash Cost and Discount Rate to assess the impact of these variables on the NPV, IRR or AISC. Based on the results of the Sensitivity Analysis the project profitability is most affected by the gold price and exchange rate.

GE21 understands that the Almas Project is economically viable and attractive based on these results.

The main risk associated to the economic model results are:

- Financial risk – price: There is a low risk regarding the gold price used reflected by the current consensus gold price applied to the project. Exchange rate can affect the ratio of Price/Cost as well;
- Financial risk – fiscal benefits: There is a low risk regarding the fiscal benefits applied to the project, since not all of them have been granted yet.

23 ADJACENT PROPERTIES

Several other exploration companies maintain land positions near Aura's Almas Gold Project area (Figure 23-1). Recent activity on these adjacent properties has been low owing to their early stage of exploration. As of October 2020, adjacent properties are held by Calango Exploração Mineral S.A., Iamgold-Brasil, M & J Mineração, Mineração Santo Expedito, and many other small companies.

23.1 ADJACENT EXPLORATION PROPERTIES

Calango Exploração Mineral S.A. holds several properties in the Almas Gold Project area, especially to North and East of the town of Almas. The nature of the exploration work conducted by Calango Exploração Mineral S.A. on their holdings is not known to Aura. Currently, Calango Exploração is not active in the district.

Many smaller companies hold mineral exploration licenses in the district, primarily for gold and some for iron ore exploration.

Mineração Santo Expedito also holds 06 blocks of exploration licenses around the Almas Gold Project but currently is not active in the district.

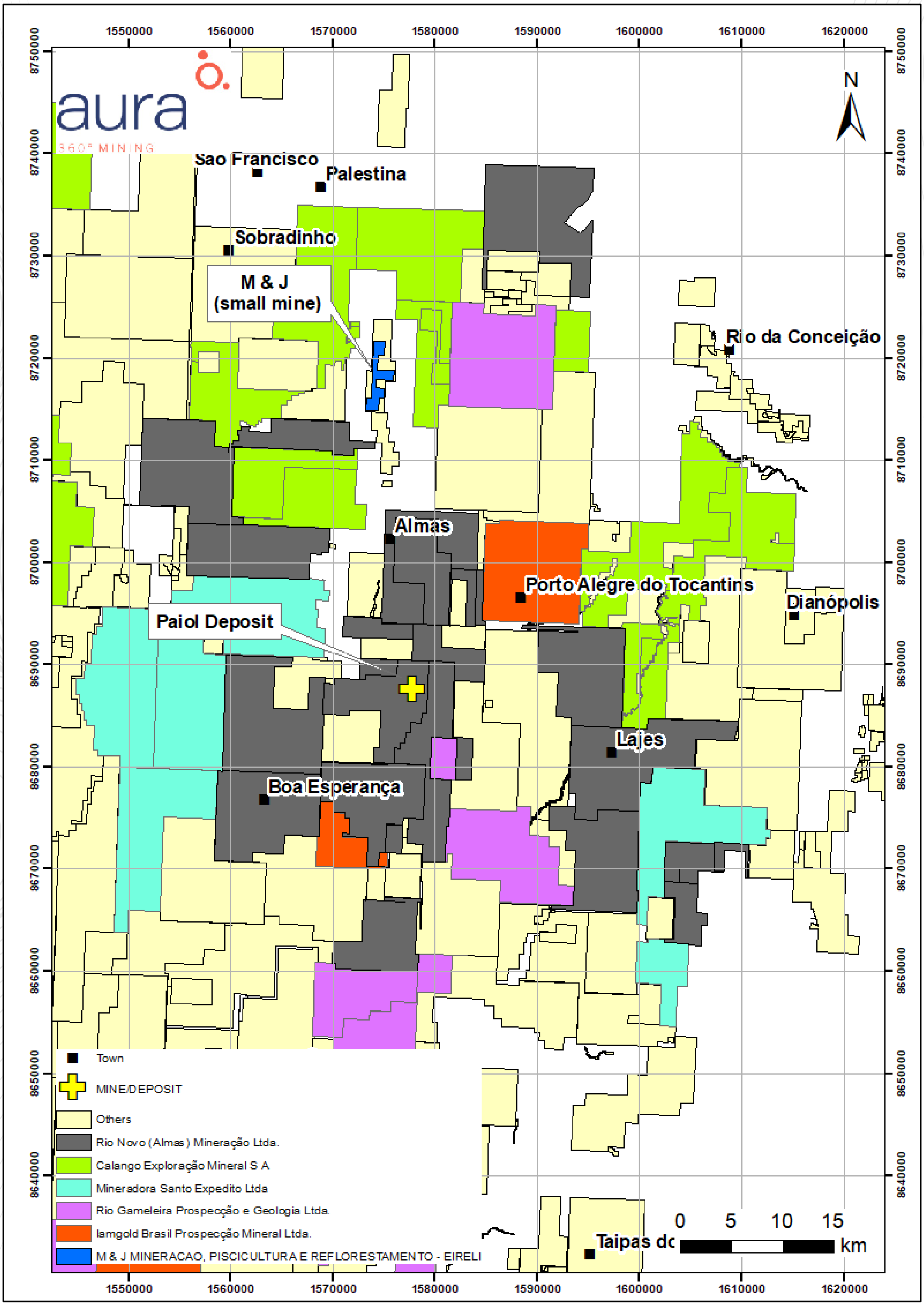
Iamgold-Brasil is currently inactive in the district but still maintains some isolated exploration licenses around the towns of Boa Esperança and Porto Alegre do Tocantins.

23.2 ADJACENT OPERATING PROPERTIES

M & J's Mineração (former Amarante Mineração) operates a small surface mining operation for gold, approximately 15 km north of the Almas town site. The operation includes several small open pits, a small oxide mill with cyanide leach plant, and a tailings disposal facility. Mining utilizes medium-sized, tracked excavators and trucks to excavate mineralized saprolite – a clay-rich, deeply weathered material. Trucks haul the ore to a central processing facility, 5 to 10 km distant from the pits. The capacity of the operation is not known but could be on the order of 100 tpd.

In addition to M & J's mine, a series of small artisanal mines exist along the length of the Almas Greenstone Belt in the Project area. These are generally surface or shallow underground operations, mining saprolite or quartz veins in weathered rock, operated by a few miners, most often, intermittently.

Figure 23-1 Mineral Properties Adjacent to Aura Properties



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 PERMITS AND LAND

The Paiol mine is at an advanced stage of permitting and most of them have already been granted and are under analysis by NATURATINS. The Environmental Assessment (EA) and other studies for the permitting are in accordance with Federal and State Standards and Regulations.

The Environmental Assessment (EA) for the permitting of the Cata Funda and Vira Saia deposits will start in 2021 and, since they will only be implemented in 2025 and 2027, there is time to carry out all the baseline and environmental permitting studies for the expected implementation deadlines. As the project is close to the Almas Town, assessment on air quality impacts and mitigation measures must be addressed with more attention within the scope of Environmental Assessment (EA).

The land use agreements for the Almas Gold Project are at an advanced stage of negotiation. The agreements for the tailings dam and water catchment are already concluded. The agreement for the construction of the metallurgical plant and Paiol mine, are advanced with the government of Tocantins. All legal procedures have been formalized and are under analysis by the Government of Tocantins for signature.

Negotiations for the Cata Funda and Vira Saia mines will begin in the second half of 2021, as these mines will start operating in 2025 and 2027 respectively.

25.2 GEOLOGY AND MINERAL RESOURCES

The deposit type and mineralization model are well understood. Although deposits are structurally controlled, but do not have strong complexity due to shearing. The ore and waste can be easily identified often by color especially in Paiol and Cata Funda where they are hosted by variety of schist units. Continuity, compared to many structural controlled deposits, is relatively good. The drilling is densely spaced and adequate for a resource estimate at the level of a Feasibility Study. The QA/QC protocols for the latest drill campaigns by Rio Novo are well designed and executed giving good confidence in the assays results even at relatively low gold levels. The classification of the resources is adequate for a Feasibility Study. There is minimal risk associated with the resource statement and if there is any, is related to local variation of tonnages and grade. Local variation of tonnages perhaps will be more evident through mine operation as the mine will produce a considerable volume of low grade materials from halo around shear zones which are excluded from ore models.

Table 25-1 shows the Measured and Indicated mineral resources which were constrained by respective optimized pits in different cut-off grades

*Table 25-1 Almas Gold Project Mineral Resources **

ALMAS MINERAL RESOURCE		Tonnes	Au (g/t)	Au (Oz)
PAIOL	MEASURED (M)	4,366,950	1.03	144,870
	INDICATED (I)	13,181,190	0.96	407,590
	M&I	17,548,140	0.98	552,460
CATA FUNDA	MEASURED(M)	482,000	1.97	30,540

	INDICATED (I)	356,000	1.39	15,920
	M&I	838,000	1.72	46,460
VIRA SAIA	MEASURED(M)	566,910	1.24	22,600
	INDICATED(I)	2,787,780	0.91	81,245
	M&I	3,354,690	0.96	103,845
Heap Leach Pad (HLP)	INDICATED (I)	1,510,090	0.88	42,680
GRAND TOTAL (M&I)		23,250,920	1.00	745,445

*Note:

1. The Mineral Resource estimates were prepared in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by the CIM Council on May 10, 2014, and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, adopted by CIM Council on November 29, 2019, using geostatistical and/or classical methods, plus economic and mining parameters appropriate to the deposit.

2. The Mineral Resource Estimate is based on an updated optimized shell using 1800 $/oz gold price and cut-off grades of 0.29 g/t, 0.34 g/t and 0.31 g/t for Paiol, Cata Funda and Vira Saia respectively.

3. Contained metal figures may not add due to rounding.

4. Surface topography based on December 31st, 2016.

5. Mineral Resources are inclusive of Mineral Reserves.

6. The Mineral Resource estimate for the Cata Funda deposit was prepared by Adam Wheeler, C.Eng., a Qualified Person as that term is defined in NI 43-101.

7. The Mineral Resource estimate for the Paiol and Vira Saia deposits and HLP were prepared Farshid Ghazanfari, P.Geo., a Qualified Person as that term is defined in NI 43-101.

25.3 MINING AND MINERAL RESERVES

The pit optimization parameters are based on the Rio Novo mining concept and updated assumptions definedby EDEM Engenharia together with Aura Minerals. The cut-off grade was calculated considering mining dilution, processing costs, G&A costs, reclamation costs, metallurgical recovery, metal price and royalties.

EDEM is of the opinion that the Almas Gold Project has a low risk due to robust mineral resources and reserves. The mineral reserve estimate applied the best practices guidelines and mining knowledge from existing goldmines operated by Aura Minerals with similar settings which will thus reduce the risk of mining the deposits.

The Mineral Reserves estimates are reported by Proven and Probable categories in Table 25-2

*Table 25-2 Almas Gold Project Mineral Reserves Summary **

ALMAS RESERVE		Tonnes	Au (g/t)	Au (Oz)
PAIOL	PROVEN	5,357,974	0.89	152,683
	PROBABLE	10,780,501	0.88	304,446
	TOTAL	16,138,475	0.88	457,129
CATA FUNDA	PROVEN	438,612	1.89	26,711
	PROBABLE	250,163	1.79	14,412
	TOTAL	688,775	1.86	41,123
VIRA SAIA	PROVEN	646,016	0.88	18,363
	PROBABLE	3,134,066	0.91	91,758
	TOTAL	3,780,082	0.91	110,122
GRAND TOTAL		20,607,332	0.92	608,373

HEAP LEACH STOCKPILE	PROVEN	-	-	-
	PROBABLE	1,275,233	0.90	36,900
	TOTAL	**1,275,233**	**0.90**	**36,900**

*Note:

1. The Mineral Reserve estimates were prepared in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by the CIM Council on May 10, 2014, and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, adopted by CIM Council on November 29, 2019, using economic and mining parameters appropriate to the deposit.

2. The Mineral Reserve Estimate is based on an updated optimized shell using 1500 $/oz gold price, average dilution of 20% , mining recovery of 100% and break-even cut off grades of 0.29 g/t Au for Paiol, 0.31 g/t Au for Vira Saia and 0.34 g/t Au for Cata Funda.

3. Contained metal figures may not add due to rounding.

4. Surface topography based on December 31st, 2016.

5. Mineral Reserve estimate for Almas Gold Project was prepared under the supervision of Luiz Pignatari, P.Eng. as a Qualified Person, competent to sign as defined by NI 43-101.

6. The ore located at the heap leach pad is classified as Probable Reserve to reflect metallurgical uncertainties related to the long period of time that this material has remained stockpiled.

The mine schedule achieved a production target of 1.3 Mtpa with a maximum annual rock movement of 19.4 Mtpa. A variable cut-off grade strategy was implemented thereby the high grades are mined in the early periods while leaving the low grades for the end of the mining sequence. The LOM sequence encompasses a 9-month pre-stripping phase at Paiol followed by 13 years of primary mining and, finally, 5 years of re-handling the low grade ore.

The contracted mining fleet involves small backhoe excavators (74-t op. weight) coupled with on-road mining trucks (22 m3 capacity). The materials will be drilled by top-hammer drill rigs in 10-m and 5-m benches.

25.4 METALLURGY AND PROCESSING

The Almas Gold Project samples selected for metallurgical testing represented various ore types and lithologies within the different deposits. In addition, an overall composite representing the first three years of operation has been tested.

Metallurgical testwork completed on the project included a comminution study; gravity recoverable gold and gravity separation tests; evaluation of bulk sulphide flotation; cyanide leaching in the CIL and CIP circuit configurations, cyanide destruction, review of potential for deleterious elements; and solid-liquid separation testing.

The projected average overall recovery for the individual ore types tested was in the range of 93-95% and for the 3-Year Composite – 93%. The selected process design criteria included overall gold recovery of 92.5% at a grind of k80 = 75 microns.

The plant will process material at a rate of 1.3 Mt/a with a design maximum head grade of 1.31 g/t Au to produce doré. The process plant flowsheet designs were based on testwork results and industry-standard practices. The flowsheet was developed for optimum recovery while minimizing capital expenditure and life of mine operating costs. The process methods are conventional to the industry. The comminution and recovery processes are widely used with no significant elements of technological innovation.

25.5 CAPITAL AND OPERATING COSTS

Capital and operating costs were estimated to the extent required for a feasibility study. Total initial project capital costs total US$72.8M and operating costs are $20.68/t processed or an average annual expenditure of US$26.24M.

25.6 INFRASTRUCTURE

The existing infrastructure for the Almas project meets operational requirements. Water supply will be provided from the Manuel Alves River. The Almas Gold Project infrastructure includes the required access, power supply, water supply, tailings storage and support facilities to support 1.3 Mtpa of ore production.

The site water balance shows the maximum requirements of make-up water will be on the order of 140 m^3/h, so the maximum need for new water will be around 185 m^3/h which is the licensing flow (outorga) requested from NATURATINS, the Tocantins State Environmental Agency.

25.7 MARKET STUDIES AND ECONOMIC ANALYSIS

GE21 analysed the market of gold study prepared by World Bank and concluded 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.

The value of US$ 1,558/oz. was adopted for the FS economic model that was developed by GE21 to support the evaluation of potential options and develop an optimal path forward for the Project.

The economic model was developed on an Excel spreadsheet-based on a financial model composed of several worksheets. Contributions from Ausenco, EY and Edem were developed using independent spreadsheets and interfaced into Aura's economic model.

Based on the assumptions adopted, the post-tax Net Present Value ("NPV") of Aura Minerals Gold Almas Project base case achieved US$183M, at 5% discount rate. The Internal Rate of Return ("IRR") reached 43.9% and the annual average EBITDA (from year 1 to year 16, at full production capacity) is US$27M. Simple payback after the start-up of operations is 2.0 years. The results of economic analysis are summarized in Table 25-3.

On analysis of the pre-tax cash flow, NPV rises to US$231Mand the IRR to 53.5%. GE21 understands that Almas Project is economically viable and attractive based on these results.

Table 25-3 Financial Results Summary (Post tax)

ITEM	UNIT	VALUE
DISCOUNT RATE (WACC)	**%**	**5.0%**
NET PRESENT VALUE – NPV	**US$M**	**182.7**
CAPEX NPV	US$M	(87.8)
Operational NPV	US$M	270.5
PROJECT IRR	**%**	**43.9%**
PROJECT PROFITABILITY INDEX		**3.1**
DISCOUNTED PROJECT PAYBACK	**Years**	**3.7**
SIMPLE PAYBACK (including start-up)	**Years**	**3.5**
SIMPLE PAYBACK (after start-up)	**Years**	**2.0**

26 RECOMMENDATION

26.1 EXPLORATION

The Almas Gold Project is hosted in a prolific greenstone belt in Tocantins state. The mineralized belts extend south of the town of Almas for up to 20 km. There are several targets within the current Almas Gold Project claims that can be explored within 20 km strike length. These targets are described in section of 7 of this report. The targets are in different level of exploration, but they can advance quickly and become a source of ore in the future for the Almas Gold Project.

The most advanced exploration target is Moro de Carneiro with several holes drilled on the property.

Morro do Carneiro is located 700 m east of Cata Funda and a preliminary review suggests the target has the most immediate potential to increase the resource base of the Almas Gold Project by adding a small deposit possibly with a little better grade and lower strip ratio.

Mineralization at Moro de Carneiro is hosted within chemical sediments which are possibly developed on top of Archean basement strata. Additional drilling, developing a resource and infill drilling is strongly recommended. Metallurgical work on this target is important to better understand the gold liberation due to its higher sulfide content.

Historically positive drilling results in the Almas Belt (Fofoca, Refresco, Vieira and Olavo targets) has the potential to be followed up with more drilling to test the continuity and grade.

There are several earlier stage targets with significant chip sample, soil and or stream sediment anomalies that were not tested by drilling.

In terms of grass root exploration, stream sediment sampling and geological reconnaissance is suggested with rock sampling to cover the southern properties around Lajes city (864416/2011; 864417/2011; 864041/2013; 864002/2018; 864003/2018 and 864005/2018) and the southeast and northwest properties around Boa Esperança city (864226/2015; 864026/2015), both regions have good history of garimpo activity, as well as the western properties around Almas city (864014/2013 and 864015/2013).

In addition, regional soil sampling on those properties will give a clear perspective for targeting geological mapping, followed by ground geophysical surveys testing the best anomalous zones. Surveys with electro-resistivity and time-domain IP, are recommended to orient exploration drilling in the zones with intermediate to high chargeability anomalies related to high values of apparent resistivity.

Among the properties with historical surface sampling, such as north of Lajes city (864004/2018) and northwest of Almas City (864415/2011) are suggested for regional soil sampling followed by the same steps of geological mapping, ground geophysical surveys and exploration drilling.

Table 26-1 shows recommended exploration budget for next 2 years for Almas Project.

Table 26-1 Almas Recommended Exploration Budget

ALMAS	TYPE	ESTIMATED COST (US$)	DD DRILLING (m)
Generate and develop new targets in the district - Geological Mapping, soil and chip sampling (preparation for drilling) - mainly on exploration concessions that are close to expiration date - 2021	EARLY STAGE / CONCESSION HOLDING	300,000	-
Delineation and extension drilling at Morro do Carneiro Target (possibility to add small higher-grade deposit to the resource base)	ADVANCED	1,320,000	3,000
Exploration drilling at Riacho do Carneiro meta-chert unit towards south (example Espinheiro Target) and step out drilling of specific historical positive drill holes along the Paiol - Cata Funda trend	EARLY STAGE	1,080,000	3,000
TOTAL		**2,700,000**	**6,000**

26.2 GEOLOGY & ADJACENT MINERAL RESOURCES

Additional infill drilling is required to convert more resources both in Paiol and Vira Saia from inferred to M&I categories. Multiple narrow shear zones can be identified in HW and FW for all three deposits. Additional infill drilling can delineate and test the continuity of these splay shear structures and related ore shoots.

At Vira Saia, the deposit model is extended to the northwest and it is recommended to convert this to the Measured and Indicated category with an infill drilling plan after production ramp up. The deposit is open to the south and north and is not completely drilled off. Further exploration may be justified to drill off the deposit.

At Paiol, the deposit narrows down toward the south but is open towards north with multiple shearing targets. Additional infill drilling will probably delineate more ounces which are not modelled and estimated in the current feasibility study. The northern part of deposit had more ounces due to the presence of high grade where there is a chance of finding more mineralized zones.

High grades at depth, despite some previous studies, does not have enough grade continuity to support any underground development and further drilling at depth is not recommended. The author believes that any further infill drilling in the near future needs to be done within the resource pit shell outline from the current feasibility study for the foreseeable future.

After de-watering of the Paiol pit, a pit geology map needs to be prepared and structures identified during core-logging need to be tied up to shear structures on the surface to better constrain the existing deposit model.

Existing lithological and alteration databases need to be reviewed and revised, and refined lithological models need to be established. Future resource estimates need to consider these updated litho-alteration models for all three deposits.

In terms of the Paiol Heap Leach pad, more drilling is required to do properly sample for the purpose of additional metallurgical studies and grain size analysis.

26.3 MINERAL RESERVE ESTIMATION

- EDEM considers that the geothecnical information is at this stage sufficient to start the operation. However, it is crucial that the operation count on an experienced dedicated geotecnical team to assure a good monitoring geotechnical program to give good support to the operation and eventually revise the Paiol pit design accordingly, after third and fourth year operation.

- EDEM recommends a maximum two pits operating simultaneously otherwise the operation it is going to be more complex and certanily costs would increased.
- EDEM considers that a single low grade ore stockpile near the processing plant it is a better strategy in terms of logistics and simplifies the low grade ore pile re-handling after 2031.
- Expand the geological investigation to increase the life of mine and thus provide additional ore feed to the processing after 2030.
- The heap leach pad schedule must strictly follow the mine planning and, if possible, try to anticipate it. The area now occupied by the heap leach pad overlaps with Waste Dump 1. Delays may force mine planners to anticipate waste haulage from the Paiol pit to the Waste Dump 2.
- Current permitted areas for waste dumps are sufficient to accommodate two waste dumps, However, the operation will be benefited if an additional area is included on east side of Paiol mine. This area for waste dump would likely intail cost reduction benefits. A trade-off study comparing an investment on a new area versus operational cost reductions is recommended.
- The final pit went a little beyond the area limit of Apoena in the pit northeast part. This would occur only in the final operation of the Paiol mine. It is recommended the completion of a trade-off to check the feasibility to add a small area in this region.
- During Vale's operation, most probably, they used to have higher cut-off grade than the 0.29 g/t Au applied to Paiol's present reserve calculation. It would be recommended that geology personnel double check the material during the partial old waste dump pile removal, which is scheduled in the present mining plan, to check for the possible presence of ore.
- A significant increase in waste removal is scheduled after year 2026. It is recommended further investigations that determines the best way to continue with the operation afterwards .
- The operation needs to follow the gold price evolution once it is planned. Due to a significative waste removal increase after the fourth year, a revision of the project is recommended to save money in the future considering lower gold prices.

26.4 MINING METHOD

- From a management point of view it not recommended to have more than two pits operating simultaneously.
- EDEM recommends grade control drilling with Down-The-Hole reverse circulation drills to support the grade control engineering.
- The pumping capacity of 150 m^3/h to dewater the Paol pit provides little time for grade control engineering to feed the plant at the start-up, according to the simulations. The grade control drilling with a DTH drill should be completed upfront to have the results ready for modeling and mine planning of the plant feed. It would help to try to increase the pumping capacity or start doing the drilling for grade control engineering with regular exploration drills with integral core sampling and diamond rotary bits. The grade control engineering to prepare the comissioning must be 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.
- We recommend 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 loading trucks operation. Alternatively, but not as efficiently , we are considering a buldozer per pit, similar to a CAT D6, because it is easier to find them in the Brazilian market.
- As transport is the most expensive mining operation task, it is recomended that the operation have extra capacity for the loading fleet and all the related equipment be kept in a good transport operational condition.
- An eletronic dispatch system is highly recomended for all operational control.

26.5 MINERAL PROCESSING AND METALLURGICAL TESTING

- Additional testwork should be considered to define the geometallurgical sample variability in more detail.

26.6 RECOVERY METHODS

- Additional leach testing is recommended to optimize leach conditions and cyanide consumption. Additional continuous cyanide detoxification tests are recommended to optimize retention time and reagent additional rates.

25.6 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITIES IMPACTS

- Continuous monitoring of the renewal schedule for the Installation License- LI for the Paiol Mine, and other permits with the environmental regulatory body - NATURATINS so there are no delays in the issuance of the environmental permits;
- Elaboration of a Physical Financial Schedule for the implementation of all Social Environmental Plans and Programs is recommended so they are able to commence as soon as the Installation License is renewed;
- Priority is given to programs that present a social scope, such as Updating the Social Diagnosis, Mapping Stakeholders, Social Management Plan, Social Communication Program and Defining Partnerships with the communities affected by the Almas Gold Project.
- To avoid significant economic impacts of mine closure in the future, the plan should be updated throughout the operation, at least five years before closure, including a detailed social management plan, community ongoing consultation and measures to mitigate the economic and social effects of mine closure.

SIGNATURE PAGE

Updated Feasibility Study Technical Report (NI 43-101) for the Almas Gold Project, Almas Municipality, Tocantins, Brazil

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28 CERTIFICATES

CERTIFICATE OF QUALIFIED PERSON

B. TERRENCE HENNESSEY, P.GEO.

As an author of this report titled "Updated Feasibility Study Technical Report (NI 43-101) for the Almas Gold Project, Almas Municipality, Tocantins, Brazil" dated March 10, 2021, with an effective date of December 31, 2020 (the "Technical Report"), I, B. Terrence Hennessey, P.Geo., do hereby certify that:

1. I am a subconsultant to, and carried out this assignment for Micon International Limited, 900 - 390 Bay Street, Toronto, Ontario M5H 2Y2, Tel.: (416) 362-5135; Fax: (416) 362-5763, e-mail: thennessey@micon-international.com
2. I hold the following academic qualifications:
3. B.Sc. (Geology) McMaster University 1978
4. I am a registered Professional Geoscientist with Professional Geoscientists Ontario (membership #0038).
5. I have worked as a geologist in the minerals industry for over 40 years.
6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and, by reason of my education, past relevant work experience and affiliation with a professional association, fulfill the requirements to be a Qualified Person for the purposes of NI 43-101. My work experience includes 7 years as an exploration geologist looking for iron ore, gold, base metal and tin deposits, more than 10 years as a mine geologist in both open-pit and underground mines and 24 years as a consulting geologist working in precious, ferrous and base metals as well as industrial minerals.
7. I have visited the Almas Gold Project in Tocantins State Brazil and Aura's core logging facility and warehouse in the town of Almas on May 3 and 4, 2019.
8. I am responsible for Sections 11 and 12 and summaries therefrom in Sections 1, 25 and 26, of this Technical Report.
9. I am independent of Aura Minerals Inc., as defined in Section 1.5 of NI 43-101.
10. I have not previously worked on the project that is the subject of the Technical Report.
11. I have read NI 43-101 and Form 43-101F1 and the portions of this report for which I am responsible have been prepared in compliance with that instrument and form.
12. 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: March 10, 2021

Effective Date: December 31, 2020;

"B. Terrence Hennessey" {signed and sealed}

B. Terrence Hennessey, P.Geo.

FARSHID GHAZANFARI, P.GEO.

As an author of this report titled “Updated Feasibility Study Technical Report (NI 43-101) for the Almas Gold Project, Almas Municipality, Tocantins, Brazil” dated March 10, 2021, with an effective date of December 31, 2020 (the “Technical Report”), 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 2, 3, 4 , 5, 6, 7, 8 , 9, 10, 14, 23, 24 and summaries there from in sections 1 and 25 of the technical report entitled “Updated Feasibility Study Technical Report (NI 43-101) for the Almas Gold Project, Almas Municipality, Tocantins, Brazil”
6. I visited the Almas Gold Project in Tocantins State Brazil and Aura’s core logging facility in many occasions between 2017 to 2019 and my last visit was between June 10 to 14 2019.
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: March 10, 2021

Effective Date: December 31, 2020

“Farshid Ghazanfari” {signed and sealed}

Farshid Ghazanfari, P.Geo.

INNA DYMOV, P. ENG.

747 Fortye Drive, Peterborough, ON, Canada, K9K 2G4

1. I, Inna Dymov, P. Eng., am employed as an independent senior consultant (metallurgy) and previously employed at SGS Minerals, Canada Inc. as a Gold Metallurgy department manager.
2. This certificate applies to the technical report titled "Updated Feasibility Study Technical Report (NI 43-101) for the Almas Gold Project, Almas Municipality, Tocantins, Brazil" dated March 10, 2021, with an effective date of December 31, 2020 (the "Technical Report").
3. I am a member of Professional Engineers Ontario, license number 90193236. I graduated from the Moscow Technical University with a Master of Engineering degree in Mineral Processing Engineering in 1979.
4. I have practiced in this profession for 34 years. I have been directly involved with the metallurgical testwork programs for process flowsheet development at the feasibility level studies for numerous gold projects in Canada and international projects.
5. As a result of my qualifications, experience, and association with a professional association, I am a Qualified Person as defined in National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).
6. I have not visited the Almas Gold Project.
7. I am responsible for Section 13 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 Almas Gold Project since 2018, during the preparation of the feasibility 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: March 10, 2021

Effective Date: December 31, 2020

"Inna Dymov" {signed and sealed}

Inna Dymov, P. Eng.

PORFIRIO CABALEIRO RODRIGUEZ, FAIG.

I, Porfirio Cabaleiro Rodriguez, FAIG, (#3708), as an author of the technical report titled "Updated Feasibility Study Technical Report (NI 43-101) for the Almas Gold Project, Almas Municipality, Tocantins, Brazil" dated March 10, 2021, with an effective date of December 31, 2020 (the "Technical Report")), prepared for Aura Minerals. (the "Issuer"), do hereby certify that:

1. I am a Mining Engineer and Director for GE21 Consultoria Mineral Ltda., which is located on Avenida Afonso Pena, 3130, 12th floor, Savassi, Belo Horizonte, MG, Brazil - CEP 30130-910.
2. I am a graduate of the Federal University of Minas Gerais, located in Belo Horizonte, Brazil, and hold a Bachelor of Science Degree in Mining Engineering (1978). I have practised my profession continuously since 1979.
3. I am a Professional enrolled with the Australian Institute of Geoscientists ("AIG") - ("FAIG") #3708.
4. I am a professional Mining Engineer, with more than 40 years' relevant experience in Mineral Resource and Mineral Reserves estimation, which includes numerous mineral properties in Brazil, including gold properties.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association as defined in NI 43-101, and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible for Chapter 22. I am also responsible for the corresponding sections within Chapters 1, 25 and 26 that are related to the foregoing Chapter of this Technical Report.
7. I am independent of Aura Minerals Inc., as defined in Section 1.5 of NI 43-101.
8. I have not previously worked on the project that is the subject of the Technical Report..
9. As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I have authored and am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I have no personal knowledge, as of the date of this certificate, of any material fact or material change which is not reflected in this Technical Report.
11. I am independent of the Issuer, applying all the tests in section 1.5 of NI 43-101.
12. I have read NI 43-101 and Form 43-101F1 – Technical Report, and the Technical Report has been prepared in compliance with such instrument and form.

Signing Date: March 10, 2021

Effective Date: December 31, 2020

"Porfirio Cabaleiro" {signed and sealed}

Porfirio Cabaleiro Rodriguez, FAIG.

TOMMASO ROBERTO RAPONI, P.GEO.

As an author of this report titled "Updated Feasibility Study Technical Report (NI 43-101) for the Almas Gold Project, Almas Municipality, Tocantins, Brazil" dated March 10, 2021, with an effective date of December 31, 2020 (the "Technical Report"), I, Tommaso Roberto Raponi, P.Eng., do hereby certify that:

1. I am employed as a Principal Metallurgist with Ausenco Engineering Canada Inc. with a business address at 11 King Street West, Suite 1550, Toronto, ON CA M5H 4C7.
2. I am a graduate of University of Toronto with a BASc in Geological Engineering, 1984.
3. I am registered as a Professional Engineer in the Province of Ontario (Reg. No. 90225970) and the Association of Professional Engineers and Geoscientists of BC (Reg. No. 23536).
4. I have worked as an independent consultant since 2016. My relevant experience is over 36 years of experience in the development, design, operation and commissioning of mineral processing plants, focusing on gold projects.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and, by reason of my education, past relevant work experience and affiliation with a professional association, fulfill the requirements to be a Qualified Person for the purposes of NI 43-101.
6. I have not personally inspected the property.
7. I am responsible for Sections 1.9, 1.10, 1.13, 17, 18.1 to 18.5 inclusive (excluding 18.4.11 and 18.4.12), 18.7.1 ,18.7.2, 18.7.3, 18.7.4, 21.2.1; 21.3.1, 21.3.1.2, 21.3.3 and 25.6 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 not previously worked on the project that is the subject of the Technical Report.
10. I have read NI 43-101 and Form 43-101F1 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 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: March 10, 2021

Effective Date: December 31, 2020

"Tommaso Roberto Raponi" {signed and sealed}

Tommaso Roberto Raponi, P.Eng.

LUIZ EDUARDO PIGNATARI, P.ENG.

As an author of this report titled "Updated Feasibility Study Technical Report (NI 43-101) for the Almas Gold Project, Almas Municipality, Tocantins, Brazil" dated March 10, 2021, with an effective date of December 31, 2020 (the "Technical Report"), I, Luiz Eduardo Pignatari, 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 Almas Gold Project in Tocantins State Brazil and Aura's core logging facility and warehouse in the town of Almas on June 6 and 8, 2017.
6. I am responsible for Sections 15, 16 and 20 of this Technical Report and summaries therefrom in Section 1.
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: March 10, 2021

Effective Date: December 31, 2020

"Luiz Eduardo Pignatari" {signed and sealed}

Luiz Eduardo Pignatari, P.Eng.

ADAM WHEELER, C. ENG.

I, Adam Wheeler, C. Eng., do hereby certify that:

1. I am an independent mining consultant, based at Cambrose Farm, Redruth, Cornwall, TR16 4HT, England.
2. I hold the following academic qualifications:

   B.Sc. (Mining) Camborne School of Mines 1981

   M.Sc. (Mining Engineering) Queen's University (Canada) 1982
3. I am a registered Chartered Engineer (C. Eng. and Eur. Ing) with the Engineering Council (UK). Reg. no. 371572.
4. I am a member in good standing of the Institute of Materials, Minerals and Mining (Fellow).
5. I have worked as a mining engineer in the minerals industry for over 38 years. I have experience with a wide variety of mineral deposits and reserve estimation techniques.
6. I have read NI 43-101 and the technical report, which is the subject of this certificate, has been prepared in compliance with NI 43-101. By reason of my education, experience and professional registration, I fulfil the requirements of a "qualified person" as defined by NI 43-101. My work experience includes 5 years as a mining engineer in an underground gold mine, 7 years as a mining engineer in the development and application of mining and geological software, and 26 years as an independent mining consultant, involved with evaluation and planning projects for both open pit and underground mines.
7. I am responsible for portion of section 14 related to Mineral Resource Estimate of Cata Funda deposit.
8. As of the date hereof, to the best of my knowledge, information and belief, the technical report, which is the subject of this certificate, contains all scientific and technical information that is required to be disclosed to make such technical report not misleading.
9. I am independent of Aura Minerals Inc and its subsidiaries other than providing consulting services.
10. I consent to the filing of the report with any Canadian stock exchange or securities regulatory authority, and any publication by them of the report.

Signing Date: March 10, 2021

Effective Date: December 31, 2020

"Adam Wheeler" {signed and sealed}

Adam Wheeler, C.Eng.