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FEASIBILITY STUDY AND TECHNICAL REPORT ON THE EPP PROJECT MATO GROSSO, BRAZIL

15°20'S LATITUDE AND 59°16' W LONGITUDE

FOR

AURA MINERALS INC.

By

P & E Mining Consultants Inc.

NI 43-101 & 43-101F1
TECHNICAL REPORT
Report No. 316

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1.0 SUMMARY

1.1 INTRODUCTION

This report titled “Feasibility Study and Technical Report on the EPP Project, Mato Grosso, Brazil” (“Report” or “Technical Report”), was prepared to provide Aura Minerals Inc. (“Aura” or the “Company”) with a National Instrument 43-101 Standards of Disclosure for Mineral Projects (“NI 43-101”) Technical Report on the Ernesto/Lavrinha/Pau-a-Pique Deposits (“EPP Project” or “Project” or “Property”), located in the southwest of Mato Grosso state, near Pontes e Lacerda in Brazil. The EPP Project is 100% beneficially owned by Aura. Aura is a public company listed on the TSX, under the symbol “ORA”.

Aura, through its Brazilian subsidiaries, acquired the EPP Project from Yamana Gold Inc. (“Yamana”) in June 2016. The Project was initially studied by Yamana from 2009 to 2011, and was put into production in early 2013 until being placed on care and maintenance in late 2014.

The EPP Project is the third gold mining operation owned by Aura in this specific region of Brazil. The Company owns the operating Sao Francisco gold mine (in production since 2006) near the town of Pontes e Lacerda and owned the Sao Vicente gold mine that ceased operations in 2014 (production since 2009).

The EPP Project consists of three deposits, two that have been planned to be mined as underground operations and the third which is planned as an open pit operation. Three additional areas will be evaluated in 2017 and 2018.

- The Lavrinha open pit and the Ernesto underground deposit are located approximately 60 kilometres ("km") south of the Company's Sao Francisco mine and 12 km south of the town of Pontes e Lacerda. The Project's process plant is located at Ernesto.
- The Pau-a-Pique underground deposit is located approximately 47 km south of the Ernesto and Lavrinha deposits and process plant.
- Three exploration areas (Nosde, Japones and Pombihnas) are within 5 km of the process plant.

This Report supports a systematic sequence to launch three gold mines starting with the Lavrinha open pit gold deposit, followed by the re-start of the Pau-a-Pique underground gold deposit and subsequently the development and production of the Ernesto underground gold deposit.

The purpose of this Report is to provide a NI 43-101 Feasibility Study and Technical Report (“the Report”) on the EPP Project. P&E understands that the Company may use this Report for internal decision making purposes and will be filed as required under applicable Canadian securities laws. The Report may also be used by the Company to support financings.

The current P&E Updated Mineral Resource Estimate presented in this Report has been prepared in full conformance and compliance with the “CIM Standards on Mineral Resources and Reserves – Definitions and Guidelines” as referred to in NI 43-101 and Form 43-101F, Standards of Disclosure for Mineral Projects and in force as of the effective date of this Report, which is July 31, 2016.

1.2 LOCATION AND OWNERSHIP

The Ernesto, Lavrinha and Pau-a-Pique gold deposits are near the town of Pontes e Lacerda, approximately 450 km west of Cuiabá, the capital of the Brazilian state of Mato Grosso. The Ernesto Deposit is approximately 12 km southeast of Pontes e Lacerda.

The Ernesto Property comprises 1,412.89 ha of 6 mining rights held (legally or beneficially) by Mineração Apoena S.A. ("Apoena"), a company wholly-owned by Aura.

On April 30, 2015, Aura announced its agreement with Serra da Borda Mineração e Metalurgia ("SBMM"), a company affiliated with Yamana, to acquire, upon completion of certain conditions, the assets and liabilities of the Project. On June 23, 2016, the Company announced that it had completed the acquisition and has assumed operation control of the Project.

Aura provided a letter dated July 31, 2016, from Ryan Goodman, VP of Legal Affairs for Aura, which states that Apoena is a wholly-owned subsidiary of the Company.

As part of the acquisition, a 2% NSR royalty is payable to Yamana on gold ounces produced from the Project with respect to up to 1,000,000 collective ounces of gold, and thereafter, a 1% NSR on gold ounces produced from the Project.

A 0.5% NSR royalty is due to each landowner (one for Ernesto/Lavrinha, and one for Pau-a--Pique), proportional to the landowner's surface rights. The Brazilian Mining Code provides that landowners are entitled to a royalty equivalent to 50% of the royalty due the government (the Financial Compensation for Exploitation of Mineral Resources – "CFEM"). The CFEM is calculated based on net income resulting from the sales of the mineral product, deducting taxes and costs of transport and insurance. In the case of gold, the rate of CFEM is 1%, thus the landowner royalty is 0.5%.

1.3 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE

The Ernesto and Lavrinha Properties are contiguous and can be accessed from Pontes e Lacerda by the federal (Brazil) highway BR-174 for 12 km and then following 2 km of gravel and dirt roads that offer year-round access to the Project. The Pau-a-Pique Deposit is approximately 73 km by road from Pontes e Lacerda, and approximately 47 km by dirt road from Ernesto. Pontes e Lacerda is approximately 450 km west of the Mato Grosso state capital of Cuiaba.

The region hosts the hot, tropical and semi-humid climate of the Mato Grosso state in west-central Brazil. The area has two well-defined seasons: one dry winter season, usually from April to October, when the temperature averages 20°C to 22°C, and a wet season that receives large amounts of rain during November to March, with daily temperatures averaging 30°C to 43°C. Average annual precipitation is estimated at 1,440 mm.

The Ernesto Property contains a 130 tonnes per hour carbon-in-leach ("CIL") process plant, which includes crushing, grinding and tailing facilities with power supplied from the national grid via a 12 km 138 kV transmission line from Pontes e Lacerda. The Ernesto Property also contains a gate house, administration offices, core shack, explosives storage facility, and the mined-out Ernesto open pit and waste rock storage area. The Lavrinha Property is contiguous to Ernesto and does not contain any infrastructure. The Pau-a-Pique Property contains an underground mine that was operated by Yamana until late in 2014, and surface facilities for administration and maintenance.

1.4 HISTORY

Gold was first discovered at the Aguapeí Gold Belt by Portuguese settlers in the 18th century, around 1734, and it was mined from primary colluvial, alluvial or placer deposits. The most significant primary gold deposits were discovered at places today known as Sao Francisco Xavier and São Vicente mines, Rio Galera, Santana, Nossa Senhora do Pilar, Aguapeí, Cágado, Santa Bárbara and Lavrinha. Since then, gold mining activities were interrupted due to difficulties in operation and exhaustion of alluvial deposits.

Modern gold mining began in 1984 during a second gold rush at Alto Guapore Gold Province (1984-1997). Artisanal miners, after exhaustion of alluvial and colluvial deposits, discovered several small primary gold deposits close to Pontes e Lacerda, including Japonês, Nosde, Lavrinha, Ernesto (Copacel), Pombinhas and Cantina/Serra Azul deposits.

Approximately 6,000 artisanal miners carried out a large number of small operations (including panning, small underground workings and small scale process plants) around Pontes e Lacerda, Vila Bela da Santíssima Trinidad and Porto Esperidião cities. Gold production data in this period are not accurate, but it is estimated that approximately 5-6 tonnes of gold was produced between 1990 and 1995. In 1992, these artisanal mining activities attracted the attention of several mining companies, including Copacel, Minopar, Anglo American, WMC, Madison do Brasil, TVX Gold/Paulo Abib and Mineração Santa Elina (“MSE”).

Copacel and Minopar, local mining companies, were the first and main owners of exploration permits in the Ernesto District in the early 1990s. In 1992, Anglo American and WMC carried out intensive surface geochemical surveys along the belt, mainly stream sediment sampling. In 1993, Madison do Brasil, after acquisition of exploration permits from Copacel and Minopar, carried out a diamond drilling program at Japones, Nosde, Lavrinha and Ernesto targets. In 1994, Madison do Brasil company assigned its mineral rights and transferred control of the exploration permits to TVX Gold. TVX Gold, in 1995, carried out additional drilling campaigns. In the same year TVX Gold transferred its mineral rights to MSE to capitalize on other business priorities. During this time MSE drilled nine more exploratory drill holes for a total of 1,711.77 m at the Lavrinha deposit and collected 683 samples.

1.5 GEOLOGICAL SETTING AND DEPOSIT TYPES

The Pau-a-Pique and Ernesto-Lavrinha Deposits are situated in the Middle Proterozoic (ca. 1.0 Ga) Aguapeí belt, a foreland fold and thrust belt that overlies the Early Proterozoic and Middle Proterozoic terrains (Geraldes et al. 2001). The Aguapeí group in the Pau-a-Pique and Ernesto – Lavrinha areas is structurally marked by reverse faults, isoclinal folds and strong penetrative axial planar cleavage, often crenulated.

The Aguapeí Group is composed of conglomerate, sandstone (arenite) and siltstone that are unconformably deposited on the underlying basement in a braided fluvial to marine depositional environment. The metasediments occur within a fold and thrust belt that is deformed under brittle-ductile conditions and are commonly in tectonic contact with the basement. Strong hydrothermal alteration and associated gold mineralization occurs in association with the lower contact of the Aguapeí Group with the underlying basement.

In the Ernesto Deposit, the contact zone between the Aguapeí sediments and the underlying basement tonalite consists of a 5 m to 25 m thick magnetite-sericite schist unit, containing lenses and elongated bodies of quartz generally concordant with the foliation, and a 1 m to 3 m thick basal layer of intensely altered, crushed and decomposed rock. The magnetite-sericite schist apparently represents strongly altered and deformed sediment, probably a hydrothermally altered and sheared metapelite (mylonite).

The Lavrinha Deposit which is closely linked to the Ernesto Deposit has been interpreted as gold-rich quartz veins and veinlets with coarse grained pyrite occurring along shallow-dipping structure. The main difference with Ernesto is the position of the mineralization in the metasedimentary sequence. Gold mineralization is located along quartz boudins in highly sericitized rock and plunges to the north.

The Pau-a-Pique Deposit occurs in close association with the contact of the meta-tonalite basement and the overlying Aguapeí Group metasediments. The tonalite is metamorphosed with a foliated structure, but preserving the original igneous texture. The rocks are metamorphosed and deformed under lower green-schist facies conditions. Muscovite schist is developed in the contact between the metatonalites and metasediments and is an important host of mineralization. The muscovite schist has S-C structures and abundant shear bounded sigmoidal veins. The schist has a strong stretching lineation oriented at N20–50W that controls the form of the deposit and sub-surface mineralization.

The Ernesto-Lavrinha Deposits consists of gold-rich quartz veins and veinlets occurring along a relatively thick, shallow-dipping structure at the base of the metasedimentary sequence and within altered sulphidic horizons in overlying meta-arenite units. The basal structure is interpreted to be a low-angle detachment fault that has been folded and faulted together with the overlying stratigraphy. Gold mineralization is located along asymmetrical anticlines and synclines that plunge gently to the north and are cut by NW and NE-trending narrow faults. The gold mineralization occurs in three zones: Lower Trap, Middle Trap and Upper Trap.

The Lower Trap mineralized zone in Ernesto is widely developed within a mylonitic zone. The mylonitic zone is a deformed version of meta-arenite which was altered and intruded by quartz veining. The mylonitic zone often resembles a healed fault zone that developed along detachment structures. Mineralization in the Lower Trap is 130 m to 210 m wide, with an average thickness of 5 m and is more-or-less continuous for at least 1,000 m along its northern plunge direction. Alteration associated with gold mineralization within the mylonitic unit includes abundant quartz veins and veinlets with coarse-grained euhedral pyrite and medium grained bipyramidal crystalline magnetite. This alteration and mineralization occurs in mylonitic zones near the base of the detachment fault.

The Upper Trap, which is widely developed in the Lavrinha Deposit, occurs in metapelitic rocks (hematite sericite schist) in dilation zones of the intensely deformed synclinal troughs. The Upper and Intermediate traps share similar alteration and mineralization suites.

The Ernesto-Lavrinha Deposits are described as detachment-style gold deposits, where typically gold mineralization is associated with low-angle to flat detachment faults, generally with a normal (extensional) sense of movement which consistently places younger units over older units.

The Pau-a-Pique gold mineralization is associated with intense hydrothermal alteration, and correlates with the occurrence of pyrite, sulphide alteration, quartz veins and sericitization. The envelope of the mineralized zone is approximately 550 m long, maximum of 15 m thick and 400 m deep in the largest extension. In the deeper levels the most common hydrothermal alteration with gold enrichment is strong albite-anorthosic quartz veining associated with chloritization and pyrite. In the shallow levels the most pervasive alteration is silicification, represented by a strong injection of quartz veins and weaker gold enrichment. The albitic alteration probably represents deeper and hot sources of the hydrothermal feeder. The Pau-a-Pique Deposit is developed within brecciated-sheared host rocks which are strongly foliated and moderately metamorphosed and can be described as structurally controlled orogenic gold lode deposit.

1.6 EXPLORATION

Both Ernesto and Lavrinha were subject to multiple exploration programs by Yamana from 2003 to 2013. The exploration programs carried out during this period included rock chip sampling, channel sampling, soil sampling, detailed geological mapping and diamond drilling. From 2003 to 2009 drill programs were carried out only on Ernesto's near-mine areas including Lavrinha. From 2009 to 2013 all exploration efforts were focused on the Ernesto District including in-fill drilling of the Lavrinha Deposit. The main goals were to define higher grade mineral resources in the Ernesto near-mine target area, mainly looking for Lavrinha open pit mineral resources.

In 2015 Aura carried out detailed geological mapping of the Lavrinha Deposit focused on outlining geological, mineralized domains and alteration. During the mapping, lack of drill information near the surface extension of the mineralized shoots was identified. Aura drilled 21 diamond drill holes for a total of 997.4 m of drilling, with 845 samples analyzed by gold fire assay at the São Francisco Mine laboratory, with check assays on the mineralized intervals from field duplicates sent to SGS Laboratories.

Exploration in Pau-a-Pique was carried out by Yamana during 2005-2006 including geological, channel sampling and face sampling from mineralized zones that were exposed by Garimpeiros (artisanal miners). Chip sampling was conducted to identify lithology and alteration. A total of 600 chips, soil and trench samples were taken in 2008.

1.7 DRILLING

11,128 m of drilling was conducted on the Ernesto mineral resource area by Yamana in 2005. In 2006, a further 7,777 m of diamond drilling was done on the Property, focusing on targets near the resource area, and included a few exploration holes. Yamana drilled 29 holes totalling 2,820 m in 2009.

In 2015, 3,076.2 m of drilling from 21 holes was conducted on the Ernesto resource area by Aura focusing only on the Lower Trap where resources were deemed to be potentially suitable for an underground operation. From these 21 holes, 15 holes were in-fill drilling to delineate existing resources and 6 other holes were geotechnical holes to assess the geotechnical characteristics of host rocks for a possible underground operation. The in-fill drilling focused on the centre of the Lower Trap deposit where the majority of previous drilling was concentrated and required limited drilling to upgrade Inferred mineral resources to the Indicated category and to provide increased confidence in the resource classification.

Yamana conducted exploration drilling on the Lavrinha Property in 2010 and 2011. 28 drill holes, totalling approximately 5,200 m were advanced surrounding the artisanal mining shafts in order to add mineral resources. In 2013, 55 drill holes totaled 10,013.13 m of diamond drilling, with 9,446 samples analyzed for gold using fire assay at ALS Chemex Laboratories, and 318 bulk density determinations were made.

In 2014, a Yamana drilling campaign at Lavrinha consisted of a total of 78 drill holes for 8,145.11 m of diamond drilling, and 5,916 samples were analyzed by gold fire assay. 48 drill holes for 4,781.31 m and 3,642 samples were analyzed at ALS Chemex Laboratories by Yamana in 2014. The remaining 30 drill holes for 3,363.80 m and 2,274 samples were analyzed by Aura in 2015 at SGS Laboratories.

In 2015 Aura identified a lack of drill information near the surface extension of the Lavrinha mineralization observed in the outcrops, which was not considered in the resource model generated by Yamana. Aura decided to carry out a confirmatory drill campaign to provide better resource definition and improved confidence in estimated grades. The campaign consisted of 21 drill holes and 997.4 m of diamond drilling, with 845 samples analyzed by gold fire assay at the São Francisco Mine laboratory, and checks on the mineralized intervals with field duplicates sent to SGS Laboratories.

Yamana conducted four drilling campaigns on Pau-a-Pique with its first two completed in 2006. 25 holes totalling 8,099.9 m were drilled. A third campaign of 14 drill holes took place in 2007, totalling 7,506.2 m. This program was focused on expanding the mineral resource along the NW strike and delineation at depth. The fourth drill campaign, carried out in 2008, was a combination of in-fill and exploratory drilling. 30 holes totalling 7,285.25 m were drilled. The main focus of the fourth campaign was to convert 51% of the 2008 Inferred resources into the Measured/Indicated categories and to define the limits of the mineral resource.

Aura conducted an underground drill campaign at Pau-a-Pique in 2015-2016. 27 holes totalling 3,160.0 m were drilled. Drilling was concentrated mainly on NW strike and NW down plunge extensions of the Pau-a-Pique main lens (P1 zone) below current development levels. Another objective was to delineate mineral resources in the SE portion of deposit (P3 and P4 zones) below mined-out levels to add and convert Inferred mineral resources to the Indicated category.

1.8 SAMPLE PREPARATION AND DATA VERIFICATION

It is P&E's opinion that sample preparation, security, analytical procedures and assay verification for both the Ernesto and Pau-a-Pique Properties drilling and sampling programs were adequate for the purposes of this Mineral Resource Estimate. It is MCB's opinion that sample preparation, security analytical procedures and assay verification for the Lavrinha Property drilling and sampling programs were adequate for the purposes of this Mineral Resource Estimate.

1.9 MINERAL PROCESSING AND METALLURGICAL TESTING

A 2010 NI 43-101 Feasibility Study by Ausenco do Brasil Engenharia Ltda ("Ausenco") prepared for Yamana describes the metallurgical testwork performed on two samples obtained from the Ernesto belt, one from the Japones area and the second sample from the Ernesto area, as well as testwork results for Pau-a-Pique mineralized material, with highlights noted below.

The Ernesto metallurgical sample had a gold grade of 4.5 g/t Au and was taken from the Intermediate Trap area. The sample underwent mineralization characterization, grinding, gravity and bottle leaching testwork. Gravity testwork results showed a 68.7% recovery of free gold with an overall mass pull of 1.72%. At the same time, the gold extraction was above 95% in all cyanidation bottle tests with no significant differences in the extraction results with or without carbon and regardless of the grinding conditions.

The Pau-a-Pique metallurgical sample had a gold grade of 5.63 g/t Au. The gravity concentration results showed a high free gold recovery at 61%. The cyanidation bottle tests showed gold recoveries between 80% and 90% without carbon; however, gold recoveries increased to above 90% in the presence of carbon averaging 94.5% for tests with top size of 0.149 mm and 0.074 mm.

The EPP process plant commenced operation in 2013 and was operated until October, 2014, receiving feed from the Ernesto open pit and the Pau-a-Pique underground mine. During its first year, the plant went through a production ramp-up stage which resulted in consistent process performance improvements over its quarters. Average plant gold recovery was 92.3% of which 41% came from gravity gold and the other 51% was extracted via the CIL circuit.

Although the ramp-up stage took place in 2013, plant performance in 2014 was not as favourable due to several issues at the mine level that resulted in a lack of consistent ore feed supply and the introduction of other feed sources from areas where artisan mining activity was taking place on the concession.

The 2016 metallurgical testwork was carried out on multiple metallurgical samples of the three deposits (Ernesto, Lavrinha and Pau-a-Pique). Samples were selected from available core and coarse rejects to represent scheduled half years according to the production forecast. The testwork was performed in two different laboratories; SGS Lakefield performed the grinding work, which consisted of SAG Power Index ("SPI") and Bond Work Index ("BWI") measurements, while SGS Geosol, in Belo Horizonte Brazil, performed the hydrometallurgical testwork.

The grinding testwork in all samples showed the ore to be relatively soft both in the coarse and fine fractions, with SPI averaging 27 minutes and the Bond Work Index ("BWI") averaging 9.3 kWh/tonne. All samples tested had a calculated treatment rate well above the design rate of 130 tph (i.e. 3,000 tpd). Therefore, the installed grinding capacity should easily handle future ore throughput forecast for the Project (i.e. between 21,500 tonnes/month and 55,000 tonnes/month) and possibly grind finer since there is available capacity in the semi-autogenous grinding ("SAG") mill.

The hydrometallurgical test programme was designed to follow the existing plant flowsheet as closely as possible. Two different grind sizes were investigated, namely 125 microns and 106 microns. For the Lavrinha and Pau-a-Pique samples the average gold recovery in the Knelson MD3 laboratory concentrator was higher for the finer grind and averaged 77.78% versus 76.4% for the coarse grind. The gravity concentrate was subsequently intensively leached for 8 and 12 hours, with the 12 hour recovery being substantially better. The gravity tailings were leached, using a CIL method, to recover the remaining gold and the results indicated that the 24 hour retention time in the plant circuit will be adequate. Overall recoveries, taking into account gravity recovery, intensive leach recovery and CIL recovery, were calculated and averaged 94.0% for the Lavrinha samples and 93.6% for the Pau-a-Pique samples.

There were problems with the Ernesto testwork in that the gold recoveries were unexpectedly low. This was thought to be due to the higher grade (twice and three times as high compared to Pau-a-Pique and Lavrinha ores) and a lack of free cyanide found at the end of the leach period. The 106 micron Knelson tailings were re-leached using a higher concentration at the start of the test and also using 100 g/t of Leach Aid. There was a substantial increase in recovery for the re-leach tests, averaging 4.36% points higher. The overall recoveries averaged 86.1% for the Ernesto samples.

For the Y3 H1 sample a complete retest was carried out, at the 106 micron grind, this being the only sample with sufficient weight remaining to allow it. The gravity recovery was down several percentage points but the intensive leach recovery increased from the previous 92.4% to 99.7% with the use of Leach Aid. This is an increase of 7.3%. In view of this result a case can be made for increasing the other intensive leach recoveries, which could make the overall recoveries for Ernesto increase to 88% levels.

1.10 MINERAL RESOURCE ESTIMATES

The Ernesto Mineral Resource Estimate was estimated at a cut-off grade of 1.5 g/t Au and is summarized in Table 1.1.

TABLE 1.1
ERNESTO DEPOSIT LOWER TRAP ZONE UNDERGROUND MINERAL RESOURCE ESTIMATE AT A CUT-OFF GRADE OF 1.5 G/T AU(1-10)

Resource Category	Tonnes (t)	Au (g/t)	Contained Au oz
Indicated	734,000	6.70	158,200
Inferred	308,000	6.30	62,400

(1) CIM Definitions were followed for the Mineral Resource Estimate.
(2) The Qualified Person for this Mineral Resource Estimate is: Richard Routledge M.Sc. (Applied), P.Geo.
(3) The Mineral Resource Estimate is estimated from surface diamond drilling and core sampling by conventional 3D block modelling based on wireframing at a 1.5 g/t Au cut-off grade and ordinary kriging grade interpolation.
(4) For the purpose of the Mineral Resource Estimate, assays were capped at 40 g/t Au.
(5) The Mineral Resource Estimate is based on a Cut-Off Grade of 1.5 g/t Au derived from an Au price: US$1,275 /oz, costs of US$33/t for mining, US$11/t for processing and US$10/t for G&A, at a 93% process recovery.
(6) A bulk density model based on rock type was used for volume to tonnes conversion with resources averaging 2.62 tonnes/m^3
(7) Mineral Resources are estimated from the 380 m EL to the 96 m EL, or from approximately 50 m depth to 150 m depth from surface.
(8) Mineral Resources are classified as Indicated and Inferred based on drill hole spacing, interpreted geologic continuity and quality of data.
(9) Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
(10) The quantity and grade of reported Inferred Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as an Indicated or Measured Mineral Resource and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured Mineral Resource category.

The Mineral Resource Estimate for the Lavrinha Deposit has been reported above a 0.5 g/t Au cut-off grade, inside an optimized pit shell with a gold price of US$1,300/oz, and is summarized in Table 1.2.

TABLE 1.2
LAVRINHA MINERAL RESOURCE ESTIMATE AT A CUT-OFF GRADE OF 0.5 G/T AU(1-8)

Resource Category	Tonnes (t)	Au (g/t)	Contained Au oz
Measured	74,000	2.31	5,500
Indicated	1,226,000	2.25	88,700
Measured + Indicated	1,300,000	2.25	94,100
Inferred	283,000	2.51	22,800

(1) CIM Definitions were followed for the Mineral Resource Estimate.
(2) The Mineral Resource Estimate for the Lavrinha Deposit was prepared under the supervision of Marcelo Batelochi, AusIMM (CP 205477).
(3) Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
(4) The quantities and grades of reported Inferred Resources in this estimation is uncertain in nature and there has been insufficient exploration to define these Inferred Resources as Indicated or Measured Mineral Resource and it is uncertain if further exploration will result in upgrading them to the Indicated or Measured Mineral Resource category.
(5) The Mineral Resource Estimate is based on an optimized pit shell using US$1,300/oz gold and at a cut-off grade of 0.50 g/t gold. Mining costs were considered at US$2.44/t and US$1.89/t for mineralized material and waste haulage, plant process costs of US$10.24/t and G&A of US$3,800,000 per year at a process recovery of 93%.
(6) A bulk density model based on rock type was used for volume to tonnes conversion with resources averaging 2.77 tonnes/m^3.
(7) Surface topography as of December 31, 2015.
(8) Contained metal may not sum due to rounding.

The Pau-a-Pique Mineral Resource Estimate was estimated at a cut-off grade of 1.5 g/t Au and is summarized in Table 1.3.

TABLE 1.3
PAU-A-PIQUE MINERAL RESOURCE ESTIMATE AT A CUT-OFF GRADE OF 1.5 G/T AU(1-10)

Resource Category	Tonnes (t)	Au (g/t)	Contained Au oz
Indicated	519,000	4.05	67,600
Inferred	117,000	4.45	16,700

(1) CIM Definitions were followed for the Mineral Resource Estimate.
(2) The Qualified Person for this Mineral Resource Estimate is: Richard Routledge M.Sc. (Applied), P.Geo.
(3) The Mineral Resource Estimate is estimated from surface and underground diamond drilling and core sampling and underground chip sampling by conventional 3D block modelling based on wireframing at a 1.5 g/t Au cut-off grade and ordinary kriging grade interpolation.
(4) For the purpose of the Mineral Resource Estimate, assays were capped at 50 g/t Au and composites >25 g/t Au were restricted to 12.5 m area of influence.
(5) The Mineral Resource Estimate is based on a Cut-Off Grade of 1.5 g/t Au derived from a Au price: US$1,275 /oz, costs of US$29/t for mining, US$11/t for processing, US$10/t for G&A and US$7/t for mill feed surface transportation, at a 93% process recovery.
(6) A bulk density model based on rock type was used for volume to tonnes conversion with resources averaging 2.77 tonnes/m^3.
(7) Mineral Resources are estimated from the 410 m EL to the 65 m EL, or from approximately 30 m depth to 500 m depth from surface.
(8) Mineral Resources are classified as Indicated and Inferred based on drill hole spacing, interpreted geologic continuity and quality of data.

(9) *Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*
(10) *The quantity and grade of reported Inferred Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as an Indicated or Measured Mineral Resource and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured Mineral Resource category.*

The combined Mineral Resource Estimate for the Project is presented in Table 1.4.

TABLE 1.4
TOTAL MINERAL RESOURCE ESTIMATE FOR THE PROJECT

Measured & Indicated	Tonnes (t)	Au (g/t)	Contained Au oz
Lavrinha	1,300,000	2.25	94,200
Ernesto	734,000	6.70	158,200
Pau-a-Pique	519,000	4.05	67,600
Total Measured & Indicated	**2,553,000**	**3.89**	**320,000**
Inferred	**Tonnes (t)**	**Au (g/t)**	**Contained Au oz**
Lavrinha	283,000	2.51	22,800
Ernesto	308,000	6.30	62,400
Pau-a-Pique	117,000	4.45	16,700
Total Inferred	**708,000**	**4.48**	**101,900**

Note: *Contained metal may not sum in the above table due to rounding*

1.11 MINERAL RESERVE ESTIMATES

The Mineral Reserve Estimate for the Ernesto Deposit was determined at a 2.35 g/t Au cut-off grade, as of an effective date of July 31, 2016, and is presented in Table 1.5.

TABLE 1.5
ERNESTO DEPOSIT: ERNESTO MINERAL RESERVE ESTIMATE(1-5)

Probable Mineral Reserve Estimate for the "Lower Trap" Portion of the Ernesto Deposit			
Reserve Category	**Tonnes (t)**	**Au (g/t)**	**Contained Au oz**
Probable	868,000	5.03	140,000

(1) *The Mineral Reserve Estimate is as of July 31, 2016.*
(2) *The Mineral Reserve Estimate was developed from the Mineral Resource Estimate model prepared by P&E. The Probable Mineral Reserves were derived from Indicated Mineral Resources.*
(3) *The cut-off grade (2.35 g/t Au) was based on a US$1,165/oz gold price, 93% metallurgical Au recovery, 99.99% payable, royalties and CEFEM tax totalling 3.5%, gold doré bar transport and refining costs totalling US$0.45 / g Au, mine direct and mine indirect costs totalling US$62.41/ t, US$10.30/t processing cost, and US$6.12/t processed for the projected share of the overall multi-mine project G&A cost that would be incurred by the proposed Ernesto underground mine project. The geological continuity of the mineralization was assessed for the cut-off grade.*
(4) *The Mineral Reserve Estimate tonnage and mined metal have been rounded to reflect the accuracy of the estimate.*
(5) *The NI 43-101 Mineral Reserve Estimate for the Lower Trap portion of the Ernesto Deposit set out in the table above has been reviewed and approved by David Orava, M.Eng., P. Eng., of P&E Mining Consultants Inc., who is a Qualified Person ("QP") , and who is independent of the Company.*

The Mineral Reserve Estimate for the Lavrinha Deposit was determined at a cut-off grade of 0.48 g/t Au and is presented in Table 1.6.

TABLE 1.6
LAVRINHA DEPOSIT: LAVRINHA MINERAL RESERVE ESTIMATE(1-7)

Reserve Category	Tonnes (t)	Au (g/t)	Contained Au oz
Proven	67,000	1.85	4,000
Probable	1,043,000	1.68	56,300
Total	**1,110,000**	**1.69**	**60,300**

(1) *CIM definitions were followed for the Mineral Reserve Estimate.*
(2) *The Mineral Reserve Estimate is as of July 31, 2016.*
(3) *The Mineral Reserve Estimate for the Lavrinha Deposit was prepared under the supervision of Marcelo Batelochi, Ausimm (CP 205477).*
(4) *The Mineral Reserve Estimate was at a cut-off grade of 0.48 g/t Au.*
(5) *The Lavrinha Mineral Reserve Estimate was at an average short-term gold price of US$1,100 per ounce.*
(6) *Bulk density average was 2.78 t/m3.*
(7) *Numbers may not add due to rounding.*

The Mineral Reserve Estimate for the Pau-a-Pique Deposit was determined at a cut-off grade of 2.40 g/t Au and is presented in Table 1.7.

TABLE 1.7
PAU-A-PIQUE DEPOSIT: PAU-A-PIQUE MINERAL RESERVE ESTIMATE(1-5)

Reserve Category	Tonnes (t)	Au (g/t)	Contained Au oz
Probable	320,000	3.24	33,300

(1) *The Mineral Reserve Estimate is as of July 31, 2016.*
(2) *The Mineral Reserve Estimate was developed from the Mineral Resource Estimate model prepared by P&E. The Probable Mineral Reserves were derived from Indicated Mineral Resources.*
(3) *The cut-off grade (2.40 g/t Au) was based on a US$1,165/oz gold price, 93% metallurgical Au recovery, 99.99% payable, royalties and CEFEM tax totalling 3.5%, gold doré bar transport and refining costs totalling US$1.56/t, mine direct and mine indirect costs totalling US$58.08/t, US$12.50/t processing cost, and US$6.44/t processed for the projected share of the overall multi-mine project G&A cost that would be incurred by the proposed Pau-a-Pique underground mine project.*
(4) *The Mineral Reserve Estimate tonnage and mined metal have been rounded to reflect the accuracy of the estimate.*
(5) *The NI 43-101 Mineral Reserve Estimate for the Pau-a-Pique Deposit set out in the table above has been reviewed and approved by Alexandru Veresezan, P. Eng., of P&E Mining Consultants Inc., who is a Qualified Person ("QP") and who is independent of the Company.*

The total Mineral Reserve Estimate for the Project is presented in Table 1.8.

TABLE 1.8 TOTAL MINERAL RESERVE ESTIMATE FOR THE PROJECT			
Proven	**Tonnes (t)**	**Au (g/t)**	**Contained Au oz**
Lavrinha	67,000	1.85	4,000
Total Proven	**67,000**	**1.85**	**4,000**
Probable	**Tonnes (t)**	**Au (g/t)**	**Contained Au oz**
Lavrinha	1,043,000	1.68	56,300
Ernesto	868,000	5.03	140,000
Pau-a-Pique	320,000	3.24	33,300
Total Probable	**2,231,000**	**3.20**	**229,600**
Total Proven + Probable	**2,298,000**	**3.17**	**233,600**

Contained metal may not sum in the above table due to rounding

1.12 MINING METHODS

1.12.1 Ernesto Underground

Due to its nature of gentle and variable shallow dip and thickness, the Ernesto Deposit will be extracted by the Drift and Fill mining method, using a combination of drifting in ore and transverse primary and secondary small stopes in a 32%:36%:32% drift/primary/secondary tonnage ratio. The deposit is relatively close to surface at a maximum depth of approximately 170 m and will be accessed by one main ramp portal, with a second portal for definition drilling access and ventilation.

Backfill material will be waste rock for secondary stopes and ore drifts and cemented rock fill (“CRF”) for all primary stopes. Waste rock to fulfill the required backfill quantities will be obtained from two sources; the primary source will be from mine waste development and the second source will from the existing Ernesto open pit waste rock storage facility.

A six month pre-production period will be followed by approximately 3.5 years of production to mine an estimated 0.87 Mt of ore at an average grade of 5.03 g/t Au. Ore production will average 800 tpd.

The majority of underground mining activities at Ernesto will use Aura’s own employees, with external contractors or suppliers to undertake the supply of explosives, piping and services, ground support consumables, cement supply for the CRF plant, and other specialised tasks. Aura will have 100% ownership of all major fixed plant components used at Ernesto. Activities such as diamond drilling and other specialized activities or Project work will be contracted.

1.12.2 Lavrinha Open Pit

Approximately 1.11 Mt of ore at an average grade of 1.69 g/t Au and 14.0 Mt of waste rock will be mined from the Lavrinha open pit over a 2.5 year period. The overall strip ratio for Lavrinha is 12.6:1 with mining conducted 365 days per year by a contractor. The contract is full service and includes providing all mining equipment, drilling, blasting, loading, hauling and maintenance. Total material movement rates for the LOM range from 15,000 to 25,000 tpd.

Conventional truck and hydraulic shovels will be utilized. Four excavators, supported by three front-end loaders, will load a fleet of ten 38-tonne trucks and five 25-tonne trucks. Ore will be transported to the primary crusher and run-of-mine (“ROM”) pad, and waste material will be hauled to a nearby waste rock storage facility.

1.12.3 Pau-a-Pique Underground

Mining at Pau-a-Pique will be conducted by a modified Avoca choke blast stoping method with ore transported to the ROM pad on surface by 30 tonne haulage trucks operating through the main ramp. Ore will be subsequently hauled on a 47 km surface road to the Ernesto processing plant. Primary access to the underground mine is via a single portal located next to the main mining office.

Approximately 0.32 Mt of ore at an average grade of 3.24 g/t Au will be mined over a 17 month period at an average of 850 tpd when the Project achieves full production. Once the deposit has been depleted most of the equipment and operators will be transferred to the Ernesto operation.

The stoping method applied to the Area 7 and Area 8, NW, and P3 and P4 ore bodies will be via Hanging Wall (“HW”) access ore drives with levels spaced at 15 m and 21 m vertical intervals, for the upper and lower areas of the deposit, respectively. The upper and lower areas will be separated by a sill pillar. Unconsolidated waste rock will be used to backfill the stopes.

The majority of underground mining activities will utilize Aura’s employees, with external contractors or suppliers to undertake the supply of explosives, piping and services, ground support consumables, truck haulage underground and on surface and other specialized tasks (i.e. site security, doré bar transportation, etc.). Aura has 100% ownership of all major fixed plant components utilized at the mine.

1.13 RECOVERY METHODS AND PROCESS DESIGN

The Project’s gold processing plant, located next to the Ernesto Deposit, was commissioned in 2012 and treated ores from Pau-a-Pique and the Ernesto open pit until its closure in December 2014. It is centrally located to these deposits and has a capacity of 3,000 tonnes per day (“tpd”) through a conventional carbon-in-leach process and is designed to treat up to 1 Mtpy feed. The process includes crushing, grinding, gold extraction/recovery and cyanide detoxification stages followed by final deposition in a tailings storage facility.

The process plant flowsheet is based on a low-risk proven technological configuration for processing gold bearing feed. A primary jaw crusher is located at the front-end of the process plant. ROM feed will be blended and fed through the plant’s primary screen. The screen oversize is crushed and the combined crushed feed is ground in a single-stage, closed-circuit SAG mill.

Approximately 25% of the mill cyclone underflow feeds a gravity-gold recovery circuit. The grinding circuit product is thickened and then pumped to a leach tank that is followed by six CIL tanks in series. CIL tailings are treated in a cyanide reduction tank where cyanide is chemically decomposed. Final tailings are pumped to a tailings dam.

Loaded carbon, recovered from the first CIL tank, reports to the desorption area. Gold is stripped from the carbon into a solution and electroplated from solution onto stainless steel cathodes. Dried cathode sludge and flux are mixed and smelted to produce gold doré.

Mill feed from Ernesto and Lavrinha will be transported to the process plant by haul trucks internally within the mine property. Mill feed from Pau-a-Pique will be transported via a public 47 km road section. This road will require ongoing maintenance by the mine.

1.14 PROJECT INFRASTRUCTURE

Most of the Project's infrastructure such as fresh water access, power line bringing energy to the different areas of the Project (including Pau-a-Pique underground mine) and access roads were built by the previous Project owner and have been preserved. The capital requirements will be further reduced by the planned reutilization and transfer of Pau-a-Pique's infrastructure and mine fleet to the newly developed Ernesto underground upon completion of the scheduled ore production at Pau-a-Pique.

The Project area is suitable for year-round mining, and has adequate access infrastructure that was developed during the previous 2013-14 operating period. Minor road maintenance work has been identified and will be carried out in early 2017.

Aura is updating the landowner agreements for resumption of ore haulage along an approximate 47 km stretch of the existing access road between Pau-a-Pique and highway BR-174. This process is well underway and no impediments are anticipated.

Fresh water for the Project is acquired from the Lavrinha Creek located 3.8 km from the processing plant and pumped at a rate of 70 m^3/hr through an 8 inch HDPE pipeline. There are two water treatment plants at the Project, one installed at the Ernesto camp with a treatment capacity of 6 m^3/h and a second water treatment plant installed at the Pau-a-Pique camp with a treatment capacity of 3 m^3/h.

A 12 km 138 kilovolt ("kV") electrical transmission line was built as part of the infrastructure for the Project which connects to the National grid from the Pontes e Lacerda substation. The Project distribution network includes a 34.5 kV transmission line to Pau-a-Pique with all other primary distribution at 13.8 kV, which is then stepped down at the various substations.

The total electrical load installed at Ernesto is currently estimated at 7.35 MW (existing plant and on-site infrastructure). When Ernesto underground mining activities start, a maximum of 2.8 MW of electrical installed load will be added to the overall consumption. The installed substation and the existing power infrastructure will be suitable to address the future energy requirements of the Project.

The total electrical load installed at Pau-a-Pique is 1.91 MW. The current transmission line is adequate to supply enough energy to the Project restart. The transformer installed at Pau-a-Pique has a 3 MVA power capacity.

The office area at Ernesto is located adjacent to the process plant and includes a main office building (which incorporates training and first-aid areas), a change house, a cafeteria, a chemical and metallurgical laboratory, a workshop and a warehouse area with a storage yard. The number of people at the Pau-a-Pique site is less than Ernesto and the size of the facilities there reflects this.

A tailings storage facility is located within the premises of the Project and is designed to store tailings from the process plant, which will process feed from the three different mines. The tailings dam crest is 6 m wide. Upstream and downstream slope ratios are 1V:2H. The tailings dam has an internal drainage system consisting of a vertical sand filter and a horizontal drainage blanket made of fine crushed stones and sand. There is a rock sump and return water pump at the drain terminus.

The tailings storage facility design accounted for a total volume of stored tailings of 5.7M m^3 over a span of 7.3 years of Project life. The original design considered three stages: Stage I with a total storage capacity of 2.3M m^3, Stage II with a storage capacity of 3.6M m^3 and Stage III with a capacity of 7.1M m^3 to support a total of 7.3 years of operation.

Stage I is currently built with a dam crest elevation of 339 m and a total storage capacity of 2.3M m^3 and a maximum safe storage capacity of 2.16M m^3. The total volume stored, as of May 2016, is 1.12M m^3 of tailings and an additional 0.4M m^3 of water for a total stored volume of 1.5M m^3, leaving an additional 0.6M m^3 of available capacity in the existing Stage I tailings storage facility.

The Company engaged Tierra Group International Ltd., an internationally recognized tailings engineering firm, to review the current Tailings Storage Facility's ("TSF") design and construction history; and based on the review, design future TSF expansions. The historical review is complete wherein Tierra Group found the existing TSF to have been designed and constructed using satisfactory industry standards of care to support initial operations. Tierra Group is currently advancing a detailed engineering investigation and design to expand the TSF.

The design work contemplates raising the dam height 3 m. (elevation 342 m), and maintaining 2H:1V upstream and downstream dam slopes. A field geotechnical investigation is defined to corroborate geotechnical parameters used in the Stage I design, and establish those for the Stage II design.

A tailings deposition plan has been developed, which prescribes adding tailings discharge points in the north and east impoundment to extend the life of the Phase II TSF to 2.3 years. Table 1.9 shows tailings storage capacity of Stages I and II.

TABLE 1.9
TIERRA GROUP'S VOLUMES AND STORAGE CAPACITY OF STAGES I AND II OF THE TAILINGS DAM

Stage	Dam Crest Elevation (m)	Tailings Discharge Elevation (m)	Incremental Volume of Dam (m^3)	Tailings Storage Cum. (Mt)	Remaining Capacity (Years)
I	339	338.5	230,000	1.76	1.0*
II	342	341.5	80,000	2.98	2.3

**Additional discharge point at the eastern end of impoundment.*

The Stage II final design will require an additional 90,000 m^3 of fill be placed downstream of the existing dam. The resultant facility will have a footprint area approximately equal to 155,000 m^2, which is nominally 5% greater than its current footprint area. Tierra Group is expected to complete the design work by January 2017.

1.15 MARKET STUDIES AND CONTRACTS

Aura does not have any forward sales or streaming gold contracts in place that are applicable to the Project, and future gold revenue will be according to spot prices on public markets.

The base case financial model for the Project utilizes a gold price of US$1,300/oz. This price remains fixed for the life of the Project. For comparison, the 48-month trailing average price for gold that existed on the effective date of this Technical Report was approximately US$1,317/oz.

Aura's wholly-owned Brazilian operating company Apoena has a contract with Umicore Brasil Ltda. to refine its gold and silver. The contract was updated on January 1, 2016, for sampling, analysis and refining services.

Apoena has a contract with Brink's - Segurança e Transporte de Valores Ltda. for the shipment of up to 120 kg of doré or $R10,500,000 value per shipment. The contract is dated November 13, 2016.

Aura has contracted Dinex Engenharia Mineral Ltda. to mine the Lavrinha open pit deposit. The contract is based on haul distances and unit costs per tonne for waste and ore applied to the Lavrinha mine plan, plus unit costs for auxiliary equipment usage. Equipment maintenance is included in the unit costs. The major equipment in the fleet is specified as Volvo excavators, CAT dozers, Scania trucks and Sandvik drills. The contract term is 24 months, and is to be done by contract phase, with Phase I at 450kt/month to the end of April, 2017, and Phase II at 750kt/month to the end of mine life.

1.16 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Aura has existing surface rights over most of the Project area either via direct ownership or agreements with landowners. Negotiations are in process for a remaining parcel in Lavrinha and a small portion of the Pau-a-Pique Project area. There are no communities or permanent dwellings within the Project footprint. Just under 234 ha of the Ernesto site's surface property held by the Company is a designated legal reserve, in compliance with the Brazil Forest Code's provisions pertaining to conservation for native vegetation in rural properties. Maintenance, monitoring and security of this legal reserve area is the responsibility of the Company.

Additional Project disturbance is primarily for Lavrinha mining and waste rock storage and is estimated to be in the order of 55 ha. Much of the Lavrinha pit area has been previously affected by smaller scale mining by others.

It is expected that noise, dust and vibration emissions from Project operations will be similar in scale to emissions during the 2013 to 2014 operating period.

Underground mining will utilize both cemented rock fill and unconsolidated waste fill in order to optimize ore recovery, and is not expected to generate waste rock for disposal at surface. The backfill process lessens the Project footprint and is also expected to minimize the potential for surface subsidence.

Acid rock characterization studies were conducted by the previous operator using samples consisting of a drillhole interval of mineralization along with the immediately adjacent 1 m of non-mineralized material. Three of the 25 Ernesto sample results and three of the 10 Pau-a-Pique sample results indicated potential for acid rock drainage. The Project cost model provides for additional test work in 2017 for tailings and waste rock.

A review of monitoring data indicates that the Company is complying with the monitoring, inspection and surveillance programs stipulated in operating licenses for Ernesto and Pau-a-Pique. Water quality monitoring results indicate that the existing facilities meet or exceed applicable federal effluent and receiving water standards.

Estimated fresh water consumption during the Project's normal operation is 70.6 m^3/h, below the permitted license limit of 100 m^3/h from the existing water intake. Approximately 130 m^3/h is expected to be recycled from the tailings impoundment to the process plant. Discharges from the Ernesto site include controlled releases of excess tailings impoundment water, in order to maintain sufficient freeboard at all times. These planned releases are expected to occur on an as-required basis throughout the Project life. The Company reports that the most recent impoundment water release occurred from July 8 to August 18, 2016 and totalled 243,242 m^3. Water discharges at Pau-a-Pique includes excess water from underground dewatering, and a minor quantity of effluent from its permitted sewage treatment plant.

Project closure costs are estimated at US$6.0M, with an additional US$1.0M allocated for supporting studies. These costs were reviewed and found to be reasonable. The cost model assumes some closure-related expenditures during the operating period for studies and closure plan updates, as well as for decommissioning of completed mine areas such as the Pau-a-Pique underground workings. The Ernesto site has a native plant nursery with facilities for seed collection, processing and storage, composting, and propagation of up to 60,000 plants per year.

The Project has the required permits and authorizations to resume and continue mining operations at the Lavrinha open pit and the Pau-a-Pique underground mine, as well as to process ore at the Ernesto plant. Pau-a-Pique had its Mining Concession (*Portaria de Lavra*) granted on December 27, 2013. The Mining Concession for Lavrinha was requested on August 21, 2016 and is under review by the Nacional de Producao Mineral ("DNPM"), which is expected to be granted in due course. While the analysis of the application for the Mining Concession is not concluded, the Project obtained, on September 9, 2016, a special authorization (*Guia de Utilização*) to mine up to 50,000 t of ore. An application for extraction of an additional 250,000 t of mineralized material was submitted to the DNPM on November 23, 2016.

The Project has valid environmental licenses for both Lavrinha and Pau-a-Pique. The permits (*Certificados de Registro* – CR's) for use of explosives and chemicals at Ernesto, and for use of explosives at Pau-a-Pique were issued on September 29, 2016.

Once the definitive Mining Concession has been issued, other pending authorizations for continued mining in Lavrinha including its definitive operating license and permit to construct a separate waste rock storage facility adjacent to the open pit, are anticipated to be issued from the State environmental authority.

1.17 CAPITAL AND OPERATING COSTS

1.17.1 Capital Costs

The development of Pau-a-Pique mine, including the Ernesto process plant and the majority of the site infrastructure, was effectively completed by the previous owner at the end of 2012. Therefore, the capital cost requirements of the Project are low.

The Lavrinha open pit is a contracted mining operation and the selection of the mining contractor has, after a rigorous competitive bidding process in Brazil, been completed. Therefore, there will not be any material capital costs associated with the operation of the Lavrinha open pit.

The Ernesto underground mine will benefit from the transferring of the existing Pau-a-Pique's mobile fleet and infrastructure since these two deposits have been scheduled sequentially. The Ernesto mine design is compatible with the existing underground mining equipment at Pau-a-Pique.

The existing tailings storage facility will undergo an additional 3.0 m raise to increase its capacity for another two years. The design of this raise was originally done by DAM Engenharia do Brazil and it is currently being re-evaluated and validated by Tierra Group.

Ernesto Underground Capex

It is anticipated that the development of the Ernesto underground gold mine will commence once the Pau-a-Pique mine's lateral development has been completed. Within the current evaluation of the Ernesto underground Project, additional mobile equipment has been included to achieve the mine production schedule and those units will be leased to purchase.

As per the current mine plan and schedule, Ernesto reaches full production after approximately six months from commencement. During this period, mining mainly consists of ore development and primary stope extraction. To expedite the planned production the Ernesto underground mine will be accessed via a twin ramp concept, with a Hanging Wall ("HW") development drift which will be primarily for definition drilling and ventilation, and a main access ramp which will serve for main haulage and fresh air intake. This arrangement will create a loop for traffic fluidity and will fulfill ventilation and secondary egress requirements.

Pre-production capital costs are estimated at US$6.36M over a five month period. The total capital cost for Ernesto has been estimated at US$23.0M which includes capitalized development, sustaining capital, allocated labour, and mobile equipment capital for the duration of the mine life. The capitalized development portion has been estimated at US$11.5M which will be required to fully develop the Ernesto underground mine including US$4.5M for pre-production and the remaining US$7.0M as sustaining capital costs required until the mine ceases operation.

Sustaining capital expenditure for the remainder of the mine life includes:

- CRF surface plant
- Office equipment and existing equipment repairs
- Road resurfacing (crushed/screen aggregates)
- Replacement of small item i.e. face pumps, fans, electrical distribution boxes

A summary of Ernesto total capital costs including pre-production and sustaining for the LOM at Ernesto is US$23.0M as shown in Table 1.10.

TABLE 1.10 **CAPITAL EXPENDITURE FOR MINING LOM AT ERNESTO**	
Capital Expenditure	**Total LOM US$M**
Capital Development Direct Cost	6.68
Indirects (Equipment, Labour, Other)	16.28
Total CAPEX	**22.97**

A closure cost for the Ernesto underground mine has been included in the consolidated financial model and was estimated at US$3.0M. This cost is not included in Table 1.10.

<u>Lavrinha Open Pit Capex</u>

The Lavrinha open pit mining operation is fully contracted and does not incur any material capital costs. Aura, using its many years of operating experience in the region, selected a reputable and reliable mining contractor for this operation.

<u>Pau-a-Pique Underground Capex</u>

In late 2014 the Pau-a-Pique underground mine was placed on care and maintenance. The existing infrastructure and installations are functional and require minimal work before mining recommences.

Sustaining capital expenditure over the mine life includes completion of outstanding work such as:

- Surface maintenance shop upgrades
- Equipment refurbishing mechanical work and associated parts
- Office equipment and existing equipment repairs
- Road resurfacing (crushed/screen aggregate)
- Small items (i.e. face pumps, fans, electrical distribution boxes).

Total Pau-a-Pique initial and sustaining capital for the LOM is estimated at US$7.8M as presented in Table 1.11**.**

TABLE 1.11 CAPITAL EXPENDITURE FOR MINING LOM AT PAU-A-PIQUE	
Capital Expenditure	**Total LOM US$M**
Preproduction	0.97
Equipment Rental	1.11
Development	5.69
Total CAPEX	**7.77**

The closure cost for Pau-a-Pique underground mine is not included in Table 1.11 but has been included in the consolidated financial model and is estimated at US$1.7M.

<u>Plant and Tailings Capex</u>

The gold processing plant was commissioned in 2012 which includes a state-of-the-art distributed control system and all associated instrumentation with all components currently fully functional.

An allowance of US$4.5M for sustaining capital projects at the plant level has been estimated over the 5.5 year LOM.

The existing tailings storage facility has capacity for one year of operation and the next dam raise was engineered by DAM Engenharia from Belo Horizonte. The estimated costs for the next raise are US$1.5M and the subsequent raise is estimated at US$2.2M for a total cost of US$3.7M over LOM.

<u>Closure Capex</u>

A total of US$7.0M has been estimated for Project closure capital at the end of the Project life.

1.17.2 Operating Costs

<u>Ernesto Underground Opex</u>

Ernesto operating cost first principle estimates have been built utilizing advance rate cycles for each heading that were applied against scheduled quantities. A summary of the Ernesto operating cost estimates is presented in Table 1.12.

TABLE 1.12 SUMMARY OF ERNESTO LOM OPERATING COST ESTIMATES		
Operating Cost Area	**US$M**	**US$/ t ore**
Mining	43.12	49.69
Mining Overhead	11.38	13.12
Total Operating Cost	**54.50**	**62.81**

Ernesto labour costs have been based on scheduled manpower requirements for the operations, in line with Aura's organizational chart. Salaries and benefit structures are calculated in accordance with current prevailing salary structures in Brazil for the prescribed employment positions. The salary structures and labour rates are compliant with the provisions required under Brazilian tax law. All-in costs have been factored into the labour rates, including bonuses, overtime, sick leave, allowances for vehicle and accommodation (where relevant), annual leave, and health insurance and medical provisions.

Ernesto and Pau-a-Pique mining costs have been developed based on a schedule of first principle developed rates for underground production, development and diamond drilling. Costs of other inputs into the mining operations, including provision of power, water and services, are based on existing contract rates with external suppliers and estimated consumption rates.

Lavrinha Open Pit Opex

The Lavrinha open pit is a contracted operation and the costs associated with ore production and waste movement have been set as presented in Table 1.13. Aura has been actively mining in this area of Brazil for over half a decade utilizing mining contractors.

TABLE 1.13
SUMMARY OF LOM CONTRACT MINING COSTS FOR LAVRINHA

Operating Cost Area	Ore (US$/t)	Waste (US$/t)
Drilling	0.38	0.22
Blasting	0.40	0.30
Loading	0.41	0.31
Hauling	0.77	0.70
Aux. Equipment	0.20	0.20
Geology	0.06	0.06
Planning	0.04	0.04
G&A (Overhead)	0.06	0.06
TOTAL Mining Operating Cost	**2.31**	**1.88**

Pau-a-Pique Underground Opex

Pau-a-Pique operating cost estimates have been developed from first principles, utilizing historical advance rates, updated contractual rates for haulage, new consumables quotes and an up-to-date study on Aura's labour rates. A summary by cost area is presented in Table 1.14.

TABLE 1.14
SUMMARY OF PAU-A-PIQUE LOM OPERATING COST ESTIMATES

Operating Cost Area	US$M	US$/ t ore
Mining	16.55	**51.72**
Mining Overhead	2.00	**6.21**
Total Operating Cost	**18.55**	**57.93**

Costs of other inputs into the mining operations, including provision of power, water and services, are based on existing contract rates with external suppliers and estimated consumption rates.

Process Plant Opex

During the first 26 months of operation, the processing plant will treat an average of 55,000 tonnes of ore per month; this average throughput will be primarily from the Lavrinha open pit and partially from the Pau-a-Pique underground. After month 27, the Ernesto underground will become the sole source of ore feed to the plant as Lavrinha and Pau-a-Pique become depleted, and this will result in a lower average monthly throughput of 21,500 tonnes per month.

The processing costs are presented in two categories: fixed and variable costs. Fixed costs include plant labour and fixed contracts to operate the plant. Variable costs include all consumables, maintenance parts, power and other variable cost components. The processing cost for the 55 Kt/month production rate is estimated at US$12.5/t, and for the 21.5 Kt/month rate is estimated at US$21.3/t, as presented in Table 1.15.

TABLE 1.15
PROCESS PLANT OPERATING COST BREAKDOWN

Cost Breakdown	55Kt/month (‘000 US$)	21.5Kt/month (‘000 US$)
Labour Cost	153.7	135.2
Contract Cost	39.7	26.5
Total Fixed Costs	**193.4**	**161.7**
Maintenance Cost	45.6	30.4
Consumables Cost	258.3	141.7
Power Cost	156.0	101.7
Contingency	32.7	21.8
Total Variable Costs	**492.6**	**295.6**
Total Monthly Cost (US$)	**686.0**	**457.3**
US$/t	**12.5**	**21.3**

Process consumables and reagents for the process plant have been calculated on budgeted consumption rates and pricing provided by suppliers for initial first fill supply.

Labour costs were defined after a “Pesquisa de Remuneracao e Beneficios” (i.e. salary survey) was conducted in early 2016 by Parametro RH, a human resources company based in Sao Paulo, Brazil. This survey provided average, maximum and minimum salaries and benefits for more than 150 employment positions based on 11 active mining companies operating in Brazil.

Maintenance costs have been estimated on planned maintenance requirements for ongoing operation of the process plant. Maintenance costs include general materials and spare parts used in the processing plant as well as small service contracts for electrical and mechanical activities. The total maintenance costs will fluctuate between US$30,400/month and US$45,650/month depending on whether the plant is running at 21.5Kt/month or 55Kt/month, respectively.

The Project has a current power supply contract with the Mato Grosso Energy Utility Company (“ENERGISA”) which is valid until the end of 2017. Under this contract, the cost per megawatt-hour (“MWh”) is R$181.6 or US$56.7 at a foreign exchange rate of US$1.0:R$3.2.

The largest power consumer across the entire Project is the processing plant, for the crushing and grinding stages. The power costs are estimated to be between US$156,000 and US$101,000 per month for 55Kt/month and 21.5Kt/month, respectively.

Gold doré bar freight and refining costs have been based on historical costs and are subject to market adjustment. The total payable for gold is 99.99% and the refining costs are estimated to be US$5.63/oz of payable gold. The gold transportation costs are estimated at US$9.44/oz of recovered gold (e.g. saleable gold).

Global G&A Costs

The Project's operational cost includes an annual fixed global G&A cost which entails all related labour, consumables, and services that are used commonly by all operating mines, as shown in Table 1.16. In addition to the global G&A, each mine and the processing plant have its own local G&A cost.

Based on the mining schedule, the Project will have the Lavrinha open pit and the Pau-a-Pique underground producing at the same time for approximately 27 months and thereafter the Ernesto underground will become the sole source of ore to the plant. Based on this schedule, global G&A costs have been broken down into the two cases.

TABLE 1.16
GLOBAL G&A COSTS – TWO MINES VS. ONE MINE OPERATING

ITEM	LAV + PPQ ('000 US$)	ERN ('000 US$)
Labour	1,614	1,406
Consumables	123	103
Contract	2,021	1,816
Others	376	332
Total Cost ('000 US$/year)	**4,134**	**3,658**

1.18 ECONOMIC ANALYSIS

1.18.1 Base Case Operating Highlights and Project Performance

- Gold price: Baseline economic evaluation: US$1,300/oz Au
- Proven and Probable Mineral Reserves: 2.3 Mt @ 3.17 g/t Au containing 233,600 oz Au
- Average Gold Production: 36,100 oz/year over approximately 5.8 years.
- Foreign Exchange Rate: 3.2:1 (BRA:USD)
- Initial CAPEX: US$18.2M (Partially funded by the Yamana Debt Facility of US$9.0M and an Aura Rights Offering in 2016 of approximately US$4.0M; including working capital and contingency)
- NPV @ 5% (after-tax): US$28.5M
- IRR (after-tax): 100%

The Project economics are comprised of three economical scenarios: 1) "Base Case" Scenario which uses current metal prices and foreign exchange rates (i.e. US$1,300/oz Au and 3.2:1 FOREX), 2) "Upside Ernesto Recovery" Secenario which considers an increase in process plant recovery from 86% to 88%, and 3) "Consensus" Scenario which considers the long-term metal prices and foreign exchange rates (i.e. US$1,350/oz Au and 3.5:1 FOREX). Table 1.17 presents the After-Tax Project economics for the "Base Case" Scenario.

Inflation has not been considered in the cash flow analysis, since the Project will be commenced over a relatively short period of time, and all costs are stated in nominal terms. Neither costs nor revenue has been escalated with any Consumer Price Index ("CPI") or other base commodities inflation.

TABLE 1.17
AFTER TAX BASE CASE PROJECT ECONOMICS

Operating Statistics	Life-Of-Mine (LOM)
Ore Tonnes	2,298,000
Au (g/t)	3.17
Plant Recovery (%)	88.7%
Gold production (payable) oz Au	207,700
Cash cost US$/oz	837
All-in Sustaining cost US$/oz	1,064
Estimated Cash Flows	**(US$ 000's)**
Gold Revenue	269,996
Government Royalties	(2,700)
Refining and Transport	(3,130)
Net Smelter Return (NSR)	**264,167**
Mining costs	(104,766)
Processing costs	(36,783)
Total Project G&A	(22,449)
Private Royalty	(6,750)
Pre-tax Cash Earnings	**93,418**
Income taxes	(8,328)
PIS/COFINS Credits[1]	8,328
After-tax Cash Earnings	**93,418**
Capital and Sustaining Capital	(38,946)
Closure Costs	(7,020)
Cash Flow to Entity	**47,452**
Debt Yamana *(Including Interest)*[2]	(11,016)
Cash Flow to Equity	**36,436**
NPV 5%	**28,517**
NPV 8%	**24,737**
NPV 10%	**22,540**
IRR	**100%**

(1) PIS/COFINS are tax credits under Brazilian Tax Regulation for exporters and those can be used to offset against income tax liabilities or refunded in cash.

(2) As previously disclosed, in order to facilitate the acquisition of the Project, the previous owner, SBMM, a company affiliated with Yamana, made available to the Company's operating entity a working capital facility of up to US$9M (the "Working Capital Facility"). The Working Capital Facility bears interest at 4% per annum on the outstanding balance. The funds advanced from the Working Capital Facility have been invested in the capital, care-and-maintenance and engineering requirements of the Project to restart the Project and to complete the NI 43-101 technical reporting. The Working Capital Facility is expected to be repaid with the initial free cash flow from the Project or will be payable in full by April 30, 2018. Should the Project not enter into production and the Company not have sufficient funds to repay the Working Capital Facility on the due

date, such amount outstanding will, at the option of Yamana, be converted into common shares of the Company at a 10% discount over the 20 day VWAP of the Company's common shares based on the period prior to the due date. At no point in time may Yamana own, beneficially or otherwise, greater than 19.9% of the issued and outstanding common shares of the Company.

1.18.2 Upside and Consensus Cases

For the "Upside Ernesto Recovery" scenario, the Ernesto ore recovery was increased from the base case of 86% to 88% to see the effects on overall Project economics. For the Ernesto 88% recovery case, the after-tax NPV at a 5% discount rate from 2016 through to completion of LOM is estimated at $31.3M and the IRR is estimated at 104%. Recovered gold over the LOM increases to 210,521 ozs compared to 207,689 ozs for the 86% recovery case.

For the "Consensus" scenario, a price forecast of US$1,350/oz gold and a long term foreign exchange rate of BRA:USD = 3.5:1 were considered, and the after-tax NPV at a 5% discount rate from 2016 through to completion of LOM is estimated at $47.7M and the IRR is estimated at 497%.

1.18.3 Economic Sensitivities (After-Tax)

Sensitivities on the after-tax base case Project economics have been analyzed by varying the gold price, opex, capex and foreign exchange rate. The results are presented in Table 1.18.

TABLE 1.18
SENSITIVITY ANALYSIS

Gold Price Sensitivity After Tax (US$M)									
US$/oz	**1,100**	**1,150**	**1,200**	**1,250**	**1,300***	**1,350**	**1,400**	**1,450**	**1,500**
NPV	-6.7	2.1	10.9	19.7	**28.5**	37.3	46.1	54.9	63.7
Net Cashflow	-5.1	5.3	15.7	26.1	**36.4**	46.8	57.2	67.6	78.0
IRR (%)	-9	10	31	59	**100**	166	288	565	1,632
NPV After Tax (US$M)									
% Change	**-15%**	**-12%**	**-8%**	**-4%**	**0%**	**4%**	**8%**	**12%**	**15%**
Capex	34.6	33.1	31.5	30.0	**28.5**	27.0	25.5	24.0	22.5
Opex	51.1	45.5	39.8	34.2	**28.5**	22.9	17.2	11.6	5.9
Net Cash Flow After Tax (US$M)									
% Change	**-15%**	**-12%**	**-8%**	**-4%**	**0%**	**4%**	**8%**	**12%**	**15%**
Capex	43.5	41.7	40.0	38.2	**36.4**	34.7	32.9	31.1	29.4
Opex	62.7	56.1	49.6	43.0	**36.4**	29.9	23.3	16.7	10.2
IRR After Tax (%)									
% Change	**-15%**	**-12%**	**-8%**	**-4%**	**0%**	**4%**	**8%**	**12%**	**15%**
Capex	146	133	121	110	**100**	91	83	76	69
Opex	1,055	435	240	150	**100**	68	46	30	17
BRA:USD Exchange Rate									

Table 1.18 Sensitivity Analysis									
FOREX				**3.0**	**3.2***	**3.5**	**3.8**		
NPV (US$M)				18.7	**36.4**	39.3	48.4		
IRR %				54	**100**	252	969		

Note: ** represents Base Case scenario*

1.19 INTERPRETATIONS AND CONCLUSIONS

P&E concludes that financial modeling of the Project has determined that the Project will be economically viable and profitable. The Lavrinha Deposit is planned to be mined by open pit method, and the Pau-a-Pique and Ernesto Deposits mined by underground methods, utilizing the existing processing plant and tailings storage area, to produce gold. This Report outlines a total Project Proven and Probable Mineral Reserve Estimate of 2.3Mt at 3.17 g/t Au containing 233,600 ozs of gold. The Project has a low initial capital cost at US$18.2M since much of the site infrastructure is already in place. Overall Project economics are strong, with an after-tax NPV of US$28.5M, an after-tax IRR of 100%, and a payback of 1.2 years using the base case metal price of US$1,300/oz Au and a BRA:USD=3.2:1 foreign exchange rate. The Project mine life is planned at 5.8 years.

P&E concludes that this Report demonstrates the viability of the EPP Project as proposed, and that further development is warranted.

The following summarizes the Technical Report conclusions, which highlight significant aspects of the Project or define Project value:

Title on the Property is in good order. Royalties exist on all deposits in the mine schedule. The area to be developed represents only a fraction of the Aura land position, and several nearby exploration targets have been identified.

The Project's local climate and geography allow for year-round mining. The Ernesto and Pau-a-Pique sites have existing suitable access for supply and services as well as for ore haulage, and there is adequate local skilled workforce availability in the region.

The Ernesto Property contains a 130 tonnes per hour CIL process plant, which includes crushing, milling and tailing facilities with power supplied from the national grid via a 12 km 138 kV transmission line from Pontes e Lacerda. The Ernesto Property also contains a gate house, administration offices, core shack, explosives storage facility, and the mined-out Ernesto open pit and waste rock dump. The Lavrinha Property is contiguous to Ernesto and does not contain any infrastructure. The Pau-a-Pique Property contains an underground mine that was operated by Yamana until late in 2014, and surface facilities for administration and maintenance.

Aura has existing surface rights over most of the Project area either via direct ownership or agreements with landowners. Negotiations are in process for a remaining parcel in Lavrinha and a small portion of the Pau-a-Pique Project area. Aura is also updating the landowner agreements for resumption of ore haulage along the 47 km access between Pau-a-Pique and Ernesto; this process is well underway. While no impediments are anticipated for concluding these pending surface rights and access road use agreements, delays could stand to affect the execution of the Project.

Regional and local geology which controls mineralization is well understood. The Ernesto-Lavrinha and Pau-a-Pique Deposits are broadly similar in host lithologies, structural style, alteration, and mineralization and all share characteristics of shear-hosted lode gold deposits.

Exploration of the Ernesto, Lavrinha and Pau-a-Pique Deposits has been comprehensive, and methodologies and practices applied are considered appropriate. Exploration drilling on the Property is extensive. Drill campaigns have been carried out by previous companies since 2005. Aura drilled the Ernesto, Lavrinha and Pau-a-Pique Deposits in 2015, focussing on in-fill drilling in the mineral resource areas.

It is P&E's opinion that sample preparation, security and analytical procedures for both the Ernesto and Pau-a-Pique Deposits drilling and sampling programs were adequate for the purposes of this Mineral Resource Estimate. It is MCB's opinion that sample preparation, security and analytical procedures for the Lavrinha Deposit drilling and sampling programs were adequate for the purposes of this Mineral Resource Estimate.

Based upon the evaluation of the QA/QC programs undertaken by Yamana and Aura, as well as P&E's due diligence sampling, P&E concludes that the data are of good quality for use in the Ernesto and Pau-a-Pique Mineral Resource Estimates. For Lavrinha, MCB had the same conclusion as P&E since the Lavrinha drilling campaigns were carried out simultaneously with Ernesto, applying the same procedures and sampling protocols.

The EPP process plant started operation in 2013 and was operated until October, 2014, receiving feed from the Ernesto open pit and the Pau-a-Pique underground mine. Samples of the three deposits (Ernesto, Pau-a-Pique and Lavrinha) were selected in 2016 from available core and sample coarse rejects to represent half years according to the production forecast for the Project. In the main, the core samples were sent for grinding testwork while the coarse rejects were sent for hydrometallurgical testing. SGS Lakefield, Canada, performed the grinding work, which consisted of SAG Power Index and Bond Ball Mill Work Index testwork, while SGS Geosol of Belo Horizonte, Brazil, performed the hydrometallurgical testwork, consisting of Gravity Recovery of Gold, bottle roll leach tests and settling testwork. The overall recoveries for the Pau-a-Pique and Lavrinha metallurgical testwork samples are very good at approximately 93%. Those for the Ernesto samples are lower than expected, at approximately 86%, even after the re-leach results are taken into account. Further work should be carried out on Ernesto material to ascertain the reasons for this. The work should investigate using finer grinds, increased cyanide levels and also the use of Leach Aid. The grinding circuit has more than adequate capacity to handle the tonnages planned for the Project. In view of this it may be advisable to investigate whether it would be beneficial to grind finer.

In P&E's opinion, the Mineral Resource Estimates for the Ernesto and Pau-a-Pique Deposits are reasonable and has been undertaken according to industry standard practice. In MCB's opinion, the Mineral Resource Estimate for the Lavrinha Deposit is reasonable and has been undertaken according to industry standard practice.

The Total Proven Mineral Reserve Estimate for the Project is 67,000 t at 1.85 g/t Au containing 4,000 oz gold. The Total Probable Mineral Reserve Estimate for the Project is 2,231,000 t at 3.20

g/t Au containing 229,600 oz gold. The Total Proven and Probable Mineral Reserve Estimate is 2,298,000 t at 3.16 g/t Au containing 233,600 oz gold.

Mining has been sequenced to start with open pit mining of the Lavrinha Deposit for a period of 28 months. Pre-production at the Pau-a-Pique underground mine starts one month after mining commences at Lavrinha, and lasts two months. Production mining at Pau-a-Pique is carried out for 17 months. Pre-production at Ernesto lasts five months and is scheduled to end when mining at Pau-a-Pique is completed. Production mining at Ernesto is then carried out for 43 months. The total LOM sequence is 69 months, or 5.8 years.

The Ernesto Deposit will be mined by a Drift and Fill method, using a combination of drifting in ore and transverse primary and secondary stopes. The orebody will be accessed by one main ramp, with a second access for definition drill access and ventilation purposes. The presence of mylonite and its thickness will require re-analysis of ground support density and maximum stope span. The Ernesto Project will use the majority of the Pau-a-Pique Project's underground mobile equipment once Pau-a-Pique operations ceased. The Ernesto cemented rockfill plant has been selected and sized to deliver the required backfill quantity and quality.

Aura has contracted the Brazilian company Dinex to mine the Lavrinha open pit Deposit. The major equipment in the fleet is specified as Volvo excavators, CAT dozers, Scania trucks and Sandvik drills. The contract term is 24 months, and is to be done by contract phase, with Phase I at 450kt/month to the end of April, 2017, and Phase II at 750kt/month to the end of mine life.

Underground mining at Pau-a-Pique will be conducted by an Avoca choke blasting stoping method. Ore will be transported up the main access ramp and then along a 47 km surface road to the Ernesto process plant.

The existing primary powerline and all electrical components (i.e. substations, etc) have been confirmed to have enough capacity to supply energy under the two operating regimes.

The tailings dam facility will undergo a 3 m raise in 2017, which will provide additional tailings storage capacity for another 2.3 years. A final raise for the remainder of the Project will require further detailed study.

The financial model is based on a gold price of US$1,300/oz. The 48-month trailing average price as of the effective date of this Technical Report was approximately US$1,317/oz. Gold revenue for the Project will be subject to spot prices. Aura, through its wholly-owned Brazilian company Apoena, has contracts with Umicore to refine its gold and silver. It also has a contract with Brink's to transport doré.

The Project has experienced and qualified environmental management staff and facilities in place. A review of the site, permits, and monitoring data indicate that Aura is complying with the monitoring, inspection and surveillance programs stipulated in operating licenses for Ernesto and Pau-a-Pique. The Project has several key operating permits in hand to allow mining and processing activities to commence. The remaining permits and authorizations are in the application process, and there is reasonable certainty of obtaining these in due course. Delays in obtaining these pending approvals may in turn, delay or otherwise affect the Project, in particular, the cost-effective mining of the Lavrinha deposit. The Project cost model provides for additional test work in 2017 for acid rock drainage studies for tailings and waste rock.

Initial capital for the Project is estimated at US$17.3M and is low since it is partially funded by the Yamana debt facility and since much of the Project infrastructure is already in place.

Operating costs for open pit mining at Lavrinha are based on the Dinex contract, and are estimated to average US$2.31/t ore and US$1.88/t waste over the LOM. Operating costs for underground mining at Pau-a-Pique and Ernesto have been developed from first principles and contain known consumable unit costs, labour rates from a salary survey and rates paid during care and maintenance, existing electrical power rates, and known costs for other services. The average cost for mining at Pau-a-Pique over the LOM is estimated at US$57.93/t ore, and for Ernesto is estimated at US$62.81/t ore. Processing costs have been developed from first principles, budgeted consumption rates, and quotations from suppliers. The processing cost for a 55 Kt/month production rate is estimated at US$12.5/t, and for a 21.5 Kt/month rate is estimated at US$21.3/t. The annual cost for Global G&A is estimated at US$4.1M under the Lavrinha/Pau-a-Pique operation and US$3.6M for the Ernesto stand-alone operation.

The after-tax NPV at a 5% discount rate from 2016 through to completion of LOM for the base case is estimated at $28.5M and the IRR is estimated at 100%, with a payback of 1.2 years. The after-tax undiscounted cash flow of the EPP Project is estimated at $36.4M over the LOM.

The Ernesto ore recovery was increased from the base case of 86% to 88% as an upside case to see the effects on overall Project economics. For the Ernesto 88% recovery case, the after-tax NPV at a 5% discount rate from 2016 through to completion of LOM is estimated at $31.3M and the IRR is estimated at 104%. Recovered gold over the LOM increases to 210,521 ozs compared to 207,689 ozs for the 86% recovery case.

Using a consensus price forecast of US$1,350/oz gold, along with a higher than base case foreign exchange rate of BRA:USD = 3.5:1, the after-tax NPV at a 5% discount rate from 2016 through to completion of LOM is estimated at $47.7M and the IRR is estimated at 497%.

1.20 RECOMMENDATIONS

P&E specifically recommends proceeding with detailed engineering and preparations for production based on the positive economics predicted by the designs and financial evaluations contained in this Technical Report.

1.20.1 Ernesto

A number of the Ernesto drill holes that cut the Mineral Resource Estimate wireframe were not fully sampled, and two holes should be deepened if possible. Modelling of a lower grade envelop in the Inferred Mineral Resource Estimate area in the northern part of the Ernesto Property is recommended to better understand geometry-continuity of the mineralized zone. The best potential to develop additional Mineral Resource Estimates for the future lies in fill-in drilling and sampling to upgrade the Inferred Mineral Resource Estimates to Indicated Mineral Resource Estimates.

Recommendation is made for all future drilling and channel sampling programs at the Project to include a more consistent approach to QC protocol for all samples to be sent for laboratory analysis.

The planned underground definition drilling program should be followed to provide additional information needed to finalize the level and stope designs prior to drifting in ore and stoping.

A Drift and Fill mining method is recommended. The performance of the access drives is sensitive to the mining sequence, effective spans established and the ground support practices. The stope span recommendations are sensitive to the ability of mine personnel to consistently tight fill the mined stopes as soon as possible after the completion of each stope. The span and ground support recommendations are sensitive to the thickness and rock mass quality of the mylonite.

It is recommended to evaluate the required crown pillar dimensions and the stability of the secondary stope pillars, including the impact of the saprolite and further analysis of the mylonite and its influence on achievable stope dimensions and ground support following the completion of the definition drilling. Additional geomechanical logging should be completed to better define the spatial variation of the rock mass quality in the immediate HW of the proposed stopes, as well as the spatial variation in the distribution of the mylonite and saprolite.

An in-situ CRF strength of 0.5 MPa is recommended. Having consistent feed material that is within the required particle size distribution specification is an important consideration in ensuring that the CRF achieves the target strength and quality on a consistent basis. A QAQC program should be put in place, using either contracted lab services or existing Aura facilities in the area, to monitor the particle size distribution of the prepared CRF aggregate, and test for the strength of the placed CRF to ensure that excessive consumption of cement does not occur.

Additional confirmatory acid rock drainage test work for waste rock in all mine areas as well as for tailings is recommended.

1.20.2 Lavrinha

MCB recommends the following:

- Organization of the drill core in the temporary shed in Pontes e Lacerda.
- Assay drill core intervals not sampled.
- A complete review of the database information and cross-referencing with original records for the drill hole and assay databases.
- Update the surface topography files with more precision.
- Additional drilling is recommended at Lavrinha to drill off the deposit in the SW of the Property towards the adjacent valley and also at the southern end of the deposit where the density of drilling is reduced and there are some lenses that can be potentially delineated near surface.
- The results of “G912-6” Geostats Standard are based on 18 assayed samples. The results indicated a slight bias in grade. It is recommended to check the certification of this standard due to the random values around the second standard deviation.
- The Lavrinha waste rock storage area design should be advanced to a detailed engineering level including elements such as foundation evaluations, design criteria, stability analysis, internal and surface drainage design.

1.20.3 Pau-a-Pique

P&E offers the following recommendations related to the Mineral Resource Estimate:

- Drill hole down hole surveys should be reviewed for implausible readings and these should be removed and the resulting re-positioning of the hole toe examined for impact on the resource wireframing.
- Additional drilling is recommended for the west target zone to identify the mineral resource potential.
- A structural study is recommended to identify and model major gold-bearing shear zones in the deposit for future exploration drill targets.
- It is strongly recommended that definition drilling be carried out in the Indicated Resources contained in the NNW lower portion of main zone P2 and the foot wall lenses P3 and P4 in the SSE portion of the deposit, before their development.

An Avoca choke blasting stoping method is recommended. P&E strongly recommends that definition drill data be available ahead of the stope extraction which subsequently must be used in the mine planning process before a particular stope is developed and mined. This will enable the mine operations to properly place the ore accesses within the stope designed boundaries and minimize stope dilution incurred during extraction, which the operation struggled with in the past.

With the objective of minimizing dilution and operating costs, the following are recommended:

- Geotechnical mapping should be undertaken during the development of the undercut and overcut for each stope. The results of the mapping should be used to plan the initial panel strike lengths.
- The panel performance should be monitored using regular CMSs and possibly instrumentation. The collected data should be used to document the actual panel dimensions and dilution. The rock mass quality of the HW and FW and the time the panel remains open should also be documented.
- The panel strike length should be adjusted based on the observed stope performance during mining.
- A final panel reconciliation should be completed for each stope and the design of future panels should be updated using the data collected from each stope.
- The mine engineering department will need to include adequate ground control staff and resources to support mine development and operations.
- Numerical stress modelling is recommended to evaluate the extraction sequence and the offset between the development and the ore body. The results of the modelling can also be used to confirm some of the inputs to the Mathews Stability Graph, as well as the stope sizing and ground support recommendations.
- Additional kinematic and numerical analyses are recommended to refine and confirm the ground support recommendations. For example, numerical modelling could be used to refine the length of the cable bolts recommended in the HW and FW of the overcuts and undercuts.
- An evaluation of the stability of the raises is recommended prior to their development.

P&E recommends that significant attention must be dedicated to stope drilling and blasting practices mainly around the drill pattern, hole spacing, firing practice, energy distribution per hole and per blast, and interdepartmental accountability/responsibility for the entire process.

It is also recommended that the 220 m Elev sill pillar extraction should be investigated. Mining of this and future sill pillars should be well understood and planned as it presents upside potential to the mine cash flow.

Relative to mine planning, mine budgeting and cost control, mine reconciliation, ground control management plan, equipment maintenance plan, and operational KPI's, P&E recommends the establishment of RACI (responsibility, accountability, controls, and implementation) charts with clear deliverables.

1.20.4 Processing Plant and Tailings Storage

The grinding circuit has more than adequate capacity to handle the tonnages planned for the Project. In view of this it may be advisable to investigate whether it would be beneficial to grind finer.

Further work should be carried out on Ernesto material to ascertain the reasons for the lower overall recovery compared to Lavrinha and Pua-a-Pique. The work should investigate using finer grinds, increased cyanide levels and a trade-off study should be performed to confirm the industrial benefits of using Leach Aid in the CIL process. Since the plant has more than enough capacity to grind finer, a series of tests should be performed to establish the optimum grind size for Ernesto ore, and then to establish the optimum leach conditions.

The following process plant recommendations are also provided:

- Continue with optimization efforts around reagent dosage, focusing on the two operating regimes outlined in the study.
- Review operating manuals to better control densities in the process, especially important for soft ores with high amounts of fines. This improvement needs to be focused at the E-Cat stage and CIL.
- Review the existing SAG mill control logic as the ore to be fed from all deposits is softer than originally expected. This logic would target the use of SAG mill speed and SAG pressure to prevent liner damage in situations where load cannot be built within the SAG mill.

Finalize the Tierra Group study, which includes a trade-off assessment of using waste rock instead of saprolite to build the next tailings storage facility raise. This study includes a better characterization of the acid generation potential testwork on the waste rock.

1.20.5 Environmental

There have been no ARD characterization tests done on tailings or Lavrinha waste rock, and it is recommended that confirmatory acid rock drainage testwork for waste rock in all mine areas be carried out, and similarly for the tailings.

It is also recommended that supporting studies and comprehensive closure plan development be initiated within the first year of operation.

2.0 INTRODUCTION

This report titled "Feasibility Study and Technical Report on the EPP Project, Mato Grosso, Brazil" ("Report" or "Technical Report"), was prepared to provide Aura Minerals Inc. ("Aura" or the "Company") a National Instrument 43-101 ("NI 43-101") Feasibility Study Technical Report on the Ernesto/Lavrinha/Pau-a-Pique Deposits ("EPP Project" or "Project" or "Property"), located in the southwest of Mato Grosso state, near Pontes e Lacerda in Brazil. The EPP Property is 100% beneficially owned by Aura. Aura is a public, TSX listed, company trading under the symbol "ORA", with its head office located at:

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Aura, through its Brazilian subsidiaries, acquired the EPP Project from Yamana Gold Inc. ("Yamana") in June 2016. The Project was initially studied by Yamana from 2009 to 2011, and was put into production in 2013 for approximately two years before being placed on care and maintenance in late 2014.

The EPP Project is the third gold mining operation owned by Aura in this specific region of Brazil. The Company currently owns the operating Sao Francisco gold mine (production since 2006) near the town of Pontes e Lacerda and had the Sao Vicente gold mine that ceased operations in 2014 (production since 2009).

The EPP Project consists of three deposits, two of which have been planned as underground mining operations and the third of which is planned as an open pit operation. Three additional areas will be evaluated in 2017 and 2018.

- The Lavrinha open pit deposit and the Ernesto underground deposit are located approximately 60 kilometres ("km") south of the Company's Sao Francisco mine and 12 km south of the town of Pontes e Lacerda. The Project's process plant is located at Ernesto.
- The Pau-a-Pique underground deposit is located approximately 47 km south of the Ernesto and Lavrinha deposits and process plant.
- Three exploration areas (Nosde, Japones and Pombihnas) are within 5 km of the process plant.

Aura is a Canadian-based company, located in Toronto, Ontario, that is focused on the exploration, development and operation of gold and base metal projects in the Americas.

This Report was prepared by P&E Mining Consultants Inc. ("P&E") at the request of Mr. Fernando A. Cornejo, Vice-President, Projects of Aura and is considered current as of July 31, 2016.

The purpose of this Report is to provide an independent, NI 43-101 Feasibility Study Technical Report ("the Report") on the EPP Project. P&E understands that the Company may use this Report for internal decision making purposes and will be filed as required under TSX regulations. The Report may also be used by the Company to support financings.

The current P&E Updated Mineral Resource Estimate presented in this Report has been prepared in full conformance and compliance with the "CIM Standards on Mineral Resources and Reserves – Definitions and Guidelines" as referred to in NI 43-101 and Form 43-101F, Standards of Disclosure for Mineral Projects and in force as of the effective date of this Report.

Mr. Richard Routledge, P.Geo., and Mr. Andrew Bradfield, P.Eng., of P&E, each a qualified person under the terms of NI 43-101, conducted a site visit of the Property on June 18-21, 2015, and June 18-23, 2015, respectively. A data verification sampling program was conducted as part of the on-site review. Mr. David Orava, M.Eng., P.Eng., of P&E, a qualified person under the terms of NI 43-101, conducted a site visit of the Property on February 16-21, 2016, and May 13-20, 2016, to review underground mining at the Ernesto Deposit. Mr. Alex Veresezan, P.Eng., of P&E, a qualified person under the terms of NI 43-101, conducted a site visit of the Property on May 17-25, 2016, and October 24-29, 2016, to review underground mining at the Pau-a-Pique Deposit. Dr. Robert Mercer, Ph.D., P.Eng., of Knight Peisold Ltd., a qualified person under the terms of NI 43-101, conducted a site visit of the Property on June 18-22, 2015, to review geomechanical and hydrogeologic aspects. Mr. Marcelo Batelochi, AusIMM (CP), of MCB Consultants, a qualified person under the terms of NI 43-101, conducted a site visit of the Property on June 21-23, 2015, to review the Lavrinha Deposit.

2.1 SOURCES OF INFORMATION

In addition to the site visit, P&E held discussions with technical personnel from the Company regarding all pertinent aspects of the Project and carried out a review of available literature and documented results concerning the Property, including internal company technical reports and maps, published government reports, company letters, memoranda, public disclosure and public information, as listed in the References at the conclusion of this Report. Sections from reports authored by other participating consultants have been summarized in this Report, and are so indicated where appropriate. Table 2.1 presents the authors and co-authors of each section of the Report, who acting as a QP as defined by NI 43-101, take responsibility for those sections of the Report as outlined in the "Certificate of Author" attached to this Report.

TABLE 2.1
REPORT AUTHORS AND CO-AUTHORS

Qualified Person	Employer	Sections of Technical Report
Dr. Richard Sutcliffe, P.Geo.	P&E Mining Consultants	6, 7, 8, 23 and Co-author 1, 25, 26
Mr. David Burga, P.Geo.	P&E Mining Consultants	4 and Co-author 1, 9, 10, 25, 26
Ms. Jarita Barry, P.Geo.	P&E Mining Consultants	Co-author 1, 11, 12, 25, 26
Mr. Marcelo Batelochi, AusIMM (CP)	MCB Consultants	Co-author 1, 9, 10, 11, 12, 14, 15, 16, 21, 25, 26
Mr. Richard Routledge, P.Geo.	P&E Mining Consultants	Co-author 1, 11, 12, 14, 25, 26
Mr. Eugene Puritch, P.Eng.	P&E Mining Consultants	Co-author 1, 14, 25, 26
Mr. Andrew Bradfield, P.Eng.	P&E Mining Consultants	2, 3, 19, 22, 24 and Co-author 1, 12, 16, 25, 26
Mr. Fernando A. Cornejo, P.Eng.	Aura Minerals Inc.	17, Co-author 1, 18, 21, 25, 26
Mr. Matthew Fuller, CPG	Tierra Group International Inc.	Co-author 1, 18, 25, 26

TABLE 2.1 REPORT AUTHORS AND CO-AUTHORS		
Qualified Person	**Employer**	**Sections of Technical Report**
Ms. Diane Lister, P.Eng.	Altura Environmental Consulting	5, 20 and Co-author 1, 25, 26
Mr. David Orava, P.Eng.	P&E Mining Consultants	Co-author 1, 15, 16, 25 ,26
Mr. Alexandru Veresezan, P.Eng.	P&E Mining Consultants	Co-author 1, 15, 16, 21, 25, 26
Dr. Robert Mercer, P.Eng.	Knight Piesold Ltd.	Co-author 1, 16, 25, 26
Mr. Bradley Howe, P.Eng.	Paterson & Cooke Canada Inc.	Co-author 1, 16, 25, 26
Mr. Graham Holmes, P.Eng.	Jacobs	13 and Co-author 1, 25, 26

This Technical Report is prepared in accordance with the requirements of NI 43-101 and in compliance with Form NI 43-101F1 of the Ontario Securities Commission ("OSC") and the Canadian Securities Administrators ("CSA"). The Mineral Resource Estimate is prepared in compliance with the CIM Definitions and Standards on Mineral Resources and Mineral Reserves that were in force as of an effective date of July 31, 2016.

2.2 UNITS AND CURRENCY

Unless otherwise stated, all units used in this Report are metric. Gold ("Au") assay values are reported in grams of metal per tonne ("g/t").

The US dollar is used throughout this Report unless otherwise specified. All metal prices are stated in US dollars.

The coordinate system used by Aura for locating and reporting drill hole information is the Universal Transverse Mercator coordinate system ("UTM"), the datum used is SAD 69, zone 21 south. The coordinates for the centre of the Property claim block are 257,000 E, 8,303,000 N. Maps in this Report use either the UTM coordinate system or latitude and longitude.

2.3 GLOSSARY AND ABBREVIATION OF TERMS

The following list shows the meaning of the abbreviations for technical terms used throughout the text of this Report.

Abbreviation	**Meaning**
"3D"	Three Dimensional
"AA"	Atomic Absorption
"ASL"	Above Sea Level
"Au g/t"	Grams Of Gold Per Tonne
"Au"	Gold
"Aura"	Aura Minerals Inc.
"CA"	Certificate of Authorization
"Capex"	Capital Cost Expenditure
"CDN"	Canadian
"CDN$"	Canadian Dollars
"CIM"	Canadian Institute Of Mining, Metallurgy And Petroleum

"cm"	Centimetre(s)
"Company"	Aura Minerals Inc.
“CRM”	Certified Reference Material
"CSA"	Canadian Securities Administrators
"Cum"	Cumulative
"DDH"	Diamond Drill Hole
"DGPS"	Differential Global Positioning System
"E"	East
“EPP”	Ernesto/Pau-a-Pique Project
“FW”	Footwall
"g/t"	Grams Per Tonne
"GPS"	Global Positioning System
"ha"	Hectare(s)
“hr”	Hour
“hp”	Horsepower
“HW”	Hangingwall
"k"	Thousands
"k$"	Thousands Of Dollars
"km"	Kilometre(s)
“KP”	Knight Piesold Ltd.
"kt"	Thousands of Tonnes
“kw”	Kilowatt
LiDAR	Light Detection and Ranging Survey
"M"	Million
"m"	Metre(s)
"M$"	Millions Of Dollars
"Ma"	Millions Of Years
"mm"	Millimetre(s)
“MW”	Megawatt
"N"	North
"N/A"	Not Applicable
"NE"	North-East
"NI 43-101"	National Instrument 43-101
"NN"	Nearest Neighbour
"OK"	Ordinary Kriging
“Opex”	Operating Cost Expenditure
"opt"	Troy Ounces Per Ton
"OSC"	Ontario Securities Commission
“oz Au/T”	Troy Ounces Gold Per Ton
“P&E”	P&E Mining Consultants Inc.
“P&C”	Paterson & Cooke Canada Inc.
“%m”	Mass Percent Solids
“Project”	The EPP Deposits
“Property”	The EPP Concessions
“RC”	Reverse Circulation Drilling
"QA/QC"	Quality Assurance/Quality Control
"QC"	Quality Control
"QP"	Qualified Person as Defined By Canadian National Instrument NI 43-101 Standards Of Disclosure for Mineral Projects
"S"	South
"SEDAR"	Website Developed by the CRA, that Provides Access to Public Securities Documents and Information Filed by Public Companies and Investment Funds in Canada
"t"	Metric Tonne(s)
"t/m^3"	Tonnes per Cubic Metre
“tpd”	Tonnes Per Day

"tph"	Tonnes Per Hour
"W"	West

3.0 RELIANCE ON OTHER EXPERTS

P&E has assumed that all of the information and technical documents listed in the References section of this Technical Report are accurate and complete in all material aspects. While P&E has carefully reviewed all of the available information presented, P&E cannot guarantee its accuracy and completeness. P&E reserves the right, but will not be obligated, to revise the Report and conclusions therein if additional information becomes known to P&E subsequent to the date of this Report.

Copies of the tenure documents, operating licenses, permits, and work contracts were not reviewed. Information on tenure was obtained from Aura and included a legal due diligence opinion supplied by Aura's Brazilian legal counsel, Mr. José Henrique Nunes Paz, and another legal due diligence opinion supplied by Aura's Canadian VP Legal Affairs Mr. Ryan Goodman. P&E has relied upon tenure information from Aura and has not undertaken an independent detailed legal verification of title and ownership of the EPP Project. P&E has not verified the legality of any underlying agreement(s) that may exist concerning the licenses or other agreement(s) between third parties but has relied on, and believes it has a reasonable basis to rely upon Aura to have conducted the proper legal due diligence.

P&E has relied upon Brazilian taxation analysis in the Project financial model by Mr. Clodomildo P. de Sousa and Mr. José Antonio Teixeira Pires, both of Aura.

A draft copy of this Feasibility Study Technical Report has been reviewed for factual errors by Aura and all Qualified Persons. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statement and opinions expressed in this Report are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of this Report.

4.0 PROPERTY DESCRIPTION AND LOCATION

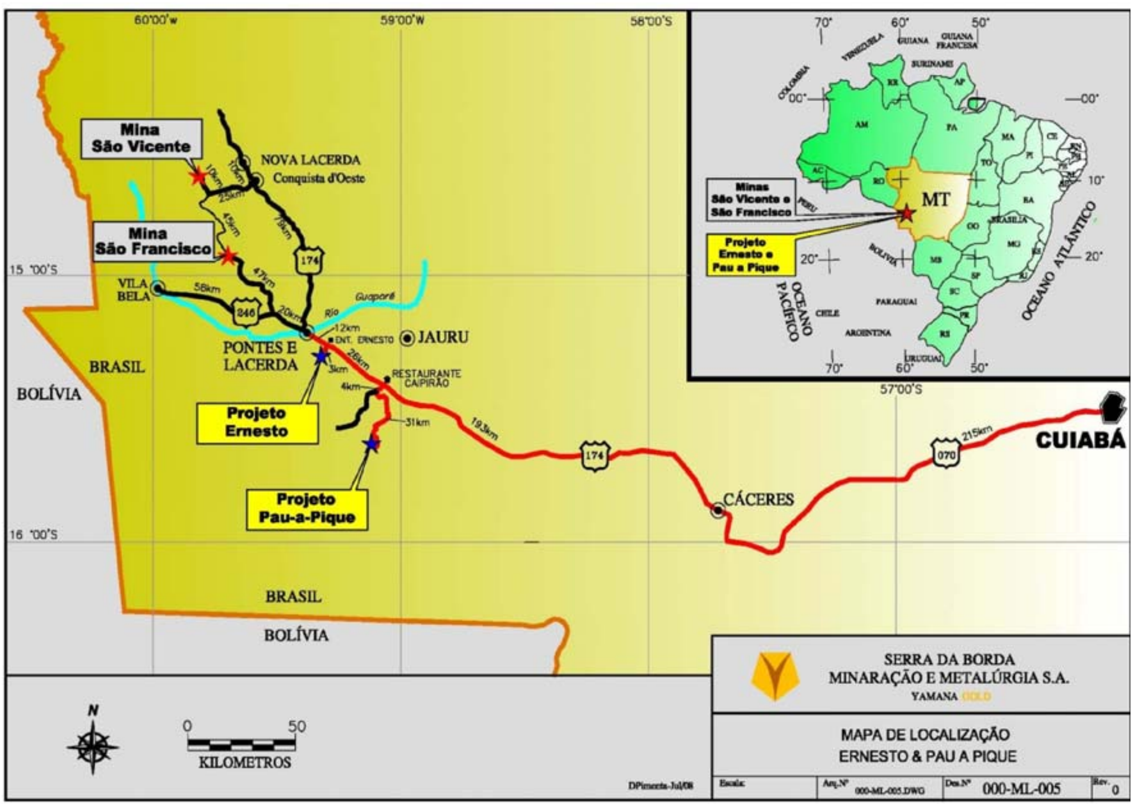
4.1 PROPERTY LOCATION

The Ernesto, Lavrinha and Pau-a-Pique gold deposits are near the town of Pontes e Lacerda, about 450 km west of Cuiabá, which is the capital of the Brazilian state of Mato Grosso. The Ernesto Deposit is approximately 12 km southeast of Pontes e Lacerda. The Project locations are shown in Figure 4.1.

The Ernesto and Lavrinha Deposits are contiguous and are reached from Pontes e Lacerda by paved road BR-174, which crosses within 2 km of the Project, and by a network of good gravel and dirt roads that offer year-round access for two-wheel drive vehicles. The Pau-a-Pique Deposit is located approximately 47 km southwest of Ernesto, and can be reached by a dirt road that runs parallel to BR-174. Figure 4.2 shows the location of the Ernesto, Lavrinha and Pau-a-Pique Properties.

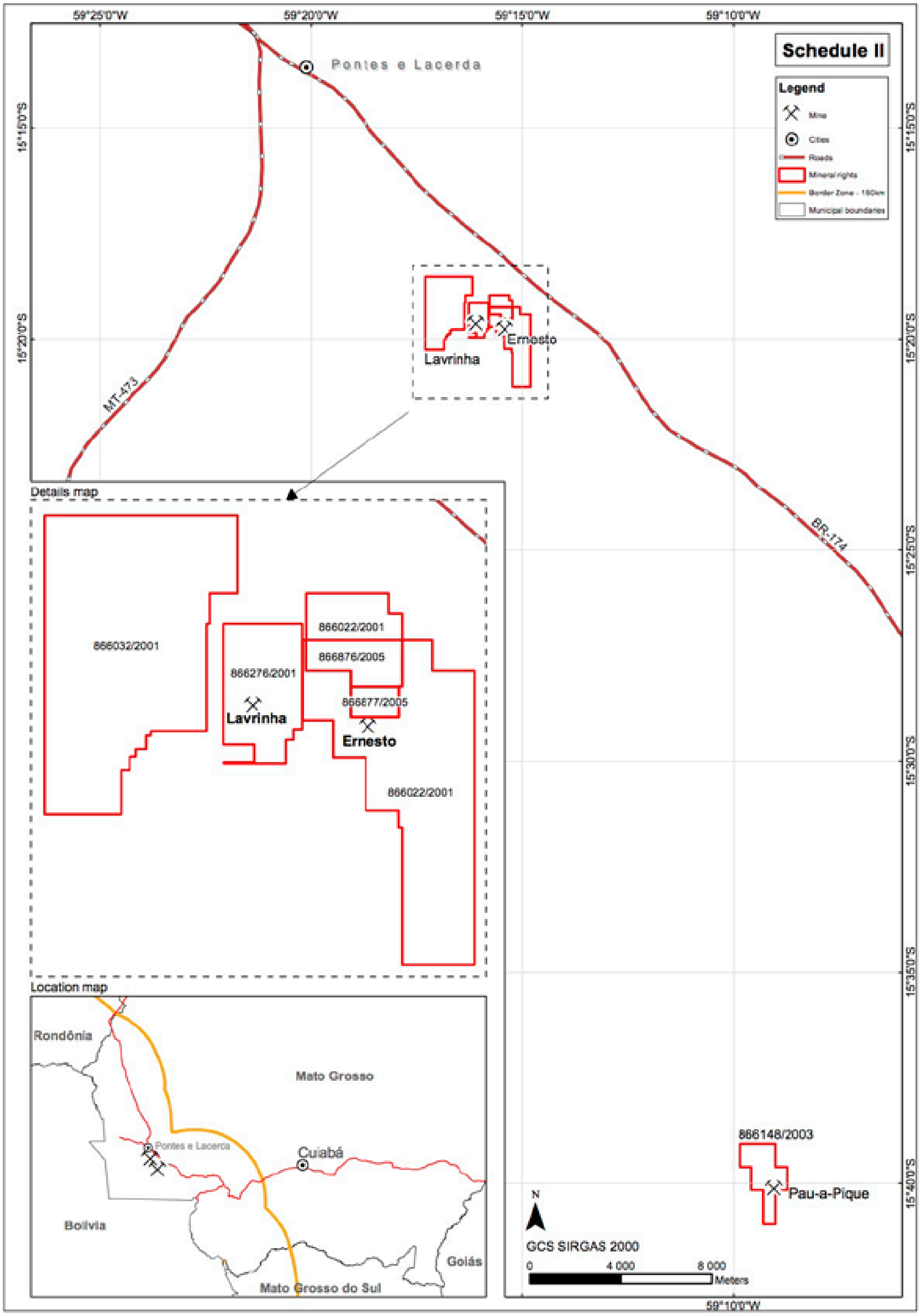
Pontes e Lacerda city has a local airport that can be used by small business aircraft. In 2010, the Pontes e Lacerda's population recorded by IBGE (Brazilian Government Census Institute) was 41,408.

Figure 4.1 Location of the Pau-a-Pique and Ernesto Project Sites



(**Source:** *MCB report 2014)*

Figure 4.2 Location of the Ernesto, Lavrinha and Pau-a-Pique Concessions



*(**Source:** Aura)*

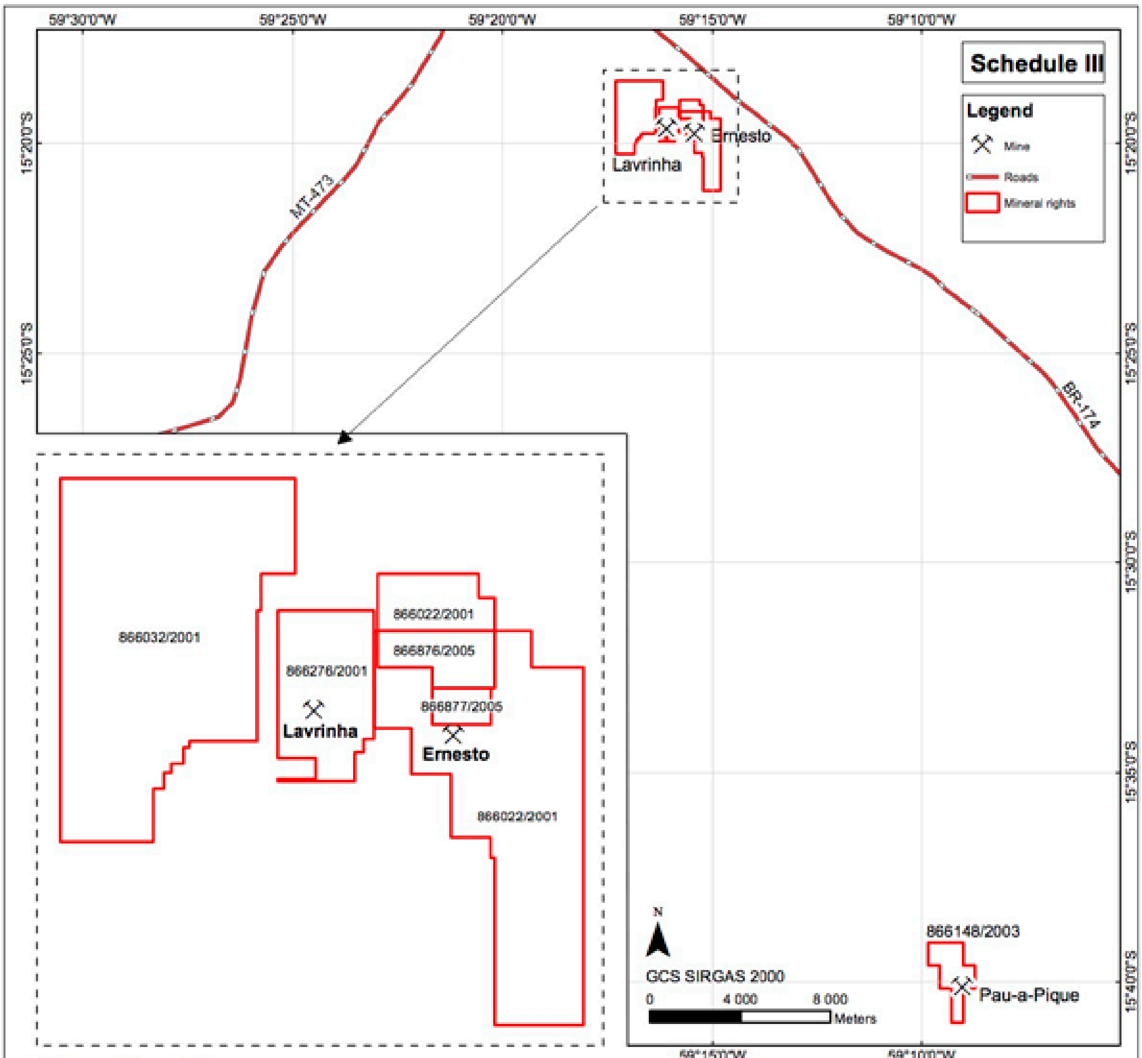
4.2 PROPERTY DESCRIPTION AND TENURE

The Ernesto Property comprises 1,412.89 ha of 6 mining rights held (legally or beneficially) by Mineração Apoena S.A. (“Apoena”), a company wholly-owned by Aura. The mining rights that cover the targets in this Report are listed in Table 4.1. A claims map is presented in Figure 4.3 that includes the coordinates of concessions, in Latitude and Longitude.

TABLE 4.1
MINING RIGHTS OF THE ERNESTO DISTRICT

Target	DNPM Process No.	Petitioner	Area (ha)	Status
Ernesto	866.022/ 2001	SBMM	375.49	Mining Concession
Ernesto	866.877/ 2005	SBMM	15.96	Mining Concession
Ernesto	866.876/ 2005	SBMM	41.63	Mining Concession
Lavrinha	866.276/ 2001	SBMM	111.63	Application for Mining Concession
Pau-a-Pique	866.148/ 2003	SBMM	374.99	Mining Concession
Japonês/Nosde	866.032/ 2001	SBMM	493.19	Application for Mining Concession

Figure 4.3 Coordinates of the Ernesto, Lavrinha and Pau-a-Pique Concessions



Vertices of Mineral Rights

866022/2001

Latitude	Longitude
-15°19'31"113	-59°15'13"474
-15°19'14"846	-59°15'13"474
-15°19'14"846	-59°15'02"958
-15°19'25"329	-59°15'02"958
-15°19'25"329	-59°14'48"224
-15°21'07"742	-59°14'48"224
-15°21'07"742	-59°15'13"461
-15°20'19"908	-59°15'13"461
-15°20'19"908	-59°15'14"697
-15°20'14"091	-59°15'14"697
-15°20'14"091	-59°15'26"073
-15°19'55"689	-59°15'26"073
-15°19'55"689	-59°15'37"392
-15°19'42"688	-59°15'37"392
-15°19'42"688	-59°15'48"123
-15°19'14"830	-59°15'48"123
-15°19'14"830	-59°15'46"967
-15°18'58"563	-59°15'46"967
-15°18'58"563	-59°15'18"079
-15°19'05"447	-59°15'18"079
-15°19'05"447	-59°15'13"474
-15°19'14"846	-59°15'13"474
-15°19'14"846	-59°15'46"997
-15°19'25"287	-59°15'46"997
-15°19'25"287	-59°15'31"341
-15°19'41"544	-59°15'31"341
-15°19'41"544	-59°15'14"643
-15°19'31"113	-59°15'14"643
-15°19'31"113	-59°15'13"474

866032/2001

Latitude	Longitude
-15°18'31"297	-59°17'17"762
-15°18'31"297	-59°16'10"718
-15°18'58"564	-59°16'10"718
-15°18'58"564	-59°16'20"487
-15°19'09"007	-59°16'20"487
-15°19'09"007	-59°16'21"627
-15°19'46"411	-59°16'21"627
-15°19'46"411	-59°16'40"958
-15°19'48"129	-59°16'40"958
-15°19'48"129	-59°16'42"588
-15°19'52"767	-59°16'42"588
-15°19'52"767	-59°16'46"069
-15°19'55"362	-59°16'46"069
-15°19'55"362	-59°16'48"249
-15°19'59"991	-59°16'48"249
-15°19'59"991	-59°16'51"306
-15°20'15"413	-59°16'51"306
-15°20'15"413	-59°17'17"762
-15°18'31"297	-59°17'17"762

866276/2001

Latitude	Longitude
-15°19'51"202	-59°16'15"727
-15°19'09"044	-59°16'15"727
-15°19'09"044	-59°15'48"209
-15°19'45"776	-59°15'48"209
-15°19'45"776	-59°15'51"233
-15°19'49"546	-59°15'51"233
-15°19'49"546	-59°15'53"842
-15°19'57"740	-59°15'53"842
-15°19'57"740	-59°16'15"684
-15°19'57"209	-59°16'15"684
-15°19'57"209	-59°16'04"920
-15°19'51"202	-59°16'04"920
-15°19'51"202	-59°16'15"727

866876/2005

Latitude	Longitude
-15°19'14"848	-59°15'13"474
-15°19'31"113	-59°15'13"473
-15°19'31"113	-59°15'14"643
-15°19'31"116	-59°15'14"643
-15°19'31"116	-59°15'31"341
-15°19'25"293	-59°15'31"341
-15°19'25"287	-59°15'31"341
-15°19'25"287	-59°15'46"997
-15°19'14"849	-59°15'46"997
-15°19'14"848	-59°15'13"474

866877/2005

Latitude	Longitude
-15°19'41"544	-59°15'14"643
-15°19'41"544	-59°15'31"338
-15°19'31"116	-59°15'31"338
-15°19'31"116	-59°15'14"643
-15°19'41"544	-59°15'14"643

866148/2003

Latitude	Longitude
-15°39'04"142	-59°09'00"615
-15°39'36"676	-59°09'00"615
-15°39'36"676	-59°08'43"826
-15°40'09"208	-59°08'43"826
-15°40'09"208	-59°09'00"614
-15°40'58"009	-59°09'00"614
-15°40'58"009	-59°09'17"403
-15°40'09"208	-59°09'17"403
-15°40'09"208	-59°09'34"192
-15°39'36"676	-59°09'34"192
-15°39'36"676	-59°09'50"981
-15°39'04"142	-59°09'50"981
-15°39'04"142	-59°09'00"615

Mine Coordinates

Mine	Latitude	Longitude
Ernesto	-15°19'44"383	-59°15'25"383
Lavrinha	-15°19'37"084	-59°16'05.221
Pau-a-Pique	-15°40'05.887	-59°09'01"783

*(**Source:** Aura)*

On April 30, 2015, Aura announced its agreement with Serra da Borda Mineração e Metalurgia ("SBMM"), a company affiliated with Yamana, to acquire, upon completion of certain conditions, the assets and liabilities of the Project. On June 23, 2016, the Company announced that it had completed the acquisition and has assumed operation control of the Project. Terms of the agreement were that Aura will issue to Yamana:

- 2,000,000 common shares of Aura,
- 3,500,000 common share purchase warrants, and
- A 2% NSR on the first 1,000,000 gold ounces produced from the Project, and thereafter, a 1% NSR on gold ounces produced from the Project.

In order to facilitate the acquisition of the Project, the previous owner, SBMM, a company affiliated with Yamana, made available to the Company's operating entity a working capital facility of up to US$9M (the "Working Capital Facility"). The Working Capital Facility bears interest at 4% per annum on the outstanding balance. The funds advanced from the Working Capital Facility have been invested in the capital, care-and-maintenance and engineering requirements of the Project to restart the Project and to complete the NI 43-101 technical reporting. The Working Capital Facility is expected to be repaid with the initial free cash flow from the Project or will be payable in full by April 30, 2018. Should the Project not enter into production and the Company not have sufficient funds to repay the Working Capital Facility on the due date, such amount outstanding will, at the option of Yamana, be converted into common shares of the Company at a 10% discount over the 20 day VWAP of the Company's common shares based on the period prior to the due date. At no point in time may Yamana own, beneficially or otherwise, greater than 19.9% of the issued and outstanding common shares of the Company.

Aura provided a letter dated July 31, 2016, from Ryan Goodman, VP of Legal Affairs for Aura, that states that SBMM was acquired through Aura's 49% owned subsidiary, Mineração Apoena S.A. 51% of Apoena is owned by Vila Bela Participacoes Ltda. Vila Bela's issued and outstanding quota capital consists of 1,000 quotas and is owned by two Brazilian Quotaholders,each with 50% ownership and 500 quotas. The Vila Bela ownership is due to a requirement of companies operating in certain areas in Brazil near the border to have 51% Brazilian ownership.

Aura has Quota Call Option Agreements with both Quotaholders. Both agreements feature the same terms, namely that Aura can become the holder of quotas, with all rights and obligations they represent, for the amount equal to the nominal value of R$1.00 (one Real) per each quota multiplied by the number of quotas held by the Quotaholders.

4.3 ROYALTIES

As part of the purchase agreement, a 2% NSR royalty is payable to Yamana on gold ounces produced from the Project with respect to up to 1,000,000 collective ounces of gold, and thereafter, a 1% NSR on gold ounces produced from the Project.

A 0.5% NSR royalty is due to each landowner (one for Ernesto/Lavrinha, and one for Pau-a-Pique), proportional to his surface rights. The Mining Code provides that landowners are entitled to a royalty equivalent to 50% of the royalty due the government (the Financial Compensation for Exploitation of Mineral Resources – "CFEM"). The CFEM is calculated based on net income resulting from the sales of the mineral product, deducting taxes and costs of transport and insurance. In the case of gold, the rate of CFEM is 1%, thus the landowner royalty is 0.5%.

4.4 ENVIRONMENTAL

To the best of knowledge and belief of P&E, after reasonable inquiry, P&E is not aware of any environmental litigation or pending fines associated with the EPP Project.

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

5.1 ACCESS

The Ernesto Property can be accessed from Pontes e Lacerda by paved road BR-174 for 12 km and then following 2 km of gravel and dirt roads that offer year-round access to the Project. The Lavrinha Property is accessed from Pontes e Lacerda by the same roads used to access the Ernesto Property.

The Pau-a-Pique Deposit is approximately 73 km by road from Pontes e Lacerda, and approximately 47 km by dirt road from Ernesto.

5.2 CLIMATE

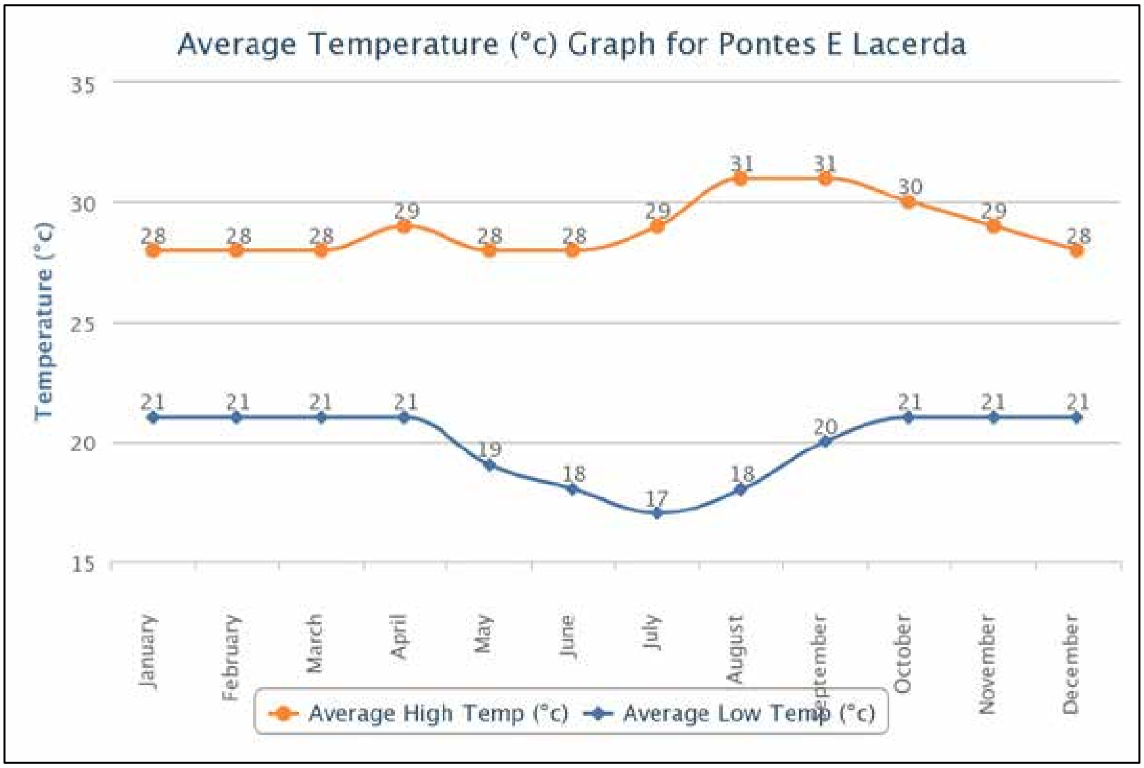
The climate in the Project area is suitable for year-round mining. The region boasts hot, tropical and semi-humid climate of the Mato Grosso province in Central West Brazil. The area has two well-defined seasons: one dry season, usually from April to October, when during the cooler dry winter the temperature averages 20°C to 22°C, and a season that receives large amounts of rain during November to March, with daily maxima ranging from 30°C to 43°C. Average monthly temperatures for Pontes E Lacerda is presented in Figure 5.1 and average rainfall data is presented in Figure 5.2. Weather data is summarized in Table 5.1.

TABLE 5.1
WEATHER DATA SUMMARY

	Pontes e Lacerda e Porto Esperidião
Annual average temperature (^{o}C)	22-24
Annual average humidity (%)	75-85
Annual average precipitation (mm)	1,440
Annual evaporation rate (mm)	1,001-1,100
Annual average potential evapotranspiration (mm)	1,201-1,400
Average summer temperature (^{o}C)	24-26
Average summer highest temperature (^{o}C)	32-34
Approx summer fraction of annual precipitation	0.7
Average winter temperature (^{o}C)	20-22
Average winter lowest temperature (^{o}C)	14-16

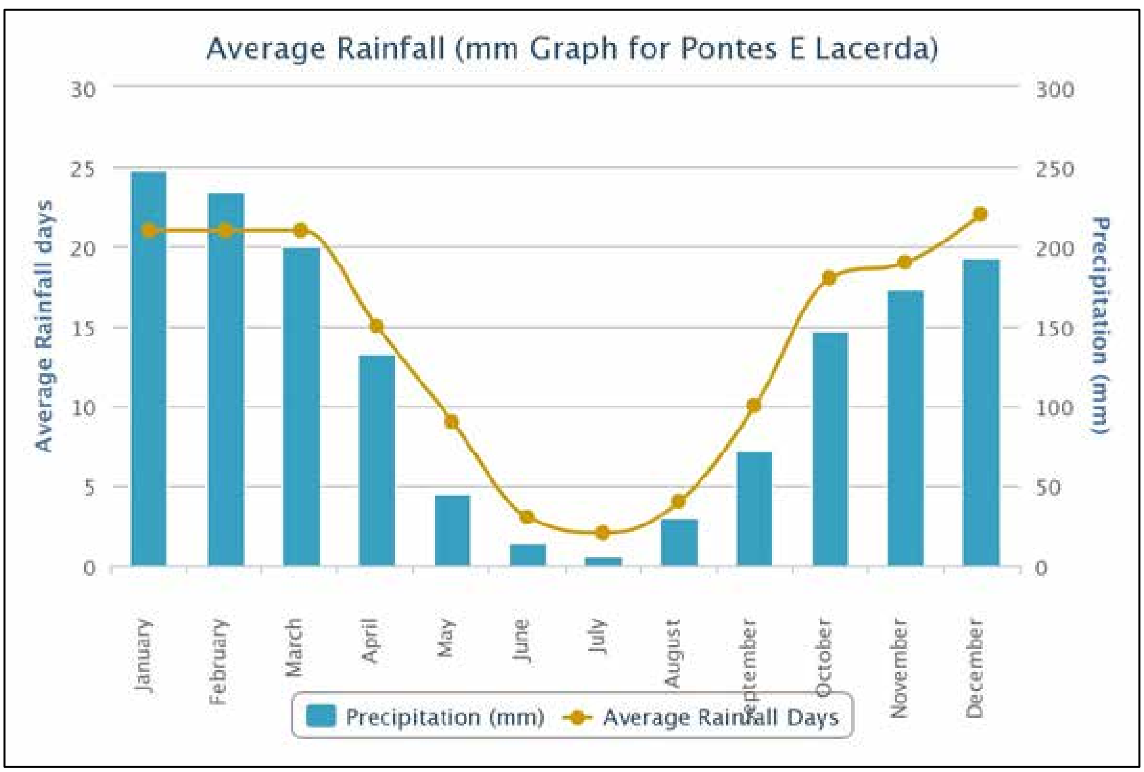
(Source: Ausenco, 2010)

Figure 5.1 Average Temperature Data, Pontes E Lacerda, Brazil



*(**Source:** www.worldweatheronline.com data averaged from 2000 to 2012)*

Figure 5.2 Average Rainfall Data, Pontes E Lacerda, Brazil



*(**Source:** www.worldweatheronline.com data averaged from 2000 to 2012)*

5.3 PHYSIOGRAPHY

The Ernesto Property is located in a range of hills that runs from northwest of Pontes e Lacerda to southeast of Pau-a-Pique (Figure 5.3). The terrain is comprised of rolling hills. The Ernesto District is covered by Amazon Forest, much of which has been cleared for livestock activity.

Locally, topographic features are characterized by flat relief and hilly highlands with elevation ranging between 280 m and 430 m. The Property is generally around 270 m above sea level.

Figure 5.3 Terrain and Relief, Ernesto and Pau-a-Pique Area (looking NW)

*(**Source:** Google Earth, 2016)*

5.4 LOCAL RESOURCES

Aura operates the São Francisco mine and operated the past-producing São Vincente mine until 2014, both in the vicinity of Pontes e Lacerda. It is thus likely that experienced personnel can be found in the local region or in the state capital Cuiabá (approximately 450 km to the east). Pontes e Lacerda city has a local airport that can be used by small aircraft. The nearest major airport is in Cuiabá.

5.5 INFRASTRUCTURE

Paved roadways exist between the deposits and the town of Pontes e Lacerda. The 47 km ore haulage route between Pau-a-Pique and state road BR-174 was established by the previous operator and utilized until mining ceased in late 2014. Construction of this gravel route mainly entailed upgrading of existing roads along with construction of some new sections. Aura is updating the landowner agreements for resumption of ore haulage; this process is well underway and no impediments are anticipated. Minor road maintenance work has been identified and will be carried out in late 2016.

The Ernesto Property contains a 130 tonnes per hour carbon-in-leach process plant, which includes crushing, milling and tailing facilities with power supplied from the national grid via a 12 km 138 kV transmission line from Pontes e Lacerda. The Ernesto Property also contains a gate house, administration offices, core shack, explosives storage facility, and the mined-out Ernesto open pit and waste rock dump. The Lavrinha Property is contiguous to Ernesto and does not contain any infrastructure. The Pau-a-Pique Property contains an underground mine that was operated by Yamana until late in 2014, and surface facilities for administration and maintenance.

Aura has existing surface rights over most of the Project area either via direct ownership or agreements with landowners. Negotiations are in process for a remaining parcel in Lavrinha and a small portion of the Pau-a-Pique Project area. There are no communities or permanent dwellings within the Project footprint.

6.0 HISTORY

Mineral exploration in the Pontes e Lacerda region began with the discovery of mineral wealth by the Portuguese bandeirantes in the late 17th and early 18th centuries. This was followed by a mining boom for gold and diamonds that peaked with exploitation of placer deposits in the Mato Grosso state in the mid-18th century (Machado and Figueiroa, 2001).

The recent history of the Ernesto and Pau-a-Pique gold deposits in Sections 6.1 and 6.2, below, is summarized from Ausenco's (2010) Feasibility Study report for Yamana.

6.1 ERNESTO AND LAVRINHA DEPOSITS

In the early 1980's, thousands of garimpeiros (artisanal miners) began recovering placer gold along the rivers and streams in the Project area; these placer deposits were exhausted by the late 1980's. In 1989, garimpeiros began mining weathered bedrock at Ernesto and surrounding areas. From these areas, approximately 60,000 oz of gold has reportedly been recovered to date. At Ernesto, garimpeiros reportedly produced 9,000 oz of gold from a small pit in a 3 m thick zone along a 200 m length and from underground workings accessed via seven declines extending 50 m to 60 m down-dip from the surface outcrop. The highest garimpeiro production was during 1991 when their best month reportedly yielded 1,600 oz of gold.

6.2 EXPLORATION HISTORY

Gold was first discovered at the Aguapeí Gold Belt by Portuguese settlers (paulistas frontiersmen) in the 18th century, around 1734, and it was mined from primary (mainly), colluvial, alluvial or placer deposits (first Gold Cycle in the Brazil Colonial times). The most significant primary gold deposits were discovered at places today known as Sao Francisco Xavier and São Vicente mines, Rio Galera, Santana, Nossa Senhora do Pilar, Aguapeí, Cágado, Santa Bárbara and Lavrinha. Since then, gold mining activities were interrupted due to difficulties in operation and exhaustion of alluvial deposits.

Modern gold mining began in 1984 during a second gold rush at Alto Guapore Gold Province (1984-1997). Artisanal miners, after exhaustion of alluvial and colluvial deposits, discovered several small primary gold deposits close to Pontes e Lacerda city, including Japonês, Nosde, Lavrinha, Ernesto (Copacel), Pombinhas and Cantina/Serra Azul deposits.

About 6,000 artisanal miners carried out a large number of small operations (including panning, small underground workings and small scale process plants) around Pontes e Lacerda, Vila Bela da Santíssima Trinidad and Porto Esperidião cities. Gold production data in this period are not accurate, but it is estimated that about 5-6 tons of gold was produced between 1990 and 1995. In 1992, these artisanal mining activities attracted the attention of several mining companies, including Copacel, Minopar, Anglo American, WMC, Madison do Brasil, TVX Gold/Paulo Abib and Mineração Santa Elina ("MSE").

Copacel and Minopar, local mining companies, were the first and main owners of exploration permits in the Ernesto District in the early 1990s. In 1992, Anglo American and WMC carried out intensive surface geochemical surveys along the belt, mainly stream sediment sampling. In 1993, Madison do Brasil, after acquisition of exploration permits from Copacel and Minopar, carried out a diamond drilling program at Japones, Nosde, Lavrinha and Ernesto targets. In 1994, Madison do Brasil company assigned its mineral rights and transferred control of the exploration permits to TVX Gold. TVX Gold, in 1995, carried out additional drilling campaigns. In the same year TVX Gold transferred its mineral rights to MSE to capitalize on other business priorities. During this time MSE drilled nine more exploratory drill holes for a total of 1,711.77 m at the Lavrinha Deposit and collected 683 samples.

6.3 PAU-A-PIQUE DEPOSIT

Similar to Ernesto, the Pau-a-Pique area was explored and mined in 18th century and mining activities were restarted in the 1980's by garimpeiros, working in alluvium and colluvium around the deposit. Later in the 1990's, the company 'Mineração Itapuã', developed an open pit and some small galleries. Small-scale garimpeiro mining still occurs in the vicinity, through the processing of alluvium and colluvium by garimpeiros associated with a cooperative that owns an area adjoining the Pau-a-Pique exploration permit.

6.4 HISTORIC EXPLORATION

6.4.1 Ernesto Deposit

The following summary of historical exploration is based on the Ausenco (2010) report. Artisanal mine workings in the Ernesto area were examined by the Brazilian company Mineração Santa Elina ("MSE"), Western Mining Corporation, Anglo American and other mining companies in 1991 and 1992. No significant programs were undertaken until 1994, when Madison acquired claims covering several garimpos and subsequently formed a 35/65 joint venture ("JV") with TVX Gold Inc. ("TVX"). TVX, as the JV operator, completed a program consisting of: regional aerial photo mapping at a scale of 1:5000; detailed mapping of several garimpo targets at a scale of 1:500; analysis of 1,160 trench samples; and analysis of 15 bulk samples (150 kg each). In late 1994, the JV drilled 46 diamond core holes (6,478 m) to test five targets and identified mineralization of interest in the Ernesto and Nosde targets. Resource estimates were made for both targets based on the drilling results.

In 1995, MSE became a third partner in the joint venture and the JV operator. MSE completed 24 vertical diamond drill holes (4,881 m) at Ernesto and 12 holes (1,359 m) in the Lavrinha target west of Ernesto. The MSE drilling helped establish and extend the down-plunge continuity of the Ernesto mineralization and a new resource estimate was prepared. The Project was subsequently abandoned in part due to declining gold prices. MSE maintained a core block of claims covering the northern down-plunge end of the Ernesto trend. After 1995, no exploration was done until Yamana consolidated and expanded the Ernesto area claims. Yamana's exploration is summarized in Sections 9 and 10 of this Report.

6.4.2 Lavrinha Deposit

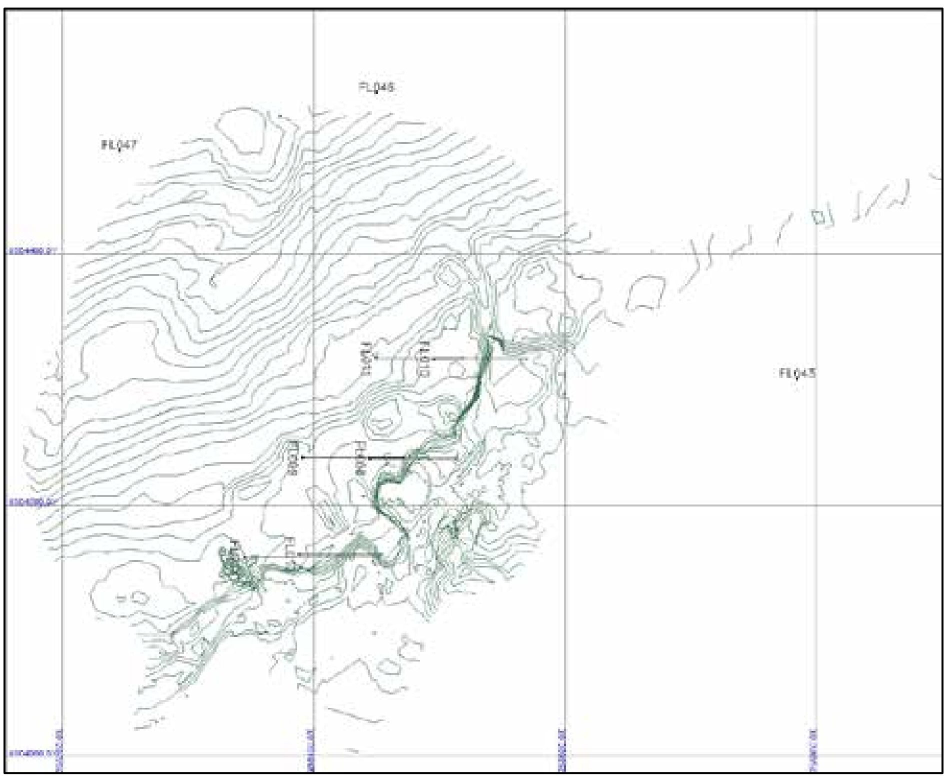
A drill campaign was carried out by TVX in 1994 and 1995 with the objective to identify satellite deposits associated with the Ernesto Deposit. This was aligned with the main TVX strategy to find gold deposits with "Large Volume/Low Grade" associated with sericitization and hydrothermal alteration.

Nine diamond drill holes were drilled for 1,711.77 m and 683 samples were analyzed for gold using fire assay. Depth of drilling varied between 100 m and 200 m, except the exploratory drill hole FL047 which went to 457.45 m. There were no survey measurements along the drill holes, therefore the holes were assumed to be at constant dip and azimuth. Six drill holes had an azimuth of 90° and dip -50° and three holes were vertical. The sample intervals of this drilling campaign were irregular as a result of varying exploration strategies to identify the deposit and mineralization and understand the exploration potential of the target. Table 6.1 and Figure 6.1 present the 1994 and 1995 drill campaign metres and locations.

TABLE 6.1
QUANTITATIVE SUMMARY OF 1994 AND 1995 LAVRINHA DRILLING CAMPAIGN

Hole ID	East	North	Elev	Az (o)	Dip (o)	Depth (m)	No. of Samples
FL008	256,444.3	8,304,238	517.55	90	-50	108.80	82
FL009	256,390.8	8,304,238	516.38	90	-50	126.45	75
FL010	256,495.7	8,304,317	515.51	90	-50	114.24	79
FL011	256,449.2	8,304,317	507.76	90	-50	109.32	63
FL012	256,388.2	8,304,161	517.4	90	-50	102.90	99
FL013	256,344.5	8,304,158	519.82	90	-50	167.40	175
FL045	256,784.4	8,304,301	475.09	0	-90	240.44	31
FL046	256,449.6	8,304,528	475.15	0	-90	2884.77	12
FL047	256,245.1	8,304,483	470.4	0	-90	457.45	67
Total	**9 Drill Holes**					**1,711.77**	**683**

Figure 6.1 Location Map of 1994 and 1995 Lavrinha Drilling Campaign



6.4.3 Pau-a-Pique Deposit

Exploration at Pau-a-Pique was initiated in 2005 by Yamana to follow up on earlier garimpeiro mining activity. Yamana's exploration is summarized in Sections 9 and 10 of this Report.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

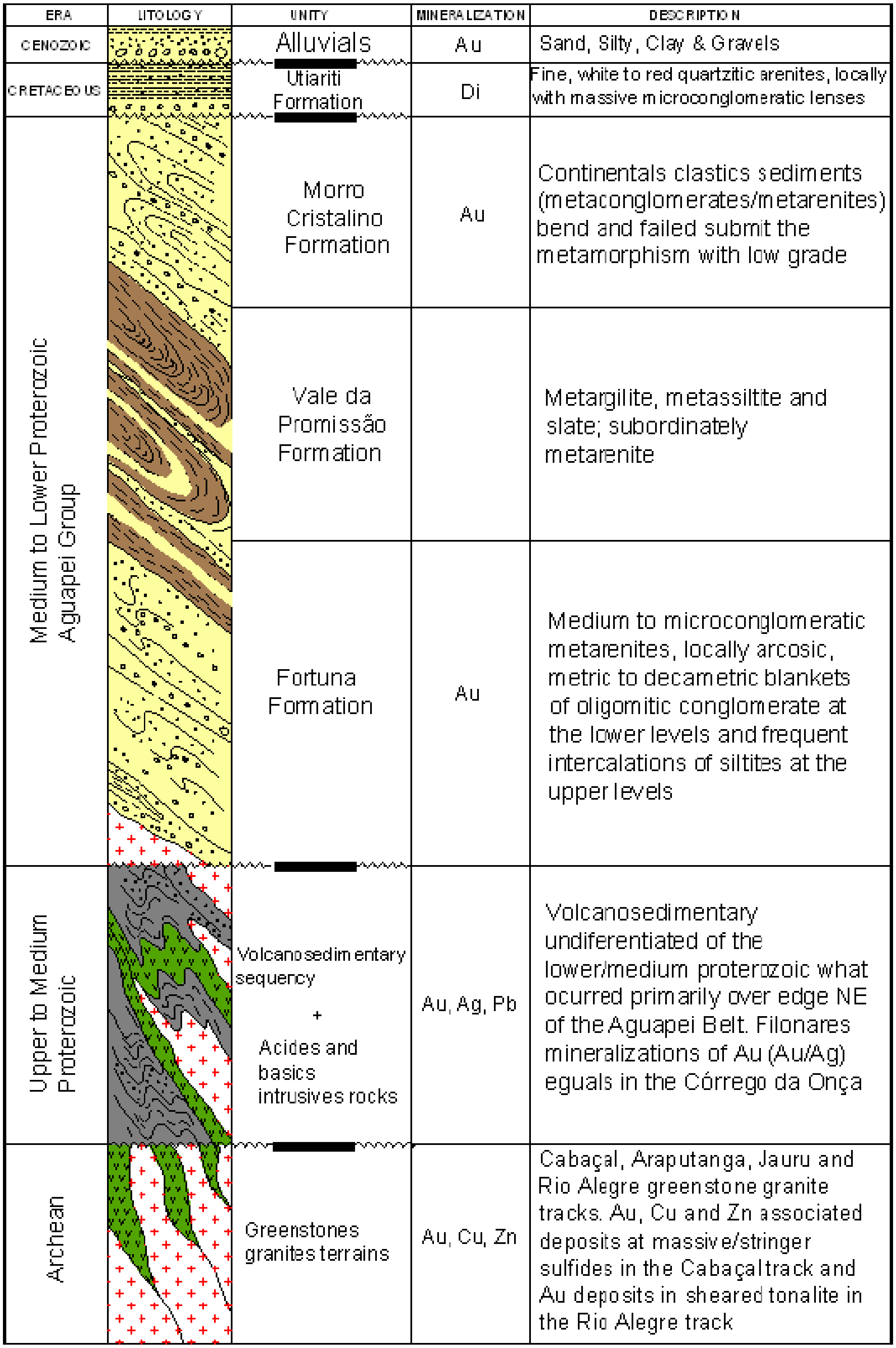
7.1 REGIONAL GEOLOGY

The Pau-a-Pique and Ernesto-Lavrinha Deposits are situated in the Middle Proterozoic (ca. 1.0 Ga) Aguapeí belt, a foreland fold and thrust belt,that overlies the Early Proterozoic Jauru (ca. 1.8 Ga) and Middle Proterozoic Rio Alegre (ca. 1.5 Ga) terrains (Geraldes et al. 2001). These terrains form part of the northwest trending Rio Negro-Juruena orogenic belt that extends for over 200 km in Mato Grosso State. The orogenic belt is the result of accretion of successive juvenile and continental Proterozoic magmatic arcs to the Archean rocks of the Amazonian Craton situated on the northeast side of the orogenic belt (Geraldes et al. 2001).

From east to west, over a distance of approximately 150 km, the Rio Negro-Juruena orogenic belt is subdivided into three major terrains (Leite and Saes, 2000). In the east, juvenile arc-related metavolcanic, metasedimentary and intrusive rocks form the Jauru terrain. Calc-alkaline granodiorite to evolved granite of the Santa Helena Batholith occurs as a central terrain and is intrusive into juvenile metavolcanic and related intrusive rocks of the Rio Alegre terrain in the west (Geraldes et al. 2001).

The linear Aguapeí Belt is a narrow northwest trending zone of shallow marine to fluvial siliciclastic sediments deposited between 1.0 and 1.4 Ga that has been deformed and metamorphosed during foreland folding and thrusting at 1.0 Ga. This tectonic event correlates with the Sunsas orogeny in Bolivia (Leite and Saes 2000). The basement beneath the Aguapeí Group is composed of volcanosedimentary rocks of the Rio Alegre Complex in the west and the Santa Helena Batholith in the east (Figure 7.1). The lower contact of the Aguapeí belt with the basement is an unconformity and usually tectonically discordant and mylonitised. This contact is an important regional control on gold mineralization.

Figure 7.1 Regional Stratigraphic Western Mato Grosso State, Brazil, Showing Position of Units that Host Gold Deposits

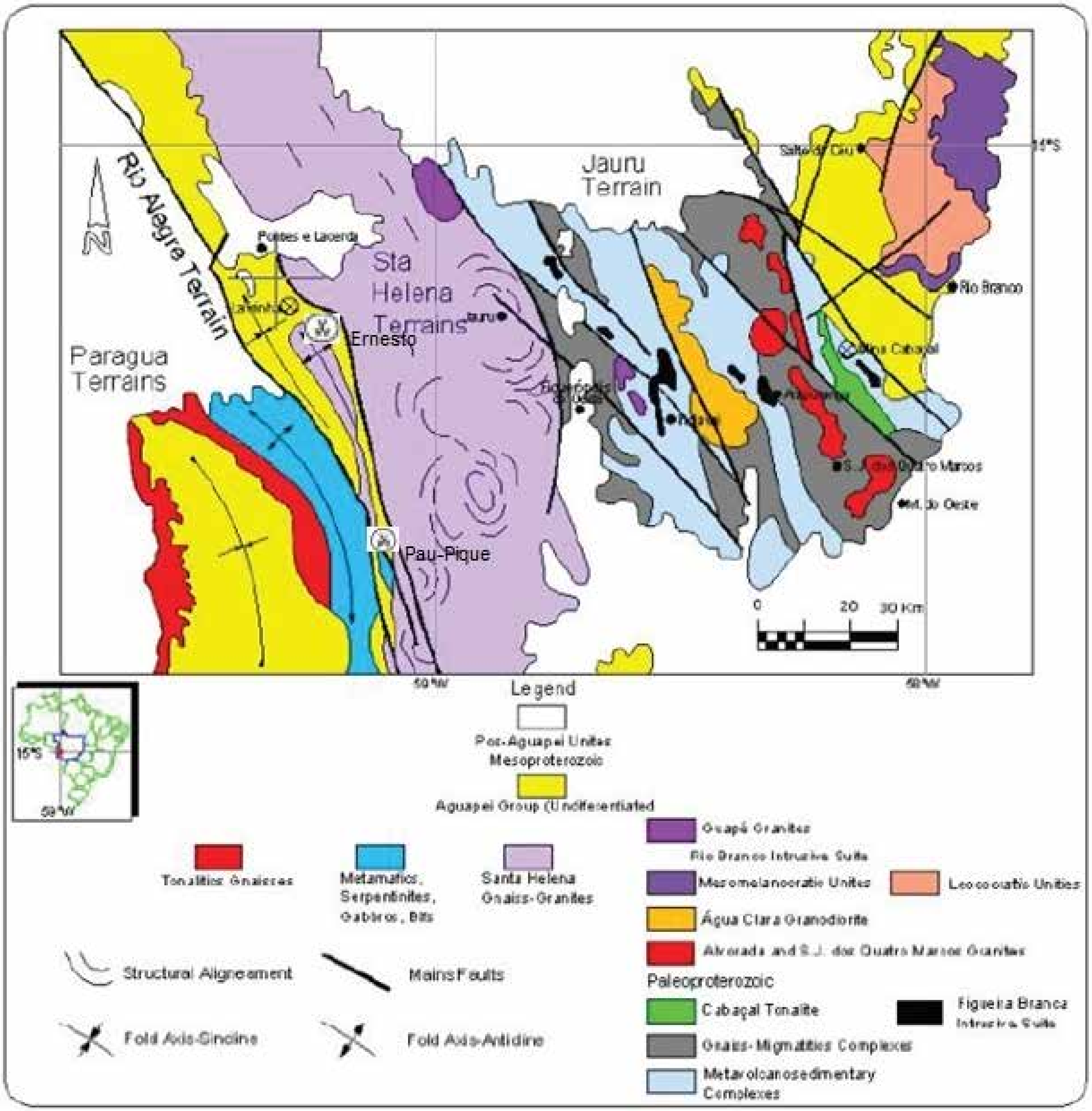


The Aguapeí belt itself can be subdivided longitudinally (north-to-south) into three compartments (Figure 7.2):

- The Northern Compartment from São Vicente to the Pillar Ruins garimpo is dominated by tightly folded Aguapeí sediments in the basement;
- The Central Compartment around Pontes e Lacerda is dominated by the Pontes e Lacerda Volcanosedimentary Sequence overthrust onto the Aguapeí Group; and
- The Southern Compartment from Ernesto–Lavrinha to Pau-a-Pique and GP3 is dominated by NNW–SSE strike synclinal linear outliers of Aguapeí rocks overlying Santa Helena granitoids.

Saes and Leite (1993) have also subdivided the Aguapeí basin laterally (west to east) into four structural domains. The westernmost Domain A is marked by unfolded or openly folded sediments with little faulting. The Domain B is marked by open cylindrical flexural-slip folding with axial planar cleavage and reverse faults. The Domain C is marked by tight, asymmetrical folds with a slightly penetrative axial planar cleavage and reverse faults. The eastern Domain D has reverse faults, isoclinal folds and strong penetrative axial planar cleavage, often crenulated. The distribution of these domains may be related to the occurrence of the major basin bounding fault (the Morro Solteiro Fault) on the eastern margin acting as a buttress during compressional deformation.

Figure 7.2 Regional Geology and Structure of Western Mato Grosso State, Brazil



7.2 LOCAL GEOLOGY

The description of local geology is largely summarized from the Ausenco (2010) report.

Volcano-sedimentary rocks of the Rio Alegre terrain represent the oldest unit in the region of the property. This terrain has a NNW trend and is subdivided into the Minouro, Santa Isabel and São Fabiano Formations. In the lower formation, melanocratic, equigranular, meta-basalts are associated with meta-cherts and banded iron-formation ("BIF"). The intermediate formation is composed of metadacite, meta-rhyolite and lesser meta-pyroclastic rocks. The top formation is composed of muscovite-schist, metacherts and BIF. The metamorphism is low-grade greenschist facies and, locally, epidote-amphibolite. The metadacites have U-Pb zircon ages of 1.52 to 1.47 Ga and considered to represent a juvenile arc environment (Geraldes et al. 2001).

The Santa Helena Batholith consists of calc-alkaline tonalite and granodiorite to evolved granitic compositions. The Batholith is elongated in a NNW direction and represents the largest felsic magmatic event in the SW portion of Mato Grosso State. The tonalite Pau-a-Pique intrusive phase has a U-Pb zircon age of 1481 ± 47 million years (Geraldes et al. 2001). The Batholith has foliated, fine to coarse grained, equigranular to porphyritic textures with biotite and subordinate hornblende and garnet. The Batholith is affected by at least three ductile to brittle-ductile deformational phases.

The Aguapeí Group is composed of conglomerate, sandstone and siltstone that are unconformably deposited on the underlying basement in a braided fluvial to marine depositional environment. The metasediments occur within a fold and thrust belt that is deformed under brittle-ductile conditions and are commonly in tectonic contact with the basement. Strong hydrothermal alteration and associated gold mineralization occurs in association with the lower the contact of the Aguapeí Group with underlying basement.

The Aguapeí Group sediments were deposited in a continental rift, possibly an aulacogen, and are divided stratigraphically from bottom to top, into the Fortuna, Vale da Promissão and Morro Cristalina formations. At the base, the Group consists of conglomerate and sandstone (rudaceo-psamitic sediments) with monomictic quartzite clast conglomerates in medium to coarse sandy matrix. Towards the top, quartz sandstones with crossbedding, plane-parallel shale packages, limestone, mudstones, with colours varying from brown to purple predominate. The uppermost part of the Group is fine to medium quartz sandstones. The lower Fortuna and Vale da Promissão Formations characterise a transitional environment with transgressive continental-marine cycles and a NNW to SSE paleo-stream direction whereas the Morro Cristalino Formation characterizes a regressive environment with SSE to NNW paleo-stream directions. The depositional environment varies from fluvial braided to marine.

7.3 REGIONAL STRUCTURAL GEOLOGY

According to Karpeta, 2006, (Exploration Implications Of The Structural Evolution Of The Santa Elina Belt, Mato Grosso, Brazil W. P. Karpeta Bastillion Limited)) the structural evolution of the Santa Elina Belt consists of three deformational events (D0–D2). The first event, D0, involved the E–W extension and rifting accompanying the filling of the basin and the fining upwards clastic sequences of the Fortuna Formation, the Vale da Promissão Formation and the Morro Cristalina Formation. The eastern margin of the Belt is the Morro Solteiro Fault (marked by the São Vicente–Pontes e Lacerda–Pau-a-Pique Lineament). This appears to have been the major normal fault bounding the rift basin.

NNW–SSE striking faults within the basin and parallel to the basin margin, such as the Longa Vida Fault, are interpreted as synthetic and antithetic normal faults affecting sedimentation within the basin. Cross-basin faults such as those terminating the belt to the north, forming the northern boundary of the Ernesto Project and the southern boundary of the Pau-a-Pique project possibly represent transfer faults, which transferred stress from one normal fault to another. All of these faults were reactivated or acted as buttresses during the D1 and D2 deformation events.

The second deformational event (D1) can be subdivided into an early D1a phase, a middle D1b phase and a late D1c phase of deformation and probably involved the E–W compression and consequential inversion of the basin. The D1a phase is characterised by thick-skinned, bedding-plane parallel thrusting (D1a) and is shown by the formation of bedding-plane parallel phyllonites (i.e. phyllitic mylonites) along the basal contact (basal decollement) and within the clastic sequence, especially along more ductile, less competent horizons. Hybrid-extension and extensional quartz vein arrays were also formed at this stage, both within and adjacent to these phyllonite horizons. In most cases, the early fabric associated with the phyllonites has been overprinted during the flexural slip movement of Phase D1b. The quartz veins also underwent boudinage during D1b. Locally thrust duplexes are preserved with the wrong sense of movement for flexural slip movement of D1b indicating this early phase of thrusting (e.g. São Vicente Pit and Ernesto Galleries).

The D1a phase was followed by thick-skinned folding (D1b), thrusting and inversion of normal faults as reverse faults. This would have occurred as the bedding plane parallel thrusts of D1a were locked (pinned) by the normal faults of D0 acting as buttresses. The folding style was flexural slip marked by inter-bed slip, folding and overprinting of the D1a foliation by a second foliation to form a local crenulation cleavage. As folding became tighter, accommodation structures such as thrusts, carinate hinges and 'saddles' on the fold hinges would be formed. However, these saddles were not filled with quartz, but with the much more ductile phyllonites, which flowed into the fold hinges. Horizontal compression would also have resulted in the formation of hybrid extensional vein arrays (e.g. São Vicente Pit, São Francisco Mine), especially near buttressing structures (the Longa Vida Fault and the Morro Solteiro Fault).

The relative ductility of the basement rocks appears to have played an important role in the folding style of the Aguapeí cover sediments. In the northern part of the basin (e.g. São Vicente), where the Aguapeí Group overlies the more ductile basement, the anticlines are tighter than the synclines in the Aguapeí Group sediments.

In the southern part of the basin (Pau-a-Pique and Ernesto), the synclines are much tighter than the anticlines because the Aguapeí Group sediments overlie the more competent Santa Helena granitoids. Once the interlimb angles of the folds exceeded 60°, the folds would tend to lock, and shortening would be taken up by steep contractional faults.

The D1c phase involved the east-over-west thrusting of the basement over the Aguapeí Group in the central part of the basin around Pontes e Lacerda. This resulted in the thrust of the basement over the northern part of Rio do Cágado Syncline, in the frontal ramp of the thrust. The northern and southern boundaries of this thrust sheet of basement are defined by D0 transfers, which now form hybrid to lateral ramps with apparent N–S directions of thrusting (e.g. the Ernesto-Lavrinha area).

7.4 DEPOSIT GEOLOGY

The deposit geology presented here is summarized from the Ausenco (2010) report.

7.4.1 Ernesto and Lavrinha Deposits

In the Ernesto-Lavrinha area, the Middle Proterozoic Aguapei sediments belong to the Fortuna Formation. The Formation consists of an 80 m thick basal felspathic to arkosic metarenite, a 20 m thick oligomictic metaconglomerate bed, and a 300 m thick series of fining upwards ortho-quartzitic metarenites and intercalated hematitic metapelites. This sequence occurs throughout the area, with the oligomictic conglomerate serving as an excellent stratigraphic marker. All of the rocks have been subjected to regional low-grade greenschist facies metamorphism but still display many well-preserved sedimentary structures such as graded bedding, cross-bedding and load deformation features.

The sediments in the area are underlain by fine-to-medium-grained, weakly foliated, hydrothermally altered tonalite. The tonalite is apparently a shallow intrusive that pre-dates the Aguapeí sediments, and is probably a part of the Lower Proterozoic volcano-sedimentary belt that forms the basement.

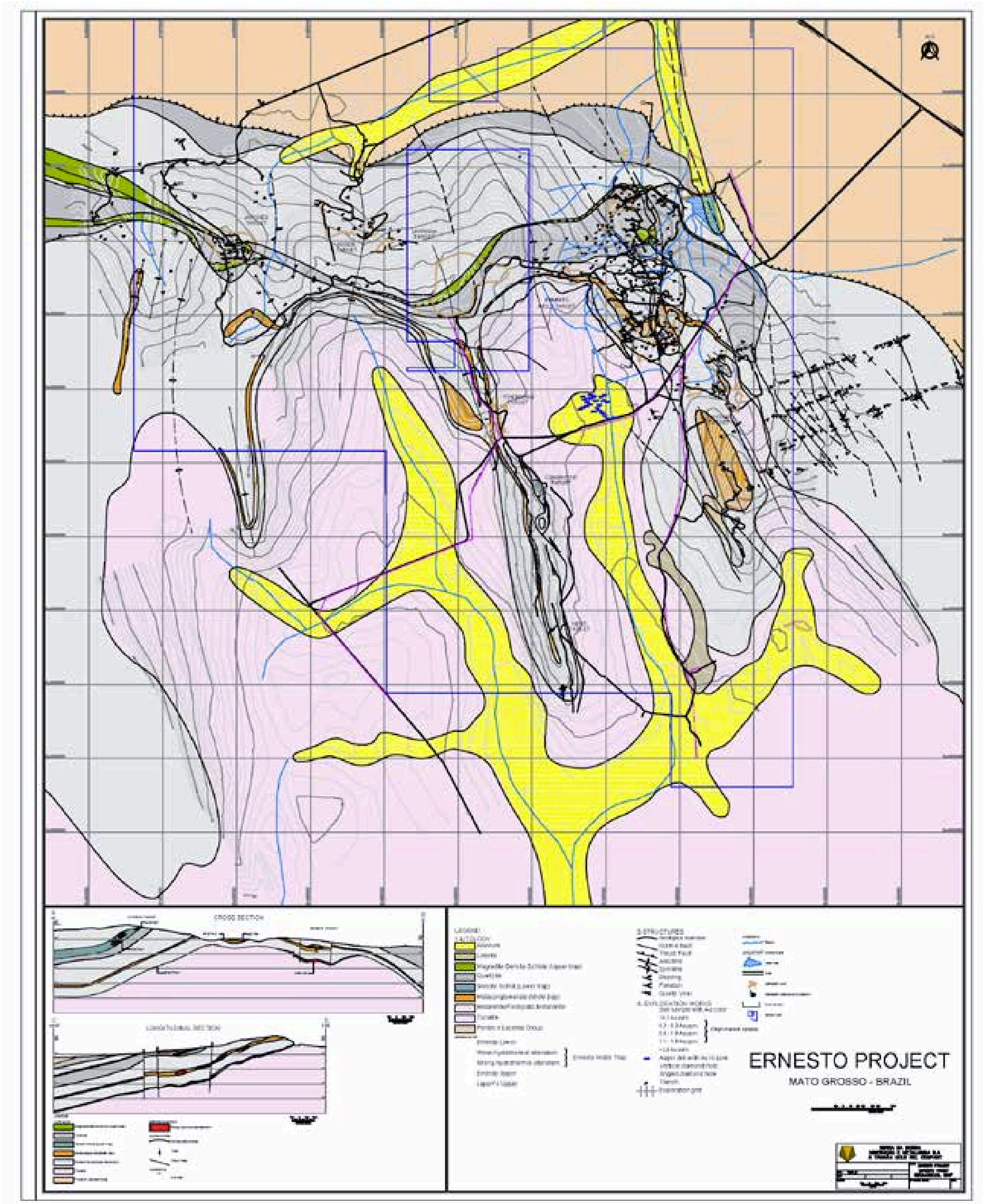
The dominant structural feature in the Aguapeí sediments in the Ernesto area is a large, internally overturned NNW-trending anticlinorium (Figure 7.3). It is comprised of a series of NNW trending asymmetrical anticlines and synclines with steep eastern flanks and gentle western flanks. In addition to these folds, there are also NW-trending high-angle faults, representing mostly brittle deformation with sinistral movement, and a final extensional rupture event trending N80ºE with vertical fractures that form small brecciated zones. Strong mineralization occurs mainly along three sub-parallel linear belts, each following one of the NNW-trending synclinal fold axes. The Ernesto Project is located along the easternmost of these belts.

The contact zone between the Aguapeí sediments and the underlying basement tonalite is observed only in drill cores and in underground artisan mine workings. The contact zone consists of a 5 m to 25 m thick magnetite-sericite schist unit, containing lenses and elongated bodies of quartz generally concordant with the foliation, and a 1 m to 3 m thick basal layer of intensely altered, crushed and decomposed rock. The magnetite-sericite schist apparently represents strongly altered and deformed sediment, probably a hydrothermally altered and sheared metapelite. The basal crushed layer possibly represents a hydrothermally altered tectonite, possibly a mylonite.

The Lavrinha Deposit is closely linked to the Ernesto Deposit as they are located less than 2 km from each other. It has been interpreted, based on mapping and drill hole logging and sampling, as gold-rich quartz veins and veinlets with coarse grained pseudomorph of coarse pyrite occurring along shallow-dipping structures. The main difference with Ernesto is the position of the mineralization along the metasedimentary sequence. The Lavrinha Deposit is developed in the Upper Trap unit. Gold mineralization is located along the quartz boudins in high sericitized rock (hydrothermal alteration) and plunges to the north. The thin mineralized lodes are from 1 m to 10 m wide, with an average thickness of 4 m and are more-or-less continuous for at least 100 m along its northern plunge direction.

Throughout the Pontes e Lacerda region, the Aguapeí contact zone forms an undulating surface dipping gently east to northeast and is thought to represent a low-angle structural detachment fault sandwiched between less-altered and less-deformed Aguapeí sediments above and an altered basement tonalite body below. This detachment contact zone has been folded together with the overlying sediments into the series of broad north-south trending folds. The detachment appears to have served as both a conduit and partial trap for mineralizing hydrothermal fluids channelled along the axes of the folds.

Figure 7.3 Geology of the Ernesto Deposit



*(**Source:** Ausenco 2010)*

7.4.2 Pau-a-Pique Deposit

The Pau-a-Pique Deposit occurs in close association with the contact of the meta-tonalite basement and the overlying Aguapeí Group metasediments. The region was affected by greenschist facies regional metamorphism, with a dynamic cataclastic metamorphism during two deformational events. The stratigraphy, structural geology, mineralization and hydrothermal alteration detailed below are summarized from Ausenco (2010).

The meta-tonalite stock that forms the basement to the overlying sediments and the Pau-a-Pique Deposit, is probably related to the Santa Helena Intrusive Suite (Figure 7.4). The tonalite is metamorphosed with a foliated structure, but preserving the original igneous texture. Different phases include coarse (>0.4 cm) and fine (0.1–0.2 cm) grain sizes that are in contact with microbreccia, protomylonite, and hydrothermal alteration zones with epidote, sericite and chlorite, that are concordant with the S1 foliation ("D1").

The metasediments, including sandstones, shales and conglomerates, are part of the Fortuna Formation of the Aguapeí Group. These sediments are immature, poorly sorted clastics, deposited in a high energy fluvial braided stream environment associated with a rift. The rocks are metamorphosed and deformed under lower green-schist facies conditions. The conglomerate pebbles up to 3 cm in diameter are generally stretched, and boudinaged, and exhibit sigmoidal deformation features. Some recrystallisation zones with quartz/sericites are formed in boudinage domains.

The contacts between the conglomerates and sandstone are gradational upwards, with very well-preserved cross-bedding. Quartz veins and strong sericitization occur generally on the primary bedding ("S0") structure. The S0 structure is parallel to the first penetrative ("S1") structure of the tonalites that was reoriented during the first deformational ("D1") event. The fine recrystallised metasandstones represent the main lithology in the 'Chapada da Serra'. They occur as a stockwork of quartz veins with comb structures and vuggy cavities, and are strongly fractured. After weathering, the silicification hinders the recognition of the remaining sedimentary structures.

Metapelites are encountered in the western portion of the 'Serra do Pau-a-Pique' parallel to the regional bedding ("S0") trends. The metapelites are characterised by fine-grained sediments that have been metamorphosed to low-grade greenschist facies. The metapelites are composed of greyish sericites (phyllites) and contain thick quartz veins that are intensively folded with disharmonic styles in shear zones. Associated pyrite occurs as euhedral to subhedral grains, up to 0.5 cm in diameter, and is associated with the margin of sheared veins.

Muscovite schist is developed in the contact between the metatonalites and metasediments and is an important host of mineralization. The muscovite schist has a lepidoblastic texture, and is composed of muscovite, quartz, chlorite, magnetite, pyrite, apatite and traces of hematite, ilmenite, chalcopyrite, chalcocite, galena and free gold associated with pyrite. The main sulphide is pyrite, which occurs in mega-crystals with diameters up to 3 cm. When the pyrite is syn-deformational, quartz/chlorite shadow zones are common at crystal margins. Quartz veins and quartz-albite occur associated with this lithology. In the southern portion of the property, the central and lower part of the deposit has a higher amount of quartz, while in the northern and upper part of the deposit, there is a higher concentration of quartz-albite-chlorite pegmatitic structures where the highest gold grades are located. The muscovite schist has S-C structures and abundant shear bounded sigmoidal veins. The schist has a strong stretching lineation oriented at N20–50W that controls the form of the deposit and sub-surface mineralization.

Biotite-schists occur in the sheared contact between the Aguapeí Group metasediments and metatonalites and within the metatonalites. The biotite schists are not mineralized. They are greenish-black to yellowish black with biotite (70%), chlorite (20%), plagioclase (5%) and quartz (5%), and accessories such as pyrite, magnetite and hematite. They are structurally conformable with the regional S1 and they are associated with high deformational zones. Possibly they are metamorphic equivalents of the metabasics within the tonalite stock. Locally, they appear with strong shear and folding, forming kink bands. The sheared quartz veins and quartz-albite are very common, alternating in accordance with the spatial disposition of the lithotype, related to the different distinct hydrothermal zones.

Quartz veins occur throughout the deposit, and represent four categories:

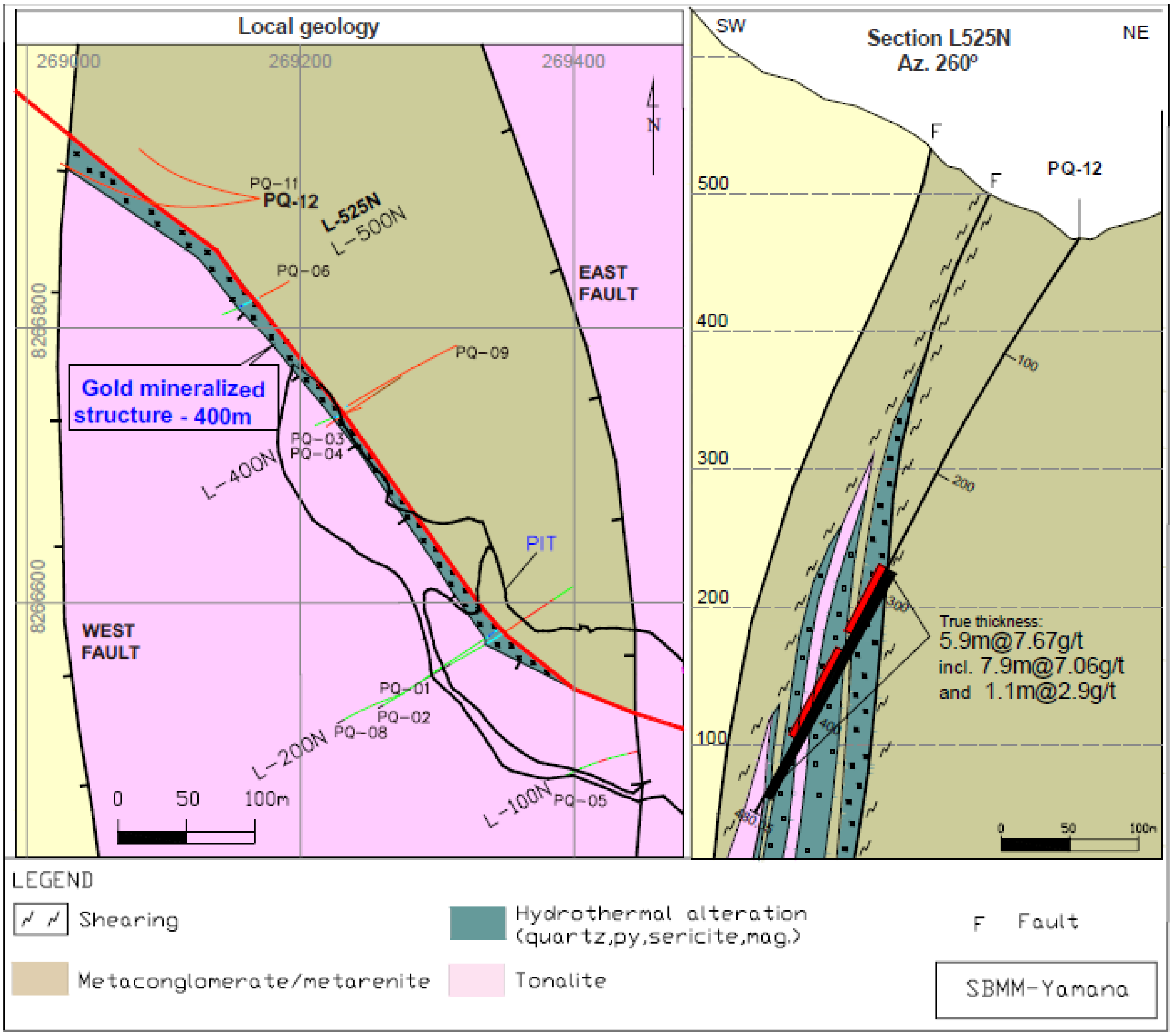
- Type 1 veins in shear zones are composed of translucent to lactescent quartz, with a lineation defined by hematite and sericite parallel to the down-dip lineation. These veins have significant dimensions on strike and width (400 m/20 m) but have marginal to low grade;
- Type 2 veins in shear zones are slightly discordant to the S1 fabric, are weakly mineralized and were the target of garimpos (artisanal mining) (L-1500N). These veins are composed of granular quartz, euhedral pyrite up to 1 cm, generally oxidised forming boxworks and reddish sericite; also presenting a significant strike and width (200 m/15 m), with an unknown depth;
- Type 3 veins are extensional and discordant. They are common in drill core samples and generally located in tectonic contacts with stockwork formations and breccias. They were deposited under a late brittle regime, composed of white lactescent quartz with comb structure. The veins are discontinuous and veined formation when hosted in the Aguapeí Group. On the basement and schist portion the most common veins are quartz-carbonate.
- Type 4 veins are without a defined structure (concordant, sub concordant and/or discordant), with pegmatoid texture, characterised by the presence of euhedral pyrite up to 5 cm long and associated with chlorite, reddish albite (Fe enrichment), translucent quartz, calcite, biotite and magnetite; and are mostly very strongly mineralized. They were probably the deepest conduits for the hydrothermal alteration, developing temperatures from almost 350ºC, with granitic affinity. The proximity to the hydrothermal pipe is considered to be an important control on mineralogy as the pressure, temperature and depth vary from the main zone conditions.

Brecciated phases are common in the contact zones between the Aguapeí Group metasediments and the metatonalites, and within the metatonalites, as well as in contact with depositional cycles between the sandstone facies and pebbled clastic rocks of the Aguapeí Group. In these cases, at the base of the deposits there is a strong quartz stockwork vein system with comb structures. The breccias developed in the ‘Aguapeí/metatonalites’ contact show oligomitic clasts of metasediments (up to 10 cm) supported by commonly reddish sericitic matrix. The clasts are generally roundish to sub-angled, indicating a strong tectonic transport. Discordant quartz veinlets with comb structures are directly associated with this lithotype in stockwork form. Breccias were formed in granitic facies contacts, showing basal facies oligomitic clasts (up to 1 cm), supported by a chloritic matrix and densely rimmed by sericitic bundle, disseminated epidote and post-quartz carbonate veinlets. These breccias normally do not host sulfides and locally, in the South Target region, show weak gold mineralization (Max: 2.05 m at 1.54 g/t Au – PQ-30).

The final deformational event (“D2”) involved strike-slip faulting within and along the margins of the belt along previous faults such as the basin bounding normal and transfer faults.

Depending on the orientation of the original fault relative to the deformation, the movement may be either dextral or sinistral. Major faults such as the Morro Solteiro Fault and the Longa Vida Fault appear to have been reactivated as strike-slip faults during the D2 event.

Figure 7.4 Geological Map and Section of the Pau-a-Pique Deposit



*(**Source:** Ausenco, 2010)*

7.5 MINERALIZATION

Descriptions of mineralization at the Ernesto and Pau-a-Pique Deposits are summarized from the Ausenco (2010) report.

7.5.1 Ernesto-Lavrinha Deposits

The Ernesto-Lavrinha Deposits consists of gold-rich quartz veins and veinlets occurring along a relatively thick, shallow-dipping structure at the base of the metasedimentary sequence and within altered sulfidic horizons in overlying meta-arenite units. The basal structure is interpreted to be a low-angle detachment fault that has been folded and faulted together with the overlying stratigraphy. Gold mineralization is located along asymmetrical anticlines and synclines that plunge gently to the north and are cut by NW and NE-trending narrow faults. The gold mineralization occurs in three zones: Lower Trap, Middle Trap and Upper Trap.

The Intermediate and Upper traps are in either permeable conglomeratic horizons or dilatent zones in the meta-arenite stratigraphy. Mineralization in the Upper and Intermediate traps is much less continuous than in the Lower Trap. The Intermediate Trap, which is about 80 m above the Lower Trap, is restricted to a conglomeratic horizon, where it intersects dilation structures developed by folding and faulting.

Mineralization in the Lower Trap is from 130 m to 210 m wide, with an average thickness of 5 m and is more-or-less continuous for at least 1,000 m along its northern plunge direction. The change in plunge angle appears to affect both the thickness and grade of the gold mineralization, with the thickest and highest-grade mineralization occurring where the plunge is less steep.

These zones have been defined by mapping and drill hole logging and sampling. The Lower Trap is within the detachment fault and includes the Ernesto resource referred to in this Report. The Lower Trap mineralized zone in Ernesto is widely developed within a mylonitic zone. The mylontic zone is a deformed version of meta-arenite which was altered and intruded by quartz veining. The mylonitic zone often resembles that of a healed fault zone that developed along detachment structures. The presence of extensional faulting in time of mineralization caused alteration footwall of tonalitic unit. The tonalite is extensively altered and represent of a weak rock which historically logged and called saprolite. However the mineralogical composition of this altered footwall unit is completely different from saprolite on surface. The footwall saprolites are mainly composed from clay minerals produced by alteration of feldspar and mica from tonalite groundmass. The dominant clay mineral is kaolinite.

The footwall saprolite is poor rock in terms of geotechnical characteristics. The alteration is gradational and kaolointic saprolite gradually changes to weakly altered tonalite. The altered tonalite has usually whitish color but groundmass of tonalite is recognizable in cores. However feldspar and micas are replaced by clay. The footwall saprolite is usually entirely replaced by clay minerals (kaolinite). Although in most cases fresh tonalite is located below footwall saprolite, in some local areas in Ernesto gradational changes from tonalite to saprolite both in upper contact with mylonite and lower contact with fresh tonalite are observed in core. This is another indicator of later movements along these detachment faults that accommodate weathering of protolith along weak contact with the mylonitic zone.

The footwall saprolite is weakly mineralized but there is no mineralization within fresh tonalite at Ernesto. This is an indicator that faulting and mineralization events are concordant.

Alteration associated with gold mineralization within the mylonitic unit includes abundant quartz veins and veinlets with coarse-grained euhedral pyrite and medium grained bipyramidal crystalline magnetite. In addition, there is saussuritization forming fine-grained sericite, chlorite, and carbonate. This alteration and mineralization occurs in mylonitic zones near the base of the detachment fault.

The Upper Trap, which is widely developed in the Lavrinha Deposit, occurs in metapelitic rocks (hematite sericite schist) in dilation zones of the intensely deformed synclinal troughs. The Upper and Intermediate traps share similar alteration and mineralization suites. The Upper Trap seems to be eroded in the Ernesto Deposit area.

Coarse gold is probably a factor in the Ernesto-Lavrinha Deposits, as indicated by the gravimetric recoveries used by the garimpeiros. Screen fire assays for gold in the seven samples collected from mineralized faces in the artisanal underground workings developed in the Lower Trap returned one sample with probable coarse gold.

7.5.2 Pau-a-Pique Deposit

The Pau-a-Pique gold mineralization is associated with the intensity of the hydrothermal alteration, and is proportional to the occurrence of pyrite, sulphide alteration, quartz veins and sericitization. Pyrite generally occurs in fresh rock, and at the exposed zones in the old pit there is a predominance of oxidized pyrites, products of surface weathering.

In the deeper levels (PQ-12), the most common hydrothermal alteration with gold enrichment is strong albite-anorthosic quartz veining associated with chloritization and pyrite. While in shallow levels (PQ-01, 02), the most pervasive alteration is silicification, represented by a strong injection of quartz veins and weaker gold enrichment. The albitic alteration probably represents deeper and hot sources of the hydrothermal feeder.

The envelope of the mineralized zone is approximately 550 m long, maximum of 15 m wide and 400 m deep in the largest extension (NW down plunge). The occurrence is confined to the contact between Aguapeí metasediments and the tonalite, which is the main prospective guide to the deposit, in addition to the NW plunge that can be seen in the longitudinal section.

In the NW down-plunge zone, gold enrichment associated with albitic-anorthosic metasomatism, pyrite, chlorite and magnetite is abundant. Alteration is wide and extensive and developed in contact with tonalite and conglomerate and within the sheared and deformed tonalite. The presence of sulphide and magnetite in sheared host rocks is favorable for gold enrichment with higher grade. Strong metasomatism which is developed mainly in contact and within hangingwall tonalite shows higher temperature mineral assemblage which represent a hyrothermal feeder for ascending metasomatic fluids.

7.6 MINERALOGY OF ORE-BEARING ROCK IN ERNESTO, LAVRINHA AND PAU-A-PIQUE

A total of five samples were submitted for X-Ray Diffraction (“XRD”) analysis to the University of Sao Paulo (Escola Politechnica de Universidade de Sao Paulo), Brazil.

Two samples were selected from each of the Ernesto and Lavrinha mineralization, and one sample from the NW zone of the Pau-a-Pique mineralization. These samples were sub-samples of the set of samples that were submitted for metallurgical tests in 2016.

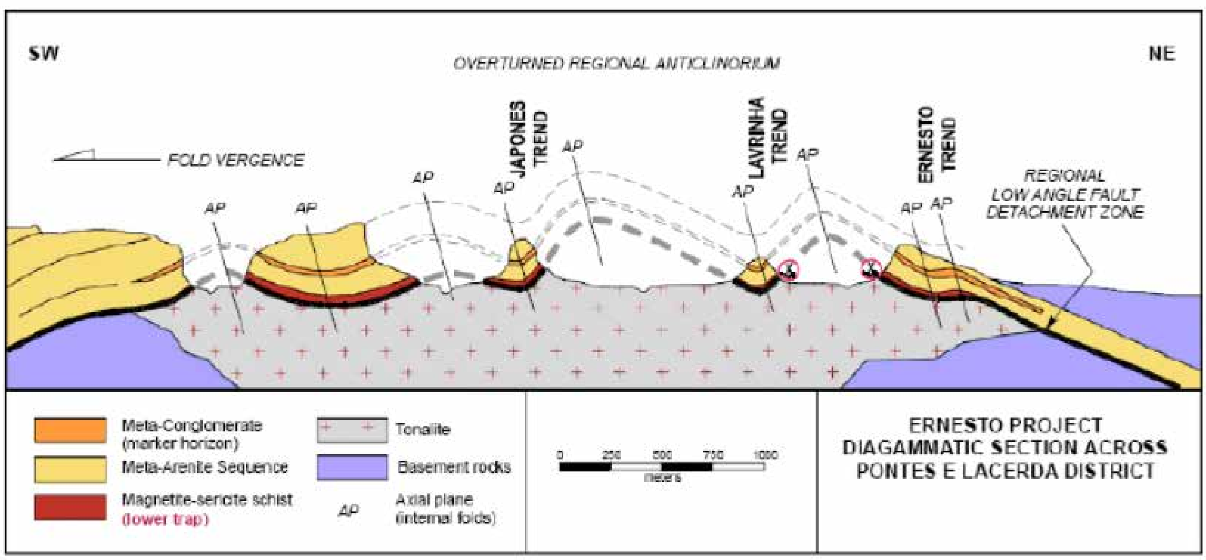
The XRD analysis indicates that the Ernesto and Lavrinha mineralized rocks are composed of 50 to 70% quartz, 25 to 40% muscovite, 5 to 7% hematite, 1 to 4% kaolinite, with minor goethite and microcline. One sample of Pau-a-Pique mineralization contained 35% quartz, 35% muscovite, 9% albite, 6% clinoclore, 5% hematite, 5% microcline, 5% carbonate, and a trace of kaolinite. Ernesto and Lavrinha show typical mica-schist mineral composition that has been metamorphosed under lower green schist facies. Most of the feldspar has been altered to sericite. Sulphide minerals such as pyrite are not identified by the XRD method, however, the presence of iron oxides in Ernesto and Lavrinha suggests that the majority of pyrite is altered and converted to iron oxides. The lithological logging of mineralized intervals supports this since in most cases pyrite is replaced by iron oxide.

8.0 DEPOSIT TYPES

Ernesto-Lavrinha is described as a detachment-style gold deposit (Figure 8.1) that typically has the following characteristics:

- Gold mineralization is associated with low-angle to flat detachment faults, generally with a normal (extensional) sense of movement which consistently places younger units over older units;
- Mineralization is commonly characterised by quartz-rich vein and veinlet zones (in the ±25% range) with magnetite or hematite, coarse euhedral pyrite (in the ±1% range), sericite, some clay mineral, some late stage calcite and gold. The gold is commonly associated with only very small amounts of silver;
- Mineralization is typically located along a 3 m to 8 m thick rubble zone or mylonite of a detachment (or thrust) fault that intersects high angle structures, either faults or folds. The detachment is commonly within a deformed zone 10 m to 30 m thick;
- The continuity of the mineralization within the detachment zone is normally quite good, extending over 100 m;
- Detachment–style gold mineralization is in altered rock parallel to anticline axes and faults;
- Multiple styles of mineralization are common with local stacked mineralized zones;
- Fluid inclusion studies indicate temperatures of formation about 200°C to 250°C.

Figure 8.1 Diagrammatic Section Across Pontes e Lacerda Gold District Deposits



The Pau-a-Pique and Ernesto UG (Lower Trap) Deposits are similar in that they both occur at the contact between the Aguapeí group and the basement meta-tonalite. At both locations, the contact is associated with shear zones and hydrothermal alteration assemblages with pyrite, sericite and hematite.

The Ernesto open pit Deposit (Middle Trap) and São Francisco Deposits are similar in that they both occur within Aguapeí Group psammitic rocks affected by hydrothermal alteration resulting in assemblages rich in silica, sericite and hematite.

The Ausenco (2010) report draws parallels between the shallow dipping Ernesto Deposit and detachment-style gold deposits in the south eastern California and to bedding plane parallel shears in Tarkwa sediments in Ghana.

Reid et al. (2012) consider Aura's São Francisco Mine, located north of Ernesto, is a shear-hosted lode gold deposit. São Francisco is located approximately 60 km northwest of Ernesto and displays similar host Aguapeí group lithologies and structural controls at the deformed basement/Aguapeí group contact. São Francisco is considered by Reid et al. (2012) as epigenetic, structurally controlled, and composed of narrow, 1 cm to 5 cm wide, and quartz veins containing free gold. The veins and vein systems and stockworks both parallel and crosscut the bedding planes and appear to represent separate but closely related mineralizing events.

The São Francisco, Ernesto-Lavrinha and Pau-a-Pique Deposits are broadly similar in host lithologies, structural style, alteration, and mineralization and all share characteristics of shear-hosted lode gold deposits.

At Lavrinha, mineralization occurs within a mata-pelitic sub-member of the Aguapei Group. Mineralization is often associated with narrow quartz vein and veinlets in phyllonitic matrix with strong sericitization and chlortization. The thickness and size of quartz veins are smaller than Ernesto and rarely exceed 1m in true thickness. Pseudomorphs of pyrite and strong sericitization with presence of quartz are good indicators for mineralization intervals.

Strong foliation and kink-band structures disrupted mineralized shoots both along strike and down dip of the deposit.

The style of mineralization and deposit type is very similar to Ernesto and detachment style faults are marked with pervasive alteration along the contacts and within a sericite schist package. Mineralization and alteration both developed in contact meta-arenite with sericite schist and also within the sericite schist.

9.0 EXPLORATION

9.1 ERNESTO

Yamana's exploration on the Ernesto Property began in 2003 and consisted of surveying, rock chip sampling, chip channel sampling, soil sampling and mapping. Geological maps are presented in Section 7.0 of this Report. Drilling is summarized in Section 10 of this Report.

9.1.1 2003 – 2009 Exploration Activities

The exploration activities carried out during 2003 to 2009 included rock chip sampling, chip channel sampling, soil sampling, detailed geological mapping, drilling and compilation work. Drill results are summarized in Section 10 of this Report. 2010 and 2011 exploration work focused on drilling the Lavrinha Deposit.

9.1.2 2012 – 2013 Exploration Activities

The exploration activities carried during 2012 and 2013 included rock chip sampling, detailed geological mapping, drilling and compilation work. Drill results are summarized in Section 10 of this Report.

The historical drilling data analysis, combined with field checks and detailed geological mapping revealed the higher grade gold mineralized zones to be controlled by hinges and fold axes plunging at low angles to the northwest or southeast (azimuth N310o or N130o) which formed during compressive events where rocks of the Pontes & Lacerda Sequence were thrust over the Aguapeí Group rocks. Structural vergence indicated a tectonic transport from NE to SW.

9.2 LAVRINHA

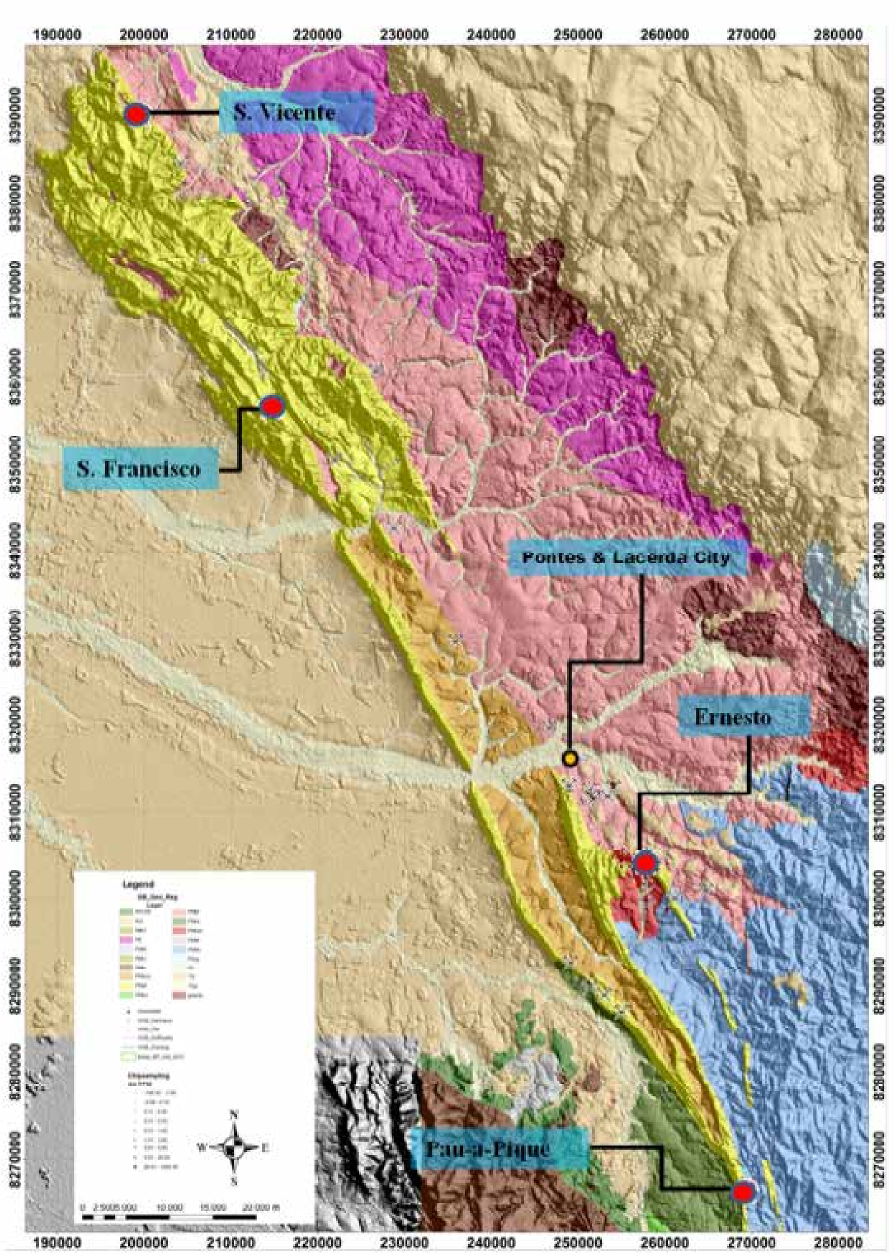
The exploration programs that were performed at the Lavrinha Deposit were closely related to ones carried out for the Ernesto Deposit, as well as the work that had been done in the Aguapeí Belt. The exploration activities are summarized below:

- Exploratory work by Yamana from 2001 to 2013 and performed in three phases;
- Phase 1: Between 2003-2009;
- Phase 2: 2010 and 2011; and
- Phase 3: Between 2012-2013.
- Infill drilling conducted by Aura.

9.2.1 Exploration By Yamana

In 2003 Yamana acquired the São Vicente and São Francisco mines from MSE and also all exploration permits related to deposits in the Ernesto camp, including Lavrinha and Pau-a-Pique (Figure 9.1).

Figure 9.1 Location Map of Main Guaporé Deposits Overlay on Regional Geology Layers



9.2.2 2003-2009 Exploration Activities

Several mineral exploration activities were carried out during the 2003-2009 period, including rock chip sampling, channel sampling, soil sampling, detailed geological mapping and additional diamond drilling. These activities followed up on historic exploration data that was generated by other mining companies in the past. During this period, drill programs were carried out only on Ernesto's near-mine areas (about 3,000 m) and at the Pau-a-Pique Deposit (24,000 m approximately) to extend and convert near-surface resources that were excavated by garimpeiros. The main goal, however, was to increase resources at the Sao Francisco Mine.

Due to budget constraints, almost all drill programs at the Ernesto near-mine areas consisted of shallow drill holes, rarely reaching 300 m depth or deeper. In the past, core sample intervals were visually selected, i.e. only strongly altered core intervals were sampled. It is estimated that at least 50% of the drill holes that were drilled in the Ernesto District before 2012 have not been sampled. In recent years (2012-2013) some intervals of historic drill holes that were not completely sampled and not visually interesting returned positive results (e.g. ER_022, ER_043, ER_124). At the time, the geologic team had not developed a good understanding of what controlled the higher grade gold mineralization (shoots). At the end of 2008 the initial Proven and Probable gold resource (open pit and underground) was estimated at about 700k oz by Yamana, from which 373k oz was related to the Pau-a Pique Deposit. Exploration work during that period was focused on adding resources to the Sao Francisco Mine. After the mine was sold to Aura, the goal was to add resources to the EPP project with less emphasis on testing additional potential areas of the district.

9.2.3 2010-2011 Exploration Activities

The exploration activities during this period were carried out only for the Lavrinha Deposit which is an important near-mine target in the Ernesto District, and contains several small artisanal mining shafts scattered along the south-central portion of the exploration permit. Geomin Ltda was the owner of the property and had attempted to add new gold resources to the Ernesto Project; the Lavrinha area was subsequently acquired by Yamana in 2010. Yamana drilled about 5,200 m in 28 drill holes around the main artisanal mining shafts and consequently added about 80k oz of additional gold resource.

9.2.4 2012-2013 Exploration Activities

During 2012-2013 all exploration efforts were focused on the Ernesto District including in-fill drilling of the Lavrinha Deposit (drill holes in the database with LV and LVR prefixes). The main goals were to define higher grade resource in the Ernesto near-mine target area, mainly looking for Lavrinha open pit possibilities to deliver mill feed (easily accessible due to close proximity) to the Ernesto Mine and also to provide geological support to mine operations. The main exploration activities included diamond drilling, rock chip sampling and detailed geological mapping.

9.2.5 Exploration Developed by Aura

In 2015 Aura carried out detailed geological mapping of the Lavrinha Deposit focused on outlining geological domains for fine meta-arenite, schist and arenites, and also mineralized domains with high hydrothermal alteration, quartz boudines associated with pseudo morphs of sulphides to better define the mineralized envelopes. During the mapping, lack of drill information near the surface extension of the mineralized shoots was identified. Aura decided to carry out 21 diamond drill holes for a total of 997.4 m of drilling. 845 samples were taken to better define geometry of mineralized bodies close to the surface and to provide better estimated grades.

Exploration on the Lavrinha Property focused on drilling which is summarized in Section 10 of this Report.

9.3 PAU-A-PIQUE

In 2005 and 2006, geological mapping of the Pau-a-Pique area was performed at a 1:1000 scale, the garimpo area was mapped at a scale of 1:500, and a scale of 1:25 was used to map channel samples and face benches where mineralized zones were exposed. 119 channel samples were collected at 2 m intervals in mineralized areas.

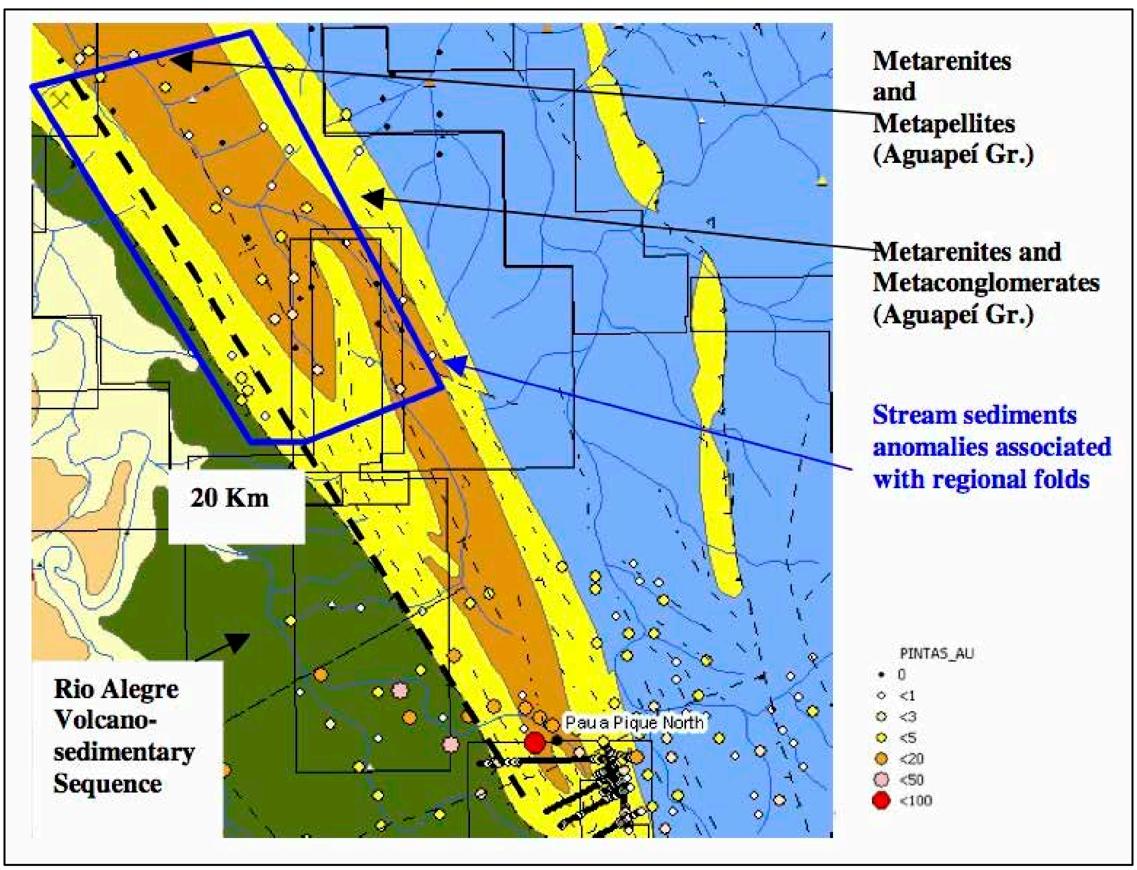
Chip sampling was conducted to identify lithologies with hydrothermal alteration. 10 kg to 15 kg samples were collected for gold analysis. A total of 600 chip, soil and trench samples were taken in 2008.

Soil sampling was conducted in the Pau-a-Pique area. The first 10 cm of each drillhole was discarded and the next 25 cm were collected and passed through a 0.6 cm sieve and put over a canvas screen. 10-15 litres of material were homogenized and collected in a plastic bag for analysis. On slopes, five holes were drilled for each sample collected, but only a single hole was drilled in flat areas.

Coarse material in drainage beds were sampled, sieved, homogenized and collected in 10 to 15 litre samples. Stream anomalies associated with regional folds were sampled, along with coarse grained sediments (Figure 9.2)

Total rock analysis was conducted on 2 chip samples and 19 core samples from mineralized areas. Samples were submitted for 32 element ICP analysis and the gold mineralization at Pau-a-Pique was found to be associated with high iron and zirconium content, a strong cobalt enrichment and positive molybdenum, copper and barium anomalies.

Figure 9.2 Stream Sediment Sample Locations – 2008



10.0 DRILLING

10.1 ERNESTO

In 2005, 11,128 m of drilling was conducted on the Ernesto resource area by Yamana. 22 holes tested peripheral target areas. In 2006, a further 7,777 m of diamond drilling was done on the Property, focusing on targets near the resource area and included a few exploration holes. 24 holes, totalling 4,295 m, were advanced in the Ernesto, Cantina, Japonês, Pombinhas and Serra Azul targets in 2007. Yamana drilled 29 holes totalling 2,820 m in 2009. The aim of the drill programs were to define areas with potential mineralization to add new gold resources or for extensions (down plunge or down dip) of shallower gold mineralization zones at targets near the Ernesto Deposit including Ernesto North, Ernesto SE, Pombinhas, Lavrinha, Open Pit1 Extension W, the Lavrinha-Nosde trend and the Japonês targets. Significant intersections from the Yamana drilling are summarized on Table 10.1. The results of the 2013 drill programs added to the Ernesto resources as well as identified other potential nearby mineralized zones (Figure 10.1).

The drill programs also helped define the understanding of some geological features of the Ernesto district. It was discovered that 60%-70% of the gold in the district was in the form of free gold, which helped determine the sampling procedures that could help improve the final analytical results.

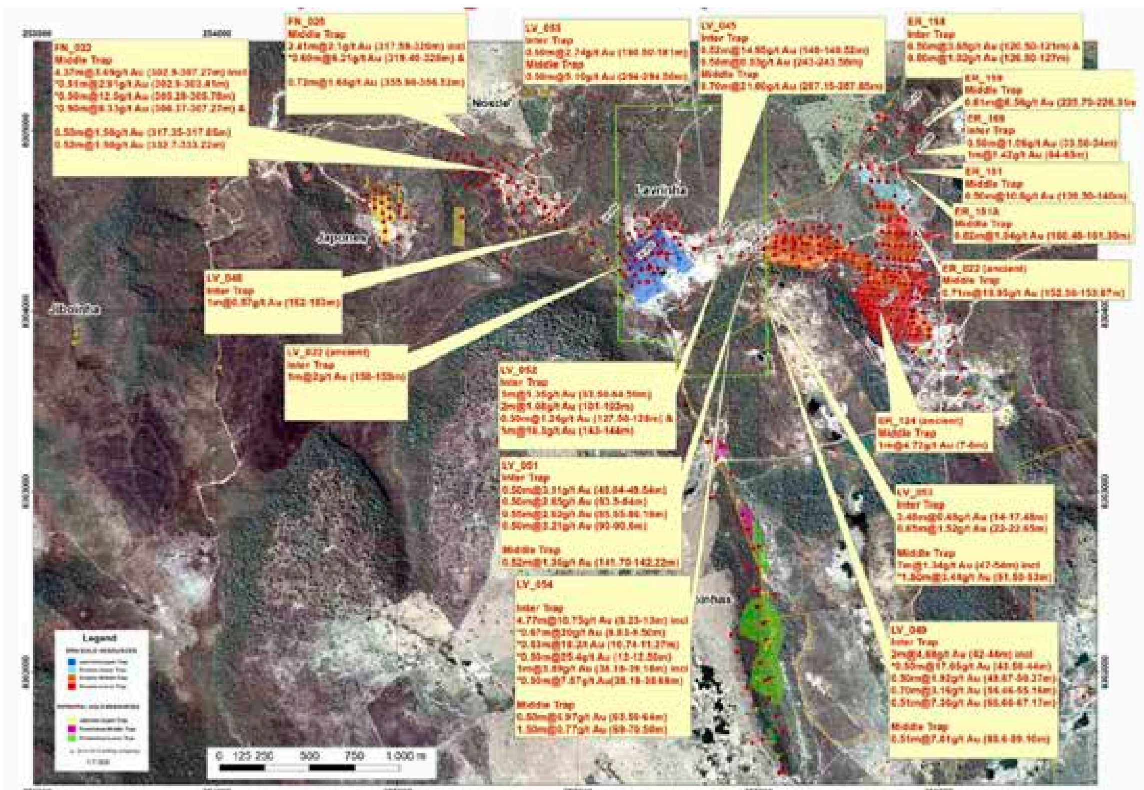
Sampling during the initial exploration drilling phases in the Ernesto district was collected systematically in 2 m intervals in the mineralized areas, ignoring geological features like thickness and grade. This resulted in dilution and the insertion of thick packages of waste zones inserted into the mineralized zones due to inconsistent geological models. More recent drilling has shown good continuity of the mineralization and that it is associated with a low angle thrust fault.

TABLE 10.1
ERNESTO SIGNIFICANT DRILLHOLE INTERSECTIONS – YAMANA DRILLING

Drillhole	From (m)	To (m)	Length	Au (g/t)	Zone
ER_022	152.36	153.07	0.71	18.95	Middle Trap
ER_043	185.00	189.00	4.00	1.57	Tonalite
Including	185.00	186.00	1.00	2.59	
ER_088	126.00	127.00	1.00	1.68	Inter Trap
ER_124	7.00	8.00	1.00	4.72	Middle Trap
ER_150	88.82	89.58	0.76	4.54	Lower Trap
ER_151	3.00	4.00	1.00	1.18	Inter Trap
	90.00	91.00	1.00	0.93	
	139.50	140.00	0.50	10.60	Middle Trap
	231.22	232.20	0.98	30.10	Lower Trap
ER_151A	160.48	161.30	0.82	1.04	Middle Trap
	214.00	215.00	1.00	0.52	
	242.89	244.94	2.05	0.98	Lower Trap
ER_153	49.85	51.17	1.32	4.28	Lower Trap
Including	49.85	50.52	0.67	6.27	
ER_158	120.50	121.00	0.50	3.65	Inter Trap
	126.50	127.00	0.50	1.02	

TABLE 10.1 **ERNESTO SIGNIFICANT DRILLHOLE INTERSECTIONS – YAMANA DRILLING**					
Drillhole	**From (m)**	**To (m)**	**Length**	**Au (g/t)**	**Zone**
ER_159	225.70	226.31	0.61	6.56	Middle Trap
ER_160	0.00	0.86	0.86	1.02	Soil
	284.21	284.76	0.55	56.00	Lower Trap
ER_169	33.50	34.00	0.50	1.06	Inter Trap
	64.00	65.00	1.00	1.42	
	259.00	260.00	1.00	0.79	Lower Trap

Figure 10.1 Ernesto Drillhole Locations - 2013



*(**Source:** Yamana, 2013)*

In 2015, 3,076.2 m of drilling within 21 holes was conducted on the Ernesto resource area by Aura focusing only on the Lower Trap where resources were deemed to be suitable for a potential underground mining operation. From these 21 holes, 15 holes were in-fill drilling to delineate existing resources and 6 other holes were geotechnical holes to assess the geotechnical characteristics of host rocks for a possible underground operation. The in-fill drilling focused on the centre of the Lower Trap Deposit where the majority of previous drilling was concentrated and needed limited drilling to upgrade Inferred mineral resources to the Indicated category and to provide increased confidence in the resource classification.

Drilling was carried out from surface utilizing the wire line method, using NQ diameter. The drill holes were surveyed with a Maxibor II, reading twice every 3 m. A 5% tolerance value was used to compare the inclination in the two runs, and was then validated in the survey report.

All samples from this drill campaign were analysed at SGS GEOSOL laboratory in Belo Horizonte, Brazil, using the fire assay method by AA finish.

Significant intersections for the 2015 drilling in Ernesto are presented in Table 10.2.

TABLE 10.2
SIGNIFICANT DRILLHOLE INTERSECTIONS (UNCUT) - ERNESTO INFILL DRILLING (2015)

Hole-ID	From (m)	To (m)	Apparent Thickness (m)	True Thickness (m)	Weighted Average Au (g/t)
P-01	137	145	8.00	6.00	15.2
P-04	124	136	12.00	10.00	12.2
P-06	124	125	1.00	0.80	3.96
P-08	100.36	104	3.64	2.48	3.13
P-10	119	122	3.00	2.25	1.63
P-11	68	75	7.00	5.25	4.31
KP15-01	137	143	6.00	5.46	21.3
KP15-06	84	97	13.00	9.73	3.11

10.2 LAVRINHA

Yamana conducted exploration drilling on the Lavrinha Property in 2010 and 2011. 28 drill holes, totalling approximately 5,200 m of drilling were advanced surrounding the artisanal mining shafts in order to add gold resources. Exploration drilling in the area surrounding the resources area including targets other than Lavrinha but the results revealed near surface mineralization along an area 750 m long and 150 m wide. In total, 10 mineralized gold horizons (NS1 to NS10) related to the Bonus Trap area were defined along the Nosde-Lavrinha NW trend.

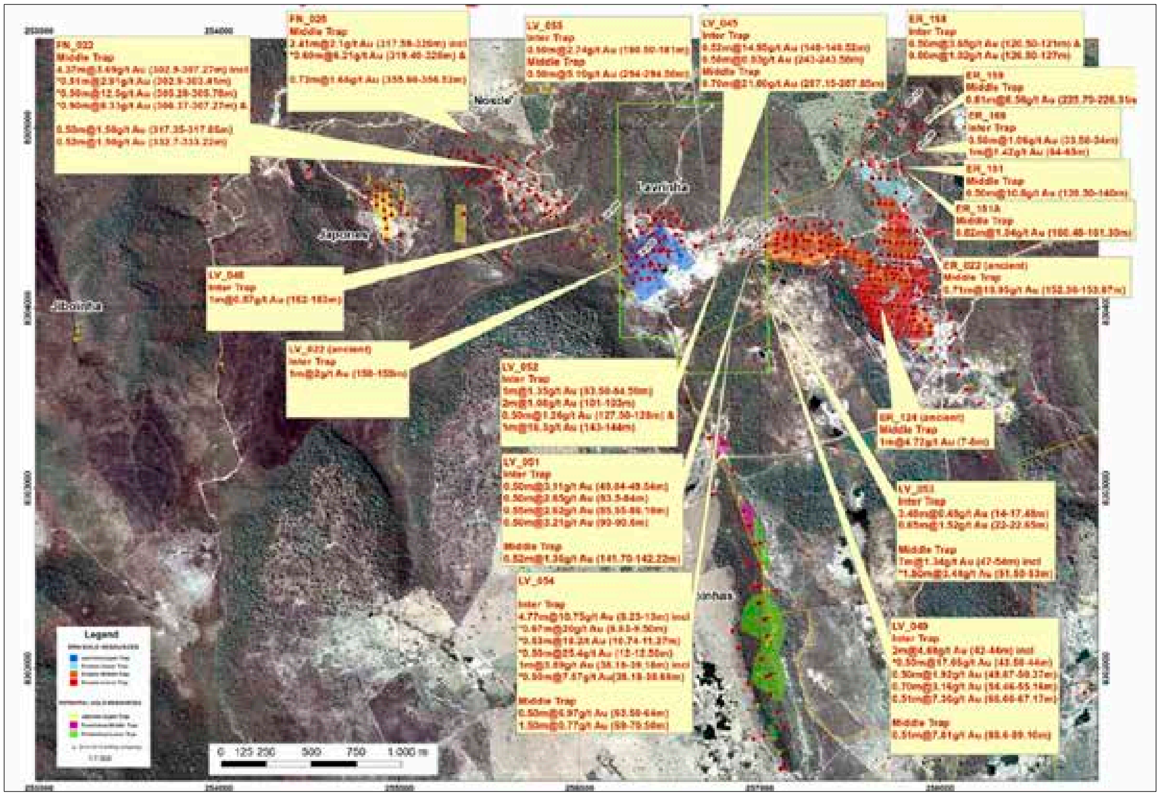
In 2013, 55 drillholes totaled 10,013.13 m of diamond drilling, with 9,446 samples analyzed for gold using fire assay at ALS Chemex Laboratories, and 318 density determinations were made. There are no survey measurements along the drillholes from LV001 to LV012, which were assumed to be of constant dip and azimuth, and from LV013 to LV055 there were measurements using the Maxibor system. Significant intersections related to drilling up to the end of 2013 on Lavrinha are listed in Table 10.3 and drillhole locations are presented in Figure 10.2. Drillholes that had samples analyzed for bulk density are noted in Figure 10.3.

TABLE 10.3
LAVRINHA SIGNIFICANT DRILLHOLE INTERSECTIONS (UP TO 2013)

<table>
<tr><th>Drillhole</th><th>From (m)</th><th>To (m)</th><th>Length</th><th>Au (g/t)</th><th>Zone</th></tr>
<tr><td>LV_022</td><td>158.00</td><td>209.00</td><td>1.00</td><td>2.00</td><td>Inter Trap</td></tr>
<tr><td rowspan="2">LV_042</td><td>208.18</td><td>209.00</td><td>0.82</td><td>1.35</td><td rowspan="2">Middle Trap</td></tr>
<tr><td>226.00</td><td>227.00</td><td>1.00</td><td>1.00</td></tr>
<tr><td>LV_043</td><td>108.00</td><td>109.30</td><td>1.30</td><td>5.20</td><td rowspan="5">Upper Trap</td></tr>
<tr><td>Including</td><td>108.00</td><td>108.57</td><td>0.57</td><td>11.20</td></tr>
<tr><td rowspan="3"></td><td>139.00</td><td>140.00</td><td>1.30</td><td>5.20</td></tr>
<tr><td>149.00</td><td>150.00</td><td>1.00</td><td>7.38</td></tr>
<tr><td>153.00</td><td>154.00</td><td>1.00</td><td>0.64</td></tr>
<tr><td>LV_044</td><td>102.39</td><td>106.35</td><td>3.96</td><td>2.69</td><td rowspan="3">Upper Trap</td></tr>
<tr><td>Including</td><td>103.26</td><td>104.54</td><td>1.28</td><td>7.00</td></tr>
<tr><td></td><td>117.78</td><td>118.95</td><td>1.17</td><td>7.64</td></tr>
</table>

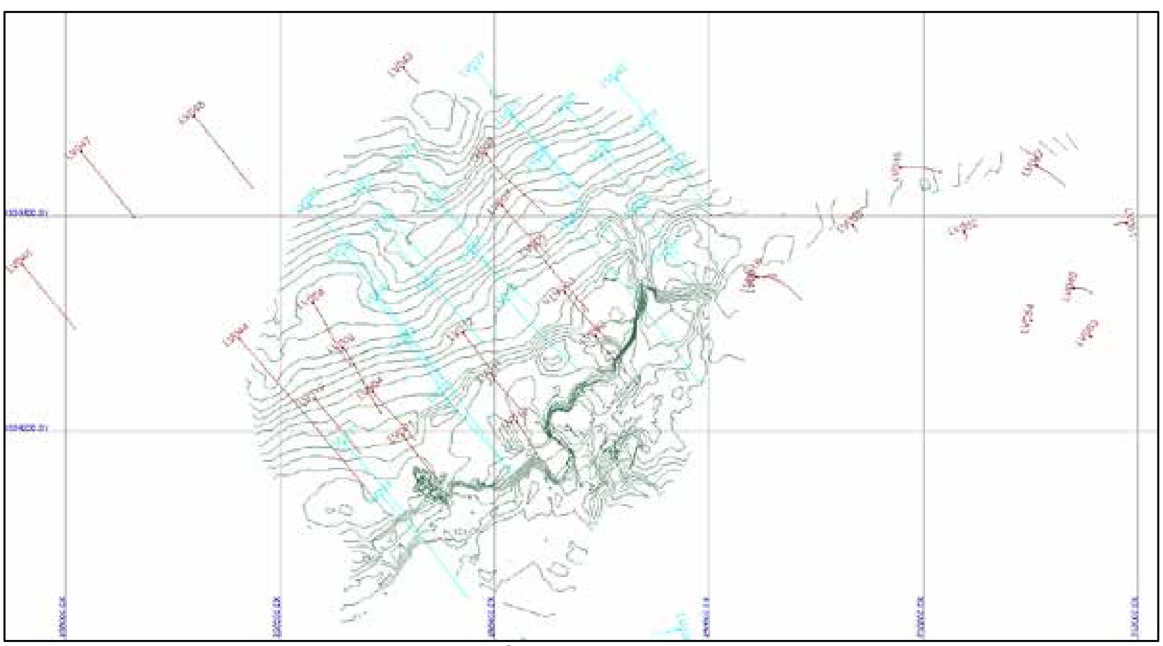
TABLE 10.3 LAVRINHA SIGNIFICANT DRILLHOLE INTERSECTIONS (UP TO 2013)					
Drillhole	**From (m)**	**To (m)**	**Length**	**Au (g/t)**	**Zone**
Including	117.78	118.36	0.58	11.90	
LV_045	121.30	122.43	1.13	0.83	Upper Trap
	140.00	140.52	0.52	14.95	Inter Trap
	243.00	243.58	0.58	0.83	
	287.15	287.85	0.70	21.60	Middle Trap
LV_047	120.29	121.00	0.71	1.15	Bonus Trap
LV_048	182.00	183.00	1.00	0.57	Inter Trap
LV_049	42.00	44.00	2.00	4.68	Inter Trap
	43.50	44.00	0.50	17.05	
	54.46	55.16	0.70	3.16	
	66.66	67.17	0.51	7.36	
	88.60	89.10	0.51	7.81	
LV_051	49.04	49.54	0.50	3.11	Inter Trap
	83.50	84.00	0.50	2.65	
	90.00	90.50	0.50	3.21	
	141.70	142.22	0.52	1.36	Middle Trap
LV_052	83.50	84.50	1.00	1.35	Inter Trap
	101.00	103.00	2.00	1.08	
	127.50	128.00	0.50	1.26	
	143.00	144.00	1.00	16.30	
LV_053	16.72	17.48	0.76	0.91	Inter Trap
	22.00	22.65	0.65	1.52	
	47.00	54.00	7.00	1.34	Middle Trap
Including	51.50	53.00	1.50	3.44	
LV_054	8.23	13.00	4.77	10.75	Inter Trap
	8.83	9.50	0.67	20.00	
	10.74	11.27	0.53	18.20	
	12.00	12.50	0.50	25.40	
	38.18	39.18	1.00	3.89	
	38.18	38.68	0.50	7.57	
	63.50	64.00	0.50	6.97	Middle Trap
	69.00	70.50	1.50	0.77	
LV_055	180.50	181.00	0.50	2.74	Inter Trap
	294.00	294.50	0.50	5.10	Middle Trap

Figure 10.2 Lavrinha Drillhole Locations – 2013



*(**Source:** Yamana, 2013)*

Figure 10.3 Lavrinha Drillhole Locations – 2013 With Density Analysis Holes



Note: *Drillholes with Au (g/t) and density (g/cm^3) analysis are in blue. The red drillholes were only analyzed for Au (g/t).*

The 2014 campaign represented in-fill drilling based on the results of the 2013 campaign. Due to changes in Yamana strategies this campaign was logged, sampled but not completely analyzed, and samples were stored in the core shed at Ernesto. Aura decided to analyze these samples in 2015 at SGS Laboratories to incorporate the results and achieve a more robust resource model.

The campaign consisted of a total of 78 drill holes for 8,145.11 m of diamond drilling, and 5,916 samples were analyzed by gold fire assay. 48 drillholes for 4,781.31 m and 3,642 samples were analyzed at ALS Chemex Laboratories by Yamana in 2014. The remaining 30 drill holes for 3,363.80 m and 2,274 samples were analyzed by Aura in 2015 at SGS Laboratories. All drillholes were to the same azimuth and dip (140°/60° at the collar) and were surveyed using the Maxibor system for each 3 m depth. 2014 significant intersections are presented in Table 10.4 and drillhole locations are presented in Figure 10.4.

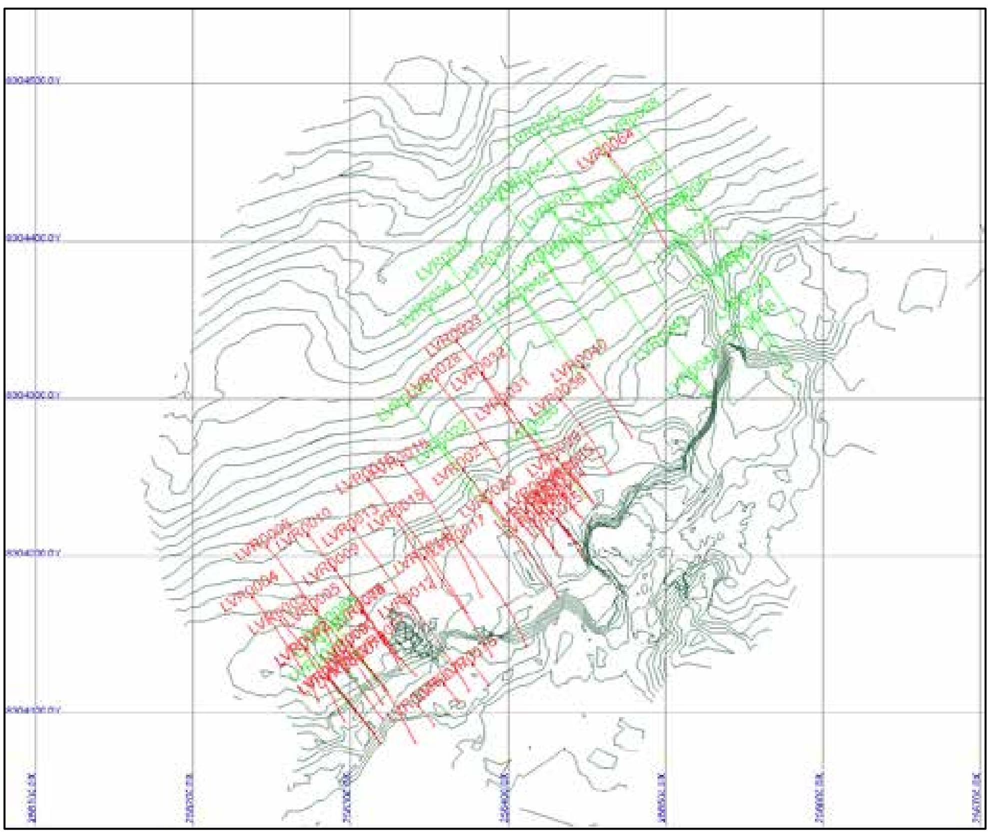
TABLE 10.4
SIGNIFICANT INTERSECTIONS - LAVRINHA 2014 EXPLORATION DRILLHOLES

Hole-ID	From (m)	To (m)	Length (m)	Weighted Average Au (g/t)
LVR0001	41	45	4	0.71
LVR0001	48	54	6	0.76
LVR0002	53	58	5	0.76
LVR0002	66	69	3	7.68
LVR0005	57	60	3	4.36
LVR0005	72	73	1	27.78
LVR0006	133	135	2	1.73
LVR0006	137	141	4	1.22
LVR0008	40	43	3	0.55
LVR0008	50	53.5	3.5	1.40
LVR0009	39	42	3	4.54
LVR0009	52	56	4	10.38
LVR0009	59	61	2	2.18
LVR0009	107	109	2	1.94
LVR0010	124	126	2	1.47
LVR0012	29.18	32	2.82	4.13
LVR0012	35	38.7	3.7	1.48
LVR0012	58	60	2	5.94
LVR0012	112	113.9	1.9	0.79
LVR0013	51	53	2	1.47
LVR0013	97	99	2	1.05
LVR0016	107	109	2	0.86
LVR0016	126	128	2	2.51
LVR0017	96	97.5	1.5	1.41
LVR0018	104	106	2	1.62
LVR0021	87	88	1	6.78
LVR0021	120	121.6	1.6	1.12
LVR0022	90	91	1	2.05
LVR0022	96	98	2	1.69
LVR0025	27	31	4	1.33
LVR0025	42	44	2	6.36
LVR0025	48	52	4	9.88
LVR0026	65	71	6	43.50
LVR0028	79	87	8	3.23

Table 10.4 Significant Intersections - Lavrinha 2014 Exploration Drillholes				
Hole-ID	**From (m)**	**To (m)**	**Length (m)**	**Weighted Average Au (g/t)**
LVR0029	51	53	2	15.72
LVR0030	91	94.3	3.3	2.17
LVR0031	75	79	4	2.49
LVR0032	73	77	4	15.18
LVR0032	99	103	4	2.03
LVR0032	109	112	3	40.87
LVR0033	59	61	2	2.23
LVR0033	87	95	8	2.39
LVR0033	106	107	1	18.01
LVR0036	36.5	38	1.5	2.32
LVR0036	48	51	3	31.14
LVR0036	77	80	3	1.03
LVR0038	114	116	2	3.46
LVR0040	61	63	2	6.30
LVR0046	78	79	1	8.33
LVR0047	120	122	2	4.12
LVR0047	128	130	2	1.52
LVR0048	59	63	4	3.31
LVR0049	52	58	6	6.13
LVR0052	68	70	2	2.49
LVR0052	102	108	6	2.88
LVR0053	97	101	4	1.53
LVR0054	79.57	80.15	0.58	17.24
LVR0054	119	121	2	9.84
LVR0056	99	101	2	1.23
LVR0057	112	114	2	10.44
LVR0063	105	106	1	2.08
LVR0069	56	57.67	1.67	4.80
LVR0070	57	60.25	3.25	30.92
LVR0072	56	63	7	6.31
LVR0073	32	33.5	1.5	6.68
LVR0073	37	41	4	2.78
LVR0073	43	47	4	1.94
LVR0074	19	20	1	29.96
LVR0076	13	15	2	1.41
LVR0077	6	8	2	2.67
LVR0077	36	38	2	2.77
LVR0078	2	4	2	1.14
LVR0078	26	28	2	1.19
LVR0079	30	33	3	4.53
LVR0080	30	32	2	2.18
LVR0081	28	29	1	115.92
LVR0081	48	49	1	6.06
LVR0082	33	38	5	5.54

TABLE 10.4 SIGNIFICANT INTERSECTIONS - LAVRINHA 2014 EXPLORATION DRILLHOLES				
Hole-ID	**From (m)**	**To (m)**	**Length (m)**	**Weighted Average Au (g/t)**
LVR0082	41	43	2	1.65
LVR0083	28	30	2	0.84
LVR0085	48	51	3	4.82
LVR0087	43	45	2	2.26
LVR0088	40	44	4	0.82
LVR0088	65	68	3	10.79
LVR0089	34	36	2	2.50
LVR0089	38.61	45	6.39	2.40
LVR0090	37	39	2	6.47
LVR0090	48	51	3	1.50
LVR0090	54	56	2	2.97
LVR0091	27	28	1	2.16
LVR0091	36	37.5	1.5	3.03

Figure 10.4 Lavrinha Drillhole Locations – 2014



Note: *Drillholes analyzed by Yamana are in red, and drillholes analyzed by Aura are in green.*

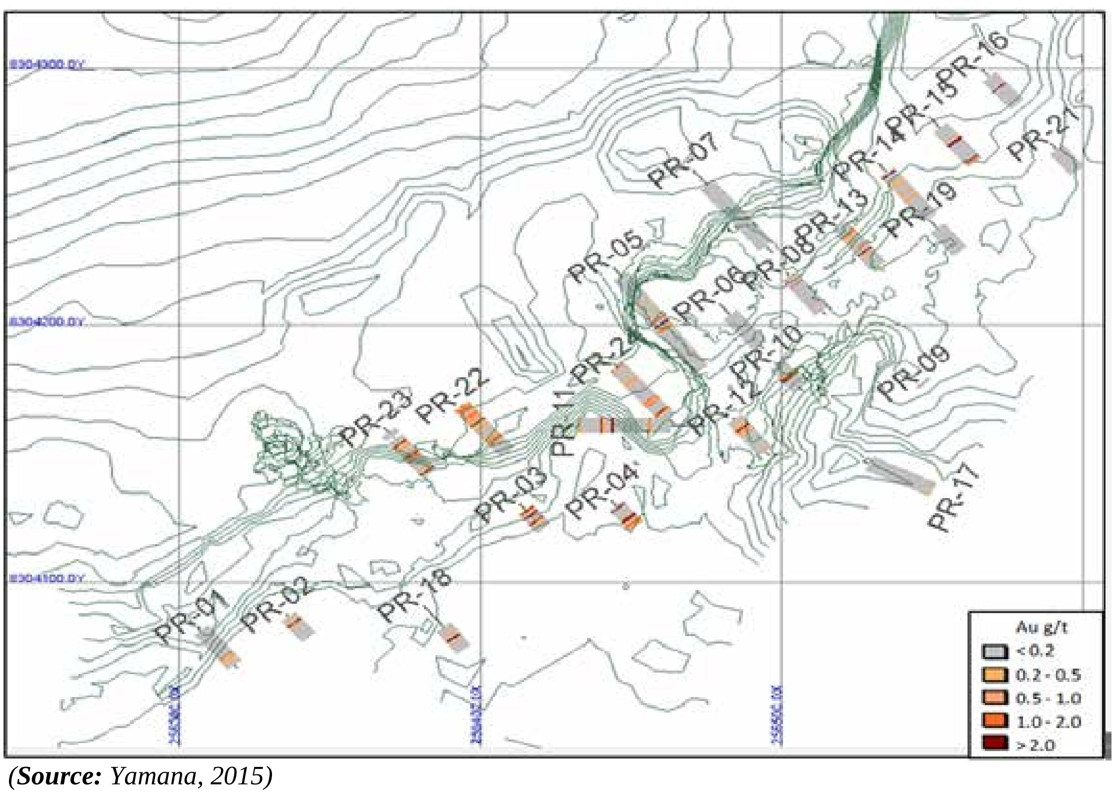
In 2015, Aura identified a lack of drill information near to the surface extension of the mineralization seen in the outcrops which was not considered in the resource model generated by Yamana. Aura decided to carry out a confirmatory drill campaign to provide better resource definition and improved accuracy of estimated grades.

The campaign consisted of 21 drill holes and 997.4 m of diamond drilling, with 845 samples analyzed by gold fire assay at the São Francisco Mine laboratory, and checks on the mineralized intervals with field duplicates sent to SGS laboratories. All drillholes were at the same azimuth and dip (140°/60° at the collar) and measured using the Maxibor survey system at 3 m intervals. 2015 significant intersections are presented in Table 10.5 and 2015 drillhole locations are presented in Figure 10.5.

TABLE 10.5
SIGNIFICANT INTERSECTIONS - LAVRINHA 2015 EXPLORATION DRILLHOLES

Hole-ID	From (m)	To (m)	Length (m)	Weighted Average Au (g/t)
PR-10	12	17	5	1.65
PR-04	6	8	2	2.91
PR-04	17	25.75	8.75	2.73
PR-16	20	22	2	1.88
PR-02	9	12	3	1.29
PR-03	7	12	5	1.44
PR-03	15	19	4	1.23
PR-12	10	13	3	1.21
PR-18	30	32	2	2.81
PR-01	29	32	3	1.38

Figure 10.5 Lavrinha Drillhole Locations - 2015

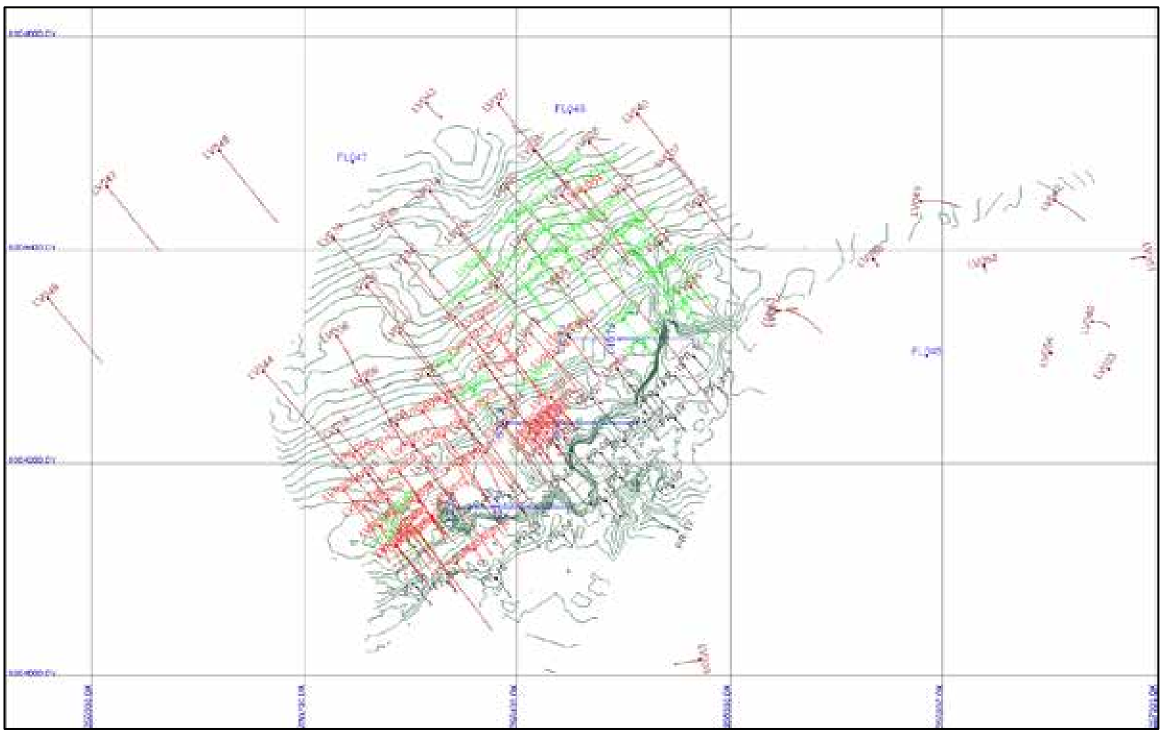


*(**Source:** Yamana, 2015)*

A summary of all drilling is presented in Figure 10.6 and Table 10.6.

TABLE 10.6 **SUMMARY OF LAVRINHA EXPLORATION DRILLHOLES**							
Campaign	**Prefix**	**Mangement**	**Drilling Company**	**Analysis**	**Drillholes**	**Metres**	**Samples**
1994/1995	FL	TVX	TVX	Nomos	9	1,711.77	683
2013	LV	Yamana	Servitec	ALS	55	10,013.12	9,466
2014	LVR	Yamana	Geosol	ALS	48	4,781.31	3,642
	LVR	Yamana	Geosol	SGS	30	3,363.80	2,274
	Subtotal				78	8,145.11	5,916
2015	PREFIX	Aura	Rede e Servitec	SGS/SF	21	997.40	845
Total					**163**	**20,867.40**	**16,910**

Figure 10.6 Lavrinha Drillhole Locations – All Years



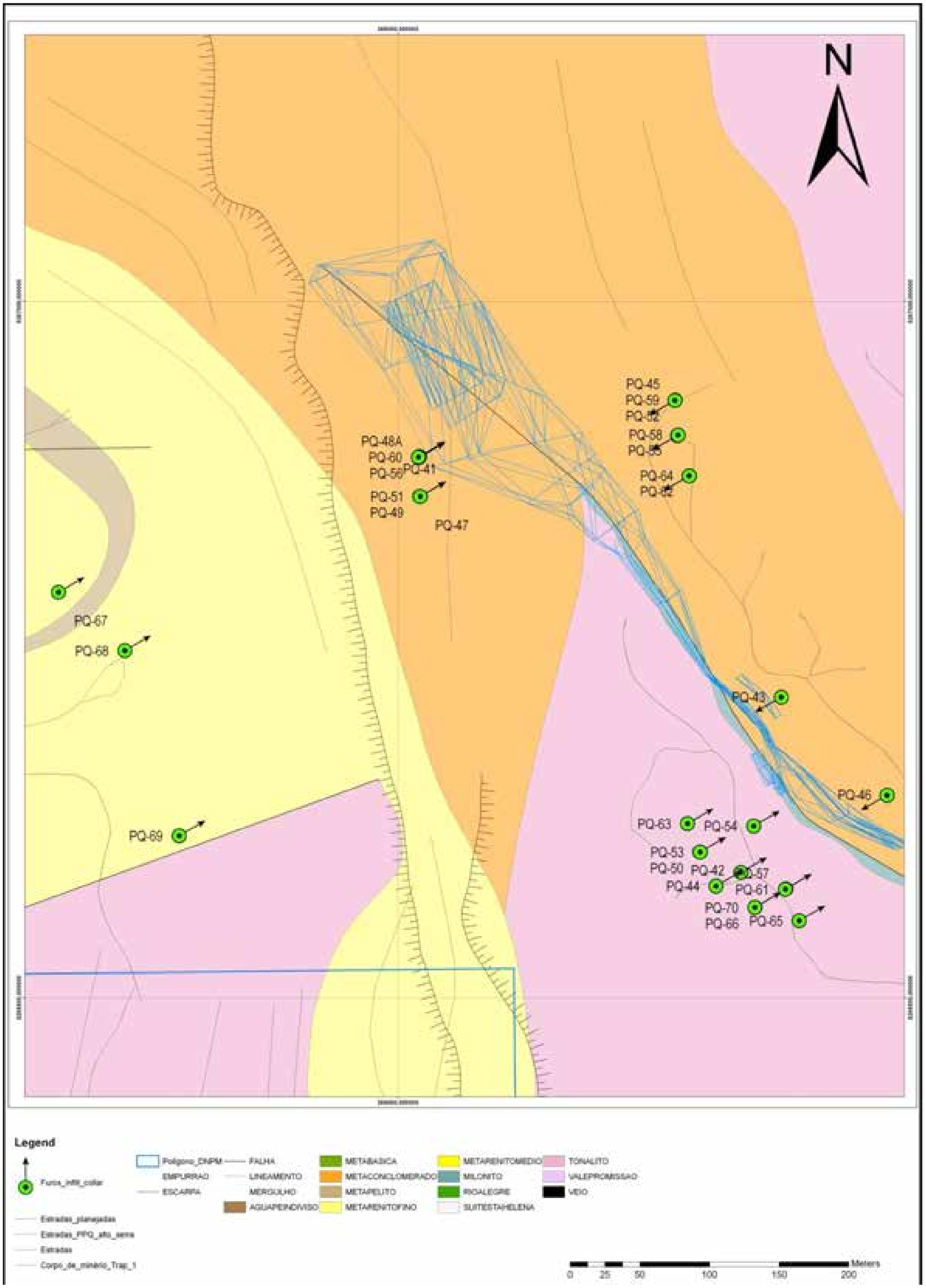
10.3 PAU-A-PIQUE

Yamana conducted four drilling campaigns on Pau-a-Pique with its first two completed in 2006. 25 holes (PQ-01 to PQ-25) totalling 8,099.9 m were drilled. A third campaign of 14 drillholes took place in 2007, totalling 7,506.2 m. This program was focused on expanding the resource along the NW strike and delineation at depth. The fourth drill campaign, carried out in 2008, was a combination of in-fill and exploratory drilling. 30 holes totalling 7,285.25 m were drilled. The main focus of the fourth campaign was to convert 51% of 2008 Inferred resources into the Measured/Indicated class and to define the limits of the resource.

The drilling at Pau-a-Pique totals 71 holes with 22,891.35 m. All drillholes were developed by REDE Energenharia utilizing the wire line method, starting out with HQ diameter and decreasing to NQ diameter in fresh rock. The drill holes were surveyed with a Maxibor II, reading twice every 3 m, with the maximum acceptable difference between the two measures of 2%. The deviation and inclination values used in the database were the mean between the two numbers.

For the drill campaigns prior to 2008, the relationship between the drillholes (that generally had a 60° inclination) and the mineralization (dipping at 85°), led to true width calculations of approximately 57%. Drillhole locations are shown in Figure 10.7 and significant intersections for the 2008 drilling are presented in Table 10.7. A cross-section, looking NE along Line 180W is presented in Figure 10.8.

Figure 10.7 Pau-a-Pique 2008 Drillhole Locations

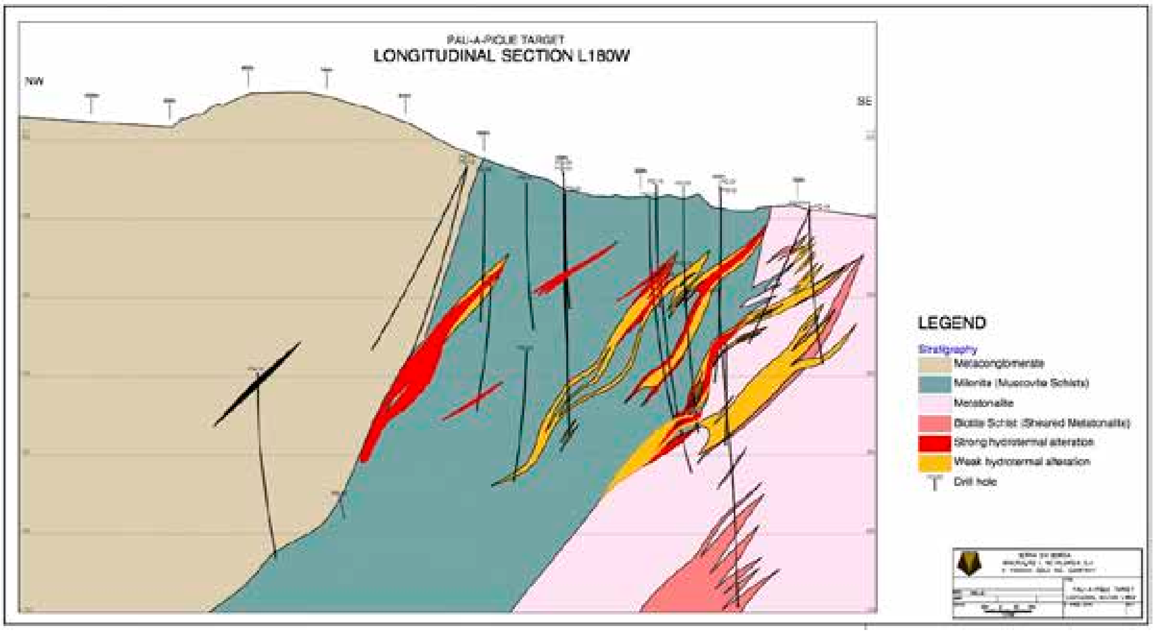


(**Source:** *Yamana, 2008)*

TABLE 10.7
PAU-A-PIQUE 2008 SIGNIFICANT DRILLHOLE INTERSECTIONS

Hole	Line	From (m)	To (m)	Length (m)	True Width (m)	Grade Au (g/t)
PQ-41	L550N	291	312	21	13	5.34
Including		297	299	2	1.23	19.3
PQ-42	L200N	152	161	9	4.8	2.45
Including		153	156	3	1.6	5.39
PQ-42	L200N	202	207.6	5.6	5.6	3.25
PQ-43	L300N	32	34	2	2	0.27
PQ-44	L200N	232.5	259	26.5	17.5	1.2
Including		232.5	236.5	4	3	3.4
PQ-45	L525N	224	233	9	-	2.66
PQ-45	L525N	241	247	6	-	1.77
PQ-45	L525N	296	304	8	-	2.02
PQ-45	L525N	302	304	2	-	5.8
PQ-47	L550N	258	268.75	10.75	6.5	7.58
Including		262	266	4	2.4	16.15
PQ-48A	L575N	233	238	5	3.5	3.36
Including		234.4	236.1	1.7	1.2	8.86
PQ-49		214	221	7	5.3	6.8
Including		218	220	2	1.5	19.5
PQ-52	L525N	165	180	15	5	3.85
Including		173	178	5	1.7	7.06
PQ-53	L225N	158	167	9	5.5	3.67
Including		163	167	4	2.5	7.54
PQ-54	L225N	99	104	5	3	3.22
PQ-55	L500N	119.1	131	11.9	6	2.44
Including		122.2	128	5.8	2.9	4.36
PQ-56	L575N	202.7	205.1	2.4	2.4	10.57
PQ-57	L175N	118	124.5	6.5	4	1.98
Including		123	124.5	1.5	0.9	5.71
PQ-59	L525N	113	129	16	10.5	8.27
Including		113	119	6	3.9	16.17
PQ-60	L575N	290	313.5	23.5	16	6.13
Including		301	312	11	7.5	8.04
PQ-61	L175N	139	159	20	15	12.57
Including		147	158	11	8.3	21.6
PQ-66	L175N	202	211	9	6	1.9
Including		202	207	5	3.33	2.87
PQ-70	L175N	225	230	5	3.15	1.3

Figure 10.8 Pau-a-Pique Cross Section – Line 180W Looking NE



*(**Source:** Yamana, 2008)*

Aura conducted a drill campaign at Pau-a-Pique in 2015-2016. 27 holes totalling 3,160.0 m were drilled. Drilling was concentrated mainly on NW strike and NW down plunge extensions of the Pau-a-Pique main lens (P1 zone) below current development levels. Another objective was to delineate mineral resources in the SE portion of deposit (P3 and P4 zones) below mined-out levels to add more ounces and also convert Inferred resources to the Indicated category.

Drillholes were collared from underground accesses by REDE Energenharia and Foraco utilizing the wire line method at BQ diameter. The drill holes were surveyed with a Maxibor II, reading twice every 3 m. A 5% tolerance value is used to compare the inclination in the two runs, and then the survey report was validated.

All samples from this drill campaign was analysed by fire assay method at the Sao Francisco mine lab. Approximately 16% of the samples were sent to SGS in Belo Horizonte to check and validate the Sao Francisco laboratory results.

A longitudinal section (looking NE) of deposit showing 2015-2016 drilling locations in red lines is presented in Figure 10.9 and significant intersections for the drilling are presented in Table 10.8.

Figure 10.9 Longitudinal Section Of Pau-a-Pique Showing 2015-2016 Drill Hole Locations

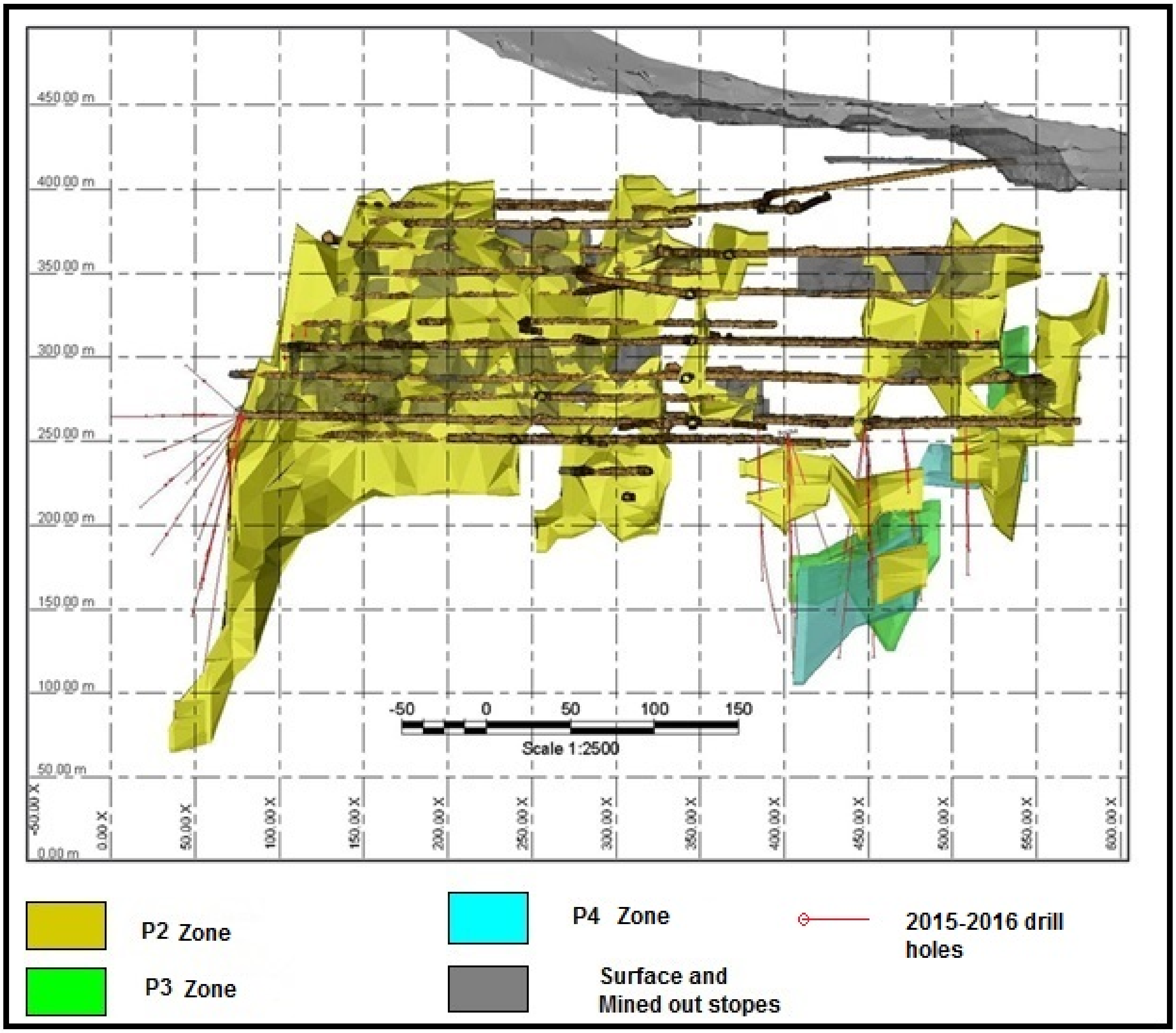


TABLE 10.8 PAU-A-PIQUE 2015-2016 SIGNIFICANT DRILLHOLE INTERSECTIONS					
Hole-ID	**From (m)**	**To (m)**	**Length (m)**	**True Width (m)**	**Grade Au (g/t)**
PPQ-527	45.00	51.00	6.00	4.62	1.91
Including	48.79	51.00	2.21	1.70	4.10
PPQ-528	47.00	49.00	2.00	1.59	5.74
PPQ-529	60.00	66.00	6.00	3.98	2.17
PPQ-530	72.00	78.00	6.00	3.51	5.24
PPQ-530	87.00	96.00	9.00	5.05	3.40
PPQ-533	72	73.27	1.27	0.73	1.94
PPQ-534	65.72	70.00	4.28	2.57	12.97

TABLE 10.8 PAU-A-PIQUE 2015-2016 SIGNIFICANT DRILLHOLE INTERSECTIONS					
Hole-ID	**From (m)**	**To (m)**	**Length (m)**	**True Width (m)**	**Grade Au (g/t)**
PPQ-535	87	96	9.00	4.81	8.70
PPQ-535	107	111	4.00	2.14	1.85
PPQ-536	76.32	80.00	3.68	2.10	3.14
PPQ-510	54	57	3.00	2.46	8.95
PPQ-510	63	67	4.00	3.32	2.10
PPQ-510	77	81	4.00	3.32	1.96
PPQ-511	35.51	39	3.49	2.45	1.66
PPQ-511	57	60	3.00	2.13	2.55
PPQ-511	73	79	6.00	4.35	5.34
PPQ-511	82	85	3.00	2.16	1.55
PPQ-512	39.5	43.6	4.10	2.70	4.10
PPQ-512	70	72	2.00	1.38	1.77
PPQ-512	94	97	3.00	2.06	5.16
PPQ-508	79	81	2.00	1.20	2.75
PPQ-508	95	102	7.00	3.60	2.32
PPQ-520	46	48	2.00	1.73	3.51
PPQ-521	101	102	1.00	0.80	9.32
PPQ-521	131	133	2.00	1.64	7.32
PPQ-506	74	77	3.00	2.52	2.04
PPQ-506	82	86.2	4.20	3.51	1.91
PPQ-504	39	40.2	1.20	0.81	4.33
PPQ-504	80	86	6.00	4.10	3.54
PPQ-504	104	106.5	2.50	1.70	11.33

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

The following description has largely been taken from Yamana's report titled, "Ernesto and Pau-a-Pique Feasibility Study Report, Revision No. 2", dated March 17, 2010 and prepared by Ausenco de Brasil Engenharia, Ltda., of Belo Horizonte, Minas Gerais, Brazil.

11.1 ERNESTO DEPOSIT

Ernesto drill core was cut in half by sawing with a diamond rock saw. Cutting was done one piece at a time with one half of the core returned to the core box for archival storage and the other half placed in a labelled cotton bag having a unique, sequential sample number written on the bag. Sample intervals were determined and marked by a geologist and samples were subsequently measured, split and bagged by a technician supervised by the geologist. Typical core samples ranged in size from 2 kg to 4 kg each. Core from all drilling programs are stored under cover at separate secure archival locations near the Project site. Samples ranged from 0.35 m to 2 m and were typically 1 m in length.

Previous companies' core samples were prepared and analysed at the Nomos Laboratory, in Rio de Janeiro, by conventional fire assay, with an AA ("Atomic Absorption") finish. Standard sample preparation at the laboratory includes drying, crushing and splitting a 300 g subset from the original sample pulp. This subset is then pulverised to 200# and a 30 g split is taken and digested in a hot aqua regia solution. Fire assay is followed by an AA finish except for samples containing more than 10 ppm gold, which are followed by a gravity finish. Nomos' detection limit for gold was 0.01 ppm.

Preparation and analysis of Yamana core samples were carried out by the São Vicente Laboratório ("São Vicente Lab") in Mato Grosso, Brazil, Mineração Fazenda Brazileiro Laboratório ("MFB Lab") in Bahia, Brazil, or Serra da Borda Mineracão e Metalurgia S.A., ("SBMM") in Ponte e Lacerda, Mato Grosso, Brazil, all of which are operated by Yamana, or sent to SGS Geosol Laboratório Ltda., ("SGS Geosol"), in Vespasiano, Minas Gerais, Brazil.

The procedure for preparation and analysis is dependent on whether the interval is considered to be mineralized or barren. If barren, conventional fire assay with AA finish is requested and, if mineralization is considered to be present, screen fire assay method is requested.

Samples analyzed by screen fire assay method (weighing approximately 8 kg) are pulverized to 150# and the resulting pulp is sieved using a 100# sieve, assuring a minimum of 30 g of retained coarse material. The undersized material is quartered and three aliquots of 50 g each are selected and assayed for gold, using fire assay followed by AA. The grade of the sample is calculated from the weighted average of the four results.

The procedure for the conventional fire assay is to crush the sample at 95% less than 10#. Using a rotary splitter, an aliquot of 1 kg is selected and milled at 95% less than 150#. Further quartering allows the separation of a 50 g aliquot, which is analyzed for Au via conventional fire assay.

SGS operates 1,650 offices and labs throughout the world. Sample processing services at SGS are ISO 17025 accredited by the Standards Council of Canada. Quality Assurance procedures include standard operating procedures for all aspects of the processing and also include protocols for training and monitoring of staff. ONLINE LIMS is used for detailed worksheets, batch and sample tracking including weights and labeling for all the products from each sample.

SGS Geosol has a quality management system ("QMS") in accordance with ISO 9001:2008 for chemical and geochemical analyzes of soil, rock and ore.

SGS Geosol's analytical quality is systematically assessed internally, as well as by participating in internationally recognized inter-laboratory proficiency testing.

Mr. Richard Routledge, P.Geo., of P&E, visited the SBMM lab in Ponte e Lacerda during a site visit to the EPP Project in June 2015 and determined the lab to be clean with modern equipment, rivalling that of commercial laboratories. Neither the SBMM nor MFB labs are certified, however they both participate in round robin testing for quality assurance and quality control purposes.

11.2 LAVRINHA DEPOSIT

The sample preparation and analysis of the Lavrinha Project are directly related to the Ernesto Project and the drilling campaigns conducted by Yamana in the past. Therefore it was decided to report in two separate sub-sections, sample analysis prior to 2014, which was conducted by Yamana, and sample analysis that was conducted by Aura in 2015.

11.2.1 Core samples from drilling before 2014

The protocols and procedures applied by Yamana (before 2014) are consolidated and reported in previous NI 43-101 reports and explained in detail. The following description has largely been taken from Yamana's report titled, "Ernesto and Pau-a-Pique Feasibility Study Report, Revision No. 2", dated March 17, 2010 and prepared by Ausenco de Brasil Engenharia, Ltda., of Belo Horizonte, Minas Gerais, Brazil.

Previous companies' core samples were prepared and analyzed at the Nomos Laboratory, in Rio de Janeiro, by conventional fire assay, with an AA ("Atomic Absorption") finish. Standard sample preparation at the laboratory includes drying, crushing and splitting a 300 g subset from the original sample pulp. This subset is then pulverised to 200# and a 30 g split is taken and digested in a hot aqua regia solution. Fire assay is followed by an AA finish except for samples containing more than 10 ppm gold, which are followed by a gravity finish. Nomos' detection limit for gold was 0.01 ppm.

Yamana core samples were prepared at the São Vicente Laboratory, operated by Yamana, or sent to SGS–Geosol laboratory, in Belo Horizonte, for preparation and analysis. The procedure for preparation and analysis depends on whether the interval is considered to be mineralized or barren. If barren, conventional fire assay with AA finish is requested to the lab. If the interval is supposed to be mineralized, a screen fire assay method is used. The procedure for the screen fire assay is to have the sample, weighing approximately 8 kg, pulverised entirely to 150#. The resulting pulp is sieved using a 100# sieve, assuring a minimum of 30 g of retained coarse material. The undersize material is quartered and then three aliquots of 50 g each are selected and assayed for gold, using fire assay followed by AA. The grade of the sample is calculated from the weighted average of the four results.

The procedure for the conventional fire assay is to crush the sample at 95% less than 10#. Using a rotary splitter, an aliquot of 1 kg is selected and milled at 95% less than 150#. Further quartering allows the separation of a 50 g aliquot, which is analysed for Au (ppb) via conventional fire assay."

11.2.2 Core samples from 2014/2015 drilling

2014 and 2015 samples were analyzed following procedures that were implemented by Mineração Apoema S.A (Aura), represented by two Operational Procedures in Portuguese: PO-MA-COR-LAB-001 (sample preparation) and PO-MA-COR-LAB-005 (fire assay chemical analysis). The following paragraphs are reported based on the operational procedures.

The samples were properly coded to match the requested form from the lab, including ID, batch, summary or code of preparation procedure and ID of company (address, date, responsible, email and phone), the same as 2014 and prior to that.

The physical preparation consisted of drying in an oven at 110°C to 120°C and crushing with a jaw crusher at P90 < 10# to 2mm, followed by splitting with a Jones riffle splitter, with 1,000 g for kept for pulverizing and the remaining material stored at the core shed. The 1,000 g was pulverized in a Pan Mill to P95 < 150# and split with a carrousel splitter into aliquots of 250g for fire assay.

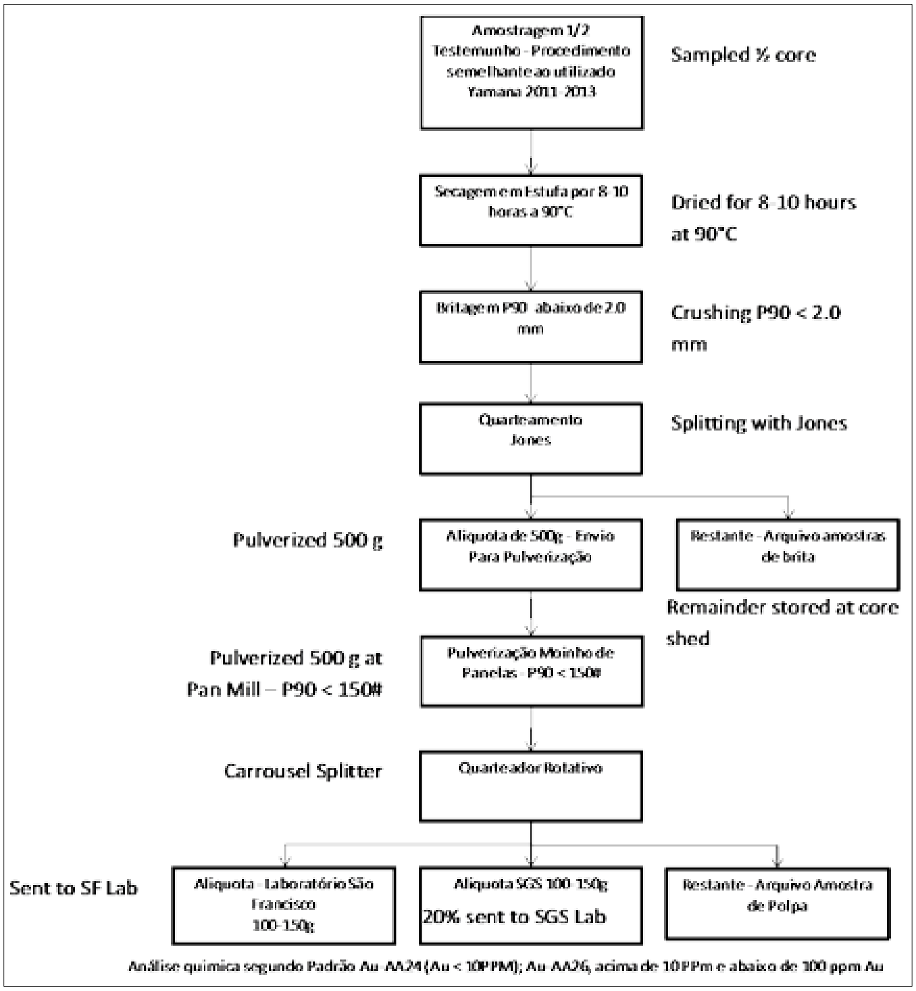
The samples from the 2014 campaign were sent to SGS, and for the 2015 campaign were extracted into two aliquots, one for the Sao Francisco lab and the other for SGS. 20% of the material allocated to the Sao Francisco lab was sent to SGS to confirm the precision of the Sao Francisco lab.

The chemical analysis carried out at the Sao Francisco lab is described in Operational Procedure "Análise de Ouro "Fire Assay" em Minérios - PO-MA-COR-LAB-005" in Portuguese and followed the AA24 ALS Chemex standard procedure of ALS Chemex, which consisted of:

- Fire assay and analysis of samples for gold by Atomic Absorption Spectroscopy ("AAS");
- The prepared sample is fused with a mixture of lead oxide, sodium carbonate, borax, silica and other reagents as required, inquarted with 6 mg of gold-free silver and then cupelled to yield a precious metal bead;
- Bead is digested in 0.5 ml dilute nitric acid in the microwave oven, 0.5 ml concentrated hydrochloric acid is then added and the bead is further digested in the microwave at a lower power setting;
- The digested solution is cooled, diluted to a total volume of 4 ml with de-mineralized water, and analyzed by AAS against matrix-matched standards;
- Detection Limit ("L/D") is set to 0.005 ppm Au and a maximum L/D of 10 ppm Au, if result is over 10 ppm Au the sample is automatically submitted for analysis using standards for a maximum detection limit of 100 ppm Au.

The flowsheet for the 2015 preparation and analysis of the Lavrinha Deposit is showed at Figure 11.1 (extracted from the original procedure in Portuguese). 2014 drill campaign samples were analyzed in 2015 and sent directly to SGS lab.

Figure 11.1 Flowsheet of 2015 Lavrinha Sample Preparation and Analysis



It is MCB's opinion that sample preparation, security and analytical procedures for the Lavrinha Project drilling and sampling programs were adequate for the purposes of this Mineral Resource Estimate.

11.3 PAU-A-PIQUE DEPOSIT

Pau-a-Pique drill core was cut in half by sawing with a diamond rock saw. Cutting was done one piece at a time with one half returned to the core box for archival storage and the other half placed in a labelled cotton bag having a unique, sequential sample number written on the bag. Sample intervals were determined and marked by a geologist and samples were subsequently measured, split and bagged by a technician supervised by the geologist. Typical core samples ranged, in size, from 2 kg to 4 kg each.

Drill core samples range from 0.31 m to 2 m and were typically 1 m in length. Channel samples range from 0.15 m to 2 m and are typically 1.2 m in length.

The MFB, SBMM, or SGS GEOSOL laboratory carried out sample preparation and analysis for Pau-a-Pique and procedures were the same as for the Ernesto Project.

Security for both Projects includes surveillance of the core yard and office facilities and up-to-date copies of the sample databases kept at the corporate office.

It is P&E's opinion that sample preparation, security and analytical procedures for both the Ernesto and Pau-a-Pique Projects drilling and sampling programs were adequate for the purposes of this Mineral Resource Estimate.

12.0 DATA VERIFICATION

12.1 SITE VISIT AND DUE DILIGENCE SAMPLING

The EPP Project was visited by Mr. Andrew Bradfield, P.Eng., of P&E from June 18 to June 23, 2015, for the purposes of completing a site visit and due diligence sampling.

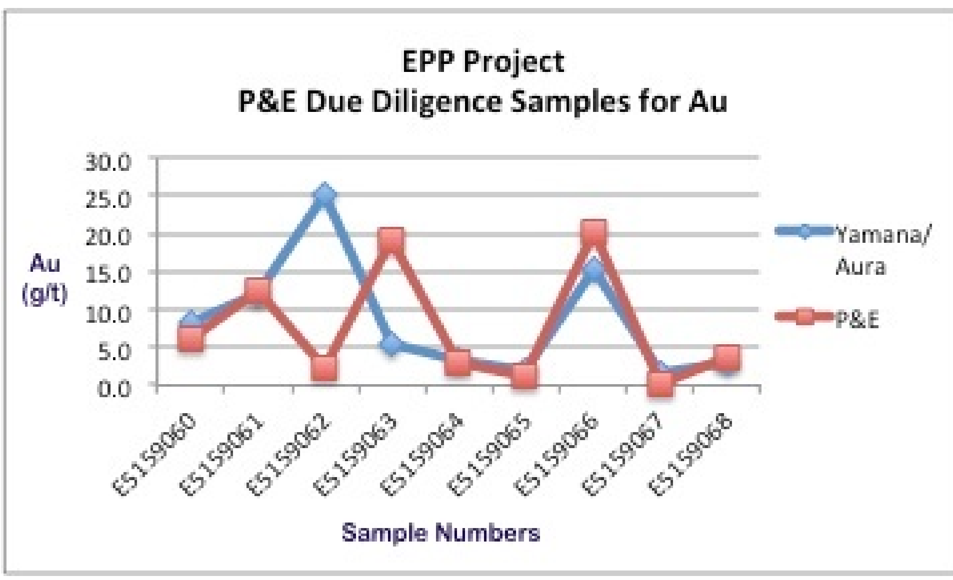
A site visit to the Project was also carried out by Mr. Richard Routledge, P.Geo., of P&E from June 18 to June 21, 2015. Mr. Routledge undertook an inspection of the Ernesto mine laboratory and obtained information pertaining to general data acquisition procedures, core logging procedures and quality assurance/quality control ("QA/QC" or "QC").

Mr. Bradfield collected nine samples from four diamond drill holes during the site visit. Samples were selected over a range of grades from the stored drill core and collected by taking a 1⁄4 split of the half core remaining in the core box. Samples were placed into plastic bags with a unique tag identification, and were placed into a larger bag for transport, via courier, to SGS Geosol for both preparation and analysis.

Samples were analyzed for gold by fire assay with an AA finish. Bulk densities were also determined on all 12 samples.

Results of the site visit due diligence samples are presented in Figure 12.1.

Figure 12.1 EPP Project Due Diligence Sample Results for Au: June 2015



With the exception of two samples (sample numbers E5159062 and E5159063), all samples match closely. Such variation in grade has been demonstrated within the deposit and the author considers the due diligence results to be acceptable.

12.2 ERNESTO QUALITY ASSURANCE/QUALITY CONTROL PROGRAM

12.2.1 2006 - 2009 Drill Programs

The following description has largely been summarized from Yamana's report titled, "Ernesto and Pau-a-Pique Feasibility Study Report, Revision No. 2", dated March 17, 2010 and prepared by Ausenco de Brasil Engenharia, Ltda., of Belo Horizonte, Minas Gerais, Brazil.

Yamana's QA/QC programs for the years 2005 to 2009 included the routine insertion of blank samples and gold standards into the sample batches sent for analysis to the laboratory.

Blanks were inserted at a rate of 1 in 40 samples and attempted to mark the end of an expected mineralized interval. The blank material used was locally sourced from white quartz vein collected nearby the Ernesto target.

One or two standards were inserted into each zone where mineralization was expected; giving an overall average of 1 in 15 standards for each in mineralized zones, and the standard grade was matched to the expected average grade of the mineralization.

A ± 1 standard deviation from the accepted mean value was permitted during monitoring of the standards and any batches with blank or standard failures were re-assayed.

12.2.2 2013-2014 Drill Programs

All drilling undertaken at the Ernesto Project throughout 2013 to 2014 was outside of the current Resource Estimate area and has therefore not been reviewed for QA/QC purposes.

12.2.3 2015 Drill Program

A total of 502 m of drill core were sampled during the Ernesto drill program and 537 samples were sent to SGS Geosol over 15 batches.

<u>Performance Of Standards</u>

Aura used five different Certified Reference Materials ("CRMs") of varying grade to monitor gold during the 2015 drill program at Ernesto: the G311-7, G312-4, G398-10, G912-6 and G997-6 standards. All standards originate from Geostats Pty Ltd, ("Geostats") of Western Australia, Australia. Multiple standards were inserted in every sample batch sent to the laboratory, with a total of 46 standards included in the 2015 QA/QC program.

Data falling within ± 2 standard deviations from the accepted mean value were passed. Data falling outside ± 3 standard deviations from the accepted mean value, or two consecutive data points falling between ± 2 and ± 3 standard deviations on the same side of the mean, were failed.

All five standards display satisfactory results for gold, with no failures recorded.

Performance Of Blanks

Aura routinely inserted blank samples into the sample stream during the 2015 drill program at Ernesto to monitor contamination. Multiple blanks were inserted into every batch sent for analysis and a total of 58 blanks were included during the 2015 QC program.

An upper tolerance limit of 0.08 g/t Au was set to monitor the blank material and all results, except one, fell below this limit. The single result that exceeded the tolerance limit returned a value of 0.126 g/t Au and no samples from this batch were included in the current Resource Estimate.

Performance Of Duplicates

A total of 49 field duplicates were routinely inserted into the sample stream throughout the 2015 QC program at Ernesto to monitor precision for gold.

A scatter plot of the original versus duplicate samples revealed moderate to poor precision for gold, acceptable and expected at this level of analysis.

12.3 LAVRINHA DATA VERIFICATION

This data verification section describes the processes used to verify exploration data in the Lavrinha Project study carried out by MCB during site visits by the QP. MCB also allocated a geologist for three months to re-log the core for alteration, lithology and to construct geological models and mineralized zones.

The objectives of the data verification were:

- Visit of the previously mined area and inspection of mineralized zones;
- Confirmation that gold is mainly associated with high hydrothermal alteration and quartz veins;
- Check core shed and facilities (logging area, density determination lab);
- Database verification of the diamond core drill data.

Marcelo Batelochi, AusIMM (CP) and director of MCB visited the Project between June 20th to 24th, 2015, and also re-visited the Project at the end of July and middle of August, 2015. The open pit mine site was inspected (Figure 12.2 and Figure 12.3), along with infrastructure (Figure 12.4), and a complete audit was performed (logging, certificates, and location) including visual inspection of core of selected drilholes: LV-15, LV-13 (Figure 12.5) and LV-03, and the São Francisco laboratory infrastructure was reviewed.

The author also checked the high grade gold intervals in the core boxes and associated hydrothermal alteration/quartz veins.

Figure 12.2 Outcrop of Lavrinha Mined Area

**Showing the high hydrothermal alteration associated with quartz veins and pseudomorph of sulfides. Rectangle in the center of the photo shows the position of the Figure 12.3 photo.*

Figure 12.3 Detail of high Hydrothermal Alteration Associated with quartz veins and pseudomorph of sulphides

Figure 12.4 Core Shed Located at Ernesto Mine

**Showing good infrastructure to preserve core from the 2014 drill campaign. The previously drilled core was stored in bad condition in a shed in the town of Pontes e Lacerda.*

Figure 12.5 Chip of quartz vein with sulphide pseudomorph in a high hydrothermal alteration zone. Drillhole LV-13 – 31.8 m depth

Based on observations of MCB and Aura, it was decided to apply several improvements to the previous geological model, mainly performing geological, structural and alteration models before modeling the mineralization lodes. This task was accompanied by an additional drill campaign to confirm extension of the mineralization specifically to the surface where artisanal mining occurred in the past. After this MCB checked and could find no evidence to suggest additional verification sampling was necessary for confirmation of the presence of gold in the area. MCB also strongly recommended organization of the core in the temporary shed in Pontes e Lacerda.

An MCB geologist, Mr. Guilherme Canedo, was dedicated to the Lavrinha Project for three months. He revisited the core, re-logged all of the holes, performed geological mapping, and checked in detail all drill hole collar locations, down hole surveys, drill recovery, paper and digital logs and density analysis. All data related to assays, density and collar coordinates were verified and no major issues were identified.

One problem was related to lack of dates for chemical analysis in the database for campaigns before 2014, which in theory interfere with the coverage of analytical standards. However, these campaigns were validated in previous QAQC reports, and therefore they can be relied on for mineral resource estimation.

12.4 LAVRINHA QUALITY ASSURANCE/QUALITY CONTROL PROGRAM

12.4.1 Introduction

Quality assurance ("QA") determines if assay data has precision and accuracy within generally accepted limits for the sampling and analytical method(s) in order to be used in a resource estimate with enough confidence. Quality control ("QC") consists of procedures used to ensure that an adequate level of quality is maintained in the process of collecting, preparing, and assaying the exploration drill samples. In general, QA/QC programs are designed to prevent or detect contamination and allow assaying (analytical), precision (repeatability), and accuracy to be quantified. In addition, a QA/QC program can disclose the overall sampling-assaying variability of the sampling method itself.

The general procedure applied at Lavrinha was created by Yamana and consisted of samples grouped into batches of 50-100. Five percent of standards, two percent of blanks, and two percent of field and pulp duplicates are included in each sample batch. QA/QC samples comprise 10% of the data. Blanks are inserted in the sample stream at the end of visible mineralization, standards are randomly inserted within mineralized intervals, and pulp and reject duplicates are randomly inserted in both mineralized and non-mineralized samples. Field duplicates are taken within mineralization. No check assays are performed by an alternate laboratory. Unfortunately this general procedure was not followed during the 2014 drill campaign and field duplicate samples were not collected.

Before the Yamana 2014 drill campaign, samples from Lavrinha sent to ALS Chemex Laboratories were included into Ernesto´s batch using the same standards, blanks and duplicates, and the QAQC analysis was performed in a similar approach (graphics and monthly internal reports).

2014 drilling was performed by Yamana, but the samples were stored in the core shed and not analyzed. In 2015 Aura decided to analyze the samples at the SGS lab in Vespasiano Minas Gerais.

Aura's 2015 drill campaign was carried out to confirm the extension of the deposit. Due to the urgency of receiving of the assay data it was decided to analyze at an internal laboratory at São Francisco Mine and to send 20% of the samples to SGS as an umpire laboratory.

A QA/QC report was prepared on monthly basis by an onsite database manager and reviewed by a project geologist. The report is also submitted to the head office for review. Batches of samples identified by QA/QC as anomalous were repeated by ALS Global at the request of the technical team.

The structure of this report consists of analysis and discussion related to two drilling campaigns carried out by Yamana (Before 2014) and sampling and drill campaigns carried out by Aura in 2015 (samples from 2014 drilling and 2015 drilling campaigns).

12.4.2 Drilling Campaigns before 2014 (Analyzed by Yamana)

The QA/QC results from drill campaigns prior to 2014 are summarized in previous NI 43-101 reports conducted by Yamana since 2010, specifically "ERNESTO E PAU-A-PIQUE - FEASIBILITY STUDY REPORT - RL-000E-00-0001" prepared by "Ausenco do Brasil Engenharia Ltda". As was mentioned in this report "The Ernesto Project (including the Lavrinha Deposit) had shown the transparency, reproducibility and reliability of the sample assay database that is appropriate for mineral resources and ore reserves evaluation. All procedures and techniques used in this analysis are according to the international exploration guidelines".

12.4.3 Contamination Evaluation – Blank Standards

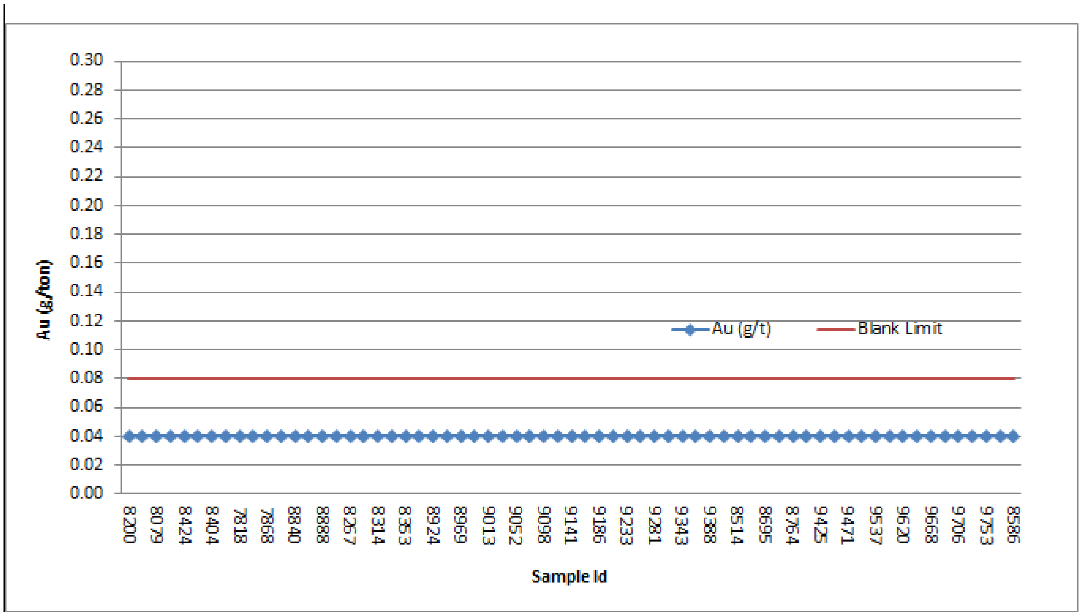
In order to control possible contamination during sample physical preparation, blank standards (analytical blanks) were inserted in the laboratory batch. These blank standards were inserted randomly in each sample batch. These samples had very low concentrations of gold (below detection limit) that allowed identification of any anomalous enrichment and indicated some contamination that required further investigation.

In the case of Lavrinha, the inserted blank standards represent 65 samples (Table 12.1) and were plotted in a control chart that showed 100% of samples below the top limit for contamination (Figure 12.6). There was no contamination during sample preparation.

TABLE 12.1
BLANK STANDARD SAMPLES

Batch	Date	Hole_id	Sample	Control I	Au (g/t)	Blank Lim	Status
LVR-FS-14-0041	25/08/2014	LVR0070	8200	Branco	0.04	0.08	Validated
LVR-FS-14-0041	25/08/2014	LVR0070	8225	Branco	0.04	0.08	Validated
LVR-FS-14-0039	02/09/2014	LVR0071	8079	Branco	0.04	0.08	Validated
LVR-FS-14-0039	02/09/2014	LVR0071	8103	Branco	0.04	0.08	Validated
LVR-FS-14-0043	04/09/2014	LVR0001	8424	Branco	0.04	0.08	Validated
LVR-FS-14-0043	04/09/2014	LVR0001	8444	Branco	0.04	0.08	Validated
LVR-FS-14-0043	04/09/2014	LVR0001	8404	Branco	0.04	0.08	Validated
LVR-FS-14-0043	04/09/2014	LVR0001	8462	Branco	0.04	0.08	Validated
LVR-FS-14-0036	08/09/2014	LVR0040	7818	Branco	0.04	0.08	Validated
LVR-FS-14-0036	08/09/2014	LVR0040	7844	Branco	0.04	0.08	Validated
LVR-FS-14-0036	08/09/2014	LVR0040	7868	Branco	0.04	0.08	Validated
LVR-FS-14-0040	08/09/2014	LVR0069	8171	Branco	0.04	0.08	Validated
LVR-FS-14-0047	09/09/2014	LVR0008	8840	Branco	0.04	0.08	Validated
LVR-FS-14-0047	09/09/2014	LVR0008	8866	Branco	0.04	0.08	Validated
LVR-FS-14-0047	09/09/2014	LVR0008	8888	Branco	0.04	0.08	Validated
LVR-FS-14-0047	09/09/2014	LVR0008	8911	Branco	0.04	0.08	Validated
LVR-FS-14-0042	10/09/2014	LVR0002	8267	Branco	0.04	0.08	Validated
LVR-FS-14-0042	10/09/2014	LVR0002	8293	Branco	0.04	0.08	Validated
LVR-FS-14-0042	10/09/2014	LVR0002	8314	Branco	0.04	0.08	Validated
LVR-FS-14-0042	10/09/2014	LVR0002	8333	Branco	0.04	0.08	Validated
LVR-FS-14-0042	10/09/2014	LVR0002	8353	Branco	0.04	0.08	Validated
LVR-FS-14-0042	11/09/2014	LVR0002	8373	Branco	0.04	0.08	Validated
LVR-FS-14-0048	15/09/2014	LVR0005	8924	Branco	0.04	0.08	Validated
LVR-FS-14-0048	15/09/2014	LVR0005	8947	Branco	0.04	0.08	Validated
LVR-FS-14-0048	15/09/2014	LVR0005	8969	Branco	0.04	0.08	Validated
LVR-FS-14-0048	15/09/2014	LVR0005	8991	Branco	0.04	0.08	Validated
LVR-FS-14-0048	15/09/2014	LVR0005	9013	Branco	0.04	0.08	Validated
LVR-FS-14-0048	15/09/2014	LVR0005	9034	Branco	0.04	0.08	Validated
LVR-FS-14-0050	17/09/2014	LVR0010	9052	Branco	0.04	0.08	Validated
LVR-FS-14-0050	17/09/2014	LVR0010	9075	Branco	0.04	0.08	Validated
LVR-FS-14-0050	17/09/2014	LVR0010	9098	Branco	0.04	0.08	Validated
LVR-FS-14-0050	17/09/2014	LVR0010	9120	Branco	0.04	0.08	Validated
LVR-FS-14-0050	17/09/2014	LVR0010	9141	Branco	0.04	0.08	Validated
LVR-FS-14-0050	17/09/2014	LVR0010	9170	Branco	0.04	0.08	Validated
LVR-FS-14-0051	19/09/2014	LVR0009	9186	Branco	0.04	0.08	Validated
LVR-FS-14-0051	19/09/2014	LVR0009	9209	Branco	0.04	0.08	Validated
LVR-FS-14-0051	19/09/2014	LVR0009	9233	Branco	0.04	0.08	Validated
LVR-FS-14-0051	19/09/2014	LVR0009	9256	Branco	0.04	0.08	Validated
LVR-FS-14-0051	19/09/2014	LVR0009	9281	Branco	0.04	0.08	Validated
LVR-FS-14-0052	21/09/2014	LVR0003	9319	Branco	0.04	0.08	Validated
LVR-FS-14-0052	21/09/2014	LVR0003	9343	Branco	0.04	0.08	Validated
LVR-FS-14-0052	21/09/2014	LVR0003	9366	Branco	0.04	0.08	Validated
LVR-FS-14-0052	21/09/2014	LVR0003	9388	Branco	0.04	0.08	Validated
LVR-FS-14-0044	21/09/2014	LVR0072	8489	Branco	0.04	0.08	Validated
LVR-FS-14-0044	21/09/2014	LVR0072	8514	Branco	0.04	0.08	Validated
LVR-FS-14-0046	23/09/2014	LVR0007	8674	Branco	0.04	0.08	Validated
LVR-FS-14-0047	23/09/2014	LVR0013	8695	Branco	0.04	0.08	Validated
LVR-FS-14-0047	23/09/2014	LVR0013	8719	Branco	0.04	0.08	Validated
LVR-FS-14-0047	23/09/2014	LVR0013	8764	Branco	0.04	0.08	Validated
LVR-FS-14-0047	23/09/2014	LVR0013	8785	Branco	0.04	0.08	Validated
LVR-FS-14-0053	24/09/2014	LVR0006	9425	Branco	0.04	0.08	Validated
LVR-FS-14-0053	24/09/2014	LVR0006	9447	Branco	0.04	0.08	Validated
LVR-FS-14-0053	24/09/2014	LVR0006	9471	Branco	0.04	0.08	Validated
LVR-FS-14-0053	24/09/2014	LVR0006	9493	Branco	0.04	0.08	Validated
LVR-FS-14-0054	26/09/2014	LVR0004	9537	Branco	0.04	0.08	Validated
LVR-FS-14-0054	26/09/2014	LVR0004	9605	Branco	0.04	0.08	Validated
LVR-FS-14-0055	26/09/2014	LVR0014	9620	Branco	0.04	0.08	Validated
LVR-FS-14-0055	26/09/2014	LVR0014	9645	Branco	0.04	0.08	Validated
LVR-FS-14-0055	26/09/2014	LVR0014	9668	Branco	0.04	0.08	Validated
LVR-FS-14-0054	29/09/2014	LVR0004	9565	Branco	0.04	0.08	Validated
LVR-FS-14-0056	29/09/2014	LVR0012	9706	Branco	0.04	0.08	Validated
LVR-FS-14-0056	29/09/2014	LVR0012	9731	Branco	0.04	0.08	Validated
LVR-FS-14-0056	29/09/2014	LVR0012	9753	Branco	0.04	0.08	Validated
LVR-FS-14-0045	29/09/2014	LVR0073	8558	Branco	0.04	0.08	Validated
LVR-FS-14-0045	30/09/2014	LVR0073	8586	Branco	0.04	0.08	Validated

Figure 12.6 Control Chart of Blank Standards before 2014 drill Campaign



12.4.4 Accuracy Analysis – Analytical Standards

The analytical standards were implemented to assess analytical accuracy and bias by comparing the assay results against the expected grade of the standard.

Lavrinha Project control for accuracy was performed by insertion of Certified Reference Materials from Geostats PTY Ltd Id G912-5, G905-6, G908-1, G912-6, G311-7, G909-2 and G997-3 which were similarly inserted for the Ernesto Project.

A low quantity of samples failed, as is presented in Table 12.2 and Figure 12.7, and there was a random bias denoting imprecision of analysis of approximately 10-15% that is acceptable for gold (Figure 12.8).

TABLE 12.2
QUANTITATIVE SUMMARY OF STANDARDS SHOWING THE LOW QUANTITY OF FAILURES

Standard type	Standard Mean	Number of Samples	Number of Failures	% failure > 2STDEV	Notifications	% Notifications
G912-5	0.38	5	1.0	20.00%	0	0.00%
G905-6	5.96	10	0.0	0.00%	0	0.00%
G908-1	0.06	13	0.0	0.00%	0	0.00%
G912-6	4.08	18	4.0	22.22%	1	5.56%
G311-7	0.36	4	0.0	0.00%	1	25.00%
G909-2	1.76	6	0.0	0.00%	1	16.67%
G997-3	1.41	9	0.0	0.00%	1	11.11%
		Total Desvios:	**5**	**Total Notificações:**	**4**	
Nº Amostras STDs	**65**	**92.308%**	OK		93.85%	OK
		7.692%	OUT		6.15%	OUT

Figure 12.7 Geostats Standard Failure

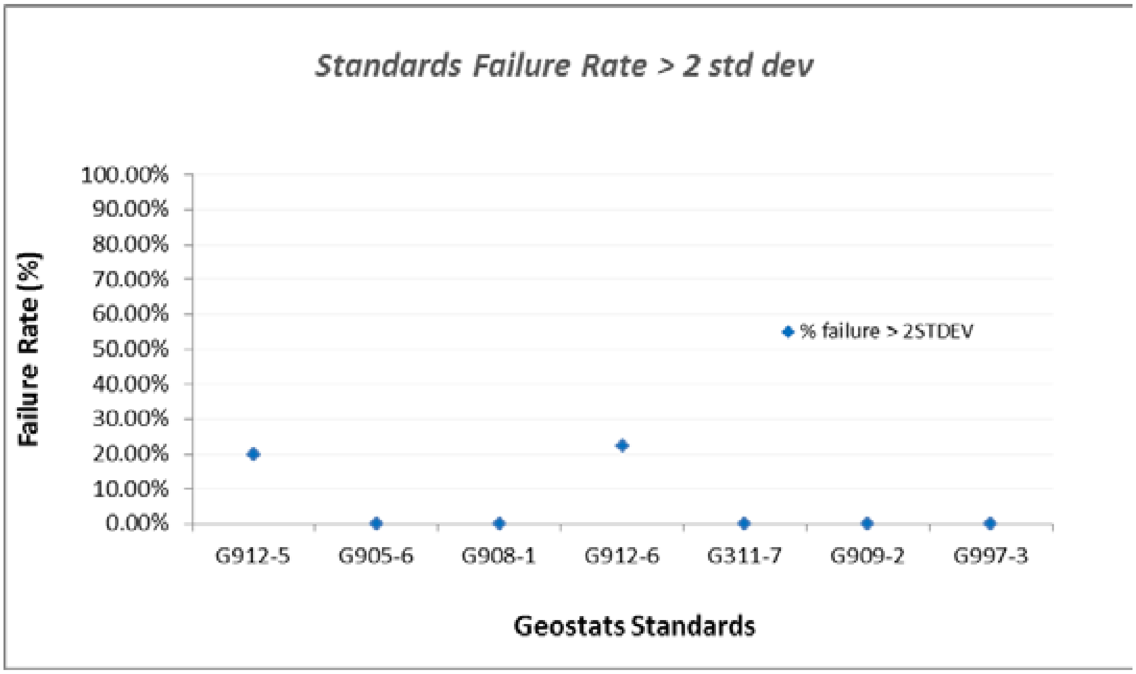
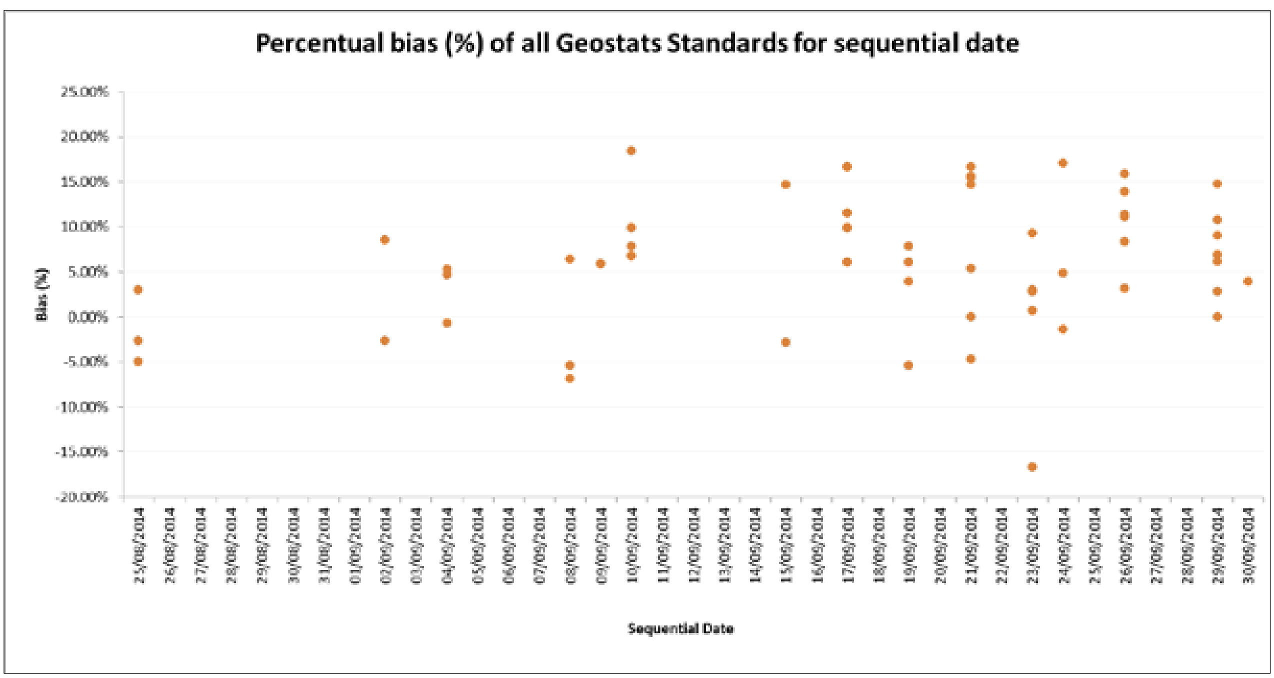


Figure 12.8 Bias Analysis of Geostats Standard for Sequential Date



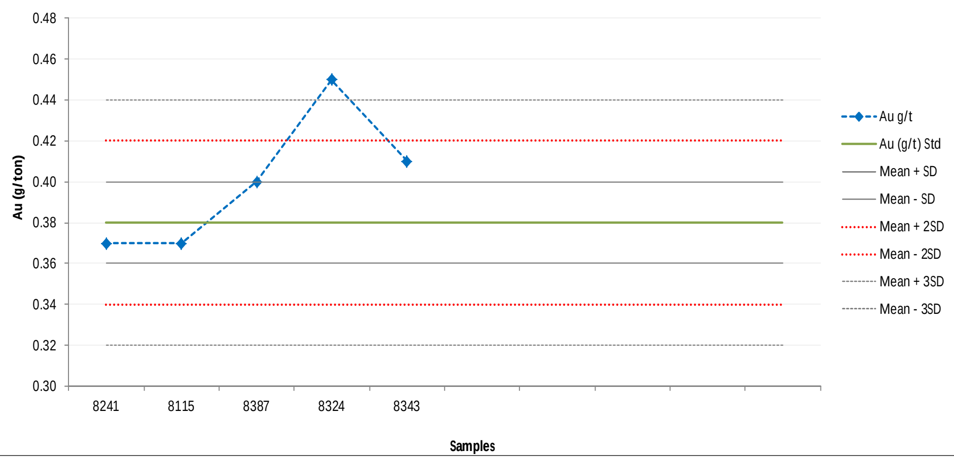
<u>"G912-5" Geostats Standard</u>

The results of "G912-5" Geostats Standard are based on 5 assayed samples. Only one sample was out of limits and was re-assayed, as shown in the Figure 12.9.

Figure 12.9 Plot of Au g/t for “G912-5” Geostats Standard

G912-5

Batch	Date	Hole_id	Sample	Standard Id	Au g/t	Au (g/t) Std	STV Dev	Mean + SD	Mean - SD	Mean + 2SD	Mean - 2SD	Mean + 3SD	Mean - 3SD	Au Dev	Au Deviation	Average results	Bias	Status
LVR-FS-14-0041	25/08/2014	LVR0070	8241	G912-5	0.37	0.38	0.02	0.40	0.36	0.42	0.34	0.44	0.32	97.37%	0.40	2.56	-2.63%	Ok
LVR-FS-14-0039	02/09/2014	LVR0071	8115	G912-5	0.37	0.38	0.02	0.40	0.36	0.42	0.34	0.44	0.32	97.37%	0.40	2.56	-2.63%	Ok
LVR-FS-14-0043	04/09/2014	LVR0001	8387	G912-5	0.40	0.38	0.02	0.40	0.36	0.42	0.34	0.44	0.32	105.26%	0.40	2.77	5.26%	Ok
LVR-FS-14-0042	10/09/2014	LVR0002	8324	G912-5	0.45	0.38	0.02	0.40	0.36	0.42	0.34	0.44	0.32	118.42%	0.40	3.12	18.42%	Reassay
LVR-FS-14-0042	10/09/2014	LVR0002	8343	G912-5	0.41	0.38	0.02	0.40	0.36	0.42	0.34	0.44	0.32	107.89%	0.40	2.84	7.89%	Ok
						0.38	0.02	0.40	0.36	0.42	0.34	0.44	0.32	0.00%	0.40	0.00	-100.00%	
						0.38	0.02	0.40	0.36	0.42	0.34	0.44	0.32	0.00%	0.40	0.00	-100.00%	
						0.38	0.02	0.40	0.36	0.42	0.34	0.44	0.32	0.00%	0.40	0.00	-100.00%	
						0.38	0.02	0.40	0.36	0.42	0.34	0.44	0.32	0.00%	0.40	0.00	-100.00%	
						0.38	0.02	0.40	0.36	0.42	0.34	0.44	0.32	0.00%	0.40	0.00	-100.00%	
				Average grade	0.40													



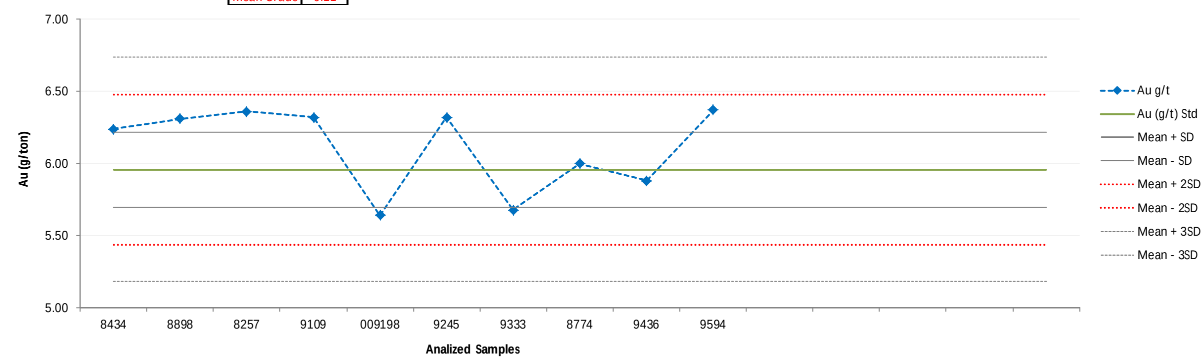
<u>“G905-6” Geostats Standard</u>

The results of “G905-6” Geostats Standard are based on 10 assayed samples. No sample was out of limits, as shown in Figure 12.10.

Figure 12.10 Plot of Au g/t for “G905-6” Geostats Standard

G905-6

Batch	Date	Hole_id	Sample	Standard Id	Au g/t	Au (g/t) Std	STV Dev	Mean + SD	Mean - SD	Mean + 2SD	Mean - 2SD	Mean + 3SD	Mean - 3SD	Au Dev	Au Deviation	Average results	Bias	Status
LVR-FS-14-0043	04/09/2014	LVR0001	8434	G905-6	6.24	5.96	0.26	6.22	5.70	6.48	5.44	6.74	5.18	104.70%	6.11	0.18	4.70%	Ok
LVR-FS-14-0047	09/09/2014	LVR0008	8898	G905-6	6.31	5.96	0.26	6.22	5.70	6.48	5.44	6.74	5.18	105.87%	6.11	0.18	5.87%	Ok
LVR-FS-14-0042	10/09/2014	LVR0002	8257	G905-6	6.36	5.96	0.26	6.22	5.70	6.48	5.44	6.74	5.18	106.71%	6.11	0.18	6.71%	Ok
LVR-FS-14-0050	17/09/2014	LVR0010	9109	G905-6	6.32	5.96	0.26	6.22	5.70	6.48	5.44	6.74	5.18	106.04%	6.11	0.18	6.04%	Ok
LVR-FS-14-0051	19/09/2014	LVR0009	009198	G905-6	5.64	5.96	0.26	6.22	5.70	6.48	5.44	6.74	5.18	94.63%	6.11	0.16	-5.37%	Ok
LVR-FS-14-0051	19/09/2014	LVR0009	9245	G905-6	6.32	5.96	0.26	6.22	5.70	6.48	5.44	6.74	5.18	106.04%	6.11	0.18	6.04%	Ok
LVR-FS-14-0052	21/09/2014	LVR0003	9333	G905-6	5.68	5.96	0.26	6.22	5.70	6.48	5.44	6.74	5.18	95.30%	6.11	0.16	-4.70%	Ok
LVR-FS-14-0047	23/09/2014	LVR0013	8774	G905-6	6.00	5.96	0.26	6.22	5.70	6.48	5.44	6.74	5.18	100.67%	6.11	0.17	0.67%	Ok
LVR-FS-14-0053	24/09/2014	LVR0006	9436	G905-6	5.88	5.96	0.26	6.22	5.70	6.48	5.44	6.74	5.18	98.66%	6.11	0.17	-1.34%	Ok
LVR-FS-14-0054	29/09/2014	LVR0004	9594	G905-6	6.37	5.96	0.26	6.22	5.70	6.48	5.44	6.74	5.18	106.88%	6.11	0.18	6.88%	Ok
						5.96	0.26	6.22	5.70	6.48	5.44	6.74	5.18	0.00%	6.11	0.00	-100.00%	
						5.96	0.26	6.22	5.70	6.48	5.44	6.74	5.18	0.00%	6.11	0.00	-100.00%	
						5.96	0.26	6.22	5.70	6.48	5.44	6.74	5.18	0.00%	6.11	0.00	-100.00%	
						5.96	0.26	6.22	5.70	6.48	5.44	6.74	5.18	0.00%	6.11	0.00	-100.00%	
						5.96	0.26	6.22	5.70	6.48	5.44	6.74	5.18	0.00%	6.11	0.00	-100.00%	
				Mean Grade	6.11													



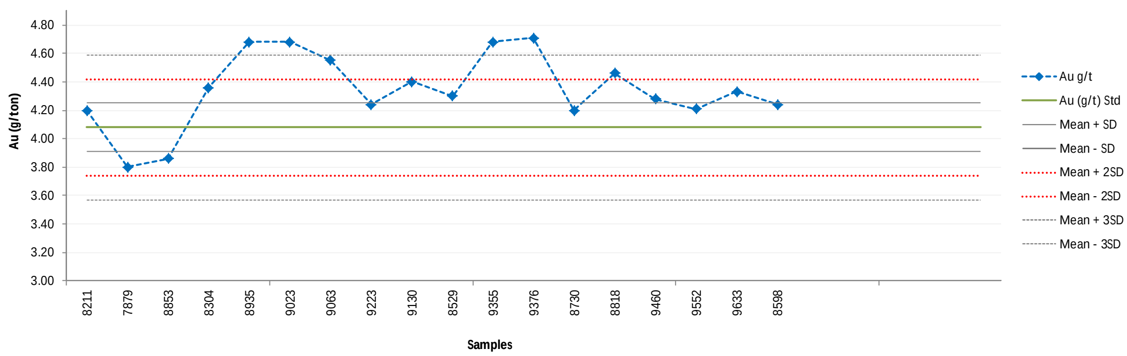
<u>“G912-6” Geostats Standard</u>

The results of “G912-6” Geostats Standard are based on 18 assayed samples. The results indicated a slight bias in grade. It is recommended to check the certification of this standard due to the random values around the second standard deviation (Figure 12.11).

Figure 12.11 Plot of Au g/t for "G912-6" Geostats Standard

G912-6

Batch	Date	Hole_id	Sample	Standard Id	Au g/t	Au (g/t) Std	STV Dev	Mean + SD	Mean - SD	Mean + 2SD	Mean - 2SD	Mean + 3SD	Mean - 3SD	Au Dev	Au Deviation	Average results	Bias	Status
LVR-FS-14-0041	25/08/2014	LVR0070	8211	G912-6	4.20	4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	102.94%	4.34	0.25	2.94%	Ok
LVR-FS-14-0036	08/09/2014	LVR0040	7879	G912-6	3.80	4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	93.14%	4.34	0.23	-6.86%	Ok
LVR-FS-14-0047	08/09/2014	LVR0008	8853	G912-6	3.86	4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	94.61%	4.34	0.23	-5.39%	Ok
LVR-FS-14-0042	10/09/2014	LVR0002	8304	G912-6	4.36	4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	106.86%	4.34	0.26	6.86%	Ok
LVR-FS-14-0048	15/09/2014	LVR0005	8935	G912-6	4.68	4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	114.71%	4.34	0.28	14.71%	Reassay
LVR-FS-14-0048	15/09/2014	LVR0005	9023	G912-6	4.68	4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	114.71%	4.34	0.28	14.71%	Reassay
LVR-FS-14-0050	17/09/2014	LVR0010	9063	G912-6	4.55	4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	111.52%	4.34	0.27	11.52%	Notificar
LVR-FS-14-0051	19/09/2014	LVR0009	9223	G912-6	4.24	4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	103.92%	4.34	0.25	3.92%	Ok
LVR-FS-14-0050	19/09/2014	LVR0010	9130	G912-6	4.40	4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	107.84%	4.34	0.26	7.84%	Ok
LVR-FS-14-0044	21/09/2014	LVR0072	8529	G912-6	4.30	4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	105.39%	4.34	0.26	5.39%	Ok
LVR-FS-14-0052	21/09/2014	LVR0003	9355	G912-6	4.68	4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	114.71%	4.34	0.28	14.71%	Reassay
LVR-FS-14-0052	21/09/2014	LVR0003	9376	G912-6	4.71	4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	115.44%	4.34	0.28	15.44%	Reassay
LVR-FS-14-0047	23/09/2014	LVR0013	8730	G912-6	4.20	4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	102.94%	4.34	0.25	2.94%	Ok
LVR-FS-14-0047	23/09/2014	LVR0013	8818	G912-6	4.46	4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	109.31%	4.34	0.27	9.31%	Notificar
LVR-FS-14-0053	24/09/2014	LVR0006	9460	G912-6	4.28	4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	104.90%	4.34	0.26	4.90%	Ok
LVR-FS-14-0054	26/09/2014	LVR0004	9552	G912-6	4.21	4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	103.19%	4.34	0.25	3.19%	Ok
LVR-FS-14-0055	29/09/2014	LVR0014	9633	G912-6	4.33	4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	106.13%	4.34	0.26	6.13%	Ok
LVR-FS-14-0045	30/09/2014	LVR0073	8598	G912-6	4.24	4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	103.92%	4.34	0.25	3.92%	Ok
						4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	0.00%	4.34	0.00	-100.00%	
						4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	0.00%	4.34	0.00	-100.00%	
						4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	0.00%	4.34	0.00	-100.00%	
						4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	0.00%	4.34	0.00	-100.00%	
						4.08	0.17	4.25	3.91	4.42	3.74	4.59	3.57	0.00%	4.34	0.00	-100.00%	
				Average grade	4.34													



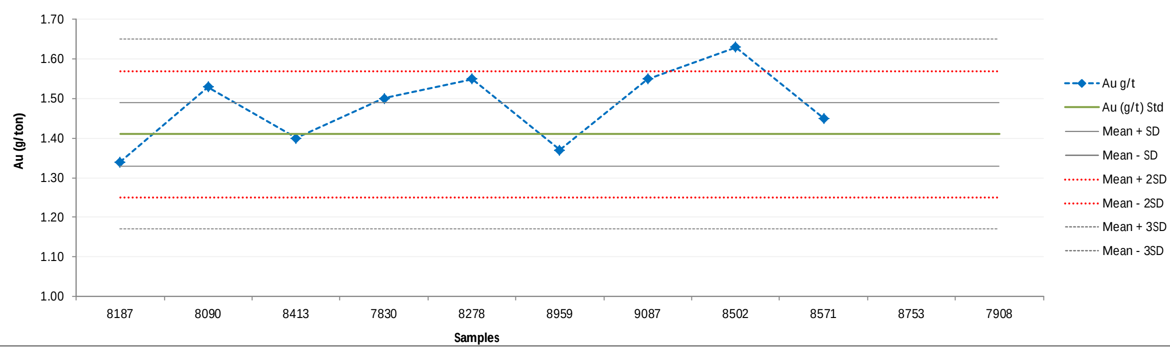
<u>"G997-3" Geostats Standard</u>

The results of "G997-3" Geostats Standard are based on 9 assayed samples, and one sample fell between the second and third standard deviation limits, but was not re-assayed. The results are shown in Figure 12.12.

Figure 12.12 Plot of Au g/t for "G997-3" Geostats Standard

G997-3

Batch	Date	Hole_id	Sample	Standard Id	Au g/t	Au (g/t) Std	STV Dev	Mean + SD	Mean - SD	Mean + 2SD	Mean - 2SD	Mean + 3SD	Mean - 3SD	Au Dev	Au Deviation	Average results	Bias	Status
LVR-FS-14-0041	25/08/2014	LVR0070	8187	G997-3	1.34	1.41	0.08	1.49	1.33	1.57	1.25	1.65	1.17	95.04%	1.25	0.67	-4.96%	Ok
LVR-FS-14-0039	02/09/2014	LVR0071	8090	G997-3	1.53	1.41	0.08	1.49	1.33	1.57	1.25	1.65	1.17	108.51%	1.25	0.77	8.51%	Ok
LVR-FS-14-0043	04/09/2014	LVR0001	8413	G997-3	1.40	1.41	0.08	1.49	1.33	1.57	1.25	1.65	1.17	99.29%	1.25	0.70	-0.71%	Ok
LVR-FS-14-0036	08/09/2014	LVR0040	7830	G997-3	1.50	1.41	0.08	1.49	1.33	1.57	1.25	1.65	1.17	106.38%	1.25	0.75	6.38%	Ok
LVR-FS-14-0042	10/09/2014	LVR0002	8278	G997-3	1.55	1.41	0.08	1.49	1.33	1.57	1.25	1.65	1.17	109.93%	1.25	0.78	9.93%	Ok
LVR-FS-14-0048	15/09/2014	LVR0005	8959	G997-3	1.37	1.41	0.08	1.49	1.33	1.57	1.25	1.65	1.17	97.16%	1.25	0.69	-2.84%	Ok
LVR-FS-14-0050	17/09/2014	LVR0010	9087	G997-3	1.55	1.41	0.08	1.49	1.33	1.57	1.25	1.65	1.17	109.93%	1.25	0.78	9.93%	Ok
LVR-FS-14-0044	21/09/2014	LVR0072	8502	G997-3	1.63	1.41	0.08	1.49	1.33	1.57	1.25	1.65	1.17	115.60%	1.25	0.82	15.60%	Notificar
LVR-FS-14-0045	29/09/2014	LVR0073	8571	G997-3	1.45	1.41	0.08	1.49	1.33	1.57	1.25	1.65	1.17	102.84%	1.25	0.73	2.84%	Ok
		LVR0013	8753	G997-3		1.41	0.08	1.49	1.33	1.57	1.25	1.65	1.17	0.00%	1.25	0.00	-100.00%	
		LVR0036	7908	G997-3		1.41	0.08	1.49	1.33	1.57	1.25	1.65	1.17	0.00%	1.25	0.00	-100.00%	
				Average grade	1.48													

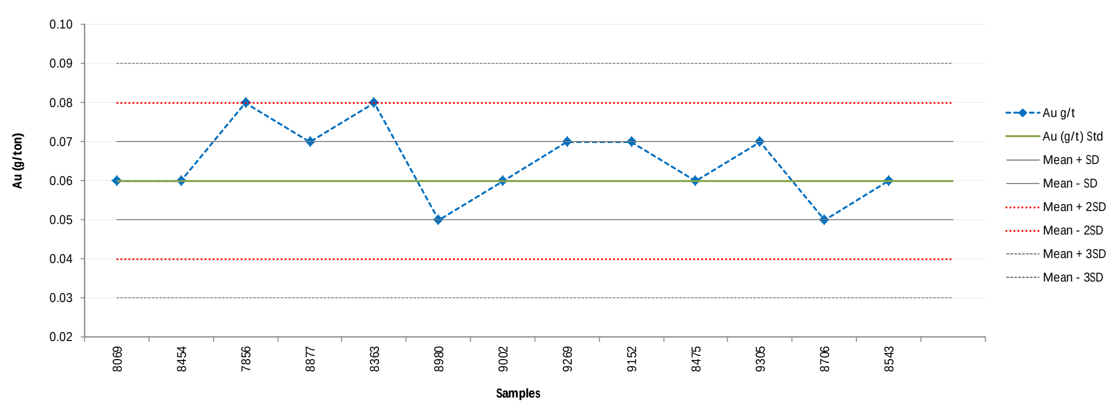


"G908-1" Geostats Standard

The results of "G908-1" Geostats Standard are based on 13 assayed samples. No standard was out of limits as shown in Figure 12.13.

Figure 12.13 Plot of Au g/t for "G908-1" Geostats Standard

Batch	Date	Hole_id	Sample	Standard Id	Au g/t	Au (g/t) Std	STV Dev	Mean + SD	Mean - SD	Mean + 2SD	Mean - 2SD	Mean + 3SD	Mean - 3SD	Au Dev	Au Deviation	Average results	Bias	Status
LVR-FS-14-0039	02/09/2014	LVR0071	8069	G908-1	0.06	0.06	0.01	0.07	0.05	0.08	0.04	0.09	0.03	100.00%	0.07	16.67	0.00%	Ok
LVR-FS-14-0043	04/09/2014	LVR0001	8454	G908-1	0.06	0.06	0.01	0.07	0.05	0.08	0.04	0.09	0.03	100.00%	0.07	16.67	0.00%	Ok
LVR-FS-14-0036	08/09/2014	LVR0040	7856	G908-1	0.08	0.06	0.01	0.07	0.05	0.08	0.04	0.09	0.03	133.33%	0.07	22.22	33.33%	Ok
LVR-FS-14-0047	09/09/2014	LVR0008	8877	G908-1	0.07	0.06	0.01	0.07	0.05	0.08	0.04	0.09	0.03	116.67%	0.07	19.44	16.67%	Ok
LVR-FS-14-0042	10/09/2014	LVR0002	8363	G908-1	0.08	0.06	0.01	0.07	0.05	0.08	0.04	0.09	0.03	133.33%	0.07	22.22	33.33%	Ok
LVR-FS-14-0048	15/09/2014	LVR0005	8980	G908-1	0.05	0.06	0.01	0.07	0.05	0.08	0.04	0.09	0.03	83.33%	0.07	13.89	-16.67%	Ok
LVR-FS-14-0048	15/09/2014	LVR0005	9002	G908-1	0.06	0.06	0.01	0.07	0.05	0.08	0.04	0.09	0.03	100.00%	0.07	16.67	0.00%	Ok
LVR-FS-14-0051	15/09/2014	LVR0009	9269	G908-1	0.07	0.06	0.01	0.07	0.05	0.08	0.04	0.09	0.03	116.67%	0.07	19.44	16.67%	Ok
LVR-FS-14-0050	17/09/2014	LVR0010	9152	G908-1	0.07	0.06	0.01	0.07	0.05	0.08	0.04	0.09	0.03	116.67%	0.07	19.44	16.67%	Ok
LVR-FS-14-0044	21/09/2014	LVR0072	8475	G908-1	0.06	0.06	0.01	0.07	0.05	0.08	0.04	0.09	0.03	100.00%	0.07	16.67	0.00%	Ok
LVR-FS-14-0052	21/09/2014	LVR0003	9305	G908-1	0.07	0.06	0.01	0.07	0.05	0.08	0.04	0.09	0.03	116.67%	0.07	19.44	16.67%	Ok
LVR-FS-14-0047	23/09/2014	LVR0013	8706	G908-1	0.05	0.06	0.01	0.07	0.05	0.08	0.04	0.09	0.03	83.33%	0.07	13.89	-16.67%	Ok
LVR-FS-14-0045	29/09/2014	LVR0073	8543	G908-1	0.06	0.06	0.01	0.07	0.05	0.08	0.04	0.09	0.03	100.00%	0.07	16.67	0.00%	Ok
				G908-2		0.06	0.01	0.07	0.05	0.08	0.04	0.09	0.03	0.00%	0.07	0.00	-100.00%	
				Average grade	0.06													

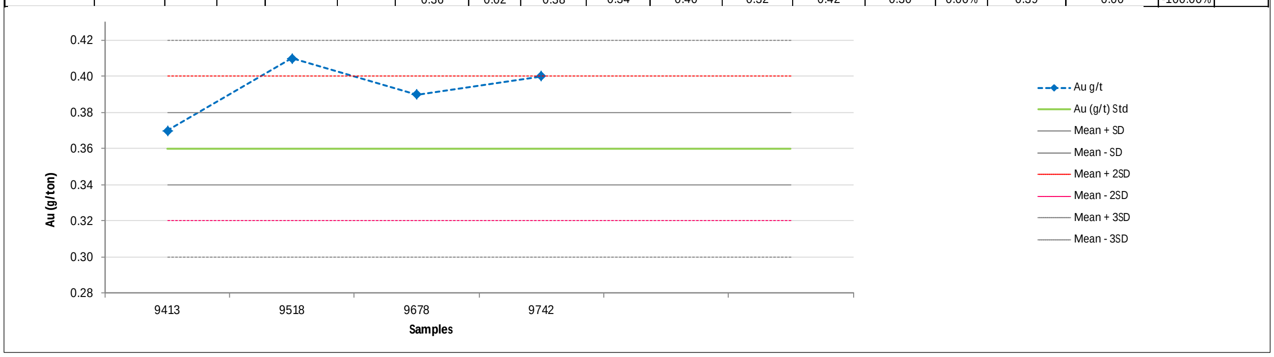


“G904-6 (G311-7)” Geostats Standard

The results of “G904-6 (G311-7)” Geostats Standard are based on 4 assayed samples, and all were above the mean grade of the standard, but only one was above the second standard deviation. The results are shown in Figure 12.14.

Figure 12.14 Plot of Au g/t for “G904-6 (G311-7)” Geostats Standard

Batch	Date	Hole_id	Sample	Standard Id	Au g/t	Au (g/t) Std	STV Dev	Mean + SD	Mean - SD	Mean + 2SD	Mean - 2SD	Mean + 3SD	Mean - 3SD	Au Dev	Au Deviation	Average results	Bias	Status
LVR-FS-14-0053	23/09/2014	LVR0006	9413	G311-7	0.37	0.36	0.02	0.38	0.34	0.40	0.32	0.42	0.30	102.78%	0.39	2.85	2.78%	Ok
LVR-FS-14-0054	26/09/2014	LVR0004	9518	G311-7	0.41	0.36	0.02	0.38	0.34	0.40	0.32	0.42	0.30	113.89%	0.39	3.16	13.89%	Notificar
LVR-FS-14-0054	26/09/2014	LVR0014	9678	G311-7	0.39	0.36	0.02	0.38	0.34	0.40	0.32	0.42	0.30	108.33%	0.39	3.01	8.33%	Ok
LVR-FS-14-0054	26/09/2014	LVR0012	9742	G311-7	0.40	0.36	0.02	0.38	0.34	0.40	0.32	0.42	0.30	111.11%	0.39	3.09	11.11%	Ok
						0.36	0.02	0.38	0.34	0.40	0.32	0.42	0.30	0.00%	0.39	0.00	-100.00%	
						0.36	0.02	0.38	0.34	0.40	0.32	0.42	0.30	0.00%	0.39	0.00	-100.00%	



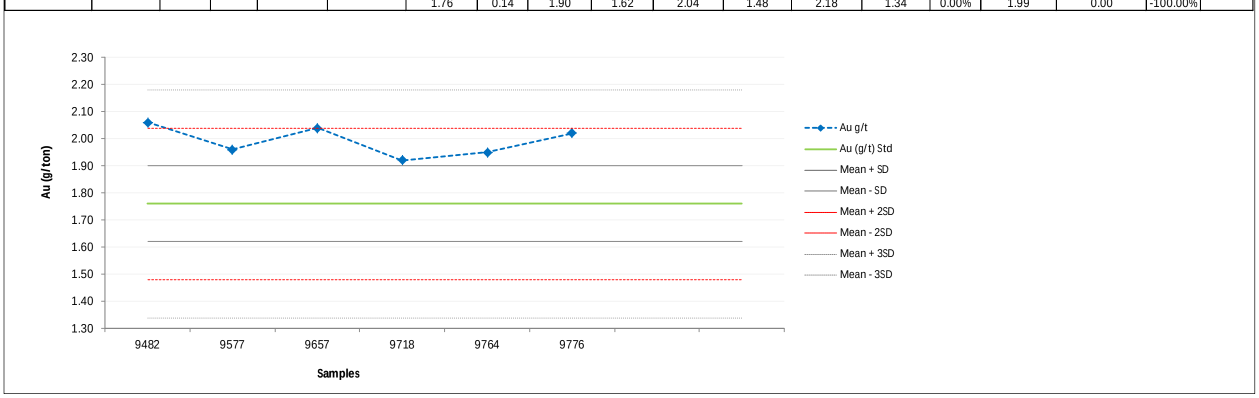
“G901-2 (G909-2)” Geostats Standard

The results of “G901-2 (G909-2)” Geostats Standard are based on 6 assayed samples, and all were above the mean grade of the standard, but only one was above the second standard deviation. The results are shown in Figure 12.15.

Figure 12.15 Plot of Au g/t for “G901-2 (G909-2)” Geostats Standard

G901-2

Batch	Date	Hole_id	Sample	Standard Id	Au g/t	Au (g/t) Std	STV Dev	Mean + SD	Mean - SD	Mean + 2SD	Mean - 2SD	Mean + 3SD	Mean - 3SD	Au Dev	Au Deviation	Average results	Bias	Status
LVR-FS-14-0053	24/09/2014	LVR0006	9482	G909-2	2.06	1.76	0.14	1.90	1.62	2.04	1.48	2.18	1.34	117.05%	1.99	0.67	17.05%	Notificar
LVR-FS-14-0054	26/09/2014	LVR0004	9577	G909-2	1.96	1.76	0.14	1.90	1.62	2.04	1.48	2.18	1.34	111.36%	1.99	0.63	11.36%	Ok
LVR-FS-14-0054	26/09/2014	LVR0014	9657	G909-2	2.04	1.76	0.14	1.90	1.62	2.04	1.48	2.18	1.34	115.91%	1.99	0.66	15.91%	Ok
LVR-FS-14-0056	29/09/2014	LVR0012	9718	G909-2	1.92	1.76	0.14	1.90	1.62	2.04	1.48	2.18	1.34	109.09%	1.99	0.62	9.09%	Ok
LVR-FS-14-0056	29/09/2014	LVR0012	9764	G909-2	1.95	1.76	0.14	1.90	1.62	2.04	1.48	2.18	1.34	110.80%	1.99	0.63	10.80%	Ok
LVR-FS-14-0056	29/09/2014	LVR0012	9776	G909-2	2.02	1.76	0.14	1.90	1.62	2.04	1.48	2.18	1.34	114.77%	1.99	0.65	14.77%	Ok
						1.76	0.14	1.90	1.62	2.04	1.48	2.18	1.34	0.00%	1.99	0.00	-100.00%	
						1.76	0.14	1.90	1.62	2.04	1.48	2.18	1.34	0.00%	1.99	0.00	-100.00%	



12.4.5 Analytical Precision Analysis – Duplicates

For the purpose of laboratory reproducibility assessment and transparency of results, duplicates were inserted randomly in each batch sample. This procedure gives a possibility to compare a pair of assay samples and correlate the results.

The dataset had a total of 325 field duplicates which were blinded for the lab and 242 pulp duplicates not blinded for the lab. Field duplicates showed poor precision with a mean difference of the duplicates around 15% while the pulp duplicates had excellent precision with a mean difference less than 2% (Table 12.3).

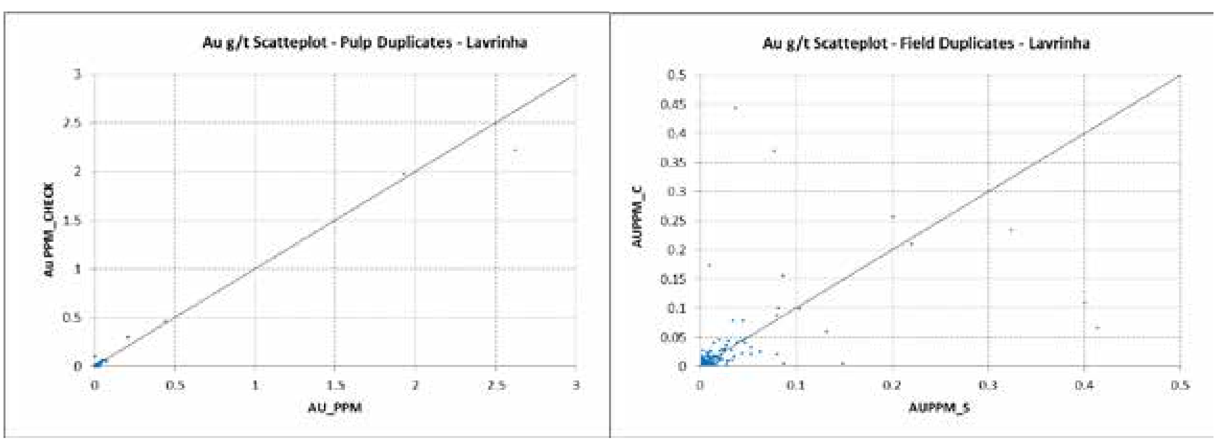
TABLE 12.3
SUMMARY STATISTICS OF PULP AND FIELD DUPLICATES

Summary Statistics - Pulp Duplicates			
Statistics	AU_PPM	Au PPM_CHECK	Difference
Total Samples	242	242	0%
Mean	0.028	0.028	-1.87%
Std Dev	0.210	0.193	-8.13%
Variance	0.044	0.037	-15.60%
Minimum	0.003	0.003	0.00%
Lower Quantile	0.003	0.003	0.00%
Median	0.003	0.003	0.00%
Upper Quantile	0.006	0.006	0.00%
Maximum	2.620	2.222	-15.19%
Correlation	**99%**		

Summary Statistics - Field Duplicates			
Statistics	AUPPM_S	AUPPM_C	Difference
Total Samples	325	325	0%
Mean	0.033	0.028	-16.7%
Std Dev	0.221	0.196	-11.4%
Variance	0.049	0.038	-21.5%
Minimum	0.003	0.003	0.0%
Lower Quantile	0.003	0.003	0.0%
Median	0.003	0.003	0.0%
Upper Quantile	0.007	0.007	0.0%
Maximum	3.240	3.230	-0.3%
Correlation	**76%**		

The difference presented in the statistical summary is more clearly shown by the data dispersion in the scatter plots in Figure 12.16.

Figure 12.16 Scatter plots of Field and Pulp duplicate samples



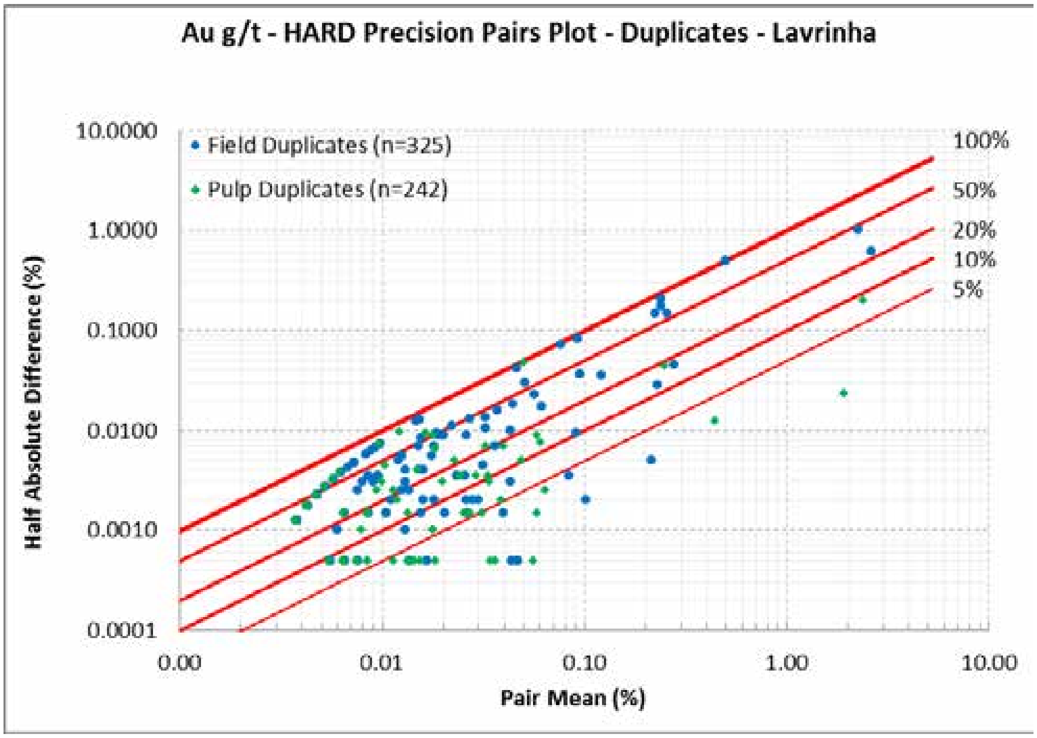
12.4.6 Precision Pair Plot (HAD) Analysis

The HAD (precision pairs plots) graphics give an idea about the precision of the assay. The formula to calculate the HAD value for a pair of samples is:

$$HAD = \frac{|X_{orig} - X_{dup}|}{2}$$

Figure 12.17 presents the precision pair plots for field and pulp duplicates and shows poor precision which is related to gold deposits with a high variability of grade and erratic distribution of free gold particles.

Figure 12.17 Precision Pairs Plots for Pulp and Field Duplicates



12.4.7 Relative Difference Plot Analysis

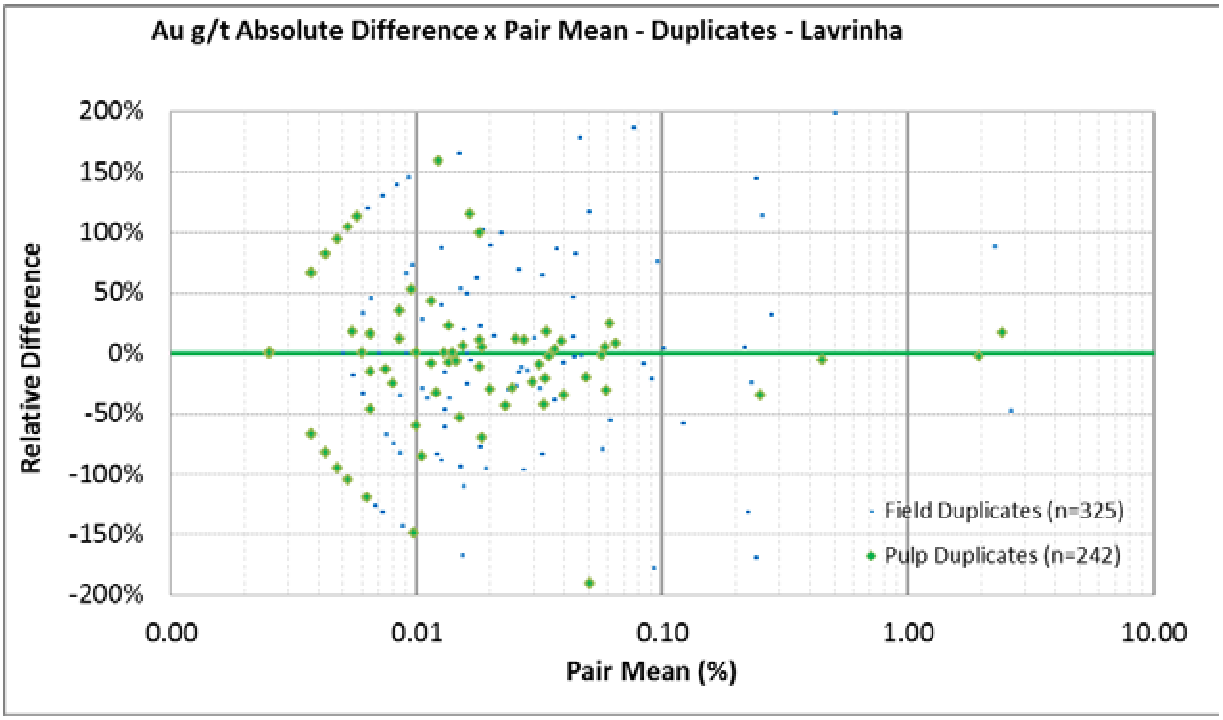
The relative difference plot was used to assess bias in duplicate data and to identify individual samples with poor precision that didn't appear in the HAD graphics. The graph is constructed by plotting the pair mean against the relative difference. The formula to calculate the relative difference is:

$$Relative\ Difference = \frac{X_{orig} - X_{dup}}{pair\ mean}$$

In the graph, if no bias exists then the differences will be symmetrical on either side of the zero line and form a "tunnel" shape, with larger relative differences at lower grades (close to the detection limit) where precision is typically poor.

Figure 12.18 shows that the relative difference of pairs has no significant bias but also confirmed the poor precision of the dataset.

Figure 12.18 Relative Difference Plots for Au g/t for Pulp and Field Duplicates

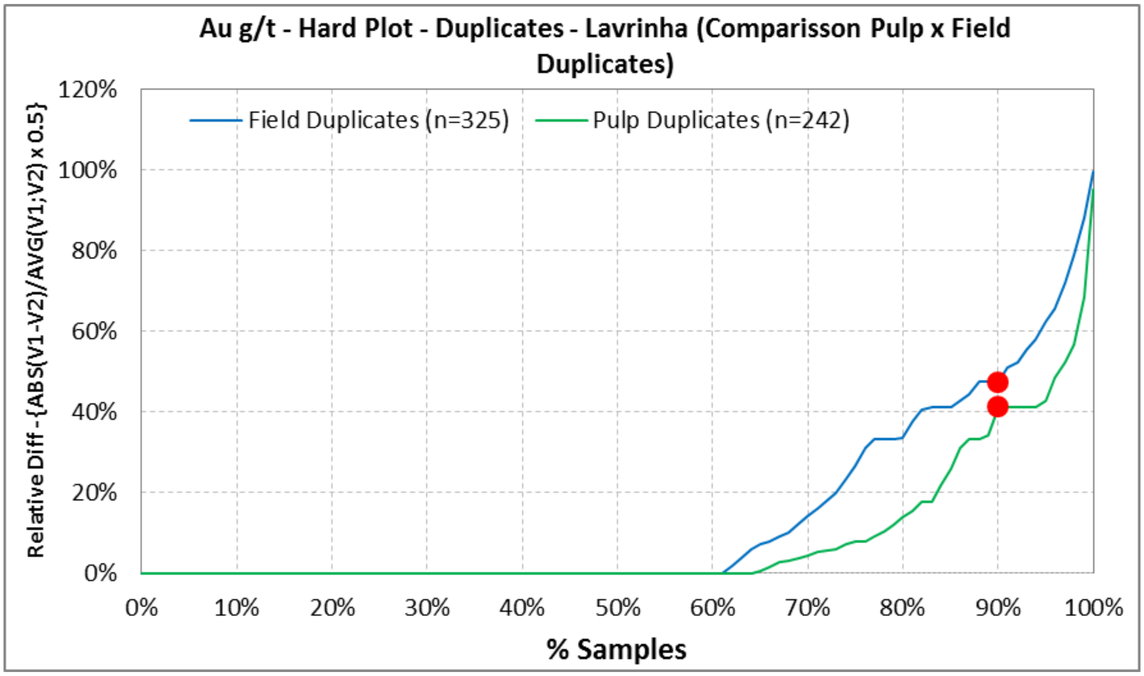


12.4.8 HARD Plot Analysis

A ranked HARD Plot is a pair analysis of duplicates which ranks all assay pairs in terms of precision levels measured as half of the absolute relative difference from the mean of the assay pairs ("HARD"), used to visualize relative precision levels (typically 90%) and to determine the percentage of the assay pair population occurring at a certain precision level. It should be noted that as the HARD statistic uses and absolute difference, a ranked HARD plot does not reveal bias in duplicate data, only the relative magnitude of differences (i.e precision). The HARD values are sorted from lowest to highest and ranked accordingly; with the rank expressed as a percentage. The ranked HARD plot is then generated by plotting the percent rank on the X-axis against the HARD value.

Figure 12.19 shows a Hard Plot comparative of Field and Pulp duplicates for Au (g/t). The points at 90% have a precision around 40% HARD, which is relatively good and confirms better precision for pulps (fine particles have more homogeneity) compared to field duplicates.

Figure 12.19 Ranked HARD plot showing the variables precision for Pulp and Field Duplicates



12.4.9 Drilling Campaigns 2015 (Analyzed by Aura)

The following descriptions are mostly excerpted from reports by Aura´s database management team Sheila Ulansky, P.Geo., in Canada and Gleidson D. Santos in Brazil.

24 holes were drilled on the Lavrinha Property in 2015 (PR-01 to PR-24). All of the core samples, including standards, duplicates and blanks, were processed (preparation and analytical testing) at the Sao Francisco mine assay laboratory between September and November 2015.

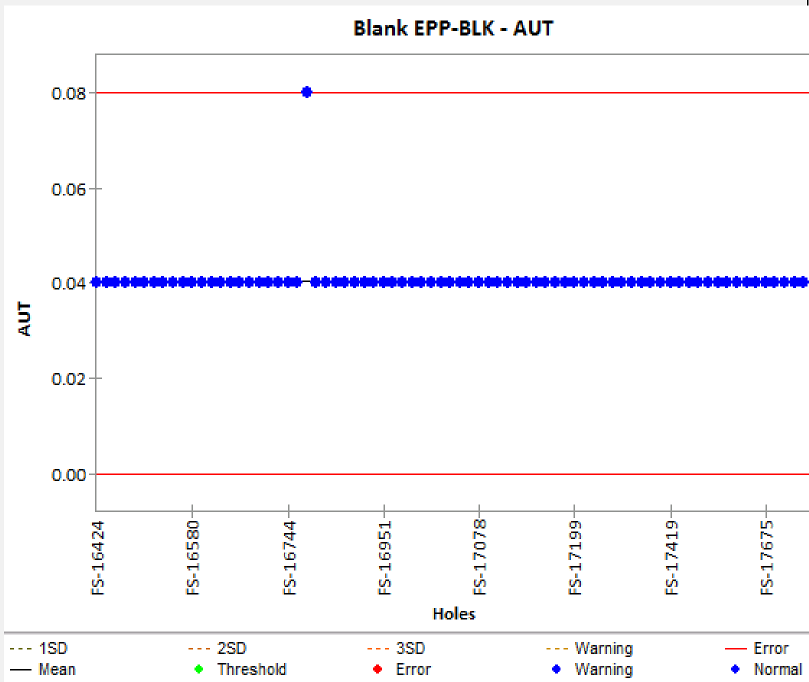
The drill campaign samples were sent to SGS (ISO certified laboratory) to check the consistency of the São Francisco laboratory ("SF lab"). There were 73 samples selected surrounding the mineralized intervals defined by logging and SF Analysis and sent as duplicates. 9 blanks and 8 standards were inserted into this batch.

12.4.10 Contamination Evaluation – Blank Standards

A total of 80 blank samples were inserted into the sample stream. The results are showed in Figure 12.20. Most of the results were within the detection limit of the SF lab. Only one sample was above the detection limit but was inside of the error limit.

Figure 12.20 QA/QC Blanks – 2015 Drilling Campaign

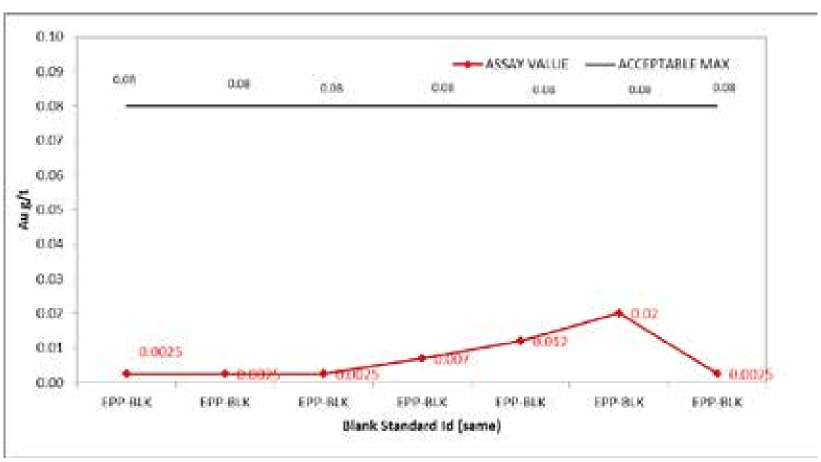
wi



# of Anlys above Threshold	80
# Outside Warning Limit	0
# Outside Error Limit	0
# of Anlys below Threshold	0
% Outside Error Limit	0
Mean	0.0405
Median	0.04
Min	0.04
Max	0.08
Standard Deviation	0.0045
% Rel. Std. Dev.	11.0423
Standard Error	0.0005
% Rel. Std. Err.	1.2346
Total Bias	n/a
% Mean Bias	n/a

Nine blank standards were sent to the SGS lab, related to duplicates. No sample above the threshold was identified to indicate any contamination (Figure 12.21).

Figure 12.21 QA/QC Blanks – 2015 Drilling Campaign – Samples analyzed at SF Lab and re-sent to SGS



12.4.11 Accuracy Analysis – Analytical Standards

A total of 79 standards aliquots (Figure 12.22 and Table 12.4) from seven certified materials (Table 12.5) were inserted into the sample batches sent to the laboratories. The results were analyzed based on control charts and statistical analysis considering as acceptable the 3rd standard deviation of Au g/t limits per certified standard.

Figure 12.22 QA/QC Standards – All standards by sequence

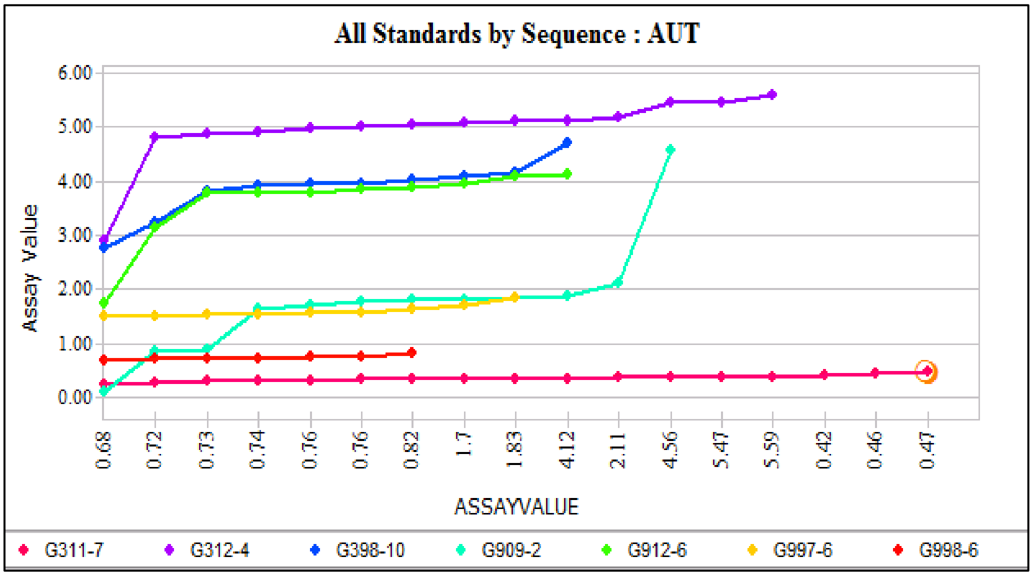


TABLE 12.4 STATISTICS OF ALL STANDARDS BY SEQUENCE							
AUT	**G311-7**	**G312-4**	**G398-10**	**G909-2**	**G912-6**	**G997-6**	**G998-6**
# of Analys above Threshold	17	14	10	12	10	9	7
# Outside Warning Limit	6	2	3	7	2	2	0
# Outside Error Limit	3	1	3	5	2	0	0
# of Anlys below Threshold	0	0	0	0	0	0	0
% Outside Error Limit	17.65	7.14	30.00	41.67	20.00	0.00	0.00
Mean	0.36	4.97	3.86	1.75	3.61	1.60	0.74
Median	0.36	5.07	3.96	1.79	3.83	1.57	0.74
Min	0.24	2.90	2.76	0.11	1.74	1.50	0.68
Max	0.47	5.59	4.72	4.56	4.12	1.83	0.82
Standard Deviation	0.06	0.64	0.53	1.06	0.71	0.11	0.04
% Rel. Std. Dev.	16.63	12.85	13.71	60.46	19.72	6.77	5.80
Coeff. of Var.	0.17	0.13	0.14	0.60	0.20	0.07	0.06
Standard Error	0.01	0.17	0.17	0.31	0.23	0.04	0.02
% Rel. Std. Err.	4.03	3.43	4.34	17.45	6.23	2.26	2.19
Total Bias	-0.10	-0.06	-0.05	-0.10	-0.11	-0.05	-0.07
% Mean Bias	-10.29	-6.32	-5.06	-9.84	-11.42	-4.83	-6.96

TABLE 12.5 CERTIFICATES OF THE REFERENCE MATERIAL				
Std	**Au Grade**	**Deviation**	**-3sd**	**+3sd**
G311-7	0.4	0.03	0.31	0.49
G312-4	5.3	0.22	4.64	5.96
G998-6	0.8	0.06	0.62	0.98
G997-6	1.68	0.08	1.44	1.92
G909-2	1.94	0.08	1.7	2.18
G398-10	4.07	0.19	3.5	4.64
G912-6	4.08	0.17	3.57	4.59

The results from the standards were mainly consistent with the expected values, although there were several standards that failed to fit within the 3rd standard deviation limits. The standards were analyzed together and if two or more standards within one certificate failed, then the decision was made to retest the entire certificate. Only one certificate (LVR_FS_15_0038) from drill hole PR-07 fits this criteria and was sent to the lab for re-assay. It still returned with several of the certified reference material sample values out of the accepted 3rd standard deviation limit.

In order to determine the best approach forward for accessing the failed standards the following points were considered:

- What standard was used and from where did it originate?
- What method was used to process the standard?
- What grade was the standard?

The standards were derived from a variety of host rocks in oxide or sulphide gold mineralization systems which are different but comparable with the Lavrinha Deposit. The standards were processed using a fire-assay digest, the same method which was used at the Sao Francisco lab.

The grades of the standards are within the observed gold range at Lavrinha although G311-7 is below the minimum cut-off grade applicable for the mineral resource. Therefore, samples that did not meet the 3rd standard deviation limit for G311-7 were not considered for re-assay. Table 12.6 shows list of the 9 standards that did not pass the 3rd standard deviation threshold.

TABLE 12.6
FAILED STANDARDS (EXCLUDING G311-7)

HOLE ID	Despatch ID	Standard ID	Assay Value	Standard Value	Standard deviation	Acceptable Min	Acceptable Max
PR-07	LVR_FS_15 _0038	G909-2	0.9	1.94	0.08	1.7	2.18
PR-07	LVR_FS_15 _0038	G912-6	1.74	4.08	0.17	3.57	4.59
PR-08	LVR_FS_15 _0037	G909-2	0.85	1.94	0.08	1.7	2.18
PR-09	LVR_FS_15 _0036	G398-10	3.24	4.07	0.19	3.5	4.64
PR-09	LVR_FS_15 _0036	G912-6	3.13	4.08	0.17	3.57	4.59
PR-11	LVR_FS_15 _0033	G398-10	2.76	4.07	0.19	3.5	4.64
PR-13	LVR_FS_15 _0035	G909-2	1.63	1.94	0.08	1.7	2.18
PR-14	LVR_FS_15 _0041	G909-2	4.56	1.94	0.08	1.7	2.18
PR-22	LVR_FS_15 _0045	G909-2	0.11	1.94	0.08	1.7	2.18

Table 12.7 shows the samples that were outside the 3rd standard deviation and the standard deviation needed for assay values to be acceptable.

TABLE 12.7
DRILL HOLES WITH STANDARDS OUTSIDE 3SD RANGE

HOLEID	G311-7	G312-4	G398-10	G909-2	G912-6	G997-6	G998-6	Comments
PR-01								
PR-02								
PR-03								
PR-04								
PR-05								
PR-06	FS-17286 5.4 SD	FS-17275 10.5 SD						Not acceptable ?
PR-07				FS-17235 13 SD	FS-17195 - 14 SD			
PR-08				FS-17163 14 SD				
PR-09			FS-17138 4.4 SD		FS-17125 5.6 SD			Not acceptable ?
PR-10								
PR-11			FS-16947 7 SD					Not acceptable ?
PR-12								
PR-13				FS-17087 4 SD				Acceptable ?
PR-14				FS-17414 32 SD				Not acceptable ?
PR-15								
PR-16								
PR-17	FS-17036 3.4 SD							Acceptable ?
PR-18								
PR-19								
PR-21								
PR-22	FS-17615 4.3 SD			FS-17591 23 SD				Not acceptable ?
PR-23			FS-17751 3.4 SD					Acceptable ?
PR-24								

In a batch of samples there are often one or two standards that do not meet the certified acceptable range. Although one-off sample cases can be overlooked, it does not raise a red flag to the accuracy and precision of the analytical laboratory.

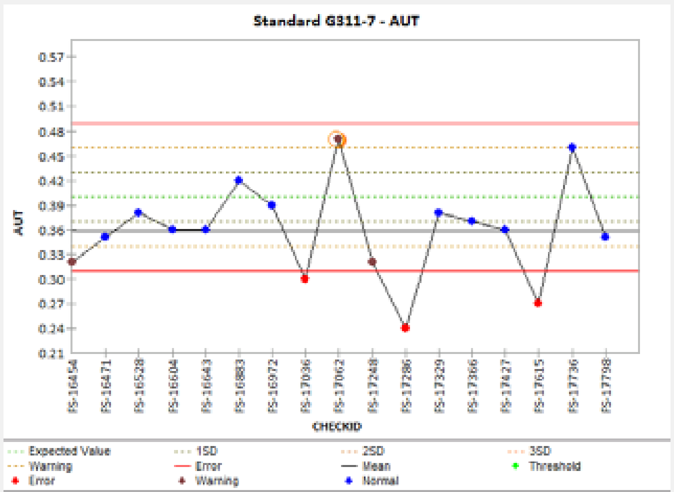
The standards as a whole should be considered together and if there is evidence of high or low bias or trends in the data, then the certificates should be re-assayed. Also, if there are obvious accuracy and/or precision errors then the whole certificate will need to be re-tested. Using this premise it was decided that if two or more standards per certificate failed then the specific certificate would be re-tested.

For all of the certificates, except one, there was no obvious trend or accuracy/precision error. The only certificate to have two failed standards >0.5 g/t Au was LVR_FS_15_0038 (PR-07).

It was noted that the other certificates have a small degree of bias (towards the lower grades). At this point it is not of concern but it should be monitored in the future. This small measure of low grade bias means that the block grades may also be interpolated lower locally.

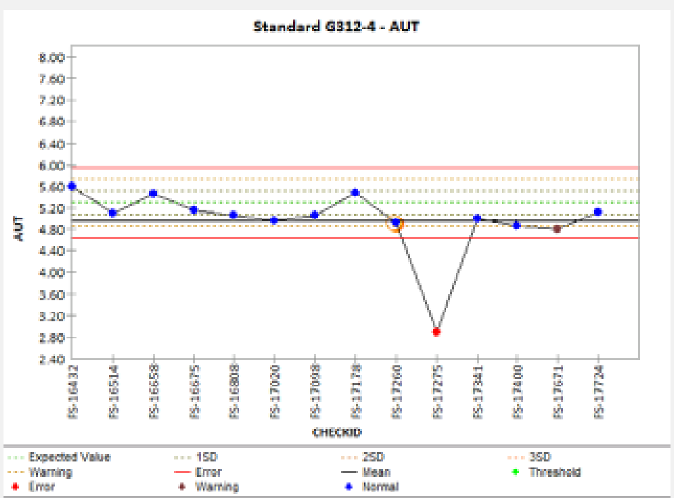
Figure 12.23 to Figure 12.29 present the control charts for the standard samples inserted into the samples batches.

Figure 12.23 QA/QC Standards – G311-7



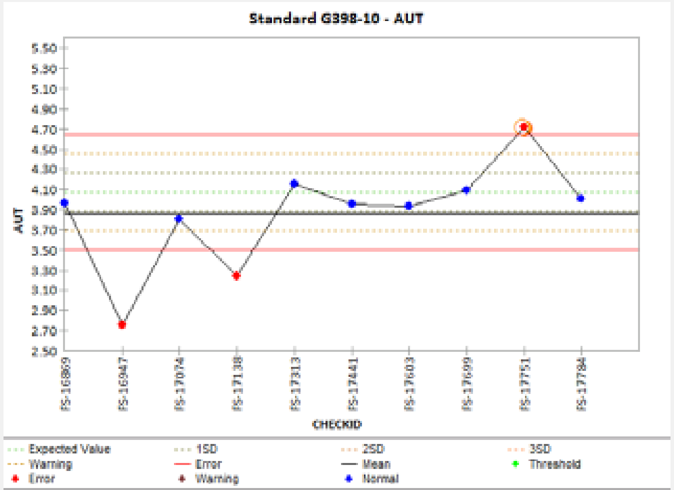
# of Anlys above Threshold	17
# Outside Warning Limit	6
# Outside Error Limit	3
# of Anlys below Threshold	0
% Outside Error Limit	17.6471
Mean	0.3588
Median	0.36
Min	0.24
Max	0.47
Standard Deviation	0.0597
% Rel. Std. Dev.	16.6306
Standard Error	0.0145
% Rel. Std. Err.	4.0335
Total Bias	-0.1029
% Mean Bias	-10.2941

Figure 12.24 QA/QC Standards – G312-4



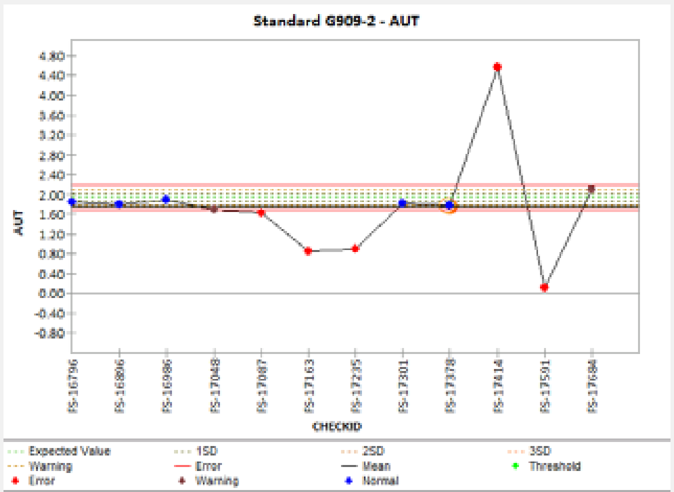
# of Anlys above Threshold	14
# Outside Warning Limit	2
# Outside Error Limit	1
# of Anlys below Threshold	0
% Outside Error Limit	7.1429
Mean	4.965
Median	5.065
Min	2.9
Max	5.59
Standard Deviation	0.6381
% Rel. Std. Dev.	12.8516
Standard Error	0.1705
% Rel. Std. Err.	3.4347
Total Bias	-0.0632
% Mean Bias	-6.3208

Figure 12.25 QA/QC Standards – G398-10



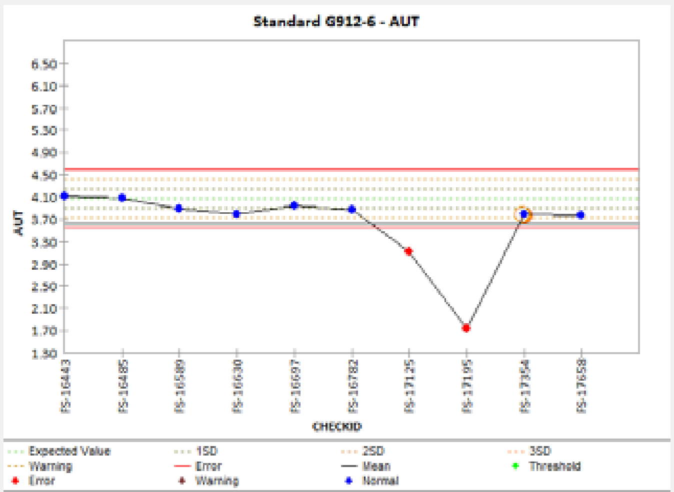
# of Anlys above Threshold	10
# Outside Warning Limit	3
# Outside Error Limit	3
# of Anlys below Threshold	0
% Outside Error Limit	30
Mean	3.864
Median	3.96
Min	2.76
Max	4.72
Standard Deviation	0.5299
% Rel. Std. Dev.	13.7129
Standard Error	0.1676
% Rel. Std. Err.	4.3364
Total Bias	-0.0506
% Mean Bias	-5.0614

Figure 12.26 QA/QC Standards – G909-2



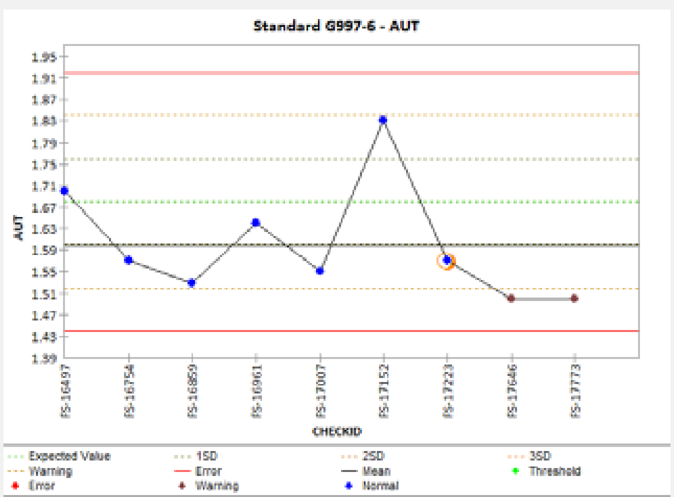
# of Anlys above Threshold	12
# Outside Warning Limit	7
# Outside Error Limit	5
# of Anlys below Threshold	0
% Outside Error Limit	41.6667
Mean	1.7492
Median	1.79
Min	0.11
Max	4.56
Standard Deviation	1.0575
% Rel. Std. Dev.	60.4572
Standard Error	0.3053
% Rel. Std. Err.	17.4525
Total Bias	-0.0984
% Mean Bias	-9.8368

Figure 12.27 QA/QC Standards – 912-6



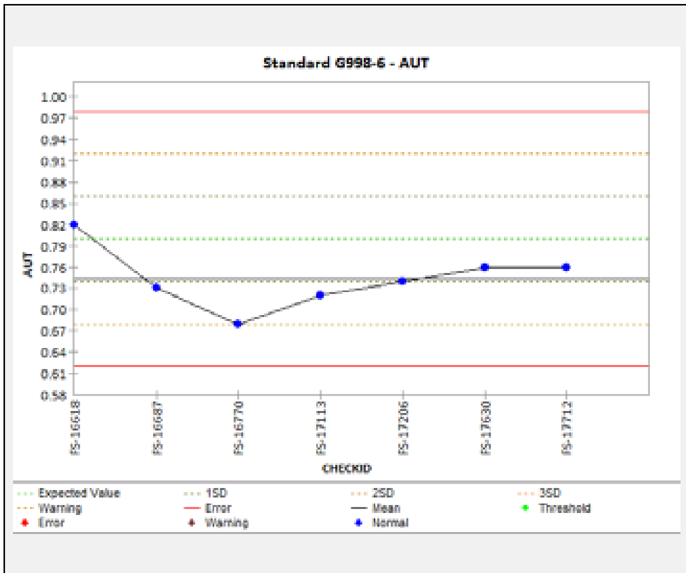
# of Anlys above Threshold	10
# Outside Warning Limit	2
# Outside Error Limit	2
# of Anlys below Threshold	0
% Outside Error Limit	20
Mean	3.614
Median	3.83
Min	1.74
Max	4.12
Standard Deviation	0.7126
% Rel. Std. Dev.	19.7166
Standard Error	0.2253
% Rel. Std. Err.	6.2349
Total Bias	-0.1142
% Mean Bias	-11.4216

Figure 12.28 QA/QC Standards – G997-6



# of Anlys above Threshold	9
# Outside Warning Limit	2
# Outside Error Limit	0
# of Anlys below Threshold	0
% Outside Error Limit	0
Mean	1.5989
Median	1.57
Min	1.5
Max	1.83
Standard Deviation	0.1082
% Rel. Std. Dev.	6.7683
Standard Error	0.0361
% Rel. Std. Err.	2.2561
Total Bias	-0.0483
% Mean Bias	-4.828

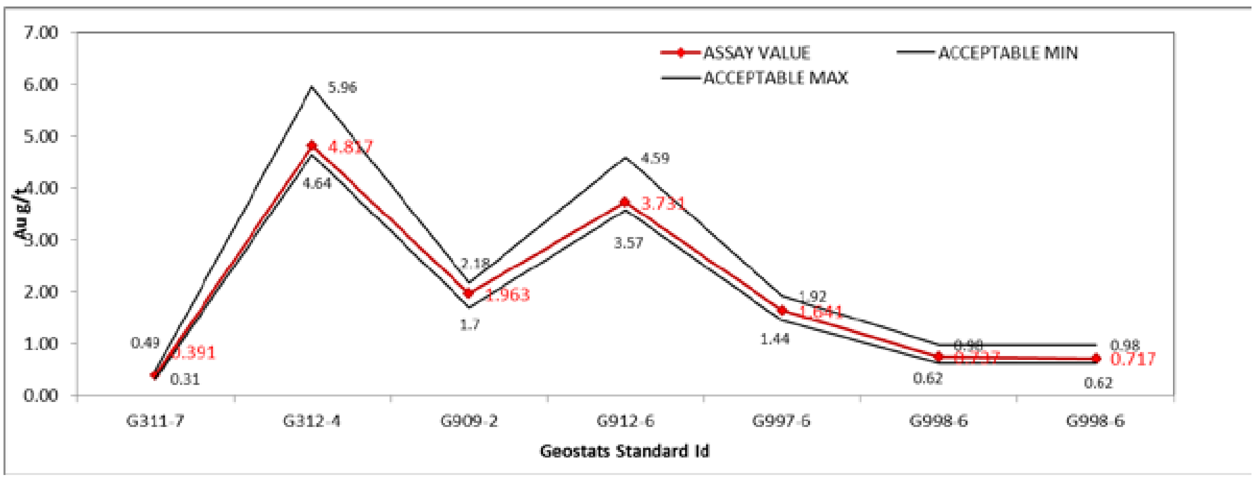
Figure 12.29 QA/QC Standards – G998-6



# of Anlys above Threshold	7
# Outside Warning Limit	0
# Outside Error Limit	0
# of Anlys below Threshold	0
% Outside Error Limit	0
Mean	0.7443
Median	0.74
Min	0.68
Max	0.82
Standard Deviation	0.0431
% Rel. Std. Dev.	5.7975
Standard Error	0.0163
% Rel. Std. Err.	2.1912
Total Bias	-0.0696
% Mean Bias	-6.9643

Eight different Geostats standards, related to duplicates, were inserted into the batch and sent to the SGS lab. The results of this batch showed that none of samples were out of the threshold limit (Figure 12.30).

Figure 12.30 QA/QC Standards – 2015 Drilling Campaign

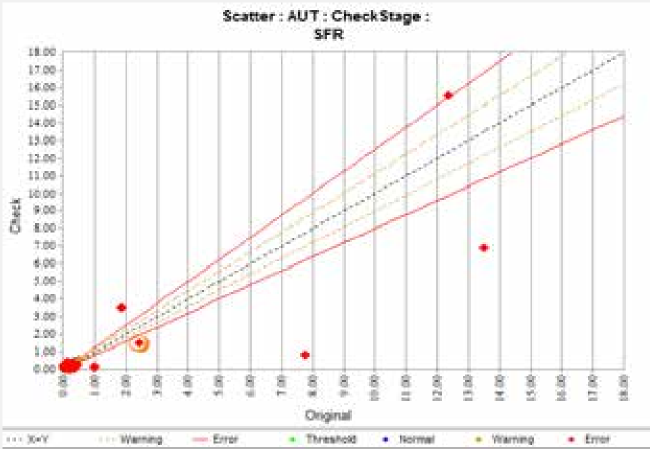


**Samples analyzed at SF Lab and re-sent to SGS*

12.4.12 Analytical Precision Analysis – Duplicates

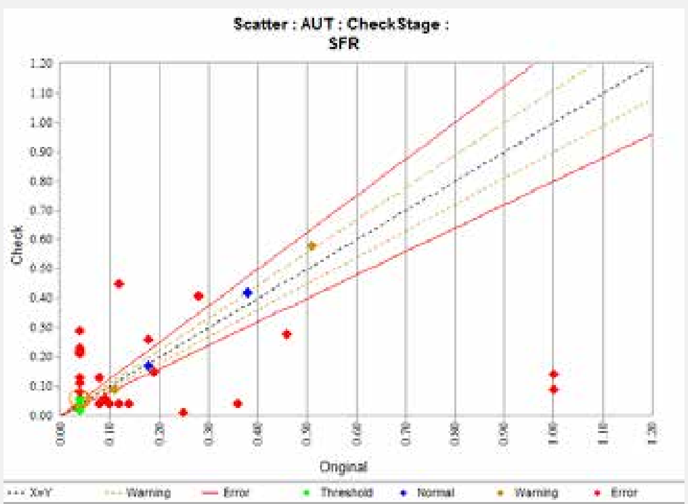
117 field duplicates were inserted into the batches sent to the SF Lab to understand the reproducibility (precision) of the preparation and analysis. The results showed 37 samples out of the limits that represented the low precision of the SF lab (Figure 12.31). For better view of the duplicates, filtered data above 1g/t Au is presented in Figure 12.32.

Figure 12.31 QA/QC Field duplicates – Duplicate of the previous sample code



# of Anlys. above Threshold	41
# of Anlys below Threshold	76
# Outside Warning Limit	39
# Outside Error Limit	37
Mean	1.08 / 0.83
Median	0.11 / 0.13
Min	0.04 / 0.01
Max	13.51 / 15.53
Range	13.47 / 15.52
Variance	8.80 / 6.76
Coeff. of Var.	2.77 / 3.18
Sample Std. Dev.	3.00 / 2.63
Bias	-0.237
Corr. Coeff.	0.84
RMA Error on Y Intercept	0.0076

Figure 12.32 QA/QC Field duplicates – upper cut-off 1.0 g/t Au



# of Anlys. above Threshold	36
# of Anlys below Threshold	81
# Outside Warning Limit	34
# Outside Error Limit	32
Mean	0.18 / 0.16
Median	0.09 / 0.11
Min	0.04 / 0.01
Max	1.00 / 0.58
Range	0.96 / 0.57
Variance	0.05 / 0.02
Coeff. of Var.	1.33 / 0.85
Sample Std. Dev.	0.24 / 0.13
Bias	-0.118
Corr. Coeff.	0.84
RMA Error on Y Intercept	0.0076

After being analyzed at the SF lab, certain samples were sent to the SGS lab (located in Belo Horizonte, MG, Brazil). A batch of 73 samples was selected based on geological and chemical results of surrounding mineralized zones to validate SF lab analysis. To compare the results the same exercise of duplicate analysis for drilling campaigns before 2014 was carried out and are presented in the following graphs:

- Scatterplot of Analysis (Figure 12.33);
- Precision Pair Plot ("HAD") Analysis (Figure 12.34);
- Relative Difference Plot Analysis (Figure 12.35);
- HARD Plot Analysis (Figure 12.36).

Figure 12.33 Scatterplot of SF x SGS inter lab Au g/t Field Results

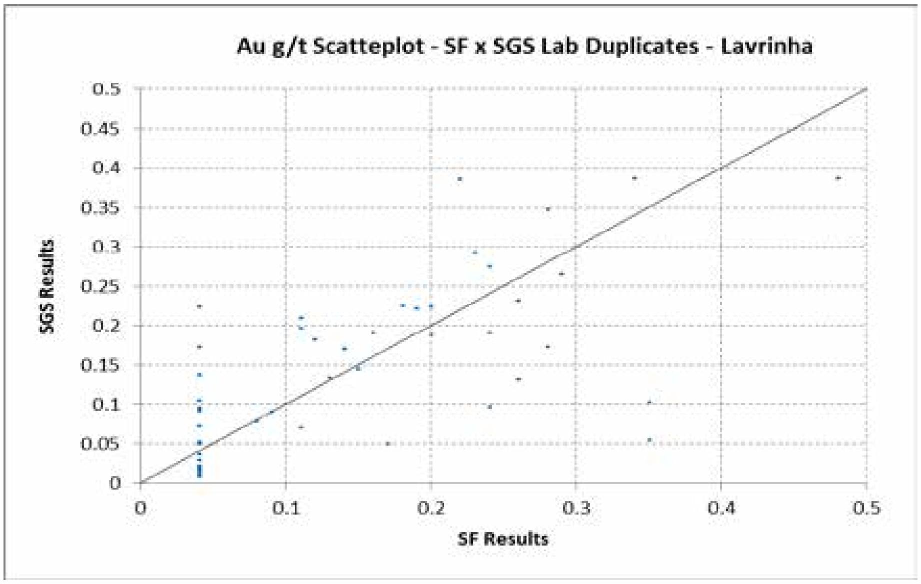


Figure 12.34 Relative Difference Plots for SF x SGS inter lab Au g/t Field Duplicates

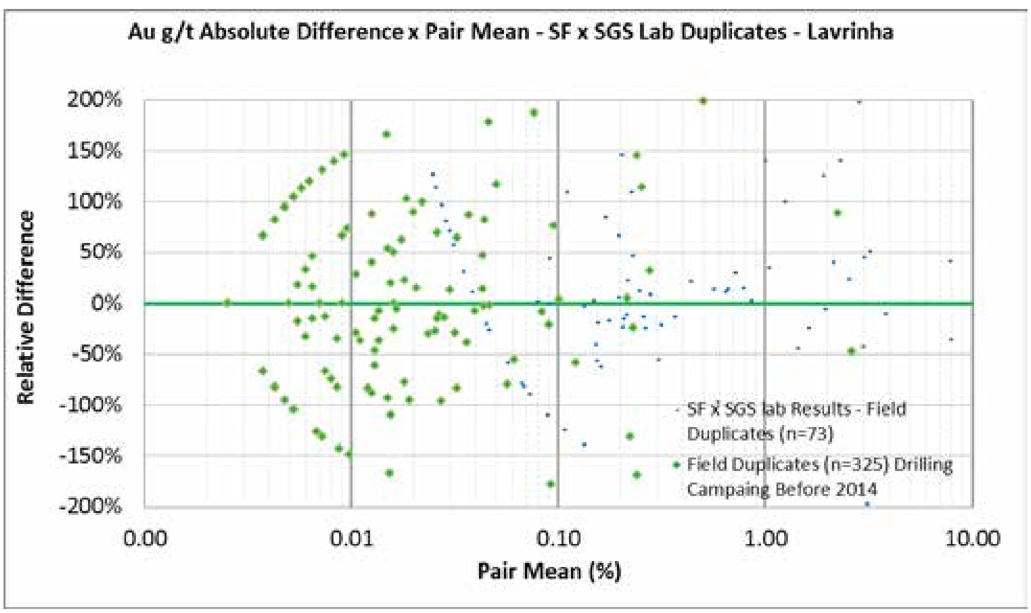


Figure 12.35 Precision Pairs Plots for SF x SGS inter lab Au g/t Field Duplicates

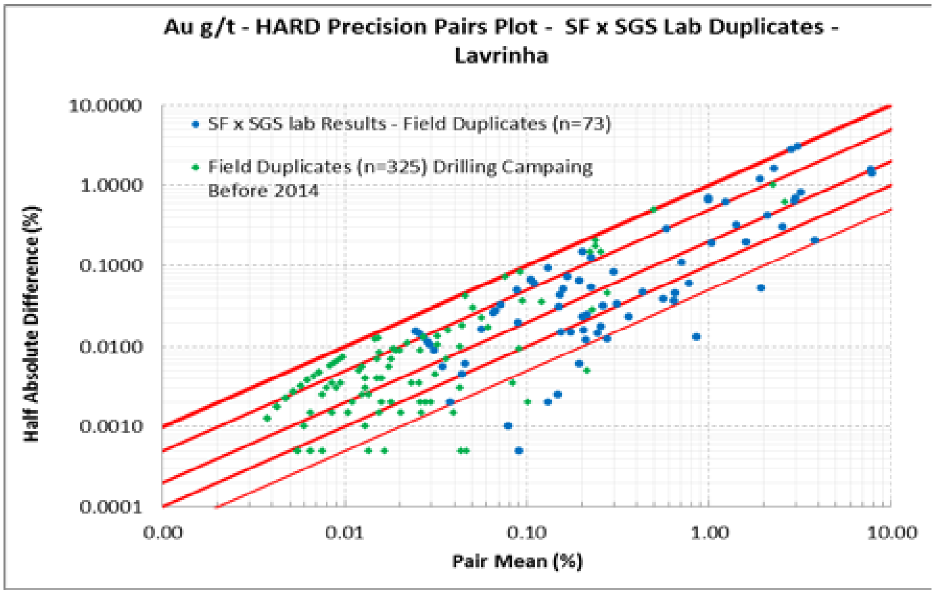
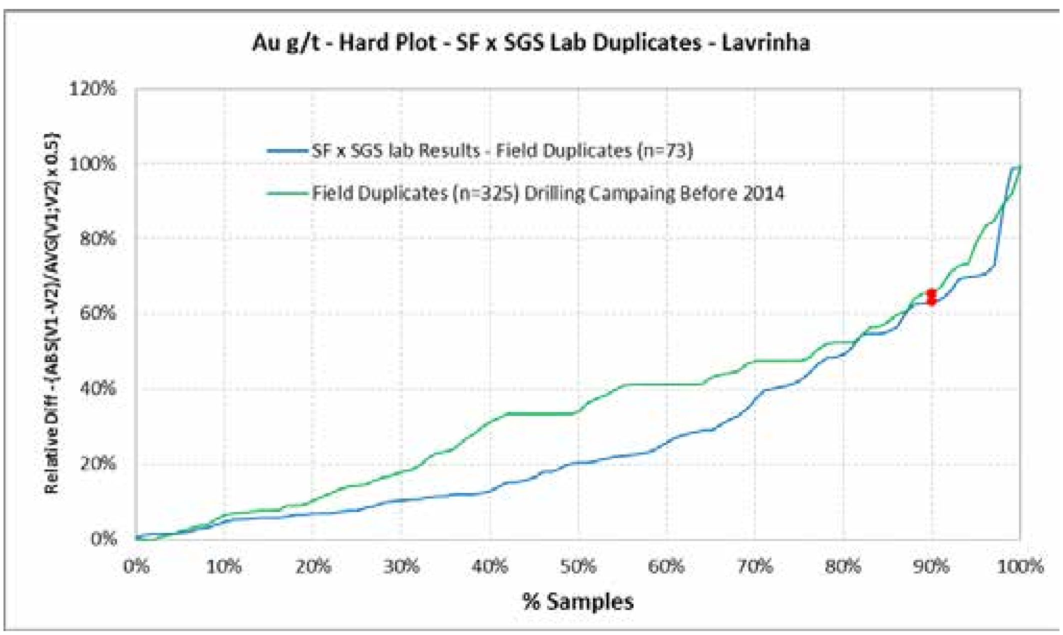


Figure 12.36 Ranked HARD plot showing the variables precision for SF x SGS inter Lab Field Duplicates



The comparison between field duplicates that were analyzed before 2014 (analyzed at ALS and excluded samples below detection limit to keep two datasets at the same basis) and 2015 drilling campaign duplicates, showed much better precision of pairs analyzed in 2015 (analysis at São Francisco and re-assay at SGS) than field duplicates analyzed at ALS laboratories in 2014, confirming the acceptable precision of 2015 Au g/t grades for the mineral resource estimation.

12.5 PAU-A-PIQUE QUALITY ASSURANCE/QUALITY CONTROL PROGRAM

12.5.1 2006 - 2009 Drill Programs

The following description has largely been summarized from Yamana's report titled, "Ernesto and Pau-a-Pique Feasibility Study Report, Revision No. 2", dated March 17, 2010 and prepared by Ausenco de Brasil Engenharia, Ltda., of Belo Horizonte, Minas Gerais, Brazil.

Yamana's QA/QC programs for the years 2005 to 2009 included the routine insertion of blank samples and gold standards into the sample batches sent for analysis to the laboratory.

Blanks were inserted at a rate of 1 in 40 samples and attempted to mark the end of an expected mineralized interval. The blank material used was locally sourced from white quartz vein collected nearby from the Ernesto target.

One or two standards were inserted into each zone where mineralization was expected; giving an overall average of 1 in 15 standards for the mineralized zones, and the standard grade was matched to the expected average grade of the mineralization.

± 1 standard deviation from the accepted mean value was permitted during monitoring of the standards and any batches with blank or standard failures were re-assayed.

12.5.2 2011 Drill Program

QC protocol for Yamana's Pau-a-Pique 2011 drill program was monitored by their Minera Fazenda Brasileiro ("MFB") Laboratory in Brazil. Laboratory protocol included the insertion of CRMs, blanks and duplicates.

Performance Of Standards

During the course of the 2011 Pau-a-Pique drill program, a total of six standards, of varying grades, were used to monitor gold assays. Multiple standards were inserted into every batch analyzed by the lab.

The author reviewed the lab's data and all standards performed satisfactorily throughout the 2011 QC program.

Performance Of Blanks

A total of 2,837 blanks were inserted into the sample stream during the 2011 year to monitor contamination. An upper tolerance limit of 0.04 g/t Au was set and only four samples returned results greater than this limit. None of these four samples were considered by the author to be of significant impact to the current Resource Estimate.

Performance Of Duplicates

A total of 4,393 laboratory duplicates were inserted into the sample stream throughout the 2015 QC program at Pau-a-Pique to monitor precision for gold.

Scatter plots of the original versus duplicate samples revealed precision to be of an acceptable level.

12.5.3 2012 Drill Program

Yamana implemented an independently monitored QA/QC program for the 2012 drill program at the Pau-a-Pique Deposit, with blanks, CRMs and duplicates routinely inserted into all batches sent for analysis to MFB lab.

Performance Of Standards

Yamana utilized four CRMs during 2012 drilling at Pau-a-Pique: the G901-2, G904-6, G996-7 and G998-6 standards, originating from Geostats Pty Ltd, of Western Australia, Australia. One to four CRMs were included with each sample shipment to the lab.

A total of 167 certified standards were inserted into the sample stream throughout the Pau-a-Pique drill program, monitoring gold only.

Data falling within ± 2 standard deviations from the accepted mean value were passed. Data falling outside ± 3 standard deviations from the accepted mean value, or two consecutive data points falling between ± 2 and ± 3 standard deviations on the same side of the mean, were failed.

Table 12.8 summarizes the different standards and their performance during the 2012 QC program at Pau-a-Pique.

TABLE 12.8
2012 PAU-A-PIQUE QC PROGRAM STANDARD PERFORMANCE FOR AU

Laboratório	Standard Type	Standard Mean	Number of Samples	Number of Failures	% Failures
MFB	G904-6	0.36	48	3	6
MFB	G998-6	0.80	43	5	12
MFB	G901-2	1.76	44	6	14
MFB	G996-7	5.99	32	2	6
Total			**167**	**16**	**10**

Investigation and subsequent re-assays were undertaken for all 16 failures noted in Table 12.8.

The author also reviewed MFB laboratory’s internal lab standards for the year. Three standards of varying grade were utilized to monitor gold during the lab’s 2012 QC program: the G303-8, G310-5 and G903-6 standards. All standards originate from Geostats Pty Ltd, of Western Australia, Australia. Multiple standards were inserted into every sample batch by the laboratory and all standards performed satisfactorily.

Performance Of Blanks

A total of 117 blanks were inserted into the sample stream during 2012 to monitor contamination, with 1 to 4 blanks inserted into a batch. An upper tolerance limit of 0.10 g/t Au was set and any batches with blank samples greater than this limit were re-assayed. A total of 13 samples returned results greater than 0.10 g/t Au during the 2012 drill program and all corresponding batches were reanalyzed.

The author also reviewed the lab's internal blanks. Multiple blanks were inserted into every sample batch by the laboratory and blank performance was considered satisfactory.

Performance Of Duplicates

MFB Laboratory's internal duplicate results were assessed for Yamana's 2012 drill program. The lab analyzed two types of duplicates, coarse reject duplicates ("CRDs") and pulp duplicates, and estimated precision by means of Thompson-Howarth ("T-H") error analysis.

A total of 2,589 CRDs were analyzed for gold and, throughout the year, these duplicates averaged a precision of 25%.

A total of 6,861 pulp duplicates were analyzed for gold and these duplicates averaged a precision of 5% throughout the year.

The author considers both approximations of precision acceptable for gold.

12.5.4 2013 Drill Program

Performance Of Standards

Yamana utilized eight CRMs during the 2013 drill program at Pau-a-Pique: the G305-2, G397-6, G901-2, G904-6, G912-5, G996-7, G997-6 and G998-6 standards. One to four CRMs were included with each sample shipment to the SBMM lab.

A total of 189 certified standards were inserted into the sample stream throughout the Pau-a-Pique drill program, monitoring gold only.

Data falling within ± 2 standard deviations from the accepted mean value were passed. Data falling outside ± 3 standard deviations from the accepted mean value, or two consecutive data points falling between ± 2 and ± 3 standard deviations on the same side of the mean, were failed.

Table 12.9 summarizes the different standards and their performance during the 2013 QC program.

Table 12.9 2013 Pau-a-Pique QC Program Standard Performance for Au					
Laboratório	**Standard Type**	**Standard Mean**	**Number of Samples**	**Number of Failures**	**% Failures**
SBMM	G305-2	0.32	33	0	0
SBMM	G397-6	3.95	9	0	0
SBMM	G901-2	1.76	12	0	0
SBMM	G904-6	0.36	12	0	0
SBMM	G912-5	0.38	8	2	25
SBMM	G996-7	5.99	44	2	5
SBMM	G997-6	1.68	36	3	8
SBMM	G998-6	0.80	35	2	6
Total			**189**	**9**	**5**

Several of the failures were most likely misallocated standards, however follow up action was taken for all 9 failures listed in Table 12.9, including investigation and subsequent re-assaying.

The author also reviewed SBMM laboratory's internal standards from August through December 2013. Eight standards of varying grade were utilized to monitor gold during the lab's 2013 QC program: the G303-8, G901-3, G910-10, G996-7 and G998-6 Geostats standards and the ITAK-563, ITAK-560 and ITAK-566 standards from ITAK of João Monlevade, Minas Gerais, Brazil. Multiple standards were inserted into sample batches by the laboratory and all standards performed satisfactorily.

Performance Of Blanks

A total of 245 blanks were inserted into the sample stream during 2013 to monitor contamination, with 1 to 5 blanks inserted into a batch. An upper tolerance limit of 0.10 g/t Au was set and any batches with blank samples greater than the set tolerance limit were re-assayed. A total of 3 samples returned results greater than 0.10 g/t Au during the drill program and all corresponding batches were reanalyzed.

The author also reviewed the lab's internal blanks for 2013. Multiple blanks were inserted into every sample batch by the laboratory and blank performance was considered satisfactory.

Performance Of Duplicates

A total of 1,969 laboratory duplicates were analyzed by SGS GEOSOL throughout the 2013 drill program at Pau-a-Pique to monitor precision for gold. The author reviewed the duplicates and a scatter plot of the original versus duplicate samples revealed satisfactory precision.

12.5.5 2014 Drill Program

Performance Of Standards

Yamana utilized four CRMs for the earlier portion of the 2014 drill program at Pau-a-Pique: the G398-10, G912-5, G996-7 and G997-6 standards. One to two CRMs were included with each sample shipment to the SBMM lab.

A total of 25 certified standards were inserted into the sample stream of the Pau-a-Pique drill program, monitoring gold only.

Data falling within ± 2 standard deviations from the accepted mean value were passed. Data falling outside ± 3 standard deviations from the accepted mean value, or two consecutive data points falling between ± 2 and ± 3 standard deviations on the same side of the mean, were failed.

Table 12.10 summarizes the different standards and their performance during the 2014 QC program. No failures were recorded for any of the four standards used.

TABLE 12.10
2014 PAU-A-PIQUE QC PROGRAM STANDARD PERFORMANCE FOR AU

Laboratório	Standard Type	Standard Mean	Number of Samples	Number of Failures	% Failures
SBMM	G398-10	4.07	8	0	0
SBMM	G912-5	0.38	3	0	0
SBMM	G996-7	5.99	6	0	0
SBMM	G997-6	1.68	8	0	0
SBMM					
Total			**25**	**0**	**0**

The author also reviewed SBMM laboratory's internal lab standards from February through December 2013. Four standards of varying grade were utilized to monitor gold during the lab's 2014 QC program: the G308-7, G312-4, G901-3 and G910-10 Geostats standards. Multiple standards were inserted into sample batches by the laboratory and all standards performed satisfactorily.

<u>Performance Of Blanks</u>

A total of 29 blanks were inserted into the sample stream for the earlier portion of 2014 to monitor contamination, with 1 to 3 blanks inserted into each batch. An upper tolerance limit of 0.10 g/t Au was set and all blanks were below this limit.

<u>Performance Of Duplicates</u>

A total of 339 laboratory duplicates were analyzed by SGS GEOSOL throughout the 2014 drill program at Pau-a-Pique to monitor precision for gold. The author reviewed the duplicates and a scatter plot of the original versus duplicate samples revealed satisfactory precision.

12.5.6 2015-2016 Drill Program

A total of 2,876 drill samples (this number includes CRM's and blanks) were analyzed for gold by the Company's São Francisco Laboratory during Aura's 2015-2016 drill program. Due to a large proportion of the CRM results falling outside of the established limits (± 3 standard deviations on the same side the accepted CRM mean), greater than 11% of the drill results were selected for re-assay at the São Francisco Lab. Coarse reject samples were re-analysed and a review of the comparison between the original versus coarse reject results is deemed to be comparable and re-assay results have replaced original results in the Pau-a-Pique database where standard failures have occurred.

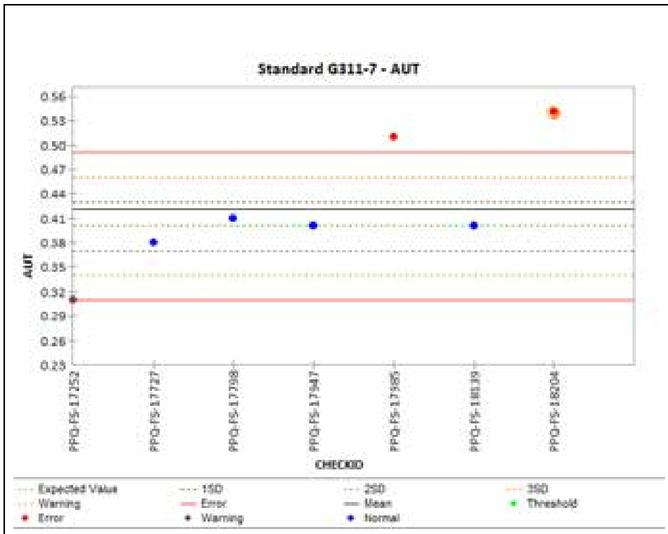
Performance Of Standards

Aura utilized six CRMs throughout the 2015-2016 drill program at Pau-a-Pique: the G311-7, G312-4, G909-2, G912-6, G997-6 and G998-6 standards (Figures 12.37 to 12.42). A total of 30 CRMs were included with the samples sent for re-assaying, monitoring gold only.

Data falling outside ± 3 standard deviations from the accepted mean value were failed.

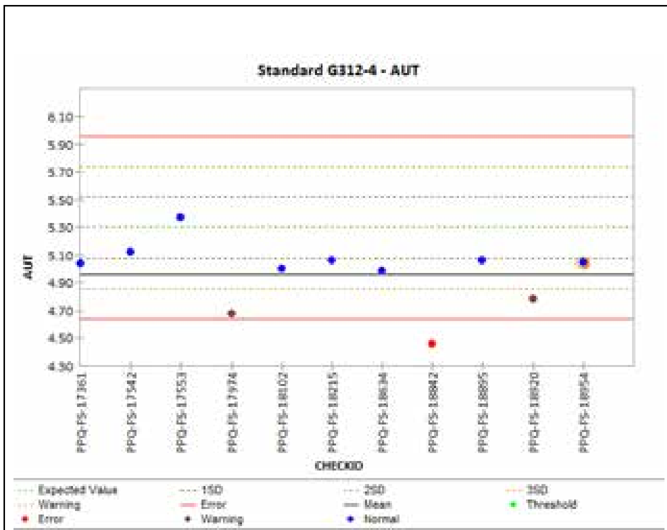
A total of five failures were noted, with no further action taken.

Figure 12.37 Performance of G311-7 Standard for Au



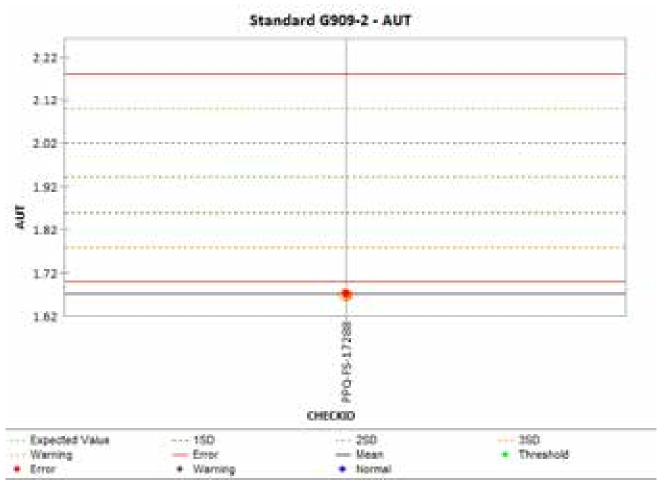
# of Anlys above Threshold	7
# Outside Warning Limit	3
# Outside Error Limit	2
# of Anlys below Threshold	0
% Outside Error Limit	28.5714
Mean	0.4214
Median	0.4
Min	0.31
Max	0.54
Standard Deviation	0.0786
% Rel. Std. Dev.	18.6554
Standard Error	0.0297
% Rel. Std. Err.	7.0511
Total Bias	0.0536
% Mean Bias	5.3571

Figure 12.38 Performance of G312-4 Standard for Au



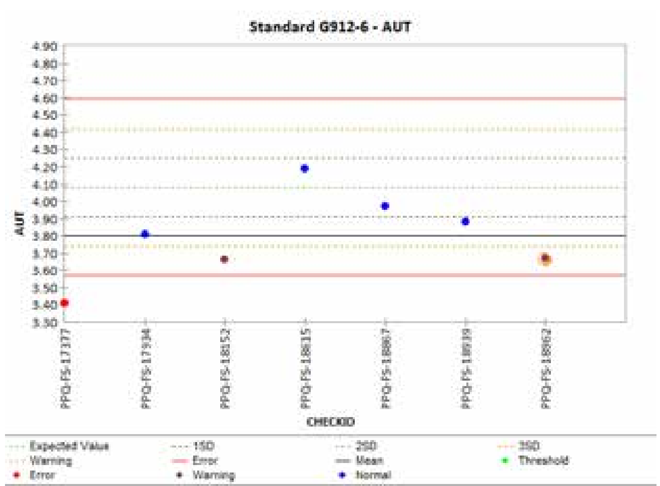
# of Anlys above Threshold	11
# Outside Warning Limit	3
# Outside Error Limit	1
# of Anlys below Threshold	0
% Outside Error Limit	9.0909
Mean	4.9645
Median	5.04
Min	4.46
Max	5.37
Standard Deviation	0.2432
% Rel. Std. Dev.	4.898
Standard Error	0.0733
% Rel. Std. Err.	1.4768
Total Bias	-0.0633
% Mean Bias	-6.3293

Figure 12.39 Performance of G909-2 Standard for Au



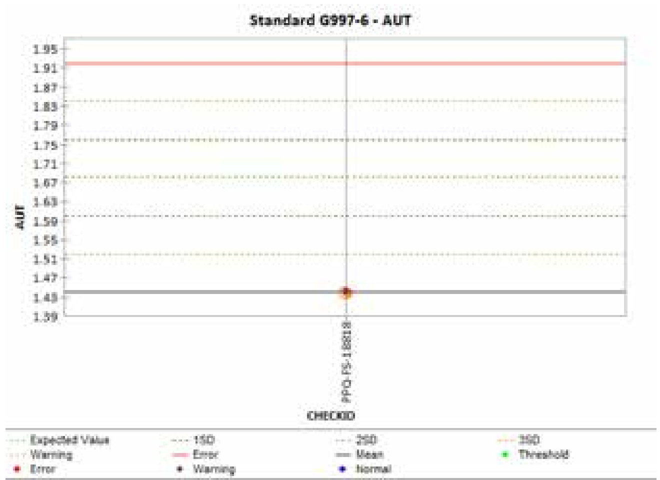
# of Anlys above Threshold	1
# Outside Warning Limit	1
# Outside Error Limit	1
# of Anlys below Threshold	0
% Outside Error Limit	
Mean	
Median	
Min	
Max	
Standard Deviation	
% Rel. Std. Dev.	
Standard Error	
% Rel. Std. Err.	
Total Bias	
% Mean Bias	

Figure 12.40 Performance of G912-6 Standard for Au



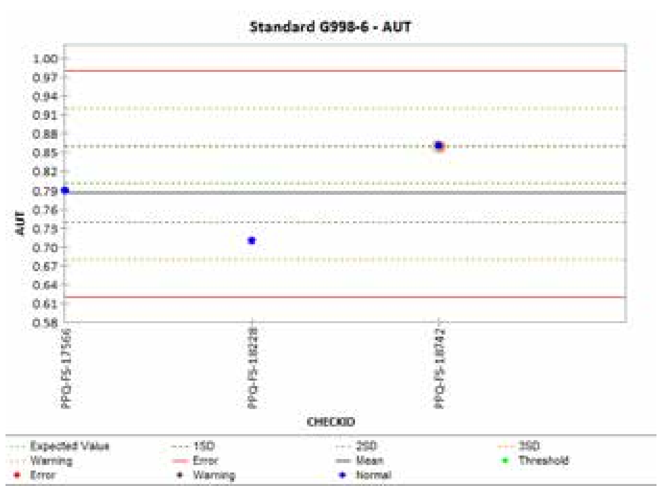
# of Anlys above Threshold	7
# Outside Warning Limit	3
# Outside Error Limit	1
# of Anlys below Threshold	0
% Outside Error Limit	14.2857
Mean	3.7986
Median	3.81
Min	3.41
Max	4.19
Standard Deviation	0.2504
% Rel. Std. Dev.	6.5909
Standard Error	0.0946
% Rel. Std. Err.	2.4911
Total Bias	-0.069
% Mean Bias	-6.8978

Figure 12.41 Performance of G997-6 Standard for Au



# of Anlys above Threshold	1
# Outside Warning Limit	1
# Outside Error Limit	0
# of Anlys below Threshold	0
% Outside Error Limit	
Mean	
Median	
Min	
Max	
Standard Deviation	
% Rel. Std. Dev.	
Standard Error	
% Rel. Std. Err.	
Total Bias	
% Mean Bias	

Figure 12.42 Performance of G998-6 Standard for Au

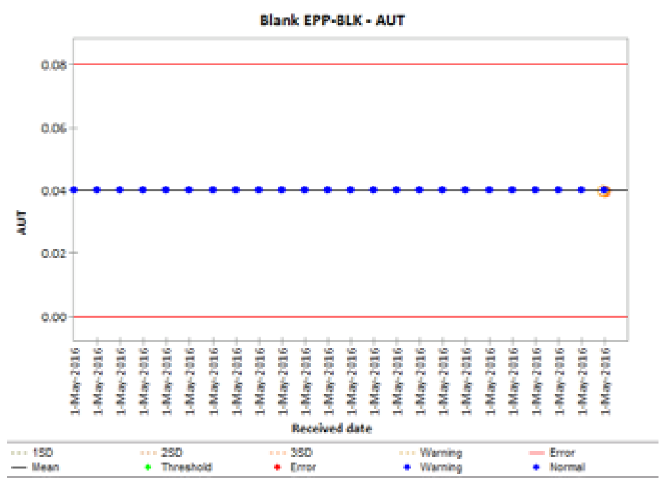


# of Anlys above Threshold	3
# Outside Warning Limit	0
# Outside Error Limit	0
# of Anlys below Threshold	0
% Outside Error Limit	0
Mean	0.7867
Median	0.79
Min	0.71
Max	0.86
Standard Deviation	0.0751
% Rel. Std. Dev.	9.541
Standard Error	0.0433
% Rel. Std. Err.	5.5085
Total Bias	-0.0167
% Mean Bias	-1.6667

Performance Of Blanks

A total of 24 blanks were inserted into the sample stream for Aura's 2015-2016 re-assaying program (Figure 12.43). An upper tolerance limit of 0.08 g/t Au was set and all blanks were below this limit.

Figure 12.43 Performance of Blanks in 2015-2016



# of Anlys above Threshold	24
# Outside Warning Limit	0
# Outside Error Limit	0
# of Anlys below Threshold	0
% Outside Error Limit	0
Mean	0.04
Median	0.04
Min	0.04
Max	0.04
Standard Deviation	0
% Rel. Std. Dev.	0
Standard Error	0
% Rel. Std. Err.	0
Total Bias	n/a
% Mean Bias	n/a

<u>Performance Of Duplicates</u>

No duplicates were assessed by the author for the 2015-2016 drill program at Pau-a-Pique.

12.5.7 2011 Channel Sampling Program

Yamana procured channel samples at Pau-a-Pique as ore control samples to support production drift development. The channel samples were analyzed at Yamana’s MFB lab from 2011 to 2013 and did not include QC samples in the sample batches prepared for analysis until 2014.

The author reviewed SBMM laboratory’s internal standards for the channel sampling undertaken at the Pau-a-Pique Deposit during 2011. The lab inserted multiple CRMs of varying grade into each batch of channel samples to monitor gold. All standards performed satisfactorily.

12.5.8 2012 Channel Sampling Program

No QC data was reviewed for the channel sampling undertaken at Pau-a-Pique in 2012.

12.5.9 2013-14 Channel Sampling Programs

<u>Performance Of Standards</u>

Yamana utilized nine CRMs for the 2013 - 2014 channel sampling programs at Pau-a-Pique: the G305-2, G397-6, G398-10, G901-2, G904-6, G912-5, G996-7, G997-6 and G998-6 standards. At least one CRM was included with each sample shipment to the lab.

A total of 57 standards were inserted into the sample stream of the Pau-a-Pique channel sampling program, monitoring gold only.

Data falling within ± 2 standard deviations from the accepted mean value were passed. Data falling outside ± 3 standard deviations from the accepted mean value, or two consecutive data points falling between ± 2 and ± 3 standard deviations on the same side of the mean, were failed.

A total of 8 standards failed during the 2013 – 2014 programs and follow up action, including investigation and subsequent re-assaying, was taken for all 8 failures.

Performance Of Blanks

A total of 64 blanks were inserted into the sample stream (at least one blank inserted into each batch) for the 2013 - 2014 channel sampling programs at Pau-a-Pique to monitor contamination. An upper tolerance limit of 0.40 g/t Au was set and all but 4 blanks were within this limit. Re-assays were carried out for all four failures recorded.

Performance Of Duplicates

No duplicates were assessed by the author for the 2013 - 2014 channel sampling program at Pau-a-Pique.

12.6 RECOMMENDATIONS AND CONCLUSIONS FOR ERNESTO, LAVRINHA AND PAU-A-PIQUE

Based upon the evaluation of the QA/QC programs undertaken by Yamana and Aura, as well as P&E's due diligence sampling, P&E concludes that the data are of good quality for use in the Ernesto and Pau-a-Pique current Mineral Resource Estimates.

For Lavrinha, MCB had the same conclusion as P&E since the Lavrinha drilling campaigns were carried out simultaneously with Ernesto, applying the same procedures and sampling protocols.

Recommendation is made however, for all future drilling and channel sampling programs at the Project to include a more consistent approach to QC protocol for all samples to be sent for laboratory analysis. QC protocol should include the insertion of QC samples (blanks, CRMs and duplicates) in the field before batches are shipped for analysis.

13.0 MINERAL PROCESSING AND METALLURGY

Samples of the three deposits (Ernesto, Pau-a-Pique and Lavrinha) were selected from available core and sample coarse rejects to represent half years according to the production forecast for the Project. In the main, the core samples were sent for grinding testwork while the coarse rejects were sent for hydrometallurgical testing. SGS Lakefield, Canada, performed the grinding work, which consisted of SAG Power Index ("SPI") and Bond Ball Mill Work Index testwork, while SGS Geosol of Belo Horizonte, Brazil, performed the hydrometallurgical testwork, consisting of Gravity Recovery of Gold ("GRG"), bottle roll leach tests and settling testwork.

The testwork was performed by SGS to confirm the metallurgical performance observed reported over the two years of operation by Yamana and to predict future treatment rates and process plant recoveries.

In providing the analysis in this Section 13 of the Report, Jacobs is relying upon the testwork performed by SGS and information furnished by Aura concerning past recoveries without assuming any responsibility for verifying, or assessing the accuracy or completeness of such information.

13.1 MINERALOGY OF ORE BEARING ROCKS IN ERNESTO, LAVRINHA AND PAU-A-PIQUE

The reader is referred to Section 7.6 of this Report and there can find the results of x-ray diffraction analyses carried out on five samples. Two samples were selected from the Ernesto orebody, two samples from Lavrinha, and one from Pau-a-Pique. These analyses were done to determine if there were any large percentages of clay-forming minerals present. The results show that there are only minimal quantities present.

13.2 HISTORIC METALLURGICAL TESTWORK

The 2010 Ausenco Feasibility Study describes the metallurgical testwork performed on two samples obtained from the Ernesto belt, one from the Japones area and the second sample from the Ernesto area, as well as testwork results for Pau-a-Pique mineralized material.

The Ernesto sample had a gold grade of 4.5 g/t Au and was taken from the Intermediate Trap, which is hosted in adjacent metaconglomerate and metarenites from the Aguapeí Group. The mineralization is associated with quartz veins/veinlets, fresh pyrite (21%) and oxidised pyrite. The presence of free coarse gold in quartz veins or boxwork (oxidised/leached pyrite) was observed.

The Ernesto sample underwent mineralization characterization, grinding, gravity and bottle leaching testwork. Gravity testwork results showed a 68.7% recovery of free gold with an overall mass pull of 1.72%. At the same time, the gold extraction was above 95% in all cyanidation bottle tests with no significant differences in the extraction results with or without carbon and regardless of the grinding conditions.

The Pau-a-Pique metallurgical sample had a gold grade of 5.63 g/t Au. The gravity concentration results showed a high free gold recovery at 61%. The cyanidation bottle tests showed gold recoveries between 80% and 90% without carbon; however, gold recoveries increased to above

90% in the presence of carbon averaging 94.5% for tests with top size of 0.149 mm and 0.074 mm.

All samples from Ernesto and Pau-a-Pique showed a large nugget effect. Table 13.1 summarises the historic metallurgical testwork. It is extracted from the 2010 Ausenco Feasibility Study.

TABLE 13.1
PREVIOUS TESTWORK RESULTS

Table 13.28 – Summary of the metallurgical tests

Test	Company responsible	Ore origin	Weight (kg)	Tests realised	Test recovery	Total recovery
1	Knelson Research	Ernesto	26	Knelson Concentrator		68.7
				P_{80} 662 µm	39.9	
				P_{80} 167 µm	21.7	
				P_{80} 74 µm	7.1	
2	Yamana	Lower Trap	17	Knelson Concentrator	13	93
				Intensive Leaching	80	
		ER-MT	20	Knelson Concentrator	22	90
				Intensive Leaching	68	
		Pau-a-Pique	101	Knelson Concentrator	54	89
				Intensive Leaching	35	
3	Yamana	ER-MT	1	Flotation 80%<0.149 mm	89	89
				Flotation 80%<0.074 mm	51	51
		Pau-a-Pique	1	Flotation 80%<0.149 mm	92	92
				Flotation 80%<0.074 mm	79	79
4	USP	Lower Trap	400	Gravimetry (Heavy Liquid)	23	96
				Leaching	73	
		ER-MT	200	Gravimetry (Heavy Liquid)	68	95
				Leaching	27	
		JP-UT	200	Gravimetry (Heavy Liquid)	55	92
				Leaching	37	
		Pau-a-Pique	200	Gravimetry (Heavy Liquid)	48	90
				Leaching	42	

13.3 HISTORIC PROCESS PLANT PRODUCTION

The EPP process plant started operation in 2013 and was operated until October, 2014, receiving feed from the Ernesto open pit and the Pau-a-Pique underground mine.

During its first year, the plant went through a production ramp-up stage which resulted in consistent process performance improvements (i.e. throughput and gold recoveries) over its quarters as shown in Table 13.2.

TABLE 13.2 2013 PROCESS PLANT STATISTICS					
2013	**Q1**	**Q2**	**Q3**	**Q4**	**Total**
Process Plant Throughput (tonnes)	146,067	183,249	197,051	263,761	790,128
Average Gold Grade (g/t)	1.08	1.33	1.14	1.17	1.15
Average Gold Recovery (%)	86%	90%	93%	97%	92.3%

Average process plant gold recovery was 92.3% from which 41% came from gravity gold and the other 51% was extracted via the CIL circuit.

Although the ramp-up stage took place in 2013, process plant performance in 2014 was not as good due to several issues at the mine level that resulted in a lack of consistent feed supply and the introduction of other feed sources from areas where artisan mining activity was taking place in the concession. 2014 operating plant results are presented in Table 13.3.

TABLE 13.3 2014 PROCESS PLANT STATISTICS					
2014	**Q1***	**Q2**	**Q3**	**Q4+**	**Total**
Process Plant Throughput (tonnes)	154,253	125,177	118,917	42,094	440,441
Average Gold Grade (g/t)	1.24	1.45	1.26	0.80	1.26
Average Gold Recovery (%)	90%	90%	85%	80%	87.7%

**Process plant did not process feed in January.*
+Process Plant operated to end of October.

13.4 RECENT METALLURGICAL TESTWORK

13.4.1 Grinding

Samples were selected from available core to represent ore to be mined from the various mining areas over the LOM. The procedure to select the samples was as follows:

- Review of existing mine plan, available core samples and spatial location intercepting the future areas of production;
- Samples were selected to represent six month production periods and sent to the SGS lab in Lakefield, Canada.
- The SPI and Bond ball mill work index for each composite was determined.

The results can be seen in Table 13.4.

TABLE 13.4 **SPI AND BOND WORK INDEX RESULTS**			
Deposit	**Sample**	**MBWI**	**SPI**
Lav	Year 1 H1	8.1	16.4
Lav	Year 1 H2	8.2	24.1
Lav	Year 2 H1	7.8	20.1
Lav	Year 2 H2	7.7	28.3
Lav	Year 3 H1	8.7	39.8
Ernesto	Year 1 H2	8.9	20.4
Ernesto	Year 2 H1	9.9	30.0
Ernesto	Year 2 H2	10.8	46.3
Ernesto	Year 3 H1	12.8	31.0
Ernesto	Year 3 H2	10.4	31.9
Ernesto	Year 4 H1	9.2	22.5
Ernesto	Year 4 H2	9.3	13.6

The Ernesto results from Year 2 Half 2 onward have been re-calculated proportionally since the mine planning periods have been changed since the samples were collected.

These results were then used to calculate the tonnage that the existing plant would be capable of milling on a daily basis in that period. The calculation was performed using the SPI produced kWh/t for the reduction to 1,760 microns and the bond formula for the part from 1,760 to 106 microns. The results of these calculations can be seen in Table 13.5.

TABLE 13.5
BOND WORK INDEX RESULTS

		Calculated	Calculated	Calculated
Lavrinha		Wi	kWh/t	t/h
Y1 H1		8.1	3.52	258
Y1 H2		8.2	4.35	230
Y2 H1		7.8	3.94	250
Y2 H2		7.7	4.75	225
Y3 H1		8.7	5.73	195
Ernesto				
Y1 H2		8.9	3.97	230
Y2 H1		9.9	4.49	205
Y2 H2		10.8	4.98	190
Y3 H1		12.8	5.72	165
Y3 H2		10.4	5.04	190
Y4 H1		9.2	4.24	220
Y4 H2		9.3	3.69	230

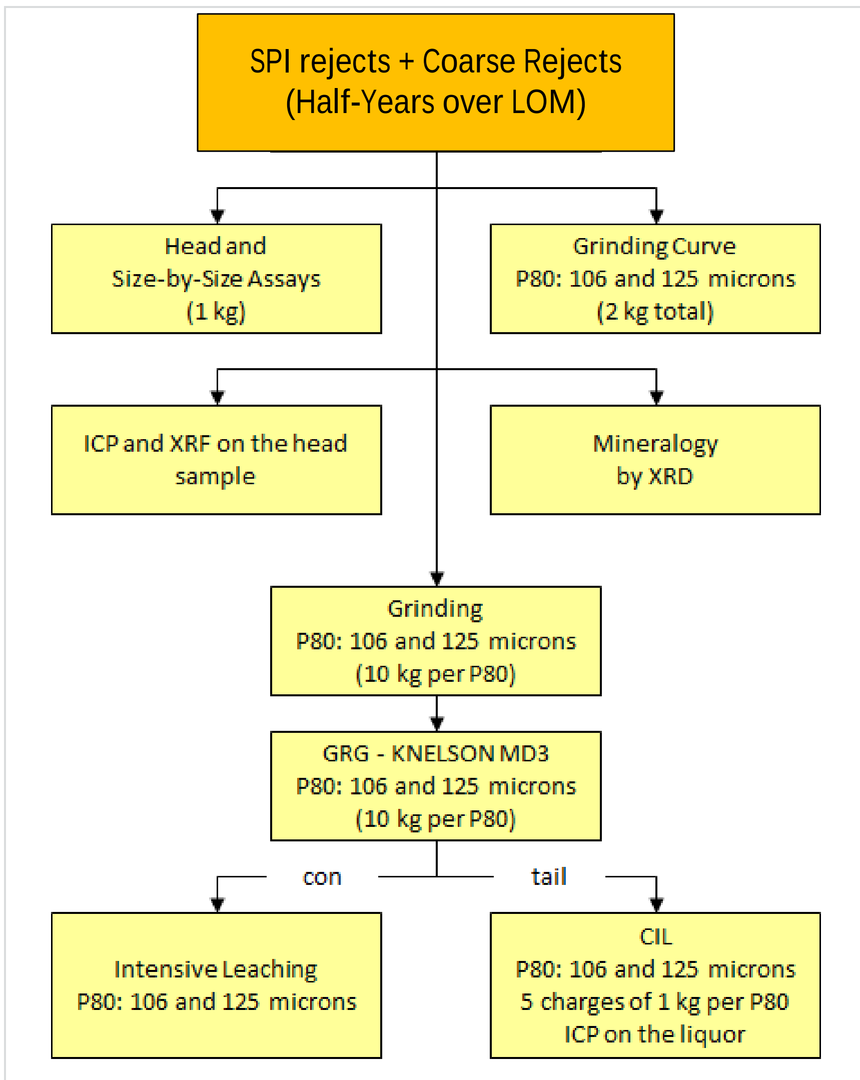
**Figure considers a 92% mechanical availability.*

From these calculations it can be seen that the tonnage figure easily exceeds the planned 3,000 tpd (or 130 tph or 90,000 tonnes/month) for all periods. Therefore, the installed grinding capacity should easily handle future ore throughput forecast for the Project (i.e. between 21,500 tonnes/month and 55,000 tonnes/month) and possibly grind finer since there is available capacity in the SAG mill.

13.4.2 Hydrometallurgy

The samples for metallurgical testing were subjected to a flowsheet similar to that used in the existing plant. Two different grind sizes were used: 106 microns and 125 microns. Each 10 kg sample was ground to the selected grind size and passed through a laboratory Knelson MD3 concentrator to recover free gold. The Knelson concentrate was then subjected to intensive leaching to recover the gold into solution. The Knelson tailings were then leached using a standard CIL technique. The test flowsheet is shown in Figure 13.1.

Figure 13.1 Metallugical Testwork Flowsheet



The results from this metallurgical program for the various deposits follow.

13.4.2.1 Pau-a-Pique Samples

The results for the testwork performed on the Pau-a-Pique samples can be seen in Table 13.6.

TABLE 13.6 METALLURGICAL TESTWORK RESULTS ON PAU-A-PIQUE SAMPLES						
Sample	**Grind Size**	**Gravity Recovery %**	**8-hr Int. Leach Recovery %**	**12-hr Int. Leach Recovery %**	**24-hr CIL Recovery %**	**48-hr CIL Recovery %**
North	125 microns	75.6	90.0	98.7	80.2	81.9
	106 microns	76.1	89.4	98.6	83.7	85.0
South	125 microns	69.0	84.5	97.4	78.6	80.6
	106 microns	65.5	87.7	98.0	80.9	81.1

13.4.2.2 Lavrinha Samples

Table 13.7 shows the results of the testwork program for the two grind sizes used on Lavrinha samples.

TABLE 13.7 METALLURGICAL TESTWORK RESULTS ON LAVRINHA SAMPLES						
Sample	**Grind Size**	**Gravity Recovery %**	**8-hr Int. Leach Recovery %**	**12-hr Int. Leach Recovery %**	**24-hr CIL Recovery %**	**48-hr CIL Recovery %**
Y1 H1	125 microns	74.1	73.8	97.5	96.4	97.0
	106 microns	76.8	79.7	98.3	96.8	96.9
Y1 H2	125 microns	73.4	74.2	97.7	91.1	84.2
	106 microns	77.3	88.5	99.1	91.2	96.6
Y2 H1	125 microns	86.7	67.8	96.3	91.6	96.3
	106 microns	85.8	66.3	90.5	95.5	97.6
Y2 H2	125 microns	n/a	n/a	n/a	n/a	n/a
	106 microns	74.8	93.7	97.8	78.8	95.0
Y3 H1	125 microns	71.5	69.6	93.4	94.0	94.8
	106 microns	74.2	65.4	88.6	95.2	96.7

In all cases the 12 hour intensive leach recovery is substantially higher than the 8 hour, which would be expected.

Most of the 106 micron figures are higher than the 125 micron figures though there are exceptions.

In all cases except one the 48 hour recovery is higher than the 24 hour recovery.

All percent recoveries are in the 90's with one exception.

13.4.2.3 Ernesto Samples

Table 13.8 shows the results of the testwork performed on Ernesto ore for the two grind sizes used.

TABLE 13.8
METALLURGICAL TESTWORK RESULTS ON ERNESTO SAMPLES

Sample	Grind Size	Gravity Recovery %	12-hr Int. Leach Recovery %	24-hr CIL Recovery %	48-hr CIL Recovery %
Y1 H2	125 microns	43.0	65.4	76.3	75.6
	106 microns	49.4	70.0	83.5**	81.5
Y2 H1	125 microns	49.3	91.6	73.1	75.6
	106 microns	51.6	93.4	81.7**	77.4
Y2 H2	125 microns	61.8	81.8	70.3	74.6
	106 microns	63.6	82.8	80.1**	77.3
Y3 H1	125 microns	34.8	80.5	72.2	78.3
	106 microns	33.8	99.7**	82.2**	82.8
Y3 H2	125 microns	55.8	91.1	80.9	84.9
	106 microns	57.3	95.1	84.2**	83.0

******Note:*** *this recovery was achieved with the addition of Leach Aid.*

During the testwork on the Ernesto samples the same conditions as those that were used on the Lavrinha and Pau-a-Pique samples were employed. This resulted in a lower leach recovery in the CIL stage than would have been expected given that cyanide levels were unchanged, although gold grades were double or triple compared to the Lavrinha and Pau-a-Pique ores. Therefore, a lack of free cyanide was indicated as the reason, thought to be so because of the higher grade.

Unfortunately, the lack of fresh core samples prohibited re-doing the testwork at higher cyanide levels for all samples with the exception of Y3H1. SGS Geosol had also stored remaining samples of the Knelson tailings which were used to repeat the CIL part of the testwork and are the results shown in Table 13.8 with the asterisks.

13.4.2.4 Overall Recoveries for all Samples

Table 13.9 shows the overall recoveries for all samples at 125 and 106 microns. The size fraction to be used in the Project, given that these ores are fairly soft and there is oversized capacity in the grinding stage, is 106 microns.

TABLE 13.9
OVERALL RECOVERIES FOR ALL SAMPLES

Lavrinha Recoveries		GRG+12hr	CIL at 24hr	Overall
Y1 H1	125 mic	72.25	96.4	97.22
	106 mic	75.49	96.8	97.95
Y1 H2	125 mic	71.71	91.1	95.94
	106 mic	76.60	91.2	97.31
Y2 H1	125 mic	83.49	91.6	95.67
	106 mic	77.65	95.5	91.21
Y2 H2	125 mic	n/a	n/a	n/a
	106 mic	73.15	78.8	93.01
Y3 H1	125 mic	66.78	94	93.57
	106 mic	65.74	95.2	90.30
106 Microns Average Global Recovery				**93.96**
Pau-a-Pique Recoveries		**GRG+12hr**	**CIL at 24hr**	**Overall**
North	125 mic	74.62	80.2	94.19
	106 mic	75.03	83.7	95.04
South	125 mic	67.21	78.6	91.57
	106 mic	64.19	80.9	92.10
106 Microns Average Global Recovery				**93.57**
Ernesto Recoveries		**GRG+12hr**	**CIL at 24hr**	**Overall**
Y2 H1	125 mic	45.16	73.1	82.22
	106 mic	48.19	81.7	87.74
Y2 H2	125 mic	50.55	70.3	72.35*
	106 mic	52.66	80.1	81.82*
Y3 H1	125 mic	28.01	72.2	50.40*
	106 mic	33.70	82.2	84.25
Y3 H2	125 mic	50.83	80.9	86.59
	106 mic	54.49	84.2	90.45
106 Microns Average Global Recovery				**86.10**

13.4.2.5 Discussion of Ernesto Results

As explained above there were problems with the Ernesto testwork in that the gold recoveries were unexpectedly low. This was thought to be due to the higher grade (twice and three times as high compared to Pau-a-Pique and Lavrinha ores) and a lack of free cyanide found at the end of the leach period.

The 106 micron Knelson tailings were re-leached using a higher concentration at the start of the test and also using 100 g/t of Leach Aid. Table 13.10 presents the comparison of the results.

TABLE 13.10 ERNESTO SAMPLE RE-LEACH CIL RESULTS GOLD RECOVERY		
Sample	**Original conditions; 150 ppm free CN no Leach Aid**	**Re-leach 300 ppm free CN and 100 g/t Leach Aid**
ERN Y1 H2	79.1	83.4
ERN Y2 H1	75.4	81.7
ERN Y2 H2	75.0	80.0
ERN Y3 H1	73.1	77.2
ERN Y3 H2	82.0	84.1

As can be seen from the results there is a substantial increase in recovery for the re-leach tests, averaging 4.36% points higher.

For the Y3 H1 sample a complete retest was carried out, at the 106 micron grind, this being the only sample with sufficient weight remaining to allow it. The gravity recovery was down several percentage points but the intensive leach recovery increased from the previous 92.4% to 99.7% with the use of Leach Aid. This is an increase of 7.3%. In view of this result a case can be made for increasing the other intensive leach recoveries, which could make the overall recoveries increase to 88% levels.

13.4.2.6 Kinetics of Leaching

As part of the test programme a kinetic test was performed on each sample tested. The two plots in Figures 13.2 and 13.3 are extracted from SGS-Geosol's interim report and are typical of all. It can be seen that a 24 hour leach time is adequate.

Figure 13.2 Pau-a-Pique Kinetic Plot Sample North at 106 Microns

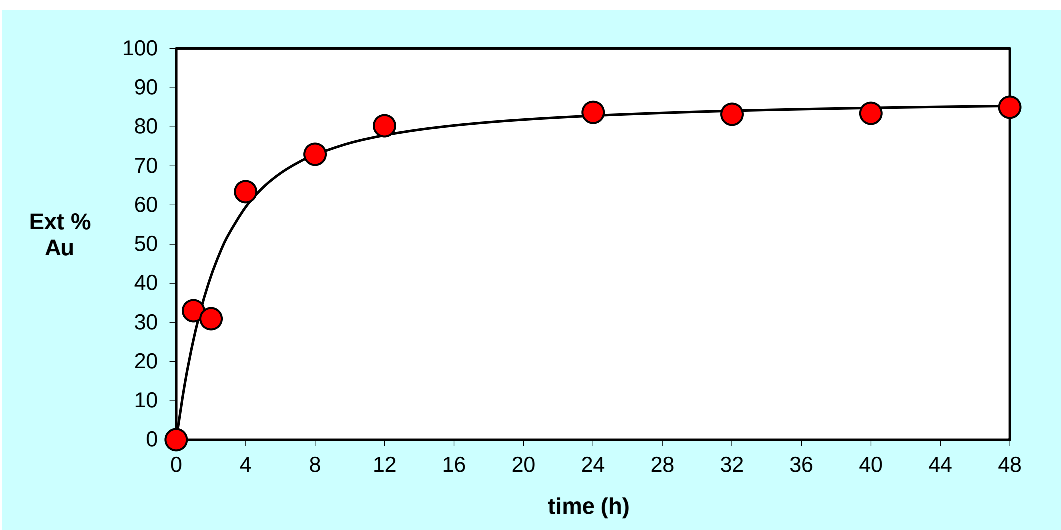
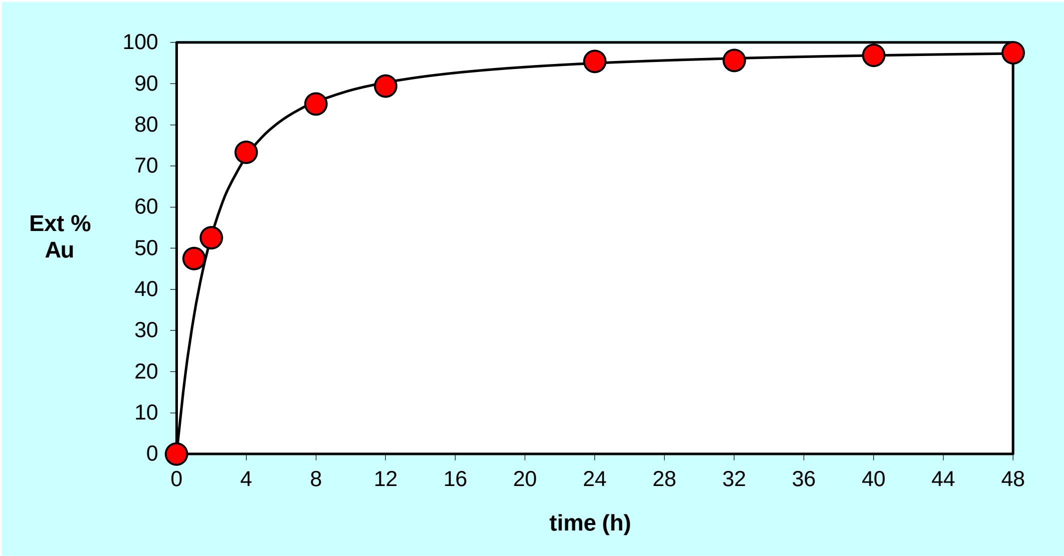


Figure 13.3 Lavrinha Kinetic Plot Sample Y2 H1 at 106 Microns



13.4.2.7 Settling Tests on Ernesto Ore

Settling tests were run on the samples of Ernesto ore tested by SGS Geosol. Each period sample was ground to 125 and 106 microns and dosed with 28 g/t of Senfloc 5210. All samples settled well with clear supernatant. No problems are expected in the plant.

13.4.2.8 Hydrometallurgy Conclusions and Recommendations

The grinding circuit has more than adequate capacity to handle the tonnages planned for the Project. In view of this it may be advisable to investigate whether it would be beneficial to grind finer.

The overall recoveries for the Pau-a-Pique and Lavrinha samples are very good. Those for the Ernesto samples are lower than expected even after the re-leach results are taken into account. Further work should be carried out on Ernesto material to ascertain the reasons for this. The work should investigate using finer grinds, increased cyanide levels and also the use of Leach Aid.

Since the plant has more than enough capacity to grind finer, a series of tests should be performed to establish the optimum grind size for Ernesto ore, and then to establish the optimum leach conditions. Since Ernesto does not come on line for some years, this work can be carried out while other ores are being processed.

14.0 MINERAL RESOURCE ESTIMATES

14.1 ERNESTO MINERAL RESOURCE ESTIMATE

14.1.1 Summary

The mineral resources for the Lower Trap zone at the Ernesto Mine Project ("Ernesto") were estimated by conventional 3D computer block modelling methods employing Dassault Systemes Geovia mining software V6.71 ("GEMS"). The mineral resource estimate is based on surface diamond drilling, core sampling and gold assaying. Assaying was performed at SGS and ALS commercial laboratories in Belo Horizonte and at Yamana mine laboratories Ernesto and MFB as well as the Aura Sao Francisco lab, all in Brazil.

Gold mineralization of the Lower Trap zone at Ernesto consists largely of free gold hosted by mylonite, muscovite schist, and quartz veins accompanied by sulphides that occur along the sheared contact between meta-tonalite and meta-arenite. Mineralization is epigenetic, hydrothermal in origin and is structurally controlled. The rock foliation and mineralized contact trend NNW and have a shallow dip of approximately -25° NNE. The contact is not uniformly planar and is subject to rolling. The Intermediate Trap zones at Ernesto were mined by open pit from 2013 to 2014. Drill hole intersections of these zones are located in the meta-arenite rocks above the Lower Trap zone, however, the Intermediate Trap zones are not included in this mineral resource estimate. The Lower Trap zone has not been mined underground except by garimpeiro (illegal miners) in small workings at one site near surface. This site is outside the current resource area. The narrow widths of the Lower Trap mineralization and depth below topography all but preclude open cast mining and the Lower Trap zone is amenable only to underground mining.

The exploration drill hole database for the Lower Trap zone underground mineral resource area contains 329 diamond drill holes totalling 47,932.22 m. Hole lengths range from 9.10 m to 615.55 m. The mineral resource is defined by 87 drill holes.

The mineral resource wireframes were constructed from mineralization intersections in drill holes at a cut-off grade of 1.5 g/t Au over a minimum vertical mining width of 2.0 m. Gold price used for the resource estimate was US$1,275/oz. Process assumptions are 93% recovery, 99.99% for payable and $15/oz Au for refining. The cut-off grade represents a marginal unit operating cut-off of US$55.10/tonne of mineralization processed based on 75% of the estimated mining cost of US$49.90/tonne. Mineralization widths are commonly narrower than minimum mining width and were "bulked out" to at least the minimum width using adjacent assays.

Assay grades were capped at 40 g/t Au. Assay composites were generated for the zone intersections from the assays captured by GEMS software in the mineralized wireframes. Equal length composites were generated dynamically at a nominal 2.0 m down-hole length. This method ensures that the grade weighting is correctly applied for bulked out domain widths but results in variable composite lengths.

Two block models were created, a lithologic model for geologic interpretation and a resource block model. The X-axis of the resource block model is rotated to 95° azimuth. Resource block size is 10 m x 10 m x 2 m vertical which is suitable for selective mining and benching methods such as room and pillar, drift and fill and mechanized cut-and fill. Ordinary Kriging ("OK") interpolation was carried out using multiple search distances and search ellipses oriented to the

NE mineralization plunge. Inverse distance squared (“ID2”) and nearest neighbour (“NN”) interpolation methods were employed for model validation.

Water immersion bulk density testing was carried out at Ernesto by Yamana for 627 core samples in 84 ER series holes and an additional 25 tests were performed during Knight Piesold geotechnical work in 2015, P&E due diligence sampling (6) in June 2015 and as a separate exercise by Aura personnel (8) carried out in February 2016. The resources are almost entirely within mylonite-sericite schist (SG 2.62) and quartz veining (SG 2.62) of the Lower Trap and thus 2.62 t/m^3 was employed as the bulk density for conversion of resource volume to resource tonnes.

Mineral resources were classified as Indicated and Inferred based drill hole spacing, confidence in the assaying and geologic confidence in the zones interpretation and grade continuity.

The total Indicated mineral resource estimated for a 1.5 g/t Au cut-off grade is 734,000 tonnes averaging 6.91 g/t Au (163,100 ounces gold). The total Inferred resource for a 1.5 g/t Au cut-off grade is 308,000 tonnes averaging 6.30 g/t Au (62,400 ounces gold).

Validation of the grade interpolation and the block model was carried out by on-screen review of grades and other block model estimation parameters versus drill hole composites, by comparison of assay, composites, zone intersections and block grades, comparison to alternate ID2 and nearest neighbour interpolations, and review of the volumetrics of wireframes versus reported resources. In P&E’s opinion, the mineral resource estimate is reasonable and has been undertaken according to industry standard practice.

The best potential to develop additional resources for an engineering study lies in fill-in drilling and sampling to upgrade the Inferred Resource to Indicated Resource.

14.1.2 Resource Database

The Ernesto Lower Trap zone has been sampled by surface diamond drilling and core sampling. Core for the surface drilling is largely NQ (47.6 mm). The Lower Trap database contains 329 holes for 47,932.22 m of which the KP15 geotechnical and P series were drilled in 2015 and funded by Aura. The other holes are historic holes drilled by Yamana. The resource is estimated from 87 holes for 13,136.49 m (Table 14.1). Five holes in the database, ERN0098 to ERN00102, have no down hole surveys and are unsampled, and one hole, ERN0056, has no variation in azimuth down hole and was not likely surveyed down hole for azimuth.

The drill holes have been drilled on a relatively wide grid of 35 m x 35 m oriented at 095° with fill-in drilling done primarily on the shallower west side of the deposit resulting in an irregular pattern overall. To the north and east, the drill hole spacing is in the order of 35 m x 100 m and is considered too wide for confidence in resource estimation at a classification any higher than Inferred.

Interval records in the database total 23,865 of which assayed intervals total 22,571 over 25,533.69 m and 1,294 records for 21,831.66 m are non-sampled despite some having sample numbers. The Lower Trap zone resource assay database totals 424 assays for 390.17 m in 87 diamond drill holes.

TABLE 14.1 SUMMARY OF DRILL HOLE DATABASE FOR ERNESTO LOWER TRAP							
Series	**Count**	**Length (m)**	**% by Length**	**Sub Vertical[1]**	**Unsampled**	**Resource**	**Survey Flag[2]**
ER	163	27,601.83	58%	143	1	36	0
ERN	100	12,914.88	27%	38	13	36	6
ERMP	31	2,650.53	6%	19	0	1	0
FE	14	1,688.03	4%	14	0	7	0
KP15	6	997.16	2%	0	3	2	0
P	15	2,079.79	4%	2	0	5	0
All	**329**	**47,932.22**	**100%**	**216**	**17**	**87**	**6**

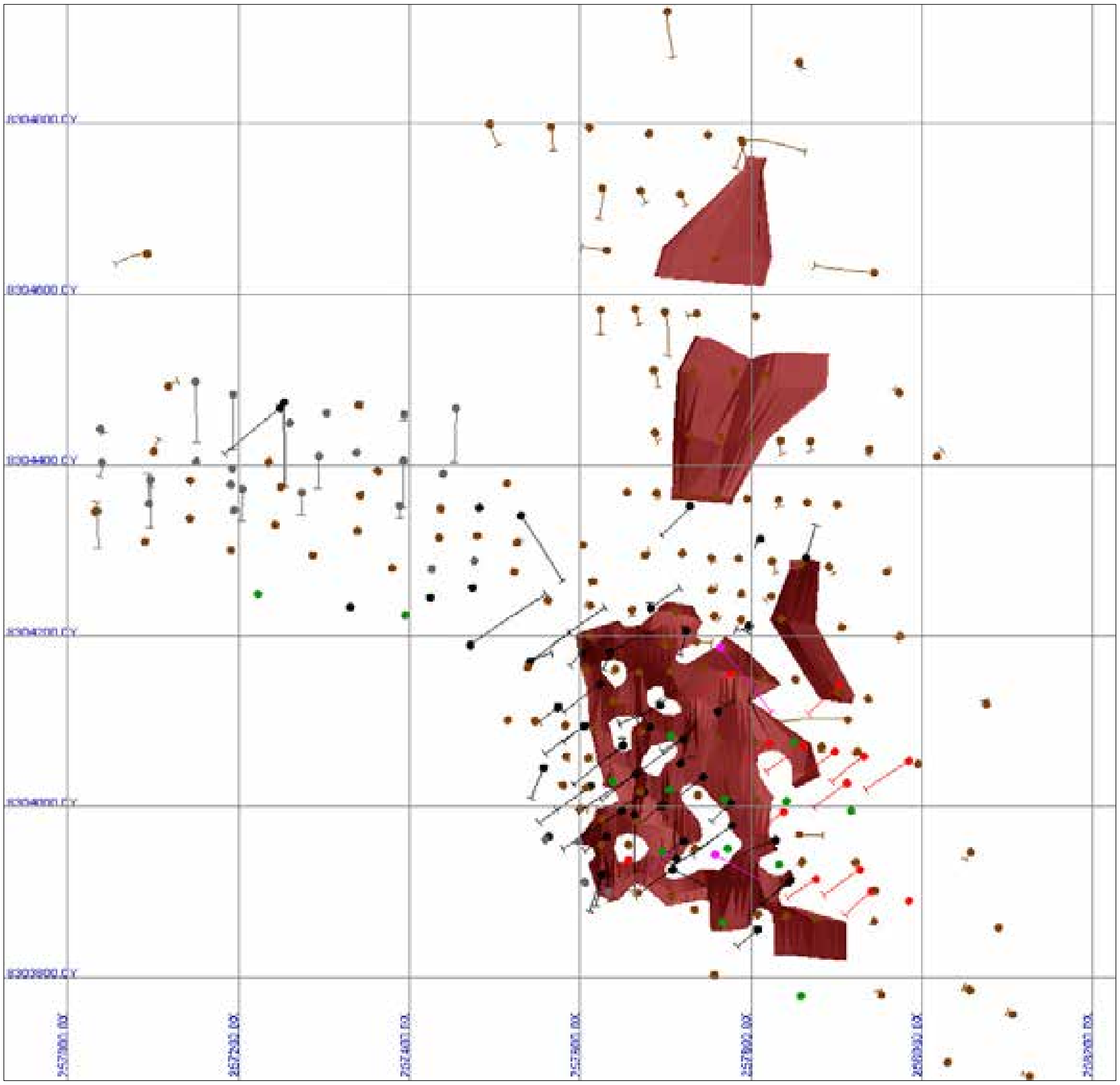
(1) Steeper than -85°; remainder inclined
(2) No down hole survey or only dip surveyed

P&E reviewed the Yamana and Aura QAQC programs and the lab internal QAQC blanks and reference standards and in P&E's opinion, the assay database is acceptable for Mineral Resource and Mineral Reserve estimation.

P&E notes that drill holes completely lacking assays in the Lower Trap zone were omitted from resource estimation. For partially assayed holes where few explicit or implicit missing assays were used for resource estimation, the missing intervals were assigned zero grade. Explicit and implicit missing assay intervals in the resource database are minimal.

Figure 14.1 shows the location of diamond drill holes in plan and the Lower Trap zone resources projected to surface.

Figure 14.1 Diamond Drill Hole Location Plan and Ernesto Lower Trap Deposit



Drill Hole Series Legend

P	
KP15	
ERN	
ER	
ERMP	
FE	

14.1.3 Lower Trap Zone Wireframing

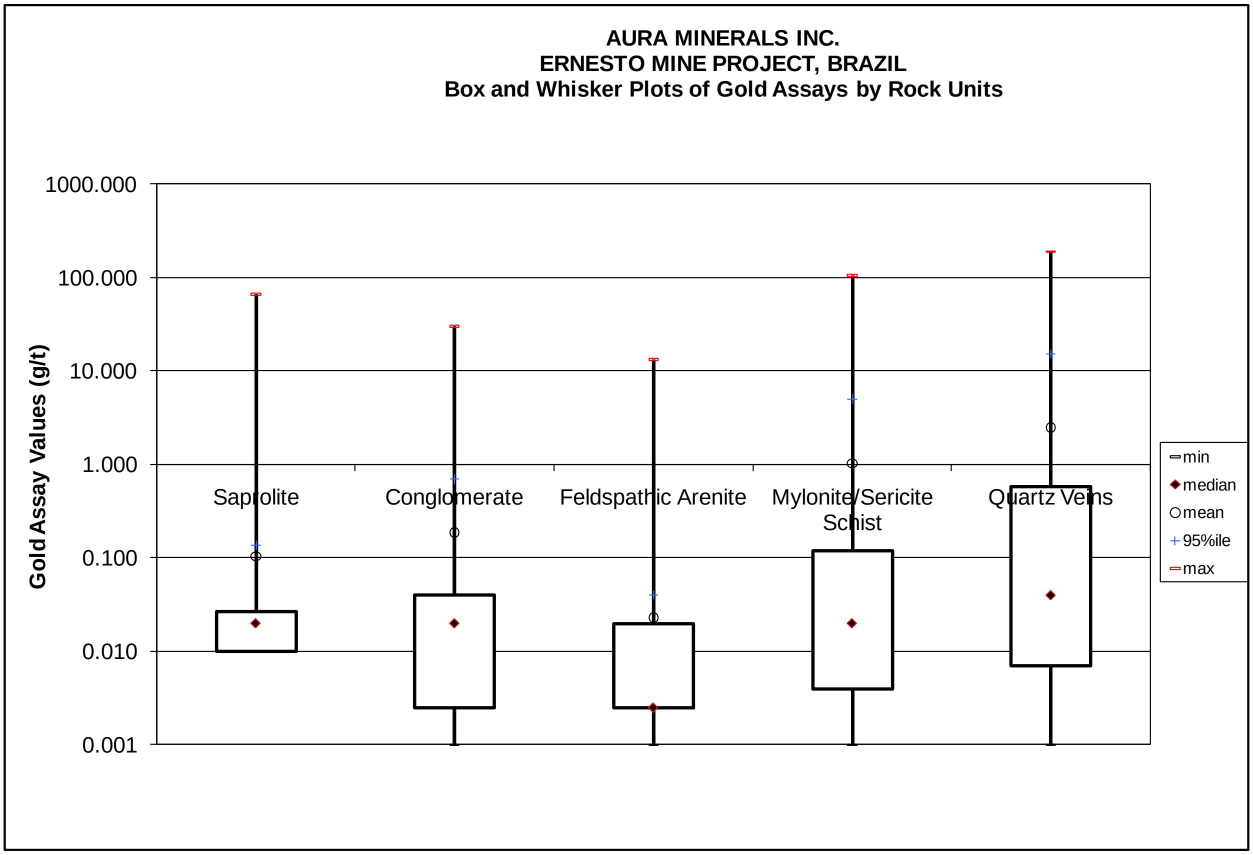
The basis for mineral zone delineation and wireframing is a cut-off grade of 1.5 g/t Au over a minimum horizontal mining width of 2.0 m. This grade is considered as a marginal cut-off for an end of mine life cost of 75% of normal mining cost (Table14.2).

TABLE 14.2 GOLD PRICE AND OPERATING COST PARAMETERS FOR RESOURCE CUT-OFF GRADE DETERMINATION	
Au Price US$/oz	$1,275
Mining Cost US$/t ore	$49.90
Marginal Mining Cost US$/t ore @ 75% of Mining Cost	$37.40
Process Cost US$/t ore	$12.50
G&A Cost US$/t ore	$5.10
Au Process Recovery	93.0%
Au Smelter Payable	99.99%
Au Refining US$/oz	$15.00
Au Cut-Off Grade g/t	1.46
Resource Cut-Off Grade Au g/t	1.50

Review of the geology, host rocks, apparent controls on gold mineralization, and distribution of assay grades in drilling brought to light the following aspects of interpretation for wireframe modelling:

- Free gold is common and gold distribution is erratic both laterally and vertically within the Lower Trap zone that consists of the narrow to broad, mylonitic and schistose contact between meta-arenite/ meta-tonalite boundary.
- Gold at resource grades is hosted mostly by mylonite/sericite schist (unit 8) and quartz veining (unit 9) (Figure 14.2) and grade distribution is strongly skewed with a high "nugget" (>50%) effect. Minor amounts of gold mineralization also occur in hanging wall and footwall rock units.
- The deposit is tabular with variable shallow dips likely related to rolls that appear to impact on the distribution of gold. Dip flattens to sub horizontal in the northern area of the resources and may reverse dip to shallow to the SW.
- Gold mineralization may be found in meta-arenite or meta-tonalite metres into the hanging wall or footwall of the contact schists, however, these are likely minor separate shears or splays off the contact shear zone and 3D continuity may not be demonstrated resulting in these isolated occurrences being ignored for the purpose of resource estimation.
- Internal dilution to make resource minimum widths is relatively high and resource intersections supported by a single assay over one metre are common and have been bulked out to make a minimum vertical mining width of 2.0 metres.
- The location of the resource intersections varies within the Lower Trap zone from hanging wall to footwall of the mylonite/schist unit. As such the continuity of the zone hole to hole may be variable and the mineralization more lensoidal in nature than assumed for the purpose of resource estimation given the relatively wide spacing of the drill holes.

Figure 14.2 Distribution of Gold by Rock Type



After review of drill hole spacing, cross sections were developed at 35 m spacing at 95° azimuth parallel to the drilling grid. Wireframing was carried out by snapping to assay limits in 3D space where cumulated assays achieved cut-off grade over the minimum mining width. Geologic interpretation and following the contact zone using the lithologic block model was a key aspect of the wireframing. A surface was also created in GEMS from lithologic data that incorporated the resource intersections and middle of the mylonite/schist unit. This trend surface was used to project the wireframe where drilling was wide-spaced. A preliminary wireframe at a 1.0 g/t Au cut-off grade over 2.0 m was also constructed and used for general guidance where drill hole spacing was wide. A mineral wireframe built by Yamana for low grade gold mineralization (0.5 g/t Au) was also available for reference.

The wireframes were extended half way to adjacent drill holes internally within the wireframed deposit or on the margins where barren or low grade holes exist. Drilling density at the margins of the wireframe is such that the perimeters of the wireframes are essentially closed off. In a few cases where sub cut-off/width material in a drill hole occurred within the zone between adjacent resource grade intersections, the wireframe was carried through to maintain zone continuity. Similarly the nominal 2.0 m width was maintained where practicable but may be less at zone inflection points.

For solids creation, the erratic nature of gold distribution and consequent spatial complexity of resource intersections and polyline rings' locations, owing to the irregular drill hole pattern, dictated that the conventional use of simple polyline splits/bifurcations was not workable. Consequently, a generalized lithology based wireframe that included the resource intersections and portions of the mylonite/schist in barren holes, was generated and then “clipped” in plan to remove the barren areas from the resource wireframe.

Volumetrics of the wireframes and estimated tonnage for a bulk density of 2.62 t/m^3 are presented in Table 14.3.

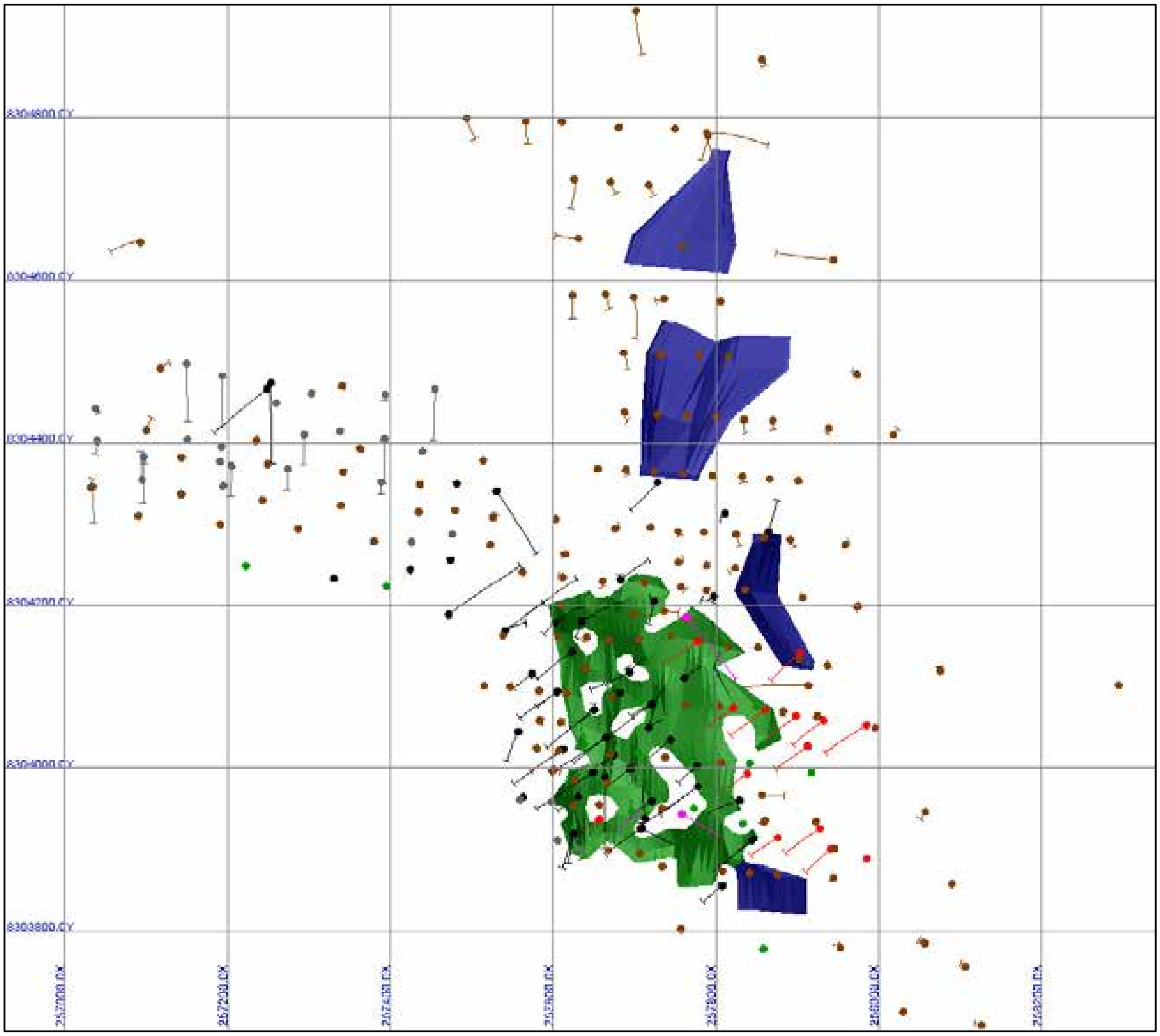
TABLE 14.3 WIREFRAME VOLUMETRICS		
Solid	**Volume (m^3)**	**Tonnes[1]**
Lower Trap (Indicated)	280,844	735,811
Lower Trap (Inferred)	117,527	307,921
Total	**398,371**	**1,043,732**

(1) Bulk density of 2.62 t/m^3.

Intersections in the wireframes were composited (“intervals”) and preliminary grades and thickness estimated from the averaged strike and dip (335°/-25°) of the Lower Trap zone. Average vertical width is estimated at 4.8 m and average true thickness is averaged at 4.3 m for the main drilled area and average vertical width is estimated at 2.6 m and average true thickness is averaged at 2.4 m for the “Inferred” wireframe where drill holes are widely spaced.

The wireframes at 1.5 g/t Au cut-off grade are shown in plan in Figure 14.3.

Figure 14.3 Wireframes Modelled at 1.5 g/t Au and 2 m Minimum Vertical Mining Thickness



Zone Legend	
	Indicated Rock Code 100
	Inferred Rock Code 200

14.1.4 Assay Statistics & Grade Capping

Assay statistics and grade distributions were examined for assays captured in the wireframes. Gold grade distributions show extended skew (Poissonian) with possibly two populations, a low grade set up to approximately 1.0 g/t and a second higher grade population. The latter represents the deposit mineralization whereas the former set may be an artefact of bulking up the zone intersections and/or varied assay detection limits.

Histograms and log-probability plots were employed to evaluate gold grade distribution and grade capping curves were utilized to show the impact of capping levels on assay average grade. The capping level indicated from the graphs is 40 g/t Au which coincides with inflections on the log-probability, coefficient of variation, % metal and % samples lost on the top cut charts, Appendix 1. 3D distribution of high grade assays was examined on-screen to ensure that "outlier" assays were not spatially correlated. Graphs are available in Appendix 1. Results of the grade capping are presented in Table 14.4. From this table, it is clear that capping has a significant impact on average grade and grade variability.

TABLE 14.4 **CORE ASSAYS CAPPING SUMMARY**	
No. of Assays	424
Average grade (g/t Au)	7.21
Coefficient of Variation	1.99
Cap Level (g/t Au)	40
No. of Assays Capped	12
% Capped	3
% Metal Lost	9.3
Average Grade of Capped Assays (g/t Au)	6.55
Coefficient of Variation Capped	1.43

Statistics for the wireframe assays are presented in Table 14.5.

TABLE 14.5
ASSAY STATISTICS

Core Assays			
Statistic	**Length (m)**	**Au g/t**	**Au g/t Capped**
Count	424	424	424
Sum	390.17	-	-
Minimum	0.35	0.00	0.00
25th Percentile	0.85	0.32	0.32
Median	1.00	2.83	2.83
75th Percentile	1.00	7.89	7.89
Maximum	2.00	190.48	40.00
Average	0.92	7.39	6.56
Weighted Mean	-	7.21	6.55
Variance	0.05	215.51	87.79
Standard Deviation	0.22	14.68	9.37
Coefficient of Variation	0.24	1.99	1.43
Skewness	0.48	6.44	2.12
Kurtosis	4.61	63.91	4.13
95th Percentile	1.11	28.40	28.40
97th Percentile	1.27	36.76	36.76
98th Percentile	1.42	45.31	40.00
99th Percentile	1.50	56.28	40.00

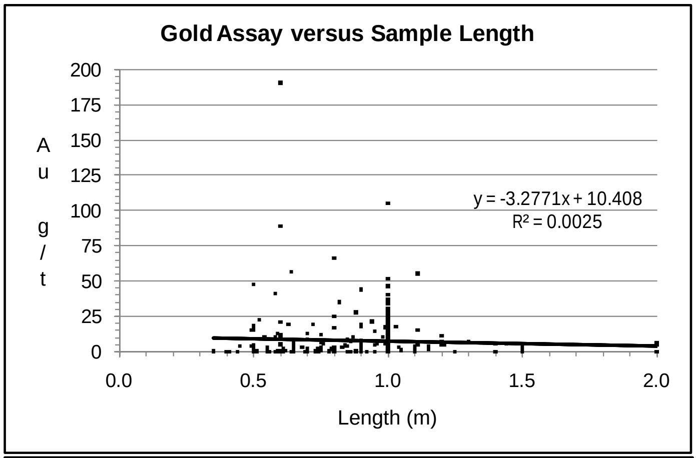
14.1.5 Compositing

Wireframes were intersected by drill holes and assays within the intersections coded. Sample lengths for assays were reviewed (Figure 14.4) and a 2.0 m composite length was selected as appropriate for the sample lengths and 2.0 m block size and to ensure that bulked out internal dilution was properly weighted in creating the composites. 100% of the core assay lengths in the resource wireframe are ≤2.0 m.

Compositing was carried out down-hole at nominal 2.0 m lengths but adjusted to equal lengths across the wireframe intercept to ensure the effect of bulking out to the minimum mining width was transferred to the composites. As such composite lengths are variable but regularization by this method is only minimally compromised.

Composite statistics are summarized in Table 14.6.

Figure 14.4 Sample Length Statistics



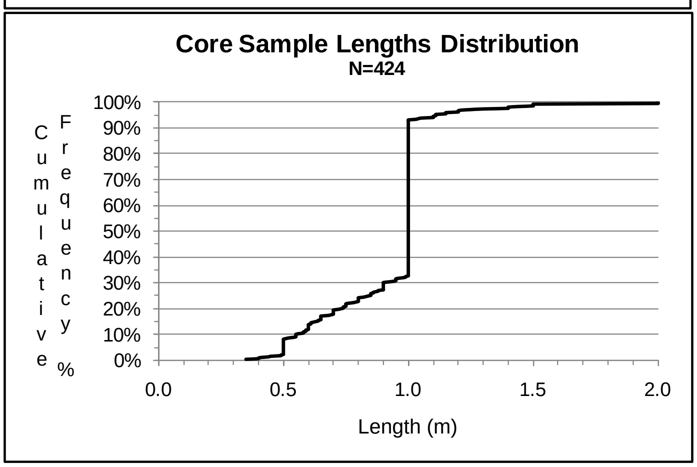


TABLE 14.6 **COMPOSITE STATISTICS**			
Statistic	**Length (m)**	**Au g/t**	**Au g/t Capped**
Count	192	192	192
Sum	388.50	-	-
Minimum	1.50	0.01	0.01
25th Percentile	1.86	2.05	2.05
Median	2.00	4.38	4.38
75th Percentile	2.05	9.23	9.23
Maximum	3.00	58.12	30.83
Average	2.02	7.35	6.64
Weighted Mean	-	7.24	6.57
Variance	0.10	77.13	42.54
Standard Deviation	0.31	8.78	6.52
Coefficient of Variation	0.15	1.20	0.98
Skewness	1.34	2.71	1.52
Kurtosis	2.34	9.82	1.95
95th Percentile	2.77	22.58	20.56
97th Percentile	3.00	28.61	22.07
98th Percentile	3.00	35.20	23.48
99th Percentile	3.00	39.13	28.34

14.1.6 Bulk Density

Underground mining has not been carried out on the Lower Trap zone and little was mined in the open pit. Consequently, there is no record of bulk densities determined from mining. Water immersion bulk density testing was carried out at Ernesto by Yamana for 627 core samples in 84 ER series holes and an additional 25 tests were performed during Knight Piesold geotechnical work in 2015, P&E due diligence sampling (6) in June 2015 and as a separate exercise by Aura personnel (8) carried out in February 2016. The resources are contained almost entirely within mylonite-muscovite schist (average bulk density 2.62 t/m^3) and quartz veining (average bulk density 2.62 t/m^3) of the Lower Trap and thus 2.62 t/m^3 was employed as bulk density for conversion of resource volume to resource tonnes. P&E notes that the bulk density range is broad for Lower Trap zone samples owing to the variable porosity of the host units. Averages and ranges in bulk density for the rock types tested are shown in Table 14.7.

TABLE 14.7 ROCK UNIT BULK DENSITIES			
Rock Type	**Unit Code**	**Average Bulk Density t/m^3**	**Range of Bulk Density t/m^3**
Sericite Schist	3	2.67	-
Quartzite	4	2.69	2.62 - 2.76
Meta-arenite	5	2.66	2.55 - 2.78
Metaconglomerate	6	2.66	1.99 - 2.90
Feldspathic Meta-Arenite	7	2.58	2.33 - 2.75
Mylonite/Schist Lower Trap Mineralization	8	2.62	1.96 - 2.98
Quartz Vein/Lower Trap Mineralization	9	2.62	2.47 - 2.88
Tonalite	10	2.66	2.33 - 2.85
Metabasalt	11	2.64	2.52 - 2.70
Mineral Wireframe	-	2.62	-

14.1.7 Trend Analysis and Variography

Grade, vertical thickness and grade-thickness contouring of wireframe drill hole intersections was carried out in plan. The contours of the grades and grade-thickness did not disclose any preferred trends other than strike and dip (Figures 14.5 to 14.7).

Down-hole linear semi-variograms were prepared for the ±2.0 m composites and assays to determine the nugget effect at a number of lag distances (Table 14.8). The resulting profiles were pure nugget with no apparent variation of 8 (h) with distance down-hole. This likely arises due to the relatively low number of samples and to bulking out to minimum mining thickness that incorporated low grade and result in high grade adjacent to low grade locally. Variance normalized 3D semi-variograms, based on spherical modelling, were then prepared for strike and dip and for the major axis of maximum continuity. Apparent nugget effect for the 3D variography was in the order of 50% to 70%. Best direction of continuity was some 20° N of the dip direction at 040°/-23°. The intermediate axis and minor axes are set from the major axis plunge with intermediate axis at 130°. Kriging profiles in normalized GEMS format were prepared from the latter semi-variograms for gold grade interpolation with overall nugget at 56%. The variography is not particularly robust due to a low number of samples but is adequate in P&E's opinion. Semi-variograms are presented in Appendix 1.

TABLE 14.8 SEMI-VARIOGRAM RANGES	
Vector	**Range (m)**
DownHole Linear	15-30
3D Strike (335°/0°)	47-64
3D Dip (065°/-20°)	60-80
3D Major Axis-Maximum Continuity (040°/-23°)	79-85
3D Intermediate Axis (130°/-43°)	45
3D Minor Axis (048°/-13°)	15-30

14.1.8 Block Model

The resource block model was set-up to encompass the mineralized wireframed areas and fringing drill holes (Table 14.9). The block size of 10 m x 10 m x 2 m vertical represents a workable size for benching mining methods and the zone widths as well as being approximately 1/3 of the detailed drill hole spacing at 35 m spacings. Model rotation is GEMS convention whereby the X axis is rotated counter clockwise to 95°. The model does not extend to surface to conserve memory for the lithology block model. The model orientation is not optimal for the strike and dip of the Lower Trap zone but is fitted to the drilling grid and facilitates visualization of the blocks and drill holes on-screen. A separate block model was created with the same XY origin, block sizes and orientation but with elevation at 450 m to extend the lithology model to the surface for mine access layout.

TABLE 14.9
RESOURCE BLOCK MODEL PARAMETERS

	X (Col)	Y (Row)	Z (Level)
Origin	257,375	8,303,715	400
Block Size (m)	10	10	2
No. of Blocks	80	135	212
Distance (m)	800	1,350	424
Rotation°	-5		
Total Blocks	2,289,600		
Volume (m^3)	457,920,000		

Figure 14.5 Grade Contours (Au g/t) in Plan View

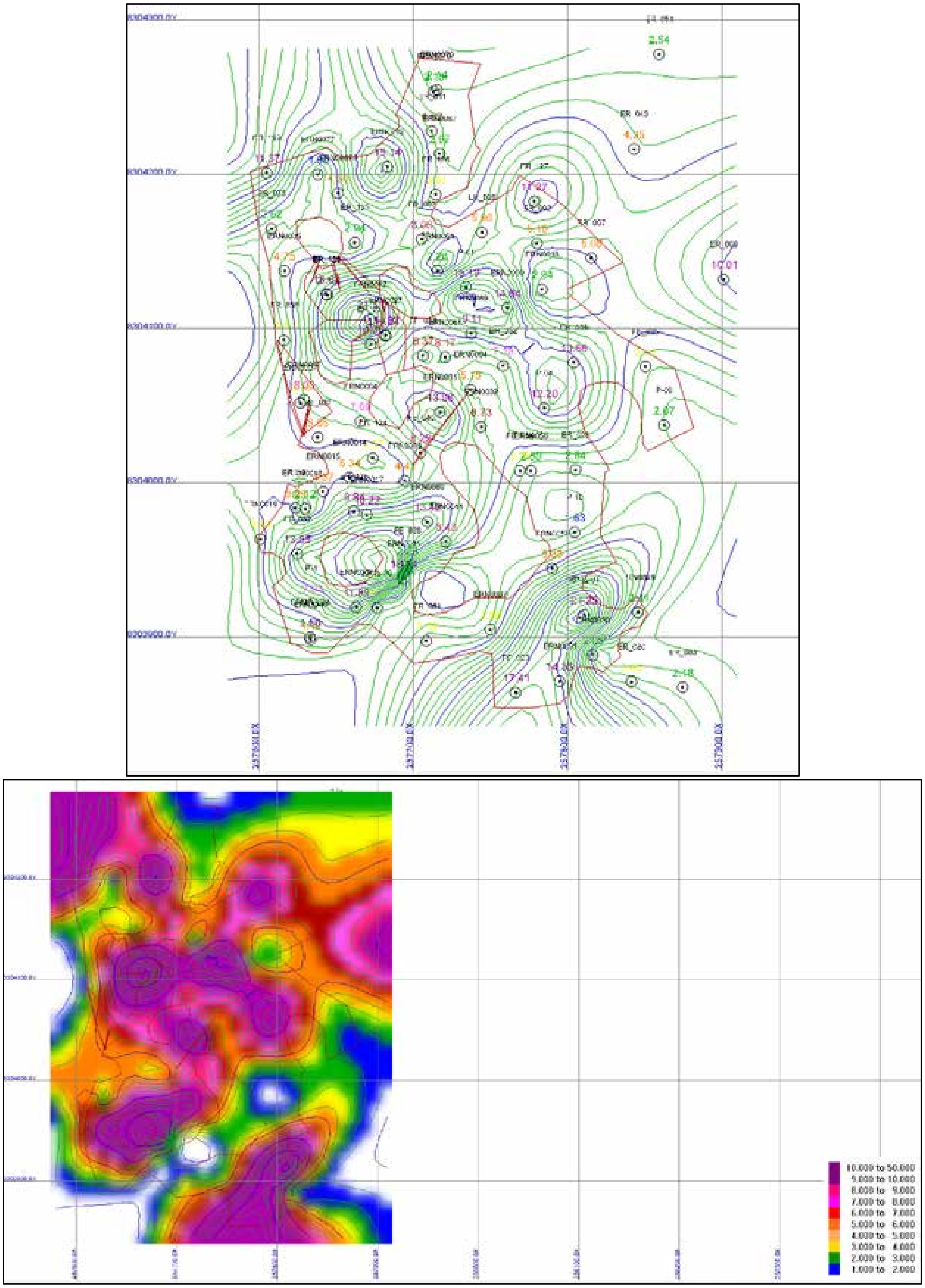


Figure 14.6 Vertical Width Contours (m) in Plan View

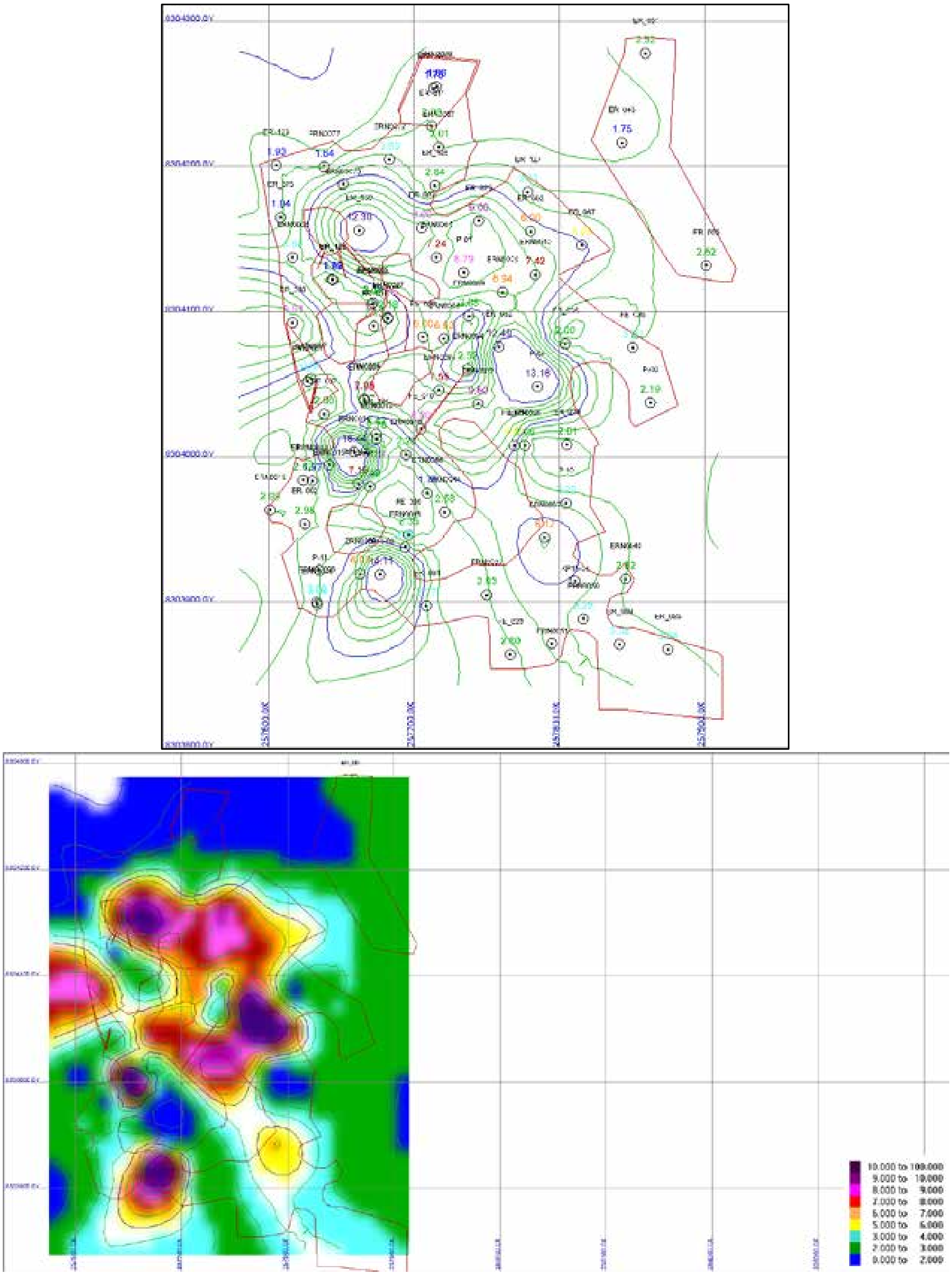
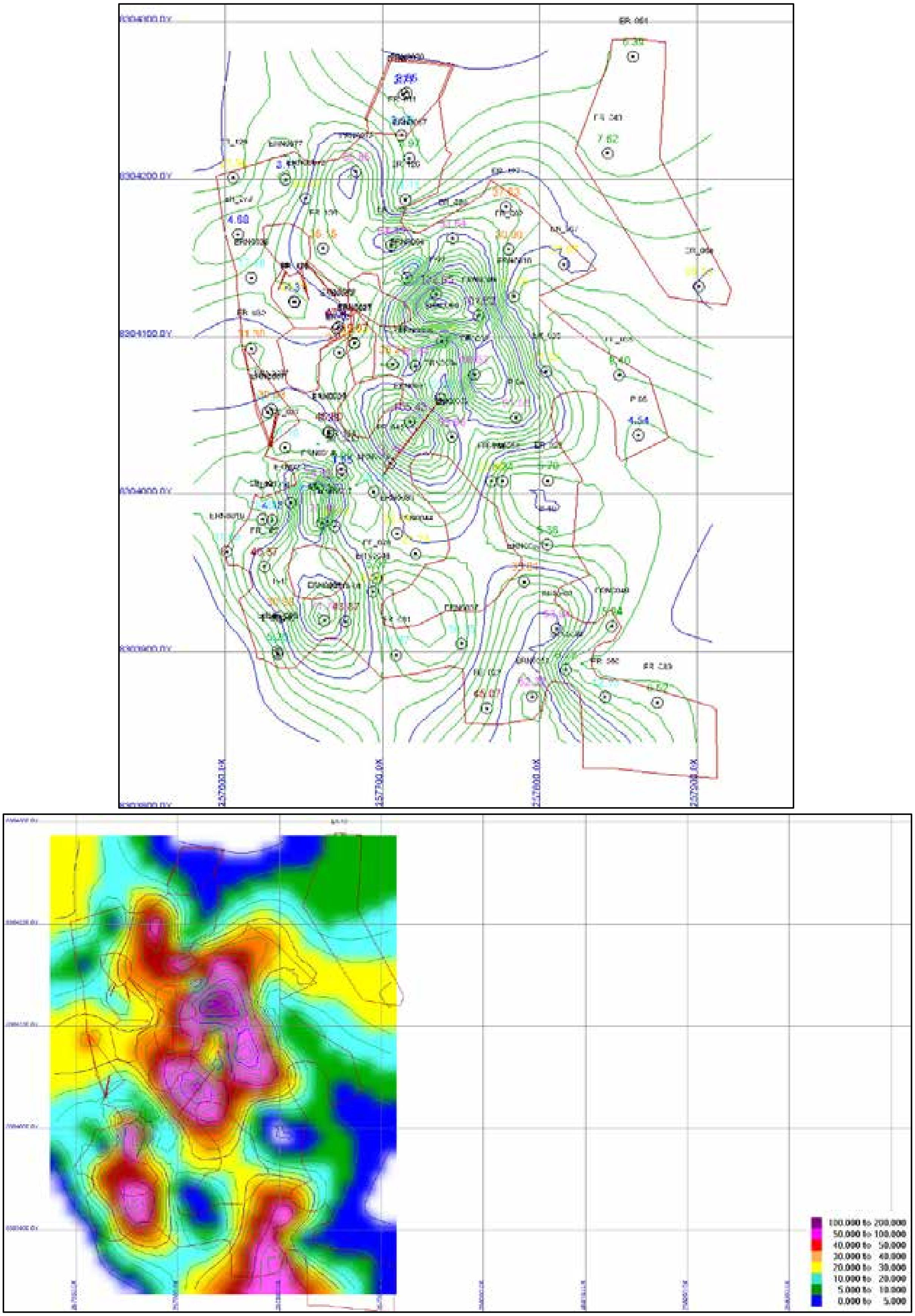


Figure 14.7 Grade-Thickness Contours (Au g/t-m) in Plan View



14.1.9 Search Strategy and Interpolation

The search strategy (Table 14.10) was designed for the anisotropic capture of an adequate number of composites in the higher density drilled area (≤35 m) and to preserve local grade diversity (i.e. not over-smooth) given a 56% nugget effect and use of Ordinary Kriging ("OK") to decluster the irregular hole pattern. The search was based on variography results and oriented by azimuth-dip-azimuth rather than by reference to the block model. The initial two interpolation passes were designed at half the variogram range but sufficient to capture holes on at least two cross sections. The wireframe in the well-drilled area, later classified as Indicated Resource, was assigned rock code 100. The Inferred Resource model was assigned rock code 200 and the two domains interpolated separately to avoid smearing across barren areas between the domains.

TABLE 14.10
SEARCH STRATEGY AND INTERPOLATION PARAMETERS

Search Ellipse	K1/K1N	K2/K2N	K3/K3N	
Azimuth1°	040/040	040/040	040/040	
Dip°	-23/-5	-23/-5	-23/-5	
Azimuth2°	130/130	130/130	130/130	
X (m)	40	80	160	
Y (m)	23	45	90	
Z (m)	8	16	32	
Pass	**Minimum#**	**Maximum#**	**Maximum# per Hole**	**Ellipse**
1	4	12	3	K1/K1N
2	2	12	-	K1/K1N
3	2	12	-	K2/K2N
4	1	12	-	K3/K3N
5	1	1	-	NN

Grade interpolation was carried out by OK in four passes with alternate check estimation methods by ID2 and NN. No declustering was done for ID2, however, results are close globally for the models. Most of the interpolation (79%) was completed in the first and second pass using the 40 m x 23 m x 10 m ellipse. 97% of the Indicated resource blocks were populated by the fourth pass. A fifth pass by NN interpolation ensured complete population of the Inferred Resource wireframes. Interpolation parameters, including number of composites used, number of holes used, distance to the nearest composite, kriging variance and interpolation pass were recorded in the block model for review and model validation.

The distribution of block grades is shown in plan in Figure 14.8 and for vertical cross sections 15N and 17N in Figures 14.9 and 14.10.

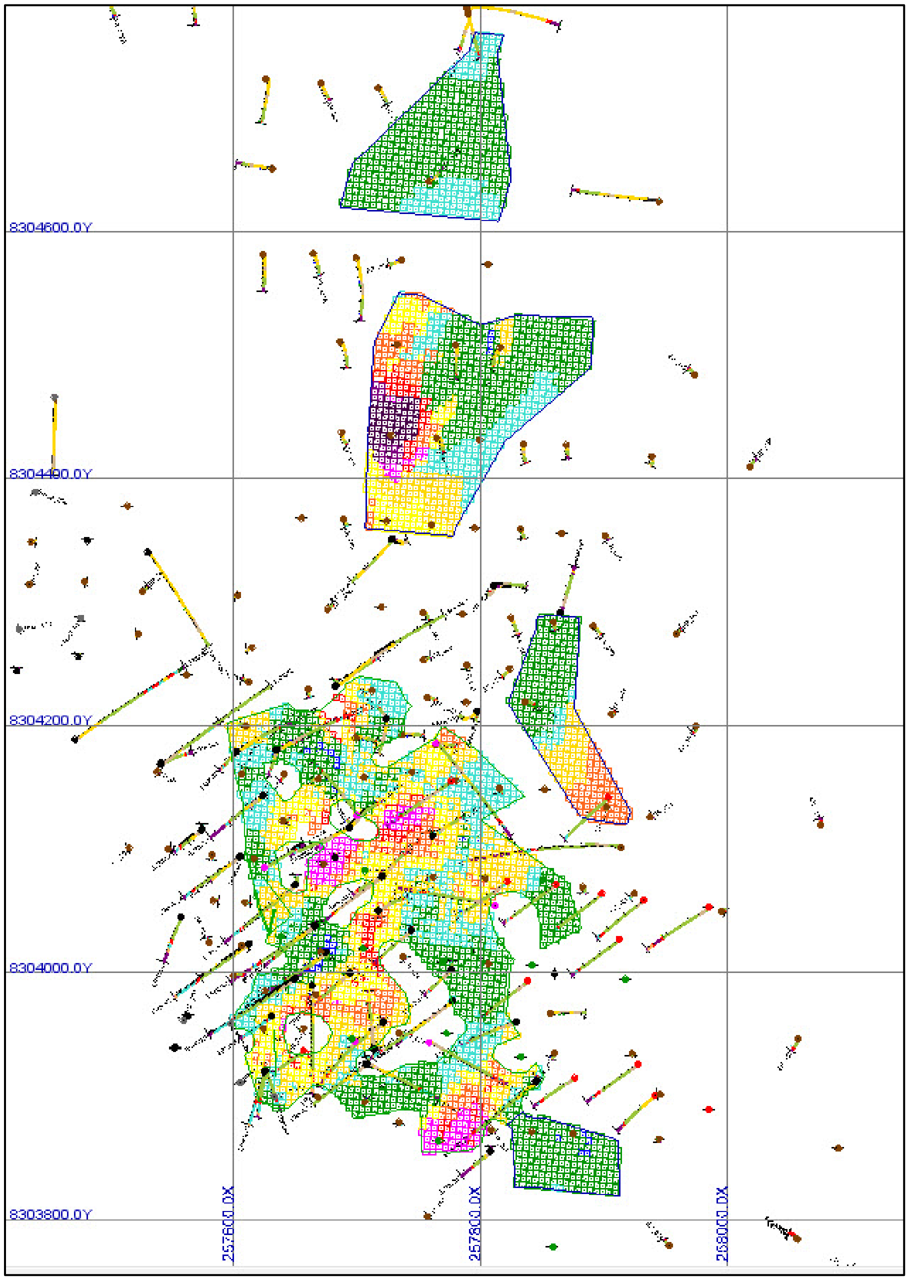
14.1.10 Resource Classification

Resource block classification was based on a review of interpolation parameters, variogram ranges and kriging variance versus distance to the nearest composite with respect to drill hole/sampling density (Figure 14.11). Most of the area of ±35 m grid drilling is classified as an Indicated Mineral Resource. The satellite areas down dip to the east and north are tested by only a few wide spaced holes and are classified as an Inferred Mineral Resource together with a small area to the SE of the Indicated Mineral Resource area where the resource is somewhat isolated from the main drilling area by barren to low grade drill intersections and is tested by only two holes. Most the main area (79%) was interpolated after Pass 2 with a search distance of 40 m, where the drill hole spacing is 35 m or less. P&E notes that kriging variance, which is a reflection of sampling geometry, versus the distance from a block to the nearest composite, is commonly used to support classification (Figure 14.12). In this case, the distance suggested is approximately 50 m and is somewhat shorter than the variography indicates, which is a reflection of 3D spatial irregularity in the intersections.

14.1.11 Model Validation

Validation of the grade interpolation and the block model was carried out by on-screen review of grades and other block model estimation parameters versus drill hole composites, review of the volumetrics of wireframes versus reported resources, by comparison of assays, composites, zone intersections and block grades, and by comparison to alternate ID2 and NN interpolations on a global basis (Table 14.11). P&E notes that assay, composites, intercept grades and block model global grades are quite comparable indicating no significant bias and thus validate the OK model.

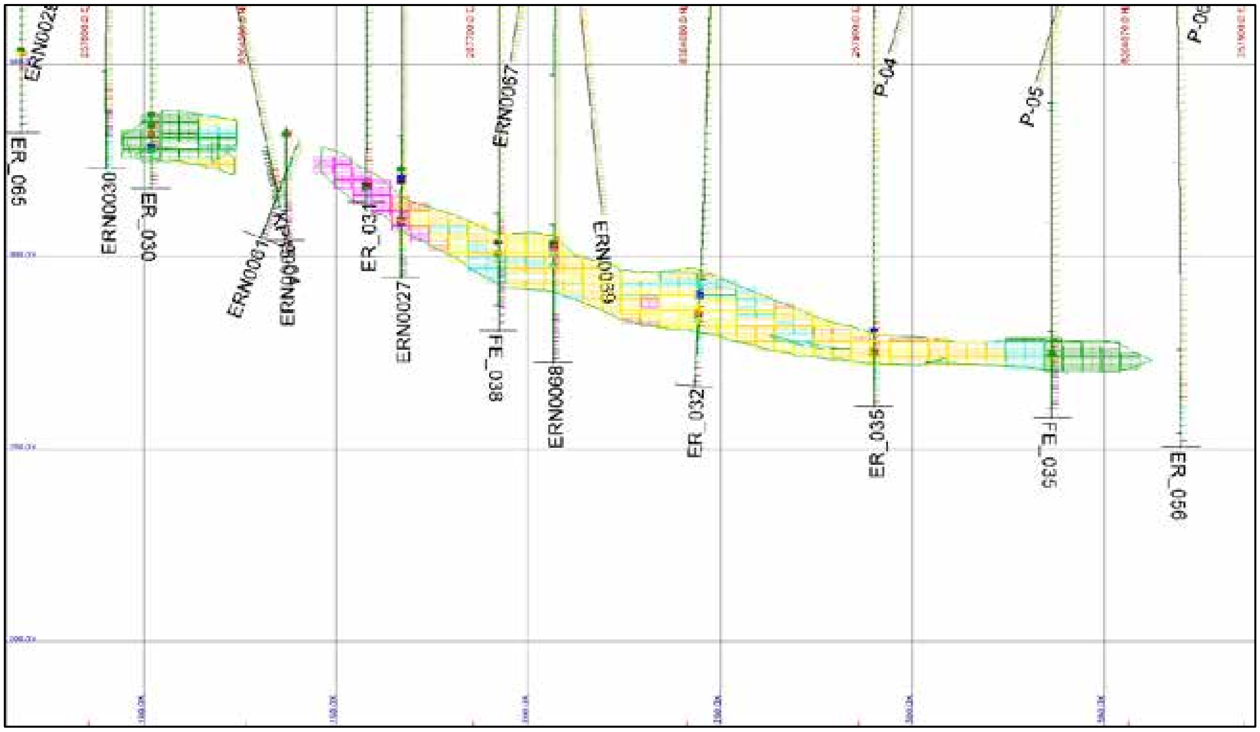
Figure 14.8   Distribution of Block Grades in Plan View



Legend (Au g/t)

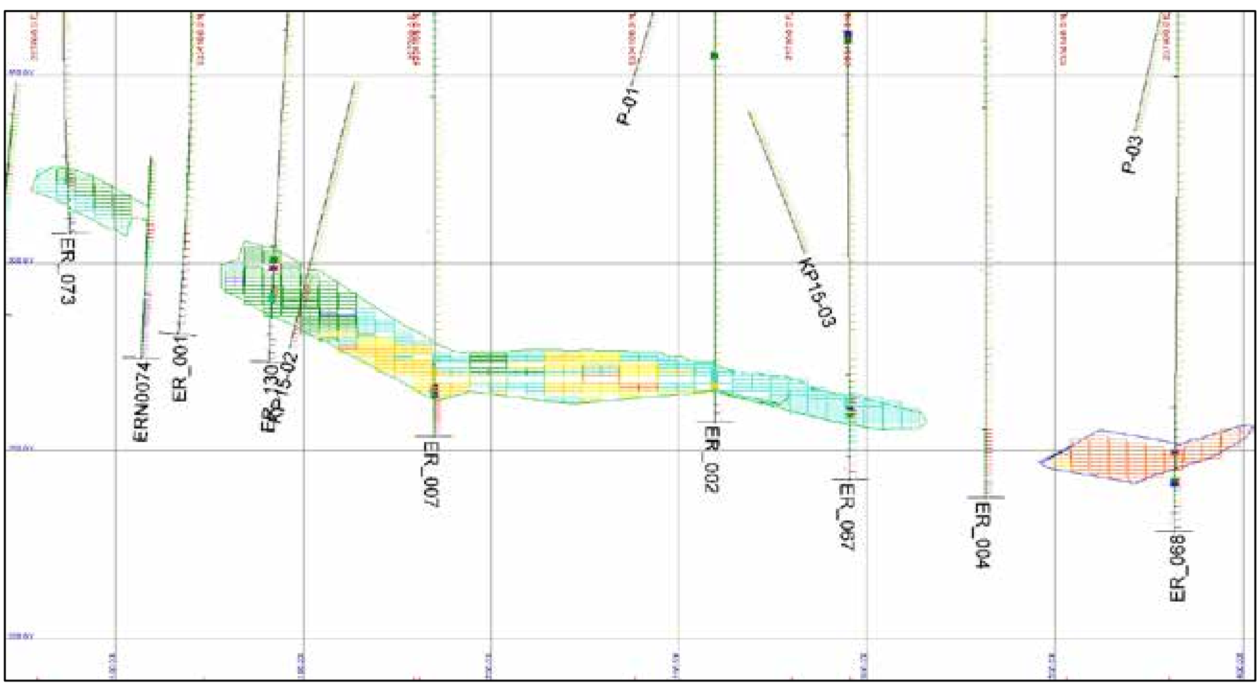
>= Lower Bound	< Upper Bound	
1.50000	2.00000	
2.00000	4.00000	
4.00000	6.00000	
6.00000	8.00000	
8.00000	10.00000	
10.00000	12.00000	
12.00000	14.00000	
14.00000	16.00000	
16.00000	10.00000	
18.00000	20.00000	
20.00000	100.00000	

Figure 14.9 Block Model Cross Section 15N (looking N)



Notes:
(1) Section corridor 24 m, not all holes on section are shown
(2) See Figure 14.8 for legend

Figure 14.10 Block Model Cross Section 17N (looking N)



Notes:
(1) Section corridor 24 m, not all holes on section are shown
(2) See Figure 14.8 for legend

Figure 14.11 Resource Classification Block Model in Plan View

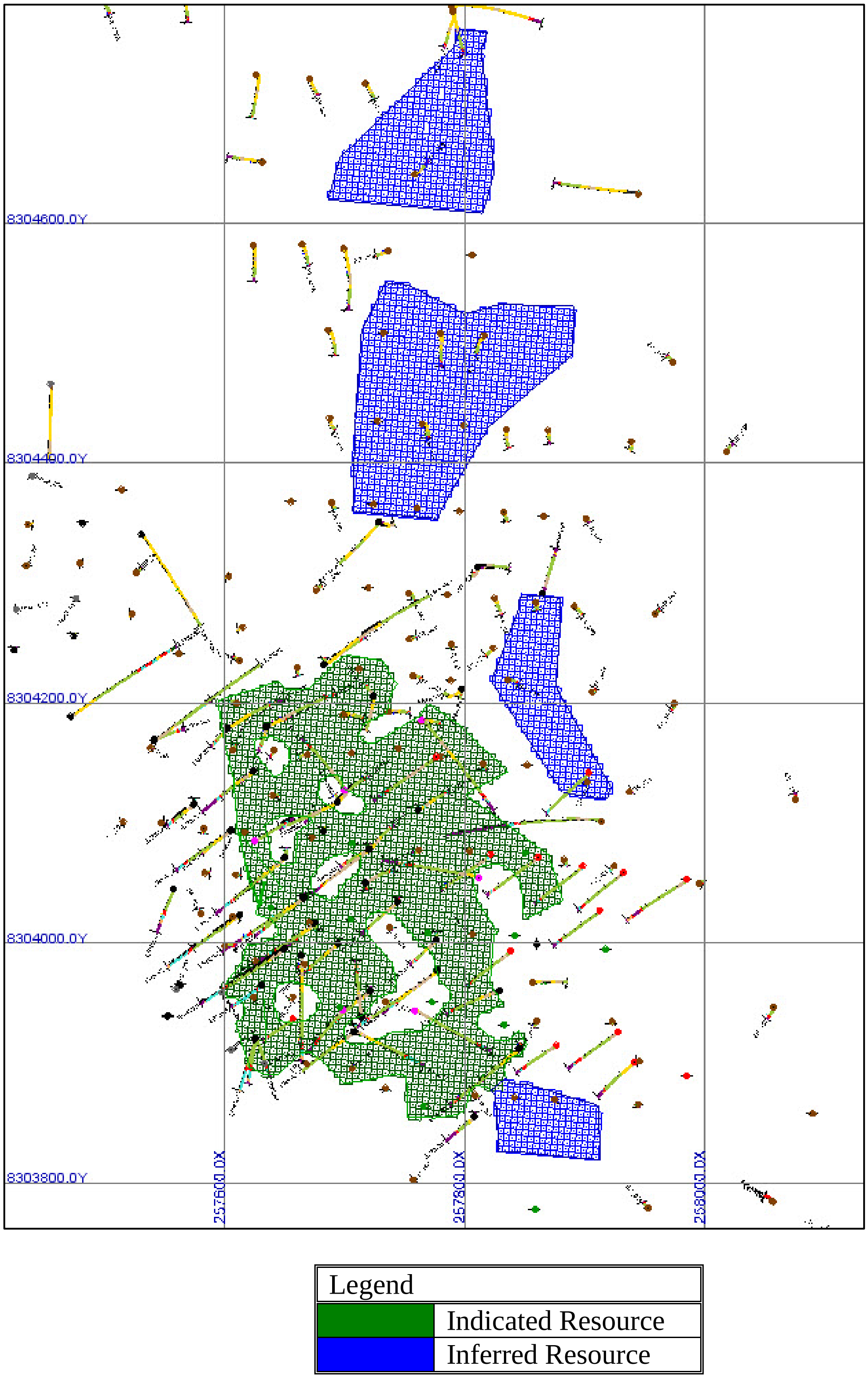


Figure 14.12 Kriging Variance versus Distance to the Nearest Composite

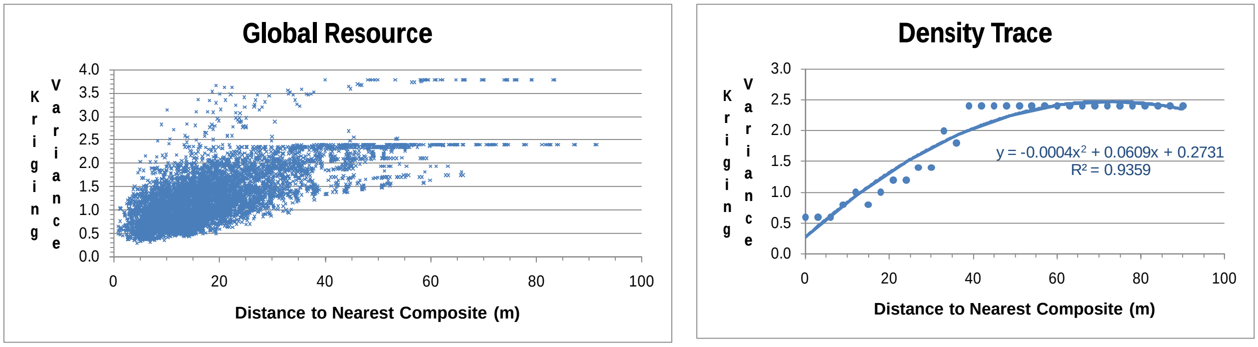


TABLE 14.11
BLOCK MODEL VALIDATION

Global Volumetrics (m³)				
Wireframes	**Reporting**	**Variance**		
398,371	398,442	0.02%		
All Zones	**Au g/t**	**Variance[1]**		
Assays	6.55	-0.3%		
Intercepts	6.53	-0.6%		
Composites[2]	6.64	1.1%		
Block Model	6.57	-		
Method	**Tonnes**	**Grade**	**Ounces**	**Variance[1]**
OK	1,043,918	6.57	220,507	-
ID²	1,043,918	6.41	215,137	-2%
NN	1,043,918	6.75	226,548	3%

Notes:

1) variance on ounces with respect to the OK block model

2) Includes explicit and implicit missing assays at zero grade

14.1.12 Resource Reporting

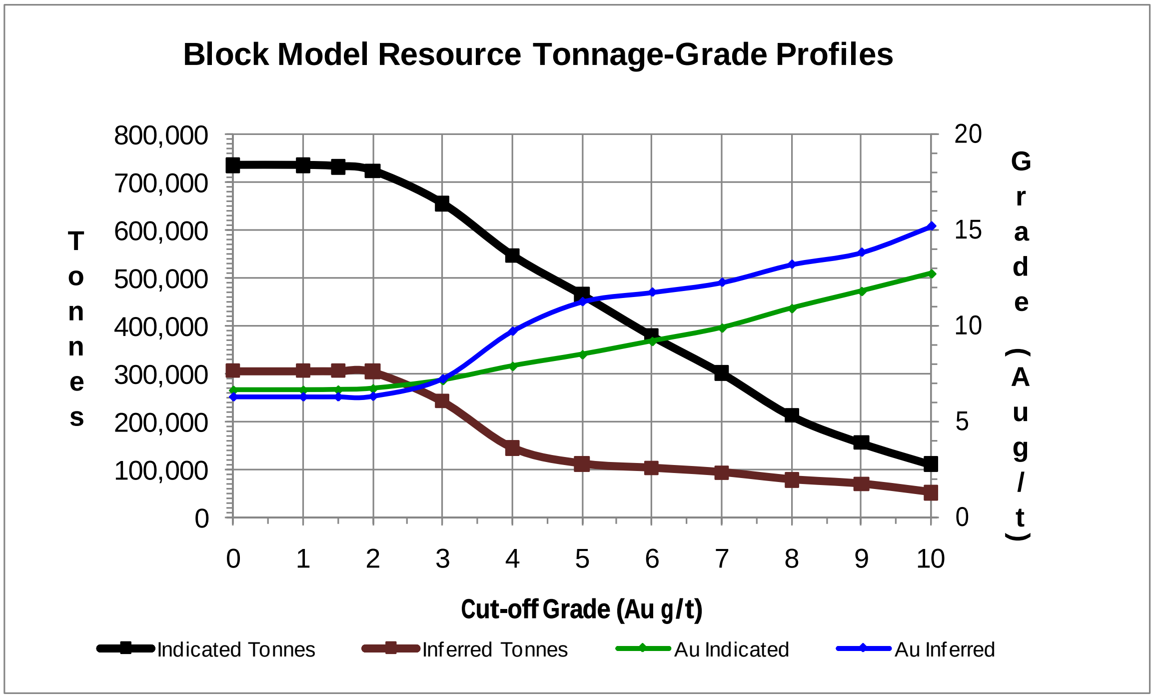
The Ernesto Mineral Resource estimated at a cut-off grade of 1.5 g/t Au is summarized in Table 14.12. Resource sensitivity to cut-off grade is presented in Table 14.13 and tonnage-grade profiles for Indicated and Inferred Mineral Resources are shown in Figure 14.13.

Table 14.12 **Ernesto deposit Lower Trap Zone Underground mineral Resource estimate At A Cut-Off Grade Of 1.5 G/T Au(1-10)**			
Resource Category	**Tonnes (t)**	**Au (g/t)**	**Contained Au oz**
Indicated	734,000	6.70	158,200
Inferred	308,000	6.30	62,400

(1) *CIM Definitions were followed for Mineral Resources.*
(2) *The Qualified Person for this Mineral Resource Estimate is: Richard Routledge M.Sc. (Applied), P.Geo.*
(3) *Mineral Resources are estimated from surface diamond drilling and core sampling by conventional 3D block modelling based on wireframing at a 1.5 g/t Au cut-off grade and ordinary kriging grade interpolation.*
(4) *For the purpose of resource estimation, assays were capped at 40 g/t Au.*
(5) *The mineral resource estimate is based on a Cut-Off Grade of 1.5 g/t Au derived from an Au price: US$1,275 /Oz, costs of US$33/t for mining, US$11/t for processing and US$10/t for G&A, at a 93% process recovery.*
(6) *A bulk density model based on rock type was used for volume to tonnes conversion with resources averaging 2.62 tonnes/m^3*
(7) *Mineral Resources are estimated from the 380 m EL to the 96 m EL, or from approximately 50 m depth to 150 m depth from surface.*
(8) *Mineral Resources are classified as Indicated and Inferred based on drill hole spacing, interpreted geologic continuity and quality of data.*
(9) *Mineral Resources, which are not mineral reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*
(10) *The quantity and grade of reported Inferred Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as an Indicated or Measured Mineral Resource and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured Mineral Resource category.*

Table 14.13 **Ernesto Mine Lower Trap Zone Underground mineral Resource Sensitivity To Cut-Off Grade**						
Cut-Off Grade	**Indicated**			**Inferred**		
	Tonnes	**Au**	**Au**	**Tonnes**	**Au**	**Au**
Au g/t	**(000's)**	**g/t**	**oz**	**(000's)**	**g/t**	**oz**
10.0	113	12.75	46,200	53	15.22	26,000
9.0	157	11.83	59,800	71	13.85	31,500
8.0	214	10.94	75,100	80	13.22	34,100
7.0	303	9.92	96,600	95	12.28	37,600
6.0	380	9.23	112,800	104	11.77	39,500
5.0	466	8.54	127,900	113	11.27	41,100
4.0	549	7.93	139,900	147	9.73	45,900
3.0	657	7.20	151,900	244	7.24	56,800
2.0	725	6.77	157,600	306	6.33	62,300
1.5	734	6.70	158,200	308	6.30	62,400
1.0	736	6.69	158,300	308	6.30	62,400

Figure 14.13 Graphic Illustration of Ernesto Resource Sensitivity to Cut-Off Grade



To assess the grade sensitivity to the modelling and overall smoothing, the internal waste areas of the Indicated Resource wireframe were included in a new solid and the tonnage and grade re-estimated for the Indicated Resource area. The modelling was also done to estimate the potential grades of development muck in the internal waste areas. The modelling of the resource mineralization trends in the waste areas was not optimized for grade and widths but merely projected through the waste areas from adjacent resource intersections (Figures 14.14 and 14.15). Table 14.14 compares results and shows the impact as an 8% gain in tonnes, 10% reduction in grade and 3% loss of gold ounces at the 1.5 g/t Au cut-off grade. P&E recommends using this impact as a downside sensitivity scenario during Project cash flow analysis.

TABLE 14.14
COMPARISON OF ESTIMATES AND MODELLING SENSITIVITY AT 1.5 G/T AU CUT-OFF GRADE

	Tonnes	Au g/t	Au oz
Waste Areas Included	790,000	6.06	153,900
Mine Plan Resource	734,000	6.70	158,200
Variance	+56.551	-0.65	-4,226
Variance%	+8%	-10%	-3%

Figure 14.14 Plan View of Resource Wireframe Showing Waste Drill Holes

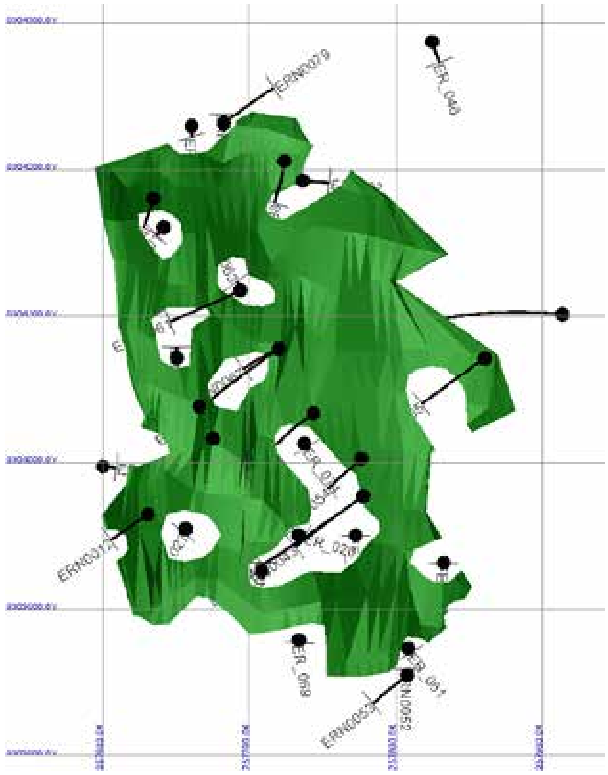
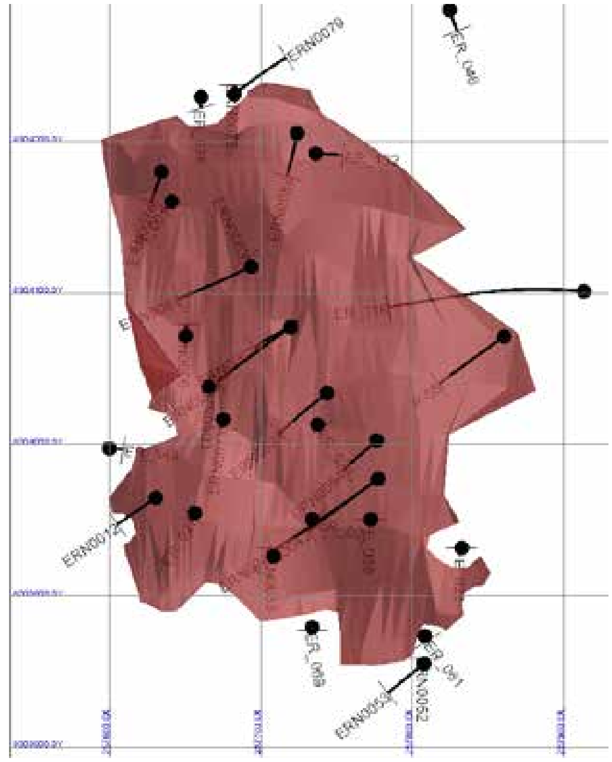


Figure 14.15 Plan View of Sensitivity Wireframe Showing Waste Drill Holes



14.1.13 Recommendations

A number of holes were drilled and cut the resource wireframe but were not sampled. Some holes were sampled up hole but not in the Lower Trap zone. The few holes with non-sampled explicit intervals within the zone also need to be sampled. Two holes did not reach the Lower Trap zone and should be deepened if possible. Holes in question are:

- ERN0076
- ERN0080
- ERN0084
- ERN0088
- ERN0074
- ERN0062
- ERN0078
- ERN0093
- ERN0098
- ERN0089 and relog bottom of hole
- ER059 and ER091 should be deepened by 50 m and 90 m respectively.

Several revisions of the drill hole and assay databases during the course of resource estimation for Ernesto indicated that the database received from Yamana had been incompletely verified and needs review and cross-referencing with original records for the drill hole and assay databases. This has been done for the Indicated Mineral Resource portion of the database but not for the fringing areas including the Inferred Mineral Resources. Down hole surveys should be thoroughly reviewed against original survey records since it is difficult to validate survey accuracy of azimuths in subvertical drill holes by simple on-screen reviews and routine examination. Completion of the relogging underway to update the lithology database should be completed for drill holes in the Inferred Mineral Resource areas.

Modelling of a lower grade envelop in the Inferred Mineral Resource area in the northern part of the Property is recommended to better understand geometry-continuity of the mineralized zone.

14.2 LAVRINHA MINERAL RESOURCE ESTIMATE

14.2.1 Drill hole Database

The Lavrinha drill hole data, which was stored and managed by Aura in a corporate SQL server utilizing Acquire software, originated from different drill campaigns that were determined by MCB to be in compliance with NI 43-101 quality control checks and data storage policies.

The raw data was exported to text files for data manipulations, population statistics, geological/alteration/resource modeling and grade estimation. MCB's opinion is that the drill hole database including Au (g/t) grades and bulk density determination (t/m^3) are valid and suitable for estimating mineral resources.

The dataset is comprised of three drilling campaigns with their respective objectives. The Lavrinha Deposit was linked to the Ernesto Deposit exploration strategies since it was considered to be its satellite deposit, therefore the same operational procedures, documentation and database management were applied.

14.2.2 Geological 3D Domain Modeling

The Lavrinha Deposit is located approximately 500 m west of the Ernesto Mine with a strike of approximately 50°NE extending approximately 400 m along strike and 500 m down plunge. It is characterized by a swarm of parallel veins, subparallel to the plunge. Mineralized lodes occur with variable thickness up to 12 m and are distributed within a thick sericite-muscovite schist unit. The maximum depth from surface that mineralization is intersected by drill holes is approximately 150 m.

Prior to geological 3D domain modeling, a gold grain study was performed within the alteration package. The gold occurrence hosted by schist and hanging wall fine meta-arenite domains, is associated with sulphide pseudomorphs and quartz veins and veinlets. The occurrence of gold is more disseminated inside meta-arenites. Based on this observation, two samples as follows were acquired using ¼ core from drillhole LV015 for analysis:

- Fine meta-arenite/schist with strong hydrothermal alteration;
- Inside the meta-arenite near the footwall.

Table 14.15 presents the results of the sub-sample study, which confirmed that gold grains are related to quartz with sulphide pseudomorphs, sericitization and high silicification within schist/fine-meta-arenite. In the meta-arenite, there were small quantities of gold where quartz boudins were absent. This information was used for re-logging of drill core before 3D domain modeling took place.

TABLE 14.15
SUB-SAMPLE STUDY

Hole ID	Original Sample			Duplicate						
	From	To	Au ppm	From	To	Description	Sample	Weight (g)	Au ppm	Au ppm Calculated
LV015	121	122	9.57	121	121.3	Highly Sericitized Schist with sulphide pseudomorphs intercalated with quartz veins between 12.27 and 12.30	FS 15959	330	9.66	20.36
				121.3	121.7	Highly Sericitized Schist with sulphide pseudomorphs and altered	FS 15960	450	44.45	
				121.7	122	Highly Sericitized Schist with sulphide marks	FS 15961	370	0.603	
	256	257	3.04	256	256.85	Meta-Arenite with sericite and almost without quartz veins and sulphide	FS 15962	1,050	0.22	0.29
				256.85	257	Meta-Arenite with sericite and quartz and sulphide pseudomorphs	FS 15963	240	0.62	

Geological interpretation and mineralized lode modeling of the Lavrinha Project were carried out using Micromine® software by MCB with technical support from Aura staff. The following tasks were performed:

- Surface geological mapping to define contacts, geological and structural features;
- Re-logging and data validation of all core focusing on lithology, hydrothermal alteration, sericitization and silicification;
- Interpretation of hydrothermal and geology features using two sets of cross-sections;
- Parallel plunge (azimuth N145° / Dip Vertical) – 6 m spacing;
- Perpendicular plunge (azimuth N55° / Dip 57°SE) – 25 m spacing for further fine tuning of the model; and
- Interpretation of mineralized lodes using 0.2 g/t Au envelope and geological contacts and alteration layers as hard boundaries.

Based on the above interpretation, trench locations were identified and an appropriate drill program was planned to confirm mineralization close to surface to provide better resolution for the mineralized model. A total of 20 drill holes were completed to more fully understand the geometry of the mineralized lodes close to surface and to construct a 3D mineralized domain model.

14.2.3 Surface Geological Mapping

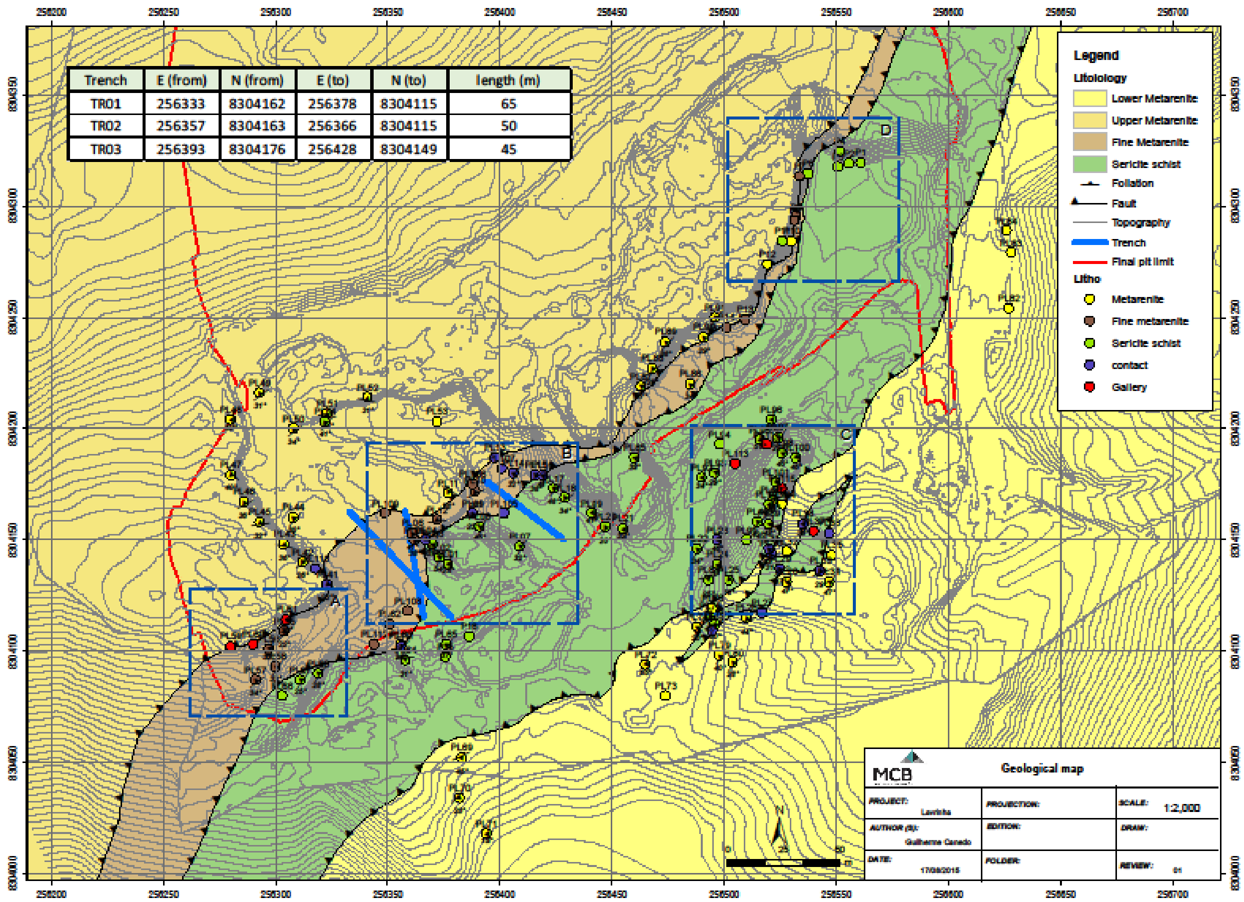
Geological mapping of the Lavrinha Deposit was performed by MCB consultant Guilherme Canedo with technical support from Aura. The objective of this mapping was to achieve a better understanding of geological controls of the mineralization and utilization as a guide for construction of the 3D mineralized domain wireframes.

Geological mapping was carried out at a scale of 1:1,000 acquiring 130 data points and information regarding geological features such as faults, fractures, mineralogy, lithology and alteration. Apparently, none of the faults that were mapped at surface have significantly displaced the mineralized lodes.

In Lavrinha, four different lithology types were identified: upper arenite, fine meta-arenite, sericite schist and lower meta-arenite (Aguapeí Group). The lithological units have a NE-SW trend and are separated by multiple shear zones.

Based on core logging review and geological information collected in the field, including three trenches, mineralization is concordant/sub-concordant with lithological layers, predominantly in two lithologies, fine meta-arenite and sericite schist (Figure 14.16).

Figure 14.16 Lavrinha Gold Deposit Geological Map with Location of 3 Trenches



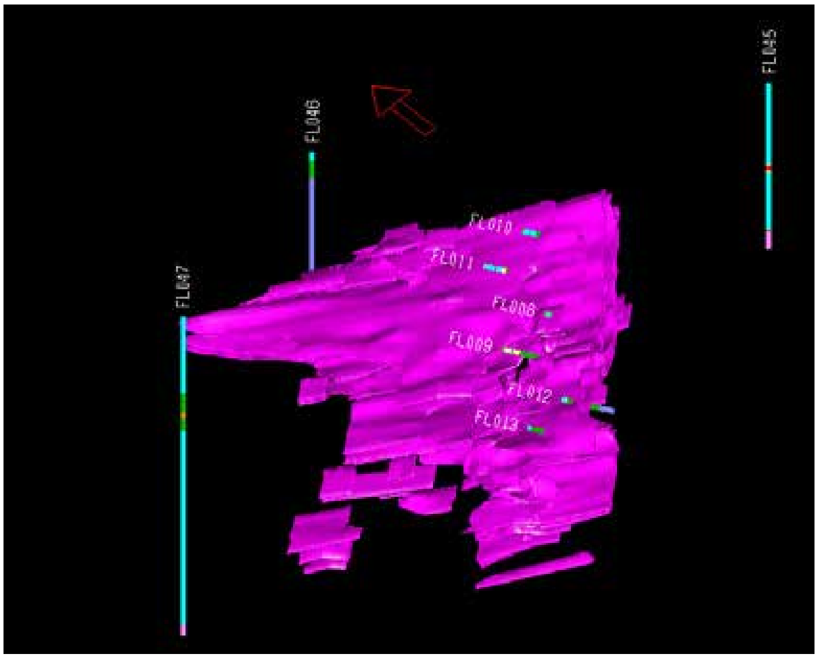
14.2.4 Core Re-Logging and Data Validation

Re-logging of drill core was part of the data validation process. During the re-logging, it was observed that most of the mineralized intervals were associated with hydrothermal alteration in the form of silica and sericite with different intensities. Considerations made in terms of re-logging of core were:

- Re-logging of a previous campaign (LVR series) was conducted based on core photos due to the core not being available to view;
- Core from the second campaign (LV series) was re-logged using all cores that were available in the core shed. This series was important particularly for the re-logging of hydrothermal alteration;
- It was not possible to re-log the first campaign holes (FL-Series) because no photos were available and just a few boxes of core remained in the core shed. It was observed in the current 3D domain model that the lithology in FL-series drill holes was very inconsistent with logging of drill holes from the other campaigns,
- FL-series holes have been excluded from the new geological and mineralization interpretation due to the following reasons:
- Core was not available;
- There was no QAQC data available to validate the assays; and
- It was not possible to check their collar locations in the field.

As a result, nine holes were excluded from the new 3D domain model, of which six were within the mineralized zone (FL-08, FL-09, FL-10, FL-11, FL-12, FL-13, FL-45, FL-46, FL-47). The spatial distribution of these holes is illustrated in Figure 14.17 by a 3D solid which represents the preliminary mineralized envelope.

Figure 14.17 Spatial Distribution of FL Drill Holes Excluded from the Geological 3D Domain Model



14.2.5 Trenches

Three trenches were excavated to define geological contacts and to allow a better understanding of the horizontal extension and continuity of the mineralization. The trenches were excavated with a backhoe and cleaned by hand to expose bedrock or caprock. One end of each trench was considered as a “drill collar” and surveyed with a total station survey instrument. From the start point, the trenches were surveyed and marked every metre and a profile was prepared for each trench.

The trenches were sampled in 2 m intervals respecting the lithological contacts. Samples were cut using a diamond blade with an average of 10 kg of material per 2 m sample. Additional care was taken to avoid sample contamination by carefully cleaning the trench walls and floors before sampling. The samples were collected in the wall in each trench and were used to gain an understanding of the distribution and continuity of mineralization.

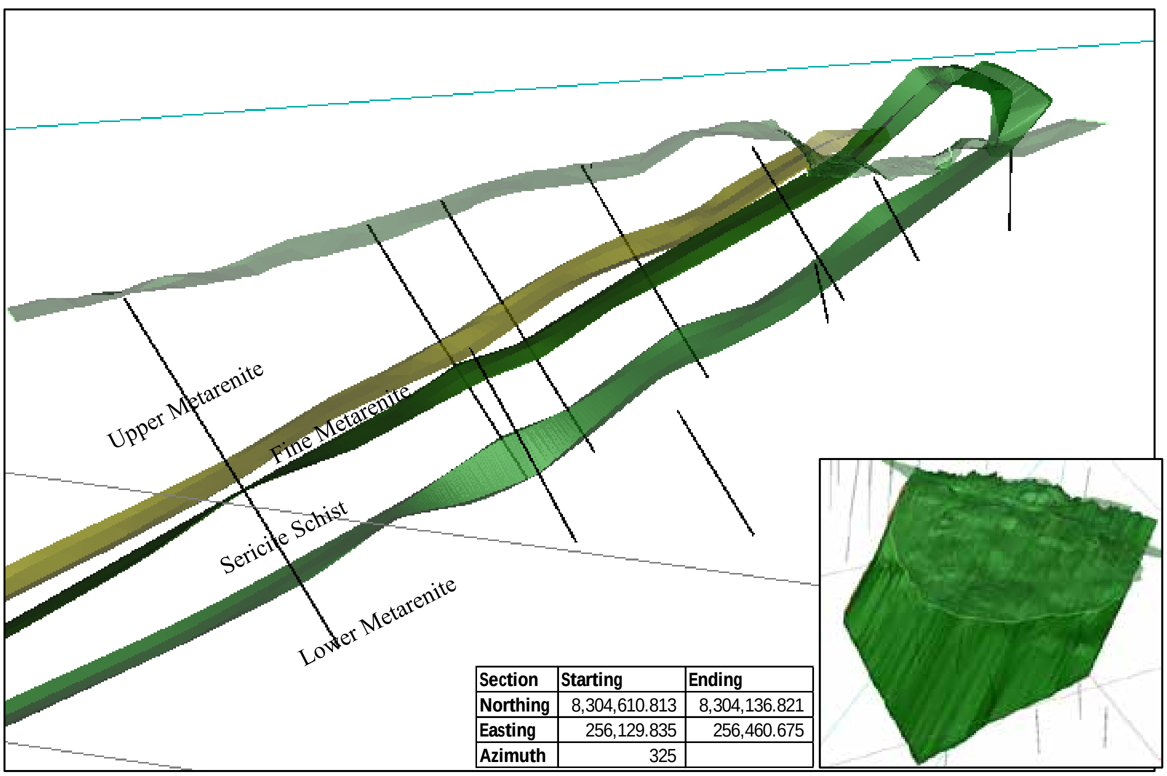
14.2.6 Geological 3D Domain Modelling

Mineralization of the Lavrinha Deposit is concordant/sub-concordant with lithological layers, predominantly in two lithologies, fine metarenite and sericite schist structures, and is associated with hydrothermal alteration zones. This approach is different from previous geological models that considered the mineralization discordant with lithology and cross-cutting sericite schist.

Geological, hydrothermal alteration and mineralized lode interpretation was carried out based on 78 vertical cross-sections along the strike direction of N320° with a 6.25 m spacing, and also irregular 25 m sections perpendicular to the plunge (inclined sections) to better define the constraining mineralized polylines to be subsequently utilized for geological 3D domain modeling.

- The sequence of geological 3D domain modeling was established as follows:
- Lithological modelling - The lithological interpretation considered four units, which consisted of the lithology types upper meta-arenite, fine arenite, sericite schist and lower meta-arenite (Figure 14.18);
- Hydrothermal alteration modeling - The hydrothermal alteration interpretation considered the occurrence of sericite and silica to define hydrothermal altered zones. These zones were confined mainly in the fine arenite and sericite schist units (Figure 14.19 and Figure 14.20);
- Mineralization modeling – The mineralized interpretation considered a 0.20 g/t Au cut-off and hydrothermal alteration and lithological models as hard boundaries. The continuity of mineralization was confined inside of the hydrothermal altered zones (Figure 14.21 and Figure 14.22).
- The polygons from these interpretations were linked to create 3D wireframe solids snapped to the drill hole intersections. These solids were checked visually on cross-sections, validated for triangulation with no inconsistency errors found.

Figure 14.18 Vertical Cross-Section of Lithological Model



Section	Starting	Ending
Northing	8,304,610.813	8,304,136.821
Easting	256,129.835	256,460.675
Azimuth	325	

Figure 14.19 Highly Hydrothermal Alteration 3D Model. Interpreted in Vertical Sections (left) and 3D Solid (right)

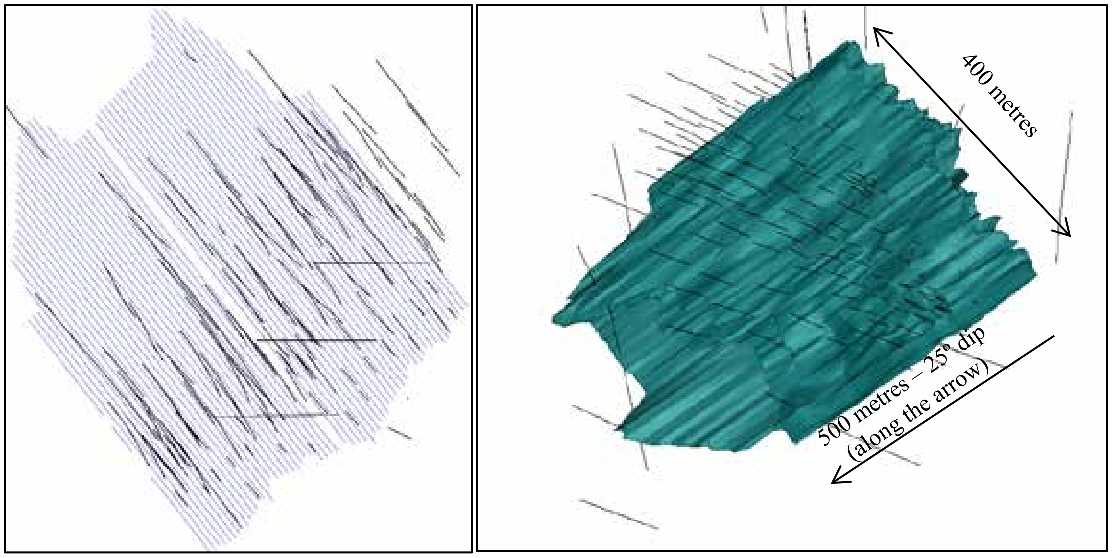
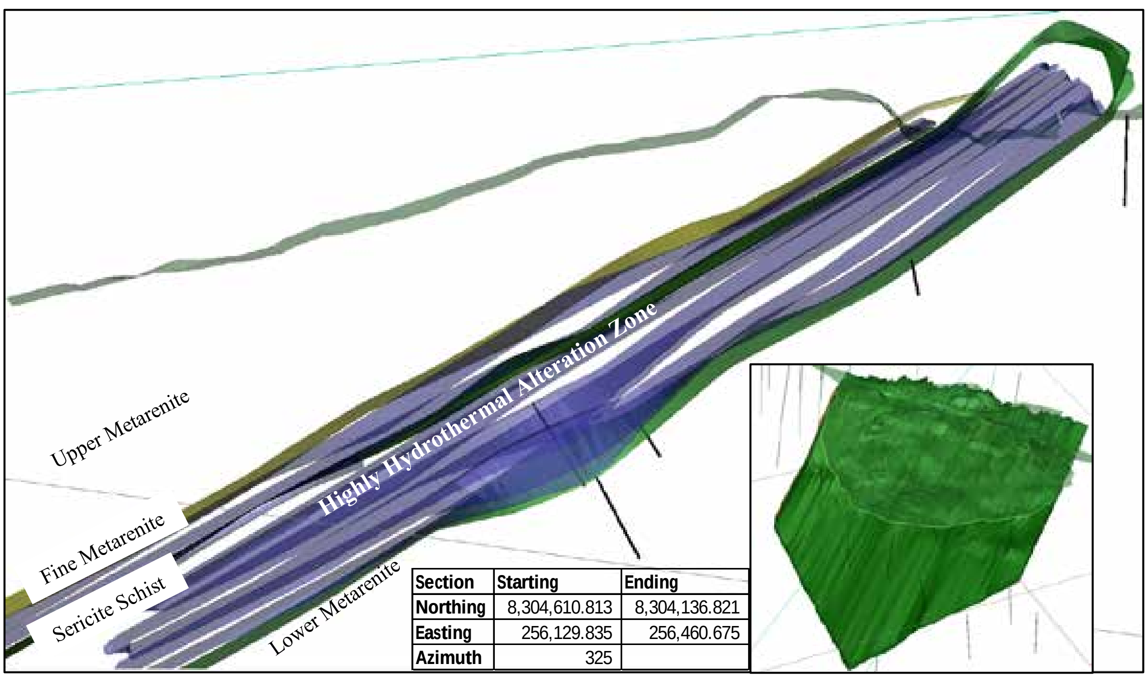


Figure 14.20 Vertical Section of Highly Hydrothermal Alteration Model



Section	Starting	Ending
Northing	8,304,610.813	8,304,136.821
Easting	256,129.835	256,460.675
Azimuth	325	

Figure 14.21 Mineralized Lode 3D model. Interpreted Vertical Sections (left) and 3D Solid (right)

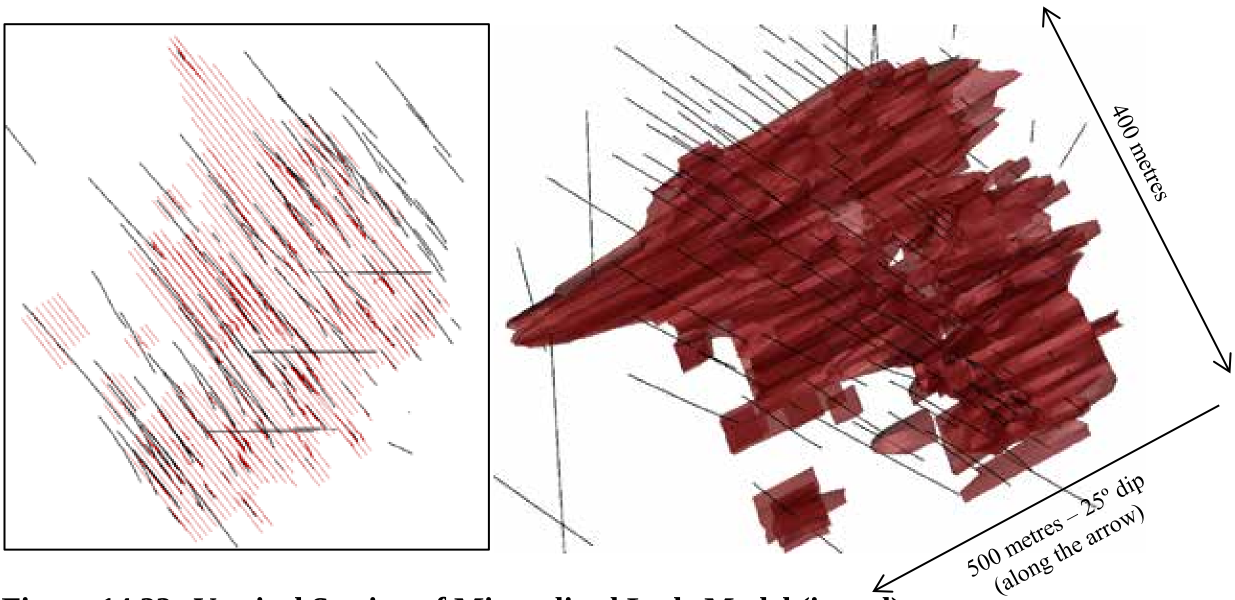
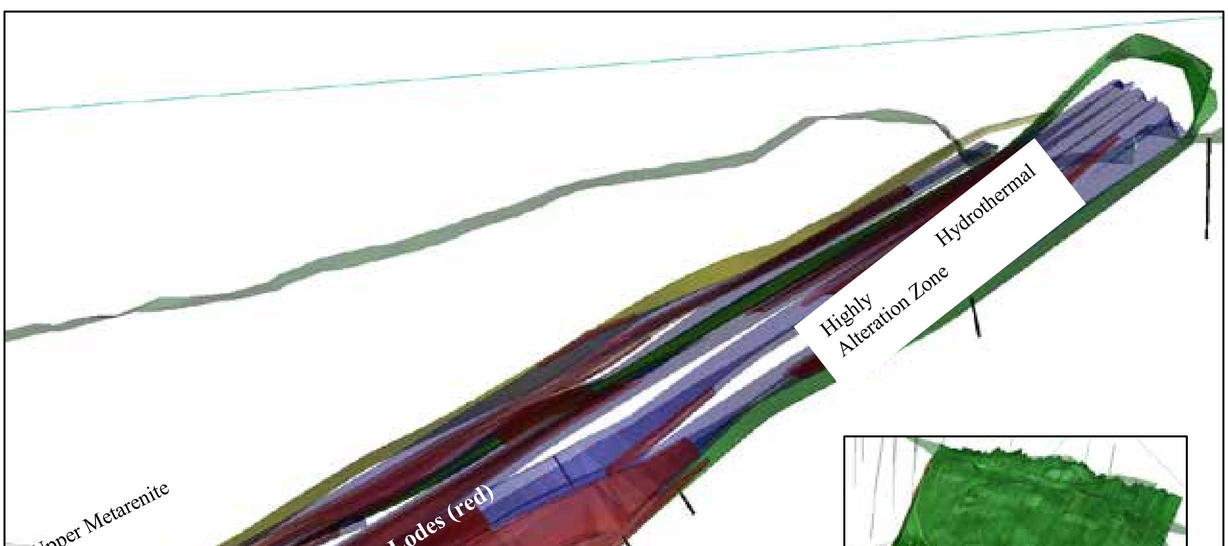


Figure 14.22 Vertical Section of Mineralized Lode Model (in red).



14.2.7 Data Analysis

Data analysis of the Lavrinha Project is an essential part of mineral resource estimation, which involves understanding the data as well as the organization of the dataset to improve the quality of estimation. This task was carried out by taking the following steps:

- Error recognition;
- Comprehensive knowledge of the statistical and spatial characteristics of all variables of interest for mineral resource estimation;

- Recognition any systematic spatial variation of variables such as grade and thickness of mineralized zones;
- Recognize and define distinctive geologic domains that must be evaluated independently for Mineral Resource Estimation;
- Identify and understand assay value outliers; and
- Evaluate similarity/dissimilarity of various types of raw data, especially samples with different origin.

The above items were relevant to all studies in particular, the identification and understanding of outliers has a significant impact on the Mineral Resource Estimation. This was discussed in detail between MCB and Aura to achieve the best outlier treatment method due to the high variability of the gold grades.

14.2.8 General Characteristics

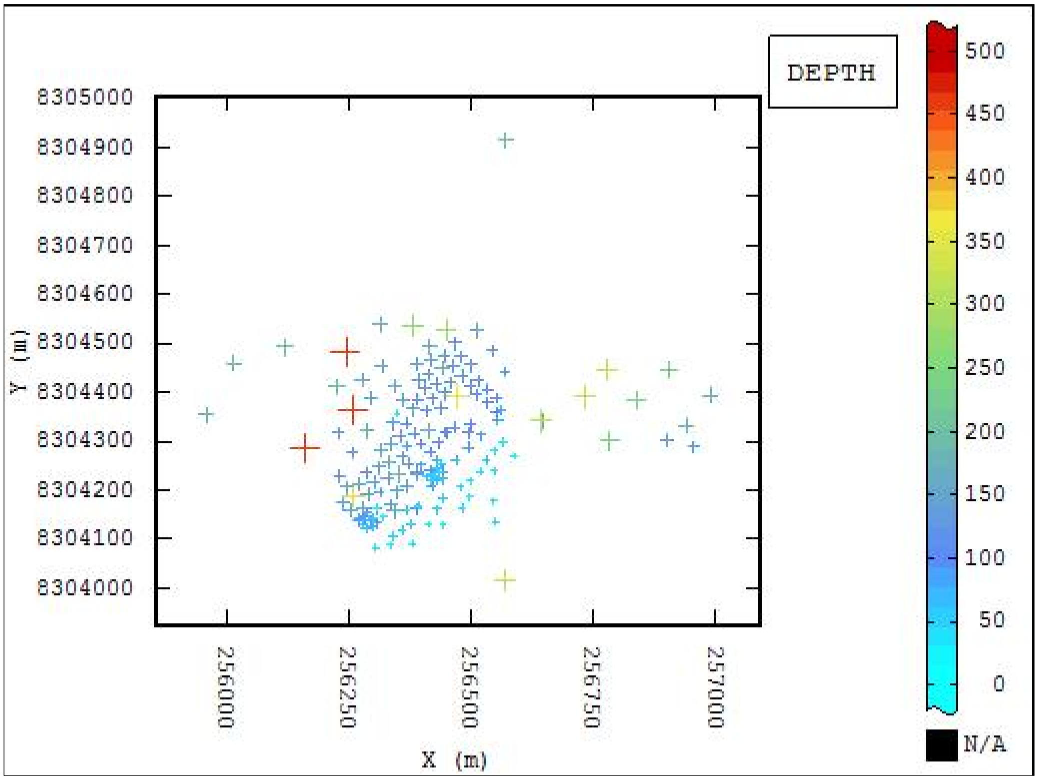
MCB undertook a series of validation and verification tests on assay data prior to the resource estimation process. Before undertaking any geostatistical analysis, a critical assessment of the available data was carried out, which did not reveal any obvious errors in data locations or assay values.

The statistical analysis of data was developed in ISATIS® v2015.2 software. The Ordinary Kriging method was applied to estimate the gold grade which was the only variable analyzed in the Mineral Resource Estimation and classification process.

14.2.9 The Dataset

Lavrinha's database stored in Access and Excel formats was provided by Aura, and totalled 165 drill holes and 20,867.41 m drilled, with a total of 20,372 samples analyzed for gold. Figure 14.23 shows a data location map (collar) in plan view (the color scale represents the total length of the drillhole).

Figure 14.23 Spatial Representation of Drillholes in Plan View



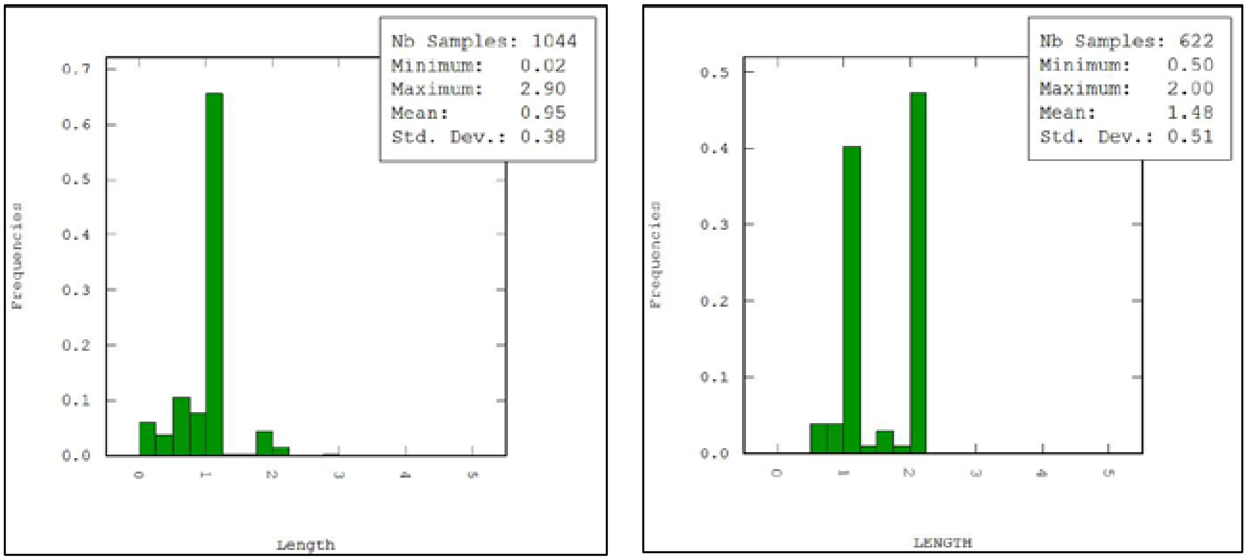
14.2.10 Compositing

The sample data of a drilling campaign is usually acquired from mineralized intervals down-the-hole. From a statistical standpoint, different lengths of samples can lead to a bias in the results. In order to unify the data set, the first hypothesis is stationary (i.e. all random variables have the same mean), assuming that the mean can be estimated by the arithmetic mean of sampled values. Thus, it is necessary to composite the data set and to unify the different sample lengths.

The dataset provided by Aura for geostatistical modeling has a total of 1,044 samples inside the mineralized zones with sample length variation between 0.02 m (snap imprecision) and 2.90 m. The methodology used for sample compositing was to create 2 m down-hole composites, starting from the collar. Short intervals less than 0.5 m were excluded to avoid bias at the ends of drill holes intersections within the geological 3D domain.

Figure 14.24 shows the lengths before (left side) and after (right side) the composites.

Figure 14.24 Histogram of Raw Assay and Composite Length Variability



Note: *The left histogram shows all non-composited assay data within mineralized domains in Lavrinha. The right histogram shows composited data excluding assay composite data with length less than 50 cm. Higher standard deviation compared to the original assays is the outcome of excluding the short composited intervals.*

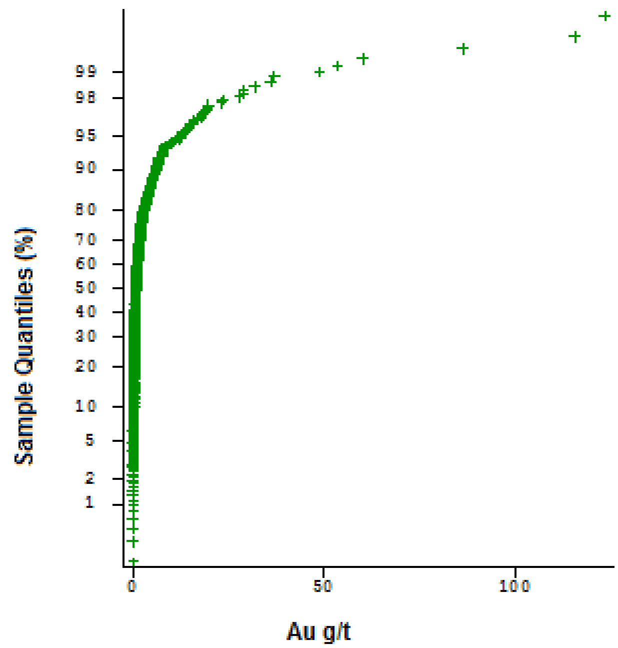
14.2.11 Grade Capping

The Lavrinha Deposit is similar to other nuggety gold deposits and shows the presence of extreme values (outliers) which can be observed by histogram data population and PP-Plots. The extreme values appear to be inconsistent with the vast majority of the data set.

The treatment of extreme values was the subject of numerous discussions between professionals involved in the Project to define the best approach to deal with them. Due to the volumetric significance of the outliers, it was decided not to apply any capping and only reduce the influence search radius of these extreme values. The steps taken to treat these extreme values are described as follows:

- Identification of the upper threshold for outliers. The threshold value was identified as Au >23.54 ppm, that represents an inflection point at Q98, as seen in the QQ Plot in Figure 14.25;
- Identify the geological 3D domains that contain outlier samples within their boundaries;
- Estimate the grade of these blocks (only these blocks) with all samples (including outlier samples) with a regular grade interpolation search strategy; and
- Remove the outlier samples from the dataset (outliers samples are considered missing) and estimate the remaining blocks using regular search criteria.

Figure 14.25 QQ Plot of Regularized Samples



14.2.12 Cluster Analysis

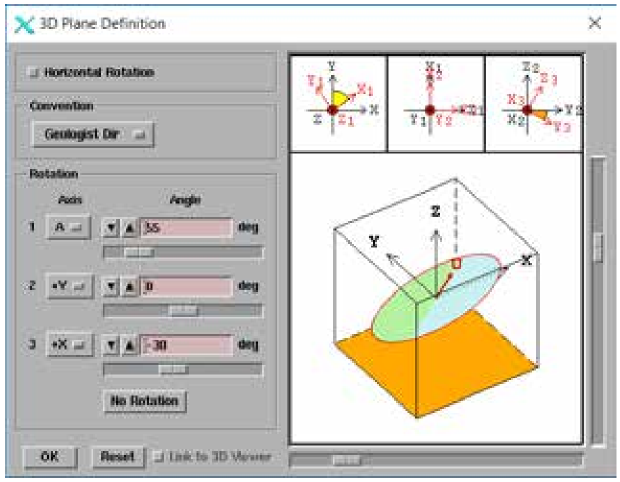
A cluster analysis was preformed but did not obtain any significant results to justify use in the resource estimation process. The de-clustered mean was not taken into account.

14.2.13 Structural Analysis

The analyses of the spatial connectivity (similarity patterns) of variables relating to a mineral deposit is normally preceded by a critical examination of the geology of the deposit and a full analysis of the data whereby a complete study of continuity can be realized efficiently.

Analysis of the spatial continuity plan was carried out according to the orientation of the mineralized zones using general strike and dip. Figure 14.26 shows the definitions with the “U” arrow being the major direction.

Figure 14.26 3D Plane Definitions for Spatial Continuity Analysis

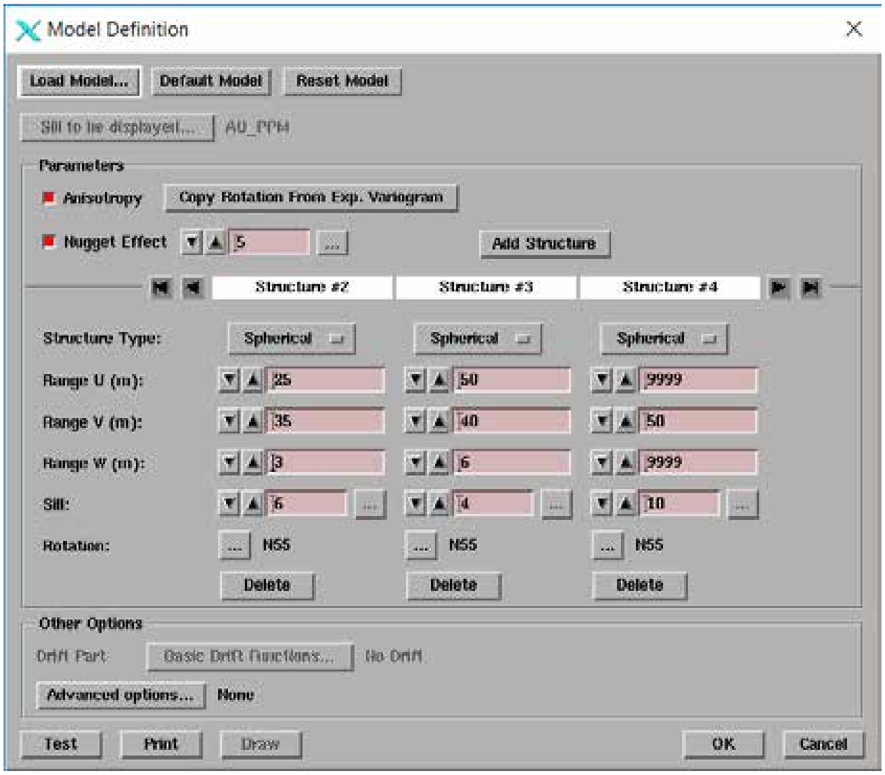


To find the spatial continuity in the horizontal direction, eight directions were calculated inside the plane. A specific upper limit for experimental variograms was set at 33 ppm (this upper limit was used only for the experimental variogram). This was an empirical value after assessing a composited dataset and creating variograms in different directions using the above anisotropy ellipsoid set-up.

14.2.14 Spatial Continuity Analysis

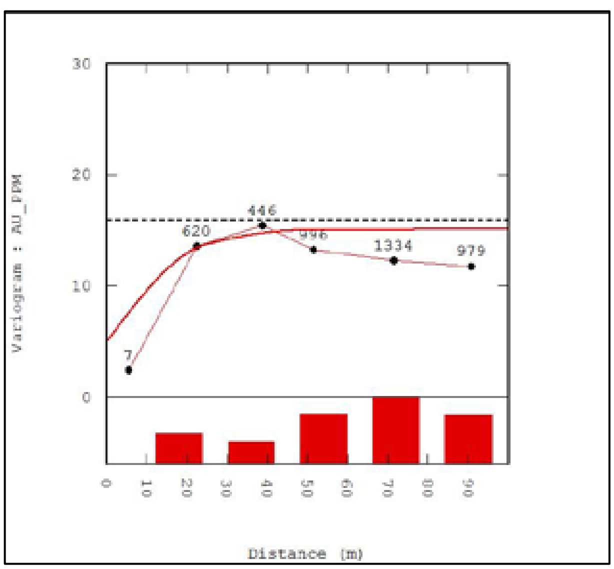
The variogram for gold represents a nugget effect equal to 5 which was obtained from the omnidirectional variogram with a 2 m lag. A zonal anisotropy feature was used to obtain a better fit function of the experimental data. The variogram function did not apply to an infinite range (replaced with the "9999" value) and it was not considered. Figure 14.27 presents the results of the spatial continuity analysis where 3 spherical structures were used (# 2, # 3 and # 4) in addition to the nugget effect.

Figure 14.27 Results of Adjusted Theoretical Variogram

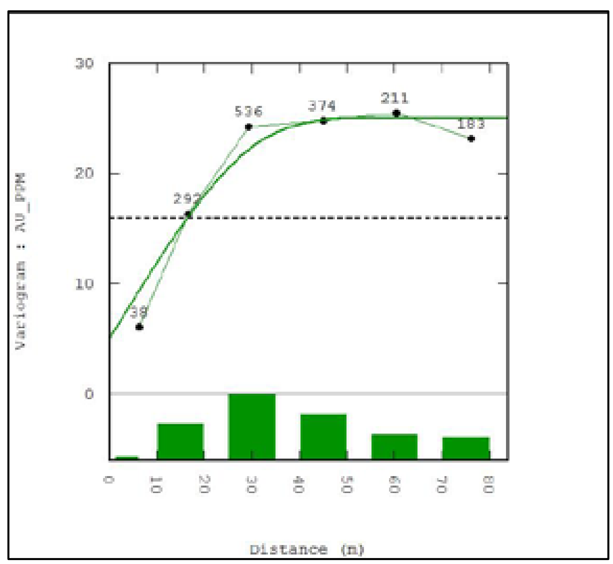


The semivariograms are presented in Figure 14.28 for (a) major continuity direction, and (b) intermediate continuity direction; and in Figure 14.29 (c) minor continuity direction variogram for the variable Au. Further details can be found in Appendix 2.

Figure 14.28 Major and Intermediate Variogram Continuity

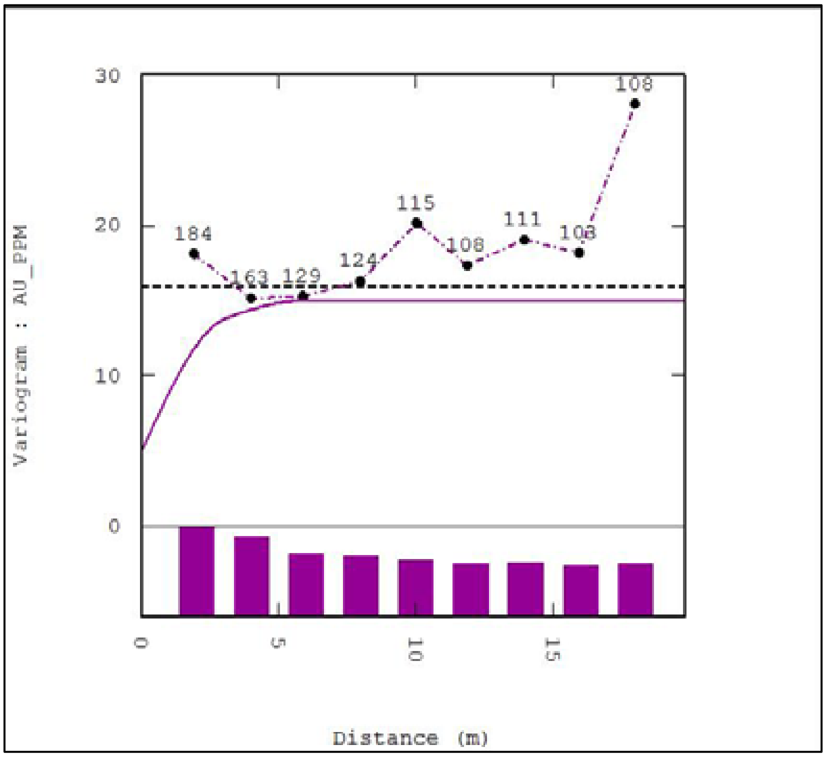


(a) Major



(b) Intermediate

Figure 14.29 Minor Continuity Direction Variogram



(c) Minor

14.2.15 Estimation and Classification

A block model was constructed using 5 m x 5 m x 2.5 m size blocks. Geostatistics was used to populate the block model grades within geological 3D domains based on composite values that were stored in the form of constrained XYZ points. Mathematically this approach is regarded as an interpolation approach.

The classification was based on three different search strategies (one for Measured, another one for Indicated and a third for Inferred) with manual adjustments to remove irregularities. Each of the search strategies is based on a proportional value obtained from the technique called Quantified Kriging Neighborhood Analysis ("QKNA").

The validation of the estimation uses three verifications: check of global mean reproduction (globally un-bias), visual inspection on cross-sections (comparing block grades with composites) and swath plots.

14.2.16 Definition of the Search Neighbourhood

For an estimation method capable of handling any number of nearby samples, the most common approach is to choose samples for estimation and to define a search neighbourhood where all available samples within the desired domain will be used. The search neighbourhood is usually an ellipsoid (3D) centered at the point (block) to be estimated. The orientation of this ellipse is normally chosen following an anisotropy pattern captured by spatial continuity analysis. If sample values are much more continuous along one direction than another, then the ellipsoid is oriented with its major axis parallel to the direction of maximum continuity. The anisotropy of the ellipsoid is usually determined from geological observation or some measurement of the spatial continuity, typically the experimental variogram. If there is no evident anisotropy then the search ellipsoid becomes a sphere.

The definition of the search strategy can have a very significant impact on the outcome of the kriging estimate. The methodologies for quantitatively assessing the suitability of a kriging neighbourhood involve simple tests. The methodology used in this Report is QKNA.

The criteria to evaluate a particular kriging neighbourhood should include the following:

- The slope of the regression of the "true" block grade on the "estimated" block grade;
- The weight assigned to the mean in Simple Kriging ("SK");
- The distribution of kriging weights themselves (including the proportion of negative weights); and
- The kriging variance.

The results of QKNA will assist in block size selection, choice of discretization and mineral resource classification decisions.

14.2.17 Slope of the Regression

Under the assumptions stated previously (that the variogram is valid and the regression is linear), it is possible to calculate the main parameters of the regression between estimated and true block grades. What can be calculated is the covariance (thus correlation coefficient) between estimated and true block grades. The slope is given in terms of this covariance and the variance of the estimated blocks by the expression:

$$a = \frac{Cov(Z_v, Z_v^*)}{Var(Z_v^*)}$$

Where:
a is the slope of the regression,
Zv is the true block grade,
Zv* is the estimated block grade.

A slope of the regression very close to 1.0 implies a conditional unbias state. In these circumstances, the true grade of a set of blocks should be approximately equal to the grade predicted by the kriging estimation.

14.2.18 Weight Assigned to the Mean in Simple Kriging

Instead of performing Ordinary Kriging ("OK"), where the sum of the weights is set to one, one could run SK where the sum of the weights is not constrained to add up to one. The remaining weight is allocated to the mean grade of the domain (the weight of the mean) and is an inversely proportional index of "screen effect". SK is also called "Kriging with known mean", and is based on the assumption that the global mean grade is known and equal to m:

$$Z_V^{*SK} - m = \sum \lambda_i^{SK} \left(Z\left(x_i\right) - m \right)$$

The weight is assigned to the global mean grade or ‘weight of the mean’ as given by:

$$\lambda_m = 1 - \sum \lambda_i^{SK}$$

As a general rule, it is preferable that the weight of the mean be close to zero. The objective in QKNA is to obtain the combination of the best slope with a minimized weight of the mean. Note that the use of SK here is solely for the QKNA method and, in general, the stationary assumptions of SK are not suited to mining grade estimation.

14.2.19 Test Results

The definition of the best search neighbourhood strategy (“QKNA”) is obtained from a series of tests (estimations). The results are presented in Figure 14.30 to Figure 14.32 and in Table 14.16.

Figure 14.30 Slope of the Regression

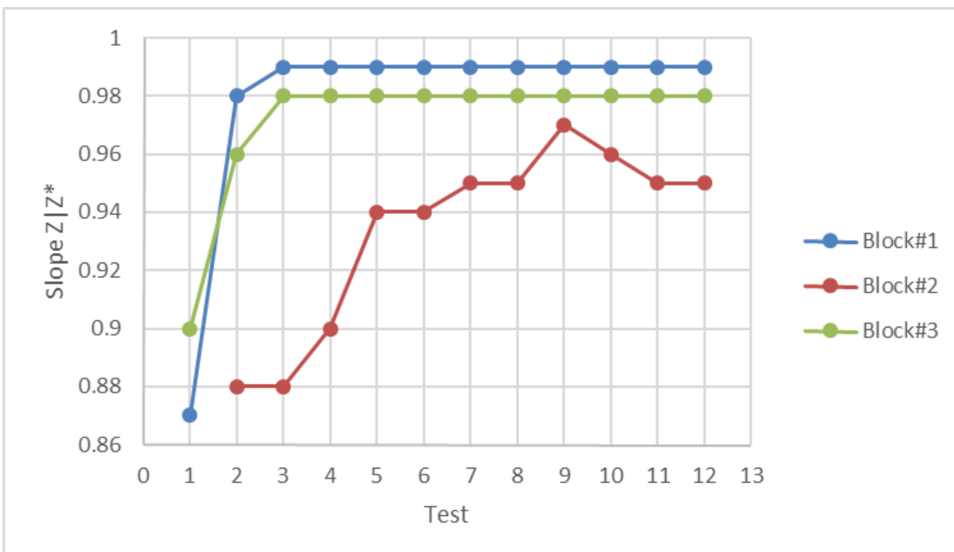


Figure 14.31 Weight Assigned to the Mean in Simple Kriging

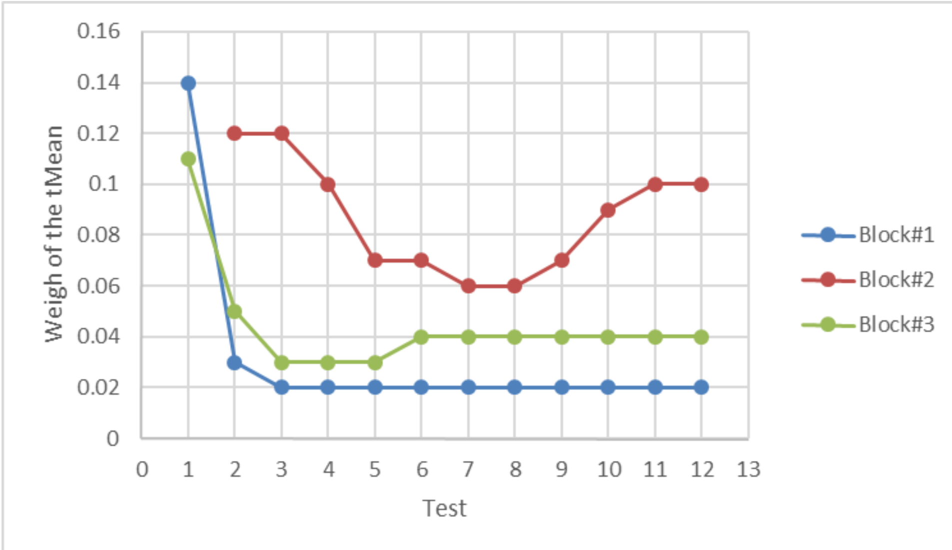


Figure 14.32 Sum of Positive Weights

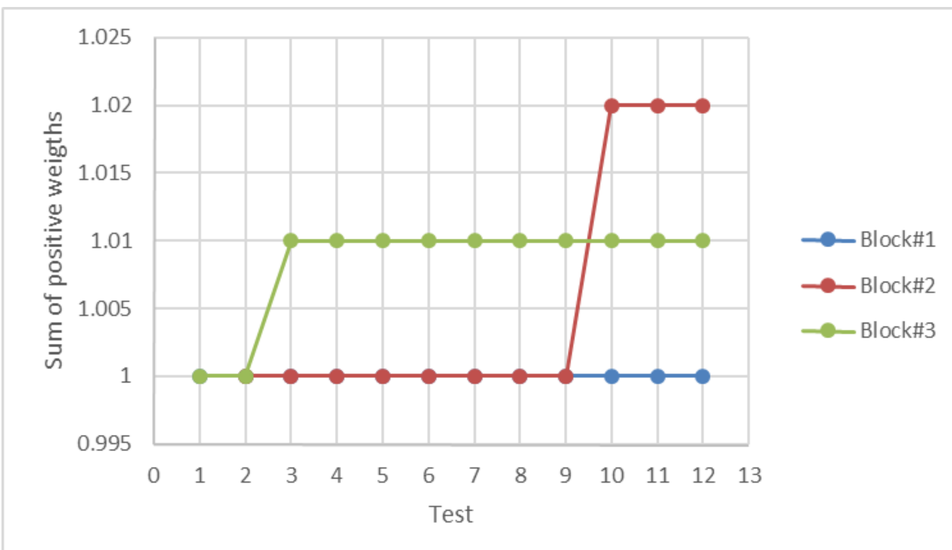


TABLE 14.16
SUMMARY OF TESTS AND THEIR RESPECTIVE SEARCH RANGES

#Test	Major (m)	Intermediate (m)	Vertical (m)
1	20	20	2
2	30	30	4
3	40	40	5
4	45	45	5
5	50	50	6
6	60	60	7
7	70	70	8
8	80	80	10
9	90	90	11
10	100	100	12
11	150	150	18
12	200	200	24

A stabilization in the curves can be clearly noted from the #5 test (especially in the Slope of Z|Z* graphic). This means that from this point, an increase in distance has no significant influence on the estimate. It also shows that the best correlation between the value of real Z and estimated Z is up to that distance.

14.2.20 Classification Standards

The mineralized lodes at the Lavrinha Project were classified in compliance of NI 43-101 standards and the guidelines published by the Council of the Canadian Institute of Mining, Metallurgy, and Petroleum ("the CIM Standards"). The classification of the Lavrinha Deposit was based on three different search estimation strategies related to Measured (Pass1), Indicated (Pass2) and a third for Inferred, and also a further manual adjustment to remove irregularities.

14.2.21 Search Parameters in Mineral Resource Estimation and Classification

One common approach to classify resources is the use of multiple search distances during interpolation. Blocks which are not estimated by the smaller search distances are then re-estimated with larger search distances. Finally, blocks which are not estimated by the first two ellipsoids are then estimated with a larger search ellipsoid. The resource is then pre-classified according to the pass number as Measured, Indicated or Inferred. A minimum number of samples within a range for the block may be considered in the analysis of resource classification.

In this Mineral Resource Estimation, the classification process of the blocks was performed as presented in Table 14.17.

TABLE 14.17
MINERAL RESOURCE ESTIMATE CLASSIFICATION CRITERIA

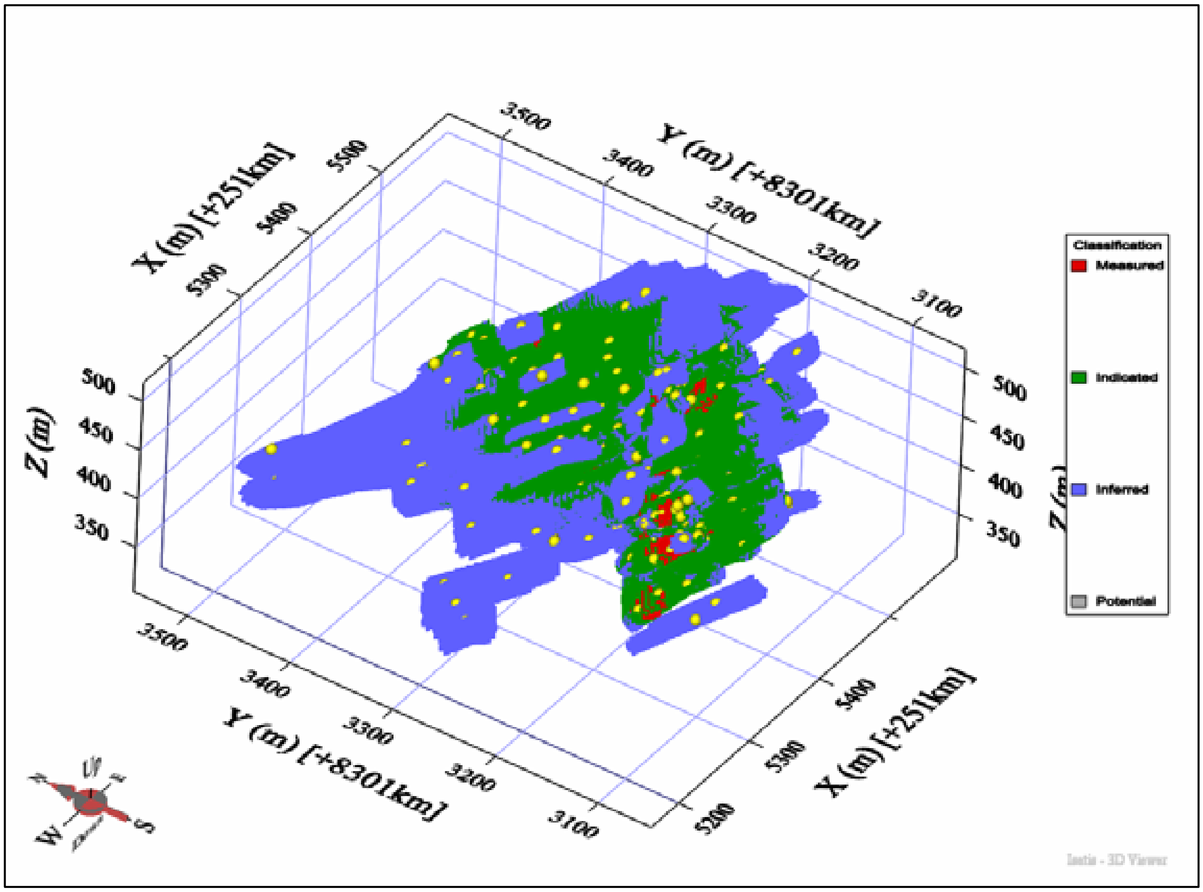
Passes	Search Ellipsoid			Samples				
	Major	Intermed.	Minor	Max	Min	Max per Hole	Max per Octant	MinNbOct
First	25	25	3	4	16	3	2	5
Second	50	50	6	3	16	3	2	6
Third	500	500	120	1	16	3	2	Na

Note: *MinNbOct – Minimum Numer of Consecutive Empty Sectors (Octants)*

14.2.22 Results of the Mineral Resource Estimation and Classification

As previously mentioned, three separate estimations were performed (Figure 14.33). Blocks estimated with the first search strategy were related to Measured Resources (identified by code 1). In the second estimation, the remaining Indicated blocks were estimated using the second search neighbourhood (identified by code 2). In the third search the remaining Inferred blocks were estimated (identified by code 3).

Figure 14.33 3D View of the Estimated Mineral Resource Blocks



Note: *Measured = 1st pass; Indicated = 2nd Pass Indicated and 3rd Inferred. This 3D view is not the final Mineral Resource Classification, it represents blocks before final processing treatment.*

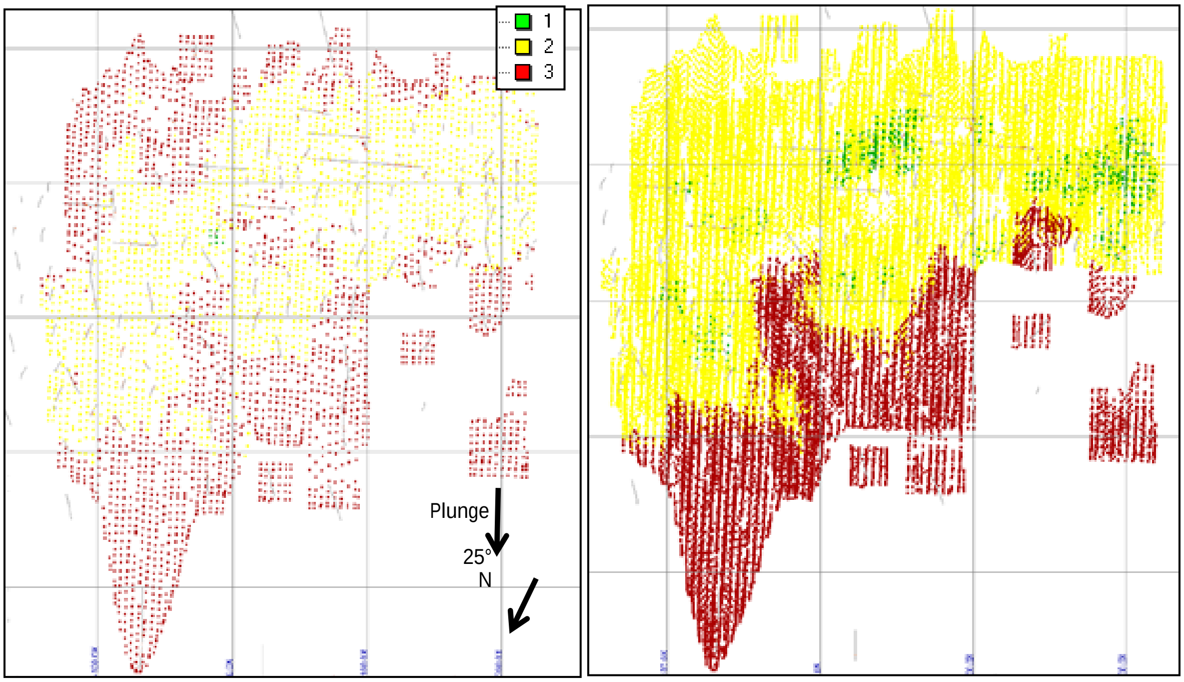
The blocks estimated in the 1st Pass (related to Measured) are located in highly sampled regions (exclusive area of interpolation). The blocks estimated in the 2nd Pass (related to Indicated) were estimated in areas with medium sampling grid (area of interpolation but with wider drilling and sampling density). The blocks estimated in the 3rd Pass (related to Inferred) were estimated with flexible search parameters and consequently lower confidence than for the first two categories. They are located in areas with limited numbers of composite data.

Based on estimated grades, no adjustments were carried out for the Measured mineral resource (Pass 1 directly corresponded to the Measured category). In Passes 2nd and 3rd the following manual adjustments were performed:

Re-classification of blocks near the surface, due to the existence of trenches, surface mapping and grade control data. The previous grade control data was not used in classification of Indicated blocks; and
Remove irregularities.

The final mineral resource figures showing the final processing affecting the 2nd and 3rd Passes is presented in Figure 14.34.

Figure 14.34 Classification 2D View Plane Projection Parallel to Plunge



Note: *Mineral resource classification after manual adjustment (a) using only estimation passes and (b) with manual adjustments. 1= Measured (green), 2= Indicated (yellow), 3= Inferred (red).*

14.2.23 Mineral Resource Grade Estimate Validations

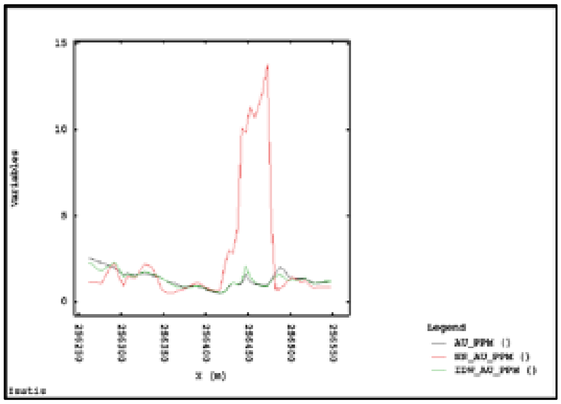
There are several tools to validate Mineral Resource Estimates, the most common is visual inspection in section and plan, and also comparison between the mean composite data values and block kriged, i.e. global non-bias check, comparing the composite data to the blocks in 2D or 3D space. Table 14.18 shows the validation per mean grades, with very low variation between Ordinary Kriging Mean Grade, IPW and Composites (outliers excluded of the mean).

TABLE 14.18
AU (G/T) VALIDATION OF ESTIMATION GRADES

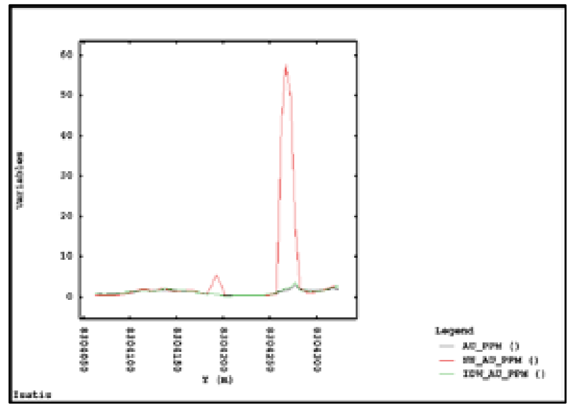
Kriging Pass	Percent Estimated	Au (g/t) - Mean Grade			Diferences (%)	
		Ordinary Kriging (OK)	IPW (P=2)	2m CMP Excl. Outliers	OK/IPW	OK/ 2m CMP
1st	3.21%	2.060	2.050	2.055	0.49%	0.26%
2nd	48.42%	2.090	2.070	2.055	0.97%	1.72%
3rd	48.37%	2.010	2.020	2.055	-0.50%	-2.18%
TOTAL	**100.00%**	**2.050**	**2.045**	**2.055**	**0.25%**	**-0.21%**

Swath Plots (Trend Analysis) is the other tool to validate the estimation performance. The results are shown in Figure 14.35, where the red line represents the grades obtained by NN, the green line represents the IDW (inverse distance weight power 2) and the black line represents the grades obtained by kriging. This validation was performed for each individual mineralized shoot. There were some mineralized shoots that did not have many samples, which prevented a proper validation.

Figure 14.35 Swath Plot Along X (a) and Y (b) Directions



(a)



(b)

Note: *The red line represents the grades obtained by NN, the green line represents the grades obtained by IDW (power 2) and the black line represents the grades obtained by kriging.*

The plot lines showed good adherence and locally validated the results of the estimates. It is important to remember that the plot lines are not strictly equal due to the estimation smoothing effect.

The basic statistical analysis of the estimated samples is shown in Table 14.19. The results show Au grades with the OK method (AU_PPM), mean distance (MDist), number of used samples (Nsamp), sum of positive weights of Kriging (SumPosWeig) and inverse square distance interpolation method (IDW_AU_PPM).

TABLE 14.19
STATISTICAL ANALYSIS OF ESTIMATED AU G/T GRADES

Variable (Isatis Notation)	Description	Kriging Pass	Statistics					
			Count	Minimum	Maximum	Mean	Std. Dev.	Variance
AU_PPM	Au (g/t) estimated by Ordinary Kriging	1	55240	0.14	22.63	2.06	1.88	3.52
		2	833368	0.07	74.26	2.09	2.12	4.48
		3	832663	0.05	33.29	2.01	1.67	2.77
IDW_AU_PPM	Au (g/t) estimated by Inverse Power Distance (IPW) - P =2	1	54760	0.05	23.88	2.05	2.04	4.16
		2	823140	0.03	115.08	2.07	2.58	6.64
		3	819527	0.05	44.74	2.02	1.88	3.55
MDist_AUPPM	Average distance of Samples used to estimate a block	1	54760	3.90	22.31	13.07	3.11	9.66
		2	823140	1.87	48.77	25.36	6.21	38.55
		3	819527	0.42	150.17	52.61	28.61	818.53
NSamp_AU_PPM	Number of Samples used to estimate a block	1	54760	3.00	8.00	4.36	0.90	0.81
		2	823140	2.00	13.00	4.57	1.54	2.38
		3	819527	1.00	16.00	5.57	3.33	11.09
Slope_AUPPM	Slope of Regression [Z|Z*] output of Ordinary Kriging	1	54760	0.48	0.99	0.83	0.08	0.01
		2	823140	0.01	0.98	0.69	0.15	0.02
		3	819527	0.00	0.98	0.52	0.26	0.07
StdDev_AUPPM	Standard Deviation - output of Ordinary Kriging	1	54760	1.49	4.09	2.62	0.46	0.21
		2	823140	1.63	5.71	3.38	0.56	0.31
		3	819527	1.73	6.40	3.97	0.74	0.55
SumPosWeig_AUPPM	Sum of Positive Weights - output of Ordinary Kriging	1	54760	1.00	1.01	1.00	0.00	0.00
		2	823140	1.00	1.04	1.00	0.00	0.00
		3	819527	1.00	1.05	1.00	0.00	0.00
VarZ_AUPPM	Kriging Variance	1	54760	10.09	18.18	13.68	1.46	2.14
		2	823140	6.73	19.29	11.95	2.05	4.20
		3	819527	4.31	25.00	12.01	6.30	39.63
WMean_AUPPM	Weight of Mean - output of Simple Kriging	1	54760	0.01	0.52	0.18	0.08	0.01
		2	823140	0.03	0.99	0.36	0.15	0.02
		3	819527	0.06	1.00	0.56	0.23	0.05

14.2.24 Bulk Density Model

Bulk density block model estimation was carried out by Nearest Neighbour ("NN") from raw data samples using hard boundaries of the lithology model, for which:

- Litho code =1 for Sericite Schist;
- Litho code =2 for Fine Metarenite; and
- Litho code= 3 for Metarenite.

The decision to subdivide the dataset based on lithology was made in compliance with Aura and MCB technical teams based on statistical analysis carried out comparing:

- Mineralization and Waste, (Figure 14.36);
- Hydrothermal Alteration, (Figure 14.37); and
- Lithological Domains, (Figure 14.38).

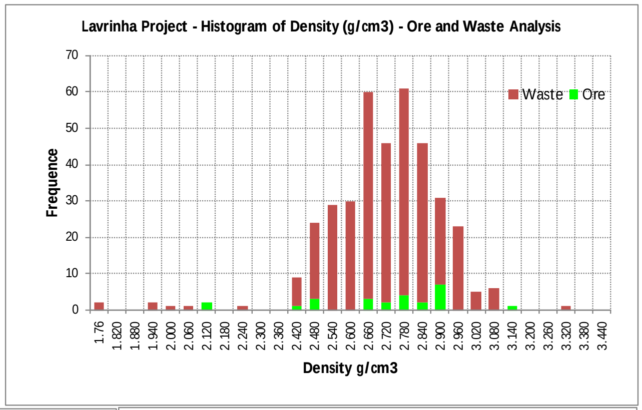
Analysis of these figures showed great similarity between the mean values as well as the distribution for the dataset which was separated according to the mineralization, hydrothermal alteration and mineralized/waste domains.

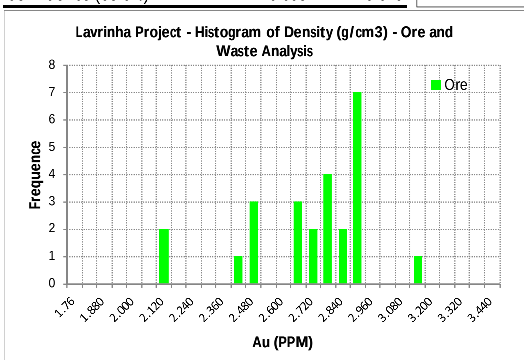
The lithology data, despite some similarities, had larger relative differences in mean values and its distributions. Furthermore, it is confirmed from the visual observations and geological characteristics that the bulk density variations have higher lithologic affinities in comparison with mineralized zones.

Figure 14.36 Bulk Density Statistical Analysis for Mineralization and Waste

Lavrinha Project - Raw Data Statistical Summary of Density Ore and Waste

Density g/cm3	Ore	Waste
Mean	2.738	2.739
SE of Mean	0.048	0.010
Median	2.810	2.755
Mode	2.930	2.690
Standard Deviation	0.24	0.18
Variance	0.06	0.03
Kurtosis	1.17	5.47
Skewness	-1.11	-1.29
Range	1.01	1.56
Minimum	2.14	1.76
Maximum	3.15	3.32
Sum	68	975
Count	25	356
Coeficient of Variation	0.088	0.067
Median/Mean	102.6%	100.6%
Confidence (95.0%)	0.095	0.019





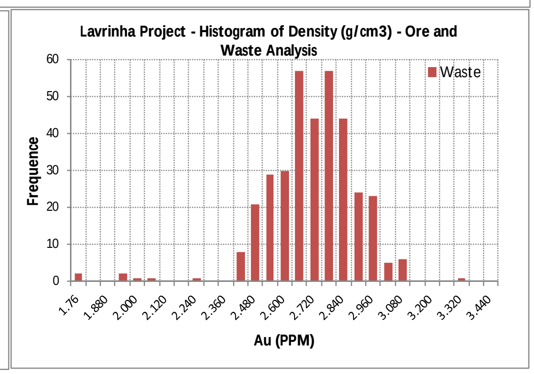
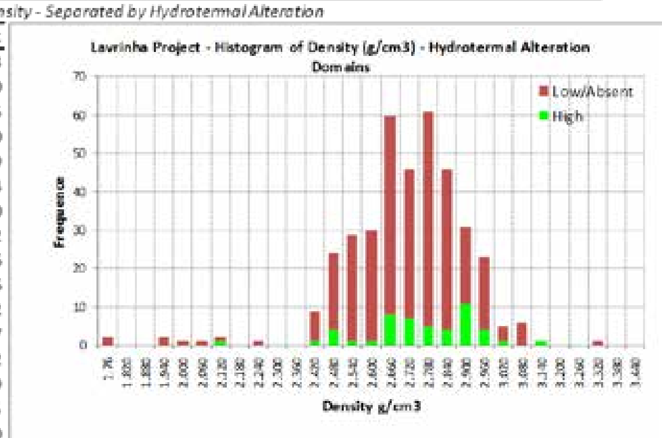
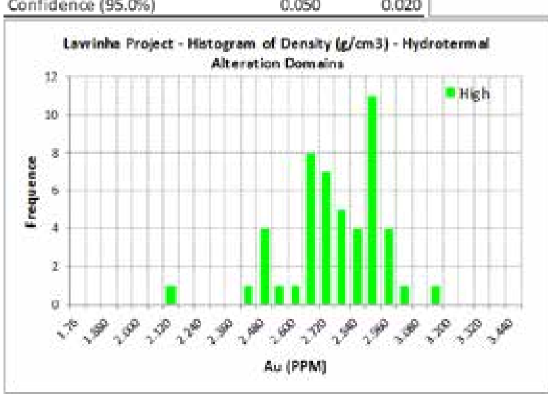


Figure 14.37 Bulk Density Statistical Analysis for High and Low/Absent Hydrothermal Alteration

Lavrinha Project - Raw Data Statistical Summary of Density - Separated by Hydrotermal Alteration

Density g/cm3	High	Low/Absent
Mean	2.782	2.733
SE of Mean	0.026	0.010
Median	2.810	2.745
Mode	2.760	2.690
Standard Deviation	0.18	0.19
Variance	0.03	0.04
Kurtosis	1.76	5.30
Skewness	-0.94	-1.32
Range	0.99	1.56
Minimum	2.16	1.76
Maximum	3.15	3.32
Sum	136	907
Count	49	332
Coeficient of Variation	0.065	0.069
Median/Mean	101.0%	100.5%
Confidence (95.0%)	0.050	0.020





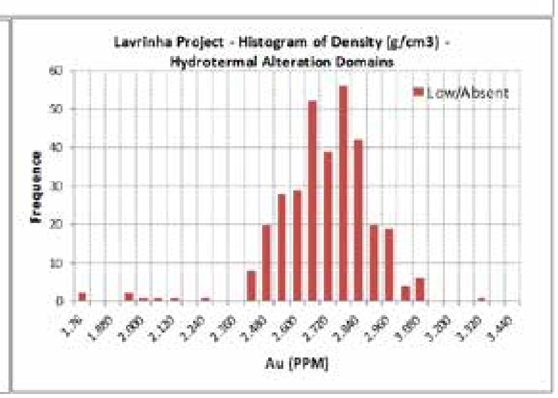
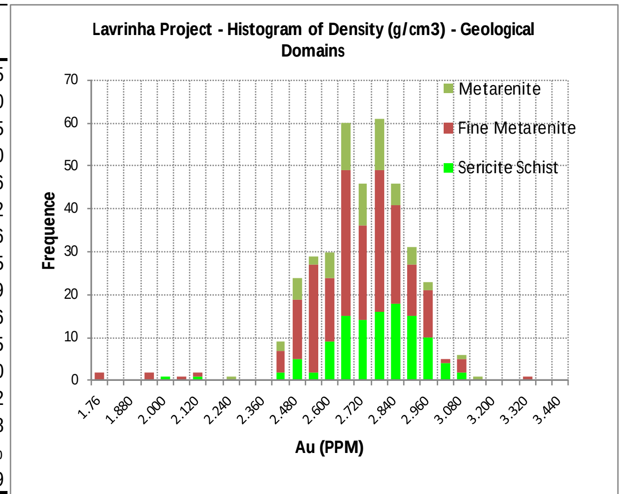
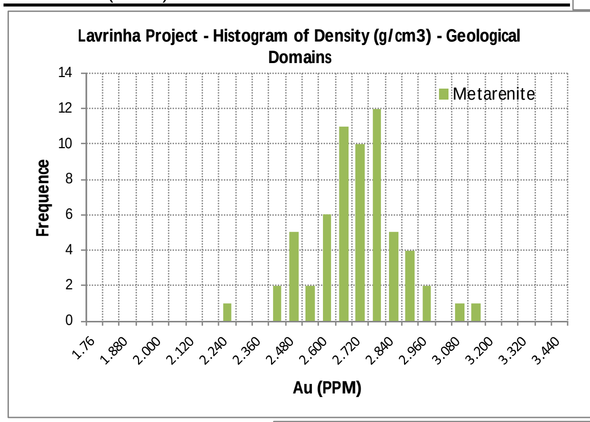


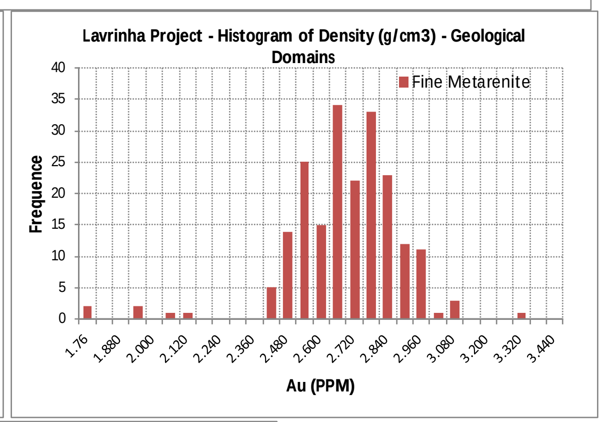
Figure 14.38 Bulk Density Analysis by Lithology

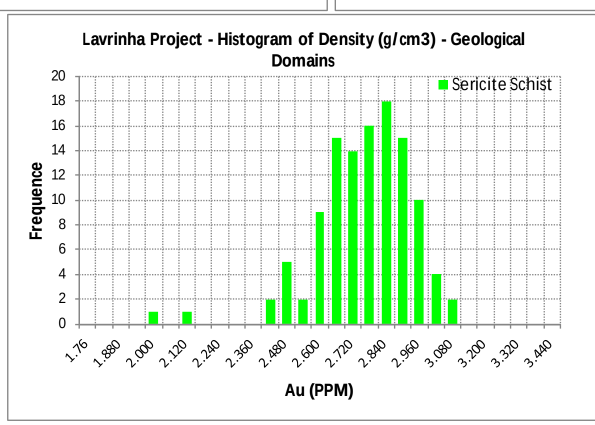
Lavrinha Project - Raw Data Statistical Summary of Density - Geological Domains

Density g/cm3	Sericite Schist	Fine Metarenite	Metarenite
Mean	2.786	2.714	2.735
SE of Mean	0.016	0.014	0.020
Median	2.810	2.720	2.745
Mode	2.810	2.690	2.820
Standard Deviation	0.17	0.20	0.16
Variance	0.03	0.04	0.02
Kurtosis	3.64	5.47	0.96
Skewness	-1.20	-1.42	-0.15
Range	1.10	1.56	0.89
Minimum	2.03	1.76	2.26
Maximum	3.13	3.32	3.15
Sum	318	556	170
Count	114	205	62
Coeficient of Variation	0.061	0.074	0.058
Median/Mean	100.8%	100.2%	100.4%
Confidence (95.0%)	0.031	0.028	0.039









The block model bulk density estimation was carried out using raw data, hard boundaries for lithology domains, and using unique neighbourhood estimator NN. The result of this estimation was checked by comparing the simple mean of the samples with the average grade of the blocks (Table 14.20).

TABLE 14.20 BULK DENSITY VALIDATION						
Lithology	**Code**	**Database**		**Block Model**		**Difference**
		No.	**Dens t/m^3**	**No.**	**Dens t/m^3**	**%**
Sericite Schist	1	114	2.79	28,416	2.80	0.57%
Fine Metarenite	2	205	2.71	6,881	2.71	-0.27%
Metarenite	3	62	2.74	10,327	2.80	2.26%
Total		**381**	**2.74**	**45,624**	**2.79**	**1.74%**

14.2.25 Mineral Resource Estimate Statement

The Mineral Resource Estimate for the Lavrinha Deposit has been reported above a 0.5 g/t Au cut-off grade, inside an optimized pit shell with a gold price of US$1,300/oz. Table 14.21 presents the economic assumptions used to define the cut-off grade. Table 14.22 presents the Mineral Resource Estimate and Table 14.23 presents the tonnage/grade sensitivity within the US$1,300/oz pit for selected cut-off grades. Figure 14.39 presents the tonnage/grade sensitivity curves for Measured, Indicated and Inferred Mineral Resources.

TABLE 14.21 MINERAL RESOURCE ESTIMATE ECONOMIC PARAMETERS		
Selling Price	**Value**	**Unit**
Gold	1,300	US$/oz
Mining Cost	Value	Unit
Mill Feed	2.44	US$/t
Waste	1.89	US$/t
Plant Costs	Value	Unit
ROM	10.24	US$/t
Recovery	Value	Unit
Mill Feed	93.0	%
Other	Value	Unit
Royalty	38.5	US$/oz
G&A	3.8	US$/t (M)
Dilution	35.0	%
Factor	0.0322	(g/oz)
Marginal Cut-off Grade	Value	Unit
Au	0.5	g/t

TABLE 14.22 **LAVRINHA MINERAL RESOURCE ESTIMATE(1-8)**			
Resource Category	**Tonnes (t)**	**Au (g/t)**	**Contained Au oz**
Measured	74,000	2.31	5,500
Indicated	1,226,000	2.25	88,700
Measured + Indicated	1,300,000	2.25	94,100
Inferred	283,000	2.51	22,800

(1) CIM Definitions were followed for the Mineral Resource Estimate.
(2) The Mineral Resource Estimate for the Lavrinha Deposit was prepared under the supervision of Marcelo Batelochi, AusIMM (CP 205477).
(3) Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
(4) The quantities and grades of reported Inferred Resources in this estimation is uncertain in nature and there has been insufficient exploration to define these Inferred Resources as Indicated or Measured Mineral Resource and it is uncertain if further exploration will result in upgrading them to the Indicated or Measured Mineral Resource category.
(5) The Mineral Resource Estimate is based on an optimized pit shell using US$1,300/oz gold and at a cut-off grade of 0.50 g/t gold. Mining costs were considered at US$2.44/t and US$1.89/t for mineralized material and waste haulage, plant processing costs of US$10.24/t and G&A of US$3,800,000 per year at a process recovery of 93%.
(6) A bulk density model based on rock type was used for volume to tonnes conversion with resources averaging 2.77 tonnes/m^3.
(7) Surface topography as of December 31, 2015.
(8) Contained metal may not sum due to rounding.

TABLE 14.23
TONNAGE/GRADE SENSITIVITY AT DIFFERENT CUT-OFF GRADES WITHIN THE US$1,300/OZ PIT

COG Au g/t	**Measured**			**Indicated**			**Measured + Indicated**			**Inferred**		
	Tonnes ('000)	**Au (g/t)**	**Ounces ('000)**	**Tonnes ('000)**	**Au (g/t)**	**Ounces ('000)**	**Tonnes ('000)**	**Au (g/t)**	**Ounces ('000)**	**Tonnes ('000)**	**Au (g/t)**	**Ounces ('000)**
> 3.00	15	5.23	2.5	260	5.20	43.5	274	5.20	45.9	88	4.64	13.1
2.75	17	4.96	2.6	302	4.88	47.3	318	4.88	49.9	104	4.37	14.6
2.50	20	4.59	2.9	343	4.60	50.8	363	4.60	53.7	116	4.18	15.6
2.25	23	4.27	3.1	395	4.31	54.8	418	4.31	57.9	128	4.01	16.5
2.00	28	3.87	3.5	468	3.97	59.7	496	3.96	63.2	135	3.91	17.0
1.75	36	3.43	4.0	558	3.63	65.1	594	3.62	69.1	148	3.73	17.8
1.50	46	3.05	4.5	670	3.30	71.0	715	3.28	75.4	190	3.26	19.9
1.25	56	2.74	4.9	791	3.00	76.3	847	2.98	81.3	210	3.08	20.8
1.00	66	2.50	5.3	940	2.70	81.7	1,006	2.69	87.0	226	2.95	21.4
0.75	71	2.38	5.5	1,084	2.46	85.7	1,155	2.46	91.2	263	2.65	22.4
0.50	74	2.31	5.5	1,226	2.25	88.6	1,300	2.25	94.1	283	2.51	22.8
0.25	85	2.07	5.6	1,323	2.11	89.9	1,408	2.11	95.5	300	2.39	23.0

Figure 14.39 Tonnage/Grade Curve within the US$1,300/oz pit – MI&I Mineral Resources

14.2.26 Recommendations

MCB offers the following recommendations:

- Assay drill core intervals not sampled;
- A complete review of the database information and cross-referencing with original records for the drill hole and assay databases;
- Update the surface topography files with more precision.
- Additional drilling is recommended at Lavrinha to drill off the deposit in the SW of the Property towards the adjacent valley and also at the southern end of the deposit where the density of drilling is reduced and there are some lenses that can be potentially delineated near surface.

14.3 PAU-A-PIQUE MINERAL RESOURCE ESTIMATE

14.3.1 Summary

The Mineral Resource Estimate for the Pau-a-Pique Mine Project ("PPQ") was estimated by conventional 3D computer block modelling methods employing Dassault Systemes Geovia mining software V6.4 and V6.71 ("GEMS"). The mineral resource estimate is based on surface diamond drilling and underground fan diamond drilling, core sampling and assaying as well as underground face channel chip sampling and assaying. Assaying was performed at SGS commercial laboratory in Belo Horizonte, at the Yamana mine laboratories Ernesto and MFB, as well as the Aura Sao Francisco mine laboratory, all in Brazil. Resources were initially estimated by P&E in October 2015. A recent in-fill drill program in May 2016 added 3,211.44 m of drilling in 28 holes (at the NNW and SSE ends of the deposit) to the previous drilling database. This additional drilling was focussed on developing Indicated resources for potential conversion to mineral reserves in an upcoming Feasibility Study.

The PPQ gold mineralization consists largely of free gold hosted in mylonite, muscovite schist, biotite schist and quartz veins accompanied by sulphides that occur along a sheared contact between meta-tonalite and meta-conglomerate. Mineralization is epigenetic, hydrothermal in origin and is structurally controlled. The mineralized contact trends SSE and dips steeply SW. The meta-conglomerate unit pinches out to the SSE where three additional zones, named herein as P1, P3 and P4, occur in the hanging wall (P1) and footwall (P3, P4) to the principal zone P2. The latter was mined from 2011 to 2014. P1, P3, P4 have not been mined. The narrow widths of the steeply dipping mineralization all but preclude open cast mining and the mineralization at PPQ is amenable to underground mining.

The exploration drill hole database for PPQ contains 491 diamond drill holes totalling 53,478.21 m and 811 channels totalling 3,878.6 m.

The gold price used for the resource estimate is US$1,275/oz. Gold recovery assumptions are 93% for process, 99.99% for payable and $15/oz Au for refining.

Three Mineral Resource wireframes were constructed from mineralization intersections in channels and drill holes at a cut-off grade of 1.5 g/t Au over a minimum horizontal mining width of 3.0 m. The cut-off grade represents a marginal operating cut-off based on 75% of the currently estimated mining cost. Mineralization widths are commonly narrower than minimum mining width and were "bulked out" to at least minimum width using adjacent assays. The 3 m minimum mining width was selected to permit use of the scooptrams currently available at the mine.

Assay grades were capped at 50 g/t Au. Assay composites were generated for the zone intersections from the assays captured by GEMS software in the mineral wireframes. Composites were prepared down hole dynamically at nominal 1.5 m down-hole. This method ensures that the grade weighting is correctly applied for bulked out lode widths but results in variable composite lengths.

Two block models were created, a lithologic model for geologic interpretation and a resource block model. The X-axis of the resource block model is rotated to 150° azimuth. The resource block size is 3 m cubed, suitable for selective mechanized mining methods. Ordinary kriging ("OK") interpolation was carried out using multiple search distances and search ellipses oriented to the NNW mineralization plunge. Area of influence for composites grading ≥25 g/t Au was restricted to 12.5 m equivalent. Inverse distance squared ("ID2") and nearest neighbour ("NN") interpolation methods were employed for model validation.

Water immersion bulk density testing for PPQ was carried out at the Ernesto mine site for 379 core samples. The bulk density varies between host rock types and accordingly a bulk density model was interpolated for resource modelling. Average bulk density for the wireframes is 2.78 t/m^3.

The Mineral Resource Estimate was classified as Indicated and Inferred based on drill hole spacing and data quality, channel sampling locations, confidence in the assaying and geologic confidence in the zones interpretation and grade continuity. P&E cautions that the Indicated Mineral Resource held in remnant pillars, sills and "skins" left in stopes may not all be recoverable pending an engineering study. The hanging and footwall lenses P1, P3 and P4 tend to be lower grade than P2 and there is no channel sampling or mining history. The lower grades and their narrower widths affect the interpretation of mineral continuity and confidence in the estimation of the Mineral Resource for these lenses is lower with respect to P2.

The total Indicated Mineral Resource Estimate for a 1.5 g/t Au cut-off grade is 519,000 tonnes averaging 4.05 g/t Au. The total Inferred Mineral Resource Estimate for a 1.5 g/t Au cut-off grade is 117,000 tonnes averaging 4.45 g/t Au.

Validation of the grade interpolation and the block model was carried out by on-screen review of block grades versus drill hole composites and other block model estimation parameters, by comparison of resource grades to the grades of assays, composites and zone intersections, comparison to alternate ID2 and NN interpolations, and review of the volumetrics of wireframes versus reported resources.

Although there is some uncertainty in the channel sample assaying arising from the lack of field QAQC for 2011 to 2013 samples, internal laboratory QAQC performance is comparable to 2014 assaying where field QAQC was done and grade distributions between chips and core samples are comparable indicating that the 2011-2013 production chip sampling is also acceptable for resource estimation. In P&E's opinion, the Mineral Resource Estimate is reasonable and has been undertaken according to industry standard practice.

14.3.2 Mineral Resource Estimate Database

The PPQ Deposit has been sampled by surface and underground diamond drilling and underground face chip sampling.

Core for the surface drilling is largely NQ (47.6 mm) and for the underground fan drilling is NQ (47.6 mm). The database contains some 528 holes for 60, 365.99 m. The PPQ Deposit exploration database contains 491 holes for 53,478.21 m of which there are 57 surface holes for 18,249.45 m and 434 underground holes for 35,228.76 m (Table 14.24). Assayed intervals for drill core total 19,708 records over 19,726.99 m.

TABLE 14.24
SUMMARY OF PAU A PIQUE DRILL HOLE DATABASE

Drill Hole Database	Count	Length (m)	% (by m)
Database	528	60,365.99	100%
Holes with No Assays, No Surveys	17	2,509.76	4%
Exploration Holes Testing Other Targets	20	4,378.02	7%
PPQ Deposit Exploration	491	53,478.21	100%
PPQ Surface	57	18,249.45	34%
PPQ Underground	434	35,228.76	66%
PPQ Fill-in 2016	28	3,211.44	6%
PPQ Resource	313	32,554.05	61%

Much of the sampling below 310 elevation (135 m depth) has been done by UG definition fan drilling resulting in a relatively high number of shallow angle and down dip intersections that make correlation hole to hole difficult/uncertain and de-regularizes the number of core samples per intercept. In addition some surface holes are 450 m to 550 m long and thus the position of the toe and resource intersections is subject to the accuracy of deviation surveys and instrument accuracy tolerances. P&E examined the down hole surveys with respect to deviation (Table 14.25). From Table 14.25 it is clear that a review of the down hole surveys is recommended and implausible readings should be removed and the resulting re-positioning of the hole toe reviewed for impact. With readings taken at 3 m intervals down hole, excessive deviation is not obvious on screen or from simple hole trace plots unless the successive differences between readings is plotted e.g. hole PQ_12 in Figure 14.40.

TABLE 14.25 DOWN HOLE SURVEY AND DEVIATION REVIEW		
Number of Holes Reviewed	318	
Number of records	10,011	
Total Length Drilled (m)	30,880.28	
Number of Unsurveyed Holes	0	
Deviation Review	**Deviation Threshold**	
	5°/30 m[1]	**10°/30 m[2]**
No. of Excessive Azimuth Deviations	519	128
No. of Excessive Dip Deviations	407	42
Minimum Azimuth Deviation °/m	-1.82	-1.82
Maximum Azimuth Deviation °/m	1.7	1.7
Minimum Dip Deviation °/m	0	0
Maximum Dip Deviation °/m	2.33	2.33
No. of Holes Affected	152	49
No. of Holes with no Azimuth Change	0	0
No. of Holes with no Dip Change	0	0

(1) Unlikely for BQ, fail for NQ
(2) Fail for NQ

Face chip “channel” sampling underground was carried out at 3 m intervals in production headings during mining operations. The channels are entered in the database as pseudo drill holes totalling 811 records for 3,878.6 m of which 91 for 422.82 m were not assayed (Table 14.26). The database contains 5,943 chip sample assays for 3,279.59 m of which 2,428 samples for 1,241.73 m were used for resource estimation. Explicit missing assay intervals in the database total 1,070 over 594.10 m.

Field QAQC work for the chip sampling was undertaken in 2014 and supported the sampling as industry standard in P&E’s opinion. No field QAQC was done for years 2011 to 2013 and two Yamana mine labs were used: Ernesto Mine lab and MFB lab. P&E reviewed the lab internal QAQC blanks and reference standards, which work was very comparable to the 2014 internal lab QAQC, and has accepted the chip sampling assays for resource estimation. P&E prepared QQ plots of the core and chip sample assays distributions for assays contained within the resource wireframes and believe the data sets are compatible for resource estimation with the channel samples showing a slight low bias with respect to the core assays.

P&E notes that drill holes (17) and channels completely lacking assays were omitted from resource estimation. Nine of these drill holes appear to have be planned but not executed since there are no down hole surveys or assays. Some 20 exploration holes drilled SW and elsewhere are not in the immediate resource area but are also included in the database. Partially assayed holes and channels with explicit or implicit missing assays were used for Mineral Resource Estimation and the explicit/implicit missing intervals were assigned zero grade.

Figure 14.40 Surface Hole PQ-12 Down Hole Survey Review

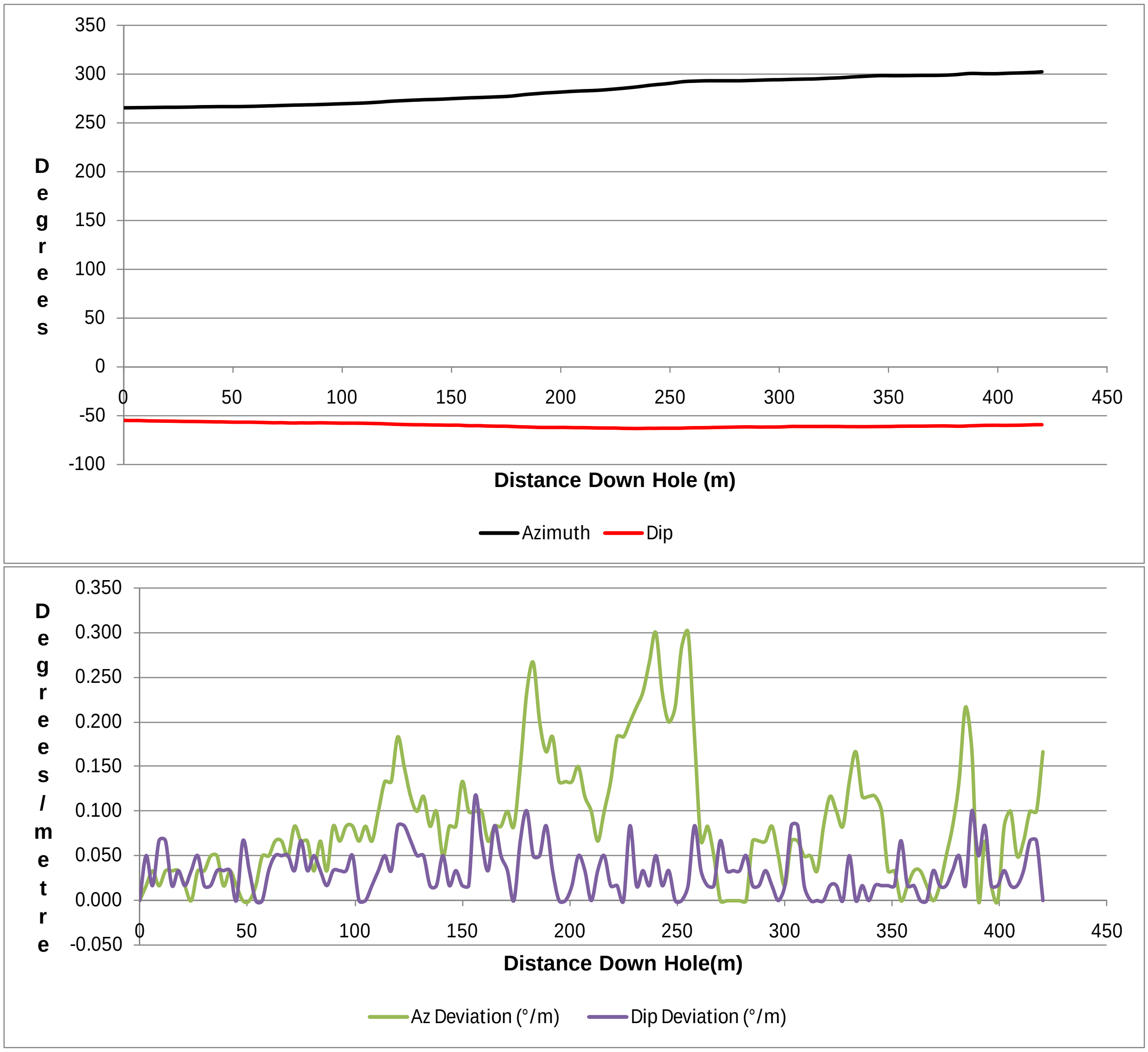


TABLE 14.26 CHANNEL DATABASE SUMMARY		
Channels	**Count**	**Length (m)**
Chip Sampled	720	3,455.88
No Assays	91	422.82
Total	**811**	**3,878.60**

Figures 14.41 to 14.43 show the diamond drill holes and channels in plan and inclined longitudinal section.

Figure 14.41 Diamond Drill Hole Location Plan and PPQ Deposit (Pre-Mining)

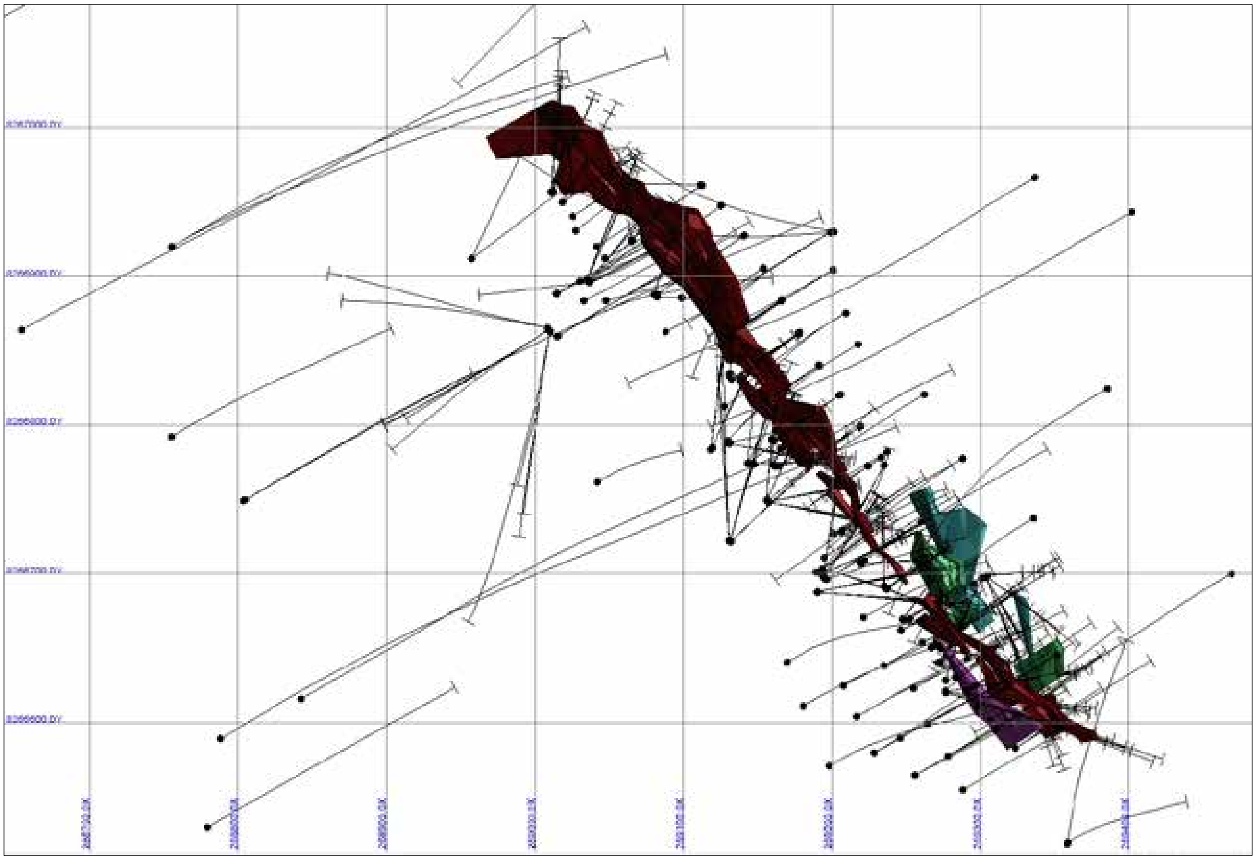


Figure 14.42 Diamond Drill Hole Location Plan, PPQ Deposit (Pre-Mining) and Workings

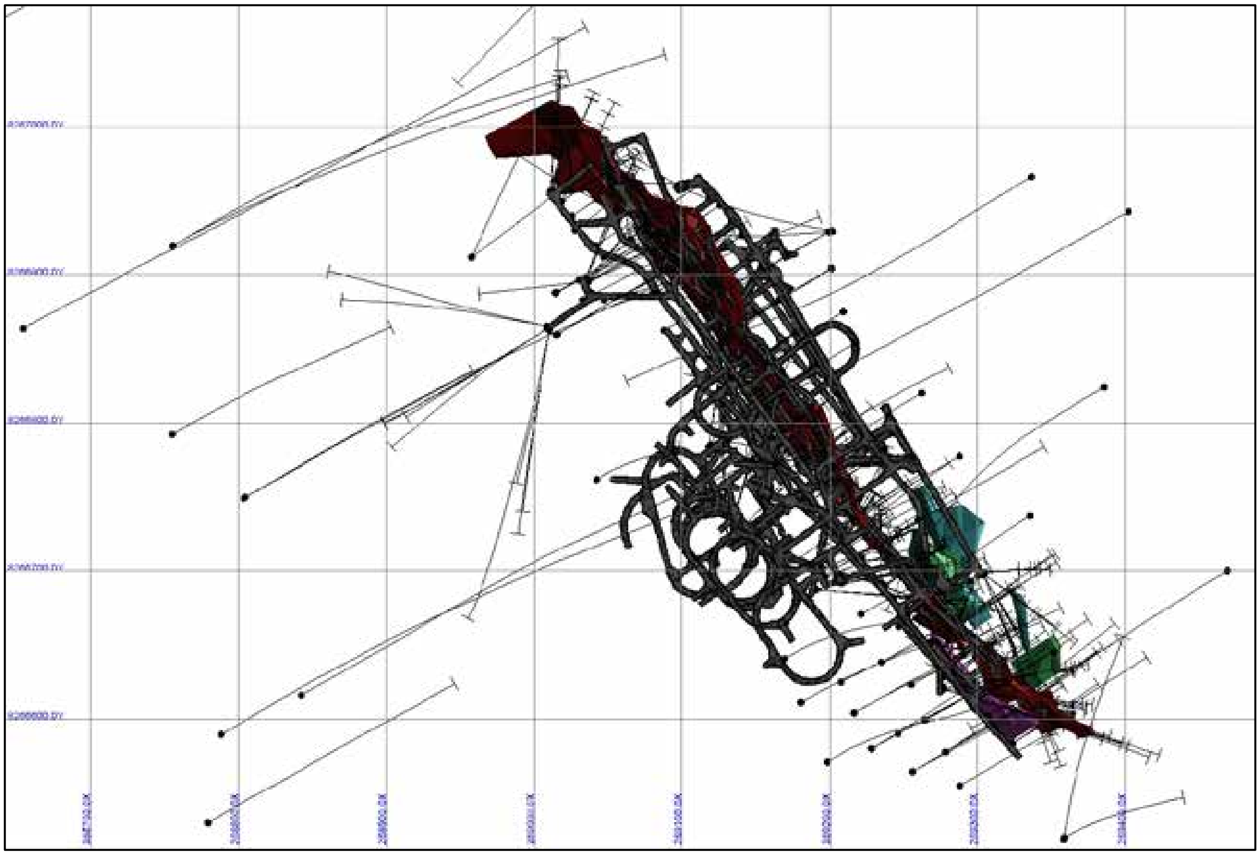
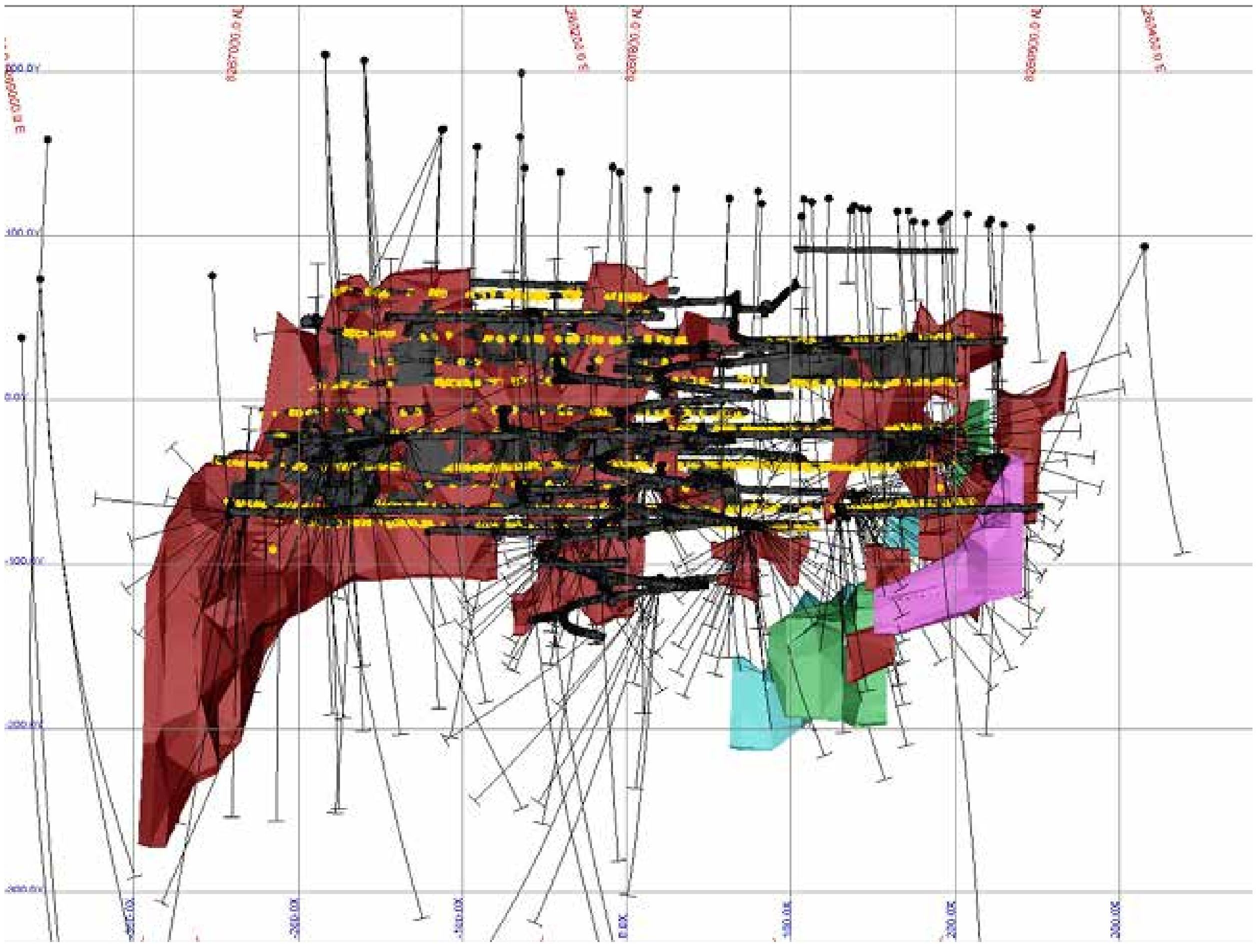


Figure 14.43 Drill Hole and Channel Sample Locations, PPQ Deposit (Pre-Mining)



*(Inclined Longitudinal Section Looking ENE)

Legend	
	Diamond Drill Hole
	Channel Sample

14.3.3 Pau A Pique Wireframing

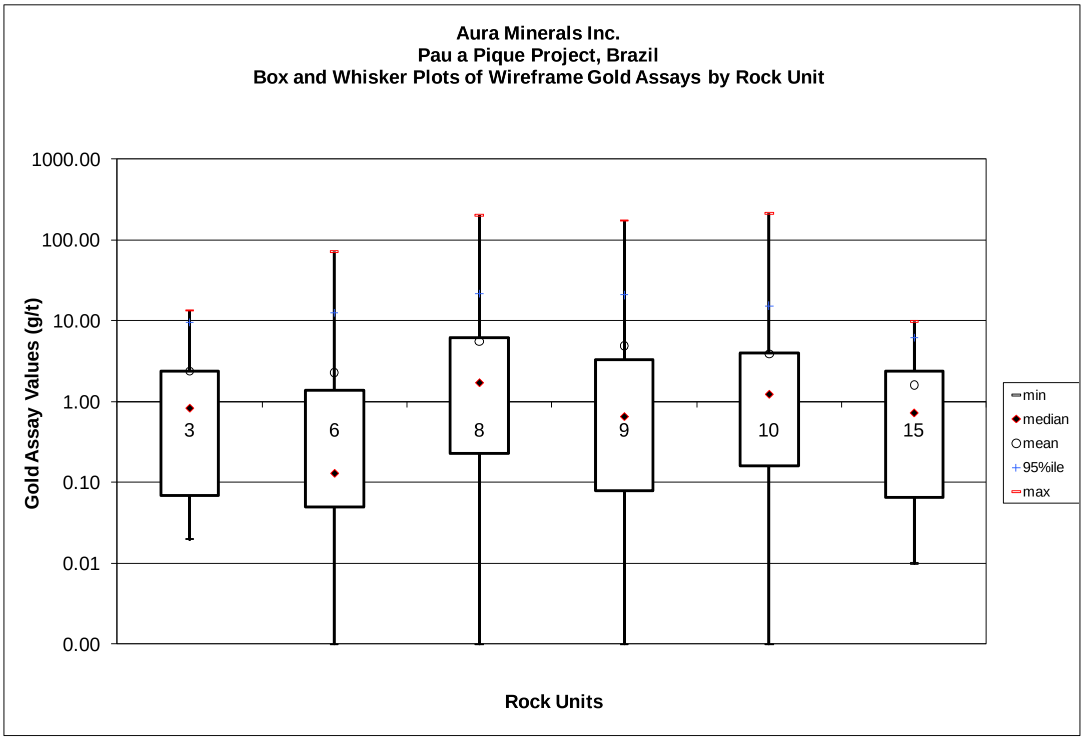
The basis for mineral zone delineation and wireframing is a cut-off of 1.5 g/t Au over a minimum horizontal mining width of 3 m. This grade is considered as an incremental or marginal cut-off for an end of mine life cost of 75% of normal mining operating cost (Table14.27).

TABLE 14.27 **GOLD PRICE AND OPERATING PARAMETERS FOR MINERAL RESOURCE ESTIMATE CUT-OFF GRADE DETERMINATION (US$)**	
Au Price /oz	$1,275
Mining Cost /t ore	$43.72
Marginal Mining Cost /t ore	$32.79
Process Cost /t ore	$12.54
G&A Cost /t ore	$5.11
Ore Haulage Cost /t ore	$8.56
Au Process Recovery	93.0%
Au Smelter Payable	99.99%
Au Refining /oz	$15.00
Au Cut-Off Grade g/t	1.57

Review of the geology, host rocks, apparent controls on gold mineralization, and distribution of assay grades in drilling and underground channel chip sampling brought to light the following aspects of interpretation for wireframe modelling:

- Free gold is common and gold distribution is erratic both laterally and vertically within the host narrow to broad, mylonitic and schistose tonalite-conglomerate contact zone, particularly at UG mining cut-off grades (≥1.5 g/t Au).
- Gold at potentially economic grades is hosted in a number of rock types in and on the margins of the contact zone (Figure 14.44) and grade distribution is strongly skewed.
- Gold mineralization at potentially economic grades and widths may be found in tonalite or conglomerate metres into the hanging wall or footwall of the contact schists, however, these are likely minor separate shears or splays off the contact shear zone and 3D continuity may not be demonstrated resulting in these isolated occurrences being ignored for the purpose of resource estimation.
- There is a high "nugget" (40%) effect and high grades may be localized in "jewel boxes" of restricted dimensions.
- Much of the sampling has been by UG fan drilling with a relatively high number of down dip and shallow angle intersections that make correlation hole to hole difficult/uncertain and de-regularizes core sampling.
- Channel face chip samples provide more regularized sampling, however, following the zone underground has proved elusive in several drifts/stopes and favourable grade intersections in drill holes above and below the levels are not necessarily matched by consistent favourable grades in the drifts.
- The wireframe cut-off results locally in narrow intersections of gold mineralization with grades >1.5 g/t on trend within the zones not meeting minimum mining width. As such these intersections are isolated and not incorporated in resources. For the generally lower grade footwall and hanging wall lenses where drilling is less intensive, the interpretation of mineralization continuity may be affected in that alternative interpretations of continuity are possible and confidence the resource interpretation is reduced.
- Internal dilution to make resource cut-off grade is relatively high.

Figure 14.44 Gold Grade Distribution in Various Host Rocks



Rock Legend	
3	Sericite/Muscovite Schist
6	MetaConglomerate
8	Mylonite/Muscovite Schist/Trap
9	Quartz Vein
10	MetaTonalite
15	Biotite Schist/Sheared Tonalite

After review of drill hole spacing, cross sections were cut at 12.5 m spacing transverse (60° azimuth) to the SSE deposit trend. Wireframing was carried out by snapping to assay limits in 3D space where cumulated assays achieved cut-off grade over the minimum mining width. Geologic interpretation and following of the contact zone using the lithologic block model was a key aspect of the wireframing. In cases where sub cut-off/width material in a drill hole occurred within the zone between adjacent resource grade intersections, the wireframe was carried through to maintain zone continuity. Similarly the nominal 3 m width was maintained where practicable but may be less at zone inflection points.

Level plans were established at the production drift elevations and polylines were constructed on levels by snapping to channel chip sample limits based on the marginal cut-off grade and minimum mining width.

The wireframes based on drilling were extended half way to adjacent drill holes internally within the wireframed deposit or on the margins where barren holes exit. Where no sampling was available at reasonable distances (12 m - 25 m) at the margins on strike as occurs in the fragmented area at the SSE area of the deposit, the wireframes were projected half the section spacing to 6 m past the drill hole intersections. The NNW area of the deposit at depth where no drilling or only very wide-spaced drilling is available, the main zone wireframe was extended as much as 33 m down dip from the lowest sampling on the production level consistent with variography and plunge direction and in keeping with conventional modelling practice where deeper drilling intersections on adjacent sections may be projected across sections representing the depth of the mineralization from the extent of known data.

For solids preparation, an attempt was made to use the GEMS facility of “2-ring solid creation” that allows for horizontal and vertical rings to be modelled together, however, the erratic nature of gold distribution and consequent spatial complexity of the rings’ locations proved too difficult for the software and caused indeterminate errors. As an alternative to the integration of the two sets of rings in the modelling, the cross section polyline rings were snapped to the level rings and used to guide the wireframing where drilling was sparse or lacking. The wireframe was subsequently created by the conventional “tied polylines” method. Use of the channels increased the density of sampling in the mined and developed areas of the deposit as well as filling in some areas deficient in drilling due to lack of access. This method is not as reliable for capture of the chip sample assays as would be use of the level polyline rings because the wireframe control points for the channels are at 12.5 m spacings not at the 3 m spacing of the individual channels.

Aura provided Datamine®-developed wireframes of the mine stopes and workings in 3D-DXF format as obtained from Yamana. These 3D solids were imported to GEMS and repaired where necessary to validate them for further use. The workings’ solids were used to “clip” the P2 Zone solid and produce a depleted wireframe representing the post-mining resources. No mining has been carried out on the footwall P3 and P4 lenses or the hanging wall P1 lens.

Volumetrics of the wireframes and estimated tonnage for a bulk density of 2.78 t/m^3 are presented in Table 14.28. P&E cautions that the difference between the undepleted and depleted volumes will not necessarily match past production due to the cut-off grades defining the current wireframes and past reserves differ, dilution impacted production and the wireframes do not necessarily directly correspond spatially to mined stopes.

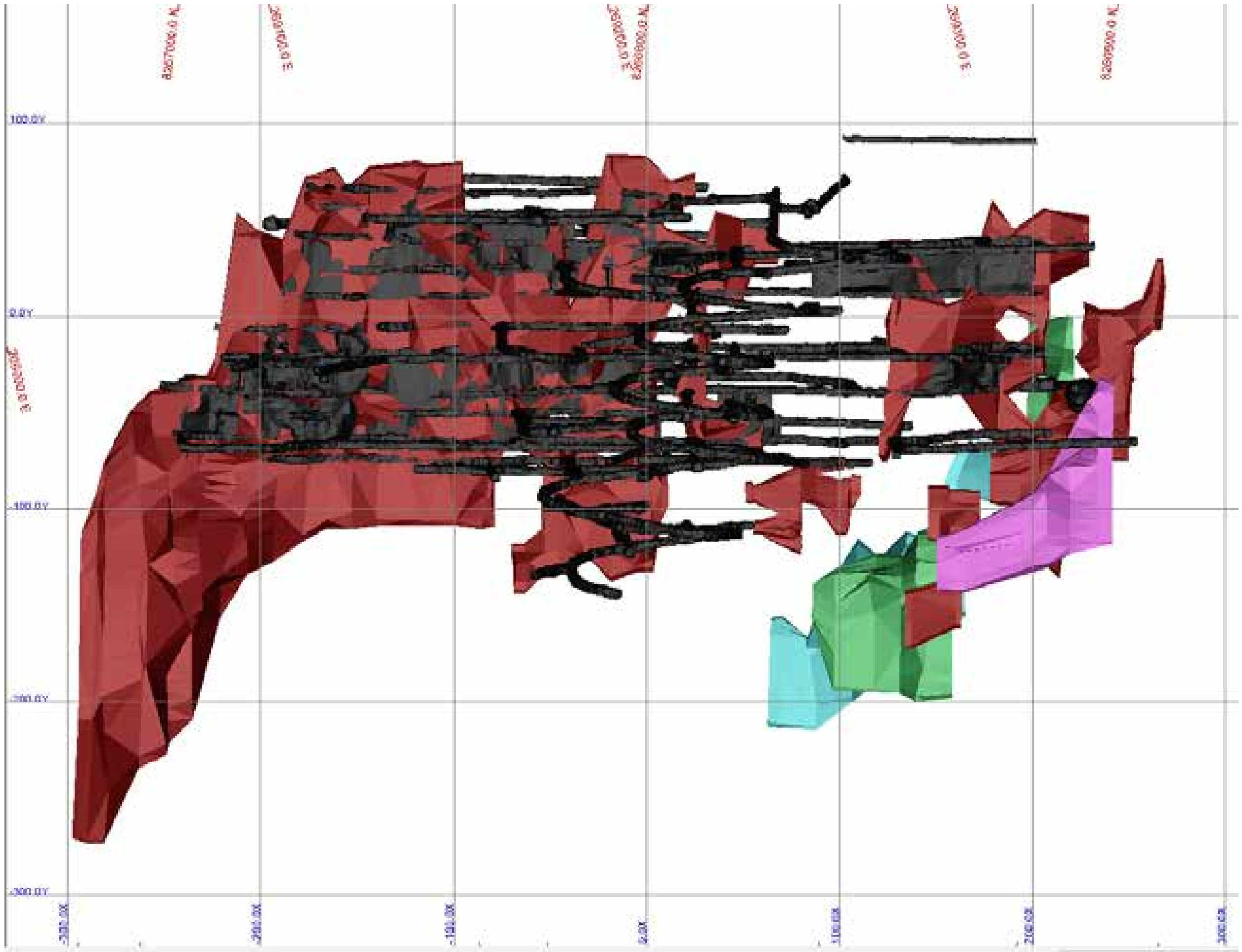
TABLE 14.28
WIREFRAME VOLUMETRICS

Solid	Volume (m^3)	Tonnes[1]
P1 (100)	14,476	40,245
P2 (200) Undepleted	294,688	819,230
P2 (200) Depleted	213,523	593,595
P3 (300)	23,819	66,217
P4 (400)	25,383	70,566
Total (depleted)	**277,202**	**770,623**

(1) Bulk density of 2.78 t/m^3

The wireframes at 1.5 g/t Au cut-off grade and workings are shown on inclined longitudinal in Figures 14.45 and Figure 14.46.

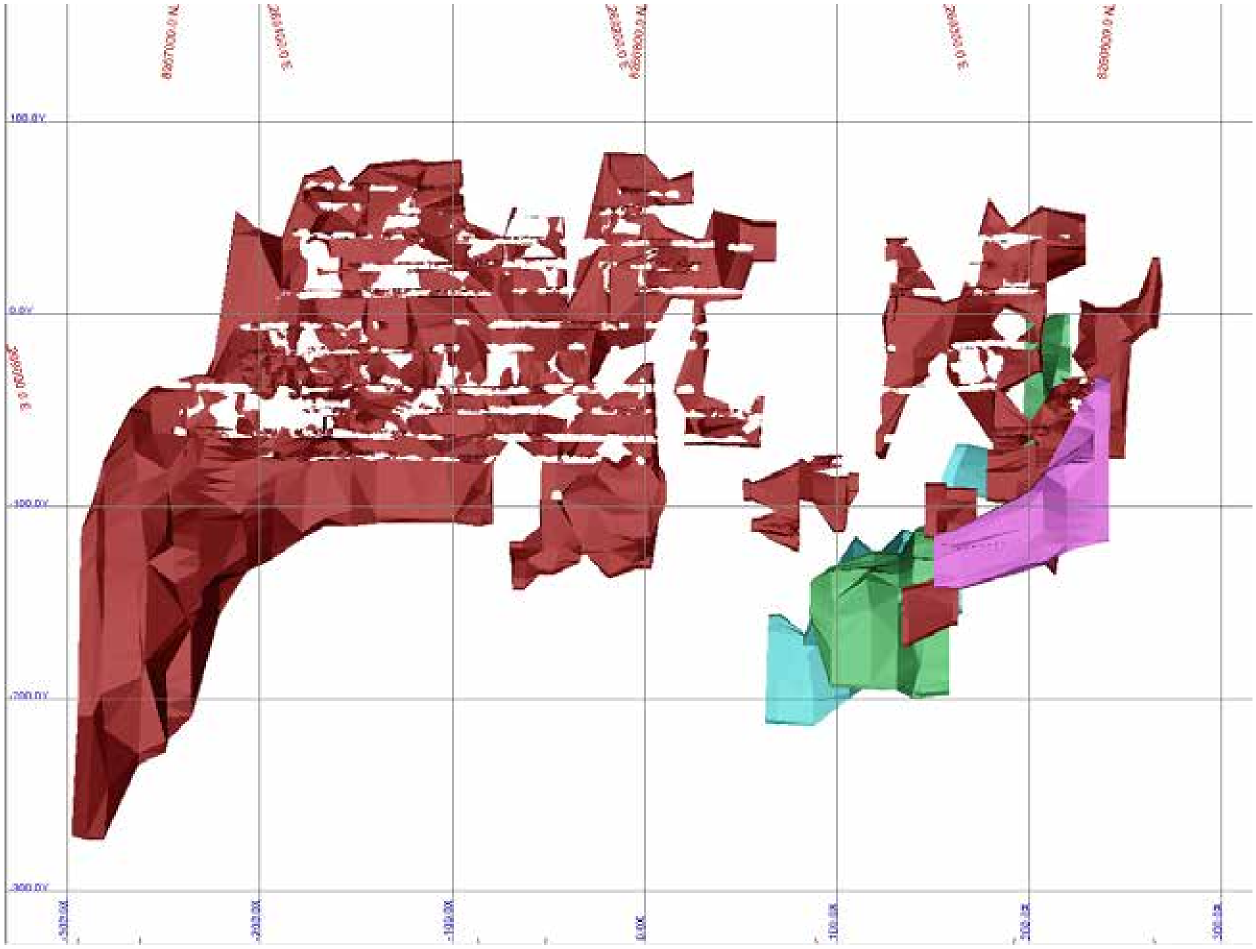
Figure 14.45 Undepleted Wireframes Modelled at 1.5 g/t Au and 3 m Minimum Horizontal Mining Width



**(Inclined Long Section Looking ENE)*

Zone Legend	
	P1 Hanging Wall (Rock Code 100)
	P2 Main Zone (Rock Code 200)
	P3 Footwall (Rock Code 300)
	P4 Footwall (Rock Code 400)

Figure 14.46 Depleted Wireframes Modelled at 1.5 g/t Au and 3 m Minimum Horizontal Mining Width



**(Inclined Long Section Looking ENE)*

14.3.4 Assay Statistics & Grade Capping

Assays captured in wireframes that were constructed pre-fill-in drilling were used for gold grade distributions and capping review. Grade distribution shows extended skew (Poissonian) with possibly two populations, a low grade set up to 1 g/t Au and a second higher grade population. The latter represents the deposit mineralization whereas the former set may be an artefact of bulking up the zone intersections and/or varied assay detection limits.

Histograms and log-probability plots were used to evaluate gold grade distribution and capping curves were utilized to show the impact of capping levels on assay average grade. 3D distribution of high grade assays was examined on-screen to ensure that “outlier” assays were not spatially correlated. Graphs are available in Appendix 3. Results of the grade capping are presented in Tables 14.29 and 14.30. From the tables, it is clear that capping has a significant impact on average grade and grade variability.

TABLE 14.29 CORE ASSAYS CAPPING SUMMARY	
No. of Assays	2,288
Average grade (g/t Au)	4.11
Coefficient of Variation	2.4
Cap Level (g/t Au)	50
No. of Assays Capped	13
% Capped	0.57
% Metal Lost	6.6
Average Grade of Capped Assays (g/t Au)	3.84
Coefficient of Variation Capped	1.8

TABLE 14.30 CHIP ASSAYS CAPPING SUMMARY	
No. of Assays	4,242
Average grade (g/t Au)	2.52
Coefficient of Variation	3.7
Cap Level (g/t Au)	50
No. of Assays Capped	19
% Capped	0.45
% Metal Lost	9.6
Average Grade of Capped Assays (g/t Au)	2.27
Coefficient of Variation Capped	2.7

Assay statistics for the current resource wireframes are presented in Table 14.31.

TABLE 14.31 ALL ZONES ASSAY STATISTICS			
Core Assays			
Statistic	**Length (m)**	**Au g/t**	**Au g/t Capped**
Count	2,298	2,298	2,298
Sum	2,228.22	-	-
Minimum	0.31	0.00	0.00
25th Percentile	0.90	0.13	0.13
Median	1.00	1.26	1.26
75th Percentile	1.00	4.36	4.36
Maximum	2.00	175.48	50.00
Average	0.97	3.59	3.37
Weighted Mean	-	4.04	3.84
Variance	0.03	91.72	49.31
Standard Deviation	0.18	9.58	7.02
Coefficient of Variation	0.19	0.19	0.19
Skewness	0.68	8.63	3.61
Kurtosis	5.59	115.81	16.16
95th Percentile	1.23	16.27	16.27
97th Percentile	1.33	23.55	23.55
98th Percentile	1.41	26.73	26.73
99th Percentile	1.55	39.47	39.47

TABLE 14.31 ALL ZONES ASSAY STATISTICS			
Core Assays			
Statistic	**Length (m)**	**Au g/t**	**Au g/t Capped**
99.5th Percentile	1.77	50.66	49.89
Channel Chip Sample Assays			
Statistic	Length (m)	Au g/t	Au g/t Capped
Count	2,371	2,371	2,371
Sum	1,211.87	-	-
Minimum	0.15	0.00	0.00
25th Percentile	0.47	0.11	0.11
Median	0.50	0.93	0.93
75th Percentile	0.50	4.41	4.41
Maximum	1.20	215.16	50.00
Average	0.51	4.65	4.19
Weighted Mean	-	4.41	3.99
Variance	0.02	148.09	65.30
Standard Deviation	0.13	12.17	8.08
Coefficient of Variation	0.26	2.62	1.93
Skewness	1.82	8.28	3.38
Kurtosis	5.32	100.03	13.09
95th Percentile	0.80	19.97	19.97
97th Percentile	0.95	27.34	27.34
98th Percentile	1.00	34.37	34.37
99th Percentile	1.00	47.18	47.18
99.5th Percentile	1.00	82.68	50.00
Combined Assay Statistics			
Statistic	Length (m)	Au g/t	Au g/t Capped
Count	4,659	4,659	4,659
Sum	3,447.00	-	-
Minimum	0.15	0.00	0.00
25th Percentile	0.50	0.12	0.12
Median	0.70	1.06	1.06
75th Percentile	1.00	4.40	4.40
Maximum	2.00	215.16	50.00
Average	0.74	4.38	4.02
Weighted Mean	-	4.18	3.87
Variance	0.08	122.97	57.10
Standard Deviation	0.28	11.09	7.56
Coefficient of Variation	0.38	2.53	1.88
Skewness	0.51	8.69	3.53
Kurtosis	-0.04	111.88	14.86
95th Percentile	1.15	18.28	18.28
97th Percentile	1.22	24.91	24.91
98th Percentile	1.30	31.27	31.27
99th Percentile	1.44	41.86	41.86
99.5th Percentile	1.60	70.32	50.00

14.3.5 Compositing

3D wireframes were intersected by drill holes, channels and assays within the intersections coded by zone. Sample lengths for assays were reviewed (Figure 14.47) and a 1.5 m composite length was selected as appropriate for the sample lengths and a 3 m cubed block size. 98.6% of the core assay lengths, and all the channel sample lengths, are ≤1.5 m.

Compositing was carried out down-hole at nominal 1.5 m lengths but adjusted to equal lengths across the zone intercept to ensure the effect of bulking out to the minimum mining width is transferred to the composites. Since such composite lengths are variable, regularization by this method is only minimally compromised and certainly the impact is far outweighed by variability in zone intercept lengths caused by fan drilling. Approximately 49% of the drill intersections in the zones intersect the zones at angles exceeding 45° indicating less than optimum intersection angles.

Composite statistics are summarized in Table 14.32.

Figure 14.47 Sample Length Statistics

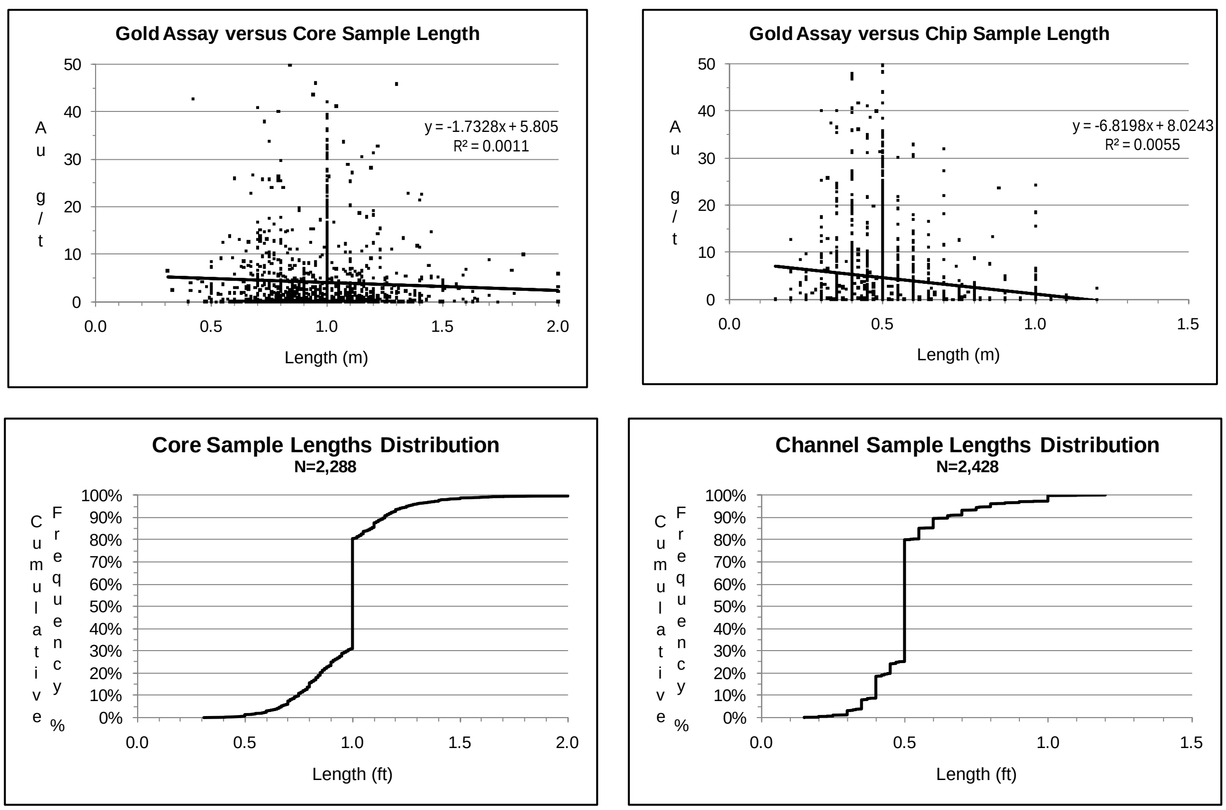


TABLE 14.32
CORE AND CHIPS COMPOSITES STATISTICS

Zone P2			
Statistic	**Length**	**Au g/t**	**Au g/t Cut**
Count	2,341	2,341	2,341
Sum	3,449.91	-	-
Minimum	0.00	0	0
25th Percentile	1.40	0.23	0.23
Median	1.49	1.44	1.44
75th Percentile	1.58	4.20	4.20
Maximum	2.25	98.73	40.08
Average	1.47	3.68	3.44
Wtd. Mean	-	3.74	3.49
Variance	0.07	47.92	29.08
Standard Deviation	0.26	6.92	5.39
Coefficient of Variation	0.18	1.88	1.57
Skewness	-2.88	5.51	2.96
Kurtosis	15.48	48.03	10.75
95th Percentile	1.80	14.34	14.23
97th Percentile	1.84	19.23	18.24
98th Percentile	1.87	23.81	21.33
99th Percentile	2.07	32.38	28.27
Zones P1, P3 & P4			
Statistic	**Length (m)**	**Au g/t**	**Au g/t Cut**
Count	256	256	256
Sum	369.24	-	-
Minimum	0.00	0.00	0.00
25th Percentile	1.41	0.35	0.35
Median	1.50	1.51	1.51
75th Percentile	1.57	3.70	3.70
Maximum	2.00	54.43	35.19
Average	1.44	2.77	2.65
Wtd. Mean	-	2.89	2.76
Variance	0.10	20.58	12.59
Standard Deviation	0.31	4.54	3.55
Coefficient of Variation	0.22	1.64	1.34
Skewness	-3.67	6.78	4.01
Kurtosis	14.32	68.47	28.45
95th Percentile	1.67	8.15	8.15
97th Percentile	1.67	10.07	10.07
98th Percentile	1.68	11.09	11.09
99th Percentile	1.77	15.18	15.18
All Zones			
Statistic	**Length (m)**	**Au g/t**	**Au g/t Cut**
Count	2,597	2,597	2,597
Sum	3,819.14	-	-
Minimum	0.00	0.00	0.00
25th Percentile	1.40	0.23	0.23
Median	1.50	1.44	1.44
75th Percentile	1.58	4.11	4.11
Maximum	2.25	98.73	40.08
Average	1.47	3.59	3.37
Wtd. Mean	-	3.66	3.42
Variance	0.07	45.30	27.51
Standard Deviation	0.27	6.73	5.24
Coefficient of Variation	0.18	1.87	1.56
Skewness	-3.02	5.63	3.04
Kurtosis	15.57	50.09	11.60
95th Percentile	1.79	13.93	13.87
97th Percentile	1.84	18.45	17.68
98th Percentile	1.86	22.91	20.87
99th Percentile	2.04	30.37	27.58

14.3.6 Bulk Density

Water immersion bulk density tests were carried out on 379 drill core samples from 30 drill holes. Samples of seven rock types were tested as well as saprolite of undisclosed precursor rock units. Averages for the rock types tested are shown in Table 14.33.

TABLE 14.33
ROCK UNIT BULK DENSITIES

Rock Unit	Average t/m^3
Biotite Schist/Sheared Tonalite	2.83
Meta-Conglomerate	2.65
Feldspathic Arenite	2.73
Mylonite, Muscovite Schist/Trap/Host Rock	2.81
Quartz Veining	2.71
Saprolite	2.44
Meta-Tonalite	2.79
Mineral Wireframe	2.78

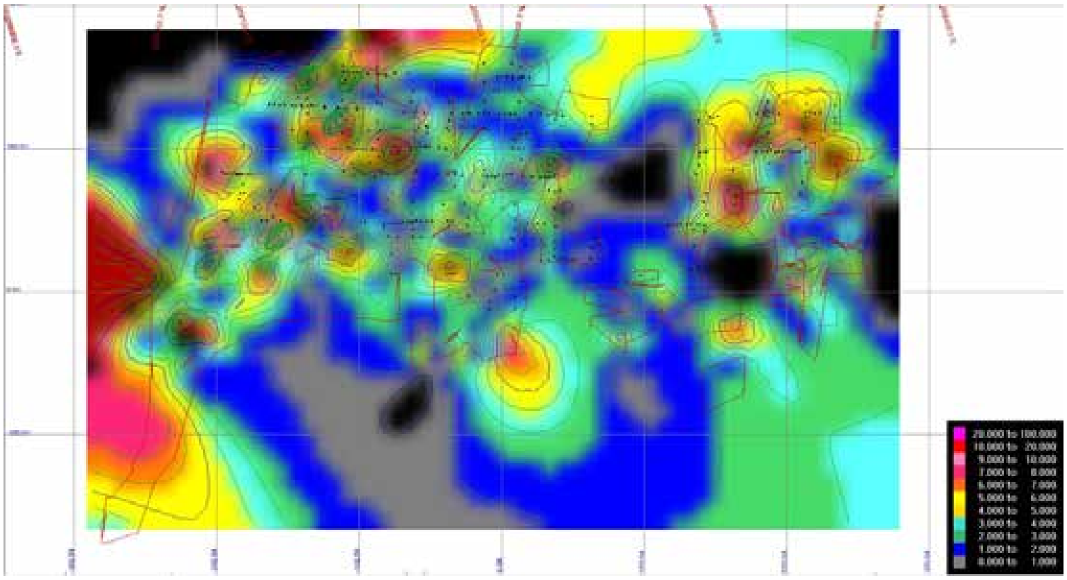
A GEMS bulk density block model attribute was prepared and populated by nearest neighbour interpolation of bulk density values assigned to the GEMS lithology table based on the average bulk density value for the rock types. Intervals in the lithology table correspond to assay intervals and thus provide detailed information on rock unit distribution. An average bulk density value of 2.72 t/m^3 was the default for minority rock types not tested. The default is the average between tonalite and conglomerate. The average bulk density for the wireframe material is 2.78 t/m^3.

14.3.7 Trend Analysis and Variography

Grade, true width and grade-thickness contouring of wireframe drill hole intersections was carried out on an inclined longitudinal section in the plane of the main zone P2 (200) wireframe. The contours of the grades and thickness disclosed a 47° plunge of the mineralization to the northwest at an azimuth of 318° (Figures 14.48 to 14.50). Variography and trend analysis was performed on data existing before the fill-in drilling was carried out. However, since the fill-in drilling contributes to only a small part of the P2 resource drilling database, results are considered acceptable.

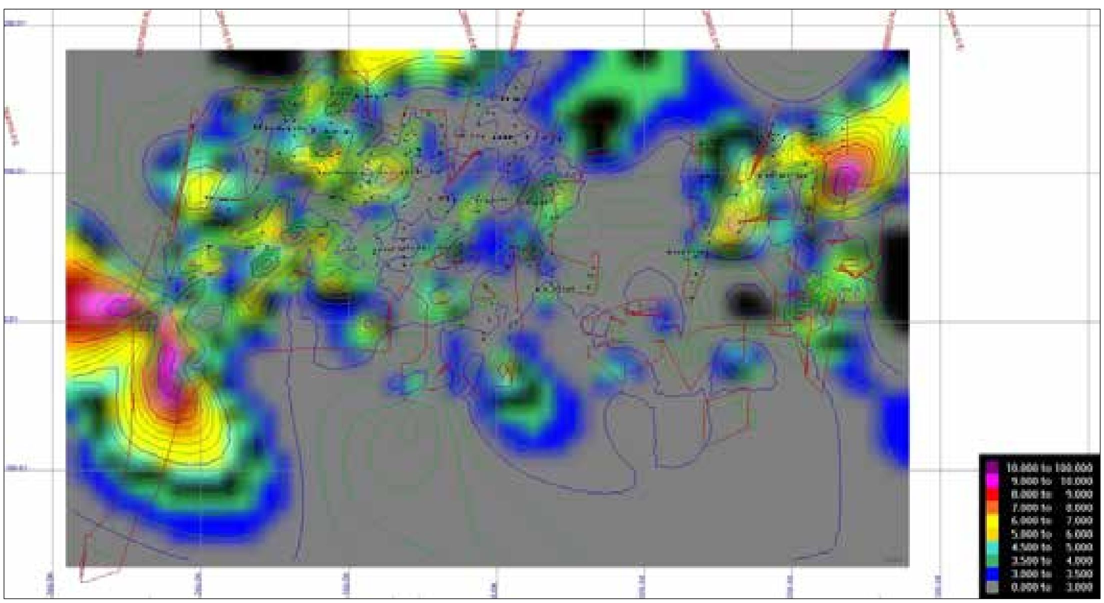
A down-hole linear semi-variogram was prepared for the 1.5 m composites to determine the nugget effect at 40%. This nugget effect was used for the 3D semi-variograms prepared to evaluate the range of continuity of gold grades. Table 14.34 lists the ranges for the semi-variograms prepared. Variance normalized semi-variograms, based on two-model nested spherical modelling, were prepared for strike and dip of the P2 zone, and for the major axis of maximum continuity at 318°/-47°, the intermediate axis at 118°/-43° and minor axis at 048°/-13°. Kriging profiles in GEMS format were prepared from the latter semi-variograms for gold grade interpolation. Semi-variograms are available in Appendix 3.

Figure 14.48 Zone P2 Grade Contours (Au g/t)*



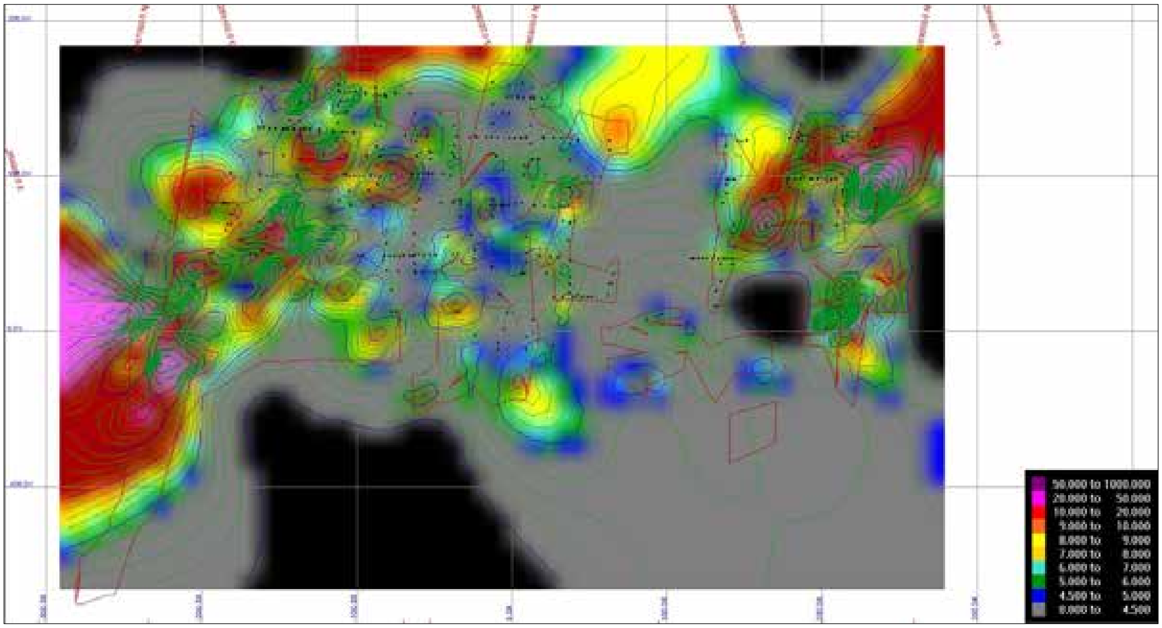
*(Looking ENE)

Figure 14.49 Zone P2 True Width Contours (m)



*(Looking ENE)

Figure 14.50 Zone P2 Grade-Thickness Contours (Au g/t-m)



**(Looking ENE)*

TABLE 14.34 SEMI-VARIOGRAM RANGES	
Vector	**Range (m)**
DownHole Linear	11
3D Strike (147°/0°)	32
3D Dip (237°/-76°)	34
3D 318°/-47° Major Axis-Maximum Continuity	50-55
3D 118°/-43° Intermediate Axis	50-55
3D 048°/-13° Minor Axis	7-8

14.3.8 Block Model

A block model was set-up to encompass the three mineralized wireframes and historic workings (Table 14.35). A block size of 3 m cubed represents a workable size for selective mining methods and the zone widths as well as being approximately ¼ of the detailed drill hole density on 12.5 m section spacing, a common industry practice. Model rotation is GEMS convention whereby the X axis is rotated clockwise 60° to 150°.

TABLE 14.35 BLOCK MODEL PARAMETERS			
	X (Col)	**Y (Row)**	**Z (Level)**
Origin	268,900	8,266,565	500
Block Size	3	3	3
No. of Blocks	222	97	166
Distance (m)	666	291	498
Rotation°	-60		
Total Blocks	**3,574,644**		
Volume (m^3)	96,515,388		

14.3.9 Search Strategy and Interpolation

The search strategy (Table 14.36) was designed for the anisotropic capture of close spaced composites in tight UG fan drilling (<12.5 m spacing) and channel chip sampling at 3 m (PK-1) as well as in wider spaced (±25 m) surface drill holes (PK-2) and to preserve local grade diversity (i.e. not over smooth) given a 40% nugget effect and use of OK to decluster, the latter important where chip samples are available. The initial interpolation pass was designed at half the variogram range but was sufficient to capture holes on at least two adjacent cross-sections for main lens P2.

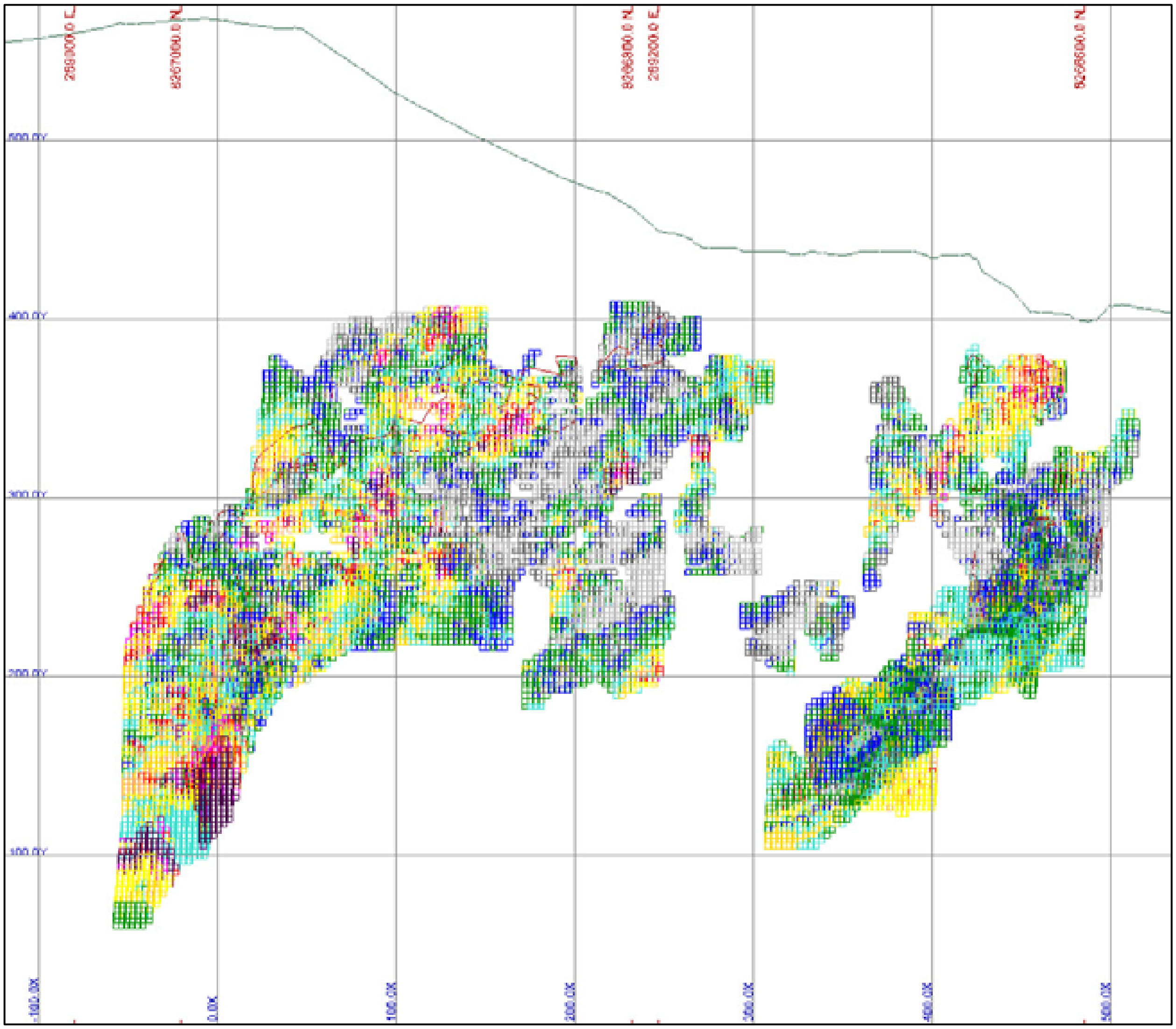
TABLE 14.36 SEARCH STRATEGY AND INTERPOLATION PARAMETERS				
	P1 Lens and P2 Main Wireframes			**P3 & P4 Lenses**
Search Ellipse	**P1**	**P2**	**P3**	**P1-P3**
Z Rotation°	5	5	5	-6
X Rotation°	82	82	82	73
Z Rotation°	47	47	47	47
X (m)	28	55	110	same
Y (m)	14	28	55	same
Z (m)	4	7	14	same
Values ≥25 g/t	12.5	12.5	12.5	same
Composites				
Pass	Minimum#	Maximum#	Maximum# per Hole	Ellipse
Pk-1	4	12	3	P1
Pk-2	4	12	3	P2
Pk-3	2	12	-	P3

Grade interpolation was carried out by OK in three passes for the main zone P2 and four passes for the other lenses. Check on the OK estimate was carried out by alternative estimation methods ID2 and NN. No declustering was done for ID2, however, the results are very close on a global basis. For the main zone P2, most of the interpolation (76%) was completed in the first pass (Table 14.37). All blocks in the wireframes were populated by the 3rd pass. Interpolation parameters, including number of composites used, number of holes used, distance to the nearest composite, and interpolation pass were recorded in the block model attributes for review and model validation.

TABLE 14.37 GRADE INTERPOLATION STATISTICS		
Zone P2		
Pass	**#**	**%**
1	15,898	76
2	3,309	16
3	1,744	8
4	0	0
	20,951	100
Zones P1, P3, P4		
Pass	**#**	**%**
1	2,396	42
2	2,602	46
3	629	11
4	62	1
	5,589	100

The distribution of block grades is shown on inclined longitudinal section in Figure 14.51.

Figure 14.51 Distribution of Block Grades



*(Vertical Longitudinal Section Looking ENE)

Legend (Au g/t)

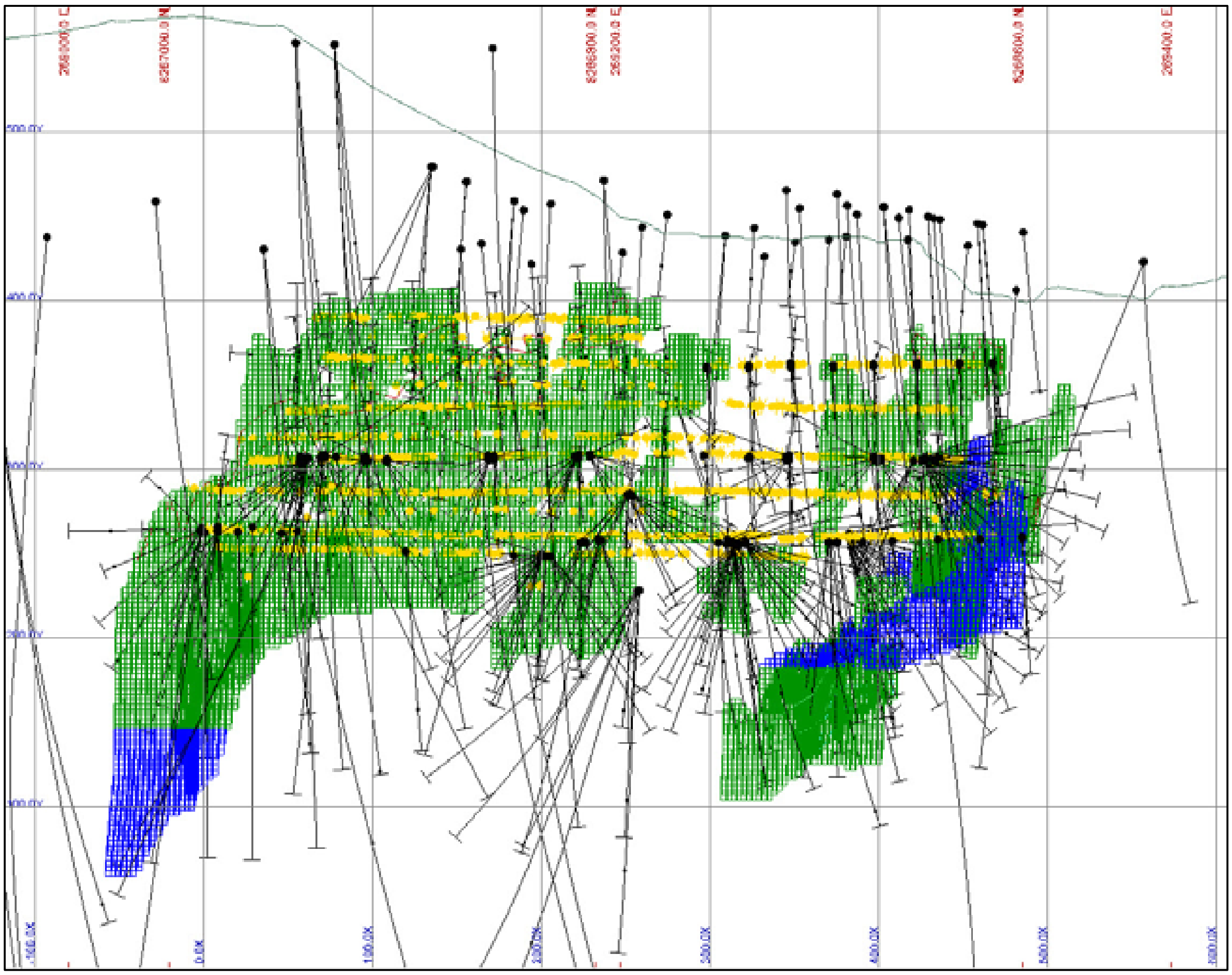
>= Lower Bound	< Upper Bound
0.00001	1.00000
1.00000	1.50000
1.50000	2.00000
2.00000	3.00000
3.00000	4.00000
4.00000	5.00000
5.00000	6.00000
6.00000	7.00000
7.00000	8.00000
8.00000	9.00000
9.00000	10.00000
10.00000	100.00000

14.3.10 Mineral Resource Classification

Resource block classification was based on a review of interpolation parameters, variogram ranges and drill hole/sampling density (Figure 14.52). Most the main zone (P2; rock code 200) is

Indicated and was 93% interpolated after Pass 2 with search distance at the variogram range of 30 m, and has a drill hole density of 25 m or tighter except for the down plunge segment at the NW extreme end of the deposit that is defined by several down dip holes only. Footwall zones P3 and P4 (rock codes 300 and 400) have few holes, with uncertain correlation because of high angle intersections to down dip drilling and are partially classed as Inferred. P&E notes that kriging variance, which is a reflection of sampling geometry, versus distance from a block to the nearest composite, is commonly used to support classification (Figure 14.53). In this case, the distance suggested is approximately 11 m and is significantly shorter than the variography indicates, which is a reflection of spatial irregularity in intersections in the zones arising from the underground fan drilling.

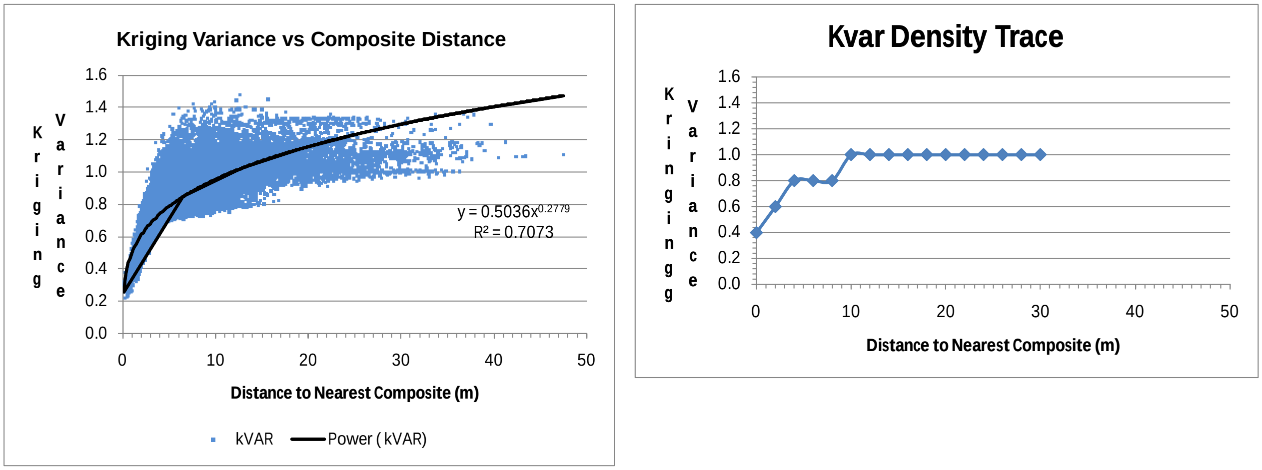
Figure 14.52 Mineral Resource Estimate Classification



(Vertical Longitudinal Section Looking ENE)

Legend	
(green)	Indicated Resource
(blue)	Inferred Resource

Figure 14.53 Kriging Variance versus Distance to Nearest Composite



14.3.11 Model Validation

P&E is aware that Yamana carried out mine reconciliations in 2013 and 2014 for PPQ during operations. However the bulk of current resources are located at depth below past mining and cut-offs during mine operation likely differ significantly from those used for the current resource estimate. P&E does not have the Yamana block model with which to compare to the P&E modelling or confirm past reconciliations. Consequently P&E has not commented on past Yamana reconciliations with respect to current P&E modelling.

Validation of the grade interpolation and the P&E block model was carried out by on-screen review of block grades versus drill hole composites and other block model estimation parameters, by comparison of resource grades to the grades of assays, composites and zone intersections, comparison to alternate ID2 and NN interpolations, and review of the volumetrics of wireframes versus reported resources (Table 14.38). P&E notes that the global wireframe intercept, composite grades and block model global grades are comparable whereas assay grades are higher than the block model by 8%. This is accounted for in part by volume-variance effect and because the assays grades do not include missing explicit and implicit intervals that are incorporated in the composites and model at zero grade. The NN model global grade is 10% higher than the OK model and within the 10% generally acceptable tolerance for difference between methods. Some of the difference is explained in that NN search is isotropic in contrast to the anisotropic search for OK and ID2 and the NN method does not employ a restricted area of influence for composites grading ≥25 g/t Au. In P&E's opinion the block model is validated.

TABLE 14.38
BLOCK MODEL VALIDATION

Global Volumetrics (m^3)		
Wireframes	**Reporting**	**Variance**
277,202	277,052	-0.05%

All Zones	Au g/t	Variance[1]
Assays	3.87	8%
Intercepts	3.67	3%
Composites[2]	3.37	-6%
Block Model	3.58	-

Method	Tonnes	Grade	Ounces	Variance[1]
OK	769,647	3.58	88,538	-
ID^2	769,647	3.59	88,761	0.3%
NN	769,647	3.94	97,441	10%

Notes:

1) With respect to the OK block model

2) Includes explicit and implicit missing assays at zero grade

14.3.12 Mineral Resource Estimate Reporting

The Pau-a-Pique Mineral Resource Estimate at a cut-off grade of 1.5 g/t Au is summarized in Table 14.39. Mineral Resource Estimate sensitivity to cut-off grade is presented in Table 14.40 and tonnage-grade profiles for Indicated and Inferred Mineral Resources are shown in Figure 14.54.

TABLE 14.39
PAU-A-PIQUE MINERAL RESOURCE ESTIMATE AT A CUT-OFF GRADE OF 1.5 G/T AU(1-10)

Zone	Indicated			Inferred		
	Tonnes (t)	**Au (g/t)**	**Contained Au oz**	**Tonnes (t)**	**Au (g/t)**	**Contained Au oz**
P1	-	-	-	31,000	3.26	3,300
P2 Main Zone	438,000	4.16	58,600	54,000	5.92	10,400
P3	27,000	3.33	2,900	24,000	3.11	2,300
P4	54,000	3.51	6,100	8,000	2.93	700
Total	**519,000**	**4.05**	**67,600**	**117,000**	**4.45**	**16,700**

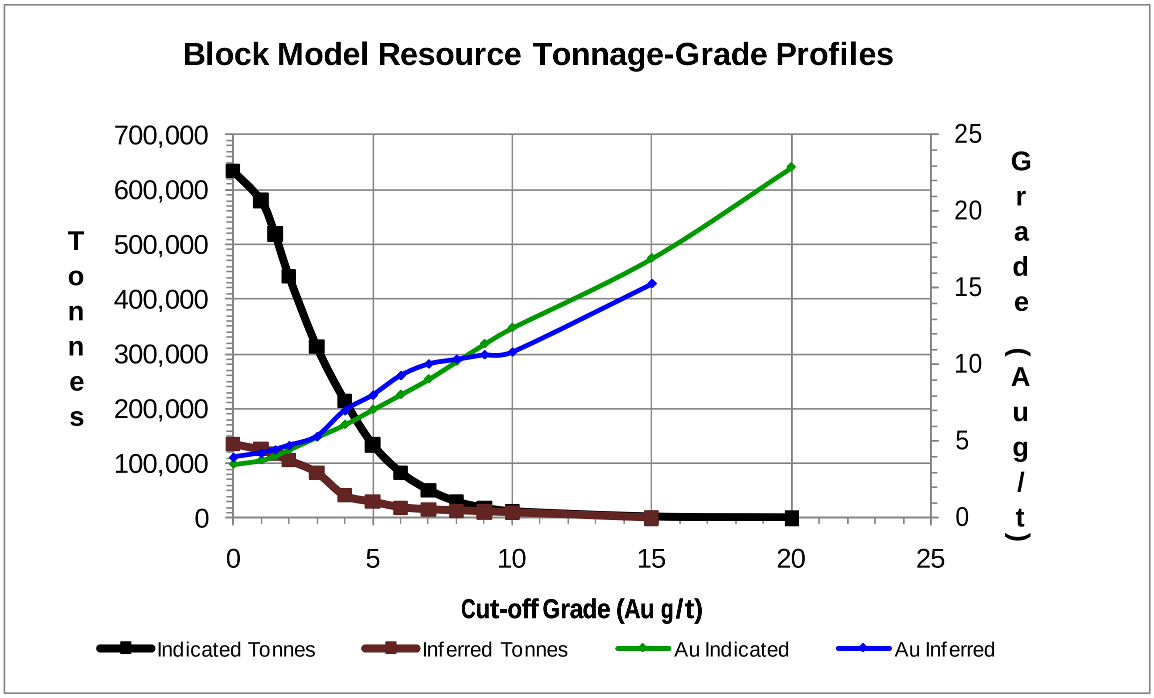
(1) CIM Definitions were followed for Mineral Resources.
(2) The Qualified Person for this Mineral Resource Estimate is: Richard Routledge M.Sc. (Applied), P.Geo.
(3) Mineral Resources are estimated from surface and underground diamond drilling and core sampling and underground chip sampling by conventional 3D block modelling based on wireframing at a 1.5 g/t Au cut-off grade and ordinary kriging grade interpolation.
(4) For the purpose of resource estimation, assays were capped at 50 g/t Au and composites >25 g/t Au were restricted to 12.5 m area of influence.

(5) *The mineral resource estimate is based on a Cut-Off Grade of 1.5 g/t Au derived from a Au price: US$1,275 /oz, costs of US$29/t for mining, US$11/t for processing, US$10/t for G&A and US$7/t for mill feed surface transportation, at a 93% process recovery.*
(6) *A bulk density model based on rock type was used for volume to tonnes conversion with resources averaging 2.77 tonnes/m^3.*
(7) *Mineral Resources are estimated from the 410 m EL to the 65 m EL, or from approximately 30 m depth to 500 m depth from surface.*
(8) *Mineral Resources are classified as Indicated and Inferred based on drill hole spacing, interpreted geologic continuity and quality of data.*
(9) *Mineral Resources, which are not mineral reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*
(10) *The quantity and grade of reported Inferred Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as an Indicated or Measured Mineral Resource and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured Mineral Resource category.*

TABLE 14.40
PAU-A-PIQUE MINERAL RESOURCE ESTIMATE SENSITIVITY
Aura Minerals Inc. - As of May 2016

Cut-Off Grade	Indicated				Inferred			
	Tonnes	Bulk Density	Au	Au	Tonnes	Bulk Density	Au	Au
Au g/t	(000's)	t/m^3	g/t	oz	(000's)	t/m^3	g/t	oz
10.0	11	2.78	12.41	4,500	10	2.81	10.81	3,500
9.0	18	2.78	11.36	6,400	12	2.81	10.64	4,000
8.0	29	2.78	10.21	9,500	14	2.81	10.34	4,500
7.0	50	2.78	9.05	14,700	15	2.81	10.04	5,000
6.0	83	2.78	8.04	21,300	20	2.81	9.28	5,800
5.0	134	2.78	7.06	30,300	30	2.81	8.00	7,600
4.0	214	2.78	6.09	41,800	41	2.80	7.00	9,300
3.0	312	2.78	5.27	52,900	84	2.80	5.31	14,300
2.0	442	2.78	4.45	63,200	106	2.80	4.72	16,100
1.5	519	2.78	4.05	67,600	117	2.80	4.45	16,700
1.0	580	2.78	3.76	70,100	126	2.80	4.23	17,100
Total Wireframe	**633**	**2.77**	**3.50**	**71,200**	**136**	**2.79**	**3.96**	**17,300**

Figure 14.54 Graphic Illustration of PPQ Resource Sensitivity to Cut-Off Grade



14.3.13 Pau-a-Pique Mineral Resource Estimate Conclusions and Recommendations

Pau-a-Pique gold mineralization consists largely of free gold accompanied by sulphides hosted in mylonite, muscovite schist, biotite schist, quartz veins as well as meta-tonalite and metaconglomerate that occur along and adjacent to sheared contacts between meta-tonalite and meta-conglomerate. Mineralization is epigenetic, hydrothermal in origin and is structurally controlled. There is a high "nugget" (40%) effect and high grades may be localized in areas of restricted dimensions.

The Mineral Resource Estimate for the Pau-a-Pique Project was estimated by conventional 3D computer block modelling methods employing Dassault Systemes Geovia mining software V6.4 and V6.71 ("GEMS"). The estimate was undertaken according to NI 43-101 standards.

CIM definitions were followed for the Mineral Resource Estimate and are based on:

- 32,554 m of surface diamond drilling and underground fan diamond drilling in 313 holes, core sampling and assaying as well as underground face channel chip sampling and assaying totalling 2,428 samples for 1,241.73 m.
- Wireframing at a 1.5 g/t Au cut-off grade over a 3 m minimum horizontal mining width.
- Ordinary kriging grade interpolation.
- Alternative estimations, using inverse distance squared and nearest neighbour methods, validate the ordinary kriging method in P&E's opinion.
- The wirefame cut-off results locally in narrow intersections of gold mineralization with grades >1.5 g/t Au on trend within the zones not meeting minimum mining width. For the generally lower grade footwall and hanging wall lenses P3 and P4 where drilling is less intensive and there is no mining history, the interpretation of

mineralization continuity may be affected in that alternative interpretations of continuity are possible and confidence of the resource interpretation is reduced.

The Mineral Resource Estimate was classified as Indicated and Inferred based on drill hole spacing and data quality, channel sampling locations, confidence in the assaying and geologic confidence in the zones interpretation and grade continuity. P&E cautions that the Indicated resources held in remnant pillars, sills and "skins" left in stopes may not all be recoverable pending engineering study.

In P&E's opinion, the Mineral Resource Estimate is reasonable and has been undertaken according to industry standard practice.

P&E offers the following recommendations:

- Drill hole down hole surveys should be reviewed for implausible readings and these should be removed and the resulting re-positioning of the hole toe examined for impact on the resource wireframing.
- Additional drilling is recommended for the west target zone to identify the mineral resource potential.
- A structural study is recommended to identify and model major gold-bearing shear zones in the deposit for future exploration drill targets.
- It is strongly recommended that definition drilling be carried out in the Indicated Resources contained in the NNW lower portion of main zone P2 and the foot wall lenses P3 and P4 in the SSE portion of the deposit, before their development.

14.4 MINERAL RESOURCE ESTIMATE FOR THE PROJECT

The total Mineral Resource Estimate for the Project is presented in Table 14.41.

TABLE 14.41
TOTAL MINERAL RESOURCE ESTIMATE FOR THE PROJECT

Measured & Indicated	Tonnes (t)	Au (g/t)	Contained Au oz
Lavrinha	1,300,000	2.25	94,200
Ernesto	734,000	6.70	158,200
Pau-a-Pique	519,000	4.05	67,600
Total Measured & Indicated	**2,553,000**	**3.89**	**320,000**
Inferred	**Tonnes (t)**	**Au (g/t)**	**Contained Au oz**
Lavrinha	283,000	2.51	22,800
Ernesto	308,000	6.30	62,400
Pau-a-Pique	117,000	4.45	16,700
Total Inferred	**708,000**	**4.48**	**101,900**

Contained metal may not sum in the above table due to rounding

15.0 MINERAL RESERVES

15.1 ERNESTO

15.1.1 Mineral Reserve Estimate Introduction

The current P&E Mineral Reserve Estimate presented in this Report has been prepared in full conformance and compliance with the "CIM Standards on Mineral Resources and Reserves – Definitions and Guidelines" as referred to in NI 43-101 and Form 43-101F, Standards of Disclosure for Mineral Projects and in force as of the effective date of this Report, which is July 31, 2016.

No Inferred Mineral Resource has been used in the Mineral Reserve Estimate.

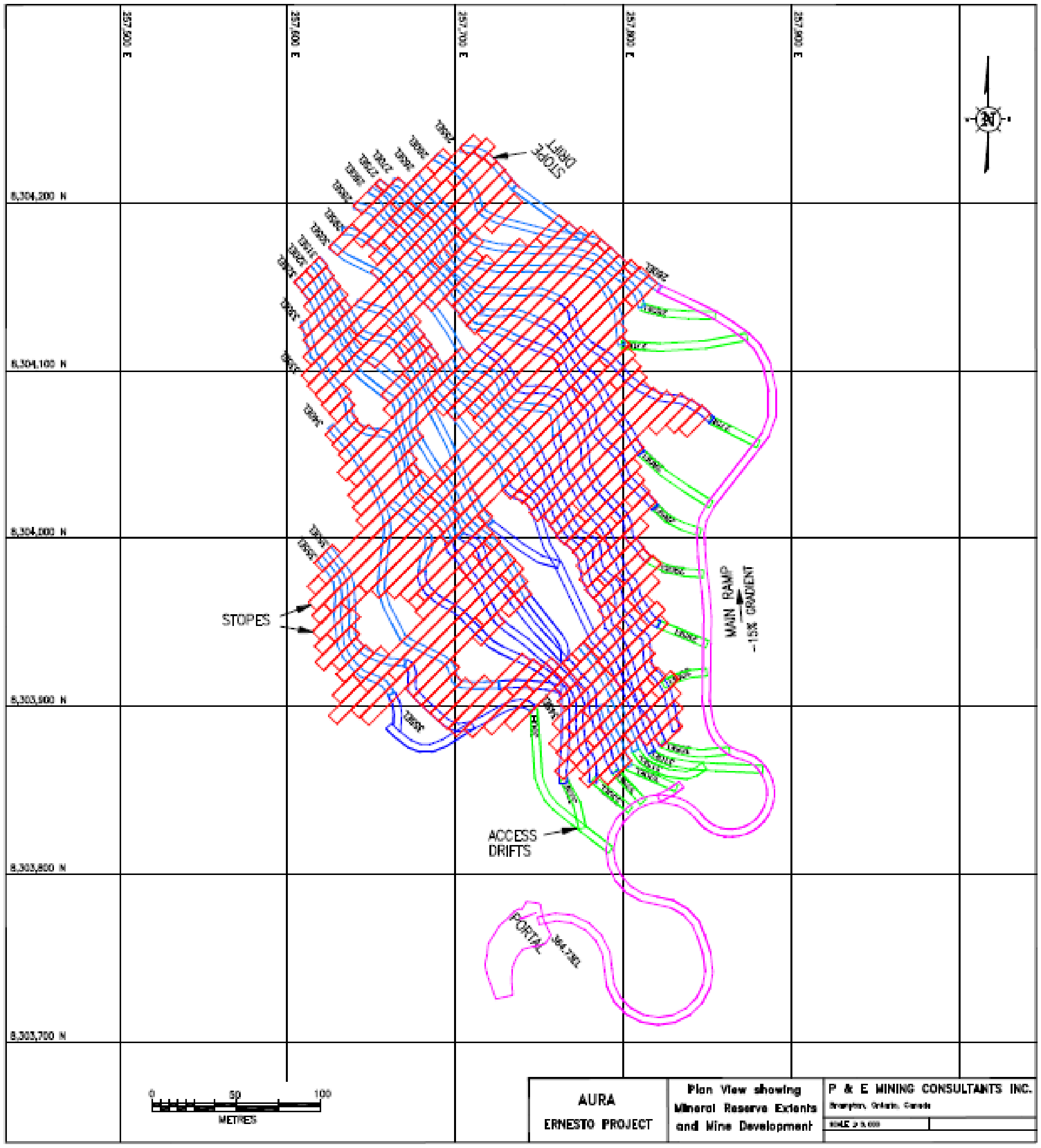
15.1.2 Mineral Reserve Estimate – Ernesto Deposit

The P&E Mineral Reserve Estimate is for the "Lower Trap" portion of the Ernesto Deposit where the "Lower Trap" portion of the Ernesto Deposit is contained by planes along the following mine sections and elevations:

- Mine sections 8,303,850N and 8,304,250N. These North oriented sections are 400 m apart.
- Mine sections 257,600E and 257,850E. These East oriented sections are 250 m apart.
- Mine elevations 255 m and 355 m. These elevations are 100 m apart.

Figure 15.1 shows a plan view of the physical extent of the proposed mine development and Mineral Reserve Estimate.

Figure 15.1 Extent of Proposed Mine Development and Mineral Reserve Estimate



15.1.3 Historical Mineral Reserve Estimate for the Ernesto Deposit

Ausenco do Brasil Engenharia Limitada (Ausenco, 2010) reported that it prepared the March 17, 2010 Feasibility Study for the Ernesto and Pau-a-Pique mines to provide Yamana Gold Inc ('Yamana') with an estimate of the capital and operating costs for the Ernesto and Pau-a-Pique gold project, and that the Feasibility Study had not been prepared to meet the requirements of the Canadian National Instrument 43-101 standard. As such, the Mineral Reserve Estimate for the Ernesto and Pau-a-Pique Deposits presented in the Ausenco (2010) Feasibility Study are considered to be not compliant with NI 43-101 requirements, and are instead considered to be historical in nature and cannot be relied upon. Ausenco (2010) indicated that:

Gold mining began in the Pontes e Lacerda area during the 18th century and that in the early 1980s, thousands of garimpeiros (artisanal miners) began recovering placer gold along the rivers and streams in the area; these placer deposits were exhausted by the late 1980s. In 1989, garimpeiros began mining weathered bedrock at Ernesto and surrounding areas. From these areas, approximately 60,000 oz of gold had been recovered to date (e.g. as of March 2010).

At Ernesto, garimpeiros had produced 9,000 oz of gold from a small pit in a 3 m thick zone along a 200 m length and from underground workings accessed via seven declines and extending 50 to 60 m down-dip from the surface outcrop. As part of a 2010 Feasibility Study, Yamana had proposed open pit and underground mining. Economic mineralization of the shallow "Middle Trap" (of the Ernesto Deposit) would be recovered by open pit mining, while underground mining would be proposed for the deeper mining ("Lower Trap"). The drift-and-fill mining method had been selected for the Ernesto underground areas in the Ausenco (2010) report with the objective of maximizing the Mineral Resource Estimate using an adequate mining method for the local geomechanical features.

The Ausenco (2010) historical NI 43-101 Mineral Reserve Estimate for the Ernesto Deposit is shown in Table 15.1 for background information purposes only.

TABLE 15.1
AUSENCO (2010) HISTORICAL MINERAL RESERVE ESTIMATE FOR THE ERNESTO DEPOSIT

Ernesto Deposit Portion	Cut-off Grade	Proven		Probable		Proven & Probable		
	Au (g/t)	kt	Au (g/t)	kt	Au (g/t)	kt	Au (g/t)	Au koz
Ernesto "Lower Trap" (Underground)	1.0	876	3.3	1,522	2.94	2,398	3.07	237
Ernesto "Middle Trap" (Open Pit)	0.3			1,771	1.75	1,771	1.75	100
Total		**876**	**3.3**	**3,293**	**2.30**	**4,169**	**2.51**	**337**

Source: *Ausenco (2010). Ernesto Deposit historic mineral estimates extracted from Table 1.10 "Total estimated mineral reserves for the Ernesto and Pau-a-Pique Project".*

15.1.4 Mineral Reserve Estimate for the "Lower Trap" Portion of the Ernesto Deposit

15.1.4.1 Mineral Resource Estimate Conversion to Mineral Reserve Estimate

For the Ernesto Deposit, the Mineral Resource Estimate was converted to a Mineral Reserve Estimate using the following approach:

P&E reviewed and inventoried the Indicated Mineral Resource contained in the resource block model for the Ernesto Deposit. During the course of the study, the HW/FW rocks were re-logged to better define the HW and FW lithologies, particularly the mylonite and saprolite zones. Given that the Mineral Resource Estimate has been classified as Indicated, and that the drill holes have been drilled on a relatively wide grid of 35 m x 35 m with fill-in drilling done primarily on the shallower west side of the deposit, a relatively wide spacing is available for mine planning, and therefore an infill drill program has been designed to better define the Mineral Reserve Estimate before production begins. The purpose of the underground definition drilling program is to provide additional information needed to finalize the level and stope designs prior to drifting in ore and stoping. The definition drilling work would be done using a just-in-time approach and as such Aura would need to efficiently carry out the associated core logging, assaying, geotechnical testing work and to timely update its geological and mineral resource model and revise level and stope design phase by phase. The definition drilling program would provide new information and data on the locations / elevations of the limits of geological zones including the altered mylonite zone; rock quality; folding / discontinuities that may be present between the existing surface diamond drill holes. The extent to which projected ore tonnages and grades, the mine schedule and estimated costs could be affected by the outcome of the definition drilling program and mine design finalization work is uncertain. Stope phases with greater surface diamond drilling density may be insignificantly affected by definition drilling results.

P&E reviewed the physical and grade distributions of the Indicated Mineral Resource blocks to provide information for the development of underground mining method options and subsequent selection of the proposed Drift and Fill ("D&F") mining method. P&E also reviewed the geology of the deposit and the upper surface of the saprolite zone as defined in a surface provided to P&E by Aura. The Inferred Mineral Resource was treated as waste.

The Ernesto Deposit would be one of a number of deposits that Aura was considering to develop and mine concurrently / sequentially. The Ernesto underground mine production would be hauled to the existing surface crusher at the Ernesto process plant and treated. A preliminary Mineral Reserve Estimate cut-off grade was developed using a projected gold price and projected mining, processing and G&A costs and excluding dilution and extraction losses and gold doré bar transportation and refining costs (Table 15.2).

TABLE 15.2
MINERAL RESERVE CUT-OFF-GRADE DETERMINATION

Unit Cost	Units	BCOG
Metal Price	US$/Oz	1,165.00
Metallurgical Recovery	%	93.00%
Payable Metal	%	99.99%
Conversion Factor	Grams to Troy oz	31.104
Mining Operating Cost	US$/t (mined)	62.41
Process Operating Cost	US$/t (processed)	10.30
G&A Operating Cost	US$/t (processed)	6.12
Refining Operating Cost	US$/t (processed)	0.16
Total BCOG Operating Cost	US$/t	78.99
Royalty	%	2.50%
CEFEM Tax	%	1.00%
ERN LOM BCOG	g/t	2.35

The preliminary cut-off grade estimate revealed that the Indicated Mineral Resource was of interest using the Drift and Fill ("D&F") mining method.

P&E developed and selected the proposed D&F mining method for use at the Ernesto Deposit in consultation with Aura. The D&F method was selected taking the following aspects into consideration:

- The typical 15° to approaching 50° dip;
- Variations in strike and mineralized zone thicknesses;
- The characteristics and rock quality of the hangingwall materials (i.e. Mylonite and meta-sediments) and the mylonite:metasediment contact;

- The characteristics and rock quality of the footwall materials (i.e. Mylonite, tonalite, saprolite (altered tonalite);
- The proposed mine layout; the proposed underground definition drilling program and the possibility of future revisions to the mine plan and production schedule depending upon the outcome of the definition drilling and assaying program;
- The proposed six phase stope development and production sequence;
- The spacing and geometry of the shanty back production levels;
- The geometry and dimensions of the proposed stopes;
- Projected groundwater inflows;
- The proposed mining equipment;
- Technical and operational factors affecting stope development and production including drilling, blasting, mucking, geotechnical mine design criterion and projected ground support requirements;
- ADT truck loading and haulage to surface including ore haulage to the mill crusher;
- Waste segregation during mucking;
- Stope and level backfilling requirements;
- Grade control;
- Primary and secondary stope sequencing taking single and multiple cut stopes into consideration;
- Advance and production rates;
- Worker health and safety; and
- Other factors.

As part of an iterative process, P&E revised the D&F mining method concept and the associated mine operating and capital cost estimates. This process resulted in the development of the D&F mining method proposed in the present Technical Report. The final stope outlines were subsequently utilized to:

a) Estimate the tonnage and gold grades of geologic materials contained within the primary and secondary stopes located between adjacent level drifts. P&E estimated the average gold grade of the primary and secondary stope outlines on each level. The vertical limits of the stopes on leach level were defined by the sill elevation of the lower level drift and the sill elevation of the upper level drift.
b) Summarize the stope outline tonnage and grade estimates:

- Reducing the Indicated Mineral Resource tonnage contained within the stope blocks between level drift sill elevations by 5% representing a 5% loss.
- Reducing the waste tonnage contained within the stope outlines between level drift sill elevations by the tonnages of waste that would be separately mined and disposed.
- Estimating the tonnages and grades of ore derived from development in ore, primary stoping and secondary stoping.
- Adding 5% dilution at 0 g/t Au to account for the projected CRF dilution of ore in the secondary stopes.

15.1.4.1.1 Ernesto Dilution

As a result of the above approach, the estimated stope tonnage also included dilution due to the excavation of mylonite in stope backs and specifically above the Indicated Mineral Resource blocks contained within stope limits and below the HW contact above the stopes, as well as from the smoothing of stope back outlines. To summarize, weak mylonite diluting material in the HW will be taken down and mined with the ore as an intentional part of the mining method. The only other type of dilution will be from CRF backfill in secondary stopes.

P&E compared the tonnage and average grade of the Indicated Mineral Resource contained within the stope outlines to the above referenced tonnage and grade estimate in order to approximate the overall dilution factor. The overall factor dilution was back-estimated to be 30% at zero grade taking into consideration the stope outlines which included shanty back drift outlines, and 5% dilution in secondary stopes due to primary stope CRF.

15.1.4.2 Mineral Reserve Estimate – Ernesto Deposit "Lower Trap"

The P&E Mineral Reserve Estimate for the "Lower Trap" portion of the Ernesto Deposit is part of the present Feasibility Study. The Mineral Reserve Estimate is a subset of the Indicated Mineral Resource Estimate for the Ernesto Deposit.

The Mineral Reserve Estimate for the Ernesto Deposit is classified as a "Probable" Mineral Reserve based on an Indicated Mineral Resource. The Mineral Reserve Estimate was not based on any Inferred Mineral Resource.

The Probable Mineral Reserve Estimate for the Ernesto Deposit was completed in compliance with the CIM Definition Standards on Mineral Resources and Mineral Reserves adopted by CIM Council, as amended. The terms "Mineral Reserve", "Probable Mineral Reserve" and "Proven Mineral Reserve" have the meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Definition Standards on Mineral Resources and Mineral Reserves adopted by CIM Council.

The Mineral Reserve Estimate for the "Lower Trap" portion of the Ernesto Deposit was estimated at a 2.35 g/t Au cut-off grade, with an effective date of July 31, 2016, and is presented in Table 15.3.

TABLE 15.3
PROBABLE MINERAL RESERVE ESTIMATE FOR THE "LOWER TRAP" PORTION OF THE ERNESTO DEPOSIT(1-5)

Mineral Reserve Category1	Tonnage (t)	Au (g/t)	Contained Au oz
Probable	868,000	5.03	140,000

1) *The Mineral Reserve is estimated as of July 31, 2016.*
2) *The Mineral Reserve Estimate was developed from the resource model prepared by P&E. The Probable Reserve was derived from Indicated Resources.*
3) *The cut-off grade (2.35 g/t Au) was based on a US$1,165/oz gold price, 93% metallurgical Au recovery, 99.99% payable, royalties and CEFEM tax totalling 3.5%, gold doré bar transport and refining costs totalling US$0.45 / g Au, mine direct and mine indirect costs totalling US$62.41/ t, US$10.30/t processing cost, and US$6.12/t processed for the projected share of the overall multi-mine project G&A cost that would be incurred by the proposed Ernesto underground mine project. The geological continuity of the mineralization was assessed for the Au cut-off grade.*
4) *The Mineral Reserve Estimate tonnage and mined metal have been rounded to reflect the accuracy of the estimate.*
5) *The NI 43-101 Mineral Reserve Estimate for the Lower Trap portion of the Ernesto Deposit set out in Table 15.3 has been reviewed and approved by David Orava, M.Eng., P. Eng., of P&E Mining Consultants Inc., who is a Qualified Person ("QP") for the purposes of this Report, and who is independent of the Company.*

15.1.4.3 Material Factors Affecting Mineral Reserve Estimation

The Mineral Reserve Estimate for the "Lower Trap" of the Ernesto Deposit has taken modifying factors such as, but not limited to, mining, metallurgical, infrastructure, permitting, economic, marketing, legal, social and governmental factors into consideration. Based on available information, P&E is of the opinion that none of these factors are likely to materially affect the development and operation of the proposed Ernesto underground mine. The Mineral Reserve Estimate for the "Lower Trap" portion of the Ernesto Deposit could be materially impacted by lower than projected gold prices, lower gold processing recovery, higher than projected operating and/or capital costs, underground definition drilling results affecting the mine design and production schedule, higher than projected dilution or extraction losses, and geotechnical and hydrogeological conditions that affect projected mine development and operations.

15.2 LAVRINHA

15.2.1 Summary

The Lavrinha Deposit Mineral Reserve Estimate, as of July 31, 2016, is presented in Table 15.4.

TABLE 15.4
LAVRINHA DEPOSIT MINERAL RESERVE ESTIMATE(1-7)

Reserves	Tonnes (t)	Au (g/t)	Contained Au oz
Proven	67,000	1.85	4,000
Probable	1,043,000	1.68	56,300
Total	**1,110,000**	**1.69**	**60,300**

1) *CIM definitions were followed for Mineral Reserves.*
2) *The Mineral Reserve Estimate is estimated as of July 31, 2016.*
3) *Mineral Reserve Estimate for the Lavrinha Deposit was prepared under the supervision of Marcelo Batelochi, Ausimm (CP 205477).*
4) *Mineral Reserve Estimate cut-off grade was 0.48 g/t Au.*
5) *Lavrinha Mineral Reserve Estimate was derived using an average short-term gold price of US$1,100 per ounce.*
6) *Bulk density average was 2.78 t/m^3.*
7) *Numbers may not add due to rounding.*

15.2.2 Pit Optimization

The Lavrinha Mineral Reserve Estimate is based on a Life-Of-Mine ("LOM") plan and process plant production schedule developed by MCB Servicos E Mineracao ("MCB").

Pit optimization has been carried out using the Lerchs-Grossman 3D ("LG-3D") algorithm in NPV Scheduler® software. This software is considered an industry standard. An economic model (with technical and economic parameters) was the main input for the pit optimization process.

The LG-3D algorithm searches for the optimal pit shell. Shell selection is a set of geologic model blocks based on the total benefit associated with the optimal pit shell. Each block in the 3D model has its economic value assessed by a benefit function that takes into account the rock type (ore or waste), production costs, price of the concentrate and mass recoveries. The pit shell benefit is calculated as the sum of the individual benefits of each block that are inside the shell. Pit slope inputs crucially affect the final shape of the shells.

The block model includes all block grades classified in the Measured, Indicated and Inferred Mineral Resource categories. In accordance with the guidelines of the National Instrument NI 43-101 on Standards of Disclosure for Mineral Projects and the Canadian Institute of Mining, Metallurgy and Petroleum Definition Standards for Mineral Resources and Mineral Reserves adopted, only those ore blocks classified in the Measured and Indicated categories are allowed to drive the pit optimizer for a Feasibility Study. Inferred Mineral Resource blocks, regardless of grade and recovery, bear no economic value and are treated as waste.

The following sections describe the economic parameters, technical parameters and other input data used in the pit optimization.

15.2.3 Grade

The gold grade estimated in the resource block model was used for pit optimization and Mineral Reserve Estimate calculation. Contaminant grades were not taken into account in the pit optimization process.

15.2.4 Dilution

15.2.4.1 Pit Optimization Analysis

It is becoming common in the industry to develop Mineral Reserve Estimate models, which essentially take into account potential dilution within the blocks, or adopt a Selective Mining Unit ("SMU") as part of the process.

For pit optimization purposes, dilution was calculated at a factor of 35%.

15.2.4.2 SMU (Selective Mining Unit)

The Lavrinha open pit Mineral Reserve Estimate model has been assessed to achieve an SMU and hence through AMS Stope Shape Optimizer v2.0 software, the mining model adopted is considered to be a 'diluted' model for SMU.

Built around the AMS Stope Shape Optimizer v2.0 software, the Stope Shape Optimizer engine ("SSO") is available in commercial products that are marketed and supported by the following mining software suppliers: Deswik, Datamine and Maptek.

The SSO is used to produce stope shapes and stope inventories from a block model that spatially represents the location of the mineralization. The SSO algorithms rely on a sub-cell block model to define the spatial location of mineralization (usually defined from a geological wireframe).

The SSO application mimics what an engineer would do, generating strings on sections, linking these to create a wireframe shape and then evaluating the wireframes against a block model.

The SSO provides a stope shape that maximizes recovered resource value above a cut-off grade while also catering for practical mining parameters such as minimum and maximum mining width, anticipated wall dilutions, minimum and maximum wall angles, minimum separation distances between parallel and sub-parallel stopes, minimum and maximum stope heights and widths, etc.

This methodology, while created originally for underground mines, can be extended to Lavrinha's Deposit to predict dilution for the LOM. During mining operations this dilution model must be updated with the actual dilution from reconciliation using a grade control model.

This feature is well established in the mining industry now and has been used worldwide.

For Lavrinha, it was assumed:

- 2 m minimum mining width;
- 10 m minimum length;
- 2.5 m bench height;
- Cut-off-Grade ("COG") varying from 0.48 g/t Au to 0.73 g/t Au.

Refer to Figure 15.2 for model sections that illustrate the SSO layout. A typical plan view is presented in Figure 15.3. Section locations are shown in the plan view presented in Figure 15.4, and six sections are presented in Figures 15.5 to 15.10.

Figure 15.2 SSO Layout

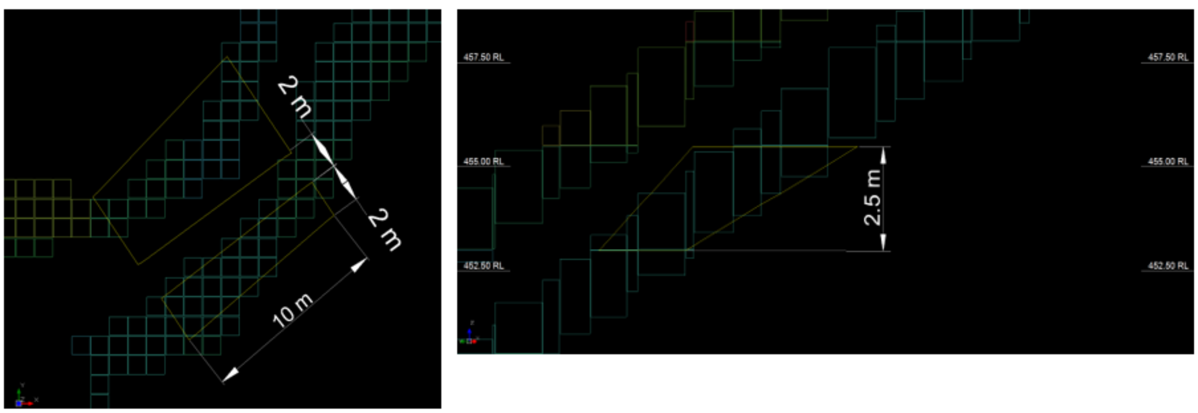


Figure 15.3 SSO Layout, Typical Plan View

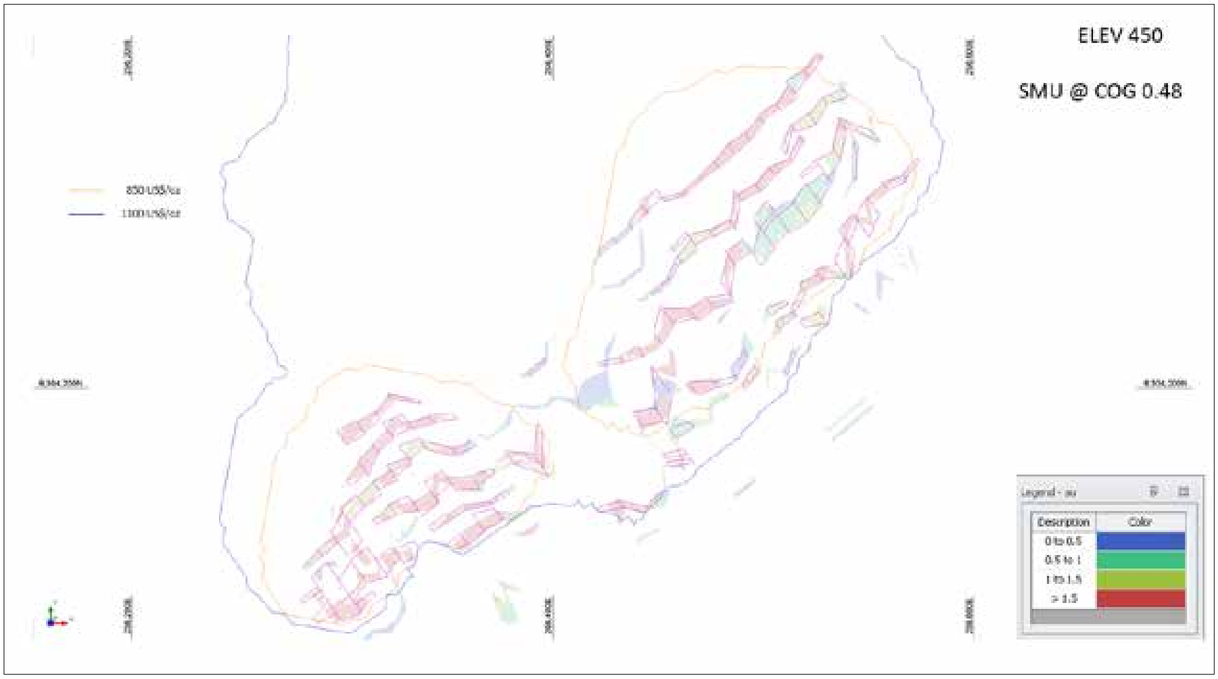


Figure 15.4 SSO Layout, Section Location

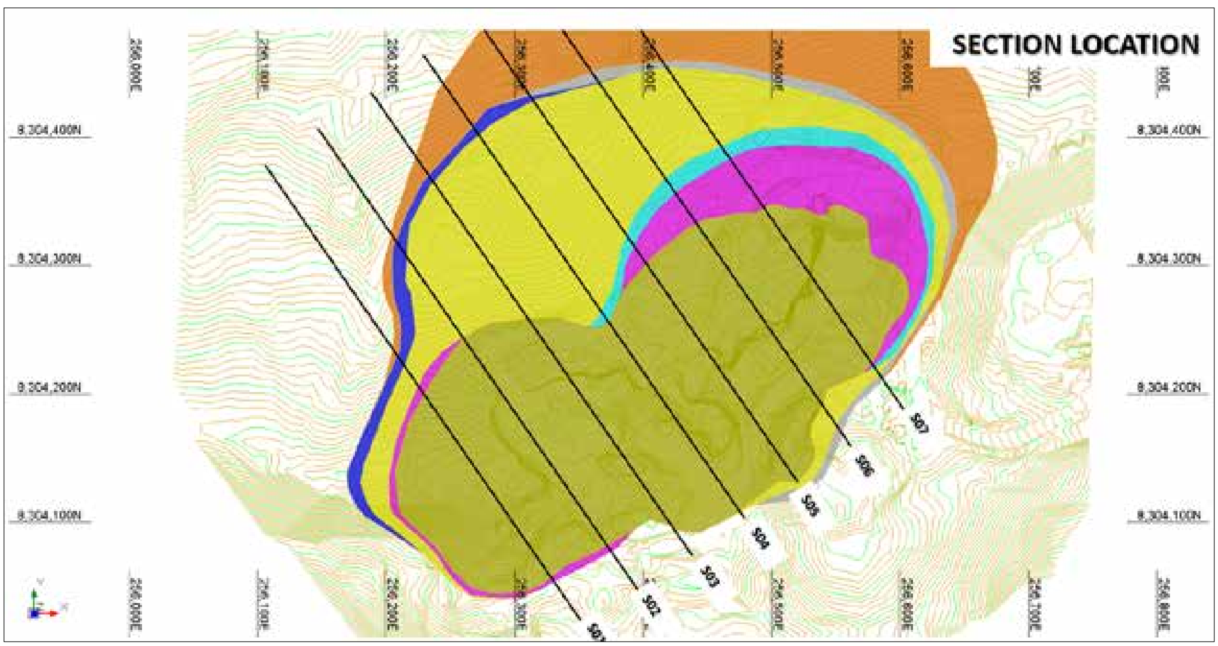


Figure 15.5 Section 01

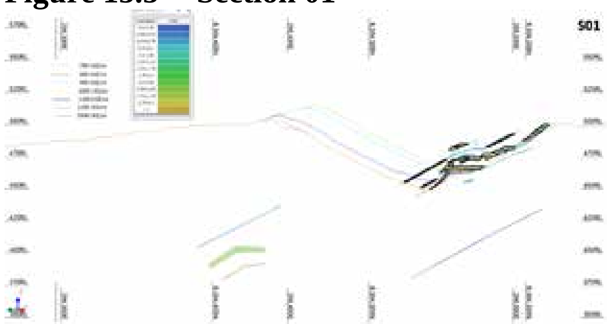


Figure 15.6 Section 02

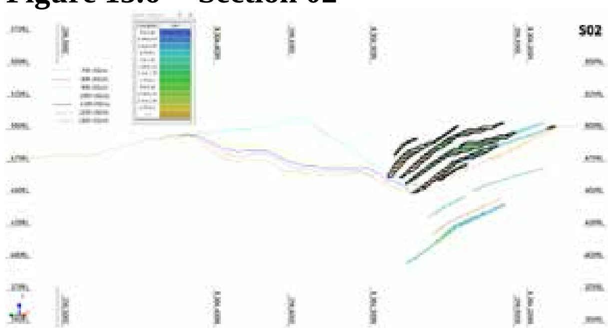


Figure 15.7 Section 03

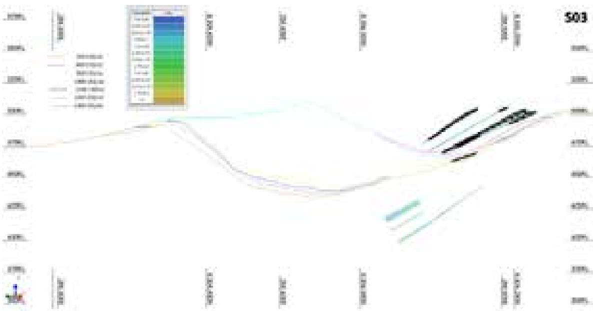


Figure 15.8 Section 04

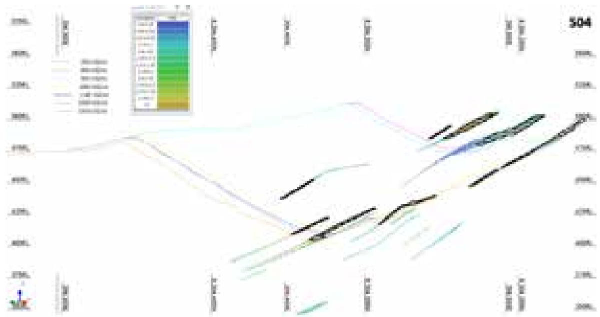


Figure 15.9 Section 05

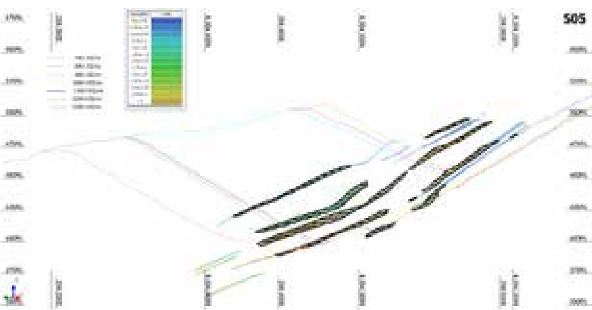
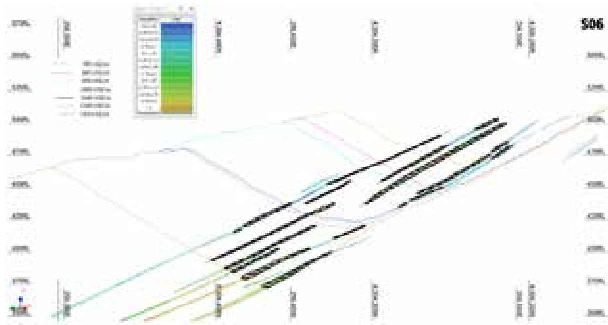


Figure 15.10 Section 06



15.2.5 Mining Recovery

Mining recovery, based on the applied SMU, has been set to 95%. This is to allow for ore losses as a result of blasting, edge effects of cutbacks and wedges of potential process plant feed left as a function of bench face advancement with pit depth.

15.2.6 Economic Criteria

A classification of the materials was prepared based on the gold grade for each block. Pit optimization was carried out using Measured and Indicated Mineral Resources above the break-even cut-off grade. This material will be sent directly for processing and the waste rock will be sent directly to the waste dump.

Note: Since the General and Administration ("G&A") costs will be shared with the EPP underground Projects, the material above 0.48 g/t Au (Low Grade Material) and below 0.73 g/t Au inside the pit shell was considered as ore.

The economic criteria for pit optimization are listed in Table 15.5.

TABLE 15.5
ECONOMIC PARAMETERS FOR PIT OPTIMIZATION

	Value	Unit
Gold Price	1,100	US$/Oz
Mining Cost	Value	Unit
Ore	2.44	US$/t
Waste	1.89	US$/t

TABLE 15.5 ECONOMIC PARAMETERS FOR PIT OPTIMIZATION		
	Value	**Unit**
Plant Costs	Value	Unit
ROM	10.24	US$/t
Recovery	Value	Unit
Ore	93.0	%
Other	Value	Unit
Royalty	38.5	US$/oz
G&A	3.8	US$/t (Mill)
Dilution	35.0	%
Factor	0.0322	(g/Troy oz)
Break-Even Cut-off Grade	Value	Unit
Au	0.73	g/t
Marginal Cut-off Grade	Value	Unit
Au	0.60	g/t
Low Grade Cut-off Grade	Value	Unit
Au	0.48	g/t

15.2.7 Sensitivity Pit Analysis: Revenue Factor

A revenue factor was applied to generate several optimized pits by varying the selling price between US$700/oz and US$1,300/oz to produce various COG's as presented in Table 15.6.

TABLE 15.6 PIT SHELL SENSITIVITY ANALYSIS, PRICE AND COG			
Sensitivity Price		**Cut-off Grade g/t Au**	
US$/Oz	**US$/g**	**Break-Even**	**Marginal**
700	22.51	1.04	0.92
750	24.11	0.97	0.86
800	25.72	0.91	0.80
850	27.33	0.85	0.76
900	28.94	0.81	0.71
950	30.54	0.76	0.68
1,000	32.15	0.73	0.64
1,050	33.76	0.69	0.61
1,100	35.37	0.66	0.58
1,150	36.97	0.63	0.56
1,200	38.58	0.60	0.54

1,250	40.19	0.58	0.51
1,300	41.80	0.56	0.49

The application of a lower revenue factor leads to ore becoming less valuable or being reclassified as waste, and this in turn leads to a smaller pit. By running pit optimizations repeatedly with incrementally different revenue factors, a range of pits from large to small is obtained. The pits so produced are often referred to as shells and are numbered from smallest to largest as presented in Table 15.7.

TABLE 15.7
PIT SHELL SENSITIVITY ANALYSIS, SUMMARY

PIT US$/oz	Ore Tonnes (t)	Au (g/t)	Au Ounces (oz)	Waste (t)	W/O	Total Material
700	571,453	2.09	38,332	4,850,272	8.49	5,421,725
750	610,354	2.06	40,498	5,223,132	8.56	5,833,486
800	642,527	2.07	42,685	5,716,018	8.90	6,358,545
850	648,850	2.06	43,053	5,796,087	8.93	6,444,936
900	677,012	2.06	44,745	6,220,372	9.19	6,897,384
950	990,044	2.12	67,520	12,803,678	12.93	13,793,722
1,000	1,002,396	2.14	68,842	13,263,991	13.23	14,266,387
1,050	1,017,322	2.13	69,644	13,503,773	13.27	14,521,094
1,100	1,032,159	2.14	70,918	13,926,933	13.49	14,959,092
1,150	1,041,186	2.14	71,783	14,284,529	13.72	15,325,715
1,200	1,055,418	2.15	72,960	14,608,586	13.84	15,664,004
1,250	1,056,736	2.15	73,008	14,621,879	13.84	15,678,615
1,300	1,342,749	2.19	94,409	22,623,078	16.85	23,965,827

An SMU model was created after the sensitivity analysis for pit gold prices from US$800/oz to US$1,150/oz varying the COG from 0.48 g/t Au to 0.73 g/t Au. The results are presented in Table 15.8.

TABLE 15.8
GOLD PRICE SENSITIVITY PIT ANALYSIS ON SMU MODEL

PIT US$/oz	SMU COG	Tonnes HG (t)	Tonnes LG (t)	Tonnes Waste (t)	SMU Tonnes SMU (t)	Ounces (oz)	Au (g/t)	Dilution (%)	Waste (t)	S.R (t/t)	Total Mined (t)
800	*0.48*	*470,684*	*35,021*	*185,211*	*690,916*	*36,354*	*1.64*	*36.62*	*5,667,629*	*8.20*	*6,358,545*
800	*0.60*	*435,943*	*20,024*	*153,113*	*609,081*	*34,513*	*1.76*	*33.58*	*5,749,464*	*9.44*	*6,358,545*
800	*0.66*	*441,112*	*14,405*	*132,026*	*587,544*	*34,475*	*1.83*	*28.98*	*5,771,001*	*9.82*	*6,358,545*
800	*0.73*	*418,263*	*5,819*	*118,805*	*542,887*	*33,974*	*1.95*	*28.01*	*5,815,658*	*10.71*	*6,358,545*
850	*0.48*	*475,324*	*35,241*	*187,460*	*698,025*	*36,659*	*1.63*	*36.72*	*5,746,911*	*8.23*	*6,444,936*
850	*0.60*	*440,133*	*20,193*	*154,755*	*615,082*	*34,780*	*1.76*	*33.62*	*5,829,855*	*9.48*	*6,444,936*
850	*0.66*	*445,238*	*14,565*	*133,692*	*593,495*	*34,750*	*1.82*	*29.08*	*5,851,441*	*9.86*	*6,444,936*
850	*0.73*	*421,505*	*5,822*	*120,311*	*547,972*	*34,191*	*1.94*	*28.15*	*5,896,964*	*10.76*	*6,444,936*
900	*0.48*	*496,166*	*36,571*	*194,659*	*727,396*	*38,123*	*1.63*	*36.54*	*6,169,989*	*8.48*	*6,897,384*
900	*0.60*	*459,412*	*21,035*	*161,142*	*641,590*	*36,124*	*1.75*	*33.54*	*6,255,795*	*9.75*	*6,897,384*
900	*0.66*	*463,669*	*15,096*	*138,704*	*617,469*	*36,097*	*1.82*	*28.97*	*6,279,915*	*10.17*	*6,897,384*
900	*0.73*	*438,876*	*5,867*	*123,543*	*568,286*	*35,511*	*1.94*	*27.78*	*6,329,098*	*11.14*	*6,897,384*
950	*0.48*	*726,681*	*61,039*	*267,162*	*1,054,882*	*58,485*	*1.72*	*33.92*	*12,738,841*	*12.08*	*13,793,722*
950	*0.60*	*676,599*	*35,536*	*223,479*	*935,614*	*55,488*	*1.84*	*31.38*	*12,858,108*	*13.74*	*13,793,722*
950	*0.66*	*675,768*	*26,080*	*189,362*	*891,210*	*55,228*	*1.93*	*26.98*	*12,902,513*	*14.48*	*13,793,722*
950	*0.73*	*648,582*	*10,052*	*177,011*	*835,645*	*54,730*	*2.04*	*26.88*	*12,958,077*	*15.51*	*13,793,722*
1000	*0.48*	*736,047*	*61,440*	*271,702*	*1,069,189*	*59,687*	*1.74*	*34.07*	*13,197,198*	*12.34*	*14,266,387*
1000	*0.60*	*685,283*	*35,816*	*227,679*	*948,778*	*56,666*	*1.86*	*31.57*	*13,317,610*	*14.04*	*14,266,387*
1000	*0.66*	*684,181*	*26,321*	*192,777*	*903,279*	*56,382*	*1.94*	*27.13*	*13,363,108*	*14.79*	*14,266,387*
1000	*0.73*	*655,940*	*10,107*	*180,494*	*846,541*	*55,835*	*2.05*	*27.10*	*13,419,846*	*15.85*	*14,266,387*
1050	*0.48*	*746,613*	*62,546*	*275,821*	*1,084,980*	*60,351*	*1.73*	*34.09*	*13,436,115*	*12.38*	*14,521,094*
1050	*0.60*	*695,511*	*36,208*	*231,162*	*962,882*	*57,306*	*1.85*	*31.59*	*13,558,213*	*14.08*	*14,521,094*
1050	*0.66*	*694,663*	*26,676*	*195,638*	*916,978*	*57,033*	*1.93*	*27.12*	*13,604,117*	*14.84*	*14,521,094*
1050	*0.73*	*665,137*	*10,131*	*183,028*	*858,296*	*56,431*	*2.04*	*27.10*	*13,662,798*	*15.92*	*14,521,094*
1100	*0.48*	*757,839*	*63,495*	*279,734*	*1,101,068*	*61,493*	*1.74*	*34.06*	*13,858,024*	*12.59*	*14,959,092*
1100	*0.60*	*706,597*	*36,916*	*234,899*	*978,413*	*58,465*	*1.86*	*31.59*	*13,980,679*	*14.29*	*14,959,092*
1100	*0.66*	*705,570*	*27,350*	*198,278*	*931,197*	*58,178*	*1.94*	*27.05*	*14,027,895*	*15.06*	*14,959,092*
1100	*0.73*	*674,778*	*10,503*	*186,107*	*871,388*	*57,515*	*2.05*	*27.16*	*14,087,704*	*16.17*	*14,959,092*
1150	*0.48*	*764,029*	*64,290*	*283,113*	*1,111,433*	*62,261*	*1.74*	*34.18*	*14,214,282*	*12.79*	*15,325,715*
1150	*0.60*	*712,642*	*37,541*	*237,866*	*988,049*	*59,221*	*1.86*	*31.71*	*14,337,666*	*14.51*	*15,325,715*
1150	*0.66*	*711,747*	*27,988*	*200,736*	*940,470*	*58,941*	*1.95*	*27.14*	*14,385,244*	*15.30*	*15,325,715*
1150	*0.73*	*679,876*	*10,862*	*188,305*	*879,043*	*58,210*	*2.06*	*27.26*	*14,446,672*	*16.43*	*15,325,715*

There is not an established rule to choose a final pit shell. It was decided to use the pit at US$950/oz for the Mineral Reserve Estimate. The selected pit generates the ROM to support Project development and any pit beyond the pit US$950/oz produces undesirable waste and stripping ratio incremental figures. Indeed, any pit choice beyond the chosen pit generates insignificant ore tonnage, creating no economic value in terms of Net Present Value ("NPV"). The selected pit supports 2.5 production years.

Mining dilution within the pit resulted in a factor of 33.92% at zero grade.

15.2.8 Pit and Mining Phase Designs

Final Pit Design

The final pit design was based on the economic shell obtained with US$950/oz and with variable pit slope angles according to geotechnical domains, ranging from inter-ramp angles of 37.5º to 42º. Table 15.9 presents the key open pit design parameters.

TABLE 15.9 PIT DESIGN PARAMETERS					
Sector	**Split**	**Berm Width (m)**	**Bench Height (m)**	**Face Slope (degrees)**	**Inter-Ramp Angle (degrees)**
11	1	9.5	10	70	37.5
12	1	1.5	10	70	42
	2	13.5	10	70	
20	1	3.25	5	45	31

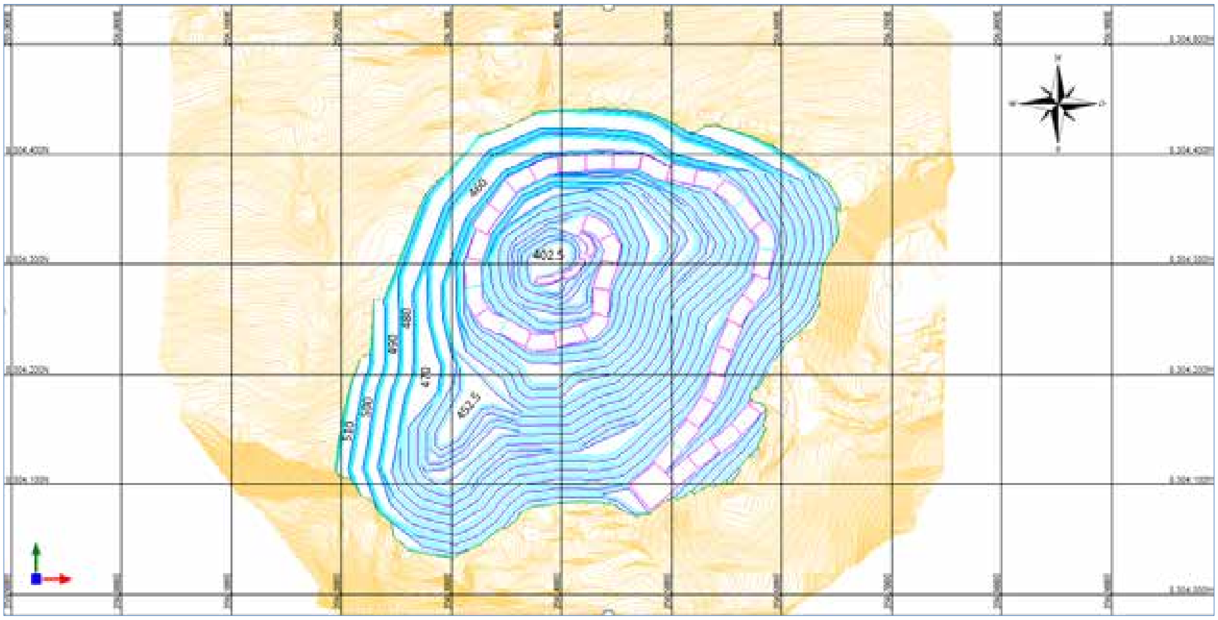
A pit haul road width of 15 m was selected to accommodate 25 and 38 tonne trucks. MCB used a 10% ramp gradient, which is common in the industry for these types of trucks. In general, the last three benches of every mining phase were designed with a single lane ramp width of 6 m.

The current mine plan is designed with 10 m high benches stacked to 20 m (i.e. double benching) for sector 12. Mining costs are based on blasting 10 m benches for the waste zones and 2.5 m slices for the ore to assure good selectivity. Since a high strip ratio is expected, this differential in drilling and blasting operations for ore and waste seems practical, as large areas will be 100% waste.

Table 15.10 and Figure 15.11 show the final pit design results. There is a single exit on the east side of the pit to access the primary crusher and the waste storage areas. The final pit is 520 m long in the SW-NE direction and 400 m wide in the NW-SE direction. The pit bottom is at 402.5 m above sea level ("ASL"). The highest wall is about 120 m on the NW side. The total footprint area of the pit is approximately 13 hectares.

TABLE 15.10 PIT OPTIMIZATION VS DESIGN RESULTS							
PIT US$950/oz	**Tonnes SMU (t)**	**Ounces (oz)**	**Au (g/t)**	**Dilution (%)**	**Waste (t)**	**S.R (t/t)**	**Total Mined (t)**
Optimization	1,054,882	58,485	1.72	33.92	12,738,841	12.08	13,793,722
Pit Design	1,110,200	60,297	1.69	33.92	14,005,170	12.61	15,115,369

Figure 15.11 Pit Design Layout



15.2.9 Tabulation of Open Pit Reserves

Table 15.11 presents the Lavrinha Mineral Reserve Estimate contained within the open pit design, and Table 15.12 presents the Mineral Reserve Estimate by bench.

TABLE 15.11
LAVRINHA MINERAL RESERVE ESTIMATE(1-7)

Reserves	Tonnes (t)	Au (g/t)	Contained Au oz
Proven	67,000	1.85	4,000
Probable	1,043,000	1.68	56,300
Total	**1,110,000**	**1.69**	**60,300**

1) *CIM definitions were followed for Mineral Reserves.*
2) *The Mineral Reserve Estimate isas of July 31, 2016.*
3) *Mineral Reserve Estimate for the Lavrinha Deposit was prepared under the supervision of Marcelo Batelochi, Ausimm (CP 205477).*
4) *Mineral Reserve Estimate derived at a cut-off grade of 0.48 g/t Au.*
5) *Lavrinha Mineral Reserve Estimate from an average short-term gold price of US$1,100 per ounce.*
6) *Bulk density average was 2.78 t/m^3.*
7) *Numbers may not add due to rounding.*

TABLE 15.12
TABULATION OF PIT CONSTRAINED MINERAL RESERVE ESTIMATE BY BENCH

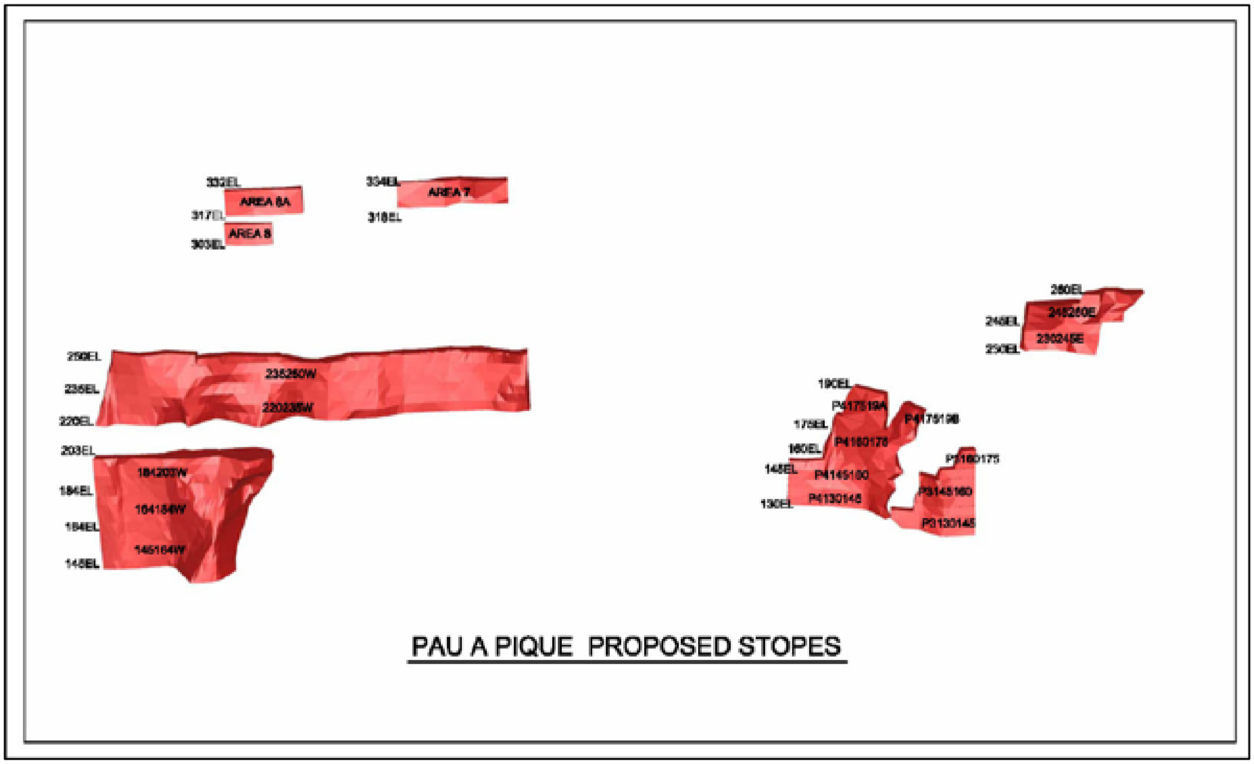
BENCH	PROVEN			PROBABLE			Grand Total		
	Tonnes	Au (g/t)	Au oz	Tonnes	Au (g/t)	Au oz	Tonnes	Au (g/t)	Au oz
512.5	-	-	-	545	0.85	15	545	0.85	15
510.0	-	-	-	1,190	0.94	36	1,190	0.94	36
507.5	-	-	-	2,310	0.82	61	2,310	0.82	61
505.0	-	-	-	7,203	1.06	244	7,203	1.06	244
502.5	2,012	1.38	90	10,707	1.24	428	12,719	1.27	518
500.0	2,760	1.67	148	23,067	1.30	962	25,827	1.34	1,110
497.5	2,631	2.09	177	31,514	1.20	1,212	34,145	1.27	1,389

TABLE 15.12 TABULATION OF PIT CONSTRAINED MINERAL RESERVE ESTIMATE BY BENCH									
BENCH	**PROVEN**			**PROBABLE**			**Grand Total**		
	Tonnes	**Au (g/t)**	**Au oz**	**Tonnes**	**Au (g/t)**	**Au oz**	**Tonnes**	**Au (g/t)**	**Au oz**
495.0	2,624	2.23	188	40,377	1.34	1,738	43,000	1.39	1,926
492.5	2,511	2.54	205	40,330	1.30	1,688	42,841	1.37	1,893
490.0	2,677	1.92	165	42,333	1.24	1,681	45,010	1.28	1,847
487.5	2,789	1.20	108	46,323	1.32	1,971	49,112	1.32	2,078
485.0	2,710	1.38	120	44,463	1.34	1,912	47,173	1.34	2,032
482.5	3,206	1.38	142	47,456	1.28	1,956	50,662	1.29	2,098
480.0	5,065	1.48	240	40,716	1.24	1,629	45,781	1.27	1,870
477.5	3,871	1.45	180	37,277	1.32	1,583	41,148	1.33	1,763
475.0	3,657	2.43	286	37,836	1.46	1,773	41,493	1.54	2,059
472.5	6,065	1.83	356	36,967	1.69	2,005	43,032	1.71	2,361
470.0	4,160	1.61	215	34,848	1.91	2,136	39,008	1.87	2,351
467.5	2,267	1.77	129	34,066	2.11	2,307	36,333	2.09	2,436
465.0	2,382	2.41	185	39,957	1.92	2,469	42,339	1.95	2,654
462.5	3,053	2.27	222	31,969	1.88	1,936	35,021	1.92	2,158
460.0	770	5.03	124	28,457	2.14	1,959	29,228	2.22	2,083
457.5	1,966	3.59	227	24,316	1.79	1,396	26,282	1.92	1,623
455.0	1,646	1.71	90	22,201	1.96	1,397	23,847	1.94	1,487
452.5	1,910	1.80	110	21,758	1.95	1,366	23,668	1.94	1,476
450.0	459	1.89	28	21,995	1.52	1,071	22,454	1.52	1,099
447.5	-	-	-	19,730	1.56	992	19,730	1.56	992
445.0	-	-	-	19,973	1.41	907	19,973	1.41	907
442.5	-	-	-	20,672	1.38	916	20,672	1.38	916
440.0	274	1.97	17	21,044	1.34	908	21,318	1.35	925
437.5	-	-	-	19,841	1.55	989	19,841	1.55	989
435.0	1,988	1.41	90	17,802	1.44	822	19,791	1.43	912
432.5	2,639	1.50	127	18,655	1.54	925	21,294	1.54	1,053
430.0	1,158	1.19	44	17,989	1.69	976	19,147	1.66	1,020
427.5	61	0.00	0	18,277	1.88	1,102	18,338	1.87	1,102
425.0	-	-	-	16,925	1.84	1,001	16,925	1.84	1,001
422.5	18	0.00	0	16,083	2.00	1,034	16,101	2.00	1,034
420.0	131	0.17	1	16,850	2.21	1,199	16,981	2.20	1,200
417.5	-	-	-	16,953	2.56	1,397	16,953	2.56	1,397
415.0	-	-	-	15,902	3.02	1,544	15,902	3.02	1,544
412.5	-	-	-	13,748	3.52	1,556	13,748	3.52	1,556
410.0	-	-	-	8,810	4.24	1,200	8,810	4.24	1,200
407.5	-	-	-	6,155	4.31	852	6,155	4.31	852
405.0	-	-	-	4,211	4.62	626	4,211	4.62	626
402.5	-	-	-	2,939	4.26	402	2,939	4.26	402
Total	**67,500**	**1.85**	**4,016**	**1,042,700**	**1.68**	**56,281**	**1,110,200**	**1.69**	**60,297**

15.3 PAU-A-PIQUE MINERAL RESERVE ESTIMATE

Mining stope mineralization wireframes were generated along level boundaries for the width of the orebody, and subsequently projected 15 m high, at various lengths along strike. The stope lengths varied from 40 m to 216 m, with the longest stopes being the two NW stopes above the sill pillar at 208 m Elev to 220 m Elev. The three stopes below the sill pillar were designed at approximate 20 m heights. This is illustrated in Figure 15.12.

Figure 15.12 Creation of Mineral Reserve Estimate Blocks for Economic Analysis



These Mineral Reserve Estimate blocks were reported from the grade model with grade and Mineral Resource Estimate category as outputs. Blocks where the ore was classified as Indicated were moved to the next phase of assessment. Blocks with stope dilution and mining recoveries were then applied to each of these shapes. Blocks that were above the break-even cut-off grade ("BCOG") were considered potential Mineral Reserves. To estimate stope dilution P&E used the geotechnical model developed by an Aura geotechnical engineer and reviewed by KP for this study with different dilution considered for different rock types located in the stope HW or FW from 1.5 m to 0.6 m. The isolated stopes where initial capital development has to be put in place for access were evaluated separately and included in Mineral Reserve Estimate figures only if deemed economic. The parameters used to assess the economic viability of the Mineral Reserve Estimate shapes are listed in Table 15.13.

The Mineral Reserve Estimate at the Pau-a-Pique Deposit is presented in Table 15.14.

TABLE 15.13
PARAMETERS USED TO ASSESS ECONOMIC VIABILITY OF THE PAU-A-PIQUE MINERAL RESERVE ESTIMATE

Reserve Parameters	Unit	Value
BCOG	Au g/t	2.40
Au Price	US$/oz	1,165
Stope Dilution	%	31
Stope Recovery	%	92
Mining Dilution (incl. ore devel)	%	23
Dilution from Backfill	%	10
Mining Recovery (incl. ore devel)	%	94

TABLE 15.14 **PAU-A-PIQUE MINERAL RESERVE ESTIMATE(1-5)**			
Reserve Category	**Tonnes (t)**	**Au (g/t)**	**Contained Au oz**
Probable	320,000	3.24	33,300

1) *The Mineral Reserve Estimate is as of July 31, 2016.*
2) *The Mineral Reserve Estimate was developed from the resource model prepared by P&E. The Probable Mineral Reserves were derived from the Indicated Mineral Resource.*
3) *The cut-off grade (2.40 g/t Au) was based on a US$1,165/oz gold price, 93% metallurgical Au recovery, 99.99% payable, royalties and CEFEM tax totalling 3.5%, gold doré bar transport and refining costs totalling US$1.56/t, mine direct and mine indirect costs totalling US$58.08/t, US$12.50/t processing cost, and US$6.44/t processed for the projected share of the overall multi-mine project G&A cost that would be incurred by the proposed Pau-a-Pique underground mine project.*
4) *The Mineral Reserve Estimate tonnage and mined metal have been rounded to reflect the accuracy of the estimate.*
5) *The NI 43-101 Mineral Reserve Estimate for the Pau-a-Pique Deposit set out in Table 15.14 has been reviewed and approved by Alexandru Veresezan, P. Eng., of P&E Mining Consultants Inc., who is a Qualified Person ("QP") for the purposes of this Report, and who is independent of the Company.*

Details on the BCOG are presented in Table 15.15.

TABLE 15.15 **BCOG DETERMINATION**		
Unit Cost	**Units**	**BCOG**
Metal Price	US$/oz	1,165.00
Metallurgical Recovery	%	93.00%
Payable Metal	%	99.99%
Conversion Factor	grams to Troy oz	31.104
Mining Operating Cost	US$/t (mined)	58.08
Process Operating Cost	US$/t (processed)	12.50
G&A Operating Cost	US$/t (processed)	6.44
Refining Operating Cost	US$/t (processed)	1.56
Total BCOG Operating Cost	US$/t	78.58
Royalty	%	2.50%
CEFEM Tax	%	1.00%
PPQ LOM BCOG	Au g/t	2.40

Details on stope dilution and mining recoveries are presented in Table 15.16. Each stope dilution was evaluated based on the stope shape proposed for extraction and geotechnical classification (Type III good ground, Type IV fair ground or Type V poor ground).

After ramp and level development was designed, each stope was assessed to check that it would be economic to mine after accounting for all operating and capital costs relevant to each individual stope. An isometric view of the ramp and level development is presented in Figure 15.13.

Figure 15.13 Isometric View of Pau-a-Pique Mine Design

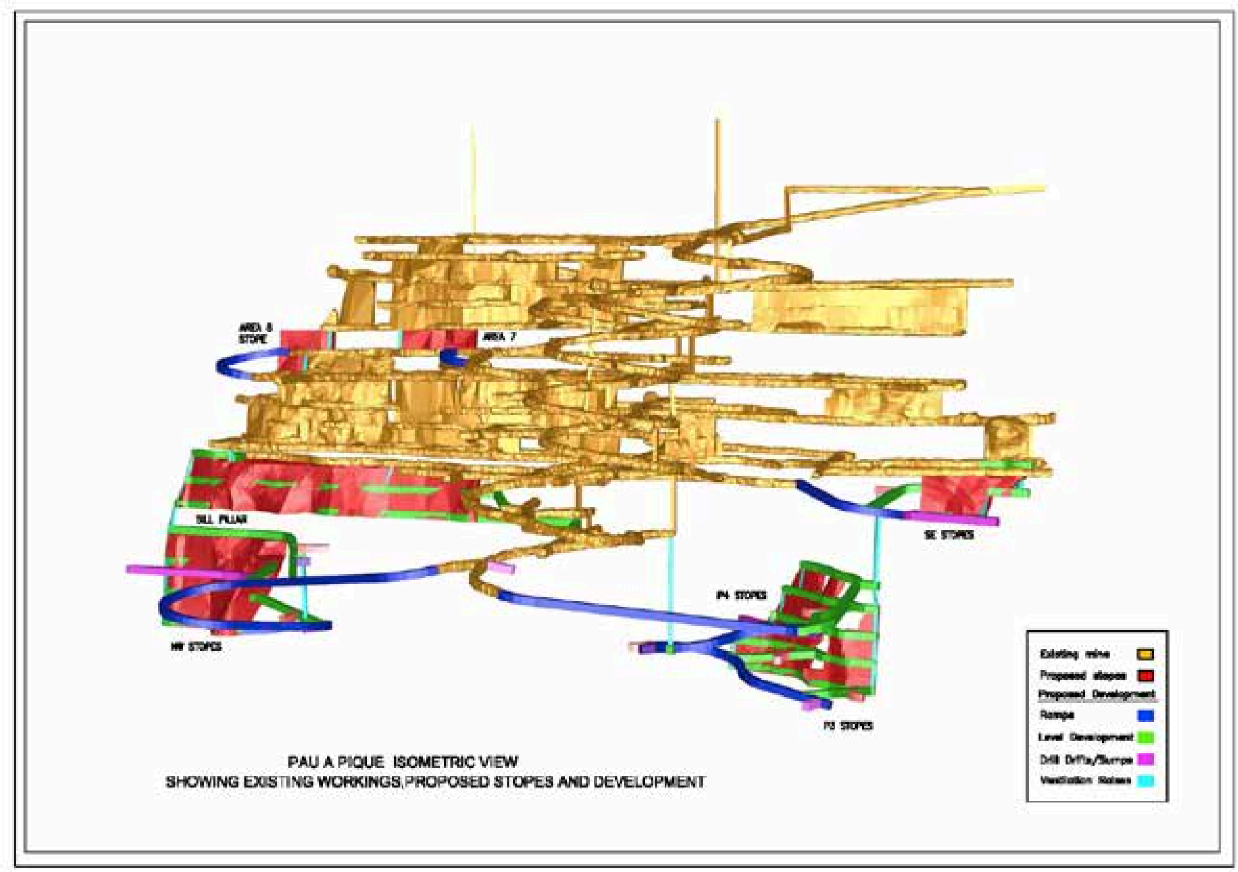


TABLE 15.16
STOPE DILUTION CALCULATIONS

Stope Annotation	Ore Insitu Tonnes	Au g/t	Gold Insitu Oz	Stope W m	Hangingwall Rock Type III	IV	V	Footwall Rock Type III	IV	V	Stope Dil %	Backfill Dil %	Stope Dil tonnes	Stope Dil Grade g/t	Mining Rec %	Rec Tonnes tonnes	Mining Rec Oz
Area7 (318-334EL)	6,392	6.04	1,242	3.80	80%	0%	20%	50%	40%	10%	30%	10%	8,951	4.41	70%	6,266	889
Area8 (317-332EL)	4,263	3.62	497	3.52	40%	60%	0%	100%	0%	0%	25%	10%	5,744	2.81	70%	4,021	363
Area8 (303-317EL)	2,218	4.30	306	3.29	0%	100%	0%	20%	80%	0%	43%	10%	3,391	2.95	90%	3,052	290
235250WC	38,290	3.65	4,499	5.51	100%	0%	0%	100%	0%	0%	11%	10%	46,288	3.05	90%	41,660	4,089
220235WC	32,462	3.70	3,858	5.37	70%	20%	10%	70%	20%	10%	19%	10%	41,880	2.91	90%	37,692	3,531
184203W	32,574	3.79	3,967	8.36	10%	70%	20%	0%	80%	20%	21%	10%	42,674	2.94	95%	40,540	3,838
164184W	26,272	4.29	3,623	7.15	30%	50%	20%	20%	50%	30%	23%	10%	34,961	3.28	95%	33,212	3,503
145164W	20,388	5.67	3,715	6.24	5%	75%	20%	0%	80%	20%	29%	10%	28,238	4.16	95%	26,826	3,587
245260SE	8,109	3.52	917	5.59	70%	30%	0%	70%	30%	0%	16%	10%	10,182	2.88	95%	9,673	896
230245SE	6,666	4.71	1,009	5.24	40%	60%	0%	40%	60%	0%	22%	10%	8,783	3.68	95%	8,344	987
P3160175	288	6.14	57	4.12	100%	0%	0%	20%	80%	0%	23%	10%	383	4.66	95%	364	55
P3145160	2,981	3.61	346	4.01	100%	0%	0%	20%	80%	0%	24%	10%	3,992	2.76	95%	3,793	336
P3130145	4,080	4.49	589	4.07	100%	0%	0%	0%	100%	0%	26%	10%	5,541	3.37	95%	5,264	571
P417519A	2,793	4.07	366	4.96	100%	0%	0%	0%	100%	0%	21%	10%	3,664	3.14	95%	3,481	351
P417519B	1,449	3.63	169	4.14	100%	0%	0%	0%	100%	0%	25%	10%	1,961	2.72	95%	1,863	163
P4160175	5,909	3.80	723	4.45	100%	0%	0%	0%	100%	0%	24%	10%	7,895	2.88	95%	7,500	695
P4145160	5,967	4.21	808	4.58	100%	0%	0%	0%	100%	0%	23%	10%	7,932	3.20	95%	7,536	776
P4130145	5,967	4.12	791	4.14	100%	0%	0%	0%	100%	0%	25%	10%	8,076	3.09	95%	7,672	761
Development Ore	71,297	3.33	7,642	3.33							0%	0%	71,297	3.33	100%	71,297	7,642
TOTAL	**278,365**	**3.92**	**35,122**	**3.99**							**23%**		**341,834**	**3.24**	**94%**	**320,055**	**33,323**

15.4 MINERAL RESERVE ESTIMATE FOR THE PROJECT

The total Mineral Reserve Estimate for the Project is presented in Table 15.17.

TABLE 15.17 TOTAL MINERAL RESERVE ESTIMATE FOR THE PROJECT			
Proven	**Tonnes (t)**	**Au (g/t)**	**Contained Au oz**
Lavrinha	67,000	1.85	4,000
Total Proven	**67,000**	**1.85**	**4,000**
Probable	**Tonnes (t)**	**Au (g/t)**	**Contained Au oz**
Lavrinha	1,043,000	1.68	56,300
Ernesto	868,000	5.03	140,000
Pau-a-Pique	320,000	3.24	33,300
Total Probable	**2,231,000**	**3.20**	**229,600**
Total Proven + Probable	**2,298,000**	**3.17**	**233,600**

Contained metal may not sum in the above table due to rounding

16.0 MINING METHODS

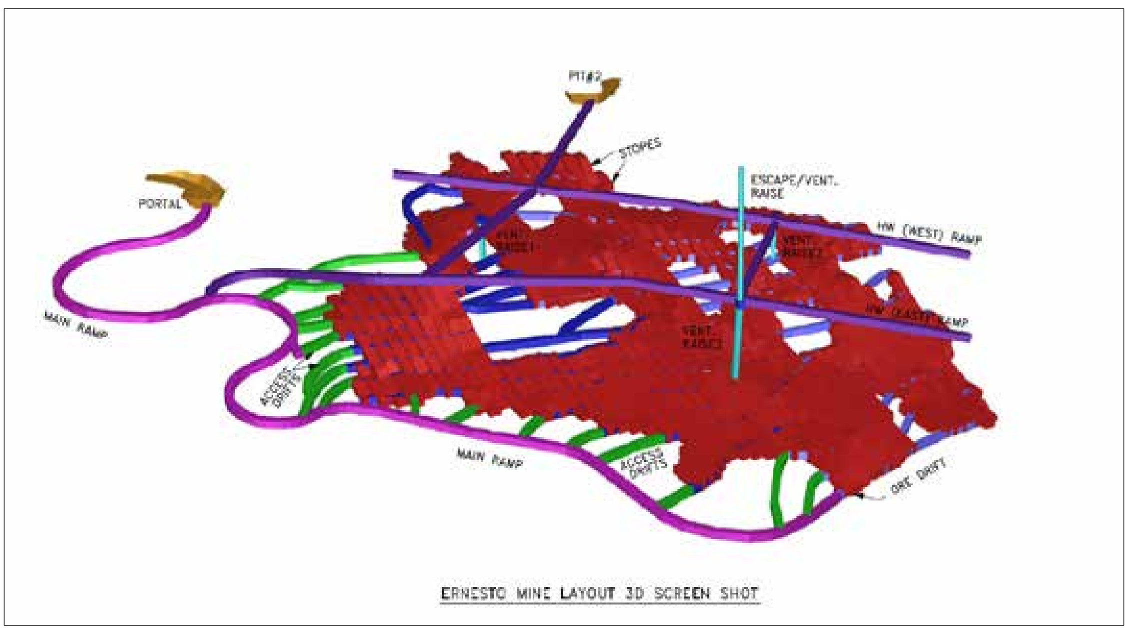
16.1 ERNESTO

The Ernesto underground mine would be mined using a drift and fill ("D&F") mining method that was selected taking into consideration:

- The variable strike, thickness and low angle dip of the mineralized zone;
- Geology; topography; geomechanical and hydrogeologic analyses and recommendations;
- Phased mine development and production, stoping and backfilling operations;
- Mine services;
- Worker health and safety;
- Regulatory requirements; and
- Current information and other aspects.

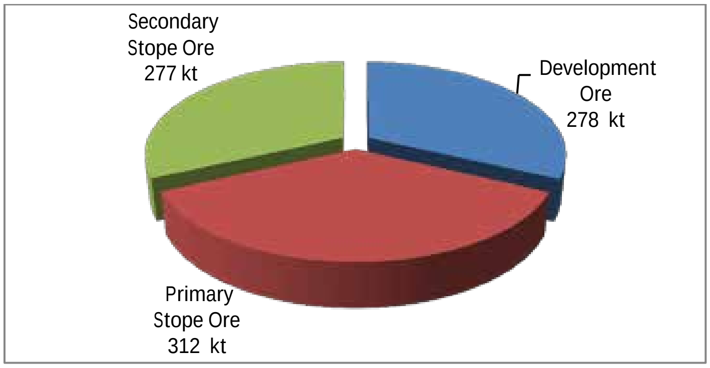
An isometric view screenshot of the Ernesto mine layout is shown in Figure 16.1. The mine layout includes a main access ramp, internal access ramps, levels, ventilation raises, and ancillary openings such as sumps and muckbays. The Pit#2 portal would be used to develop and access the definition drilling headings in the metasediment hangingwall formation.

Figure 16.1 Ernesto Mine Isometric View



The distribution of ore tonnages produced through development in ore, primary D&F stopes, and secondary D&F stopes is shown in Figure 16.2.

Figure 16.2 Ore Tonnages Produced in Development and Primary and Secondary Stopes



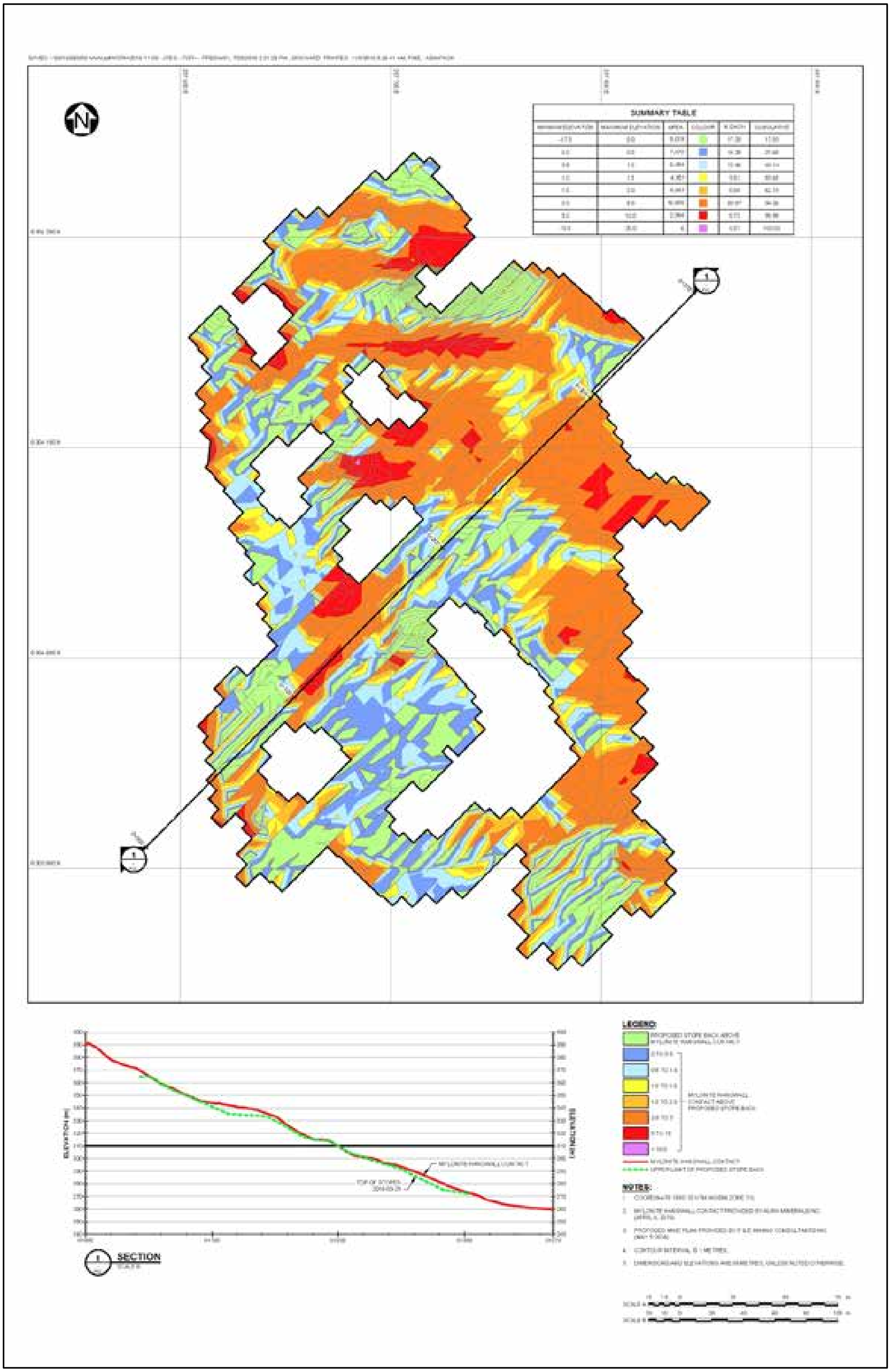
16.1.1 Ernesto Mineral Deposit Characteristics

The Ernesto mine would be developed to exploit the mineralized Lower Trap zone generally bounded by mine sections 8,303,792 NW and 8,304,172 NW and the 255 m and 365 m mine elevations.

The Ernesto Lower Trap zone is a gentle but variable dip and variable thickness deposit that strikes northwest and typically dips 15° to approaching 50° to the northeast.

The Lower Trap mineralization occurs within a mylonite-magnetite-sericite schist zone which is located between hanging wall metasediments and the footwall tonalite. The mylonite rock zone is 6 to 28 m thick and averages about 6 m thick. Saprolite (altered tonalite) is often present between the footwall tonalite and the lower contact of the Lower Trap zone. The saprolite is typically 10 m to 15 m thick in the centre of the proposed mining area and is thinner along the northwest, southwest and southeast margins of the proposed mining area. In addition, rafts of unaltered tonalite are at times present between the saprolite and Lower Trap zone. Figure 16.3 presents a plan diagram of the mylonite thickness contours above the stope backs.

Figure 16.3 Plan Diagram of the Mylonite Thickness Contours Above the Stope Backs



The gold mineralization within the mylonite-sericite schists occurs in quartz lenses, veins and veinlets that are situated within the Lower Trap structure, and occasionally in altered sulphidic mineralization occurring locally in the hanging wall meta-arenite unit. The Mineral Resource Estimate model for the Ernesto Lower Trap Deposit is comprised of gold mineralized mylonite-sericite schist blocks but does not include gold mineralization that occasionally occurs in the local hanging wall meta-arenite unit.

The proposed mine production plan and the Mineral Reserve Estimate were developed based on the Mineral Resource Estimate model and mine planning aspects including “shanty back” type drifts and stope designs where poor rock quality mylonite on the back of stopes would be mined back to a projected competent mylonite:metasediment hangingwall contact. In areas where the rock is more competent, rock type I to III (RMR 40 to 100), the drift and stope backs have been designed in mylonite. Overall, 25% of the stopes were predicted to have metasediments in the back. 30% would have mylonite in the back but would be proactively mined back to the metasediments. 45% would have mylonite in the back, of which 22% was expected to be of rock type IV or V (RMR of 0 to 40).

The geomechanical rock types (i.e. I, II, III, IV and V) used to analyze the EPP deposits are standard classes associated with different ranges of rock mass quality:

- Type I is RMR 80 to 100
- Type II is RMR 60 to 80
- Type III is RMR 40 to 60
- Type IV is RMR 20 to 40
- Type V is RMR 0 to 20.

The Indicated Mineral Resource and Mineral Reserve portion of the Lower Trap Deposit were drilled at an approximate 35 m spacing drill program. A proposed underground 10 m x 10 m spacing underground definition drilling program would provide Aura with additional information on the geology and mineralization, gold grades, lithology boundaries, and geotechnical information necessary for level development and final stope planning.

16.1.2 Geomechanical and Hydrogeological Input

In 2015, Knight Piesold Ltd. (“KP”) was engaged by P&E to complete a geomechanical and hydrogeological site investigation and provide Feasibility Study support recommendations for the proposed underground development at Ernesto. A report titled “Ernesto Deposit Geomechanical and Hydrogeological Input for Feasibility Study NB201-508/1-1” was completed in November, 2016. The work completed included:

- Reviewing the available geological, structural, geomechanical, and hydrogeological information for the deposit
- Completing a geomechanical and hydrogeological site investigation program
- Characterizing the encountered rock masses and defining geomechanical domains
- Developing a geotechnical block model using Vulcan software
- Completing stability analyses and providing rock mechanics design input for the proposed underground mine
- Estimating groundwater inflows to the underground mine.

The information provided by others for this study included:

- Topography provided by Aura (June 5, 2015)
- Exploration drillhole database for the Ernesto Deposit, including RQD and RMR data, provided by Aura (March 1, 2016 and April 6, 2016)
- Lithology models provided by Aura and P&E (February 10, 2016), including separate mylonite models (April 6, 2016) and saprolite models (April 6, 2016)
- Fault surfaces provided by Aura (January 21, 2016)
- Geotechnical solids provided by Aura (March 9, 2016)
- Proposed mine plan provided by P & E (March 29, 2016 and updated April 6, 2015 and May 5, 2016)
- Previous geotechnical and hydrogeological studies provided by Aura (June 5, 2015)

16.1.2.1 Geological Setting

16.1.2.1.1 General

The geological setting of the ore bodies is important for underground mine design and influenced the design of the completed site investigations. Background information on the main lithologies, mineralization, and large-scale structure is provided in the following sections. Unless otherwise noted, the information is summarized from discussions with Aura site geological staff and a Feasibility Study Report commissioned by Yamana Gold Inc. (Ausenco do Brasil Engenharia Ltda, 2010).

16.1.2.1.2 Main Lithologies

The Ernesto Deposit is located on the Aguapeí orogenic belt, which extends about 200 km NW from the south margin of the Amazonian Craton and Mato Grosso State. The deposit consists of an assemblage of sub-horizontal lithologies that dip at a shallow angle to the northeast. The lithologies that make up the Ernesto Deposit are described below, in order of increasing depth:

- Meta-Sediments - The Meta-arenite (“MA”), Meta-conglomerate (“MC”) and Feldspathic Meta-arenite (“MAF”) form the hangingwall of the deposit. In particular, the feldspathic meta-arenite forms the hangingwall (“HW”) in the immediate vicinity of the deposit. It can be locally altered, with inclusions of vesicles.
- Mylonite/Magnetite Sericite Schist (“SCH”) - The mylonite represents a shear zone along the contact between the meta-sediments and the tonalite. Locally it is known as the “Lower Trap” and hosts the mineralization at Ernesto. The mylonite is characterized by heavy sericite alteration and pervasive shearing, with lenses of quartz that generally follow the structure. The mylonite is typically between 5 m and 25 m thick.
- Basal Saprolite (“BS”) - The basal saprolite consists of altered tonalite along the footwall (“FW”) contact of the mylonite. The basal saprolite varies in thickness from a few metres to more than 15 m. The characteristics of this unit are variable, but it generally becomes gradually less altered and more competent with increasing distance from the mylonite. In some areas, a waste pillar of unaltered tonalite is present between the mylonite and the basal saprolite.

- Tonalite (“TON”) - The tonalite forms the basement rock of the regional sequence and the FW of the deposit. It is a fine-to-medium grained and weakly foliated intrusive volcanic.

16.1.2.1.3 Mineralization

Mineralization at Ernesto consists of gold-rich quartz veins and veinlets occurring within the Mylonite and occasionally within altered sulfidic horizons in the overlying meta-arenite units.

16.1.2.1.4 Large Scale Structures

The following large-scale structures have been observed at the Ernesto Deposit:

- SE-NW - The mylonite shear zone strikes southeast-northwest and dips at between approximately 20 and 50° to the northeast. This represents the dominant structural orientation at the deposit. The foliation and the bedding in the meta-sediments are parallel to the mylonite. Numerous faults and shears with this orientation have also been identified.
- W-E - Sub-vertical faults cross-cut the deposit striking west-east.
- NNE-SSW - Sub-vertical faults cross-cut the deposit striking north northeast - south southwest.

16.1.2.2 Site Investigation

16.1.2.2.1 General

A geomechanical and hydrogeological site investigation program was completed between July and September, 2015 and included the following:

- Six (6) oriented and triple-tubed diamond drillholes with a combined meterage of 997 m. Detailed geomechanical logging of the core was completed
- Detailed geomechanical logging of existing exploration core. Approximately 416 m of core was logged from four (4) drillholes
- Installation of six (6) vibrating wire piezometers in two (2) drillholes
- Collection and laboratory strength testing of drill core
- Surface mapping of existing open pit walls in the vicinity of the deposit.

The drillhole details for the site investigations at the Ernesto Deposit were initially proposed by KP and subsequently modified by KP, P&E and Aura. The drillholes associated with the site investigation program are shown on Figure 16.4 and summarized in Table 16.1.

Figure 16.4 Ernesto Site Investigation Drill Holes

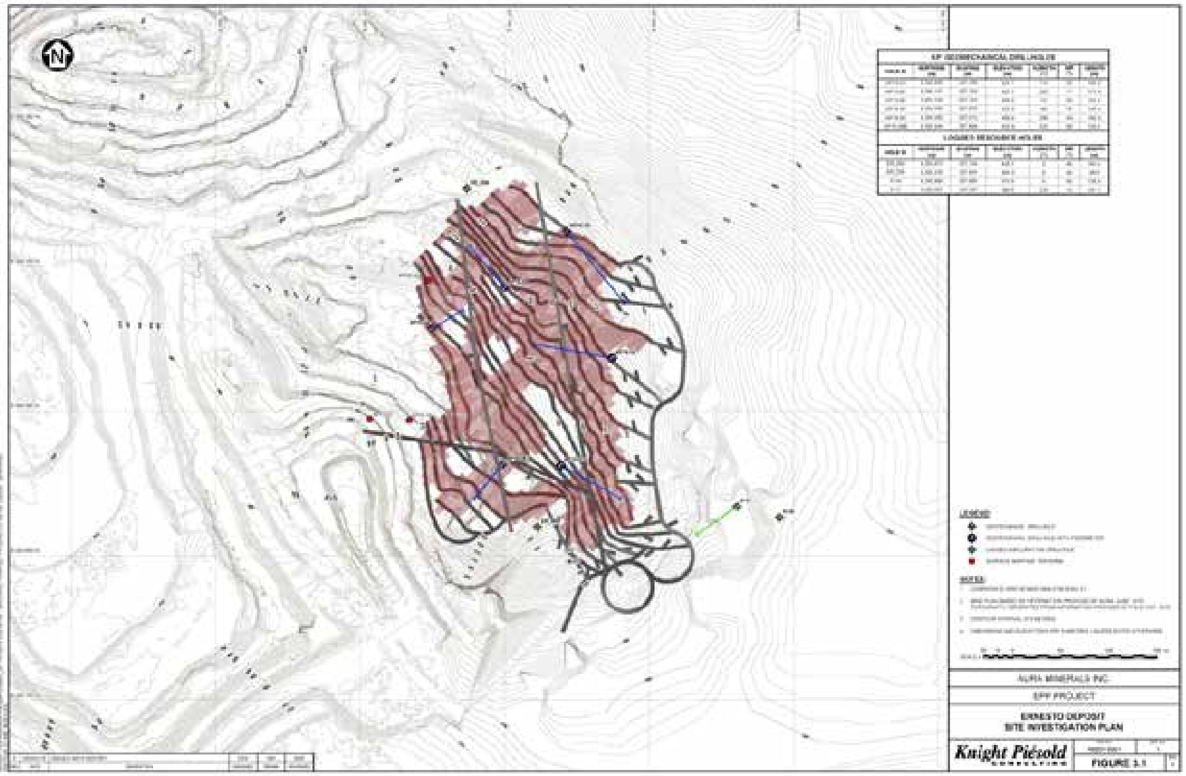


TABLE 16.1
SUMMARY OF ERNESTO SITE INVESTIGATION DRILL HOLES

Knight Piésold
CONSULTING

TABLE 3.1

AURA MINERALS INC.
EPP PROJECT

ERNESTO DEPOSIT GEOMECHANICAL AND HYDROGEOLOGICAL INPUT FOR FEASIBILITY STUDY
SUMMARY OF SITE INVESTIGATION DRILLHOLE DETAILS

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Deposit	Actual Drillhole Name	Drillhole Details						Abandoned	Instrumentation	Core Orientation	Interval with Geomechanical Logging		
		Collar Coordinates			Azimuth	Dip	Length				Depth from	Depth to	Length
		Northing	Easting	Elevation									
		(m)	(m)	(m)	(°)	(°)	(m)				(m)	(m)	(m)
ERNESTO	KP15-01	8,303,943	257,758	426.7	119	65	168.0	-	Y	Y	0.0	168.0	168.0
	KP15-02	8,304,127	257,700	423.1	320	71	175.4	-	-	Y	0.0	175.4	175.4
	KP15-03	8,304,186	257,764	408.0	141	60	192.9	-	-	Y	0.0	192.9	192.9
	KP15-04	8,304,085	257,625	426.2	80	75	145.4	-	-	Y	0.0	145.4	145.4
	KP15-05	8,304,055	257,812	409.9	280	64	189.3	-	Y	Y	0.0	189.3	189.3
	KP15-06	8,303,947	257,700	425.8	220	70	45.6	Y	-	Y	0.0	45.6	45.6
	KP15-06B	8,303,944	257,698	425.9	220	68	126.2	-	-	Y	0.0	126.2	126.2
	ER-038	8,304,230	257,661	404.3	0	90	164.2	-	-	-	0.0	44.0	44.0
											92.0	132.0	40.0
											154.1	164.2	10.2
	ER-069	8,303,879	257,734	426.70	0	90	99.4	-	-	-	25.8	75.9	50.1
								-	-	-	90.0	99.4	9.3
	P-16	8,303,889	257,985	375.5	0	90	138.0	-	-	-	23.1	138.0	114.9
	P-17	8,303,901	257,941	386.6	235	70	147.7	-	-	-	0.0	147.7	147.7

\[Table 3.1 - Summary of SI Drillholes 20150716.xlsx]Drillhole Summary

NOTES:
1. AZIMUTH AND DIP VALUES PRESENTED IN TABLE ARE AVERAGED FROM MAXIBORE DEVIATION SURVEYS OF ENTIRE DRILLHOLE.
2. DRILLHOLES SURVEYED BY AURA.

0	[illegible]	ISSUED WITH REPORT [illegible]	CAV	DDF
REV	DATE	DESCRIPTION	PREP'D	RVW'D

16.1.2.2.2 Geomechanical Drillholes

16.1.2.2.2.1 General

Six oriented and triple-tubed geomechanical drillholes were completed as part of the site investigation program. The drilling was carried out by Rede Serviços de Perfuração (“Rede”) and Foraco International SA (“Foraco”) under the direct supervision of KP and Aura site staff. The drillholes were completed using HQ3 diameter drilling equipment, except for KP15-01 which was reduced to NQ diameter due to adverse ground conditions encountered in the mylonite. Geology logs for the geomechanical drillholes were provided by Aura. Core photos for some of the geomechanical drillholes were provided by Aura and the remainder were completed by KP.

16.1.2.2.2.2 Oriented Core Drilling

Core orientation was undertaken on each of the geomechanical drillholes to determine discontinuity orientations within the encountered rock units. Core orientation was completed with the Reflex ACT III tool. This tool has emerged as the industry standard for core orientation due to its performance and ease of use. Difficulties are still occasionally encountered in obtaining consistent orientation data in highly fractured, broken, or poor quality rock.

KP staff were present at the drill rig to supervise the drilling and core orientation procedures for each drillhole. Aura geologists were trained by KP staff to assist in supervising core orientation as a cost-saving measure part-way through the program. Aura staff assisted in the supervision of drillholes KP15-01, KP15-03 and KP15-05.

All drill supervisors collected core orientation parameters to assess the quality of the collected data on an on-going basis. The collected parameters included the percentage of each drill run oriented, the angular difference between orientation lines on successive runs, Total Core Recovery (“TCR”) and Rock Quality Designation (“RQD”).

All drillholes were surveyed by Aura upon completion in order to determine the final collar coordinates and downhole deviation. The surveys were completed using a Reflex Maxibor. The drillhole discontinuity orientation data were corrected for any significant drillhole deviation (>5°) using the results of the surveys.

16.1.2.2.2.3 Geomechanical Logging of Oriented Core

Detailed geomechanical logging was completed at the drill and the core shack by KP staff. Rock mass parameters were input directly into KP’s electronic logging spreadsheet. Standard logging procedures were modified to ensure the project specific conditions were appropriately captured. Detailed logging parameters were collected to characterize downhole variations in the rock mass quality using both the Rock Mass Rating (RMR89) system (Bieniawski, 1989), and the Tunnelling Quality Index (NGI-Q) system (Barton et. al., 1974).

Field estimates of Uniaxial Compressive Strength (“UCS”) were made by KP and Aura staff with the assistance of a Schmidt Hammer. The Schmidt Hammer values were collected using procedures adapted from ASTM standards (ASTM D5873).

16.1.2.2.3 Geomechanical Logging Of Exploration Core

Additional detailed geomechanical logging was completed on core from four NQ diameter exploration drillholes. The logging was undertaken to provide a better overall characterization of the Ernesto rock units without the need for additional drilling.

It should be noted that only un-cut intervals of the historical drillholes were logged. Core cutting reduces both the apparent quality of the rock mass and the surface area on which the discontinuities can be characterized. No orientation data was collected from the exploration core.

16.1.2.2.4 Surface Mapping

Surface mapping of bench faces was conducted at several areas within the open pit adjacent the deposit to collect discontinuity orientations and better characterize the engineering properties of the near-surface rock masses. Opportunities for surface mapping were limited by the number and quality of exposures. Detailed RMR89 assessments were completed from appropriate exposures using line mapping techniques. This technique is best suited to the characterization of blocky rock masses with discrete discontinuities. Three (3) line mapping traverses were completed during the program. This technique is most similar to the procedure used for logging core and direct comparisons between the two methods are possible.

Surface mapping locations are shown (as red dots) on Figure 16.4.

16.1.2.2.5 Laboratory Testing

Core samples were collected from each of the geomechanical drillholes for each significant rock unit. These samples were selected for Uniaxial Compressive Strength ("UCS"), Triaxial Compressive Strength and Brazilian Indirect Tensile Strength testing. In all cases, more samples were collected than were needed for testing. This type of oversampling accommodates sample substitutions, breakage and/or changes in the planned testing program. Sample selection for lab testing involved the following:

- Identifying samples that were representative of the overall rock mass
- Selecting samples that do not have any obvious weaknesses or features that are likely to cause anomalous results (i.e., planar fabrics, large clasts, voids, etc.)
- Selecting samples from a range of drillholes and depths

The final sample selection process was completed at site by KP staff. The laboratory testing was carried out by the Instituto de Pesquisas Tecnológicas ("IPT") in São Paulo, Brasil. The test results are included in the appendices of the July, 2016, KP report. The tested samples were returned to Aura for assaying.

The laboratory testing is summarized below:

- Uniaxial Compressive Strength - A total of 59 laboratory UCS tests were completed as part of the 2015 program. Tests were completed on each of the major lithologies encountered at the deposit to determine the intact rock strength. The results suggest that the intact rock strengths at Ernesto vary significantly by lithology and range from <25 to >250 MPa.

- As much as possible, outlying results were identified and eliminated from further consideration. Tested samples could not be reviewed as they were returned to Aura for assaying.
- The Schmidt hammer rebound number for each tested UCS sample was compared to the laboratory result and a correlation between the two was developed for the Ernesto Deposit. The Schmidt hammer correlation was used to calibrate the rebound numbers collected during logging. The calibrated values were then used to provide a continuous UCS estimate along each drillhole. This estimate was also used to update the RMR89 values determined during the detailed geomechanical logging.
- Triaxial Compressive Strength - A total of 44 triaxial compressive strength tests were completed as part of the 2015 program. The tests were completed on each of the major lithologies encountered at the deposit, and were used to determine the intact rock strength under prescribed levels of confinement. The results are used to estimate the mi parameter that is required for the Hoek-Brown failure criterion, which allows the strength of a jointed rock mass to be estimated. The levels of confinement selected for testing were based upon the intact strength of the rock and the confinements expected within the slope using criteria established by Read (2005) and Read and Stacey (2009). Confining stresses of 10 MPa, 20 MPa and 30 MPa were specified for testing.
- Brazilian (Indirect) Tensile Strength - These tests were completed on each of the major lithologies encountered at the deposit, and were used to estimate the intact rock tensile strength for each unit.

As much as possible, outliers and results influenced by planar weaknesses were eliminated from the data. In some instances, the foliation was found to have impacted the test results. In general, the results are well distributed; however, there is some scatter that has been attributed to variations in composition.

16.1.2.2.6 Hydrogeology

16.1.2.2.6.1 General

The hydrogeological component of the site investigation program was designed to characterize the hydrogeological regime in the vicinity of the proposed underground development. The hydrogeological site investigations consisted of the installation of six (6) vibrating wire piezometers and the completion of hydraulic packer testing within the various rock units.

16.1.2.2.6.2 Hydraulic Conductivity Testing

Hydraulic conductivity testing was conducted during the advancement of the drillholes to characterize the encountered lithologies and large-scale structures. The tests were completed by KP staff using a hydraulically inflated Standard Wireline Packer System ("SWiPS") produced by Inflatable Packers International ("IPI") and supplied by Rede. Polymer-based drilling additives Polyplus, Polysafe 600 and Celutrol HV1 were used in several drillholes due to the encountered drilling conditions. In all cases prior to testing, the drillhole was flushed with fresh water to completely displace the water inside the hole. This was done in an attempt to remove any drilling additives (if used) or debris that could potentially influence the test results. Even so, the effects of drilling may still result in estimated hydraulic conductivities lower than the true values.

Constant head tests were completed over intervals ranging from approximately 7 m to 39 m in length. A mechanical flow meter with a minimum accuracy of 1 L was used to measure the injection rate for the constant head tests. The injection rate was calculated based on the volume of water discharged from a small graduated container when the flow rates were too low to measure reliably with the flow meter. A pressure transducer was installed in a housing unit below the packer tool to monitor the pressure during testing. Manual water level readings were taken prior to and following testing to monitor the water level recovery from drilling and testing.

16.1.2.2.6.3 Vibrating Wire Piezometer Installations

Vibrating wire piezometers were installed in drillholes KP15-01 and KP15-05 in order to measure groundwater elevations and gradients.

The vibrating wire piezometers and associated equipment were manufactured by Geokon Inc. ("Geokon") and supplied by G5 Engenharia LTDA ("G5"). Each installation was connected to a Model 8002-4 data logger in a fibreglass enclosure to allow for the collection of data at regular intervals.

The vibrating wire piezometers were installed using one-inch PVC pipe as a guide and tremie pipe. The depths at which the vibrating wire piezometers were installed were selected based on observations made during drilling. After the instrumentation was lowered into the drillhole, a validation check was performed to confirm that the piezometers were calibrated properly. The drillhole was then grouted in place using a cement-bentonite slurry produced at the drill rig.

The locations of the installations are shown on Figure 16.4.

16.1.2.3 Rock Mass Characteristics and Domain Definition

16.1.2.3.1 General

One of the main objectives of the site investigation program was to obtain information on the geomechanical characteristics of the encountered rock masses. Rock mass characteristics are divided between intact material properties and the characteristics of the discontinuities. This section describes the geomechanical characteristics of the main rock units of the Project. The characteristics of each unit are summarized in Table 16.2.

16.1.2.3.2 Intact Rock Properties

The following intact rock properties have been estimated for each domain based primarily on the results of the laboratory strength testing:

- Unconfined Compressive Strength ("UCS")
- Triaxial Compressive Strength
- Brazilian Tensile Strength
- Unit weight
- Young's Modulus
- Poisson's Ratio

In some cases, where a domain was a minor unit or where limited intervals of a domain were encountered, the intact rock properties have been selected based on field estimates or published values. These cases are noted in Table 16.2.

16.1.2.3.3 Rock Mass Quality

The rock mass quality of each major lithology has been characterized using the Rock Mass Rating (RMR89, Bieniawski, 1989) and NGI-Q (Barton et. al., 1974) rock mass classification systems. The characterization is based upon the detailed geomechanical logging and field UCS estimates completed during the geomechanical site investigation programs.

The rock mass quality typically ranges from FAIR to GOOD (i.e., RMR89 values typically ranging from 60 to 75), though the mylonite/ore and saprolite are of worse quality (i.e., RMR89 values of 25 to 55). Discontinuities typically have slightly rough surfaces spaced 60 to 600 mm apart. Most have no infill or a thin infill (commonly sericite, iron oxide, or clay), and show slight to no weathering. Aperture typically ranges from <0.1 mm to 1.0 mm.

The RMR89 and Q' design parameters are the 30th percentile values of the distribution for each lithology and are included in Table 16.2.

TABLE 16.2
DESIGN VALUES

Knight Piésold CONSULTING

TABLE 4.1

AURA MINERALS INC.
EPP PROJECT

ERNESTO DEPOSIT GEOMECHANICAL AND HYDROGEOLOGICAL INPUT FOR FEASIBILITY STUDY
DESIGN VALUES

Print Nov/08/16 8:32:34

Unit Group	INTACT ROCK PROPERTIES			ROCK MASS PROPERTIES		
	mi	UCS (MPa)	Unit Weight (g/cm3)	RMR_{89}	GSI	Q'
Meta-arenite	22	120	2.6	65	60	5.1
Meta-conglomerate	30	220	2.7	65	60	4.7
Feldspathic Meta-arenite	22	120	2.6	60	55	3.8
Mylonite / Ore	9	20	2.7	30	25	0.4
Basal Saprolite	-	5	1.8	25	20	0.04
Waste Pillar	-	15	2.6	50	45	2.3
Tonalite	21	160	2.7	65	60	5.5

I:\201\00508\01\A\Report\1 - SI and Feas Input\1 - Final\Tables\[Table 4.1 Updated Design Parameters Mar 8, 2016.xlsx]Rock Mass Parameters

NOTES:

1. mi VALUES ARE BASED ON LABORATORY TESTING. DATA IS NOT AVAILABLE FOR THE BASAL SAPROLITE AND THE WASTE PILLAR.
2. RMR AND Q' VALUES REPRESENT 30TH PERCENTILE OF DATA BY RUN WEIGHTED BY LENGTH.
3. GSI CALCULATED USING THE FORMULA: GSI = RMR_{89} - 5.
4. THE INTACT ROCK PROPERTIES FOR THE META-ARENITE AND FELDSPATHIC META-ARENITE ARE BASED ON THE RESULTS FROM BOTH UNITS.

0	[illegible]	ISSUED WITH REPORT [illegible]	[illegible]	[illegible]
REV	DATE	DESCRIPTION	PREP'D	RVW'D

16.1.2.3.4 Discontinuity Orientations

Three dominant discontinuity orientations have been identified at the Ernesto Deposit based on the oriented core and surface mapping data:

- Joint Set C - A joint set striking southeast-northwest and dipping shallowly to the northeast. This is the dominant structural orientation and corresponds to the bedding in the sediment and the foliation of the rock mass. This joint set is oriented sub-parallel to the ore body.
- Joint Set A - A joint set striking west-east to southwest-northeast and dipping steeply to the south to southeast. In some instances, the dip of the discontinuities varies through vertical, resulting in pole concentrations on both sides of the stereonet. This joint set cross-cuts the ore body.
- Joint Set B - A joint set striking approximately north-south and dipping steeply to the west. In some instances, the dip of the discontinuities varies through vertical, resulting in pole concentrations on both sides of the stereonet. This joint set cross-cuts the ore body.

Note that the drillhole discontinuity orientation data were filtered to exclude discontinuities from runs with run-on-run consistency in the FAIR (20 to 35°) or POOR (>35°) ranges. In addition, all discontinuities flagged as possible breaks or labelled as a vein or veinlet were removed from the data set based on a review of their impact on the results.

16.1.2.3.5 Domain Definition

As part of the design process, the encountered rock masses are often grouped into geomechanical domains in order to simplify the stability analyses. Each domain contains rock masses with similar engineering characteristics and that are expected to perform similarly during mining. Several possible domain definitions were considered, including:

- Lithology - The potential for variation in rock mass quality between the main lithologies was evaluated.
- Lithology and Localized Zones of Reduced Rock Mass Quality - The potential for variation in rock mass quality within select lithologies was evaluated. In particular, the potential for a transition or shear zone in the meta-sediments in the immediate HW of the mylonite/ore was evaluated. The potential for a similar transition between the Basal Saprolite and the Tonalite was also considered.

The suitability of each domain definition was evaluated through a comparison of their individual engineering characteristics. Ultimately, it was not possible to identify sufficient evidence for the transition zones considered and the lithology definition was selected as being most appropriate for the current study. As a result, the following domains were defined:

- Meta-arenite
- Meta-conglomerate
- Feldspathic Meta-arenite
- Mylonite / Ore
- Basal Saprolite
- Waste Pillar
- Tonalite

The design properties selected for each domain are summarized in Table 16.2.

16.1.2.3.6 Geomechanical Block Model

A geomechanical block model was created to support the mine design process. The model was developed using the following inputs:

- The detailed RQD and RMR89 data from the six geomechanical drillholes and four exploration drillholes logged by KP during the site investigation program.
- A database of RQD and RMR89 data from 157 historical exploration drillholes. The RQD data was collected by Yamana. The RMR data was primarily based on a review of core photos completed by Luis Navarro of Aura. In a few instances, this data has been supplemented with more detailed historical data collected by Yamana.
- A 3D model of the mylonite produced by Aura and P&E.
- A 3D model of the saprolite produced by Aura and P&E.

The mylonite and saprolite models were used to create domains within the block model. The RQD and RMR89 data were not interpolated between domains. This was done to reflect the significant contrast in rock mass quality between these units and the more competent meta-sediments and tonalite. From a review of the data and discussions with Aura, the RQD data is considered to be less reliable than the RMR89 data. The block model was sent to Aura in April, 2016.

16.1.2.4 Mine Design Input

16.1.2.4.1 General

The general underground mining approach is understood to be as follows, based on discussions with P&E and Aura:

- Mining Method: Drift and Fill.
- Stope Sizing: Stopes are 3.5 m to 7 m wide, and 5 m high.
- Backfill: Cemented Rock Fill ("CRF")
- Overall Access: A ramp from surface. Secondary egress is via a ventilation drift with an adit in the existing open pit.
- Access Sizing: The main ramp is 4.5 m wide and 4.5 m high. The access drives are 4 m wide and 4 m high.
- Depth: Approximately 55 to 165 m below ground surface ("mbgs").

16.1.2.4.2 Expected Rock Mass Quality

The quality of the rock masses in which the proposed mine openings will be established is fundamental to the expected performance of the openings. The rock masses forming the stopes and access drives were evaluated using the lithology models developed by Aura and P&E:

- The back is expected to be primarily in the mylonite and feldspathic meta-arenite
- The walls are expected to be primarily in the mylonite/ore
- The floor is expected to be primarily in the mylonite, waste pillar and saprolite

The distribution of rock mass quality in the immediate back of the stopes was estimated using the geotechnical block model. The model suggests that there will be significant variation in the rock mass quality in the back. This is consistent with the expected distribution of rock mass quality in the mylonite. Aura has completed an assessment of the spatial distribution of mylonite and estimates that 44% of the stopes in the current mine plan will have mylonite in the back. However, the geotechnical block model does not have the resolution to capture this variability in the mylonite.

To better define the rock mass quality in the stope backs and walls, a more detailed assessment of the rock mass quality within the mylonite was completed based on the results of the detailed geomechanical logging. Approximately 50% of the mylonite is expected to have a rock mass quality of RMR89 40 or less (equivalent to Types V, IV). Rock masses in this range are expected to require additional ground support to manage ravelling and progressive failures (e.g. shotcrete).

The conditions in the floor of the stopes will be managed at an operational level. It is understood that P&E has incorporated a working floor of rockfill and/or CRF in areas where saprolite is expected in the floor of the stopes.

16.1.2.4.3 Stope Dimensions

Stope dimensions for drift and fill mining were evaluated. Achievable back spans were evaluated using empirical design methods. Both the Span Design Curve (Wang et al, 2000) and the Unsupported Span Curve (Barton, 1976) were used.

The following back spans are thought to be achievable under standard 2.4 m (8 ft) long primary ground support based on the expected rock mass quality:

- Type II or Better: 7 m
- Type III: 3.5 m
- Type IV or V: 3.5 m

Larger spans, up to a maximum of 9 m, are likely achievable in areas with rock mass quality of Type III or better with the use of longer ground support (e.g. 3.6 m Super Swellex). In irregular openings or intersections, the span should be measured using the inscribed circle approach.

Where possible, the stope back should be established in the meta-sediments instead of the less competent mylonite. A shanty profile cut to the bedding is expected to result in improved back performance within the meta-sediments.

The base stope height of 5 m was proposed by P&E. When multiple cuts are taken, the exposed wall height should not exceed 10 m. Note that a 10 m wall is tall for typical practice in the Canadian mining industry and will need to be monitored and managed carefully.

16.1.2.4.4 Extraction Sequencing

The proposed mine plan and extraction sequencing divides the deposit into six mining blocks or phases. Each mining block corresponds to a range of elevations within the deposit. Within each mining block, stopes will be mined in horizontal drifts approximately perpendicular to the strike of the deposit. A primary-secondary sequence has been proposed. All of the primary stopes within a given mining block would be mined first, starting with the stopes near the decline. The secondary stopes would then be mined on retreat back towards the decline. The stopes closest to the decline would not be mined until the end of the mine life in order to protect the decline.

These stopes would be mined on retreat (i.e., bottom-up). The primary stopes would be backfilled with CRF, while the secondary stopes would be backfilled with uncemented fill.

The extraction sequencing is currently planned at a relatively high level. It is understood that the detailed sequencing on a stope by stope basis will be developed once definition drilling is completed.

KP has provided the following comments on the extraction sequencing:

A pillarless retreat sequence is preferable to a primary - secondary sequence from a rock mechanics perspective as it will:

- Eliminate the pillars between stopes and reduces the stand-up time of the access drives
- Reduce the effective span of the access drives and improve their performance
- Reduce maintenance of the access drives and intersections

Adjacent primary stopes should not be mined or left open concurrently. The secondary stope pillars that would result are not expected to be stable in the mylonite in all cases. Note that the decision to use a primary-secondary sequence instead of the pillarless retreat sequence was made after the completion of KP's work and the stability of the secondary stope pillars has not been evaluated.

The stopes should be backfilled as soon as possible after mining. The backfill should be tight to the back and extend to the wall of the access drives. This will help limit the effective span of the access drives and will help maintain the integrity of the secondary stope pillars.

Stopes directly opposite each other in the access drifts should not be mined or left open concurrently. Similarly, the initial rounds of the secondary stopes should not be taken during development of the access drive. These measures will limit the effective span of the access drives.

16.1.2.4.5 Ground Support

Ground support recommendations are discussed below. The recommendations have been developed for three different categories of openings, depending upon their anticipated service life, span and importance (i.e., consequences if access to the excavation was interrupted).

The three categories are as follows:

- Long-Term Access Development (i.e., the ramp)
- Short-Term Access Development (i.e., access drives)
- Drift and Fill Stopes.

The recommended support systems were based on Canadian mining practice and experience in similar mines. Specific considerations are discussed below.

Ground Control Concerns - At the planned mining depths, the main ground control issues are expected to be associated with controlling zones of reduced rock mass quality and recognizing and controlling wedges or blocks in the back and walls. Potential causes of ground control issues include:

- The thickness and quality of the Mylonite in the back. The properties of this unit have a significant impact on the ground support recommendations. Aura has estimated that 22% of the stopes have mylonite with a rock mass quality of Type IV or V in the back. These stopes will require shotcrete.
- Random structures intersecting the mine openings. These interactions could result in the formation of wedges that overtop the recommended ground support.
- Locally reduced rock mass quality associated with faults. Several faults intersect the mine openings. These faults represent zones of reduced rock mass quality and will likely require additional ground support.
- Larger spans, particularly those associated with intersections between the stopes and the main access drives.
- Multiple cuts in the drift and fill stopes, resulting in increased stope wall heights. The increased height will limit access to the upper portion of the wall and will increase the likely size and frequency of any instabilities. Shotcrete should be applied to the walls of the upper cut prior to establishing the lower cut.
- Additional Ground Support - The ground support recommendations represent the basis for a minimum ground support standard. The purpose of a minimum ground support standard is to safely accommodate the most commonly encountered ground control issues. Adverse ground conditions will require the use of additional ground support. The support elements used under adverse conditions will vary, but are expected to include longer tendons, shotcrete and/or mesh straps. Based on the available rock mass information, enhanced support will likely be required when random features are encountered that can form wedges or when zones of reduced rock mass quality are intersected. On this basis, it is considered reasonable to assume that an additional 5% of the stopes and access drives will require shotcrete.

16.1.2.4.6 Cemented Rockfill Strength

The mix design for the cemented rockfill was completed by Paterson and Cooke ("P&C"). Guidance on the required strength of the CRF was provided by Pakalnis & Associates under sub-contract to KP. An in-situ CRF strength of 0.5 MPa was recommended, assuming the following considerations:

- Drift and Fill mining with a 5 m cut height
- Tight-filling of the stopes to the back using a rammer jammer. The CRF should be placed at an angle steeper than 45° to limit the effect of cold joints.
- Controlled blasting in proximity to any placed CRF
- Documentation of the CRF placement and quality control testing.

16.1.2.4.7 Review of Proposed Underground Mine Plan

The proposed underground mine plan was reviewed to provide guidance on mine layout and geometry with respect to ground control and rock mechanics issues. Two reviews were completed on interim versions of the mine plan that incorporated a combination of drift and fill and longhole open stoping. The reviews included comments on:

- Placement of infrastructure and accesses relative to the proposed mine workings and regions of reduced rock mass quality (e.g. the saprolite)

- The crown pillar between the proposed underground mine and the existing open pit
- Offsets between adjacent openings.

16.1.2.5 Estimate of Groundwater Inflows to the Underground Mine

16.1.2.5.1 General

One of the objectives of the site investigation program was to improve the characterization of the hydrogeological regime in the vicinity of the proposed underground mines at the Ernesto Deposit. The hydrogeological regime at the Project was previously studied by Artois (2009) and Schlumberger (2011) and groundwater inflows to the underground mine were estimated. The groundwater inflow estimate has been updated as part of the current study to reflect the additional data collected during the site investigation program.

16.1.2.5.2 Review of Previous Groundwater Inflow Estimate

Groundwater inflows to the Ernesto Project underground workings have been previously estimated by Artois (2009) to range between 4 and 210 m^3/hr). These inflows were estimated from the modified Dupuit equation (Equation 1) for radial groundwater flow to a well in an unconfined aquifer (Artois 2009):

$$Q = \frac{\pi K(H^2 - h^2)}{\ln(R_O/r_w)} \qquad \text{(Equation 1)}$$

Where:

Q	=	Inflow [m3/s]
K	=	Hydraulic Conductivity [m/s]
H	=	Groundwater System Static Water Level [m]
h	=	Groundwater Level at Underground Workings [m]
RO	=	Drawdown Cone Radius of Influence [m]
rw	=	Radial Footprint of Underground Workings [m]

At the time of the Artois (2009) assessment, hydraulic conductivity data was not available for the lithologies expected in the vicinity of the underground workings. A theoretical range of possible hydraulic conductivity values was estimated using a method from Hoek and Bray (1981, Equation 2), which defines the hydraulic conductivity of a fractured rock environment based on the fracture frequency and aperture of the fractures:

$$K = \frac{ge^3}{12\text{vb}} \qquad \text{(Equation 2)}$$

Where:

K	=	Hydraulic Conductivity [m/s]
g	=	Gravitational acceleration [m/s^2]
e	=	Fracture aperture thickness [m]
v	=	Coefficient of kinematic viscosity [m^2/s]
b	=	Spacing between fractures [m]

Schlumberger (2011) also used this method to estimate a range of theoretical hydraulic conductivities for the Ernesto Project. However, Equation 2 represents a simplification of groundwater flow through natural fractured rock environments and is generally regarded as a theoretical basis for fracture flow, rather than a tool to estimate hydraulic conductivity (Wyllie and Mah 2004). Hydraulic conductivity values calculated using this method tend to be overestimated relative to observational data as the method does not provide an assessment of fracture connectivity or connectivity to a groundwater recharge source.

Water level data in the vicinity of the underground workings was not available when the Artois (2009) inflow assessment was completed. As a result, the groundwater table was estimated using other information from the site, including the location of springs, historical open pit pond elevations, water levels in open exploration boreholes, and shallow overburden monitoring wells. The resulting static water level may not reflect pore pressures at the greater depth of the proposed underground workings.

Seasonal variability as a result of the unimodal dry/rainy season annual precipitation pattern was not addressed in this study. Approximately 75% of the average annual precipitation at the Project occurs during the rainy season between November and March (Artois 2009, Schlumberger 2011).

16.1.2.5.3 Updated Inputs to the Groundwater Inflow Estimate

16.1.2.5.3.1 General

The parameters used in the Artois (2009) groundwater inflow assessment were updated to reflect the current mine plan and the hydrogeological data collected during the 2015 geotechnical site investigation.

16.1.2.5.3.2 Hydraulic Conductivity

Estimates of the hydraulic conductivity of the rock mass were developed from the results of the nine packer tests were completed during the 2015 site investigation. The estimates ranged between 2 x10-8 m/s and 5 x10-7 m/s with a geometric mean value of 8 x10-8 m/s. These estimates are on the low end of the range of the theoretical values calculated by Artois (2009) and Schlumberger (2011), which suggests that Equation 2 may overestimate hydraulic conductivity values at the Project.

16.1.2.5.3.3 Groundwater Elevations

Groundwater pore pressure data was collected from vibrating wire piezometers installed in drillholes KP15-01 and KP15-05. The pore pressure data was compared to rainfall data provided by Aura from a rain gauge installed at the Ernesto processing plant.

KP15-01 - Pore pressure data indicate that groundwater elevations rose from 350 masl to 365 masl during rain events in January 2016.

KP15-05 - Pore water pressure data indicate that groundwater elevations rose from 360 masl to 382 masl during the same rain events.

The data from both vibrating wire piezometer installations indicate a sustained, rapid increase in pore pressures following a large 100 mm rain event that occurred on January 17, 2016. Pore pressures increased until the first week of March, after which they began to subside. This pore pressure response is not fully understood and other factors (changes in surface runoff response, partial saturation of overburden, etc.) may be important.

Based on the data from the vibrating wire piezometers, water level elevations of 355 masl and 380 masl were selected as approximate dry and rainy season static water levels, respectively.

Note that the vibrating wire piezometer data is currently only available for the period from September 2015 to May 2016. A longer record is required to reliably estimate dry and rainy season water level elevations.

16.1.2.5.3.4 Mine Plan

The mine plan and extraction sequence provided by P&E on June 16, 2016 were used to simulate the change in mine geometry over the life of the mine. The proposed sequence consists of six phases of mining. For the purpose of the groundwater inflow estimate, the minimum mining level elevation for each phase of mining was assumed to be the groundwater level in the immediate vicinity of the mine workings. The elevation of the lowest phase was used if two or more phases are intended to be mined concurrently.

16.1.2.5.4 Updated Groundwater Inflow Estimate

The groundwater inflow assessment completed by Artois (2009) using Equation 1 was updated to reflect the additional data collected during the site investigation program. The minimum and maximum groundwater inflow estimates range from 1 m^3/hr to 94 m^3/hr.

The updated groundwater inflows are consistent with the general range of inflows estimated by Artois (2009). The reduced maximum and average inflow estimates reflect the updated mine plan and the lower hydraulic conductivity values determined from packer testing of the deposit rock masses. The hydraulic conductivity of the rock mass is expected to be the largest source of uncertainty in the groundwater inflow assessment.

The groundwater inflow estimates are based on the expected extents of mining at the end of each phase and are not cumulative. The estimates assume that the mined stopes are backfilled with cemented waste rock before commencing a subsequent phase and that any mined stopes below the active level are allowed to flood.

The updated groundwater inflow estimate was intended to provide a check on the previous high-level inflow estimate from Artois (2009) and reduce uncertainty in the input parameters. Transient groundwater inflows, such as those associated with the intersection of a water-bearing structure, are not considered. It is important to emphasise that the groundwater inflow estimate is subject to uncertainty and represents a best estimate. It is reasonable to assume that, in practice, the inflow rate could vary by an order of magnitude. If the operation is sensitive to the potential range of groundwater inflows, a more detailed assessment is recommended as the basis for the design of a dewatering system.

16.1.2.6 Operational Considerations

The performance of the underground openings and the implementation of the rock mechanics recommendations provided in this Report will be influenced by the operational considerations discussed below.

Successfully implementing the proposed mine plan will require operational discipline and an ongoing commitment to planning and data collection, including the use of instrumentation.

The mine engineering department will need to include adequate ground control staff, both during mine development and operations. The duties of the ground control staff will include:

- Discussing and signing-off on all of the excavation layouts
- Making regular visits underground to resolve ground control issues, observe ground conditions and document how they are changing over time
- Developing a ground support quality control program and completing regular ground support audits
- Adjusting the support package to accommodate the encountered ground conditions
- Working with ground support suppliers to ensure that there is a reliable supply of the correct ground elements
- Working with operations staff to optimize drilling and blasting practices
- Developing a suitable program to collect relevant rock mass information on an ongoing basis
- Ensure that any required stability analyses are undertaken in a timely fashion
- Design and coordinate the installation of underground instrumentation (e.g., in the crown pillar, wide spans, areas with potential ground control issues, etc.)
- Completing quality control testing on the backfill
- Documenting backfill placement, including location, cure time, mix design and quality control testing
- Drilling and blasting practices directly influence excavation and backfill performance and are closely related to both the frequency of ground control issues and the amount of ground support required. As such, improvements to drilling and blasting practice will increase safety, reduce costs and increase productivities.
- The proposed mining method and sequence requires that the stopes be tight filled. If gaps are present between the CRF and the back of the stope, shotcrete pillars or another form of span interrupter will need to be placed in the gaps while mining the adjacent stope.
- The stope cycle time from the first production blast to the completion of backfilling should be minimized. Limiting the cycle time will improve the performance of the stope (including reduced dilution), as well as that of adjacent stopes.
- The design recommendations presented in this Report should be refined as necessary during the next level of design and/or operations based on the actual rock mass conditions encountered and the observed performance of the mine openings.

16.1.2.7 Geomechanical and Hydrogeologic Summary

16.1.2.7.1 Conclusions

Underground rock mechanics design recommendations have been provided for the Ernesto Deposit on:

- Achievable opening dimensions
- Extraction sequencing
- Ground support
- Cemented rockfill strength
- Infrastructure and access placement
- Various other operational considerations
- Groundwater inflows to the proposed underground mine have also been estimated.

The recommendations, and the analyses on which they are based, are appropriate for feasibility level design. The provided design recommendations are based upon the currently available geological, structural, geomechanical and hydrogeological data. The completed stability analyses suggest that the recommendations are reasonable and appropriate. The recommendations assume that controlled blasting and proactive geotechnical monitoring will be undertaken along with an ongoing commitment to geomechanical and hydrogeological data collection and analyses. Maintaining flexibility in the mine plan will be important to accommodate any ground control issues.

16.1.2.7.2 Risks And Opportunities

A number of potential risks and opportunities for the successful implementation of the design recommendations have been identified over the course of the reviews and analyses.

Potential risks include:

- Access Drives - The performance of the access drives is sensitive to the mining sequence, effective spans established and the ground support practices. Larger effective spans and increased stand-up time will increase the likelihood for instabilities, increased ground support and rehab requirements, and decreased production.
- Backfill - The stope span recommendations are sensitive to the ability of mine personnel to consistently tight fill the mined stopes as soon as possible after the completion of each stope. Poor quality or delayed backfill will cause the main access drives to deteriorate and will make mining of the secondary stopes more difficult.
- Mylonite - The span and ground support recommendations are sensitive to the thickness and rock mass quality of the mylonite. If the mylonite is generally thicker or of poorer rock mass quality than currently expected, increased ground support will be required and a higher proportion of the stopes will need to be mined with a 3.5 m back span. The definition drilling proposed by Aura and P&E will be an important step in improving the understanding of this unit.
- Secondary Stope Pillars - The stability of the secondary stope pillars has not been evaluated. The secondary stope pillars are expected to be founded on the saprolite

in some areas. There is a risk that the pillars could fail into an adjacent open primary stope. The performance of the pillars will be dependent on the pillar geometry and rock mass quality, the detailed excavation sequencing, backfill practices, and the presence and thickness of any saprolite.

- Crown Pillar - The stability of the crown pillar has not been evaluated in detail. The crown pillar will be established late in the mine life. The proposed mining method, mine geometry and sequence limit the effect of potential instabilities associated with the crown pillar and provide an opportunity to gain experience with the deposit rock masses before the pillar is established. Additional engineering studies should be completed prior to establishing the crown pillar.
- Groundwater Inflows - The groundwater inflow estimates are sensitive to the hydraulic conductivity of the rock mass and the potential for intersections with water-bearing structures. The actual groundwater inflows could vary from the estimates, potentially requiring additional dewatering.

Potential opportunities include:

- Fibrecrete - There may be an opportunity to use fibrecrete instead of the combination of shotcrete and mesh. Depending on a number of factors, this could improve cycle time.
- Mylonite - The span and ground support recommendations are sensitive to the thickness and rock mass quality of the mylonite. If the mylonite is generally thinner or of better rock mass quality than currently expected, less ground support will be required and a higher proportion of the stopes can be mined with the full 7 m back span.
- Mining Sequence - The development of a detailed mining sequence may identify opportunities to improve expected opening performance and productivity. For example, a pillarless retreat sequence is expected to improve the performance of the access drives.

16.1.2.7.3 Future Work

Future work should include more detailed analyses based on additional or updated data for the deposits in order to support the next stage of engineering.

Additional data requirements include:

- Updating the existing 3D lithological models, including the saprolite and mylonite models, to incorporate the results of any additional exploration drilling and/or an improved understanding of the deposit geology.
- Expanding the existing 3D structural model to encompass the full extents of the proposed mining area as well as sub-horizontal features.
- The rock mass characteristics in the immediate vicinity of the crown pillar should be better defined during the next phase of design or during the early stages of mining.
- Additional geomechanical logging should be completed to better define the spatial variation of the rock mass quality in the immediate HW of the proposed stopes, as well as the spatial variation in the distribution of the mylonite and saprolite. The definition drilling currently proposed by Aura could be used for this purpose.

- Additional hydrogeological data should be collected if the project economics or operating conditions are sensitive to the groundwater conditions and groundwater inflow estimate. For example, the completion of additional packer testing and the installation of additional vibrating wire piezometers could be used to refine the hydrogeological characterization and evaluate the potential for spatial variability.
- The groundwater pore pressure data from the existing vibrating wire piezometers should be recorded and reviewed on a regular basis.
- The encountered rock mass quality and observed opening performance should be documented during development of the proposed definition drill drives and the initial stages of underground mining. This represents an opportunity to refine the stope dimensioning and ground support prior to the start of production.
- A detailed extraction sequence should be developed for the proposed stopes.

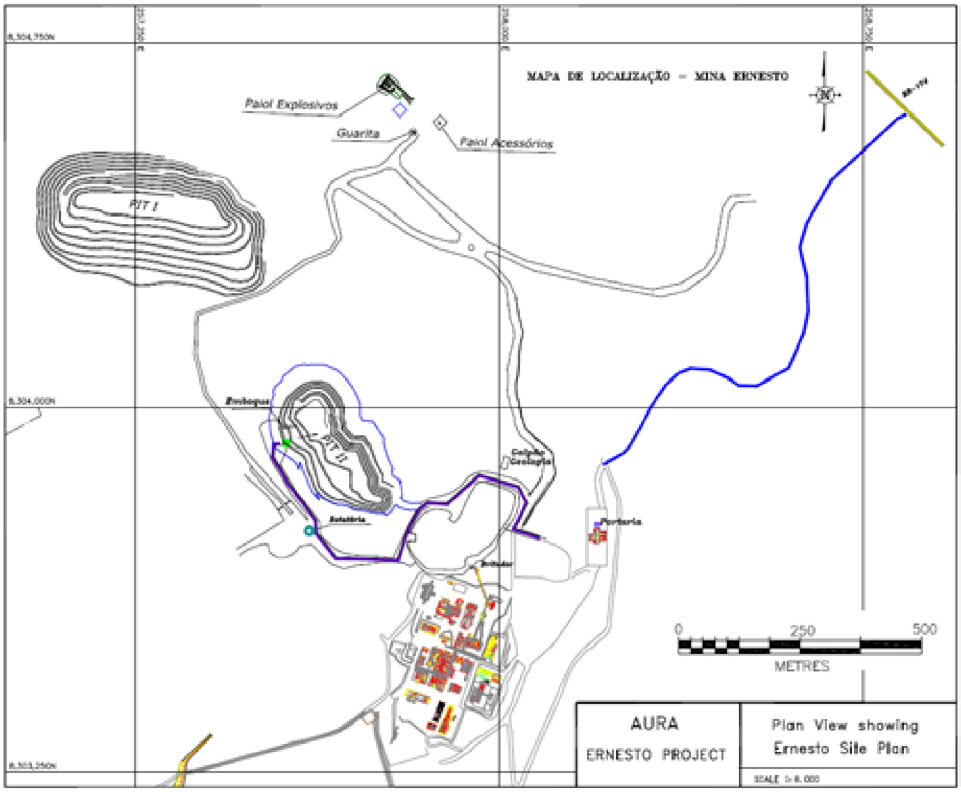
For detailed design, the domain definition, stability analyses, recommendations, and groundwater inflow estimate should be updated to account for the results of the additional site investigations and any changes to the geological models, large-scale structural interpretations and/or underground mine plan. Additional analyses are also recommended to advance the recommendations to support detailed design. These analyses include:

- Evaluating the required crown pillar dimensions
- Reviewing the detailed extraction sequencing once it had been developed
- Evaluating the stability of the secondary stope pillars, including the impact of the Saprolite
- Further analysis of the mylonite and its influence on achievable stope dimensions and ground support following the completion of the definition drilling
- Developing a transient groundwater inflow model to better account for variations in groundwater inflows as mining progresses (if the mining operations are sensitive to the possible range of inflows)
- Finally, the review of the underground mine design identified a number of considerations that should be addressed during the next stage of design and engineering.

16.1.3 Ernesto Site Layout

A general site plan is shown in Figure 16.5.

Figure 16.5 Ernesto General Site Plan



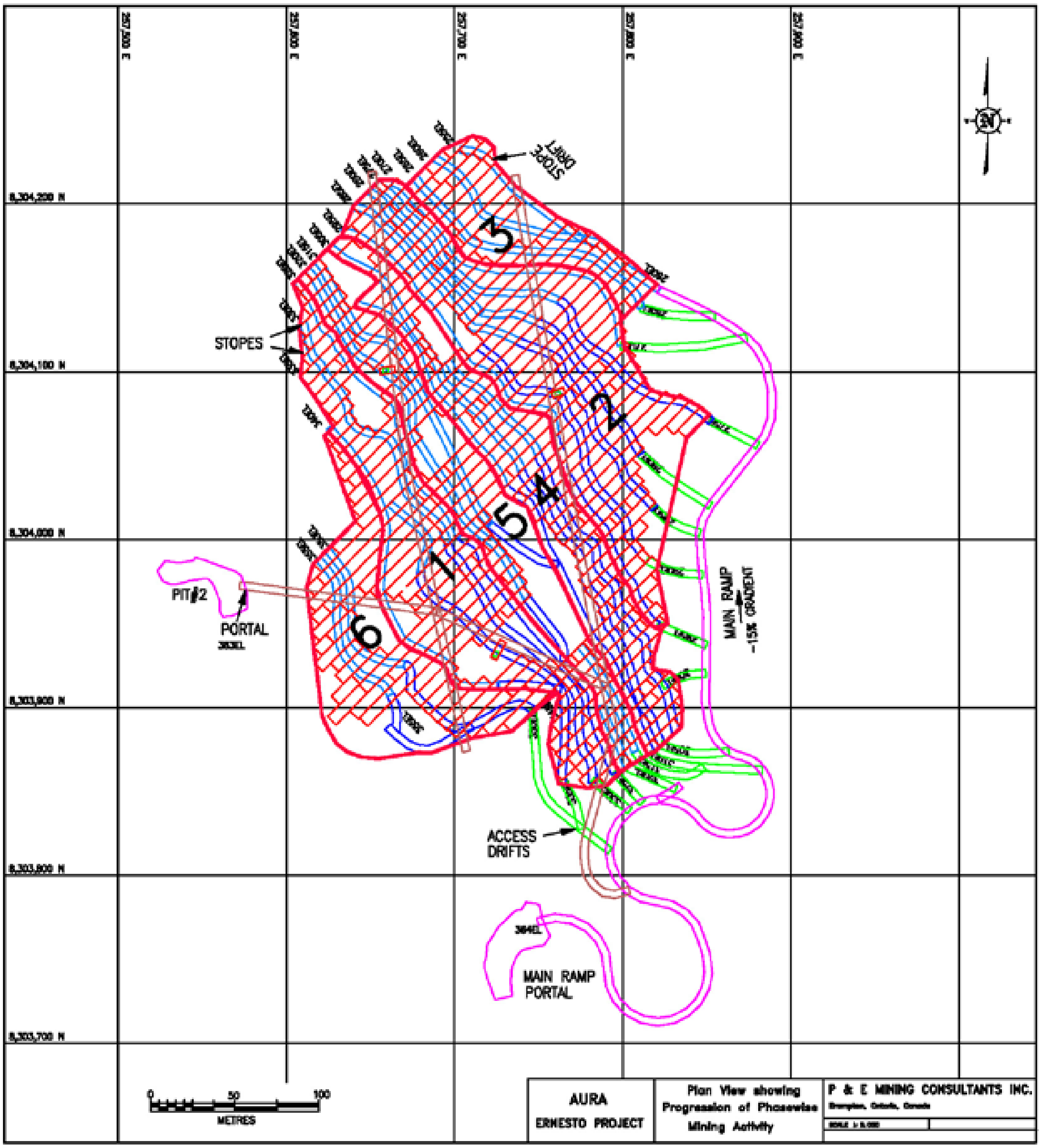
16.1.4 Ernesto Underground

16.1.4.1 Hangingwall Development For Definition Drilling

The underground definition drilling program would be used to provide additional geological, grade and geotechnical information for production stope planning. The drilling would be conducted from the East and West definition drill headings and an interconnecting heading that are to be developed in the HW metasediments. These headings would be developed from a portal to be collared into the upper section of the pit wall in historic Pit #2. Aura would also drive an internal ramp to connect the definition drill workings to the main ramp.

There would be six stope development phases as shown in Figure 16.6. The definition drilling for each phase would commence in advance of stope development to provide Aura's mine planners with additional information on the geology, local structures, gold grades, ore zone thicknesses, hangingwall and footwall elevations, geotechnical conditions, etc. required for detailed level development and final stope planning and production scheduling, and to precisely position the drifts with the right hand shoulder of the drive located at the planned HW contact position. The new information to be collected as part of the definition drilling program would supplement the existing database. A 10 m x 10 m drill pattern was selected and is believed to be reasonable at this point in time, with the final pattern to be a compromise between locating the ore drifts precisely and the cost of incurring dilution.

Figure 16.6 Mine Phases 1 to 6



The definition drilling would initially be done using Aura's diamond drills and equipment. A diamond drill contractor would provide additional drills and drillers as required to meet the definition drilling schedule. The BQ size definition drill holes would be grouted once completed. The definition drilling meterage and projected ore tonnes to be extracted from each phase are shown in Table 16.3.

TABLE 16.3 DEFINITION DRILLING METRES AND PROJECTED ORE TONNES BY PHASE					
Phase	**Definition Drilling**	**Development Ore**	**Primary Stope Ore**	**Secondary Stope Ore**	**Total Ore**
	(m)	**(kt)**	**(kt)**	**(kt)**	**(kt)**
1	6,291	58	53	56	167
2	9,722	96	79	83	258
3	3,081	18	21	22	61
4	6,527	61	47	49	157
5	3,622	26	27	28	81
6	2,496	19	85	39	143
Total	**31,739**	**278**	**312**	**277**	**867**

16.1.5 Mine Development

The D&F mining method was selected to address deposit variations along strike, the low dip angle range, ore thickness variation and projected geotechnical conditions and ground support requirements and other mine operations aspects. The stopes and levels are laid out in a step room and pillar type arrangement. Sample mine cross-sections are shown in Figures 16.7 and 16.8. In addition:

- The main ramp portal would be developed in an existing rock face. The sill elevation at the portal was selected to position the portal and mine decline in rock above basal saprolite. The main decline (-15% gradient) would be driven generally parallel to and along the eastern boundary of the deposit. A probe hole would be proactively drilled at the main ramp portal site to confirm as appropriate the absence of saprolite along the ramp alignment prior to collaring the portal.
- The levels would be accessed from the main ramp and established at nominal 5 m vertical intervals. The levels would be driven with shanty-type backs to suit local mylonite, mylonite:metasediment contact conditions. The stope outlines would be finalized taking into consideration definition drilling and face mapping, sampling and assay data. The currently proposed stope outlines include the level drift shanty backs. The level drifts in the eastern portion of the deposit would be driven 4.5 m H x 4.5 m W, and 4.5 m H x 4 m W in the western portion of the deposit (not including the triangular dimensions for the shanty back) in order to control dilution when developing in ore.
- The D&F stopes would be developed between adjacent levels. The primary stopes would be developed between rib pillars and promptly tight backfilled using cemented rock fill (“CRF”). The rib pillars located between the primary stopes are to be mined as secondary stopes after the CRF has sufficiently cured. Mined out secondary stopes would be backfilled using uncemented rock fill. The stope designs were developed taking geotechnical criterion and mine operating requirements into consideration and as such the D&F method includes scaling and ground support measures designed to protect workers.
- The mine layout includes a number of mine ventilation headings. Fresh air would be drawn into the mine via the main ramp and distributed to the stoping levels. Three ventilation raises would be used to exhaust air from the stoping areas

upwards to the hangingwall drill headings. A main ventilation fan would draw air from the hangingwall drill workings and exhaust it to the atmosphere.

- The main ramp and the hangingwall workings would also be connected by an internal ramp. To provide miners with an independent escapeway, an escape raise complete with an air flow bulkhead would be driven from the hangingwall drill drift horizon to surface. The escape hoist available at the PPQ mine would be re-installed in this raise.
- A ventilation bypass drift would be driven between the East and West hangingwall drill drifts. This drift and sets of airflow control bulkheads would be used to redirect upcast contaminated airflows around active definition drilling workplaces.
- The deposit is scheduled to be mined using an orderly six phase sequence. Ore production in each phase would be achieved through level drift development in ore, and primary stoping and secondary stoping operations. Primary stope development would typically commence as level development progresses on the particular level. The secondary stopes in each phase would be mined after the completion of primary stope mining and backfilling and would be done on a retreat basis progressing back towards the main ramp.

Figure 16.7 Mine Cross-Section 8,303,862 NE

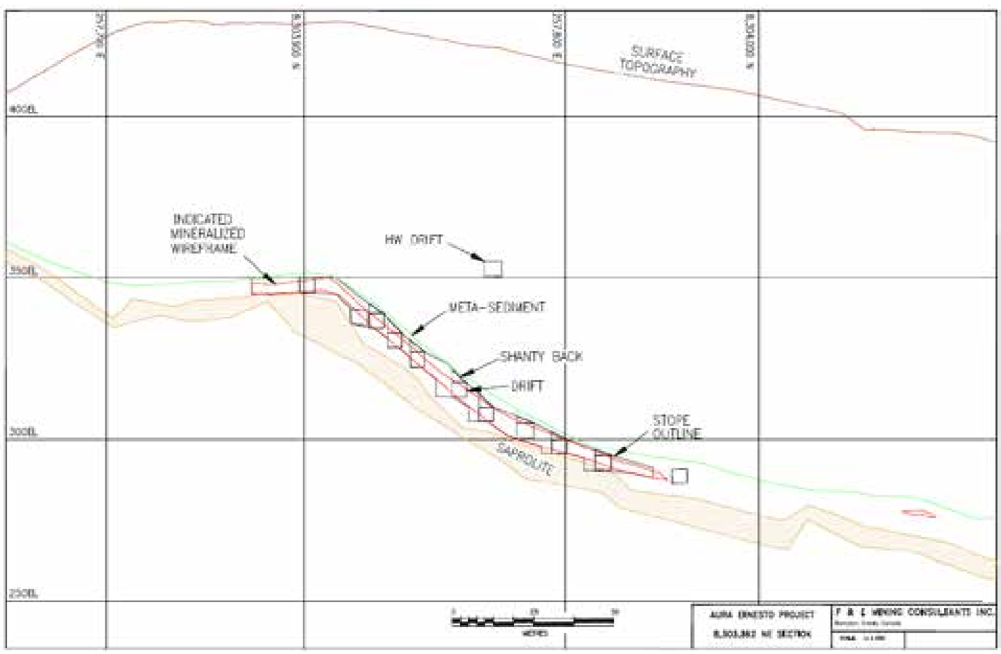
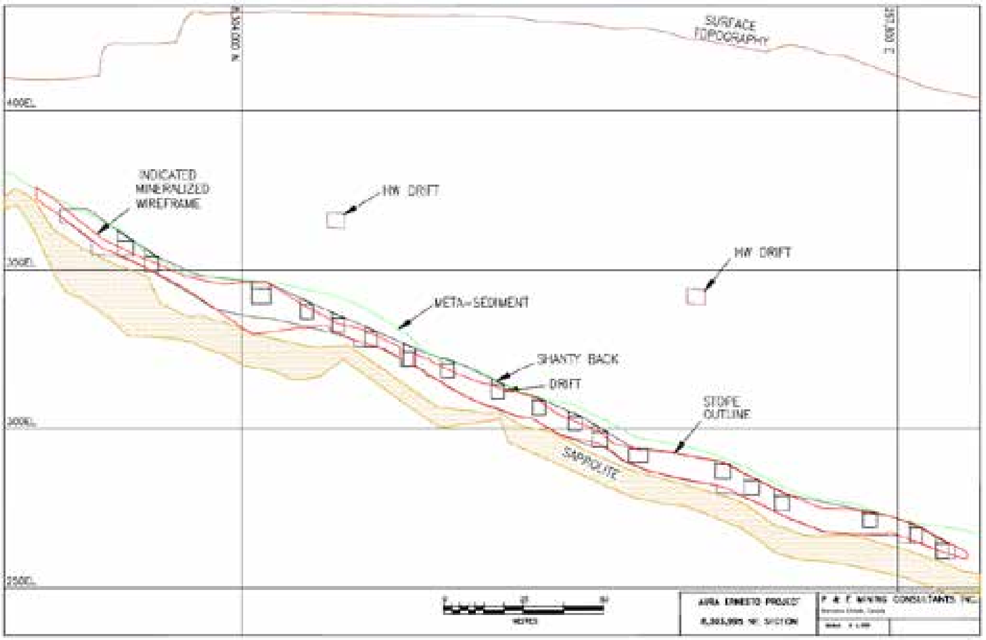


Figure 16.8 Mine Cross-Section 8,303,995 NE



The projected mine development quantities are summarized in Table 16.4 with additional details shown in Tables 16.5 and 16.6. The development work would be done using Aura equipment and personnel with the possible assistance of a mine raise contractor.

TABLE 16.4
ERNESTO MINE DEVELOPMENT SUMMARY

Item	Projected Quantities[A]	Gradient
Main Ramp and Level Development		
Main ramp boxcut excavation	7,457 t	-15%
Main ramp	780 m	-15%
Access headings developed off the main ramp	981 m	varies
Total level development in waste	318 m	Typically +1%
Total level development in ore	4,441 m	Typically +1%
Ventilation raises	175 m	
Hangingwall Development		
Pit #2 excavate rock to create portal face	7,000 t	0.5%
Pit #2 decline	119 m	-13.2%
West hangingwall definition drill ramp	353 m	-12.7% & -16.4%
East hangingwall definition drill ramp	340 m	-1.8% & -17.9%
Interconnecting ramp and ventilation cross-cuts	224 m	varies

[A] 5% allowance for the excavation of sumps, electrical cut-outs and other ancillary mine excavations is included in the estimated costs.

TABLE 16.5
ERNESTO MAIN RAMP AND LEVEL DEVELOPMENT QUANTITIES

	Main Ramp and Access Headings						Main Level Drifts					Ventilation
Rev. 13	**Main Ramp**		**Access Hdgs**		**Level Access Drifts**		**Waste Devt**	**Waste Devt**	**Ore Devt**	**Ore Devt**	**Total Level devt**	**Raises**
Elevation	**(4.5mx4.5m)**		**(4.5mx4.5m)**		**(4.5mx4.5m)**		**(4.5m H x4.5m W)**	**(4.5m H x 4.0m W)**	**(4.5m H x4.5m W)**	**(4.5m H x 4.0m W)**	**Ore & Waste**	**(3m x 3m)**
	(m)	**Grade**	**(m)**	**Grade**	**Metres**	**Grade**	**(m)**	**(m)**	**(m)**	**(m)**	**(m)**	**(m)**
												9 m (fan)
383EL												VR1=24m
368.93EL	Portal											
334.41EL	233	-15%	116	15%								
365EL												
360EL												
355EL					23	-10.2%				99	99	
350EL			101	15%					103	94	197	
345EL									0	0	0	
340EL					35	14.6%		30	0	179	209	
335EL	28	-15%	30	-9.3%			22		149	52	222	
330EL	28	-15%	26	-2.4%			35		141	168	344	VR3=21m
325EL	9	-15%	22	-11.8%			42	21	132	166	361	
320EL	25	-15%	30	-12.7%	43	-12.8%		35		246	281	
315EL	103	-15%	78	11.2%	46	14.4%		29		236	265	
310EL			32	-3.9%				14	185	131	330	
305EL	24	-15%	39	-10.4%				21	186	158	365	
300EL	52	-15%	25	-5.5%					192	78	270	
295EL	17	-15%	29	-12.8%				29	131	159	319	
290EL	43	-15%	32	-7.9%					174	0	174	VR2=44m
285EL	25	-15%	29	-12.8%					135	131	266	VR4=77m
280EL	18	-15%	50	-12.2%					102	134	236	
275EL	46	-15%	29	-14.1%					122	138	260	
270EL	71	-15%	73	1.6%						188	188	
265EL	23	-15%	42	-1.0%				16		151	168	
260EL	37	-15%						24		139	163	
255EL					50	-10.0%				41	41	
Total	**780**		**783**		**198**		**99**	**219**	**1,751**	**2,690**	**4,759**	**175**
							Level waste devt	**318**	**Level ore devt**	**4,441**		

TABLE 16.6 ERNESTO HANGINGWALL DEVELOPMENT QUANTITIES			
Item	**Dimensions**	**Gradient**	**Projected Quantities**
Pit #2 initial rock excavation		0.5%	7,000 t
Pit #2 decline	4.5 m x 4.5 m	-13.2%	119 m
West definition drill ramp	4.5 m x 4.5 m	-12.7% & -16.4%	353 m
East Drill definition ramp	4.5 m x 4.5 m	-1.8% & -17.9%	340 m
Main East-West connection	4.5 m x 4.5 m	-14.9%	111 m
Ventilation cross-cuts	4 m x 4 m to 4.5 m x 4.5 m	Typically 1%	113 m

16.1.6 Drift and Fill Mining

The D&F stopes would be developed using the sequence described in Table 16.7. The alignment and elevations of the drifts and the physical limits of the stopes would be finalized by the mine staff based on the results of the underground definition drilling program which will provide additional information on geology, gold grades, local structure, discontinuities, contacts, etc. During operations the mine geologist would also inspect the drift and stope faces and collect samples for assaying at the Ernesto mine’s laboratory.

TABLE 16.7 DRIFT AND FILL MINING CYCLE		
Item	**Primary D&F Stopes**	**Secondary D&F Stopes**
Access drifts	Access drifts would be developed across the deposit following the strike. The drifts would be spaced a nominal 5m vertical intervals. Shanty backs would be established in mylonite or along the mylonite:metasediment contact. The drifts would be developed using two boom electric hydraulic jumbos, 10 t LHD units, articulated dump trucks (ADT) and ground support installation and ancillary equipment. This equipment would also be used to mine the stopes.	
Stope dimensions	The stopes would be developed between adjacent levels using the mine development equipment. The stopes are designed to be 7 m wide but the span will be reduced when in intervals of Type III, IV or V rock.	
Stope production	The miners would advance into each stope from the upper and lower access drifts. The back of the back of the stopes would be secured using rock bolts, screen and shotcrete as required. In stopes where the single faces are sufficiently high to excavate the ore, this process would be repeated until the stope is mined out. This method allows the mine to use common equipment for development and production, and provides the mine with needed flexibility in the stopes. In stopes where the vertical height of the ore is higher than a round the ore would be extracted using multiple cuts. Depending upon the ore zone geometry, three or more stopes that connect on section would be mined in sequence. The initial cut would be made starting from the highest elevation drift, and followed by another cut developed off the next lower drift and the process repeated until the connected stopes are mined out. Using this approach miners always work under secured backs. Additional ground support would be installed in the stope walls when the height of the stope exceeds a single cut.	

TABLE 16.7
DRIFT AND FILL MINING CYCLE

Item	Primary D&F Stopes	Secondary D&F Stopes
Blasting and mucking	The stope rounds would be loaded using conventional packaged emulsion cartridges and ANFO and initiated using non-electric caps. The blasted ore would be mucked using a 10 t LHD then loaded into an ADT and hauled across the level and up the ramp to the surface crusher. In the event of poor quality rock and difficult ground conditions in a stope, it can be mined at a narrower width using the smaller LHD (e.g. a 6.7 t LHD) and track loader included in the mine equipment fleet.	
Mine services	Mine services (i.e. electrical power, compressed air, process water, ventilation ducting) would be installed in the access drifts. Portable refuge stations and a designated escapeway would be provided for use in the event of an emergency.	
Backfilling	The primary stopes would be backfilled using CRF. The mine equipment fleet includes a small bulldozer and a backfill rammer to assist in placing CRF tight to the back.	The secondary stopes would be backfilled using uncemented rock fill obtained from surface or underground waste development.
Other aspects	Fill fences would be constructed for primary stopes and for secondary stopes when there is a need to maintain access across the particular level.	Secondary stopes would be used to mine the pillars situated between the primary stopes. Secondary stope development would commence after the CRF placed in the primary stopes has sufficiently cured. Specified ground support would be installed in CRF walls exposed in secondary stopes.

Based on a review of the proposed stope limits and the projected upper surface of the basal saprolite material it is projected that the mine working would infrequently intercept minor amounts of basal saprolite. The current development program and mining method incorporates controls aimed at assisting the mine in dealing with saprolite on development sills / stope footwalls, where:

- The upper surfaces of the saprolite / altered mylonite will be revisited and confirmed as appropriate as part of the proposed definition drilling program.
- As part of the Pit # 2 portal development work, historic Pit #2 is to be dewatered and kept dry over the operating life of the mine. This is to help reduce the possibility of water entering the historic Yamana portal which had been developed into saprolite.
- Mine water intercepted on the levels in the Ernesto mine would be collected in sumps and pumped to the main sumps which would be excavated in rock off the main decline.
- Saprolite if exposed on drift and stope sills could be capped using waste rock or CRF. The mine equipment fleet includes a tracked bulldozer, a tracked loader and a backhoe which could allow miners to work in soft underfoot conditions should they occur from time to time.

16.1.7 Mine Operations

The mine would operate on a four six-hour shifts per day, six days per week basis and would not operate on Sundays or designated holidays.

16.1.7.1 Development and Production Drilling

Development and stope drift rounds would be drilled using conventional two boom electric-hydraulic jumbos.

16.1.7.2 Blasting

The development and stope rounds would be loaded with a combination of emulsion cartridges and ANFO and initiated using non-electric caps. Explosives, blasting agents and detonators would be stored in secure magazines on surface, and transported underground using blasting trucks equipped to facilitate blasthole loading. Blasting operations would be done by trained and qualified personnel.

Blasting times would be unrestricted during early development. It may become more efficient to implement a blast schedule once production is underway. In keeping with good practice, other personnel who could potentially be affected by blasting operations would be advised and access to blast sites would be prevented by guarding. Mine supervisors equipped with CO gas monitors would inspect and approve post-blast re-entry to work areas. The mine would have written health and safety procedures on what to do in the event of a misfire or other hazardous conditions.

16.1.7.3 Ground Support

Ground support would be installed in accordance with specifications developed by Knight Piesold and possible refinements to be introduced by the mine's geotechnical staff as the mine gains site-specific underground experience.

Scaling in development and stope headings would be done using scissorlifts. Bolts (i.e. typically resin rebar and swellex bolts) and welded wire mesh would be installed using Sandvik DS311 type bolting machines. Shotcrete would be mixed on surface, transported underground and transferred to a wet shotcrete machine equipped with a remote spray arm.

16.1.7.4 Mucking and Haulage

The mine's 10 t capacity LHD units would be used to muck development and stope headings, and load the nominal 28 t articulated dump trucks. These LHD units would typically muck a blasted round back to a muck bay in order to provide access to the active face for scaling and ground support installation work.
The mine would also have a 6.7 t capacity LHD for use in narrow width stopes. Stope ore would be hauled up the main ramp to the surface crusher. The mine equipment fleet also includes a tracked loader and a small backhoe to assist the mine in mucking stopes in the event of soft or slippery underfoot conditions.

The mine trucks would be used to backhaul CRF and clean waste rock from surface to the primary and secondary stopes respectively. Two trucks would be equipped with ejector boxes to facilitate the dumping of backfill into stopes.

16.1.7.5 Backfill Studies

The Ernesto underground mine plan completed by P&E relies on backfill as a ground support medium. The primary stope voids within the Ernesto underground mine are to be filled using cemented rock fill (“CRF”).

Backfill would be produced at the CRF plant to be constructed proximal to the main ramp portal. During the mine production start-up stage, and prior to the commissioning of the CRF plant, CRF would be produced by adding cement slurry to waste rock in the mine trucks.

Paterson & Cooke Canada Inc. (“P&C”) completed a Feasibility Study of the Ernesto backfill system for the EPP Project. The report was titled “Ernesto Backfill Feasibility Study, P&C Project No.: PAE-32-0164, Feasibility Design Report”, dated July 21, 2016, and was supplemented by a testwork report titled “Cemented Rock Fill Testwork Summary”, dated February 16, 2016. The executive summary of the P&C report is presented below.

P&C’s final scope of work included the following:

- Development of the backfill design criteria;
- Technical support and guidance of backfill test work completed in Brazil;
- Input into the backfill method selection;
- Specification and layouts of a surface CRF preparation plant;
- Design of truck load-out/wash-down pad;
- Design of a water supply system (pump and pipeline) to deliver water from the Ernesto mill site to the CRF mixing plant location; and
- Preparation of an operating cost estimate as well as input to Aura’s capital cost estimate.

16.1.7.5.1 Design Criteria

To maintain schedule with the mining, the following backfill capacities and strength targets were set and agreed upon:

- The nominal design production rate of the CRF plant is 45 t/h and will operate at an average utilization rate of approximately 30%.
- The original CRF target strength was 0.5 MPa after 28 days. However, based on the strengths achieved and the requirements of the mining cycle, the target cure time was reduced to 7 days.

16.1.7.5.2 Test Work Summary & Recipes

For sample logistics and project schedule reasons, the Universidade Federal de Minas Gerais (“UFMG”) completed the CRF test work for the Ernesto feasibility study in Belo Horizonte, Brazil. P&C developed the mix designs and supervised the casting of the initial UCS cylinders created using waste rock sourced from the Ernesto waste rock dump. The following summarizes the main outcomes of the test work which influenced the design process.
Two samples of Ernesto waste rock were tested:

- Type 1: Crushed and screened to minus 3” (75 mm). The majority of cylinders were cast using this sample.
- Type 2: Crushed and screened to minus 3” (75 mm) with – 5 mm size fraction removed.
- Type 1 sample contained 34% fines (passing 10 mm), considered to be higher than the acceptable limit (30%) for CRF.

Predominant minerals in Type 1 sample are Quartz, Muscovite, Feldspar. The presence of muscovite may pose challenges to mixing efficiency. Due to the large presence of fines, the w:c ratio required to obtain a product with sufficient workability was higher than initially expected.

Final CRF recipe for the feasibility study is Type 1 rock at 93%m solids, 4%m binder content (Votorantim CPIIZ-32). The recipe may need to be adjusted in the field based on the material properties of the prepared aggregate as well as workability requirements to achieve ideal compaction of the CRF within the stopes.

16.1.7.5.3 CRF Plant Site Overview

The CRF plant site is directly 260 m North East of the Ernesto process plant, located adjacent to the proposed Ernesto portal area and existing waste dump facilities. The CRF plant site topography is relatively flat, as the area has previously been levelled and graded. The location was chosen due to its central location between the waste piles, the underground access (to promote quick cycle times), and the surface infrastructure and services.

16.1.7.5.4 CRF Preparation Process Overview

P&C considered two concepts for the production of CRF: a conventional semi-mobile batch plant and a cement slurry mixing plant, using a colloidal mixer to prepare the cement slurry. The battery limits, in general, were:

- P&C – Specification of preparation plant, truck load-out/wash-down pad, and process water system.
- Aura – Preparation and delivery of the aggregate to the CRF trucks and surface infrastructure.
- P&E – Haulage and underground placement of the CRF.

Aggregate for the CRF will be sourced from the existing waste rock dumps that were generated during mining of the Ernesto open pits. When CRF is required, a surface loader will feed the waste rock through a static grizzly, to remove all particles larger than 3” (76 mm). This material falls on to a transfer conveyor, which will discharge on to a vibratory screen sourced from Aura’s São Francisco mining operation. Excess fines are removed and the finished product is stockpiled to be loaded on to the CRF trucks as required. The screening plant from São Francisco was reviewed by Aura’s engineering team and found to be suitable for producing the required aggregate particle size distribution; However, P&C has not reviewed or viewed the equipment. This is seen as a reasonable opportunity to re-use existing equipment.

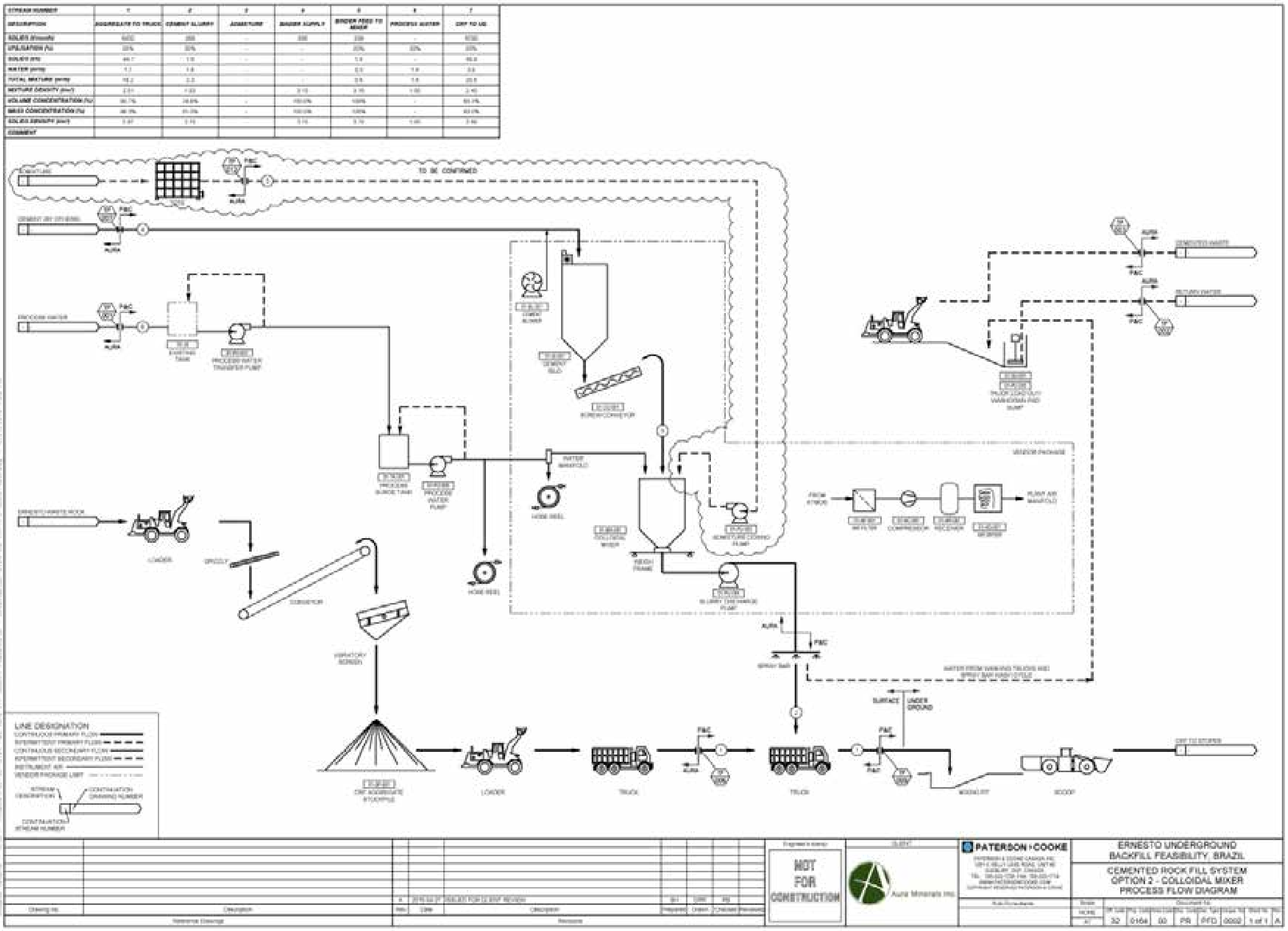
Note that subsequent to P&C completing its study it was decided to revise the aggregate preparation equipment mentioned in the P&C report. The cost to move and set up the equipment from Aura’s São Francisco operation was estimated to be higher than anticipated. A lower cost alternative was quoted by a Brazilian company Engemaxx Engenharia e Gerenciamento

(“Engemaxx”). The front-end of the system is comprised of a Simplex Globalmix (“Simplex”) SX60PSM system, which is made up of a Simplex Vibratory Feeder SXAV-0727, a Simplex Jaw Crusher SXBM-6240 and a SXPI-3012 / 3D Simplex Inclined Vibrating Screen that works in closed circuit with the jaw crusher. The Engemaxx quote provides for general infrastructure, an aggregate plant, a cement plant using a colloidal mixer, a truck wash facility, engineering and installation.

Figure 16.9 presents a process flow diagram of the backfill plant. Cement slurry is prepared by a containerized mixing plant using a colloidal mixer to provide the required mixing of the cement and water. Water is added to the mixer until the required water weight is achieved, then cement is dosed in to the mixer from a bulk silo via a screw feeder until the required cement weight is achieved. After being mixed for approximately four minutes, the slurry is pumped to a shower bar, suspended over a concrete pad sloped towards the sump designed as part of the truck load-out/wash-down pad. The slurry is sprayed over the aggregate placed in the underground CRF truck box and percolates through the aggregate during transport to the underground.

CRF trucks will also transport the CRF underground and discharge into mixing pits located off the ramp near the level accesses. Scooptrams will finish mixing the CRF in the mixing pits, then place the CRF in to the stopes. An underground dozer or rammer jammer will then compact the fill within the stope.

Figure 16.9 CRF Plant Process Flow Diagram



16.1.7.5.5 Recommendations

The following recommendations are made and should be resolved as a first priority during the detailed design phase:

Particle Size Distribution ("PSD") of the CRF Aggregate: Having consistent feed material that is within the required specification is an important consideration in ensuring that the CRF achieves the target strength and quality on a consistent basis. There is an opportunity to increase the maximum particle size to 5 inch (127 mm); however, strength test work should be completed to ensure that no loss in strength or segregation is observed.

Backfill Production Requirements and Placement Strategy: Prior to the procurement of the CRF preparation system, it will be important to confirm the plant capacity can support the requirements of the underground mine while ensuring sufficient catch-up capacity to make up for periods of system downtime. The current placement strategy requires many steps before ultimate placement in the stopes. The actual cycle time should be confirmed so that the operating cost estimate can be as accurate as possible.

QAQC Program: Although the system as summarized in this Report does not include a laboratory, a QAQC program should be put in place, using either contracted lab services or existing Aura facilities in the area, to monitor the PSD of the prepared CRF aggregate, and test for the strength of the placed CRF to ensure that excessive consumption of cement does not occur.

Cement Supply Contract: Investigate/negotiate a robust cement supply contract with a closer bulk transfer port. Backfill placement is directly linked to cement delivery. Discussions should be held with potential suppliers to determine cement delivery schedules. The cement system has been specified with one storage silo; an additional silo and screw conveyor can be procured as part of the vendor with no modifications made to the mixing system.

16.1.7.6 Backfill Operational Considerations

The primary stopes would be promptly backfilled with CRF once mined out. The CRF backfilling process is summarized in Table 16.8.

TABLE 16.8
PRIMARY STOPE CRF BACKFILLING PROCESS

Activity	Process
Stope Preparation	The backs and walls of the primary stope would have been secured during stoping operations. The workplace would be re-inspected to ensure it is safe for workers to enter. The following work would typically be carried out prior to backfilling. The workers would construct a CRF berm across the width of the stope at its base. A bulkhead would then be constructed to separate the stope from the lower drift located at the base of the stope. The purpose of the bulkhead is to contain CRF backfill within the stope and prevent spillage into the lower drift.

TABLE 16.8 PRIMARY STOPE CRF BACKFILLING PROCESS	
Activity	**Process**
	Depending upon ground conditions at a bulkhead location, and the mining access requirements, workers could alternatively choose to not construct a bulkhead and instead CRF backfill the stope and the associated section of the lower drift. If acccess needs to be re-established on the lower level, the workers could mine through the cured CRF in order to re-establish access on the lower drift.
CRF Transport	The articulated dump trucks ("ADT"s) would typically haul ore and some waste rock to surface, and backhaul CRF when returning underground. The CRF would be loaded into the trucks at the nominal 45 tph capacity CRF plant and hauled underground. The CRF would be dumped directly into a stope, or into a CRF bay where it would be reclaimed and hauled to a stope using 6.7 t or 10 t capacity LHD units. Two of the mine trucks would be equipped with ejector boxes. An ejector box allows an operator to end dump a truck into a stope without having to raise and tilt the box – this also avoids the cost of over-excavating the back at truck dumping locations.
CRF Placement	CRF would be brought to a stope through a drift located up-gradient from the fill bulkhead at the base of the stope. The CRF would be dumped in the stope by a truck or LHD. A track dozer would then be used to distribute the CRF in the stope below the CRF dump level elevation. The CRF would be tight-filled against the stope back and walls using a LHD unit equipped with a ramming boom (e.g. a rammer-jammer machine). In addition: Care would be taken to reduce CRF segregation. The mine equipment fleet includes a track loader that can also be used to transport CRF within the stope. The CRF target strength is 0.5 MPa after 7 days based on the strengths achieved in testing and mining cycle requirements. Drifts that are designated to be CRF backfilled would be backfilled using the stope CRF placement equipment.
CRF Curing	The CRF in a backfilled primary stope needs to cure for at least 7 days before an adjacent secondary stope can be developed. Based on the project schedule available CRF cure time is typically 28 days or longer.
Truck Washing Station	The truck boxes would be washed using recycled water at the high pressure hose truck washing station to be constructed on surface near the CRF plant.

Secondary stopes would be promptly backfilled with unconsolidated rock fill to be obtained from the surface screening plant and underground waste rock development. The process that would be used to backfill the secondary stopes with unconsolidated rock fill is summarized in Table 16.9.

TABLE 16.9 SECONDARY STOPE UNCONSOLIDATED ROCK FILL PROCESS	
Activity	**Process**
Stope Preparation	The backs and walls of the secondary stope would have been secured during stoping operations. The workplace would be re-inspected to ensure it is safe for workers to enter. Secondary stopes are scheduled to be backfilled on a retreat basis. As such, the construction of backfill bulkheads at the base of secondary stopes would not normally be required.
Rock Fill Transportation	The ADTs would back haul waste rock from surface to a mined-out secondary stope. The rock would be dumped directly into a stope, or into a muck bay where it would be reclaimed and hauled to a stope using 6.7 t or 10 t capacity LHD units. Two of the ADTs would be equipped with ejector boxes. Waste rock from underground development would also be used to backfill secondary stopes.
Rock Fill Placement	The waste rock would be brought to a stope through an upper drift and dumped in the stope by the truck or LHD. A track dozer would then be used to distribute the fill in the stope below the CRF dump level elevation. The fill would be pushed against the stope back and walls using a LHD unit equipped with a ramming boom (e.g. a rammer-jammer machine).
Other	Controls such as water sprays would be used to control dust levels during rock backfill operations.

16.1.8 Mine Equipment

The proposed mine equipment fleet is shown in Table 16.10.

TABLE 16.10 MINE EQUIPMENT FLEET						
Item	**Source**	**Make, Model**	**Number in mine equipment fleet**			
			Year 1	**Year 2**	**Year 3**	**Year 4**
2 boom electric hydraulic jumbo	ProcuredA	Atlas Copco Boomer 282	1-3	3	3	3
10 t LHD unit	1 Procured. 1 PPQ mine.	Sandvik LH410	1-3	3	3	3
6.7 t LHD unit	Procured	Sandvik LH307	1	1	1	1
28 t articulated dump truck	Procured	Volvo Articulated Truck A30FB	1-2	3	3	3
Rockbolter	PPQ mine	Sandvik DS311	1-2	2	2	2
Blast truck	PPQ mine	Dux	1-2	2	2	2
Scissorlift – scaling	PPQ mine	Dux	1-2	2	2	2
Scissorlift – mine services	Procured	Dux	1-2	2	2	2
Shotcrete transmixer	Procured	Source locally	1-2	2	2	2
Shotcrete machine	PPQ mine	Dux	1-2	2	2	2
Bulldozer	Procured	Caterpillar D5	1	1	1	1
Tracked loader	Procured	Source locally	1	1	1	1
Backhoe	Procured	Source locally	1	1	1	1
Backfill rammer	Procured	Source locally	1	1	1	1
Nipping flatbed truck	Procured	Source locally	1	1	1	1

TABLE 16.10 MINE EQUIPMENT FLEET						
Item	**Source**	**Make, Model**	**Number in mine equipment fleet**			
			Year 1	**Year 2**	**Year 3**	**Year 4**
Mine grader	Procured	Source locally	1	1	1	1
Fuel and lube truck	Procured	Source locally	1	1	1	1
Personnel carrier	Procured	Source locally	1-2	2	2	2
Supervisor & maintenance vehicles	Procured	Source locally	5	5	5	5
Jacklegs and stopers	Procured	Source locally	6	6	6	6
Refuge stations	1 PPQ mine. 1 Procured.	Source locally	2	2	2	2
See capital cost and cashflow for procurement (i.e. purchase, rental, lease-to-own) details. Two injector boxes would be sourced locally in Year 1.						

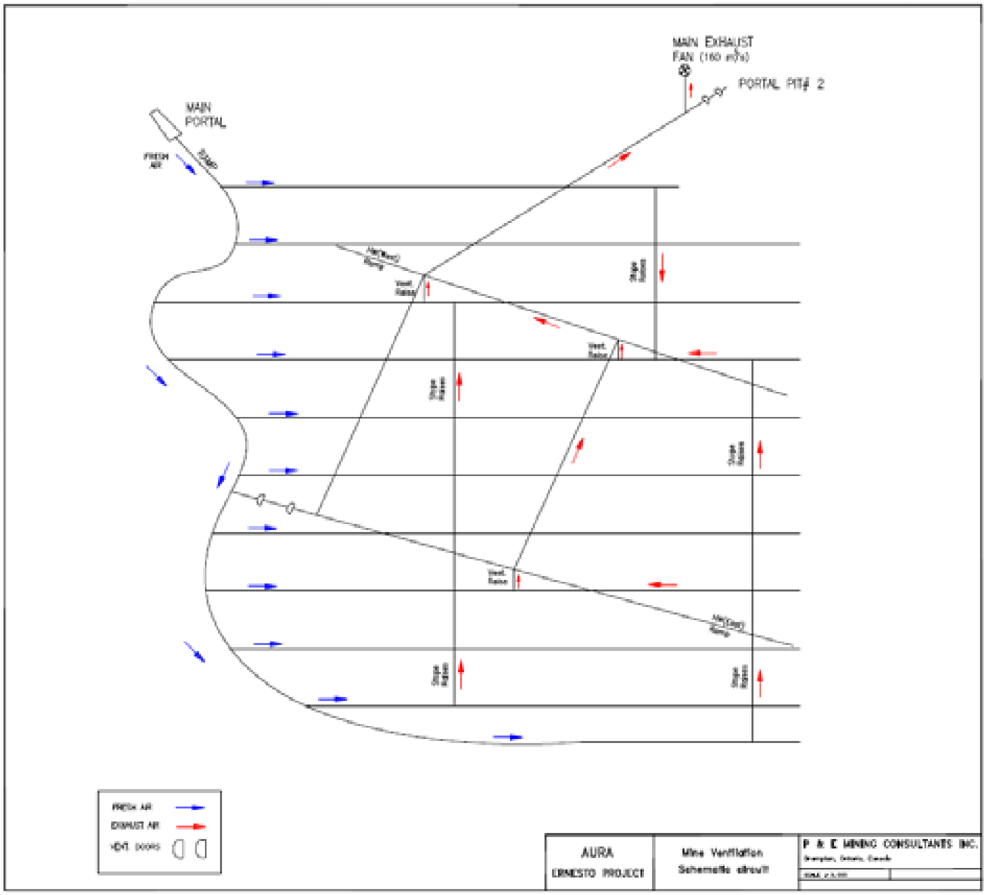
16.1.9 Mine Services

16.1.9.1 Mine Ventilation

The mine ventilation system would provide fresh air to the mine workings and help maintain a safe working environment. The main ventilation fans in place at surface at Pau-a-Pique would be moved and installed at Ernesto as they become available. The Ernesto ventilation requirements were estimated for the initial development and mine production stages taking regulatory requirements under the Brazilian Mining Regulatory Standard – NRM 06, a VentSim program based estimate, worker health and safety aspects, and mining sector ventilation practices into consideration.

In the proposed mine ventilation network (Figure 16.10) fresh air would be drawn down the main ramp and distributed to the levels and stopes. Air movement would occur between levels via mine openings developed between levels within stope outlines. Air would be exhausted upwards from the levels and stopes to the hangingwall workings via three nominal 3m x 3 m ventilation raises. A series of vent doors and a ventilation bypass heading would be used to divert contaminated air around active definition drilling workplaces. The main exhaust fan would be located on surface and at the collar of a raise extending from the hangingwall access decline to surface. Ventilation control doors would also be installed at the Pit #2 portal, in the access ramp that extends from the main ramp to the East hangingwall drift, and at the base of the escape raise extending from the hangingwall horizon to surface.

Figure 16.10 Mine Ventilation Circuit Schematic



During the initial development stage, fans installed at the Pit #2 and main ramp portals would provide fresh air via flexible ducting to the underground development workplaces. The permanent exhaust fan would be installed after a ventilation connection is established between the hangingwall and main ramp workings. The projected ventilation requirements and connected primary, secondary and auxiliary ventilation fan loads are shown in Table 16.11.

TABLE 16.11
PROJECTED MINE VENTILATION REQUIREMENTS AND CONNECTED LOADS

Stage	Initial Mine Development	During the transition from development to Phase 1 production	Mine Production
Mine ventilation	80 m^3/s in total. (40 m^3/s at each portal)	80-160 m^3/s.	160 m^3/s.
Connect Fan Loads			
Primary fan(s)	150 kW in total. (75 kW at each portal)	225 kW if one axial vane exhaust fan is used. The use of two parallel axial vane fans would facilitate maintenance and help reduce power costsA.	
Secondary fans	56 kW	112 kW	224 kW
Auxiliary fans	56 kW	112 kW	300 kW
Total connect fan load	262 kW	449 kW	749 kW
Estimated draw on working days	262 kW	449 kW	599 kW
At least one third of the main ventilation system would need to be operated when the mine is not active such as on Sundays and on National holidays.			

16.1.9.2 Mine Water Supply

During the initial mine pre-production stage the mine would obtain its process water from Pit #2, and from the mill. The mine water pond to be located on surface in the main ramp portal area is scheduled to be constructed as part of the pre-production work. Once constructed, this pond would allow the mine to recycle process water. Bottled potable water would be procured locally.

16.1.9.3 Mine Dewatering

The groundwater inflow rates during each mine development phase were estimated taking precipitation during the dry and rainy seasons, three hydraulic conductivity estimates, and the mine plan and schedule into consideration. The geometric mean and maximum groundwater inflow rates are shown in Tables 16.12 and 16.13, respectively. The inflow rates for the final three stoping phases take ramp pillar recovery into account.

The groundwater inflow would be collected in sumps and pumped to the surface mine water pond. The mine would use Metso submersible pumps to be obtained from the PPQ mine and procure additional larger capacity pumps to increase the mine dewatering capacity to suit the projected maximum inflows. The Metso pumps are capable of 115 m^3/hr, can pump a total head of approximately 91 m, and draw 100 hp. Pit #2 would be kept dewatered with the water pumped to the surface mine water pond located near the main ramp portal. A separate pump would be used to pump excess water from the mine water pond to the mill. The mine would use process water recycled from the mine water pond.

TABLE 16.12 PROJECTED GEOMETRIC MEAN GROUNDWATER INFLOW RATES				
Phase	**Dry Season**		**Rainy Season**	
	7 Months/year		**5 Months/year**	
	m^3/hr	**USgpm**	**m^3/hr**	**USgpm**
1	4	18	8	35
2	10	44	14	62
3	11	48	15	66
4	11	48	15	66
5	11	48	15	66
6	11	48	15	66

TABLE 16.13 PROJECTED MAXIMUM GROUNDWATER INFLOW RATES				
Phase	**Dry Season**		**Rainy Season**	
	7 Months/year		**5 Months/year**	
	Maximum Inflow		**Maximum Inflow**	
	m^3/hr	**USgpm**	**m^3/hr**	**USgpm**
1	25	110	48	211
2	62	273	84	370
3	72	317	94	414
4	62	273	84	370
5	62	273	84	370
6	62	273	84	370

16.1.9.4 Electrical Power Distribution

The projected connected electrical loads are shown in Table 16.14. Electrical power would be obtained from the existing main substation at the Ernesto mill. A pole line would be constructed from the power disconnects at the mill to the main ramp and Pit #2 portal areas. The substations to be installed at the portals and in the mine would be obtained from the PPQ mine. The capital cost estimate allows for the replacement of two dry transformers with capacity 750 KVA/ 60 Hz / 13.8Kv / 440 V.

TABLE 16.14 PROJECTED MINE POWER REQUIREMENTS				
Item	**Load (kW)**			
	Year 1 (months 3 -12)[A]	**Year 2**	**Year 3**	**Year 4**
Projected Connected Load				
Mobile equipment	411 - 762	762	762	762
Pumps	70 - 570	570	570	570
Air compressor	200	200	200	200
Office and shop	400	400	400	400
Ventilation fans	262 - 749	749	749	749
Other loads	150	150	150	150
Projected Connected Load	1,493 - 2,831	2,831	2,831	2,831

TABLE 16.14 PROJECTED MINE POWER REQUIREMENTS				
Item	**Load (kW)**			
	Year 1 (months 3 -12)[A]	**Year 2**	**Year 3**	**Year 4**
Projected Utilized Load on work days	870 – 1,648	1,648	1,648	1,648
Projected Utilized Load on non-work days	140 - 514	514	514	514
[A]. The mine would use a diesel-powered portable generator and portable air compressor in months 1 and 2.				

16.1.9.5 Compressed Air Plant

The mine compressor plant would be obtained from the PPQ mine. It consists of two Atlas Copco compressors, each rated at 782 cfm with 132 Kw motors.

16.1.9.6 Mine Communications System

The mine would be equipped with a phone system connecting the stoping area, office, shop, CRF plant, escape raise and refuge stations.

16.1.9.7 Portable Refuge Stations

Two portable refuge stations would be provided for use in the event of a fire or other emergency. The refuge stations would be located underground in areas where there is a possibility that persons may not be able to reach a mine exit in a reasonably short time. The refuge stations would be located in a fresh air circuit and away from hazard areas such as a fuel storage area. The mine personnel would be made familiar with the location of the stations and relevant procedures including the mine's emergency preparedness and response plan. The estimated costs include mine escapeway and refuge station location signage costs.

16.1.9.8 Mine Maintenance Facilities

The surface shop would include a roof-covered concrete working floor and storage container walls and be constructed midway between the proposed main ramp portal and existing ERN office complex. The mine equipment would be serviced at the surface shop. Underground equipment would be refuelled at the diesel fuel tank on surface or underground using the fuel - lube truck.

16.1.9.9 Mine office and dry

The existing Ernesto mine office would be used. The existing change facilities would be expanded to create the mine dry.

16.1.10 Mine Personnel

The total mine workforce over the life of mine is shown in Table 16.15. Details are provided in Tables 16.16 to 16.19.

TABLE 16.15 **PROJECTED TOTAL MINE LABOUR FORCE**				
Area	**No. of personnel on the mine payroll**			
	Year 1	**Year 2**	**Year 3**	**Year 4**
Mine Management and Supervision	5-8	8	8	8
Mine Operations	10-128	128	128	128
Mine Maintenance	13-32	32	32	32
Technical Services	9	16	16	15
Total	**37-177**	**184**	**184**	**183**

TABLE 16.16 **MINE MANAGEMENT AND SUPERVISION**				
Position	**No. of personnel on the mine payroll**			
	Year 1	**Year 2**	**Year 3**	**Year 4**
Mine Manager	1	1	1	1
Mine Captain	1	1	1	1
Analyst Control	1	1	1	1
Mine Shift Supervisors	1-4	4	4	4
Mine Safety Coordinator	1	1	1	1
Total	**5-8**	**8**	**8**	**8**

TABLE 16.17 **MINE OPERATIONS LABOUR**				
Position	**No. of personnel on the mine payroll**			
	Year 1	**Year 2**	**Year 3**	**Year 4**
Drill Jumbo Operator	1-12	12	12	12
Blasters	2-16	16	16	16
LHD Operator	1-16	16	16	16
Truck Driver	1-16	16	16	16
Scaler	1-16	16	16	16
Bolting Machine Operator	1-8	8	8	8
Shotcrete Transmixer Operator	1-8	8	8	8
Shotcreter	1-8	8	8	8
Auxiliary Operator – Development	1-8	8	8	8
Auxiliary Operator – Stoping	0-16	16	16	16
Backfill Pusher	0-4	4	4	4
Total	**10-128**	**128**	**128**	**128**

TABLE 16.18 MINE MAINTENANCE LABOUR				
Position	**No. of personnel on the mine payroll**			
	Year 1	**Year 2**	**Year 3**	**Year 4**
Mine Maintenance Supervisor	0-1	1	1	1
Maintenance Planner	0-1	1	1	1
Maintenance Advisor – Drill Jumbo	1	1	1	1
Lead Mechanic	1-4	4	4	4
Mine Mechanic	2-4	4	4	4
Mechancial Technician	4-6	6	6	6
Lead Electrician	1-4	4	4	4
Electrical Technician - Intermediate	2-4	4	4	4
Electrical Technician – Junior	2-4	4	4	4
Welder	0-1	1	1	1
Lubricator	0-1	1	1	1
Bit Sharpener	0-1	1	1	1
Total	**13-32**	**32**	**32**	**32**

TABLE 16.19 TECHNICAL SERVICES LABOUR				
Position	**No. of personnel on the mine payroll**			
	Year 1	**Year 2**	**Year 3**	**Year 4**
Supervisor (mining engineer)	1	1	1	1
Stope planning engineer		1	1	1
Mine technician		1	1	1
Blasting specialist	1	1	1	1
Rock mechanics engineer	1	1	1	1
Rock mechanics technician		1	1	1
Mine geologist	1	1	1	1
Definition drilling geologist	1	1	1	
Geological technician / samplers	2	5	5	5
Surveyor	1	1	1	1
Survey assistants	1	2	2	2
Total	**9**	**16**	**16**	**15**

16.1.11 Mine Development and Production Schedules

16.1.11.1 Mine Development Schedule

As shown in Figure 16.11 the mine would be developed and operated over a four year time line. Ore production is scheduled to be done in six phases referred to as Phases 1 to 6. The phase sequence was developed taking the projected number of ounces of gold available in each phase and other mine planning aspects into consideration. Ore production in each phase would be comprised of level development in ore; primary stoping with CRF backfill; and secondary stoping with uncemented rock backfill. The level development corresponding to each phase is shown in Table 16.20.

Figure 16.11 Mine Development Schedule

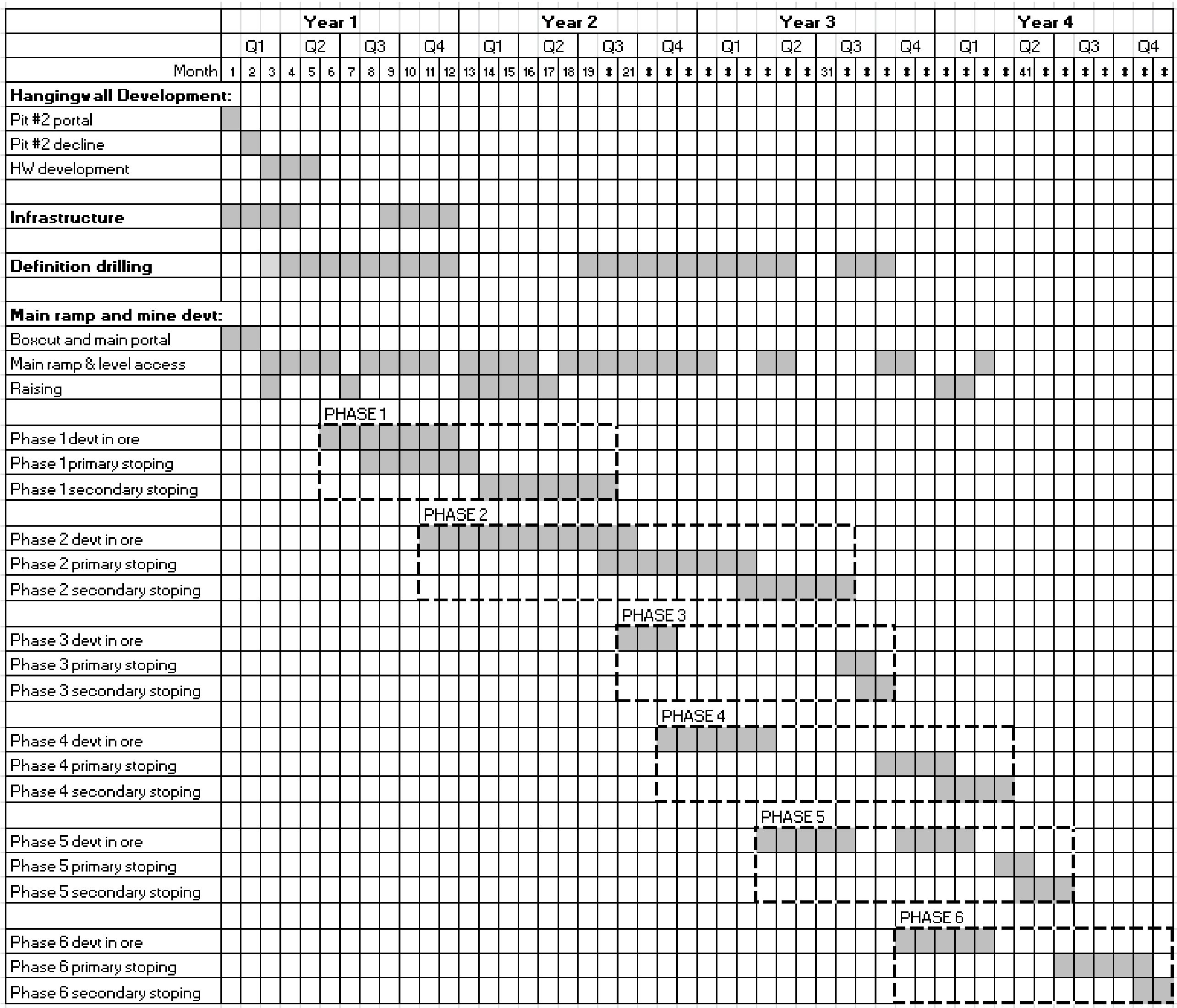


TABLE 16.20 LEVELS DEVELOPED IN EACH PHASE						
	Level development completed in each Phase					
Levels	**Phase 1**	**Phase 2**	**Phase 3**	**Phase 4**	**Phase 5**	**Phase 6**
355						
350						
340						
335						
330						
325						
320						
315						
310						
305						
300						
295						
290						
285						
280						
275						
270						
265						
260						
255						

The mine development and production schedules were developed taking work activities, estimated quantities, estimated performances, an average of 26 working days per month, and other mine planning aspects into consideration. Development in waste and ore and primary and secondary stope production were scheduled taking the work cycles and activities (i.e. drilling, blasting, mucking, ground support, services), number of equipment and equipment performance, maximum of six underground working hours per worker per day, expected delays, expected conditions, health and safety and other factors into consideration. The mine development and production schedule was developed based on the performance / production rates shown in Table 16.21. The mine would commence the initial development work with one two boom electric hydraulic jumbo. Two additional jumbos would be added to the fleet when additional faces become available in the hangingwall headings and the main ramp development commences.

TABLE 16.21 PROJECTED PERFORMANCE / PRODUCTION RATES		
Activity	**No. of development headings**	**Performance / Production Rate**
Main ramp development	Single heading Double heading	110 m / month or 4.3 m/day. 160 m / month or 6.2 m/day.
Shanty back level development	Multiple heading	150 m / month or 5.8 m/day.
Primary stoping		250 tpd per jumbo.
Secondary stoping		300 tpd per jumbo.

16.1.11.2 Mine Production Schedule

The mine production schedule is shown in Table 16.22.

TABLE 16.22
MINE PRODUCTION SCHEDULE

Month	Development Ore		Primary Ore		Secondary Ore		Total Ore Production		Average ore grade
	(t/month)	(g / t Au)	(t/month)	(g / t Au)	(t/month)	(g / t Au)	(t/month)	(t/day)	(g / t Au)
6	8,153	4.83	-		-		8,153	314	4.83
7	16,076	4.97	-		-		16,076	618	4.97
8	11,538	5.07	4,250	4.13	-		15,788	607	4.82
9	7,488	5.27	7,500	4.13	-		14,988	576	4.70
10	5,999	4.74	11,125	4.13	-		17,124	659	4.34
11	5,371	4.23	11,500	4.13	-		16,871	649	4.16
12	10,421	4.05	10,500	4.13	-		20,921	805	4.09
13	6,899	3.98	8,298	4.13	-		15,197	585	4.07
14	12,445	4.77	-		9,300	3.94	21,745	836	4.41
15	11,704	5.66	-		9,450	3.94	21,154	814	4.89
16	15,273	6.32	-		8,700	3.94	23,973	922	5.45
17	11,105	6.39	-		4,350	3.94	15,455	594	5.70
18	10,963	6.56	-		6,000	3.94	16,963	652	5.63
19	10,667	6.45	-		11,700	3.94	22,367	860	5.13
20	8,805	6.47	7,875	5.96	6,332	3.94	23,012	885	5.60
21	9,138	5.94	12,125	5.96	-		21,263	818	5.95
22	7,715	5.83	10,785	5.96	-		18,500	712	5.91
23	3,532	4.07	14,125	5.96	-		17,657	679	5.58
24	11,251	5.30	10,375	5.96	-		21,626	832	5.62
25	15,181	5.25	9,750	5.96	-		24,931	959	5.53
26	15,181	5.25	9,750	5.96	-		24,931	959	5.53
27	13,552	5.30	4,537	5.96	7,050	5.67	25,139	967	5.52
28	6,984	5.83	-		15,600	5.67	22,584	869	5.72
29	1,838	6.38	-		15,600	5.67	17,438	671	5.75
30	4,344	6.38	-		17,400	5.67	21,744	836	5.82
31	4,344	6.38	-		19,500	5.67	23,844	917	5.80
32	1,671	6.38	11,500	5.47	8,138	5.67	21,309	820	5.62
33	-		9,403	5.47	12,000	5.21	21,403	823	5.32
34	-		11,250	4.83	9,948	5.21	21,198	815	5.01
35	5,461	5.88	16,250	4.83	-		21,711	835	5.09
36	9,895	5.63	14,000	4.83	-		23,895	919	5.16
37	8,443	5.37	5,453	4.83	10,200	4.60	24,096	927	4.92
38	3,708	6.02	-		17,250	4.60	20,958	806	4.85
39	3,759	5.29	-		19,500	4.60	23,259	895	4.71
40	-		17,100	5.56	2,351	4.60	19,451	748	5.44
41	-		9,806	5.56	3,900	5.30	13,706	527	5.49
42	-		-		23,400	5.30	23,400	900	5.30
43	-		18,500	4.28	951	5.30	19,451	748	4.32
44	-		19,500	4.28	-		19,500	750	4.28
45	-		19,500	4.28	-		19,500	750	4.28
46	-		19,500	4.28	-		19,500	750	4.28
47	-		7,572	4.28	16,874	2.89	24,446	940	3.32
48	-		-		21,600	2.89	21,600	900	2.89
Total	**278,904**	**5.48**	**311,829**	**4.95**	**277,094**	**4.67**	**867,827**		**5.03**

16.1.12 Pre-production Development

The pre-production development works would be completed over a six month time line where the key work activities include:

- The development of the Pit #2 portal, the hangingwall decline and the hangingwall definition drill headings required in the near term.
- The development of the main ramp portal and associated mine infrastructure including but not limited to the surface shop, power distribution, mine water pond, mine dry, etc. The main ramp would be driven down to the Phase 1 stoping levels (i.e. the 325, 330, 335 & 340 m levels).
- Initial Phase 1 level development in ore.

16.1.12.1 Phase 1

Phase 1 ore production would be derived from:

- Development in ore on the 325, 330, 335 and 340 levels.
- Primary stopes developed between the 325 – 330, 330-335, and 335-340 levels.
- Secondary stopes developed between these same levels.

Phase 1 ore production is shown in Table 16.23 where the grade of ore in stopes is the average grade of ore between the upper and lower levels that define the top and bottom of the stopes. The ore tonnage and grade estimates include a 5% ore loss factor. The secondary stope ore tonnage and grade estimates include 5% dilution at zero grade for expected CRF and footwall material dilution. Dilution from material mined within stope outlines in backs and sills is also accounted for.

TABLE 16.23
PHASE 1 ORE TONNAGES AND GRADES

Phase 1	Ore	Grade
Total ore production:	(t)	(g/t Au)
Development in ore	57,847	4.88
Primary stopes	53,173	4.13
Secondary stopes	55,832	3.94
Total Phase 1 ore	166,853	4.33

Phase 1	Dev't Ore	Grade
Development in ore:	(t)	(g/t Au)
340 level	12,703	4.26
335 level	11,106	4.42
330 level	19,190	4.87
325 level	14,848	5.78
Total development in ore	57,847	4.88

Phase 1	Stope Ore	Grade
Stope production:	(t)	(g/t Au)
335 level	31,671	4.07
330 level	63,098	4.16
325 level	30,948	4.16
Subtotal	125,717	4.13
Less ramp pillar	19,370	4.13
Subtotal	106,347	4.13
Primary stopes	53,173	4.13
Secondary stopes	55,832	3.94

16.1.12.2 Phase 2

Phase 2 ore production is shown in Table 16.24 where the grade of ore in stopes is the estimated average grade of ore between the specific two levels. The ore tonnage and grade estimates include 5% ore loss, and 5% dilution in secondary stopes. Dilution from material mined within stope outlines in backs and sills is also accounted for.

TABLE 16.24
PHASE 2 ORE TONNAGES AND GRADES

Phase 2	Ore	Grade
Total ore production:	**(t)**	**(g/t Au)**
Development in ore	96,483	5.76
Primary stopes	79,322	5.96
Secondary stopes	83,288	5.67
Total Phase 2 ore	259,093	5.79

Phase 2	Dev't Ore	Grade
Development in ore:	**(t)**	**(g/t Au)**
295EL	19,284	3.94
290EL	12,208	5.27
285EL	18,360	5.91
280EL	16,637	6.86
275EL	17,591	6.35
270EL	12,403	6.54
Total development in ore	96,483	5.76

Phase 2	Stope Ore	Grade
Stope production:	**(t)**	**(g/t Au)**
295EL	0	
290EL	28,027	5.11
285EL	52,232	5.30
280EL	31,889	7.28
275EL	38,422	6.40
270EL	36,634	5.93
Subtotal	187,204	5.96
Less ramp pillar	28,560	5.96
Subtotal	158,644	5.96
Primary stopes	79,322	5.96
Secondary stopes	83,288	5.67

16.1.12.3 Phase 3

Phase 3 ore production is shown in Table 16.25 where the grade of ore in stopes is the average grade of ore between the upper and lower levels that define the top and bottom of the stopes. The ore tonnage and grade estimates include 5% ore loss, and 5% dilution in secondary stopes. Dilution from material mined within stope outlines in backs and sills is also accounted for.

TABLE 16.25
PHASE 3 ORE TONNAGES AND GRADES

Phase 3	Ore	Grade
Total ore production:	**(t)**	**(g/t Au)**
Development in ore	18,007	5.50
Primary stopes	20,903	5.47
Secondary stopes	21,948	5.21
Total Phase 3 ore	60,859	5.38

Phase 3	Dev't Ore	Grade
Development in ore:	**(t)**	**(g/t Au)**
265EL	9,533	6.55
260EL	6,540	4.83
255EL	1,934	2.64
Total development in ore	18,007	5.50

Phase 3	Stope Ore	Grade
Stope production:	**(t)**	**(g/t Au)**
265EL	15,786	6.11
260EL	19,120	5.89
255EL	6,900	2.82
Subtotal	41,806	5.47
Primary stopes	20,903	5.47
Secondary stopes	21,948	5.21

16.1.12.4 Phase 4

Phase 4 ore production is shown in Table 16.26 where the grade of ore in stopes is the average grade of ore between the upper and lower levels that define the top and bottom of the stopes. The ore tonnage and grade estimates include 5% ore loss, and 5% dilution in secondary stopes. Dilution from material mined within stope outlines in backs and sills is also accounted for.

TABLE 16.26
PHASE 4 ORE TONNAGES AND GRADES

Phase 4	Ore	Grade
Total ore production:	**(t)**	**(g/t Au)**
Development in ore	60,932	5.30
Primary stopes	46,953	4.83
Secondary stopes	49,301	4.60
Total Phase 4 ore	157,186	4.94

Phase 4	Dev't Ore	Grade
Development in ore:	**(t)**	**(g/t Au)**
310EL	20,420	5.53
305EL	22,211	5.58
300EL	18,301	4.71
Total development in ore	60,932	5.30

Phase 4	Stope Ore	Grade
Stope production:	**(t)**	**(g/t Au)**
310EL	-	-
305EL	19,437	6.10
300EL	36,648	4.63
295EL	37,821	4.37
Subtotal	93,906	4.83
Primary stopes	46,953	4.83
Secondary stopes	49,301	4.60

16.1.12.5 Phase 5

Phase 5 ore production is shown in Table 16.27 where the grade of ore in stopes is the average grade of ore between the upper and lower levels that define the top and bottom of the stopes. The ore tonnage and grade estimates include 5% ore loss, and 5% dilution in secondary stopes. Dilution from material mined within stope outlines in backs and sills is also accounted for.

TABLE 16.27
PHASE 5 ORE TONNAGES AND GRADES

Phase 5	Ore	Grade
Total ore production:	**(t)**	**(g/t Au)**
Development in ore	25,987	6.73
Primary stopes	26,906	5.56
Secondary stopes	28,251	5.30
Total Phase 5 ore	81,144	5.84

Phase 5	Dev't Ore	Grade
Development in ore:	**(t)**	**(g/t Au)**
320EL	11,618	7.16
315EL	14,369	6.38
Total development in ore	25,987	6.73

Phase 5	Stope Ore	Grade
Stope production:	**(t)**	**(g/t Au)**
320EL	22,299	4.37
315EL	15,866	4.89
310EL	15,647	7.94
Subtotal	53,812	5.56
Primary stopes	26,906	5.56
Secondary stopes	28,251	5.30

16.1.12.6 Phase 6

Phase 6 ore production is shown in Table 16.28 where the grade of ore in stopes is the average grade of ore between the upper and lower levels that define the top and bottom of the stopes. The ore tonnage and grade estimates include 5% ore loss, and 5% dilution in secondary stopes. Dilution from material mined within stope outlines in backs and sills is also accounted for.

TABLE 16.28
PHASE 6 ORE TONNAGES AND GRADES

Phase 6	Ore	Grade
Total ore production:	**(t)**	**(g/t Au)**
Development in ore	19,648	4.69
Primary stopes	84,572	4.27
Secondary stopes	38,474	2.89
Total Phase 6 ore	142,694	3.96

Phase 6	Dev't Ore	Grade
Development in ore:	**(t)**	**(g/t Au)**
355EL	6,014	5.29
350EL	13,634	4.43
Total development in ore	19,648	4.69

Phase 6	Stope Ore	Grade	
Stope production:	**(t)**	**(g/t Au)**	
355EL	18,762	2.50	
350EL	11,047	4.35	
340EL	43,475	2.93	
Subtotal	73,285	3.04	
Ramp Pillar - Phase 1	19,370	4.13	Primary & pillar weighted average 84,572 t @ 4.27 g/t Au
Ramp Pillar - Phase 2	28,560	5.96	
Primary =	36,642	3.04	
Secondary =	38,474	2.89	

16.2 LAVRINHA

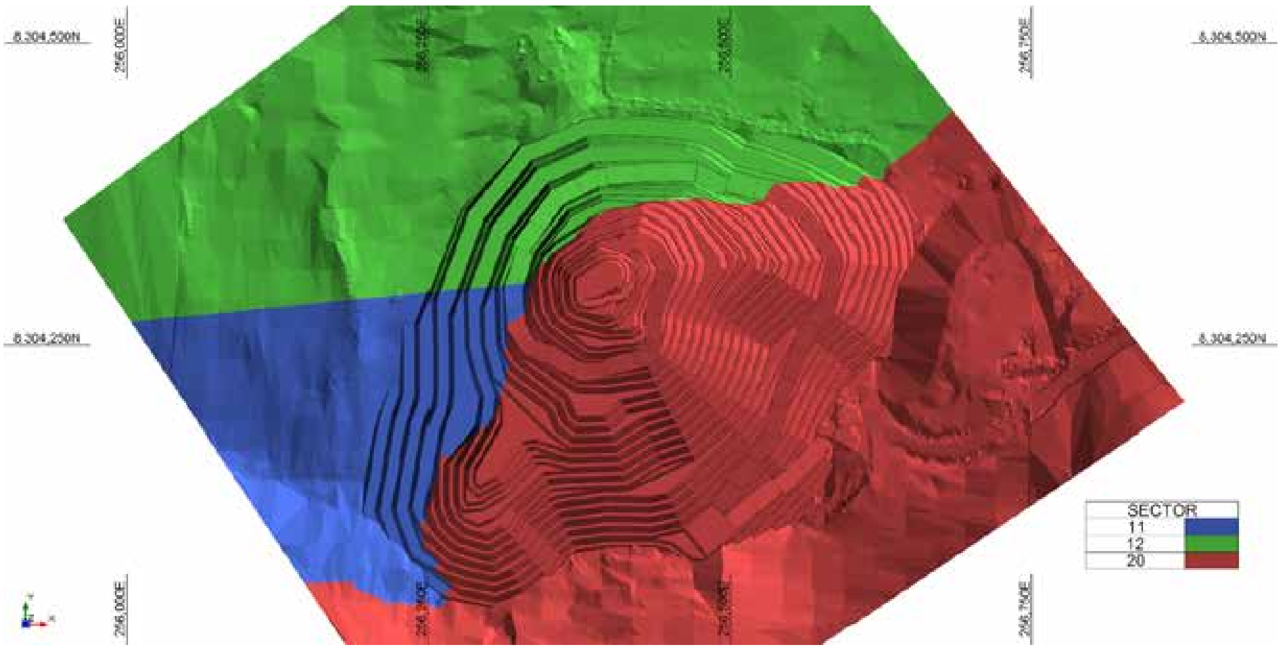
16.2.1 Ground Conditions and Slope Stability

16.2.1.1 Overall Slope Angles

MCB contracted the firm WALM Engenharia E Tecnologia Ambienta to develop a geotechnical investigation and pit slope conditions study. The study was carried out in November/December, 2015, and a geotechnical report in Portuguese (WBH 87-15-MCB1-RTE-0001) was issued. The final pit walls were divided into three main sectors and the slope angles, varying from 31° to 42°, were used in the pit optimization, as presented in Table 16.29. The sectors are presented in plan view in Figure 16.12.

TABLE 16.29 INTER-RAMP SLOPE ANGLES BY SECTOR	
Sector	Inter-Ramp Angle (degrees)
11	37.50
12	42.00
20	31.00

Figure 16.12 Plan View of Geotechnical Sectors



16.2.2 Life of Mine Plan

16.2.2.1 Mining Phases Design

MCB designed a set of four mining stages (phases) for the Lavrinha open pit. NPV Scheduler® software has been used for pushback definition. Table 16.30 and Figures 16.13 to 16.19 show the stage designs.

TABLE 16.30 OPEN PIT STAGE SUMMARY						
Stage	Ore			Waste		Total Mined (t)
	Tonnes (t)	Au (g/t)	Au (oz)	Tonnes (t)	S.R (w/o)	
1	170,334	1.36	7,456	842,749	4.95	1,013,083
2	229,939	1.57	11,596	2,222,907	9.67	2,452,847
3	315,831	1.67	16,989	4,316,219	13.67	4,632,050
4	394,096	1.91	24,256	6,623,294	16.81	7,017,389
Total	**1,110,200**	**1.69**	**60,297**	**14,005,170**	**12.61**	**15,115,369**

Figure 16.13 Mining Stages – 3D View

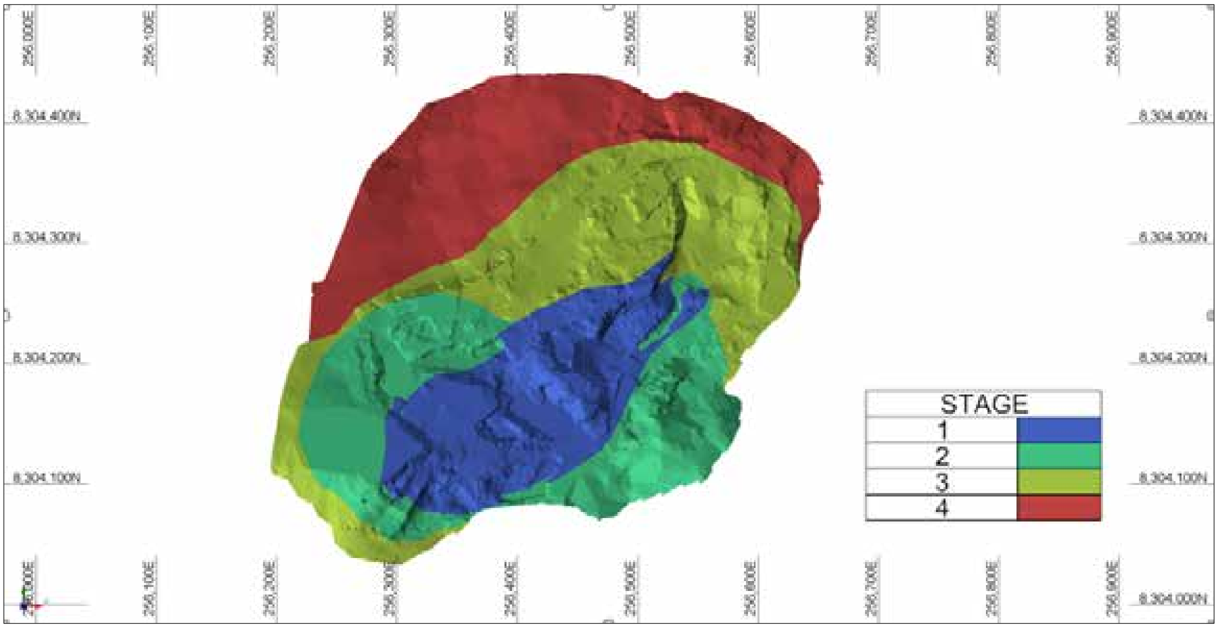


Figure 16.14 Mining Stages, Plan Views

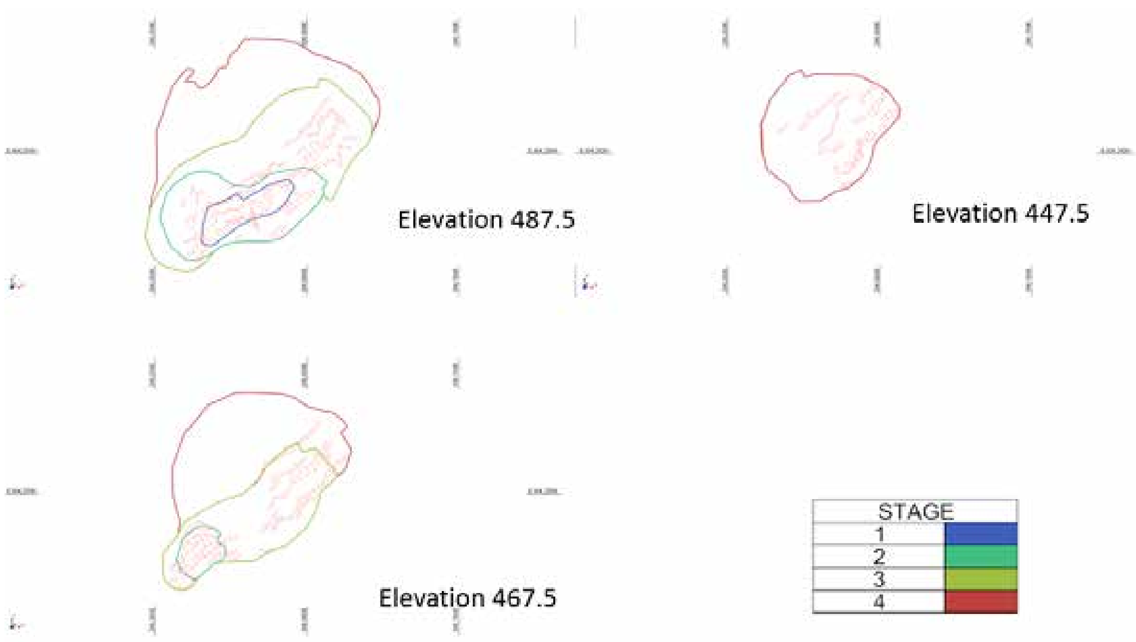


Figure 16.15 Mining Stages, Section View

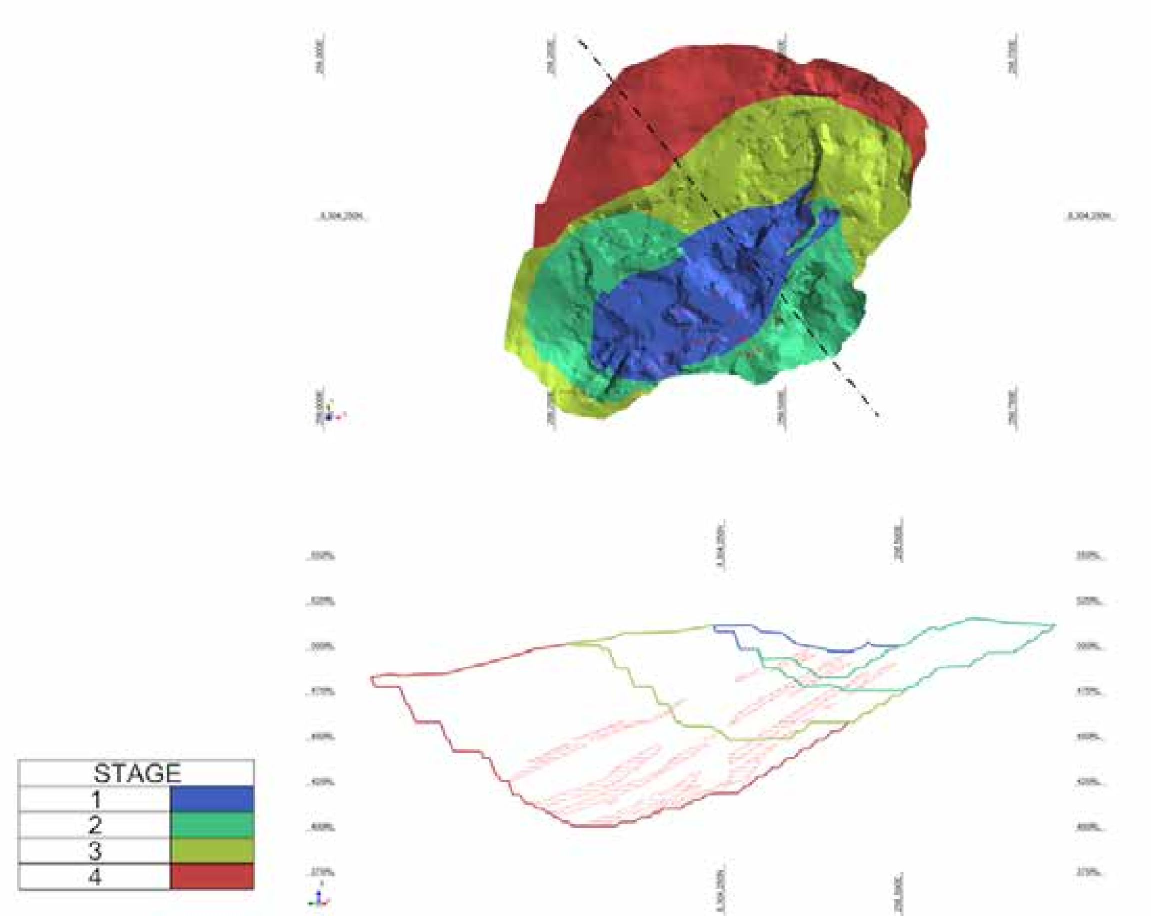


Figure 16.16 Stage 1 Layout, Plan View

Figure 16.17 Stage 2 Layout, Plan View

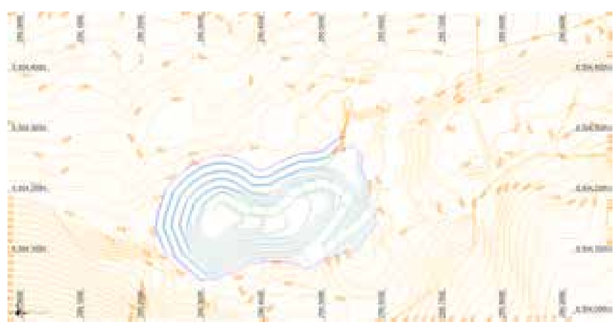


Figure 16.18 Stage 3 Layout, Plan View

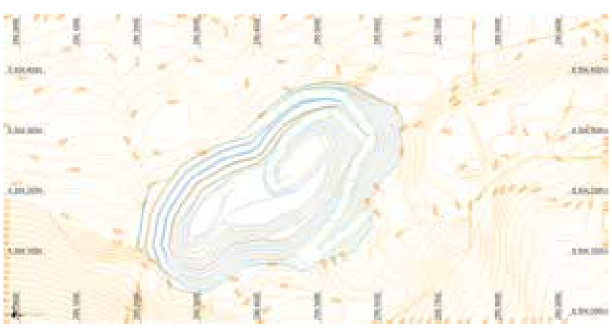
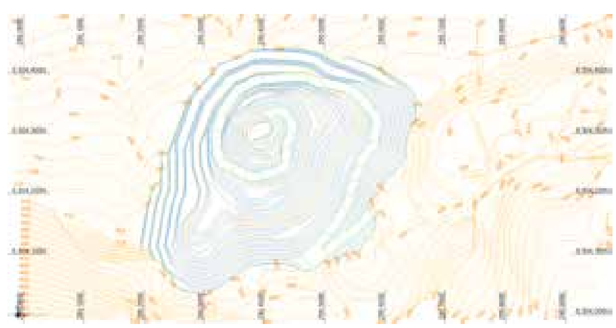


Figure 16.19 Stage 4 Layout, Plan View



16.2.2.2 Mine Production Schedule

A mine production schedule was developed to show the ore tonnes, metal grades, total material and waste material by quarter over the LOM. The distribution of ore and waste contained in each of the mining stages was used to develop the schedule, assuring that criteria such as continuous ore exposure, mining accessibility, and consistent material movements were met.

Deswik® software was selected to run the mining sequencing because it integrates all the functionalities in one package and utilizes a number of different Mixed Integer Linear Programming ("MILP") and Dynamic Programming techniques to work on detailed schedules.

Deswik® software checks if the operation can achieve the maximum NPV and at the same time it provides information about the equipment requirements, blended ore products and truck requirements.

Several runs at various proposed total material movement schedules were done to determine a satisfactory production schedule strategy. It is important to note that this program is not a simulation package, but is a tool for calculation of the mine schedule and haulage profiles for a given set of phases and constraints that must be set by the user.

The mine plan developed by MCB includes dilution based on the SMU model factored by 33.92%. MCB determined a 95% ore mining recovery after considering the proximity of the economic cut-off grade to the background gold content of the rock.

Ore was classified into two categories:

- High Grade Ore: > 0.73 g/t Au
- Low Grade Ore: 0.48 g/t Au to 0.73 g/t Au

16.2.3 Production Schedule Summary

The schedule is based on 90,000 high grade ore tonnes per quarter delivered to the processing plant. Table 16.31 shows the quarterly mine ore production and material movement. The limit on the ore production is the number of benches that are possible to mine in a year in a single phase, or vertical development per phase.

TABLE 16.31
ORE PRODUCTION SCHEDULE SUMMARY

	YEAR 1				YEAR 2				YEAR 3	
	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2
ROM										
Tonnes Ore	123,008	126,676	104,913	138,126	141,320	90,244	114,472	133,025	127,529	10,887
Au (g/t)	1.40	1.34	1.44	1.51	1.65	2.20	1.51	1.48	2.56	4.41
Ounces	5,535	5,442	4,841	6,699	7,480	6,392	5,542	6,345	10,478	1,545
Tonnes Waste	1,226,992	1,223,324	1,545,087	2,078,591	2,108,680	2,132,984	2,135,528	1,091,197	442,288	20,498
W/O	10.0	9.7	14.7	15.0	14.9	23.6	18.7	8.2	3.5	1.9
Tonnes	1,350,000	1,350,000	1,650,000	2,216,718	2,250,000	2,223,227	2,250,000	1,224,221	569,818	31,385
By Grade										
HG - Tonnes (t)	110,000.0	105,211.9	91,546.5	120,000.0	120,000.0	80,114.0	96,195.4	120,000.0	120,000.0	10,380.3
HG - Au (g/t)	1.50	1.49	1.55	1.65	1.83	2.41	1.68	1.58	2.68	4.59
HG - Ounces	5,308	5,029	4,576	6,351	7,076	6,218	5,192	6,082	10,351	1,533
LG - Tonnes (t)	9,591	18,843	12,566	16,021	19,638	6,718	17,430	12,217	5,423	507
LG - Au (g/t)	0.62	0.62	0.63	0.62	0.61	0.62	0.61	0.64	0.59	0.72
LG - Ounces	190	378	256	322	386	133	340	252	103	12
By Litho										
Tonnes Ore	123,008	126,676	104,913	138,126	141,320	90,244	114,472	133,025	127,529	10,887
FINE ARENITE	6,595	13,002	13,417	17,512	16,747	3,517	42,053	19,516	1,255	
LOWER ARENITE		8,648	8,494	23,412	3,375	16,614	5,988	10,830	13,301	573
SCHIST	116,413	105,026	83,002	97,203	121,198	70,112	66,430	102,679	112,973	10,314
HG - Au (g/t)	1.50	1.49	1.55	1.65	1.83	2.41	1.68	1.58	2.68	4.59
FINE ARENITE	1.26	1.43	2.31	1.63	1.97	1.33	1.44	0.97	1.05	
LOWER ARENITE		2.14	1.36	1.84	1.21	2.72	2.24	1.94	1.78	1.00
SCHIST	1.51	1.43	1.45	1.60	1.83	2.40	1.74	1.62	2.81	4.80
Litho Proportion	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%
FINE ARENITE	5.36%	10.26%	12.79%	12.68%	11.85%	3.90%	36.74%	14.67%	0.98%	0.00%
LOWER ARENITE	0.00%	6.83%	8.10%	16.95%	2.39%	18.41%	5.23%	8.14%	10.43%	5.26%
SCHIST	94.64%	82.91%	79.11%	70.37%	85.76%	77.69%	58.03%	77.19%	88.59%	94.74%

Figure 16.20 presents the ore feed by HG and LG category. Figure 16.21 presents the contained gold grade and ounces of the feed. Figure 16.22 presents the total material mined over the LOM.

Figure 16.20 Ore Feed by Category (HG and LG)

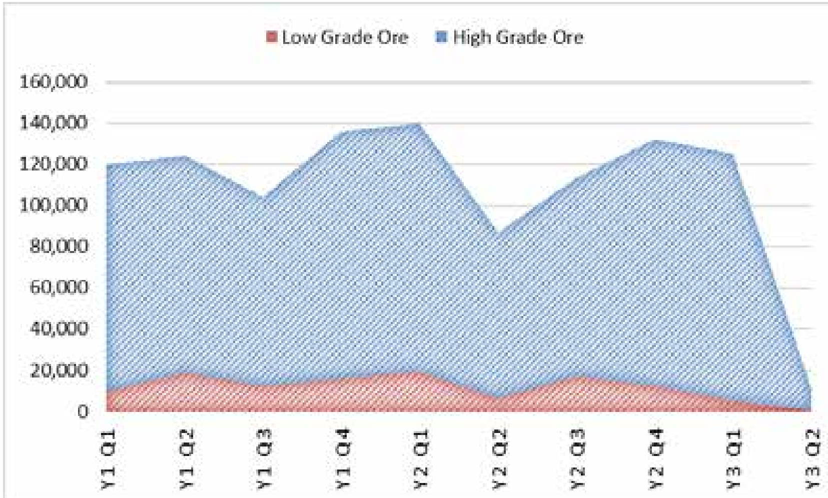


Figure 16.21 Scheduling Results - Contained Ounces and Au Grade

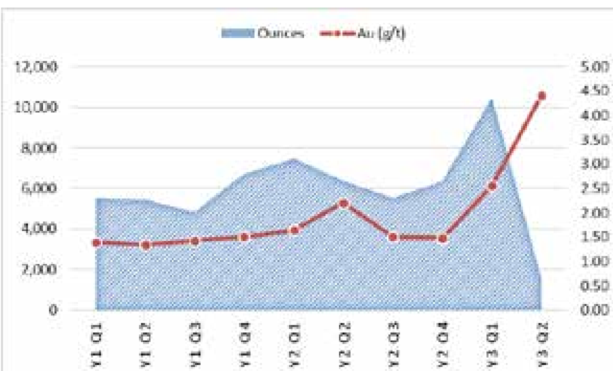
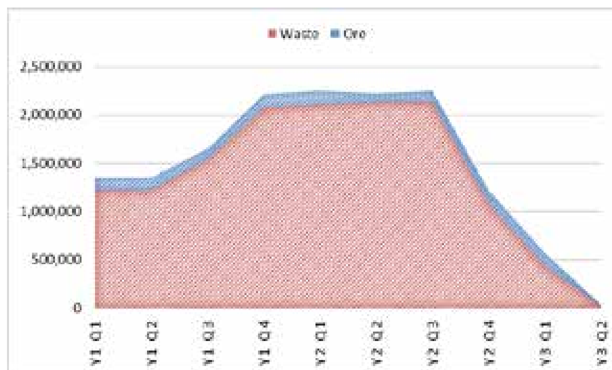


Figure 16.22 Total Material Mined



16.2.4 Production Schedule Details

Table 16.32 and Figures 16.23 to 16.34 present the detailed production schedule over the LOM.

Table 16.32
Production Schedule Comparison

SCHEDULE	YEAR 1 Q1	YEAR 1 Q2	YEAR 1 Q3	YEAR 1 Q4	YEAR 2 Q1	YEAR 2 Q2	YEAR 2 Q3	YEAR 2 Q4	YEAR 3 Q1	YEAR 3 Q2	YEAR 1	YEAR 2	YEAR 3	TOTAL
Tonnes Ore	123,008	126,676	104,913	138,126	141,320	90,244	114,472	133,025	127,529	10,887	492,723	479,061	138,416	1,110,200
Au (g/t)	1.40	1.34	1.44	1.51	1.65	2.20	1.51	1.48	2.56	4.41	1.42	1.67	2.70	1.69
Ounces	5,535	5,442	4,841	6,699	7,480	6,392	5,542	6,345	10,478	1,545	22,517	25,758	12,023	60,298
Tonnes Waste	1,226,992	1,223,324	1,545,087	2,078,591	2,108,680	2,132,984	2,135,528	1,091,197	442,288	20,498	6,073,995	7,468,388	462,786	14,005,170
W/O	10.0	9.7	14.7	15.0	14.9	23.6	18.7	8.2	3.5	1.9	12.3	15.6	3.3	12.6
Tonnes	1,350,000	1,350,000	1,650,000	2,216,718	2,250,000	2,223,227	2,250,000	1,224,221	569,818	31,385	6,566,718	7,947,449	601,203	15,115,369
DESIGN	**Q1**	**Q2**	**Q3**	**Q4**	**Q1**	**Q2**	**Q3**	**Q4**	**Q1**	**Q2**	**YEAR 1**	**YEAR 2**	**YEAR 3**	**TOTAL**
Tonnes Ore	124,404	125,444	109,980	135,359	144,277	88,597	116,693	131,884	123,145	10,942	495,188	481,451	134,087	1,110,726
Au (g/t)	1.40	1.34	1.40	1.55	1.67	2.18	1.49	1.49	2.60	4.28	1.42	1.67	2.74	1.69
Ounces	5,594	5,393	4,940	6,728	7,748	6,199	5,589	6,328	10,300	1,505	22,656	25,865	11,804	60,326
Tonnes Waste	1,241,970	1,225,562	1,554,607	1,979,091	2,189,341	2,158,022	2,114,928	1,101,650	456,642	22,558	6,001,230	7,563,942	479,200	14,044,371
W/O	10.0	9.8	14.1	14.6	15.2	24.4	18.1	8.4	3.7	2.1	12.1	15.7	3.6	12.6
Total Tonnes	1,366,374	1,351,006	1,664,587	2,114,450	2,333,618	2,246,619	2,231,621	1,233,534	579,786	33,500	6,496,418	8,045,392	613,287	15,155,097
DIFFERENCE	**Q1**	**Q2**	**Q3**	**Q4**	**Q1**	**Q2**	**Q3**	**Q4**	**Q1**	**Q2**	**YEAR 1**	**YEAR 2**	**YEAR 3**	**TOTAL**
Tonnes Ore	1.1%	-1.0%	4.8%	-2.0%	2.1%	-1.8%	1.9%	-0.9%	-3.44%	0.51%	0.50%	0.50%	-3.13%	0.05%
Au (g/t)	-0.1%	0.1%	-2.7%	2.5%	1.5%	-1.2%	-1.1%	0.6%	1.79%	-3.09%	0.12%	-0.08%	1.35%	0.00%
Ounces	1.1%	-0.9%	2.0%	0.4%	3.6%	-3.0%	0.9%	-0.3%	-1.71%	-2.60%	0.62%	0.42%	-1.82%	0.05%
Tonnes Waste	1.2%	0.2%	0.6%	-4.8%	3.83%	1.17%	-0.96%	0.96%	3.25%	10.05%	-1.2%	1.3%	3.5%	0.3%
W/O	0.1%	1.2%	-4.0%	-2.8%	1.70%	3.05%	-2.85%	1.83%	6.92%	9.49%	-1.7%	0.8%	6.9%	0.2%
Tonnes	1.2%	0.1%	0.9%	-4.6%	3.72%	1.05%	-0.82%	0.76%	1.75%	6.74%	-1.1%	1.2%	2.0%	0.3%

Figure 16.23 Production Schedule, Tonnage Comparison

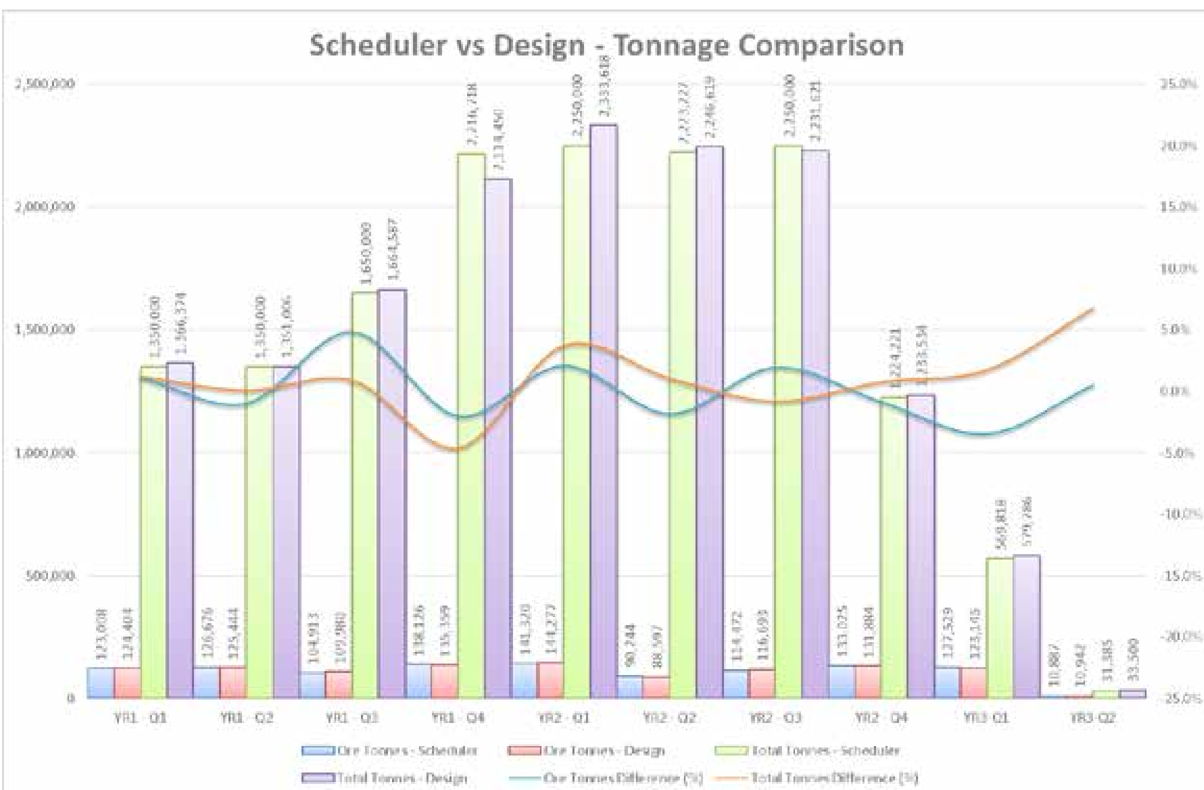


Figure 16.24 Production Schedule – Grades and Ounce Comparison

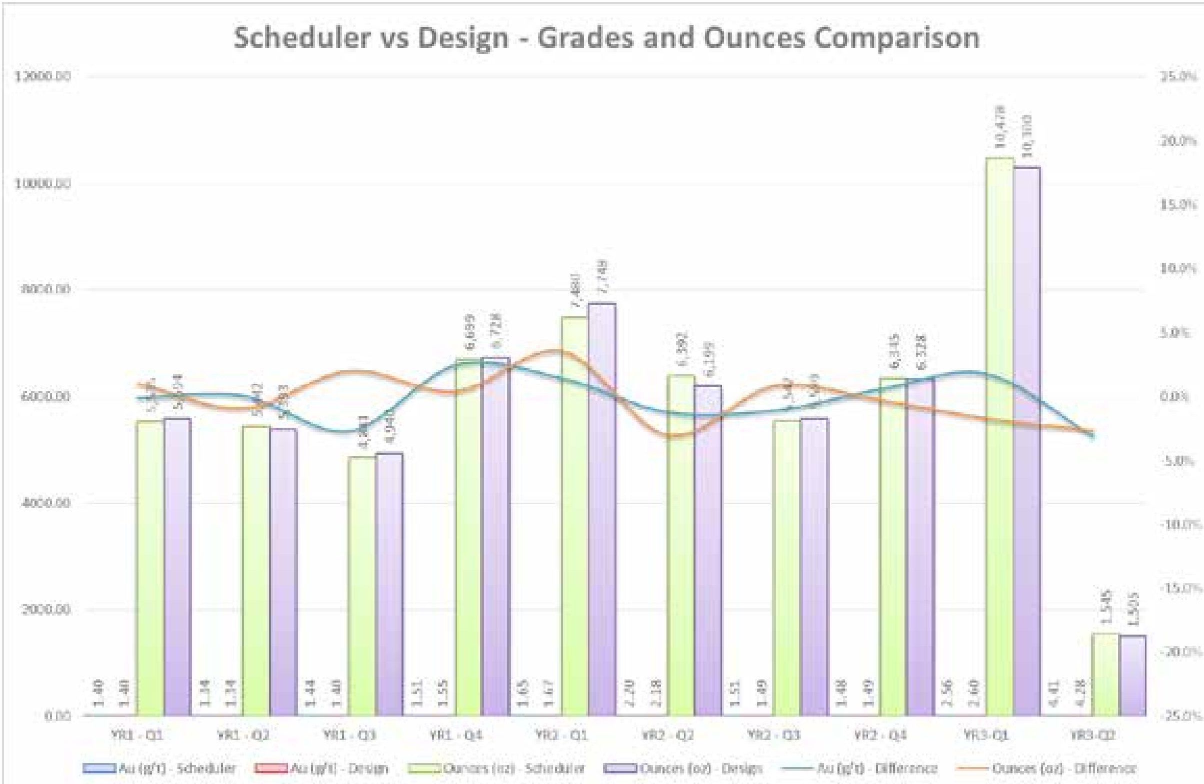


Figure 16.25 Schedule Y1-Q1

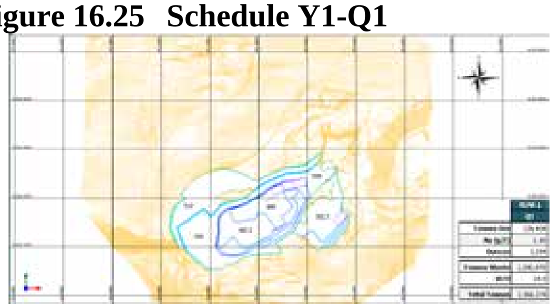


Figure 16.26 Schedule Y1-Q2

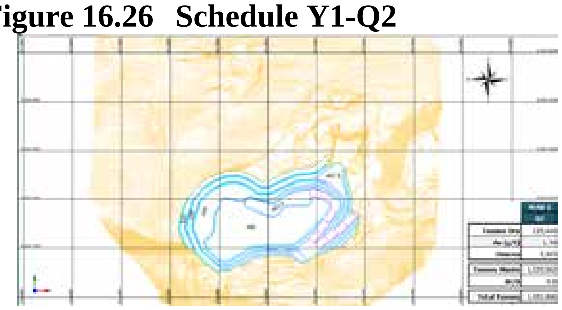


Figure 16.27 Schedule Y1-Q3

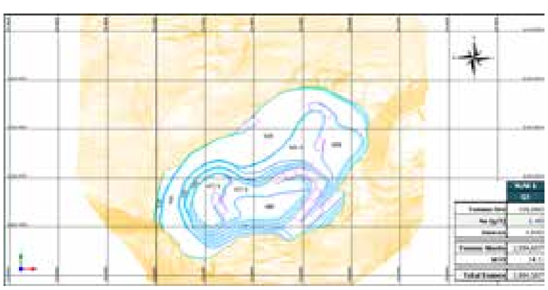


Figure 16.28 Schedule Y1-Q4

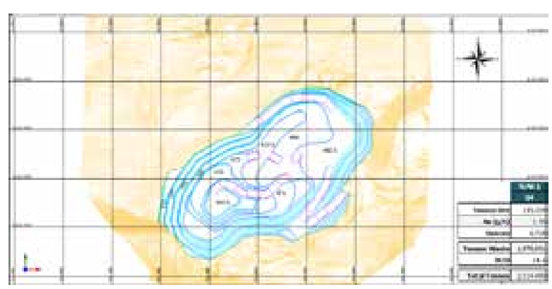


Figure 16.29 Schedule Y2-Q1

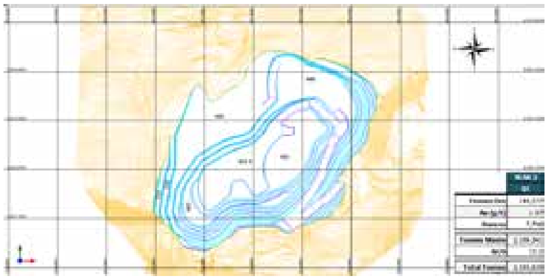


Figure 16.30 Schedule Y2-Q2

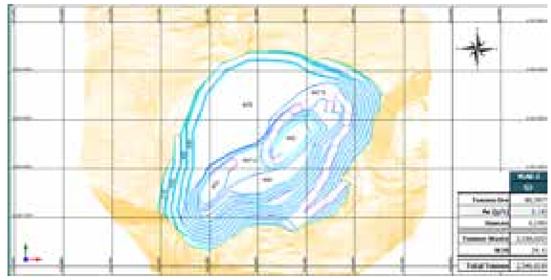


Figure 16.31 Schedule Y2-Q3

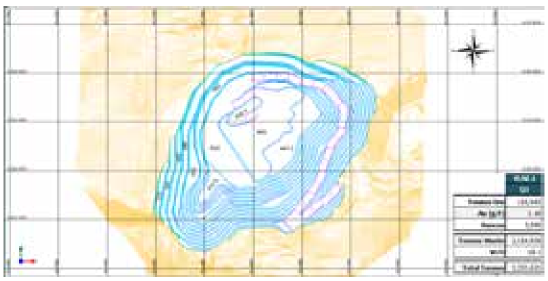


Figure 16.32 Schedule Y2-Q4

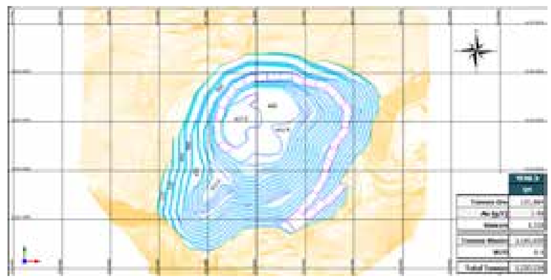


Figure 16.33 Schedule Y3-Q1

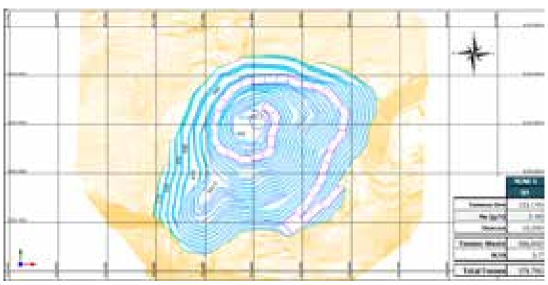
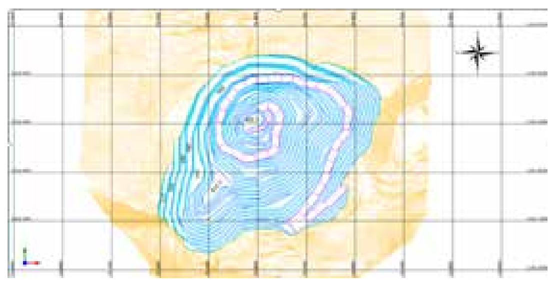


Figure 16.34 Schedule Y3-Q2



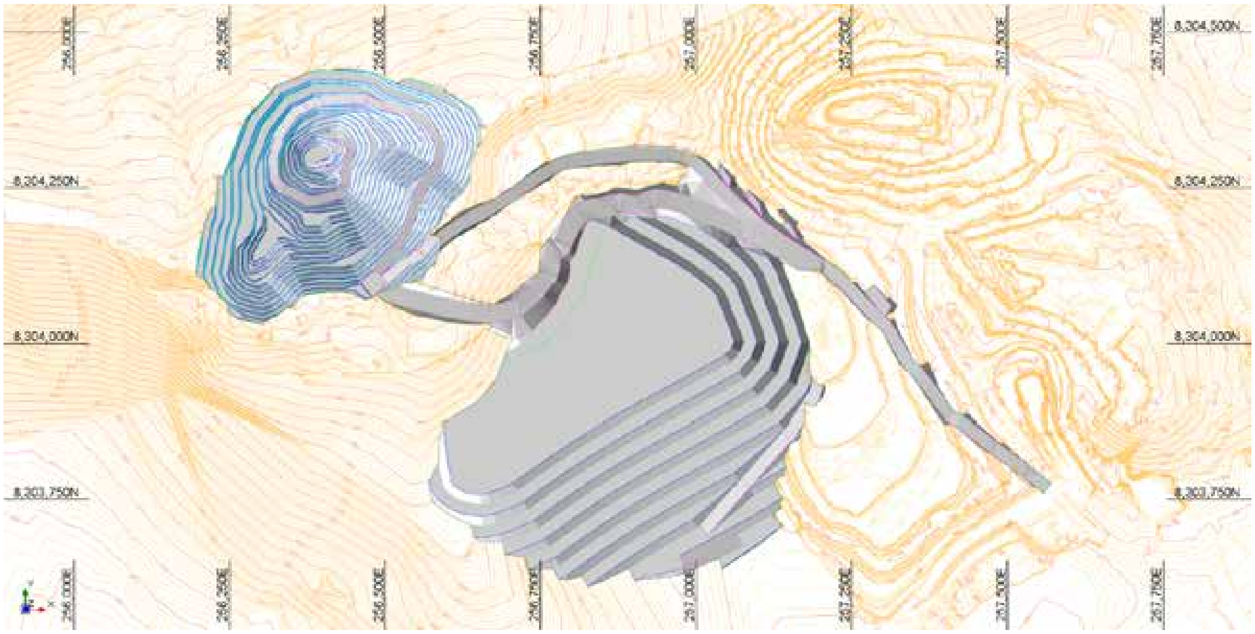
16.2.5 Waste Rock Storage

A waste rock storage area at the east of the pit was designed for the Lavrinha Project. The waste storage facility layout was conceptually designed by MCB to fulfill the total required waste rock volume.

The total demand for waste disposal is approximately 8.7 Mm^3 swell volume (swell factor equal to 1.4). Figure 16.35 presents the waste rock storage facility layout.

MCB recommends that the waste rock storage area design be advanced to a detailed engineering level including elements such as foundation evaluations, design criteria, stability analysis, internal and surface drainage design.

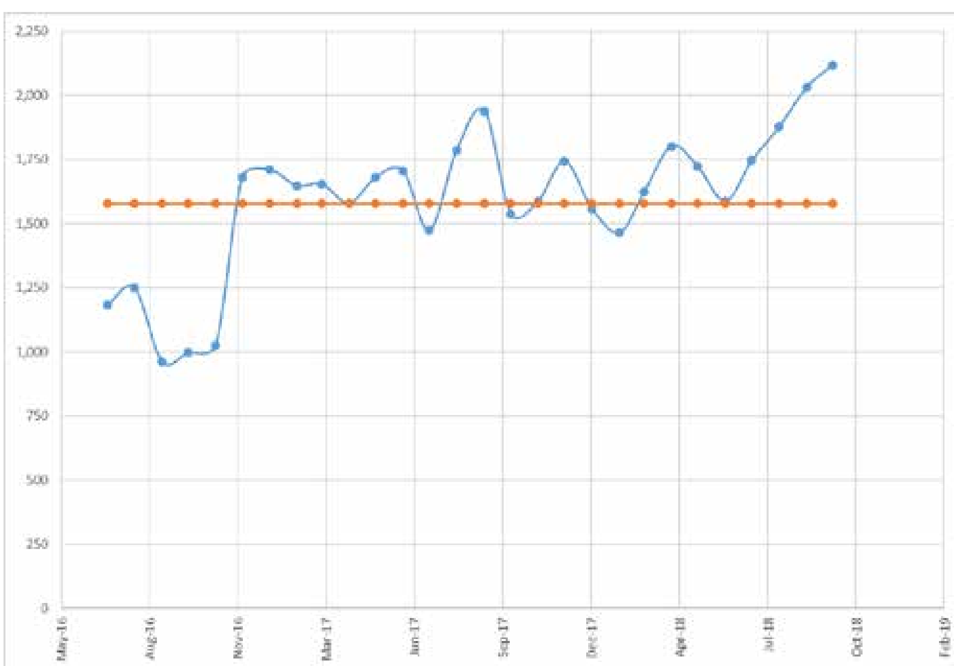
Figure 16.35 Waste Rock Storage Layout



16.2.6 Average Hauling Distance

Landform and Haulage Scheduler® ("LHS") from Deswik® software was used to calculate the Average Hauling Distance ("AHD") using a strategy to minimize the distance. The monthly average distances are presented in Figure 16.36.

Figure 16.36 Monthly Average Haul Distances (m)



16.2.7 Grade Control

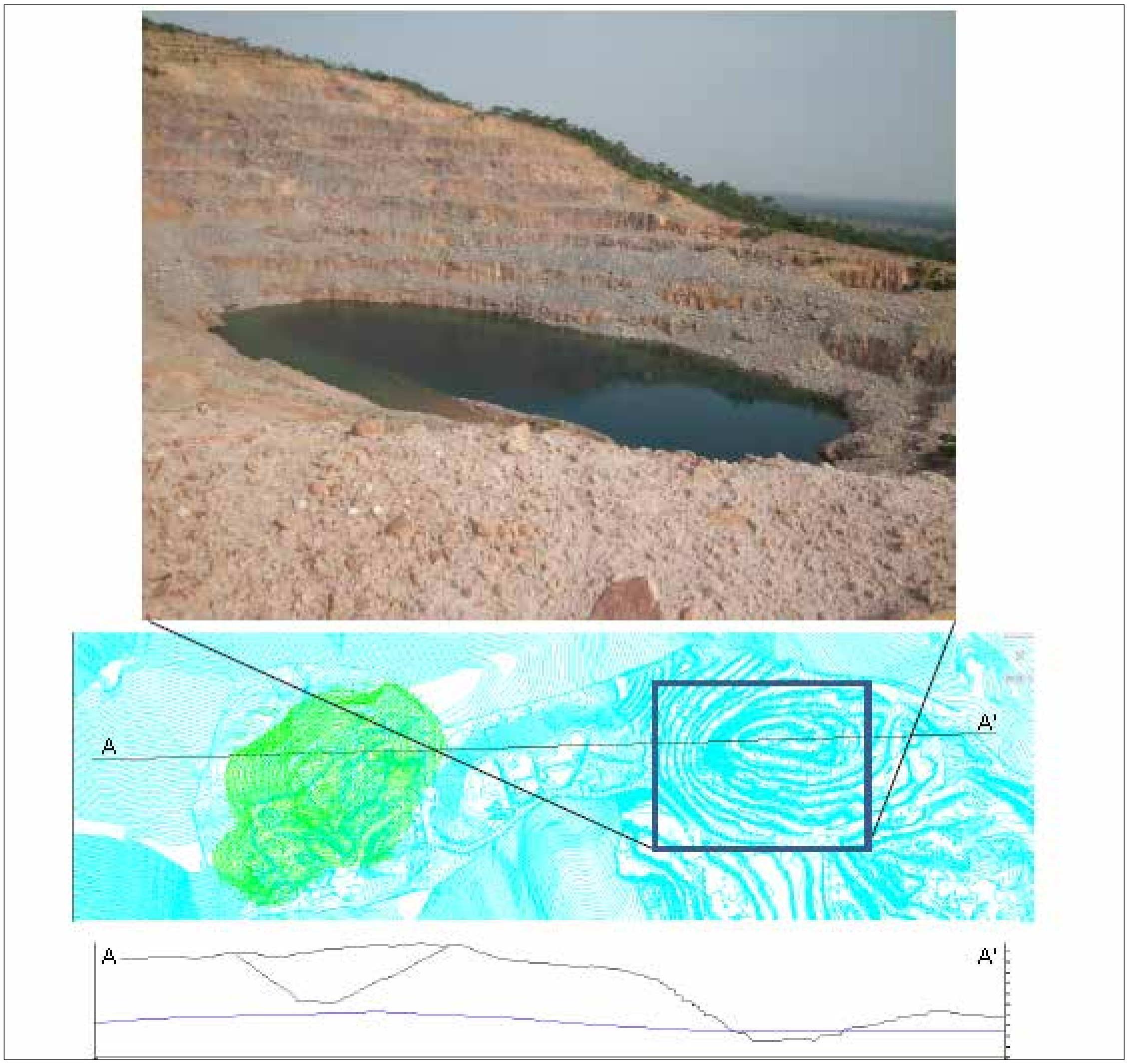
Careful grade control must be carried out during mining to minimize misplaced ore. These efforts should include the following standard procedures:

- Implement an intense and systematic program of sampling, mapping, laboratory analyses and reporting.
- Utilize specialized in-pit, bench sampling drills for sampling well ahead of production drilling and blasting.
- Use of excavators and benches no higher than 2.5 m (as presently being planned) to selectively mine ore zones.
- Maintain top laboratory staff, equipment, and procedures to provide accurate and timely assay reporting.
- Utilize trained geologists and technicians to work with excavator operators in identifying, marking, selectively mining and dispatching ore and waste.

16.2.8 Hydrogeolocial Model

A hydrogeological model was not generated for the Lavrinha Project. The water table at the Ernesto pit (Figure 16.37) that is located near Lavrinha´s pit (850 m distance apart), is on elevation 347 m, approximately 50 m below Lavrinha’s bottom pit elevation (402.5 m). This study must be implemented during mining operations.

Figure 16.37 Longitudinal Section Intersecting Ernesto and Lavrinha Pits



16.2.9 Stormwater and Pit Dewatering System

Stormwater and pit dewatering systems were not designed in this study. The LOM has an estimated life of 2.5 years. Throughout this period the surfaces of the pit shall be subject to rain which may induce erosion processes on the benches. Additionally, stormwater and groundwater flows towards the pit bottom could result in operational problems.

These situations require a stormwater drainage and a pit dewatering system designed to control the flows along the benches subject to erosion. These systems also help in minimizing the input of water to the pit bottom, guiding water out of the pit where possible.

Minimum maintenance at sites close to the pit bottom is achieved by a pit dewatering system in which the pumps and sumps are operated together to keep sites dry.

The final bench designs must include permanent drainage structures for operation with a large discharge capacity.

The dewatering system must be reviewed over time while the pit is deepened.

16.2.10 Mine Equipment

Lavrinha open pit mining is by typical truck and shovel operations. Contractor mining operations have been scheduled in order to meet the production targets. Dinex Engenharia Mineral Ltda ("Dinex") has been contracted to mine the Lavrinha open pit.

The mine fleet is composed of small equipment which are compatible with the production level of the Lavrinha mine.

The current mining operation as well as the current block model work with a maximum bench height of 10 m.

The trucks haul waste to the nearest waste dump (adjacent to the pit). The ore is hauled to the crusher site.

Two haul truck sizes were selected: 25 t haul trucks for ore and 36 t haul trucks for waste, with compatible loading equipment.

The fleet is complemented with drilling rigs for ore and waste, as 100% of material is defined as hard rock. Auxiliary equipment includes track dozers, motor graders and water trucks.

The fleet size was calculated by MCB to support Lavrinha open pit demand. The required fleet size is presented in Table 16.33.

TABLE 16.33
MINE EQUIPMENT FLEET

Fleet	# Units
Truck 25 tonne	5
Truck 38 tonne	10
Front-End Loader	3
Hydraulic Shovel	4
Drill 2"	1
Drill 5 1/2"	2
Track Dozer D8	3
Track Dozer D6	1
Grader CAT16	1
Loader CAT 980	1
Lowbed Trailer	1
Lube Truck	1
Water Truck	2
Backhoe Excavator CAT349	1
Supply Convoy Truck	1

16.2.11 Trade-off Study, Owner Versus Contractor

For the purpose of this study both ownership or contractor equipment have the same expected lifetimes so it is proper to simply compare the present value of expenses of the two options.

Total Expenses Cash Flow (cost + investment) has been analyzed for the ownership and contractor options. The analysis took into account the fleet sizing, operational costs ("OPEX") and capital expenditure ("CAPEX"). Additionally the differential cash flow has been calculated to demonstrate gains and losses and the best option to be implemented. Tables 16.34 to 16.36 present annual differential cash flows.

TABLE 16.34
DIFFERENTIAL CASH FLOW 2016

		2016					
TONNES x 1000		472.5	472	473	473	472	472
Own Fleet		Jul	Aug	Sep	Oct	Nov	Dec
Disbursements	US$ 10^3	**16,171**	**1,081**	**421**	**428**	**418**	**789**
CAPEX	US$ 10^3	15,751	650	0	0	0	350
Operational Cost	US$ 10^3	421	431	421	428	418	439
OPEX US$/t		0.89	0.91	0.89	0.91	0.88	0.93
Outsourcing		Jul	Aug	Sep	Oct	Nov	Dec
Disbursements	US$ 10^3	**1144**	**952**	**949**	**953**	**948**	**1017**
Mobilization/Demobilization	US$ 10^3	107	0	0	0	0	0
Operational Cost	US$ 10^3	1037	952	949	953	948	1017
OPEX US$/t		2.19	2.01	2.01	2.02	2.01	2.15
Differential Cash Flow	US$ 10^3	**-15,028**	**-129**	**528**	**525**	**530**	**229**

TABLE 16.35
DIFFERENTIAL CASH FLOW 2017

		2017											
TONNES x 1000		473	472	787.5	812	753	773	787	768	793	778	783	800
Own Fleet		Jan	Feb	Mar	Apr	Mai	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Disbursements	US$ 10^3	**443**	**437**	**5,438**	**1,354**	**656**	**679**	**1,103**	**1,121**	**1,168**	**1,102**	**1,111**	**1,117**
CAPEX	US$ 10^3	0	0	4,770	650	0	0	0	0	0	0	0	0
Operational Cost	US$ 10^3	443	437	668	704	656	679	1,103	1,121	1,168	1,102	1,111	1,117
OPEX US$/t		0.94	0.93	0.85	0.87	0.87	0.88	1.40	1.46	1.47	1.42	1.42	1.40
Outsourcing		Jan	Feb	Mar	Apr	Mai	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Disbursements	US$ 10^3	**850**	**1024**	**1364**	**1416**	**1322**	**1349**	**1375**	**1341**	**1382**	**1359**	**1367**	**1457**
Mobilization/Demobilization	US$ 10^3	0	0	0	0	0	0	0	0	0	0	0	0
Operational Cost	US$ 10^3	850	1024	1364	1416	1322	1349	1375	1341	1382	1359	1367	1457
OPEX US$/t		1.80	2.17	1.73	1.74	1.76	1.74	1.75	1.75	1.74	1.75	1.75	1.82
Differential Cash Flow	US$ 10^3	**406**	**587**	**-4,074**	**62**	**666**	**670**	**272**	**220**	**214**	**257**	**256**	**340**

TABLE 16.36
DIFFERENTIAL CASH FLOW 2018

		2018								
TONNES x 1000		**751**	**740**	**850**	**607**	**346**	**319**	**251**	**199**	**153**
Own Fleet		**Jan**	**Feb**	**Mar**	**Apr**	**Mai**	**Jun**	**Jul**	**Aug**	**Sep**
Disbursements	US$ 10^3	**1,053**	**1,043**	**1,202**	**908**	**571**	**552**	**555**	**403**	**300**
CAPEX	US$ 10^3	0	0	0	0	0	0	0	0	0
Operational Cost	US$ 10^3	1,053	1,043	1,202	908	571	552	555	403	300
OPEX US$/t		1.40	1.41	1.41	1.50	1.65	1.73	2.21	2.02	1.96
Outsourcing		**Jan**	**Feb**	**Mar**	**Apr**	**Mai**	**Jun**	**Jul**	**Aug**	**Sep**
Disbursements	US$ 10^3	**1431**	**1295**	**1545**	**1227**	**749**	**729**	**626**	**554**	**1145**
Mobilization/Demobilization	US$ 10^3	0	0	0	0	0	0	0	0	64
Operational Cost	US$ 10^3	1431	1295	1545	1227	749	729	626	554	1081
OPEX US$/t		1.91	1.75	1.82	2.02	2.16	2.28	2.49	2.78	7.08
Differential Cash Flow	**US$ 10^3**	**377**	**252**	**342**	**319**	**178**	**177**	**71**	**151**	**845**

Considering a monthly discount rate equal to 1%, the differential cash flow analysis considering onwership vs contractors indicates the following:

- Net Present Value of the Costs: US$-11.45 M
- Internal Rate of Return: -3%

Based on information provided by Aura on the Dinex contractor cost quote, and a budget spreadsheet provided by Aura on ownership costs, and considering MCB's experience in similar projects, it was concluded that the contractor has a significant advantage providing an NPV gain of US$11.45 M for the Project when compared to ownership costs.

16.2.12 Manpower

Lavrinha open pit operations will be on a schedule of 24 hours per day on 3 shifts, 365 days per year. For most operating positions there are four work crews with three on site at any time working three 8 hour shifts per day.

Mining operating manpower is based on approximately four operators for each operating position. Mining manpower for operations, maintenance, and technical services is estimated at 230 staff, employees, and contractors. MCB considers the manpower estimates to be reasonable.

16.2.13 Mine Infrastructure

Lavrinha has all necessary infrastructure for an open pit mine operation including a truck shop, truck wash facility, warehouse, fuel storage and distribution facility, explosives storage and magazine sites, electrical power distribution and substations to support construction projects and mine operations.

16.3 PAU-A-PIQUE

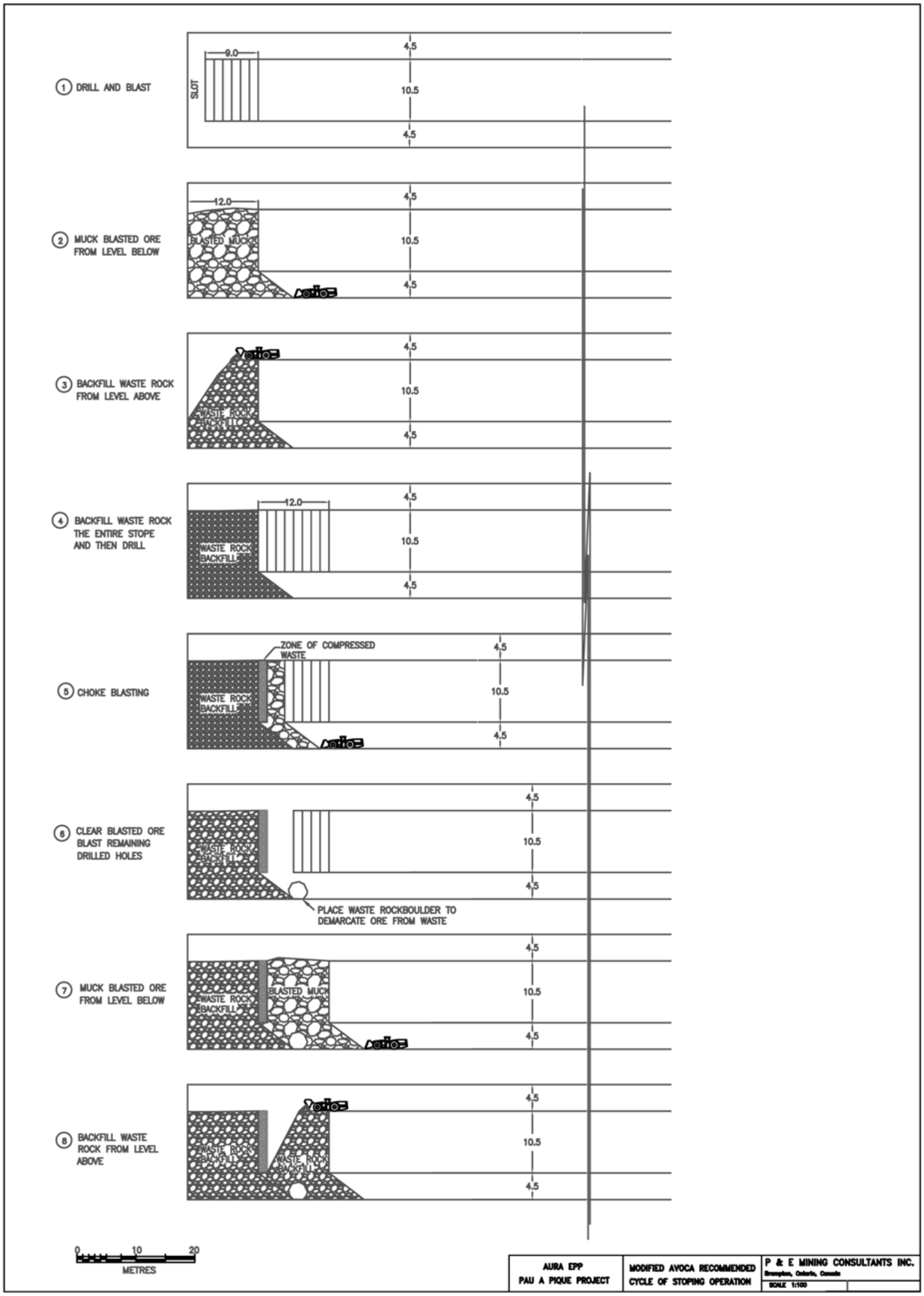
16.3.1 Introduction

Mining at the Pau-a-Pique underground gold Project will be conducted by a modified Avoca choke blasting stoping method, with ore transported to the ROM pad on surface by 30 tonne

ADT haul trucks operating through one decline. Ore will be subsequently hauled on a 47 km surface road to the Ernesto processing plant. Access to the underground mine is via a single portal located next to the main mining office.

Avoca with choke blasting stoping will be employed to extract the Area 7 and 8, NW, SE and P3 and P4 ore bodies. The stoping method applied to these ore bodies is via HW access ore drives with levels spaced at 15 m and 21 m intervals, for the upper and lower areas of the deposit, respectively. The upper and lower areas will be separated by a sill pillar. Unconsolidated rock fill ("URF") will be used to backfill the stopes over the LOM. Figure 16.38 illustrates the mining method.

Figure 16.38 Avoca Choke Blasting Mining Method Proposed for Pau-a-Pique



The majority of underground mining activities utilise Aura's employees, with external contractors or suppliers to undertake the supply of explosives, piping and services, ground support consumables, truck haulage underground and on surface and other specialised tasks i.e. site security etc. Aura has 100% ownership of all major fixed plant used at the mine, with the exception of haulage trucks and temporary maintenance plant used by the haulage contractor to service the trucks and other supplied mobile equipment for the project development, stoping activities and other project work.

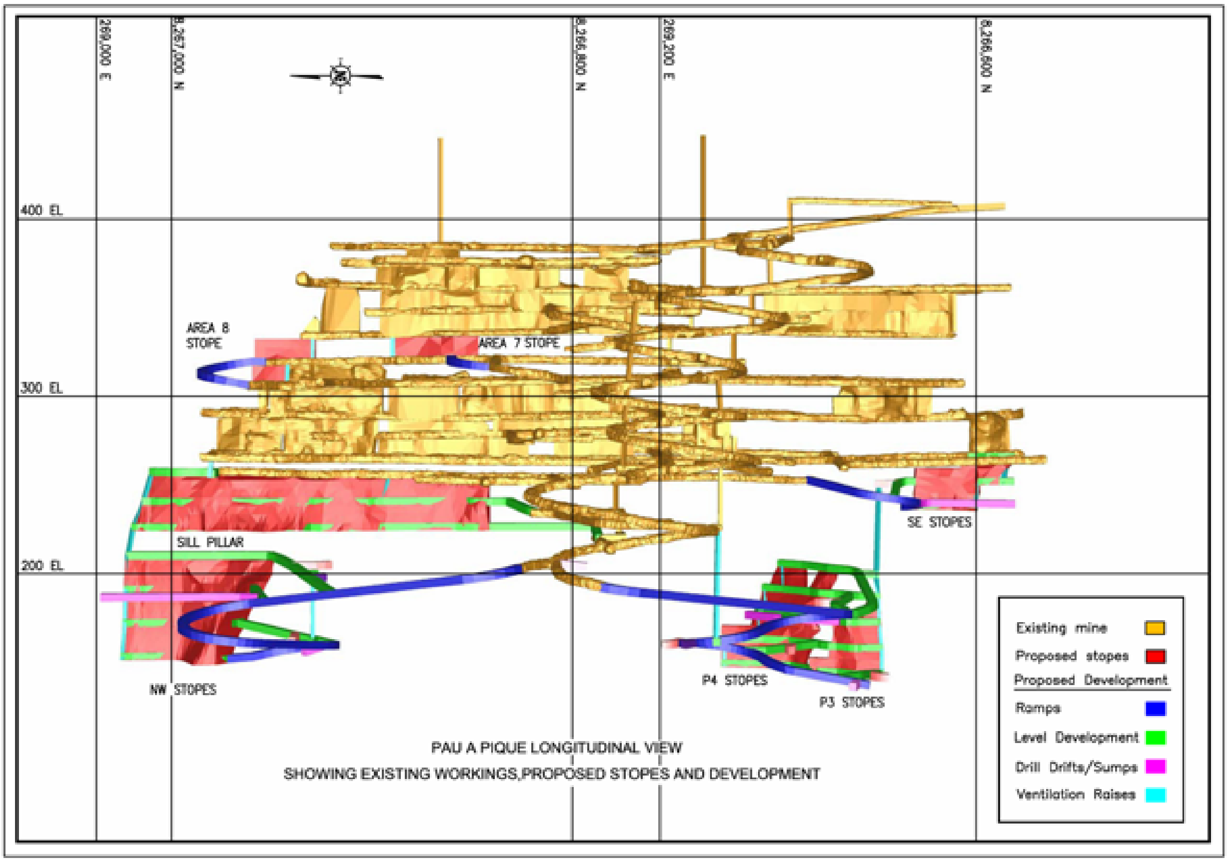
16.3.2 Mine Description

16.3.2.1 Mine Access

Access to the mine is by a 47 km surface road from the Ernesto process plant and a single decline over the first 2,400 m with a second decline to the P3 and P4 ore bodies. The existing nominal 4.5 m x 4.5 m profile decline of some 1,200 m at 1:7 (15%) gradient must be scaled and checked and reconditioned if required to become the main haulage ramp and access for the mine. Development of additional decline 4.5 m x 4.5 m also at 1:7 gradient, will recommence to provide service access and haulage for NW stopes above the sill pillar. Access to Area 7 and 8 stopes requires some 150 m of additional waste development mainly to re-establish Area 7 access due to a ground fall at the existing crosscut and level drive intersection, and second egress establishment in the vicinity of Area 8 stope. The SE ore body will be accessed via a short decline located in the HW.

A longitudinal projection of the mine access and development is presented in Figure 16.39.

Figure 16.39 Longitudinal Projection of the Pau-a-Pique Mine Access and Development



Within the current mine plan and evaluation, additional development has been included to accommodate ore definition drilling. The additional development was included not only in the design but also into the schedule and mine sequence to properly cater for the information turnaround from the drilling program. The drilling will help define the stope shapes and will guide the best location of ore drifts. P&E has scheduled a 12 m x 12 m drill spacing which equates to approximately 12,000 m of definition drilling. P&E strongly recommends that definition drill data be available ahead of the stope extraction which subsequently must be used in the mine planning process before a particular stope is developed and mined. This will enable the mine operations to properly place the ore accesses within the stope designed boundaries and minimize stope dilution incurred during extraction, which the operation struggled with in the past.

Definition drilling components have been costed and included in the current study (mine plan schedule) and sequenced to ensure that the data obtained from drill holes will be available ahead of the design and planning tasks approximately one month before any stope or ore mining area will be accessed.

16.3.2.2 Ventilation

Downcast ventilation is through the 4.5 m x 4.5 m decline which will be augmented by one 3 m x 3 m escape raise. No cooling or heating is required for the supplied fresh air due to location of the Pau-a-Pique Project. Total air volume required over the LOM is currently estimated at some 180 m3/sec.

The primary ventilation circuit has been highlighted and reflects the planned ventilation network at the end of the mine life. A VentSim software visual model of the final mine design has been performed to produce the underground ventilation diagram.

The ventilation simulation is based on a latest mining design and equipment HP distribution for the Pau-a-Pique Project. The primary objective of this set of analysis was to confirm adequacy of the capacity of the existing primary fans, used as exhaust.

A worst case scenario was considered assuming furthest points of the east and west side of developments were ventilated and fresh air was partitioned at the bottom-most ramp using natural splitting. A total of 180 m^3/s of return air was exhausted through the combination of two 45 m^3/s in parallel (90 m^3/s) at each return air raise. The fan configuration used is shown in Figure 16.40.

Figure 16.40 Pau-a-Pique Ventilation Fan Configuration

Total fresh air intake of 180 m^3/s was maintained as the same as when the mine was previously operated with a breakdown of 163 m^3/s through the main ramp and 17 m^3/s through the egress. Conservatively, it was assumed that there were three working areas each requiring 60 m^3/s of fresh air.

Current analysis did not show any auxiliary fans as those were shown adequate and a local area analysis could be done quickly. Following assumptions were used in the ventilation network simulations:

- Density of air as standard density of 1.2 kg/m^3
- Drifts were sized as 4.5 m (w) x 4.5 m (h)
- Raises were sized as 3.1 m diameter
- Ramps were sized as 4.5 m (w) x 4.50 m (h)

- Two primary exhaust fan branches (at each return air raise to surface) with a flow of 90 m^3/s

Results of the ventilation simulation are illustrated in Figure 16.41, and the exhaust fan performance curve is shown in Figure 16.42.

Figure 16.41 Pau-a-Pique Ventilation Model

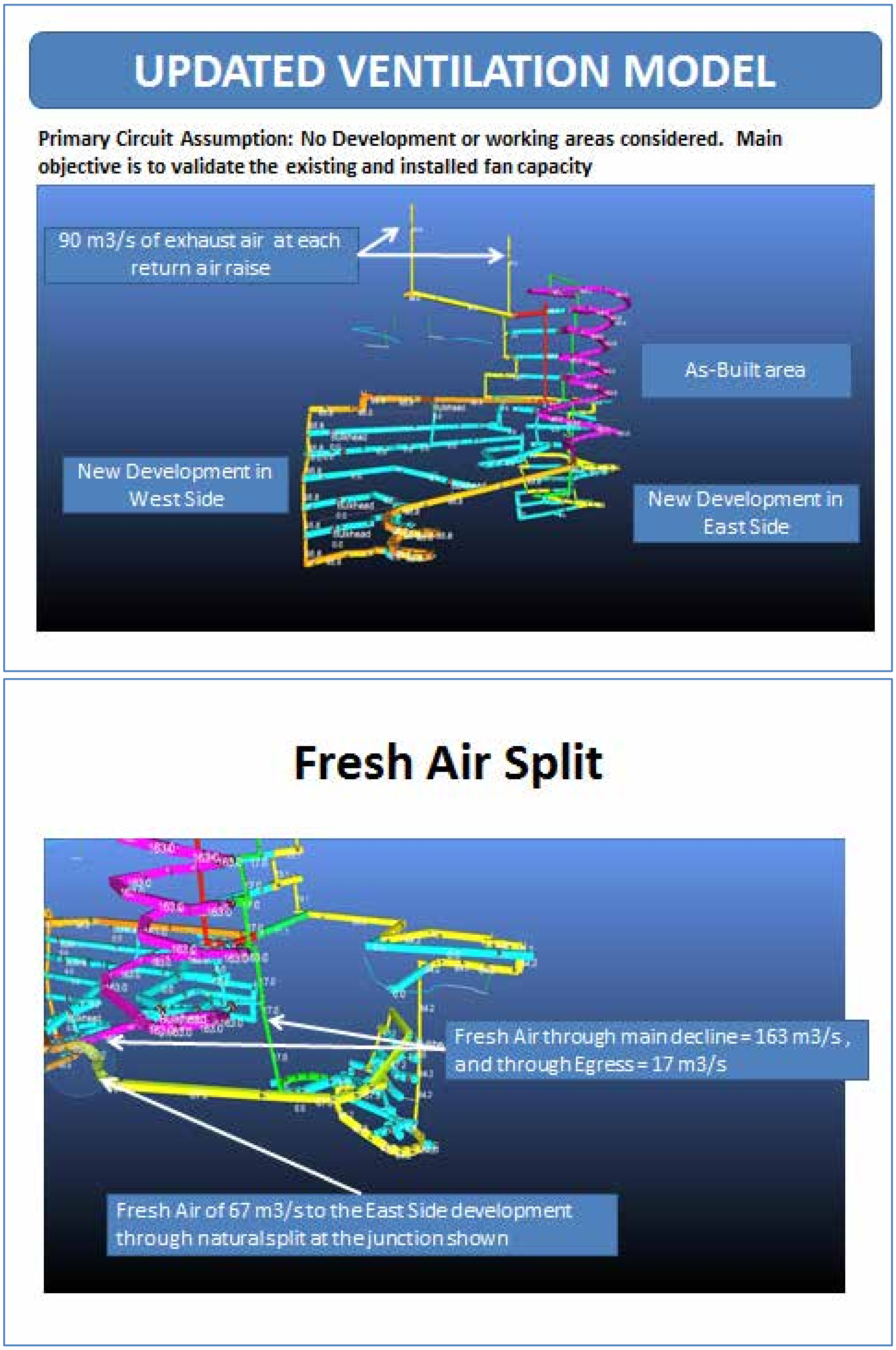
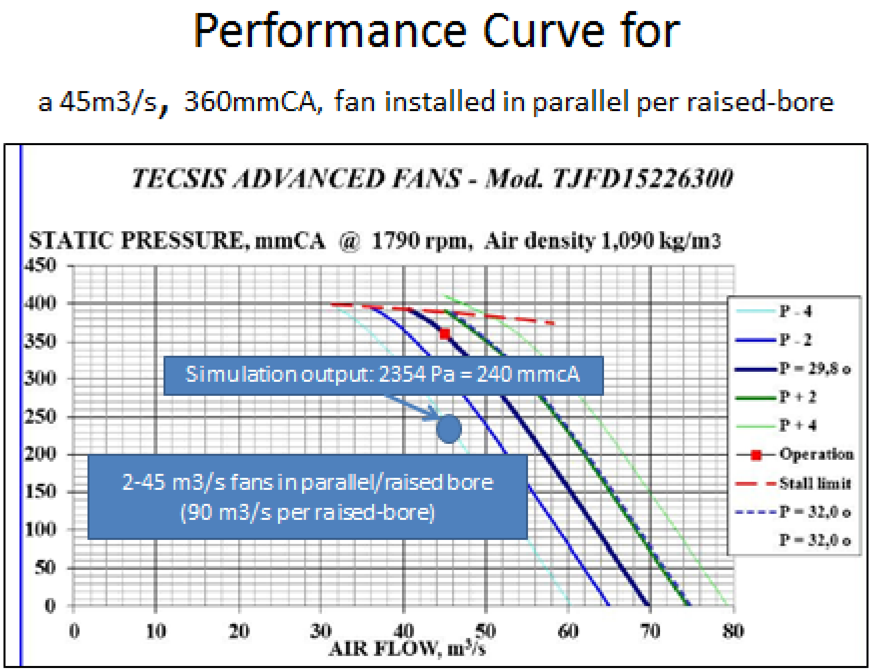


Figure 16.42 Pau-a-Pique Exhaust Fan Performance Curve



The primary fans available for the Pau-a-Pique Project are adequate to implement a safe ventilation plan for the proposed project. The approach was conservative and a more realistic study could optimize air requirement and further implementation of Ventilation on Demand ("VOD") would save operating costs during mine production.

16.3.3 Mine Development

Ore and waste will be loaded on each level directly into underground haulage trucks at a purpose-built loading bay. Haulage from the levels to the surface stockpiles will be by 30t capacity ADT surface haulage trucks. Development waste will initially be hauled to surface and stored underground as much as possible but as stoping activity advances it is planned to be tipped directly into stope voids as backfill material as well as to produce aggregate for roads.

All development face drilling will be done by a two boom electric hydraulic jumbo, drilling each round to ~4.1 m in depth. All waste headings are supported with resin rebar (back), swellex inflatable bolts (walls) and welded wire mesh. All ore drives are supported with resin rebar (back), swellex inflatable bolts (walls) and welded wire mesh first pass ground support and second pass one layer of shotcrete at thickness dictated by the ground conditions and 6 m grouted cable bolts.

Additional ground support will be applied if ground conditions dictate, usually for intersections or larger excavation areas. Development faces are charged using an ITH or scissor lift equipment, utilising air-loaded ANFO and packed emulsion cartridges.

16.3.4 Mine Production

The mining method is Avoca with choke blasting for all stopes which are filled with unconsolidated waste rock. Stopes will be arranged in an inverted triangle pattern with bottom stopes being extracted first and up to last level along strike. Stopes will be taken in sequence from the bottom level upwards to sill pillars. Sill pillar extraction on retreat at the end of the sequence presents an upside potential and will be assessed geotechnically during mining operations. Sill pillars have been excluded from this evaluation.

16.3.5 Mining Method

16.3.5.1 Avoca Stopes

The Area 7 and 8, SE, NW and P3 and P4 Mineral Resource Estimate (at 2.40 g/t Au BCOG) contain 309Kt of mineralized material at an average grade of 3.76 g/t Au. This has been converted into a mine inventory of 320Kt based on Avoca choke blasting stope mining with a mill feed grade of 3.24 g/t Au including mining dilution and stope recovery. This equates to a mine life of approximately one and a half years based on an average mining extraction rate of 213Ktpa including initial production ramp-up.

To maximise the extraction tonnage and reduce stope dilution, open voids will be backfilled immediately with waste rock to support stope walls. Due to the fact that in the past stope dilution was an ongoing problem for the operation, within the current LOM plan the maximum open span for each stope will be closely monitored using a hydraulic radius approach to minimize the HW/FW surface exposed as well as to reduce the time between ore extraction and void backfilling. Waste rock necessary for backfill will be sourced from ongoing development stored underground as necessary to reduce haulage from surface and therefore traffic congestion on the main access ramp.

The total waste produced by the mine from ongoing development will not suffice the backfill requirements and falls short by some 44Kt, so this quantity will be hauled back from the Ernesto waste stockpile to the Pau-a-Pique mine via a 47 km surface road. Broken ore from the stopes will be extracted with conventional and tele-remote mucking techniques.

Once the stope is complete, it is filled with waste rock from development. Due to sequencing in extraction of NW stopes it is envisioned that there will be one 12 m height sill pillar left behind in order to have a continuous stope production below the 220 m Elev within the NW ore body. The sill pillar contains approximately 5,000 oz Au and can potentially be extracted at the end of the mine life once additional geotechnical work has been performed i.e. 3D geotechnical modelling and stope stress analysis, corroborated with information obtained from a definition drilling program, and represents upside potential for the mine. Current reserve figures exclude the sill pillar tonnage and ounces.

During site visits, P&E reviewed previous practices and stope designs and assessed jointly with Aura’s site blasting technician and geotechnical engineer the ongoing issue of high stope dilution before and after the Avoca choke blasting method was introduced. It became relevant that the implementation of this variation of the Avoca mining method was not fully completed or optimized. Since this happened over a very short period of time it was understood that the mine technical services and operations department did not have sufficient time to address all aspects of the mining method change. The change occurred towards the end of Yamana ownership and approximately four months before Pau-a-Pique ceased operation. Also there is little relevant information captured over this period relative to method benefits, reduction in dilution, stope cycle, HW and FW behaviour after blasts, etc that can be included in the current evaluation.

Therefore, it has been widely agreed that once the mining operation resumes at Pau-a-Pique a staged approach will be adopted and implemented regarding extraction of stopes using Avoca choke blasting. P&E recommends that initially, significant attention must be dedicated to stope drilling and blasting practices mainly around the drill pattern, hole spacing, firing practice, energy distribution per hole and per blast, and interdepartmental accountability/responsibility for the entire process.

P&E recommends the first few stopes be treated as test stopes with a new drill pattern and spacing to be developed jointly by Aura technical services and operations management by analysing in greater detail the contributing factor to the stope over-break in the HW and FW area. It is possible that a larger hole spacing and pattern is required. This will subsequently put less explosive energy into the blast holes and possibly deliver the same fragmentation results due to the fact that the majority of the ore across the mine is classified as geotechnical type IV and type V with only the HW and FW varying and improving with depth.

The fan drilling technique tends to put higher energy at the toe of drilled holes and when blasted these holes impact the stability of stope walls differently than parallel drill holes. P&E recommends that stope drilling be undertaken using a parallel drilling technique as much as possible.

Another initiative that can be assessed and implemented if it demonstrates its viability is a smooth wall blasting technique in order to control the HW and FW over-break. Sometimes it can be costly but the reduction in dilution could very easily offset the increase in cost per tonne of stoping.

P&E recommends that the technical services department should develop, implement and closely monitor a stope extraction process. This should include, but not be limited to, stope design, drilling and blasting, stope closure, and stope reconciliation using a CMS survey technique to properly determine the over/under break of the HW and FW.

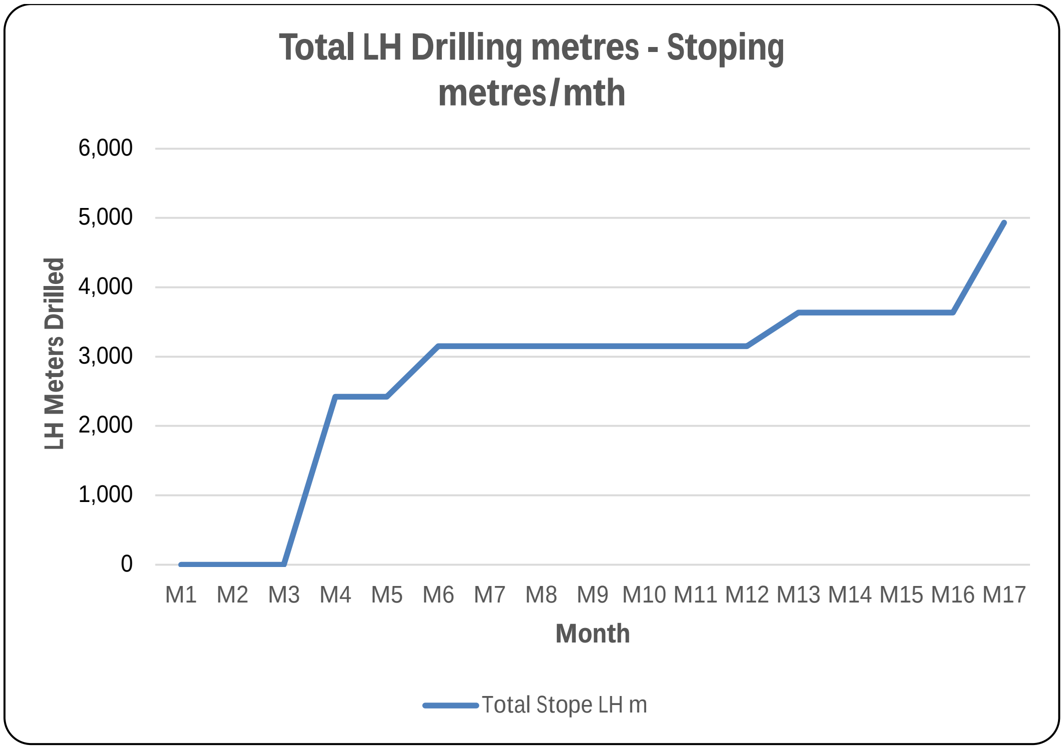
16.3.6 Drilling

Production drilling at Pau-a-Pique will be carried out predominately by a “down-the-hole” longhole drill rig already onsite and readily available with minimal mechanical refurbishing work. This rig drills holes of up to 30 m in length, and can drill several hundred metres of production holes per day.

Production drilling will mainly be undertaken with down-hole production drill rigs, drilling 64 mm diameter holes from a single drill drive located in the hanging wall of the stopes using fan rings. Up-holes are planned to be drilled using the same drill rig, drilling 64 mm upper holes mainly when attempting to extract Area 7 and 8 stopes as well as for existing and future sill pillars if deemed economic and safe to be mined. A second top-hammer rig will be used for cable bolting and miscellaneous drilling work i.e drain holes, service holes, etc.

Estimated production drilling per month is presented in Figure 16.43.

Figure 16.43 Production Drill Metres Per Month



16.3.7 Blasting

Special attention is required to the planning of blasting due to the potential of excessive dilution if not controlled and adjusted properly, especially from stopes walls (HW/FW) and unconsolidated backfill. A dedicated crew will transport and charge the blast holes according to a charge instruction signed-off by planning, geotechnical, geology and operations departments. This approach is recommended by P&E as an immediate measure to control/minimize stope dilution and to engage each department in the operations process. Explosives will be stored on site in secure explosive magazines in accordance with Brazilian regulations.

The primary blasting agent to be used at Pau-a-Pique is ANFO. The explosive is air-loaded into the drilled blast holes using a MEMU charge-up wagon. Cartridge boosters, Nonel and two (2) mantopim safety fuse assembly detonators will be used to initiate blasts.

The blast is fired at the end of the shift only from underground using the mantopim safety fuse assembly system. In this fuse assembly system, the detonators are tied in with detonating cord, and this, in turn is connected to Nonel in-the-hole detonators. Once all personnel are evacuated from the mine the shift boss/shift supervisor will give the go-ahead to fire the blast from underground. This is a control process and only when clearance is given by the person in charge will the blasters initiate the blast.

16.3.8 Loading and Hauling

Daily ore production at Pau-a-Pique is projected at around 750 tpd when in full production. Production levels will initially be approximately 200 tpd in the first year, increase to 400 tpd in the second year and 750 in the third year. This requires 1 to 2 active stopes to maintain these production levels, as well as development ore. All mucking and loading underground is done using 10 tonne LHD underground loaders. Rock is loaded into 30 tonne ADT surface haul trucks (2 to 3 required), and is then hauled to the surface via access declines.

A long haul (47 km) to the Ernesto mill will take place using the same 30 tonne ADT surface trucks in accordance with the overall milling plan and is recommended to take place in convoy during daylight (increased visibility) due to the fact that the road crosses cattle farm land.

16.3.9 Backfill

All stopes at Pau-a-Pique will be filled with development waste rock. This also provides short cycle times and production efficiencies when removing waste from underground. Minimal waste is hauled to the surface. Some waste may be screened at the Ernesto CRF plant and hauled back to Pau-a-Pique to be used as underground roadbed aggregate as well as to maintain the 47 km haul road in working condition during mine life, subject to geochemical testwork and suitability. Taking into account that the mine is located in sub-tropical weather conditions and substantial rainfall occurs it is paramount that the surface access road is maintained in good order. This is the only access to the mine and any disruptions in materials supply and haulage of ore will impact the economics of the Project.

Within the actual economic assessment, it is assumed that there will be no access restrictions or disruptions to mine supplies or ore haulage during the mine life.

16.3.10 Ground Support

Ground support is a high priority at Pau-a-Pique, not just in terms of safety, but also to assure continuity of operations. It is recommended that a site-wide ground control regime be set up by geotechnical staff prior to recommencement of mining operations. This regime classifies all headings and drives in terms of their mining rock mass rating, blast damage, stress change, life of excavation, usage frequency and excavation size and will have independent ground support standard design which will only change if geotechnical conditions dictate.

All main access development will be reinforced by resin rebar bolts (2.4 m long) at the back and swellex bolts (2.4 m long) on the walls with a pattern of 1.5 m by 1.5 m and welded wire mesh. When poor ground conditions are encountered shotcrete will be applied and the thickness will increase to ensure the stability.

Waste level accesses, cross cuts in waste, muck bays, sumps, electrical cut-outs, and other waste development will be will be reinforced by resin rebar bolts (2.4 m long) at the back and swellex bolts (2.4 m long) on the walls with a pattern of 1.8 m by 1.8 m and welded wire mesh. Shotcrete will be applied only to crosscuts and level access and thickness will be sized to ensure stability as recommended by the site geotechnical expert if ground conditions dictate.

All ore drive development will be reinforced by resin rebar bolts (2.4 m long) at the back and swellex bolts (2.4 m long) on the walls with a pattern of 1.5 m by 1.5 m, welded wire mesh and shotcrete in one or two layers depending on rock type encountered to ensure stope stability and continuous ore extraction.

Static 6 m, 9 m or 10 m cable bolts are also used to provide ground support where large unstable wedges are formed and in all ore drives to ensure the integrity of ore access. Where back failure is predicted, cable bolts will be installed to reduce dilution and maintain the integrity of those accesses once the stope below has been extracted. Generally, all intersections are reinforced by 6 m cable bolts. Holes for both bolt types will be drilled by a production longhole drill rig, and the cables installed by the service crew.

16.3.11 Knight Piesold Rock Mechanics Report

A report by Knight Piesold entitled “NB16-00368 PPQ Updated Underground Rock Mechanics Recommendations” dated November 7, 2016 follows.

16.3.11.1 Introduction

Knight Piésold Ltd. (“KP”) was retained to provide rock mechanics support for the resumption of underground mining at Pau-a-Pique. KP previously provided underground rock mechanics design input for the Pau-a-Pique Deposit (KP, 2016). The mine design has been updated over the past few months to reflect the results of an infill drilling program completed by Aura. At Aura’s request, KP has updated specific aspects of the underground rock mechanics design input to reflect the changes to the mine design.

The recommendations summarised are intended to support a resumption of underground mining operations at the Pau-a-Pique Deposit and are pragmatic in nature. They were not completed to a consistent level of design. They should be refined as additional data is collected during mine operations.

16.3.11.2 Background

16.3.11.2.1Available Data

The recommendations are based on the following information:

- Proposed mine plan (provided by P&E Mining Consultants Inc. (P&E) in June 14, 2016)
- Previously mined stopes and development (provided by Aura in August, 2015)
- Ore body model (provided by Aura in September, 2015)
- Geotechnical drillhole database (developed by Yamana and reviewed by Aura in June, 2015)
- Geotechnical model (provided by Aura in June, 2015 and updated in September, 2015)
- Records of historical stope performance (provided by Aura in October, 2015)
- Previous geotechnical reports by AMC Consultants Pty. Ltd., Itasca Consulting Group and SBVS (provided by Aura in July, 2015)
- Observations made by senior KP personnel during site visits completed on June 21, 2015 and August 10, 2015.

The available data was reviewed and summarised in a PowerPoint presentation.

16.3.11.2.2Geology

The Pau-a-Pique Deposit consists of four ore lenses (P1 to P4) located in a shear zone at the contact between a meta-tonalite and a meta-conglomerate. The shear zone consists of weak sericite schist, mylonite and fault zones. The assemblage strikes approximately Northwest-Southeast and dips to the Southwest at 80°. Mineralization is associated with quartz veining within the shear zone. The main lithologies at the Pau-a-Pique Deposit are summarized below, arranged in order from the hangingwall (“HW”) to the footwall (“FW”):

- Meta-Tonalite
- Mylonite
- Sericite Schist
- Mylonite
- Meta-Conglomerate.

The meta-tonalite and meta-conglomerate become increasingly sheared as the mylonite is approached.

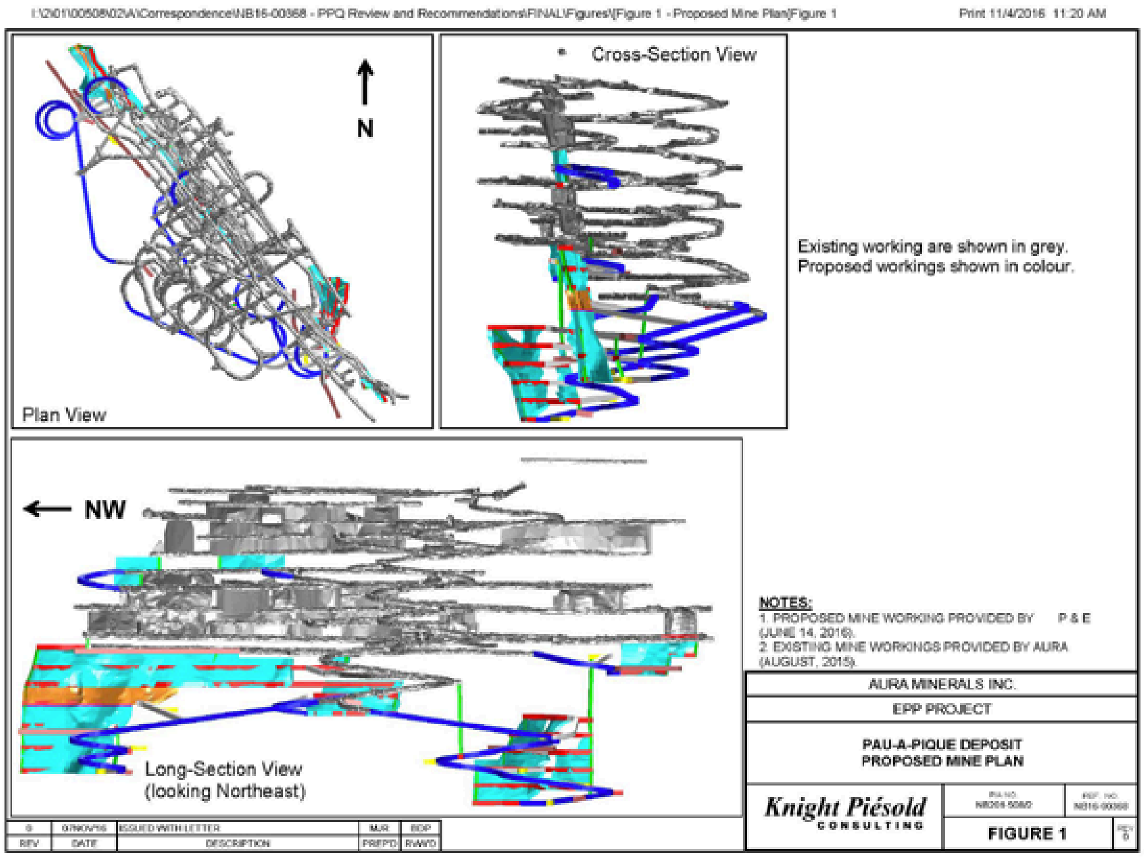
16.3.11.2.3 Mining Method

Several mining methods have been proposed or trialled at the Pau-a-Pique Deposit. The recommendations provided are based on the following mining strategy selected by Aura and P&E:

- Mining Method - Predominantly Avoca with choke blasting. One stope (Area 7) will be mined on retreat using uphole open stoping
- Stope Sizing - Stopes are approximately 4 to 10 m wide (HW to FW) with a sublevel spacing of 15 m to 21 m (resulting in stope heights of 19.5 to 25.5 m when the 4.5 high overcuts are considered). Undercuts/overcuts have a maximum span of 4.3 m. In areas where the ore body is wider than 4.3 m, fan drilling will be used to blast the full width of the orebody.
- Backfill - Predominantly uncemented rockfill. The uphole stope will not be backfilled
- Overall Access - The existing ramp from surface. The ramp is collared in the meta-conglomerate (FW) but crosses the ore body and is primarily located in the meta-tonalite (HW).
- Access Sizing - The ramp drives are 4.5 m wide and 4.5 m high
- Depth - Approximately 50 to 335 m below ground surface (400 to 115 m above sea level)

The proposed mine plan is shown on Figure 16.44. Note that KP refers to the open portion of an Avoca stope (i.e. the span between the active face of the stope and the backfill) as a panel. Also note that the Central stopes and the bottom P4 stope were subsequently removed from the mine plan.

Figure 16.44 PPQ Mine Plan



16.3.11.3 Review of 2015 Geotechnical Model

The geotechnical model developed for Pau-a-Pique in 2015 was reviewed as part of the previous study. The process through which the model was developed is summarized below.

Yamana technical staff collected RQD and RMR89 data for numerous infill and exploration drillholes at Pau-a-Pique.

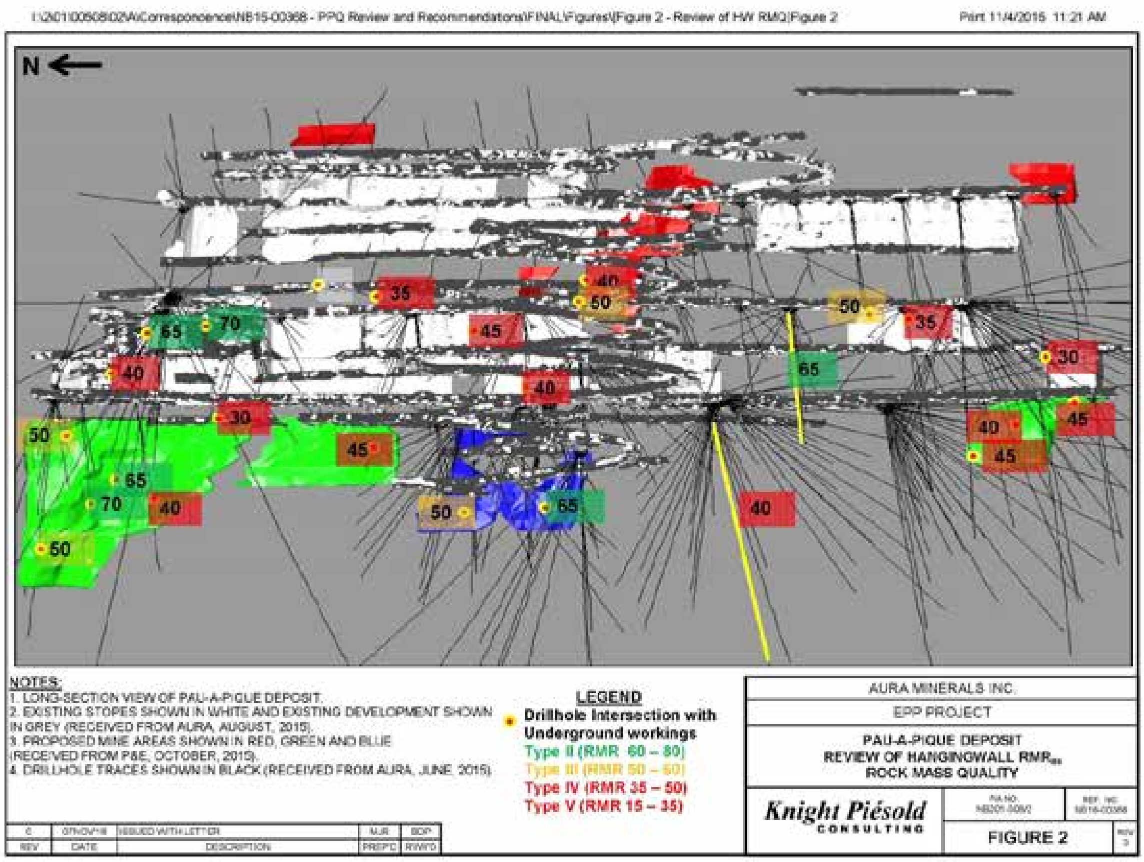
The drillhole data was reviewed by the site geotechnical engineer (Sr. Luis Navarro). Sr. Navarro found the data to be unreliable, with intervals of significantly different rock mass quality grouped into a single classification. As a result, select drillholes were re-evaluated. The rock mass quality was defined by RMR89 category (i.e. Type I through V, representing Very Good through Very Poor rock) based on a review of the core photos from these drillholes.

The revised drillhole data were then used to define solids representing the approximate distribution of the Type V (Very Poor) and Type IV (Poor) rock masses in the vicinity of the ore body. The rock mass outside of these solids was assumed to be Type III (Fair).

It is important to note that the extents of the 2015 geotechnical model developed by Aura are limited by the spatial distribution of the available drillhole data. The model is also focussed on the deposit below approximately 200 m below ground surface (250 m above sea level), as the majority of new mining is proposed for this area.

KP reviewed the 2015 geotechnical model using core photos from 26 drillholes distributed over the extents of the model. The objectives of the review were to assess the reliability of the model and to evaluate potential trends in the rock mass quality with depth. The review focussed on the rock mass quality in the immediate HW and FW of the existing and proposed stopes. The results of the review are summarized on Figure 16.45.

Figure 16.45 PPQ Rock Mass Quality



The rock mass classification approach used by KP results in RMR89 values 10 to 20 points higher than those used for the Type IV and Type V rock masses in the geotechnical model. Similar values were obtained for the Type III or better rock masses. The discrepancy appears to be primarily related to differences in the application of the groundwater rating. The model considered groundwater ratings ranging from Damp to Flowing. The approach preferred by KP is to assume a rating of Dry during rock mass classification and to account for groundwater during the subsequent stability analyses. This approach ensures that the effects of water are not accounted for twice. Based on the review, and for the purposes of the completed analyses, the range of RMR89 values associated with each rock mass quality Type were adjusted to bring them into line with the approach used by KP. Neither the model nor the underlying geotechnical logging data were modified.

The following comments on rock mass quality were provided based on the 2015 geotechnical model and the review of core photos by KP:

- The rock mass quality is variable within the immediate vicinity of the ore body

- The rock mass quality in the immediate (0 to 5 m) HW of the stopes is typically Type IV or Type V
- In some areas, such as the northern end of the deposit, there is an improvement in rock mass quality with depth. In these areas, the HW of the stopes is likely to be of better rock mass quality than Type IV or V
- In other areas, such as the southern end of the deposit, there is no improvement in rock mass quality with depth and the rock mass quality may even decrease. In these areas, the HW of the stopes is likely to continue to consist of rock mass quality Type IV or V.
- The geotechnical model was subsequently updated by Aura based on the results of additional infill drilling completed in 2016. The updated model has not been reviewed as part of this work. It is understood that it was developed using the same methods as the original model and has been expanded to cover all of the proposed new mining areas.

16.3.11.4 Review of Historical Mining

The Pau-a-Pique Deposit was mined by Yamana between 2012 and 2014. Several mining methods were trialled, including:

- Mechanized Sublevel Shrinkage Stoping - This method was proposed during the initial feasibility study completed by Yamana. It is not clear if it was trialled at the start of mining. The method was ultimately rejected due to concerns over dilution.
- Modified Avoca - This method was used for the majority of mining at Pau-a-Pique. This method was associated with significant dilution (exceeding 100%). Initially, the strike length of the panels was not controlled. With the arrival of Sr. Navarro in July, 2014, the length of each panel was defined based on the anticipated rock mass conditions in the panel HW and FW. This helped to manage dilution. The sublevel spacing was also varied between approximately 15 and 25 m in an attempt to further manage dilution.
- Avoca with Choke Blasting - The final method trialled by Yamana. This method was used during the last five months of operations at Pau-a-Pique. The sublevel spacing was initially 22 m and was reduced to 12 m for the final three panels in an effort to reduce dilution.

Limited mining and rock mass performance records are available for the historical mining:

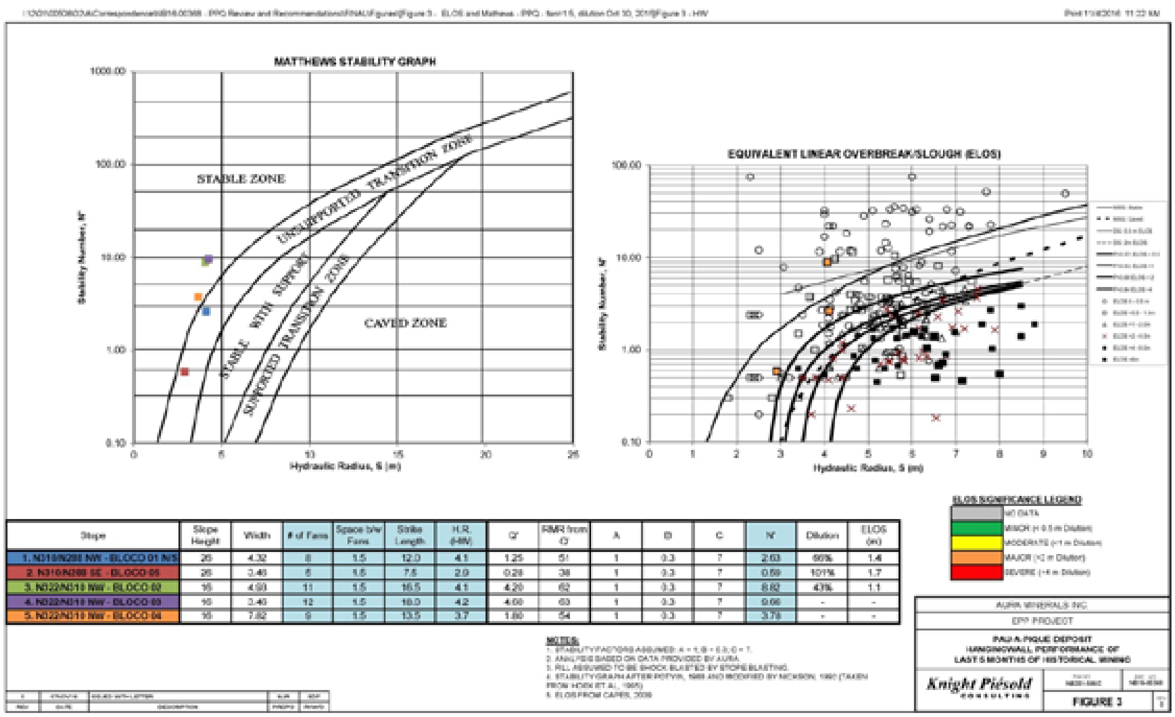
- The planned stope shapes are available for all of the mined stopes
- Cavity Monitoring Scans ("CMS") are available for 16 panels. The majority of the panels for which a CMS is available were mined using modified Avoca, but Avoca with choke blasting was used for at least one of them. Dilution and recovery were calculated by Yamana for each of these panels using the CMS.

Estimates of the achieved panel dimensions and the open span between the active face and the backfill, as well as qualitative assessments of dilution and the HW rock mass quality, were developed by Sr. Navarro for 42 of the modified Avoca panels.

The historical stope performance was reviewed in order to provide guidance on likely future stope performance. The review focussed on the five panels mined during the last five months of operations, as this was the period when Avoca with choke blasting was trialled. It is important to note that CMSs and dilution estimates are only available for three of the five panels. The performance of earlier stopes was also reviewed, but the data could only be used qualitatively since the records were incomplete.

The historical performance of the stopes was evaluated using the Mathews Stability Graph (after Potvin, 1988 and modified by Nickson, 1992). Note that the default design zones on the graph (e.g. Stable with Support) are not necessarily valid for the rock masses at Pau-a-Pique, but the principles underlying the method are applicable. The limited historical data suggests that the boundary between the "Stable Zone" and the "Unsupported Transition Zone" divides the panels that have performed relatively well from those that have performed relatively poorly (Figure 16.46.

Figure 16.46 PPQ Stope Stability Analysis



The historical records of dilution were evaluated using the Equivalent Linear Overbreak/Slough ("ELOS") relationships developed by Capes (2009). The panels mined during the last 5 months of operations had, on average, 1.5 m of dilution on each of the HW and FW surfaces. Less than 1 m of dilution was predicted for these stopes using the ELOS relationships. The results of the evaluation suggest that typical ELOS relationships will likely underestimate actual dilution. This may be due to the tendency of the stope HW and FW to break back to the contact between the Type IV and Type III rock masses.

16.3.11.5 Design Input

16.3.11.5.1Open Stope Dimensions

Achievable open stope dimensions for Avoca with choke blasting were evaluated based primarily on the performance of the panels mined during the last five months of operations. The boundary between the "Stable Zone" and the "Unsupported Transition Zone" on the Mathews

Stability Graph was used to evaluate achievable stope dimensions based on the results of the review of the historical mining.

The current mining strategy incorporates cable bolts in the HW and FW of the ore drives to limit the effective HW and FW spans. These cable bolts were not always effective during the previous operations at Pau-a-Pique. In order to evaluate the sensitivity of the stope dimensions to the effectiveness of the cable bolts, several scenarios were considered and the effective HW and FW spans varied accordingly. The analyses are summarised in Table 16.37.

TABLE 16.37
SUMMARY OF RECOMMENDED PPQ STOPE GEOMETRY

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CONSULTING

TABLE 1

AURA MINERALS INC.
EPP PROJECT

PAU-A-PIQUE DEPOSIT
SUMMARY OF RECOMMENDED STOPE GEOMETRY

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Sub-Level Spacing	Rock Mass Quality	Achievable Strike Length (m)		
		HW / FW Cable Bolts Effective	HW / FW Cable Bolts not Effective	HW / FW Cable Bolts Partially Effective (Recommended Design Case)
15 m	V	4.5	4.5	4.5
	IV	13.5	9	9
	> IV	37.5	15	18
21 m	V	4.5	4.5	4.5
	IV	9	7.5	7.5
	> IV	18	13.5	15

I:\201\00508\02\A\Correspondence\MB16-00368 - PPQ Review and Recommendations\FINAL\Tables\[Table 1 Stope Dimension Summary June 17, 2016.xlsx]Table 1

NOTES:

1. ACHIEVABLE STRIKE LENGTHS BASED ON STABILITY GRAPH METHOD AND A BACK-ANALYSIS OF THE LAST FIVE MONTHS OF MINING AT PPQ.
2. HW / FW CABLES BOLTS REFER TO 3 ROWS OF 10 m LONG CABLE BOLTS INSTALLED IN THE HANGINGWALL AND FOOTWALL OF THE STOPE ORE DRIVES.
3. THE STOPE HEIGHT WAS VARIED BASED ON THE EFFECTIVENESS OF THE HW / FW CABLE BOLTS. IF THE CABLE BOLTS ARE EFFECTIVE, THE EFFECTIVE STOPE HEIGHT WAS REDUCED TO THE SUB-LEVEL SPACING LESS THE HEIGHT OF THE UNDERCUT (4.5 m). IF THE CABLE BOLTS ARE NOT EFFECTIVE, THE EFFECTIVE STOPE HEIGHT WAS INCREASED TO THE SUB-LEVEL SPACING PLUS THE HEIGHT OF THE OVERCUT (4.5 m). IF THE CABLE BOLTS ARE PARTIALLY EFFECTIVE THE EFFECTIVE STOPE HEIGHT WAS SET AT THE SUB-LEVEL SPACING.
4. STRIKE LENGTHS REPORTED IN MULTIPLES OF 1.5 m FANS OF BLASTHOLES.

The following panel strike lengths are thought to be achievable based on past performance, and the current understanding of the rock mass quality associated with each rock mass category (Figures 16.47 and 16.48). The panel strike lengths assume that the cable bolts installed in the HW and FW of the ore drives are partially effective.

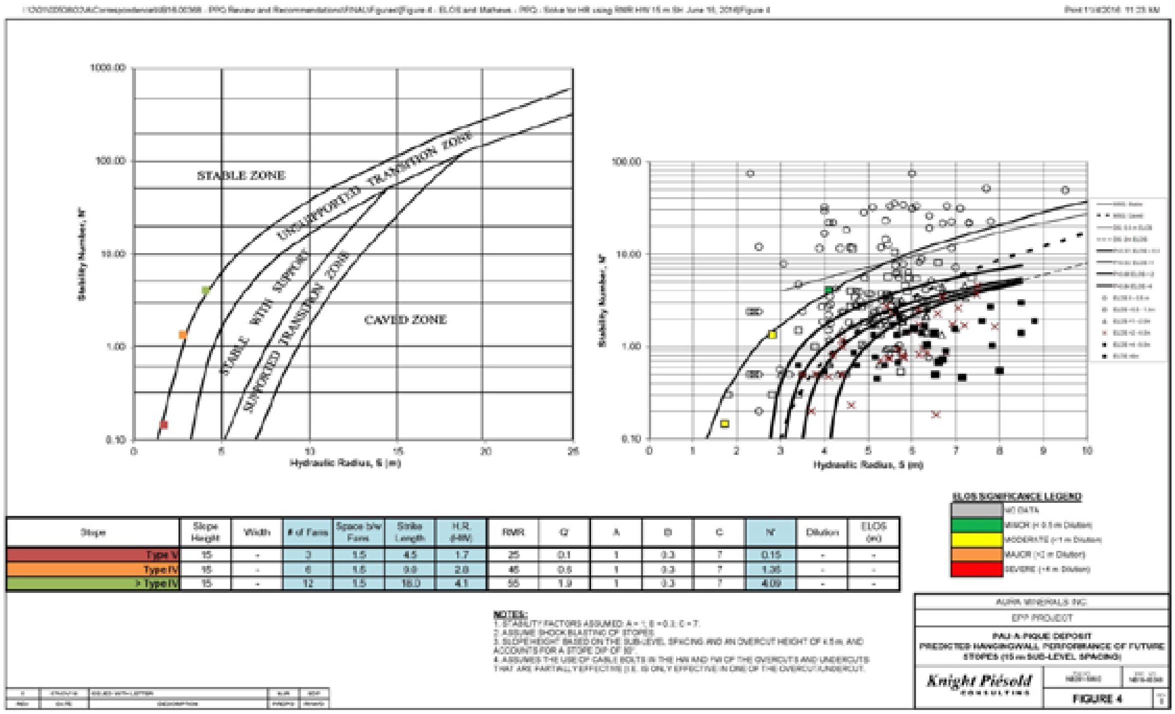
For a 15 m sublevel spacing:

- HW/FW in Type V Rock Mass - 4.5 m
- HW/FW in Type IV Rock Mass - 9 m
- HW/FW in Type III or Better Rock Mass - 18 m

For a 21 m sublevel spacing:

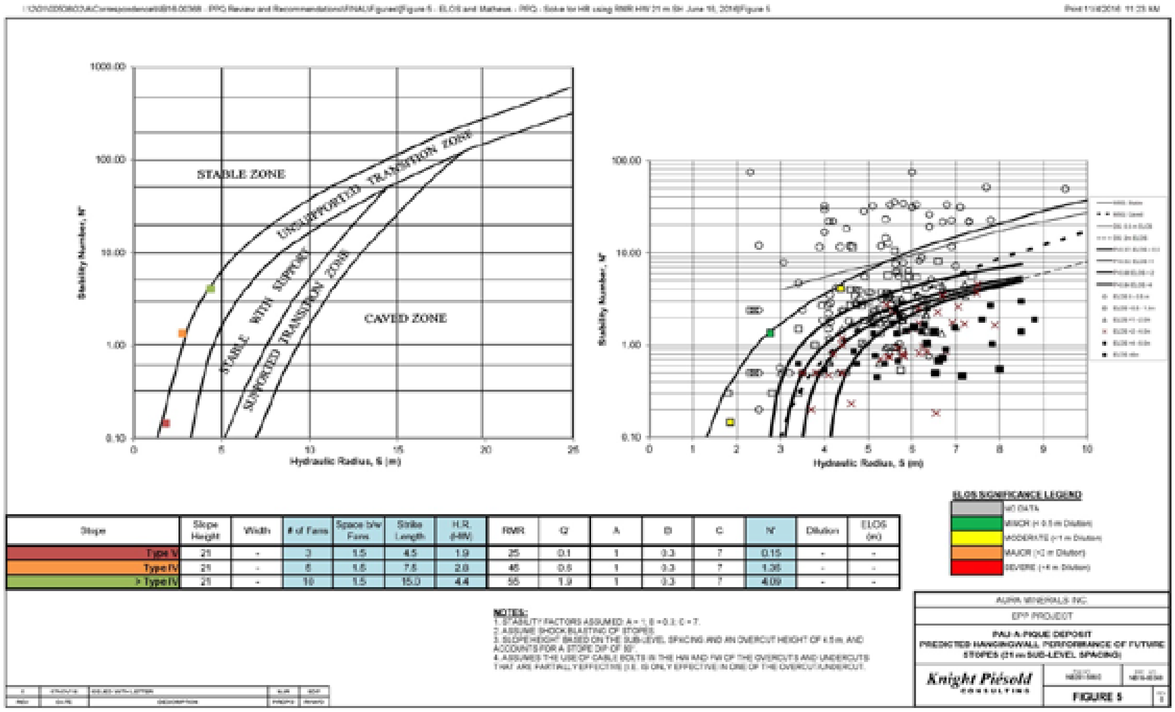
- HW/FW in Type V Rock Mass - 4.5 m
- HW/FW in Type IV Rock Mass - 7.5 m
- HW/FW in Type III or Better Rock Mass - 15 m

Figure 16.47 Predicted Hangingwall Performance of Future PPQ Stopes, 15 m Sublevel Spacing



Stope	Stope Height	Width	# of Fans	Space b/w Fans	Strike Length	H.R. (HW)	RMR	Q'	A	B	C	N'	Dilution	ELOS (m)
Type V	15	-	3	1.5	4.5	1.7	25	0.1	1	0.3	7	0.15	-	-
Type IV	15	-	6	1.5	9.0	2.8	45	0.6	1	0.3	7	1.36	-	-
> Type IV	15	-	12	1.5	18.0	4.1	55	1.9	1	0.3	7	4.09	-	-

Figure 16.48 Predicted Hangingwall Performance of Future PPQ Stopes, 21 m Sublevel Spacing

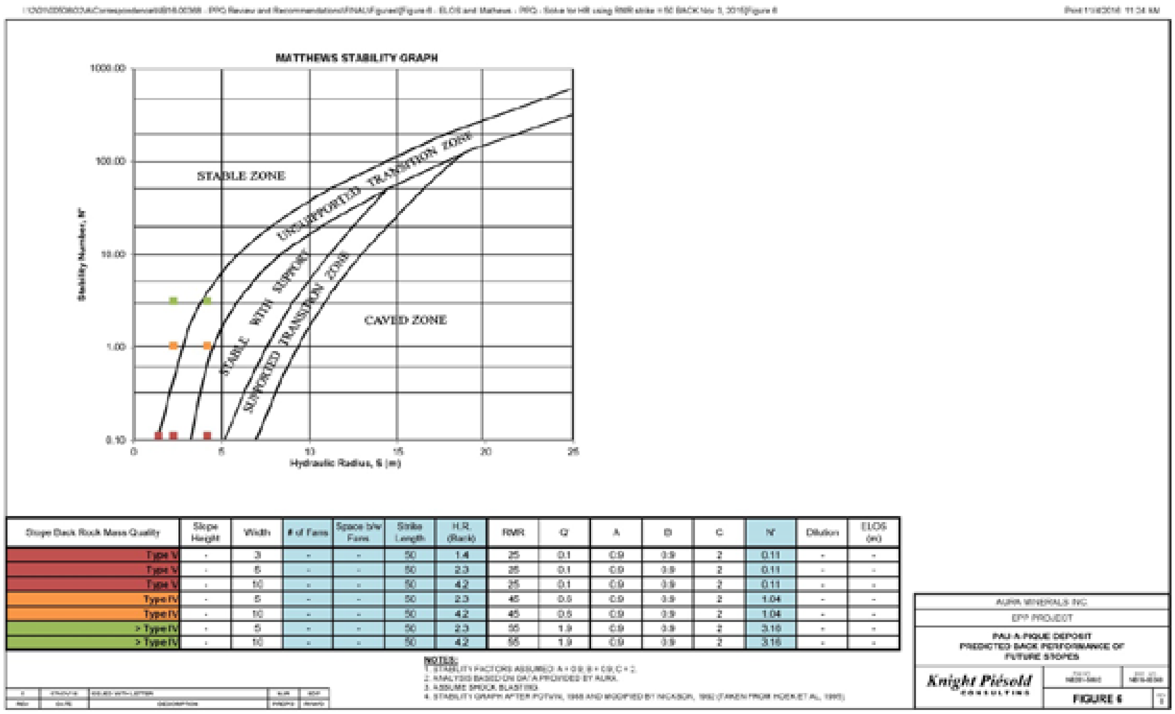


Achievable back spans for the stopes were also evaluated using the relationship between rock mass quality and span underlying the Mathews Stability Graph. This approach was supplemented with the empirical design method developed by Grimstad and Barton (1993) for the overcuts and undercuts. This method is most relevant for drives with man-entry and was used as it more reliably accommodates lower rock mass qualities. The average width of the stopes proposed for Pau-a-Pique range from 2.1 to 8.6 m. As a result, two main cases were considered: a 5 m back span and a 10 m back span. Approximately 70% of the stopes could be mined with a back span of 5 m or less. The results of the two approaches are summarized below:

The Mathews Stability Graph approach (Figure 16.49) suggests that a 5 m back span should be achievable in all rock masses, though it is probably the upper bound for the Type V Rock Mass using standard ground support. A 10 m back span is not recommended based on historical practice.

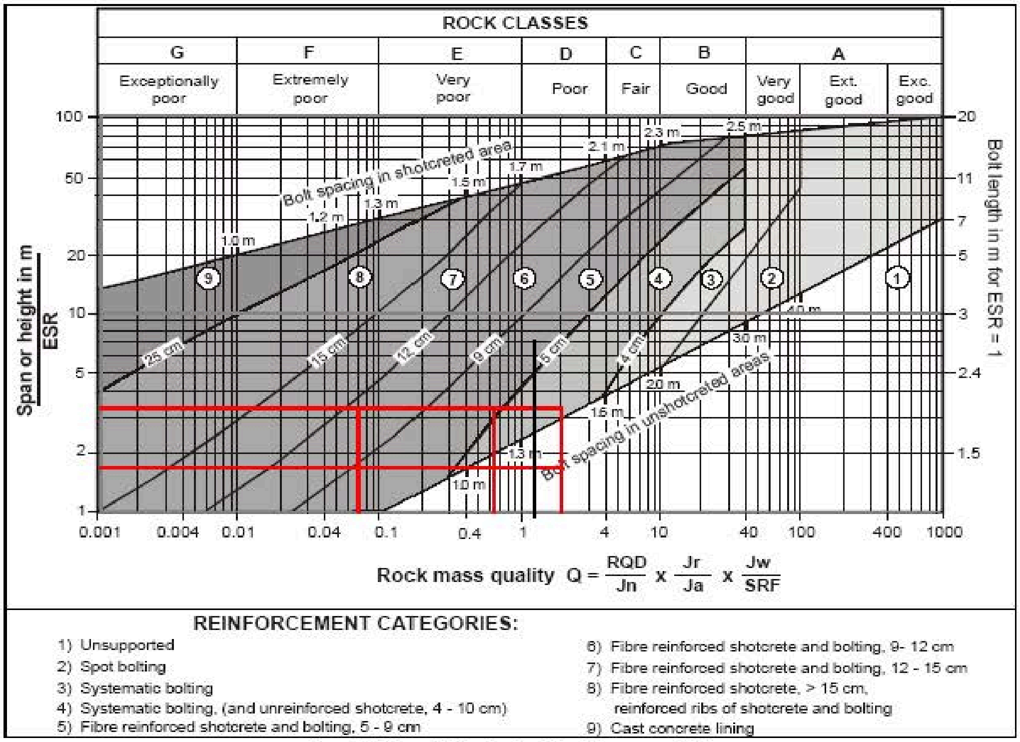
The Barton and Grimstad approach (Figure 16.50) suggests that a 5 m back span is achievable in all rock masses, but would require the use of 2 to 3” of shotcrete in the Type IV and V Rock Masses.

Figure 16.49 Predicted Back Performance of Future PPQ Stopes



Stope Back Rock Mass Quality	Slope Height	Width	# of Fans	Space b/w Fans	Strike Length	H.R. (Back)	RMR	Q'	A	B	C	N'	Dilution	ELOS (m)
Type V	-	3	-	-	50	1.4	25	0.1	0.9	0.9	2	0.11	-	-
Type V	-	5	-	-	50	2.3	25	0.1	0.9	0.9	2	0.11	-	-
Type V	-	10	-	-	50	4.2	25	0.1	0.9	0.9	2	0.11	-	-
Type IV	-	5	-	-	50	2.3	45	0.5	0.9	0.9	2	1.04	-	-
Type IV	-	10	-	-	50	4.2	45	0.5	0.9	0.9	2	1.04	-	-
> Type IV	-	5	-	-	50	2.3	55	1.9	0.9	0.9	2	3.16	-	-
> Type IV	-	10	-	-	50	4.2	55	1.9	0.9	0.9	2	3.16	-	-

Figure 16.50 Predicted Back Performance of Future PPQ Stopes Using Grimstad & Barton



Span/ESR and Q Determination Summary Table

Stope Back Rock Mass Quality	Span (m)	ESR	Span/ESR	RMR	Q'	Jw	SRF	Q Low	Q High
TYPE V	5	3	1.7	25.0	0.07	1	1	0.07	0.07
	10	3	3.3	25.0	0.07	1	1	0.07	0.07
TYPE IV	5	3	1.7	45.0	0.64	1	1	0.64	0.64
	10	3	3.3	45.0	0.64	1	1	0.64	0.64
TYPE III	5	3	1.7	55.0	1.95	1	1	1.95	1.95
	10	3	3.3	55.0	1.95	1	1	1.95	1.95

NOTES:

1. ESR = 3 WAS USED (TEMPORARY MINE OPENING).
2. Q' = RQD/Jn x Jr/Ja.
3. J_w = 1.0 (DRY).
4. SUPPORT GRAPH AFTER GRIMSTAD AND BARTON, 1993 (TAKEN FROM PALMSTROM AND BROCH, 2006).

REV	DATE	DESCRIPTION	PREP'D	RVW'D
0	01NOV'16	ISSUED WITH LETTER	MJR	BDP

AURA MINERALS INC.

EPP PROJECT

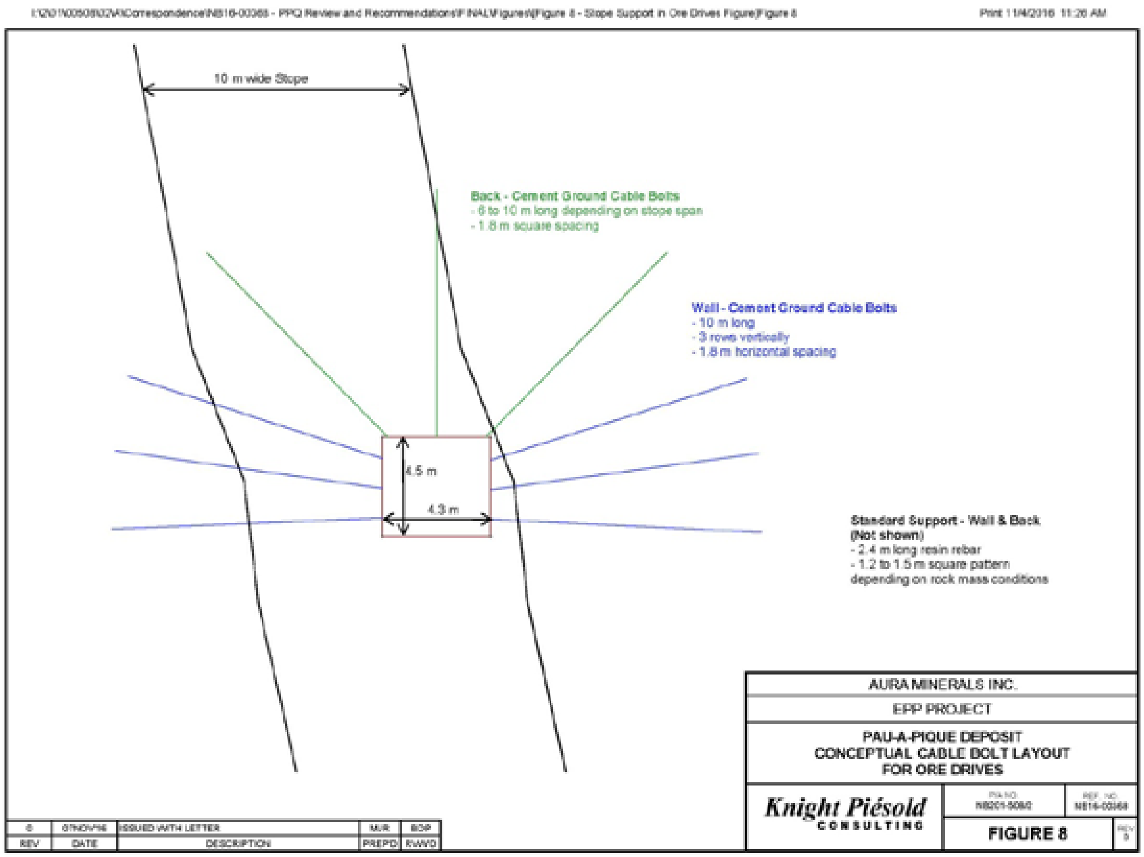
PAU-A-PIQUE DEPOSIT
PREDICTED BACK PERFORMANCE OF FUTURE STOPES USING GRIMSTAD & BARTON

Knight Piésold CONSULTING

FIGURE 7

The current mining strategy proposed by P&E limits the back span of the ore drives to 4 m. In areas where the ore body is wider than 4 m, fan drilling from a 4 m overcut will be used to blast the full width of the ore body. In these cases, additional ground support (i.e. cement-grouted cable bolts) will be installed in the back of the overcut, and the ore will be mucked using remotely operated equipment. A conceptual layout for the cable bolts is shown on Figure 16.51.

Figure 16.51 PPQ Conceptual Cable Bolt Layout For Ore Drives



16.3.11.5.2Expected Dilution

The expected dilution has been estimated based on the historical stope performance and proposed mining strategy using the ELOS method (Capes, 2009). The unplanned dilution for the panels mined during the last 5 months of operations ranged from 44% in the Type III rock mass to 101% in the Type IV rock mass. The panels had, on average, 1.5 m of dilution on each of the HW and FW surfaces. This is thought to be a reasonable estimate of average dilution for the proposed future mining. It is important to note that this estimate assumes both a regular stope geometry and that the ore drives do not undercut the HW or FW.

The contact between the Type IV and Type III rock masses often marked the limit of dilution for earlier panels. As such, the extents of the Type IV rock mass in the HW and FW of the panels are thought to represent a likely upper bound on dilution in regions of poor rock mass quality. The following considerations are expected to reduce dilution:

- Improving the placement of the ore drives
- Improving ground support practices
- Improving drilling and blasting practices
- Controlling the panel span
- Limiting the stand-up time of the panels

These effects are difficult to quantify, and any improvement is expected to be incremental rather than revolutionary. Improvements in the order of 5 to 10% have been discussed with Aura and P&E and are considered a reasonable starting point for planning purposes.

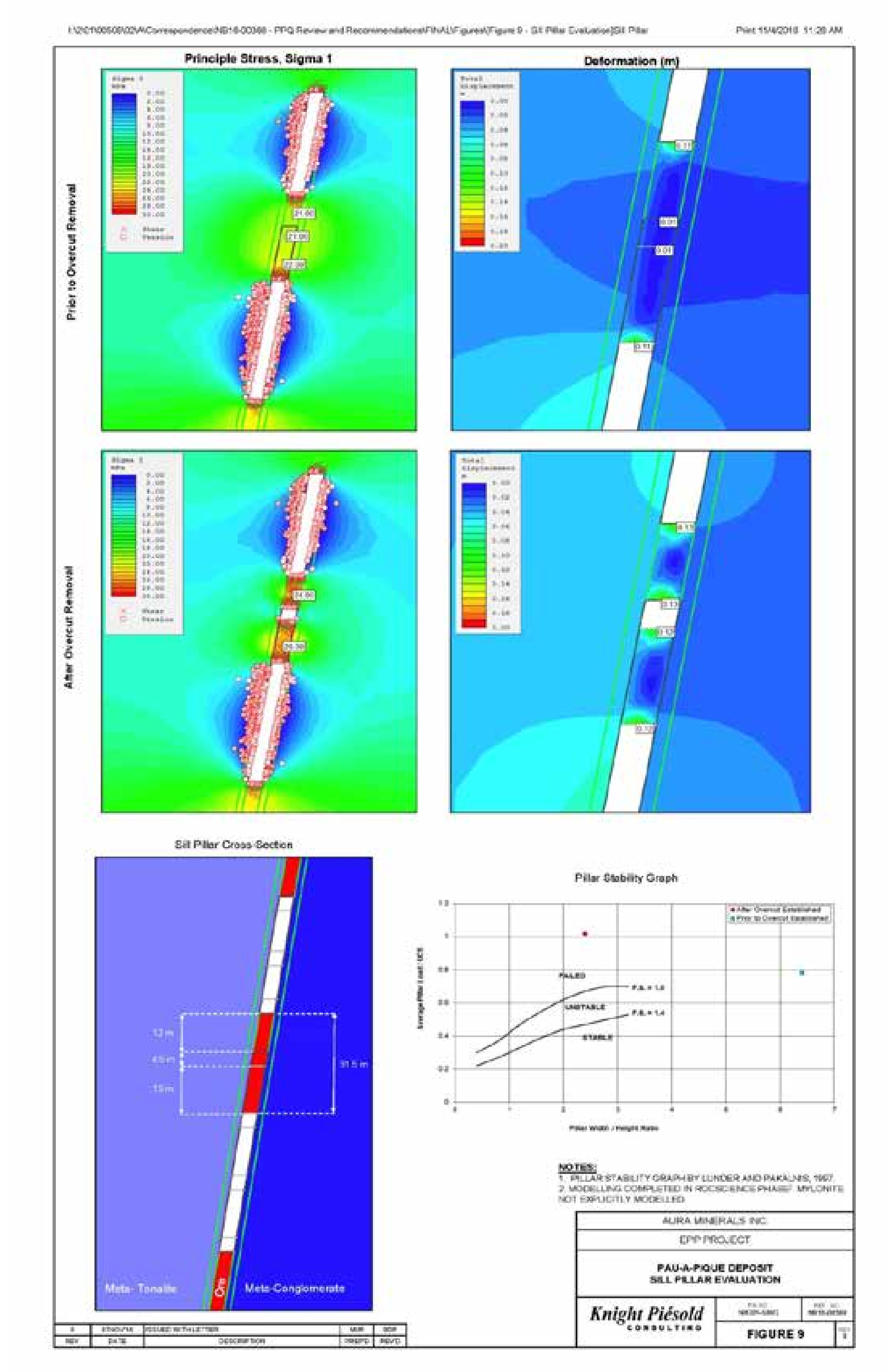
16.3.11.5.3Sill Pillar

A permanent sill pillar has been incorporated into the proposed mine plan between 220 masl and 208 masl at the northwestern end of the mine. The ore body is approximately 5 m wide in this area. The sill pillar is intended to allow the concurrent mining of multiple blocks at different elevations by preventing the migration of uncemented backfill from a previously mined portion of the overlying mining block into the underlying mining block. As part of the proposed mining sequence, the overlying mining block will be completely mined out before mining of the underlying mining block begins.

The expected performance of the sill pillar was evaluated based on previous mining experience at Pau-a-Pique, the results from 2D perfectly plastic numerical models developed in Phase2, and the empirical Pillar Stability Graph developed by Lunder and Pakalnis (1997). The results of numerical analyses previously completed by Itasca were also reviewed.

The evaluation suggests that the sill pillar will yield once the overcut at 203.5 EL is established (Figure 16.52). The pillar is expected to deform rather than fail in a brittle manner, and the main ground control challenge will likely be managing the deformation and associated ravelling during the extraction of the stope immediately below the sill pillar. Mining and ground support practices will need to be adjusted as needed to account for the observed excavation and pillar performance.

Figure 16.52 PPQ Sill Pillar Evaluation



The following considerations should be taken into account when developing the sill pillar:

- The ore drive at 203.5 masl should be established as late as possible in the mine sequence and the span of the drive be kept to a minimum within the proposed sill pillar. It will be important to employ development and mining strategies that maintain control of the overcut.
- Cable bolts should be installed in the back of the ore drive at 203.5 EL immediately after the installation of the primary support in order to help manage deformation of the sill pillar. The cable bolts will also resist a block failure along the mylonite by anchoring the ore to the more competent meta-tonalite and meta-conglomerate. The length and spacing of the cable bolt should be confirmed when the ore drive is established, but 6 m cables on a 1.8 x 1.8 m pattern could be used for planning purposes. 0-gauge straps should be installed between the cables. Depending on rock mass performance, there may be some benefit to installing yielding cables.
- Two multi-point borehole extensometers ("MPBXs") should be installed in the back of the ore drive 203.5 EL to monitor deformation within the sill pillar
- The stopes directly above and directly below the sill pillar should not be mined concurrently. This is consistent with the current mine design.
- The mining of the stopes immediately below the sill pillar should be completed in a timely way to minimize the stand-up time of the pillar

16.3.11.5.4Ground Support

The ground support recommendations are summarised in Table 16.38 and discussed below. Note that P&E did not follow all of KP's ground support recommendations, and in particular preferred the use of less expensive rebar rockbolts in place of swellex since the mine life is short.

Recommendations have been developed for six different categories of openings, depending upon their anticipated service life, span and importance (i.e., consequences if access to the excavation was interrupted). The categories are as follows:

- Ramp
- Waste Drive Intersections
- Waste Drives
- Ore Drive Intersections
- Ore Drives
- Stope Support in Ore Drives

TABLE 16.38
PPQ GROUND SUPPORT RECOMMENDATIONS BY KNIGHT PIESOLD

Knight Piésold
CONSULTING

TABLE 2

AURA MINERALS INC.
EPP PROJECT

FAU-A-PIQUE DEPOSIT
GROUND SUPPORT RECOMMENDATIONS FOR AVOCA AND UPHOLE STOPE MINING

Openings		Expected Rock Mass Quality	Standard Support										Additional Support Requirements
			Back					Walls					
			Bolts/Cables			Mesh	Shotcrete	Bolts/Cables			Mesh	Shotcrete	
Type	Span		Type	Length	Spacing			Type	Length	Spacing			
Ramp	4.5 m	Type II or Better	Resin Rebar	2.4 m	1.8 m	Galvanized Welded Wire Mesh	-	Resin Rebar	1.8 m	1.8 m (Within 1.5 m of floor)	See Additional Support Requirements	-	- Mesh to be installed to at least the first row of wall bolts - Check scaling/inspection program required if mesh does not extend to within 1.5 m of the floor
Waste Drive Intersections	6.4 m	Type III or Better	PM12 (Swellex)	2.4 m	1.8 m	Welded Wire Mesh	-	PM12 (Swellex)	1.8 m	1.5 m (Within 1.8 m of floor)	Welded Wire Mesh (Within 1.5 m of floor)	-	- Cement grouted cables to be installed on a case-by-case basis if large wedges are identified. Spacing not to exceed 2 m. Length not to be less than the span divided by 2.5 and should be adequate to stabilize the observed wedges. - Welded Wire Mesh should be installed on corners of intersections
Waste Drives	4.5 m	Type III or Better	PM12 (Swellex)	2.4 m	1.5 m	Welded Wire Mesh	See Additional Support Requirements	PM12 (Swellex)	1.8 m	1.5 m (Within 1.5 m of floor)	Welded Wire Mesh (Within 1.5 m of floor)	See Additional Support Requirements	- Shotcrete required as the drives approach the HW and FW contact of the orebody. Assume 5 m offset for costing purposes. - Shotcrete to within 1 m of the floor
Ore Drive Intersections	6.7 m	Type III or Better	PM12 (Swellex)	2.4 m	1.2 m to 1.5 m	Welded Wire Mesh	See Additional Support Requirements	Resin Rebar	2.4 m	1.2 m to 1.5 m (See Additional Support Requirements)	Welded Wire Mesh (See Additional Support Requirements)	See Additional Support Requirements	- 6-gauge straps between cables - 6-gauge straps wrapping corners - Shotcrete (5 cm) to be applied as required, to floor - Bolts and Mesh to extend to within 1 m of the floor when drive is parallel to schistosity and to extend to within 1.5 m of the floor when drive is crossing the schistosity
			Cement Grouted Cables	6 m	1.8 m								
		Type IV or V	PM12 (Swellex)	2.4 m	1.2 m	Welded Wire Mesh	10 cm	Resin Rebar	2.4 m	1.2 m (See Additional Support Requirements)	Welded Wire Mesh (See Additional Support Requirements)	5 cm (To Floor)	- 6-gauge straps between cables - 6-gauge straps wrapping corners - Bolts and Mesh to extend to within 1 m of the floor when drive is parallel to schistosity and to extend to within 1.5 m of the floor when drive is crossing the schistosity
			Cement Grouted Cables	6 m	1.8 m								
Ore Drives	4 m	Type III or Better	PM12 (Swellex)	2.4 m	1.2 m to 1.5 m	Welded Wire Mesh	See Additional Support Requirements	Resin Rebar	2.4 m	1.2 m to 1.5 m (Within 1.5 m of Floor)	Welded Wire Mesh (Within 1.5 m of Floor)	See Additional Support Requirements	- Shotcrete (5 cm) to be applied as required, to floor - 10 cm of Shotcrete to be applied on back if uphole will be drilled in the back (e.g. Area 7 of the mine plan)
		Type IV or V	PM12 (Swellex)	2.4 m	1.2 m	Welded Wire Mesh	5 cm	Resin Rebar	2.4 m	1.2 m (Within 1 m of Floor)	Welded Wire Mesh (Within 1 m of Floor)	5 cm (To Floor)	- 10 cm of Shotcrete to be applied on back if uphole will be drilled in the back (e.g. Area 7 of the mine plan)
Stope Support in Ore Drives	2 to 10 m	Type III, IV or V	Cement Grouted Cables (See Additional Support Requirements)	6 m to 10 m	1.8 m	-	-	Cement Grouted Cables	10 m	3 Rows Vertically 1.8 m Horizontally	-	-	- 6-gauge straps between cables - Cement grouted cables to be installed in the back of the ore drives in cases where the HW-FW span of the underlying stope is greater than 5 m - The length of the cement grouted cables should be adjusted on a case-by-case basis to ensure adequate support of the back, based on the full HW-FW span of the underlying stope

NOTES:
1. EXCAVATION HEIGHTS ARE 4.5 m.
2. SPANS BASED ON INSCRIBED CIRCLE METHOD.
3. THE EXPECTED ROCK MASS QUALITY IS BASED ON THE TYPES DEFINED BY THE GEOTECHNICAL MODEL. BASED ON A REVIEW OF THE CORE PHOTOS, TYPE II WAS CONSIDERED TO BE EQUIVALENT TO AN RMR RATING OF 55 OR BETTER.
4. THE ROCK MASS QUALITY OF THE BACK AND THE WALLS SHOULD BE ASSESSED SEPARATELY.
5. BOLTS TO BE KEPT [illegible] OFF FACE.
6. ALL WELDED WIRE MESH TO BE #6 GAUGE AND HAVE A 10 cm (4") APERTURE.
7. SHOTCRETE SHOULD BE APPLIED IN ADVANCE PRIOR TO INSTALLING MESH.
8. CEMENT GROUTED CABLES SHOULD HAVE A MODIFIED GEOMETRY (e.g. BULGE CABLES), WITH NO MORE THAN 0.5 m BETWEEN BULGES.
9. CEMENT GROUTED CABLES SHOULD BE INSTALLED ON ADVANCE (i.e. AT THE SAME TIME AS THE REBAR).
10. CEMENT GROUTED CABLES TO BE PLATED.
11. BOLT AND CABLE LENGTHS AND SPACINGS TO BE ADJUSTED ON A CASE BY CASE BASIS TO ACCOUNT FOR ADVERSE CONDITIONS.

The recommended support systems were based on Canadian mining practice and the following considerations:

- Ground Control Issues - At the planned mining depths, the main ground control issues are expected to be associated with controlling zones of reduced rock mass quality and recognizing and controlling wedges or blocks in the back and walls of the excavations.
- Minimum Support Requirements - The ground support recommendations represent the minimum support required, and are intended to safely accommodate the most commonly encountered ground control issues. Adverse ground conditions will require the use of additional ground support. The recognition of adverse ground conditions will be the joint responsibility of both engineering and operations staff. Based on the available rock mass information, enhanced support will likely be required within the mylonite, in larger span areas, and when random features are encountered that can form wedges.

Span and Tendon Length - Bolt lengths have been based on the following:

- The size and purpose of the proposed excavations
- The expected thickness of the mylonite
- Rules of thumb, experience, and accepted Canadian mining practices
- Additional Support Requirements - As previously noted, adverse ground conditions will require additional ground support. The support elements used will

vary, but are expected to include cement-grouted cables, shotcrete and 0-gauge mesh straps.

16.3.11.6 Review of Proposed Mine Plan

An underground mine design has been developed for the Pau-a-Pique Deposit by P&E. The underground mine design was subsequently reviewed by KP for rock mechanics considerations including:

- Agreement between the proposed mine geometry and the rock mechanics recommendations
- The presence of adverse underground geometries, such as 4-way intersections and small offset distances between openings
- Extraction sequencing.
- The review process builds off of reviews of the previous mine plan. There are a few outstanding issues that will need to be re-evaluated during the next stages of mine design and planning.

16.3.11.7 Summary and Recommendations

The overall mining and development strategy is believed to be suitable from a rock mechanics perspective given the expected rock mass conditions and the available historical data. The management of dilution will be critical to the success of the proposed mining approach. Dilution will need to be managed through a combination of the following factors:

- The placement of the ore drives
- Ground support practices
- Drilling and blasting practices
- Panel span
- Stand-up time of the panels

Based on the results of the completed work, the following recommendations are provided:

- The proposed stope dimensions are based on limited mining experience and will need to be refined during the initial mine operations. A key advantage of Avoca mining is that the panel strike length can be adjusted as mining progresses based on the observed panel performance. The following are recommended:

    - Geotechnical mapping should be undertaken during the development of the undercut and overcut for each stope. The results of the mapping should be used to plan the initial panel strike lengths.
    - The panel performance should be monitored using regular CMSs and possibly instrumentation. The collected data should be used to document the actual panel dimensions and dilution. The rock mass quality of the HW and FW and the time the panel remains open should also be documented.
    - The panel strike length should be adjusted based on the observed stope performance during mining
    - A final panel reconciliation should be completed for each stope and the design of future panels should be updated using the data collected from each stope

- The mine engineering department will need to include adequate ground control staff and resources to support mine development and operations
- Numerical stress modelling is recommended to evaluate the extraction sequence and the offset between the development and the ore body. The results of the modelling can also be used to confirm some of the inputs to the Mathews Stability Graph, as well as the stope sizing and ground support recommendations.
- Additional kinematic and numerical analyses are recommended to refine and confirm the ground support recommendations. For example, numerical modelling could be used to refine the length of the cable bolts recommended in the HW and FW of the overcuts and undercuts.
- An evaluation of the stability of the raises is recommended prior to their development.

16.3.12 Dewatering

The mine is located in the state of Mato Grosso where the average rainfall is 1,527 mm with the most rain during Jan-Apr and Nov-Dec, and a dry period during May – Oct. The driest month is July when in average there will be around 11 mm of rainfall as seen in Table 16.39.

TABLE 16.39
AVERAGE MONTHLY RAINFALL AT PAU-A-PIQUE

Month	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
mm	262	242	229	143	61	22	11	21	48	95	162	231

The process water for the mine will be supplied via existing piping and a surface water tank. Water discharged from underground operations will be discharged at a surface clearing sump and will return to underground for drilling usage. The ore moisture and underground rock will absorb approximately 5% water. The balance water from the mine would therefore be discharged in the existing water dam and later on into the surroundings once the quality of the water meets the state regulations (as per environmental closure and remediation plan). The mine service water reticulation system is designed using the ring main principal, similar to the power, to allow a certain degree of redundancy and reduce downtime due to any loss of water at any work area.

Used underground service water enters the dewatering system where it is settled before being pumped back to the surface mine water dam. The water is then recycled back into the mine as required. It has been estimated that three (3) new sumps are required when the main ramp advances below 320 m Elev. As per current design one sump will be constructed at 320 m Elev and two (2) additional sumps at the bottom of each orebody, one (1) on 145 m Elev for NW stopes and one (1) on 115 m Elev for P3 and P4 stopes.

16.3.13 Mine Power

The existing power supply off the national grid is envisioned to suffice the needs for power supply for underground and surface installations for Pau-a-Pique over the LOM. The surface HV power supply installation and substation is 13.8KVA @ 60Hz. The mine site has been designed to handle future loads as per the LOM plan and there is no need for additional power at Pau-a-Pique. The underground distribution network will be 1,000V with step-down transformers being used to connect to the surface HV power network.

16.3.14 Mine Production

16.3.14.1 Production Plan

The basis for the stope extraction at Pau-a-Pique is defined by longitudinal retreat using Avoca choke blasting and a bottom-up pattern with unconsolidated fill to maximize ore recovery and reduce dilution in the stopes. From this classification, production proceeds by:

- Mining a bottom stope;
- Filling the bottom stope void with unconsolidated backfill;
- Mining the next stope above;
- Filling the stope above with unconsolidated backfill; and
- Continue mining the stopes above until the panel has been exhausted (panels could be as high as 2 to 5 stopes).

On this basis, stope extraction can utilize only bottom-up extraction sequencing due to the mining method and backfill type restrictions.

The focus for production when full mining commences is the NW deep and P3 and P4 stopes. Commencing stope extraction from these ore bodies enables mine operations to extract stopes concurrently and increase the number of stopes available for extraction. The tonnes of ore mined per month is presented in Figure 16.53 and the monthly gold ounces produced is presented in Figure 16.54.

Figure 16.53 LOM Ore Production Profile

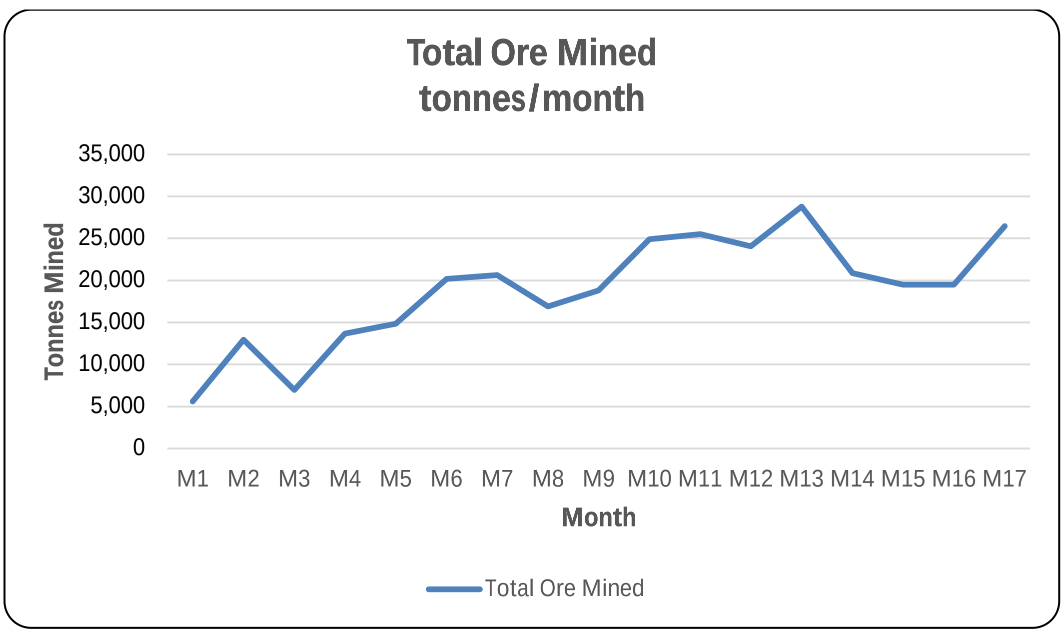
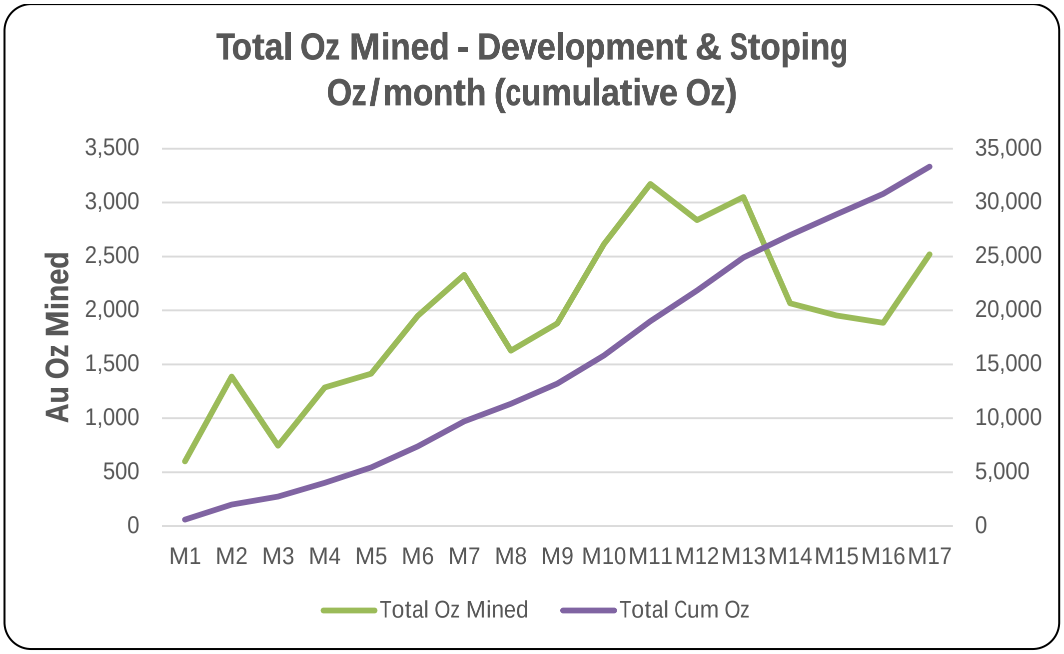


Figure 16.54 Total Au Oz Mined LOM



Initial ramp-up requires the majority of mine development to be completed in order to access all mining areas, this takes place mainly in the first year of operation. This allows continuous stoping ore production as a minimum to be mined at any period ensuring production ramps up to 750 tpd. The required mobile fleet is onsite and additional equipment will be sourced within Brazil. Lead time for each class of equipment is perceived very short after discussions with major equipment suppliers in the country. Lead time is not a schedule constraint to the Pau-a-Pique mine plan.

Development ore supplements the overall production tonnage in years one and two until all development is completed. Towards the end of the LOM ore production increases as the maximum numbers of stopes are available and are not dependent on development being completed before the next stope can commence.

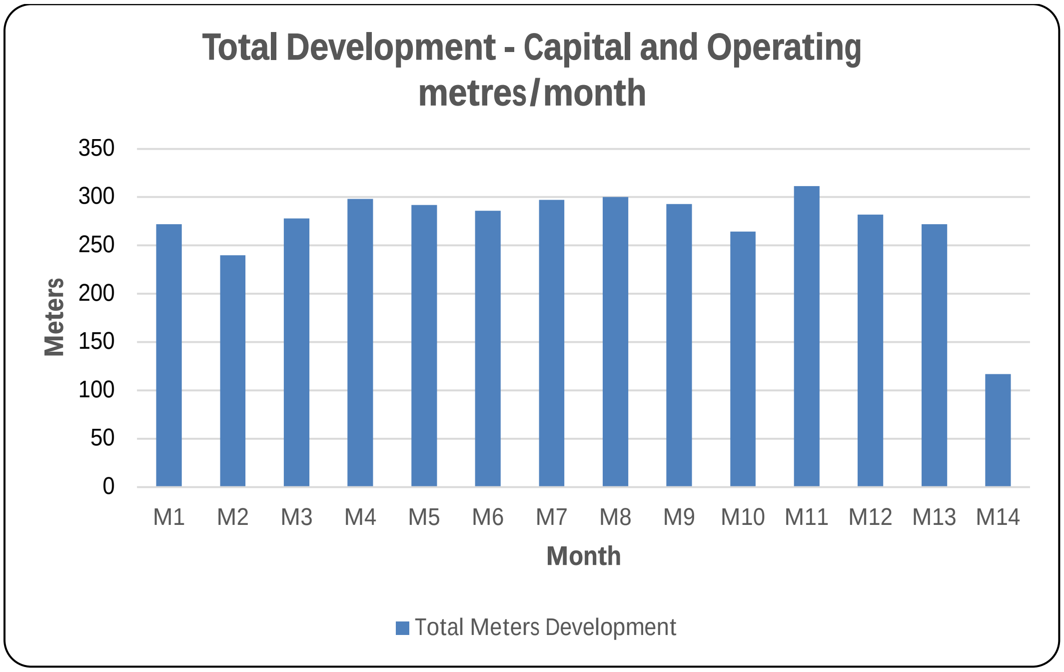
16.3.15 Development Plan

The Pau-a-Pique development plan will focus on developing the SE orebody decline, NW above the sill pillar and at the Area 7 and 8 stopes to open new stoping horizons and to enable continuous ore production at constant levels of approximately 750 tpd. NW deep and P3 and P4 orebody declines will be developed towards the end of mine life. Diamond drill drives will be prioritized to enable definition drilling and grade control modelling before ore level development and stoping activities commence.

Level development will be focused on the scheduled mining sequence to ensure that all waste and ore access drives, capital infrastructure, and slots are completed ahead of stoping requirements. The primary focus will be to develop and establish the SE and NW above the sill pillar to maintain stope ore production until the NW deep and P3 and P4 are in production. Simultaneous development of the levels above will be completed as late as possible (without delaying stoping) to allow resources to be diverted to the main decline below 220 m Elev to expedite the development of the next stoping horizon.

Waste rock development metres per month are presented in Figure 16.55.

Figure 16.55 LOM Horizontal Development Plan



16.3.16 Backfill Plan

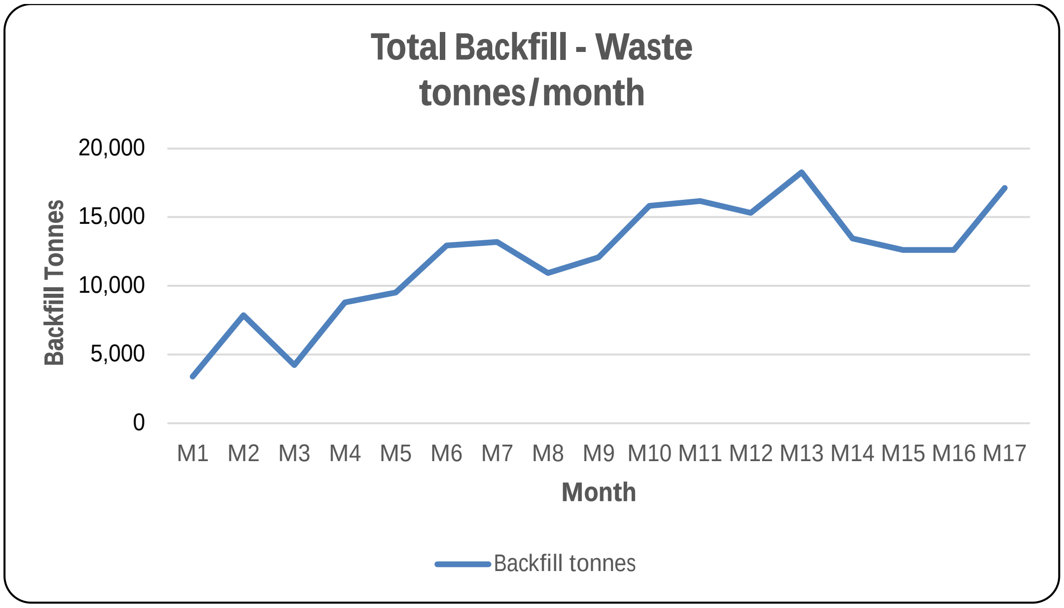
Unconsolidated rock fill is preferred as the most economic method of backfill at Pau-a-Pique. This is a continuation of the backfill practice at Pau-a-Pique that was done by previous owner Yamana. It has been determined by P&E in conjunction with KP that no major change is required regarding backfill approach or material, but changes are required in the way stope extraction is cycled and in the maximum designed open span.

Backfilling of stopes must be completed as quickly as possible so the next maximum span stoping panel is not delayed reducing total ore production and potentially having the void opened for too long which has been recognized as one of the major factors in stope dilution. Due to low ore production and the availability of three (3) 10t LHD's during the first two years it is envisioned that backfill rates will not affect stope cycle in any ways.

Back haulage from the Ernesto waste rock storage area must be timed and scheduled in more detail once the mine is back into operation to avoid stope cycle changes or production disruptions. P&E recommends that backfill requirements should be detailed and incorporated within the short and medium term detail mine plans to better determine waste haulage, underground storage locations and quantities, total material produced and shortfall, and appropriate timing for waste backhaul from the Ernesto waste rock storage area to Pau-a-Pique.

Monthly backfill requirements are presented in Figure 16.56.

Figure 16.56 Backfill LOM Requirements



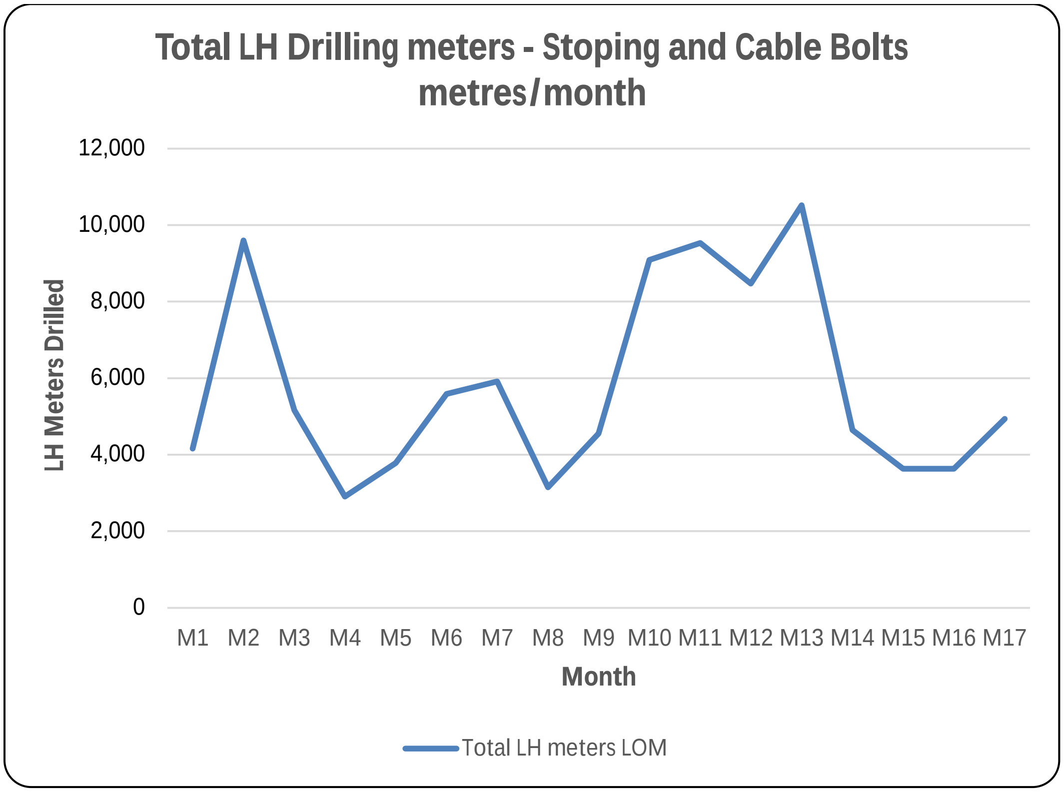
16.3.17 Longhole Drilling Plan

All longhole drilling will be completed using the existing two (2) top hammer ("TH") fan drilling equipment. Production drilling will require 15 m to 25 m long holes depending on the distance between levels at diameters up to 64 mm. These will be drilled from the upper level ore access or lower level as required.

One of the TH drills will be required for cable bolting and up-holes that may be required to be drilled at 89 mm as a result of larger diameter cable bolts recommended by the geotechnical consultant for additional support. Any additional longhole drilling i.e. service holes, drain holes, etc. that may be required by the operation could employ either drills, this depends largely on their availability at the time these holes will be required.

Figure 16.57 presents monthly longhole drilling requirements.

Figure 16.57 LOM Drilling Requirements for Longhole



16.3.18 UG Capital Development – Declines and Access

Mine capital development consists of the continued development of all ore bodies via the access decline, including new level accesses, muck bays, sumps, ventilation and other infrastructure. Considerable capital development is also required to set up Pau-a-Pique lower zones in the NW below the sill pillar and P3 and P4 ore bodies. A summary of capital development over the LOM is presented in Table 16.40.

TABLE 16.40
LOM CAPITAL DEVELOPMENT METRES

Capital Development	Metres
Access Ramp	1,040
Level Access Drifts	1,260
Vertical Development	181
Total Development	**2,481**

A schedule of development is presented in Table 16.41.

TABLE 16.41 **PAU-A-PIQUE DEVELOPMENT METRES PER MONTH**															
Development Metre Description/Month	**M1**	**M2**	**M3**	**M4**	**M5**	**M6**	**M7**	**M8**	**M9**	**M10**	**M11**	**M12**	**M13**	**M14**	**LOM**
Main Access Ramp	68	-	102	149	97	68	150	165	149	92	-	-	-	-	1,040
Level Access Cross-Cut – Waste	21	-	33	82	116	120	52	135	54	-	69	102	61	-	845
HW Diamond Drilling Drifts – Waste	55	-	-	55	45	25	-	-	55	-	-	-	-	-	235
Muck Bays, Sumps, Other Devel. Waste	24	-	-	-	-	12	12	-	-	24	32	32	24	20	180
Escape Raise - Waste	-	-	-	-	-	-	14	-	-	-	51	15	15	31	126
Vent Raise - Waste	-	-	14	-	-	-	-	-	-	-	-	-	-	41	55
Subtotal Capital Development	168	0	149	286	258	225	228	300	258	116	152	149	100	92	2,481
Drift - Ore	104	240	129	12	34	61	69	-	35	148	159	133	172	25	1,322
Raise - Ore	-	-	26	-	12	-	23	-	-	15	30	-	37	47	190
Subtotal Operating Development	104	240	155	12	46	61	92	0	35	163	189	133	209	72.1	1,512
Total Development	**272**	**240**	**304**	**298**	**304**	**286**	**320**	**300**	**293**	**279**	**341**	**282**	**309**	**164.1**	**3,993**

16.3.19 Mobile Equipment

Mobile equipment required for the LOM is based on the required productivity levels to achieve the desired mine production. All activities are to be carried out underground by owner labour and equipment, with the exception of diamond drilling, which will be the responsibility of the drilling contractor. Table 16.42 lists the minimum requirements for the equipment fleet to achieve the LOM plan.

TABLE 16.42
MOBILE EQUIPMENT REQUIREMENTS FOR LOM

Item	Equipment Description	UM	No of Units	Op Cost $US/hr
Mine Heavy Equipment				
1.0	Jumbo 2BHYD	each	1	$104.34
2.0	LHD Toro 410	each	3	$75.57
3.0	UG Bolter	each	1	$95.60
4.0	DL UG Longhole Drills	each	2	$75.57
5.0	DUX Scissorlift	each	2	$62.00
6.0	Shotcrete Robarm Jumbo	each	2	$110.17
7.0	30 T ADT Truck	each	3	$104.96
Mine Auxiliary Equipment				
8.0	Mine LV's	each	4	$5.00
9.0	Mine UG Personnel Carriers	each	2	$10.00
10.0	Mine Shotcrete Transmixer	each	1	$35.00
11.0	Mine Forklift	each	1	$35.00
12.0	Mine Grader/Dozer	each	1	$45.00
13.0	Mine Maintenance Toolcarrier	each	1	$20.00
14.0	DE 130 - DD Machine	each	3	$5.60

16.3.20 Maintenance

16.3.20.1 Surface Fixed Plant Maintenance

Infrastructure to support the underground mining operations at Pau-a-Pique includes the following:

- UG Workshop including wash-down and environmental facilities
- Shotcrete bag storage facility
- Secure store warehouse and yard
- Office complex including change rooms and emergency first aid facilities
- Compressor (compressed air) facilities
- Dewatering and water settling facilities
- Surface main fans
- Electrical substation and power reticulation
- Security fences and infrastructure
- Access roads
- Surface ore and waste dumps
- Communication infrastructure.

It is expected that the above surface infrastructure can support the requirements of the LOM without any major rebuild or replacement with minimum equipment addition. An overhead crane or lifting facility within the workshop area has been costed in the site setup lump sum.

16.3.20.2 Underground Fixed Plant Maintenance

- Underground infrastructure to support the operations at Pau-a-Pique includes the following:
- Underground fan and ventilation installations
- Underground pumps and dewatering installations
- Underground electrical reticulation and substations
- Underground services
- Haulage roads and accesses
- Second egress
- Refuge fixed/portable chambers
- Communication infrastructure.

The above infrastructure will be maintained by Aura except where specialized trades are required i.e. communication devices and amplifiers repairs, high voltage terminations, etc. Additions to current underground infrastructure have been estimated and costed within fixed plant calculations. Replacement of existing units have been included in the sustaining capital cost to operate the mine.

16.3.20.3 Underground Mobile Fleet Maintenance – Owner and Contractor

All maintenance and repairs of heavy vehicles and machinery will be carried out under the Aura Maintenance Management Plan. All major heavy equipment supplied by Aura will maintained by its employees including all associated parts, tires, hydraulics, and ground engaging tools ("GET"). Mobile machinery is maintained in the surface workshop adjacent to the mine office by Aura's qualified personnel comprised of a Maintenance Foreman, Mechanics, Welders, and Auto Electricians, etc.

Mobile machinery supplied by the contractor will be maintained in Aura's surface workshop adjacent to the mine office by the contractor's qualified personnel. All associated parts, tires, hydraulics, and GET are included in the contractor rates. No replacements are required for contractor equipment due to the short mine life.

17.0 RECOVERY METHODS

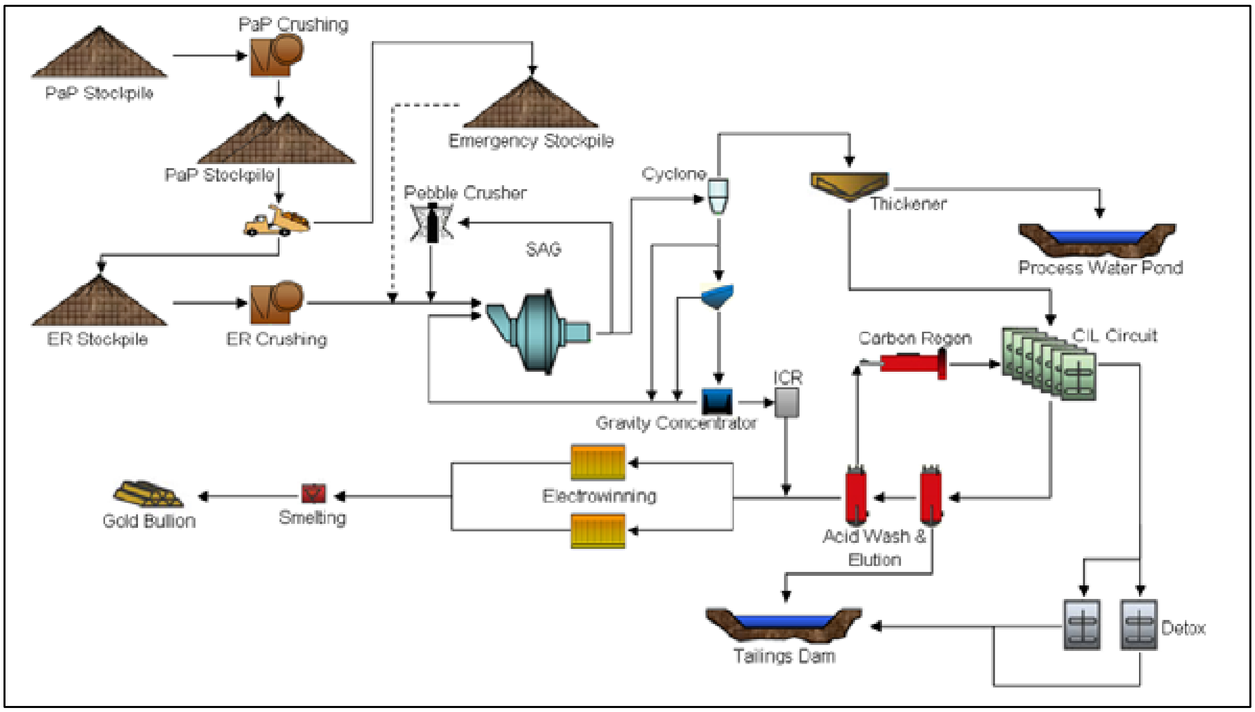
The EPP gold processing plant is located next to the Ernesto Deposit and is designed to treat up to 1 Mtpy feed. The flowsheet (Figure 17.1) is based on a low-risk proven technological configuration for processing gold bearing ore.

A primary crusher located at the front-end of the process plant. Run-of-mine ("ROM") feed will be blended and fed through the plant's primary screen. The screen oversize is crushed and the combined crushed feed is ground in a single-stage, closed-circuit semi-autogenous grinding ("SAG") mill.

Approximately 25% of the SAG mill cyclone underflow feeds a gravity-gold recovery circuit. The grinding circuit product is thickened and then pumped to a leach tank that is followed by six carbon-in-leach ("CIL") tanks in series. CIL tailings are treated in a cyanide reduction tank where cyanide is chemically decomposed. Final tailings are pumped to a tailings storage facility.

Loaded carbon, recovered from the first CIL tank, reports to the desorption area. Gold is stripped from the carbon into a solution and electroplated from solution onto stainless steel cathodes. Dried cathode sludge and flux are mixed and smelted to produce gold bullion.

Figure 17.1 EPP Process Flowsheet



17.1 CRUSHING

Haul trucks deliver run-of-mine feed to the ROM pad. A front-end loader is used to feed the crusher feed bin, which is fitted with static grizzly bars with an aperture of 600 mm.

The primary crusher grizzly feeder, which has an aperture of 150 mm, withdraws feed from the crusher feed bin at an average rate of 130 tph and discharges oversize material to the 1,045 mm x 840 mm primary jaw crusher, which operates at a final product size of 80–100 mm.

Grizzly feeder undersize and primary crusher product are combined on the primary crusher product conveyor, which discharges to the feed hopper. The feed hopper can also receive feed from a loader. A belt magnet removes tramp steel from the primary crusher product conveyor.

17.2 GRINDING

Crushed feed, pebbles, cyclone underflow, and gravity concentrator tailings are directed to the SAG mill feed chute. The SAG mill dimension is 5.8 m in diameter, 5.8 m long and has an installed motor power of 2.65 MW.

The recirculating load is 4.5 times the mass of the newly crushed feed. The SAG trommel screen (15 mm aperture) removes oversize (‘pebbles’) from the mill discharge at an estimated rate of 33 tph. The pebbles are directed to the pebble recycle conveyor, which includes a belt magnet to remove tramp steel and a metal detector located after the magnet.

SAG trommel screen undersize slurry flows to the cyclone feed hopper, where water is added to adjust the solids content to cyclone requirement. The primary cyclone feed pump feeds the cyclone cluster. Cyclone overflow flows to the trash screen bypass box. Cyclone underflow reports to the cyclone discharge distributor box, which directs half of the underflow to the concentrator scalping screen (2 mm aperture) and the remainder of the underflow to the SAG mill feed chute. Concentrator scalping screen oversize (estimated to be half the scalping screen feed) returns to the SAG mill feed chute, while the undersize feeds the gravity concentrator.

Gravity concentrate is discharged intermittently to an intensive cyanidation reactor (“ICR”) package unit. The ICR unit intermittently discharges leached concentrate to the cyclone feed hopper and discharges pregnant solution to the gold room.

Process water is added at the following points in the grinding area:

- SAG mill feed chute dilution
- SAG trommel screen spray
- Cyclone feed hopper dilution
- Concentrator scalping screen spray
- Gravity concentrator fluidization (flush).

Two sump pumps in the grinding area discharge intermittently to the cyclone feed hopper to return spillage to the feed system.

17.3 E-CAT THICKENER

The trash screen bypass box normally feeds the trash screen (0.8 x 18 mm aperture), but also allows slurry to bypass this screen. Trash screen oversize is directed to the trash bunker, while trash screen undersize slurry flows to the pre-leach thickener feed box. Dilute flocculant is added to the pre-leach thickener feed box and to the thickener feed well.

The CIL feed pump discharges thickened slurry to the CIL feed distributor. The feed to the CIL circuit is 50-52% solids by weight after the addition of lime, barren eluate and the flow from the gold room sump pump. The average flow rate of the CIL feed is 176 m^3/h (260 tph). Thickener overflow flows to the pre-leach thickener overflow tank and is then pumped to the process water pond.

The CIL feed sampler takes a primary sample from the trash screen underflow stream. The primary sample passes to the CIL feed secondary sampler, and the sample discarded from the secondary sampler reports to the spillage sump. The pre-leach thickener sump pump discharges intermittently to the pre-leach thickener feed box.

17.4 CARBON-IN-LEACH

The CIL area includes seven tanks in series: one leach tank followed by six CIL tanks for a total residence time of 24 hours.

The CIL feed distributor box normally discharges to the leach tank, but can also discharge to the first CIL tank. Launders allow the slurry flow to bypass any of the CIL tanks. The final CIL tank discharges to the cyanide reduction feed box.

Sodium cyanide solution is added to the leach tank. Cyanide can also be added to CIL tanks 1 and 2. A cyanide analyzer monitors the leach cyanide concentration.

Each tank has a mechanical agitator and air sparge point. Each CIL tank has a 0.8 mm slotted-aperture inter-tank screen for retaining carbon. Carbon transfer pumps in CIL tanks 2 to 6 periodically transfer carbon to the preceding CIL tank. The loaded carbon recovery pump in CIL Tank 1 periodically discharges slurry to the 0.8 mm x 0.8 mm aperture loaded-carbon recovery screen. Loaded carbon (screen oversize) reports to the acid wash column and loaded carbon recovery screen undersize returns to CIL Tank 1.

Barren carbon and new carbon (slurry) report to the 1.0 mm slotted-aperture barren carbon sizing screen. Oversize carbon reports to CIL Tank 6 or, alternatively, to CIL Tank 5. Barren carbon sizing screen undersize flows to the cyanide reduction feed box. Two sump pumps in the CIL area discharge intermittently, either to the trash screen bypass box, or to the cyanide reduction feed box.

17.5 ADR PLANT

17.5.1 Acid Wash

The acid-wash cycle starts when the acid-wash column is full of loaded carbon. Concentrated hydrochloric acid is added to raw water to a concentration of 3% (w/w) in the column. The acid wash column is then allowed to soak.

During the soak, the acid dissolves any acid-soluble fouling agents (mainly calcium carbonate, $CaCO3$) that have adsorbed onto the carbon. After the soak, the carbon is rinsed with raw water. The acid column rinse water and soak solution both discharge to the tailings pump hopper. Rinsed carbon is transferred from the acid wash column to the elution column by using raw water.

17.5.2 Elution

The strip solution pump draws solution from the strip solution tank and primes the elution column. Sodium hydroxide and sodium cyanide are added to the strip solution at a concentration of 3% w/w each.

During preheating, the strip solution is pumped through the recovery heat exchanger (cold side), through the primary heat exchanger (cold side), and into the elution column. Solution leaving the elution column returns directly to the strip solution tank. The LPG-fired solution heater heats thermal oil, which is pumped in a closed circuit through the primary heat exchanger (hot side) indirectly heating the strip solution.

Elution starts when preheating is complete. During elution, solution leaving the elution column passes through the recovery heat exchanger (hot side) and to one of the eluate tanks. Elution continues until six elution column bed volumes ("BV") have reported to the eluate tank. Raw water makes up the strip solution tank level during elution. Solution heating stops for the last BV, cooling the column and the carbon.

After cooling, the eluted carbon is transferred to the regeneration feed dewatering screen, or bypasses to the carbon quench tank. The desorption sump pump discharges to the CIL feed distributor box.

17.6 CARBON REGENERATION

Excess water drains from the carbon in the 1.0 mm slotted-aperture regeneration feed dewatering screen. A screw feeder withdraws carbon from the feed hopper and feeds the carbon regeneration kiln. Regenerated carbon is quenched with water in the carbon quench tank and is then pumped to the barren carbon sizing screen by the carbon transfer pump.

A plant operator uses the activated carbon hoist to lift carbon bulk bags and add carbon to the 2.5 m^3 carbon attrition tank. The new carbon transfer pump discharges attritioned carbon to the barren carbon sizing screen.

17.7 GOLD ROOM

Pregnant eluate and ICR pregnant solution report to the eluate tanks. The eluate pumps feed electrowinning cells during electrowinning and transfer barren eluate to the CIL feed distributor box at the end of an electrowinning cycle. Electrowinning cell discharge flows to the eluate return hopper and is then pumped back to the eluate tank(s).

A plant operator uses the gold room hoist to lift loaded cathodes up over the off-line cell. The operator then uses a high-pressure cleaner to loosen and wash the cathode slimes into the cell. The slimes flow into the pan filter and filtered sludge is put into an oven to dry. Dry sludge is mixed with flux and smelted in the barring furnace about once a week. Bullion bars are cleaned, sampled, weighed, and stored in a vault. The electrowinning cell exhaust fan vents air and mist from the cells to atmosphere outside the building during the electrowinning process. The furnace exhaust fan vents air from the furnace to the outside atmosphere during smelting. The gold room has four vent fans to ensure adequate ventilation.

The gold room sump pump discharges to the CIL feed distributor box.

17.8 CYANIDE REDUCTION AND TAILS PUMPING

Lime slurry, sodium metabisulphite solution, and copper sulphate solution are added to CIL tailings slurry in the cyanide reduction feed box, which discharges to the cyanide reduction tank. The reduction tank has a mechanical agitator and four low-pressure air sparges. The reduction tank overflows into the carbon safety screen feed box, which feeds a 1.0 mm slotted-aperture carbon safety screen. Oversize reports to a drum or box, while screen undersize flows to the tailings pump hopper and is pumped to the tailings storage facility.

The tails sampler takes a primary sample from the reduction tank overflow stream. The primary sample passes to the tails secondary sampler, and the secondary sampler discard reports to a spillage sump. There are two cyanide-reduction area sump pumps. The one that receives the secondary-sampler discard discharges to the cyanide reduction feed box and the other discharges to the carbon-safety-screen feed box.

17.9 DISTRIBUTED CONTROL SYSTEM

The processing plant is fully automated and controlled through a distributed control system ("DCS") Siemens PCS7. This system utilizes a modern process with open, flexible and scalable architecture as its system platform.

At the same time, this specific DCS model benefits from the innovative Advanced Process Library ("APL") which integrates many years of experience both with design engineers and plant operators as an excellent basis for functionally optimized design with high operational efficiency.

The DCS system has three operating modes: Automatic, Manual and Expert.

- The Manual mode allows control room operators to over-write setpoints in the process and manually regulate controlled variables.
- The Automatic mode launches the basic PLC/DCS control strategies implemented for the different areas.
- The Expert mode is a third mode implemented in the DCS for future integration of an advanced control system package, better known as an "Expert System", which is considered the highest rank for automation in processing plants.

The Siemens PCS7 comes with one engineering station which is exclusive for development and manipulation of the control strategies coded into the DCS, two redundant servers in case of communications failure or any other problems, and three client stations provided for control room operators to control the plant.

Figures 17.2 and 17.3 provide computer screen snapshots of two existing user interfaces implemented in the DCS.

Figure 17.2 Crushing and Grinding DCS Interface

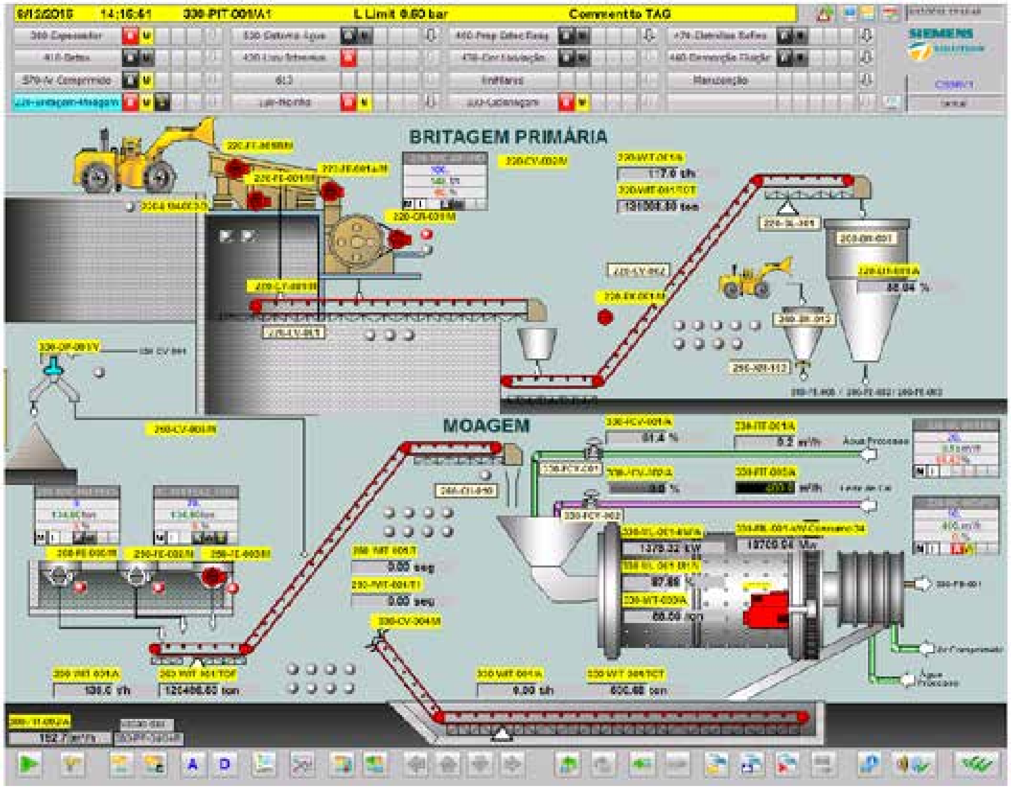
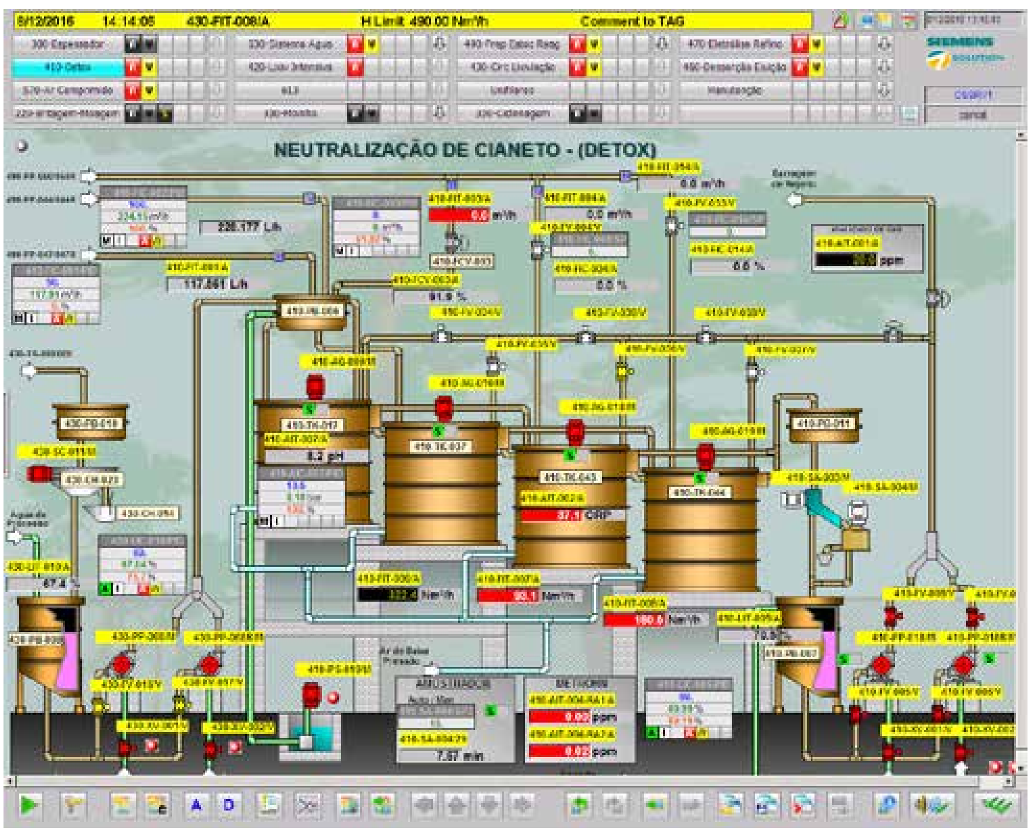


Figure 17.3 Detox DCS Interface



18.0 PROJECT INFRASTRUCTURE

18.1 SITE ACCESS AND CONTROL

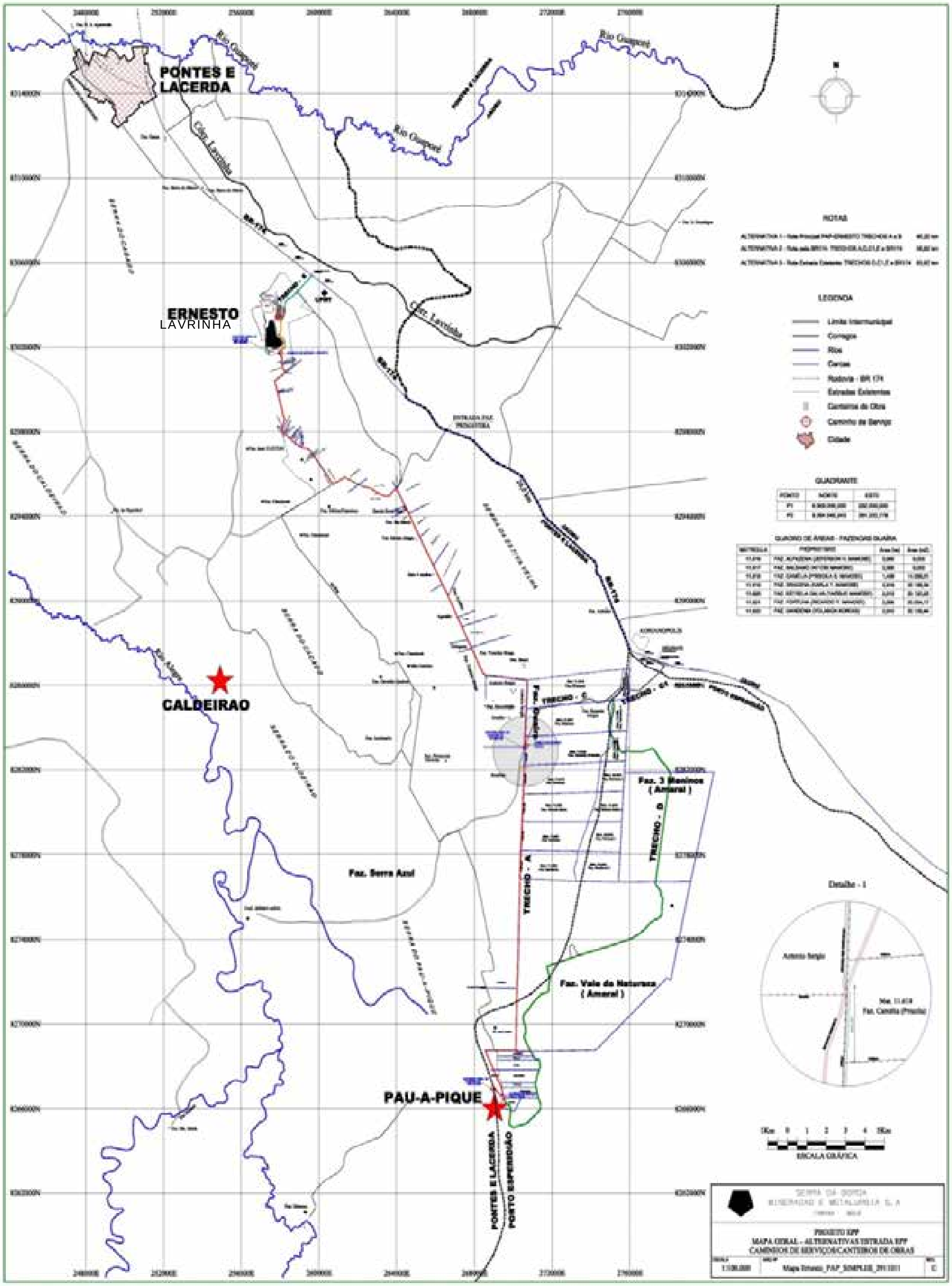
The Ernesto and Lavrinha Deposits are contiguous and are located 12 km south of the town of Pontes e Lacerda which is approximately 450 km west of the city of Cuiabá (the capital of Mato Grosso state, Brazil). The Pau-a-Pique Deposit is located approximately 47 km southwest of Ernesto.

The federal (Brazil) highway BR-174, which connects Cuiaba with Pontes e Lacerda is also used to reach Ernesto and Lavrinha from Pontes e Lacerda, which crosses within 2 km of the Project. Aside from highway BR-174, the Project is served by a network of good gravel and dirt roads that offer year-round access for two-wheel drive vehicles.

Ore from Ernesto and Lavrinha will be transported to the process plant by haul trucks internally within the mine property using internally maintained roads.

Pau-a-Pique ore will be transported via a public 47 km road section. This road will require to be maintained over time by the mine, at a frequency no longer than a bi-weekly basis. Figure 18.1 shows the location of the Ernesto, Lavrinha and Pau-a-Pique Deposits and the various access roads.

Figure 18.1 Location of the Ernesto, Lavrinha and Pau-a-Pique Access Roads



Pontes e Lacerda has a local airport that can be used by small business aircraft. The nearest international airport with connecting flights to all major cities in Brazil and internationally is located in Cuiaba.

18.2 WATER SERVICES

Lavrinha Creek ('Córrego Lavrinha') located near highway BR-174, and approximately 3.8 km from the processing plant, is the source of fresh water for the Project.

Fresh water is pumped by two 100 hp Imbil pumps model INI50315 with a nominal flow of 70 m^3/h (maximum capacity 120 m^3/h) via an 8" HDPE pipeline to a fresh water tank located in the premises of the processing plant.

The fresh water tank has four main water pumps (e.g. raw water pump; low-pressure gland water pump; high-pressure gland water pump; and fire-water pumps) providing fresh water to the following addition points:

- Reagents
- Crushing
- Desorption
- Gold room
- Water truck stand-pipe.

The low-pressure gland water pump provides gland service water to the following items and points of use: grinding; carbon transfer pump; pre-leach thickener; and lime pump.

The high-pressure gland water pump provides gland service water to the tailings pumps.

Electric and/or diesel fire-water pumps provide water to fire hydrants and hose reels throughout the site, when required. The fire-water jockey pump maintains pressure in the fire-water ring main pipeline.

Process water is recovered from the tailings dam by two 75 hp Flygt pumps, model BS 2290010 HT, through a 1 km 6" HDPE line at a nominal rate of 85 m^3/h (maximum capacity of 234 m^3/h).

Figure 18.2 shows the routing of the fresh water line and the recycling water from tailings dam lines. The lower portion of the raw water tank is reserved for use as fire-water.

There are two water treatment plants in the Project, one installed in the premises of Ernesto with a treatment capacity of 6 m^3/h and a second water treatment plant installed in Pau-a-Pique camp with a treatment capacity of 3 m^3/h. These two plants provide potable water to all kitchens, toilets, hand basins, change houses and showers ('ablutions'), offices and process plant safety showers throughout the Project.

The potable water tank at the Ernesto plant has a capacity of approximately 100 m^3. The treated water tank at Pau-a-Pique has a capacity of approximately 50 m^3. The relatively low retention times ensure the treated water will not go stale.

Figure 18.2 Fresh Water and Reclaimed Water Piping Routing

Source: Google Earth, edited by Aura.

18.3 ELECTRICAL POWER SUPPLY

In 2010, Ausenco and Dalben Consultoria were retained to evaluate options for power supply as part of a Feasibility Study. A direct energy supply connecting to the Pontes e Lacerda substation was recommended as the most cost effective, reliable and best environmental option and as such, a 12 km 138 kilovolt ("kV") transmission line was built for the Project which connects from the Pontes e Lacerda substation.

The main substation from ENERGISA, which is the Power Utility Company of the Mato Grosso State, is located in the mine property and receives the 138 kV power from the transmission line, which is then downgraded to 34.5 kV for internal distribution. The internal power distribution at 34.5 kV voltage is as follows:

- 48 km transmission line to Pau-a-Pique;
- An internal connection of approximately 50 m connects to the Ernesto main substation, which is operated and maintained by site personnel.

From the Ernesto main substation, power is once again converted to 13.8 kV which feeds to the main plant distribution circuit. The transformer installed in the Ernesto substation has 10/12.5 MVA power capacity, using natural oil-forced-air cooling. Under forced ventilation, this transformer guarantees a 25% reserve of power for future expansions of the Project.

The total electrical load installed in Ernesto is currently estimated at 7.35 MW (existing plant and on-site infrastructure). When Ernesto underground mining activities start, a maximum of 2.8 MW of electrical installed load will be added to the overall consumption. The installed substation and the existing power infrastructure will be suitable to address the future energy requirements of the Project.

The total electrical load installed at Pau-a-Pique is 1.91 MW. The current transmission line is adequate to supply enough energy for the Project restart. The transformer installed at Pau-a-Pique has a 3 MVA power capacity. Table 18.1 summarizes the entire Project electrical load.

TABLE 18.1
EPP PROJECTED ELECTRICAL LOAD

	Ernesto Site				Pau-a-Pique	
Transformer capacity (at main substation)	10 MVA (12.5MVA w/ forced ventilation)				3 MVA	
Total electrical load installed (MW)	Lavrinha	Ernesto UG	G&A and others	Plant & workshop	G&A and others	UG
	10.15				1.91	
	N/A	2.8	1.0	6.35	0.5	1.5

18.4 TAILINGS DAM

18.4.1 Existing Tailings Dam

In 2016, Tierra Group International, Ltd. ("Tierra Group") was retained to evaluate the detailed engineering design for the existing Stage I Ernesto e Pau-a-Pique ("EPP") tailings dam and also to analyze its physical stability in its current condition.

The tailings dam was built in 2010, based on the Stage I engineering detail design developed by the consulting company DAM Projetos of Engenharia ("DAM") based in Belo Horizonte, Brazil. The earth-fill dam was built using compacted saprolite to its current crest height 339 m ASL.

The dam is 19 m high, with the upstream and downstream dam constructed to 2H:1V (horizontal:vertical) slopes. The impoundment area is not lined with geomembrane. Tailings are discharged along the upstream dam face to create a tailings beach upstream of the dam and a supernatant pool in the upstream impoundment basin.

Tierra Group visited the EPP site between October 23 and 26, 2016 to inspect the tailings storage facility ("TSF") and review data pertaining to the Stage I detailed engineering design. The data included a Stage I engineering design report, construction drawings, "As Built" reports and drawings, geotechnical investigation and laboratory analyses, hydrology, and operations reports, etc.

Using this data, Tierra Group performed a data gap analysis to confirm that sufficient information is available to design the required Stage II dam raise.

To complete the gap analysis, Tierra Group performed the following reviews:

- The hydrology, hydraulics, and water balance were reviewed for accuracy and practical application to the existing conditions.
- A geotechnical review of the engineering properties used to design the dam. Slope stability analyses were performed to determine current factors of safety ("FOS") against dam slope instability considering dam construction and foundation strengths. The slope stability analysis verified that in its current state the dam exceeds minimum slope stability FOS recommended by the Canadian Dam Association ("CDA") guidelines.
- The tailings discharge plan was reviewed and updated to maximize storage at the current dam crest elevation (339 m ASL) and an updated tailings distribution plan was developed and presented in a series of drawings.

Through the course of the Stage I dam design review Tierra Group found that appropriate standards of engineering practice and care were used to design the Stage I tailings dam. Tierra Group has commenced a detailed engineering design for the Stage II dam raise to dam crest 342 m ASL.

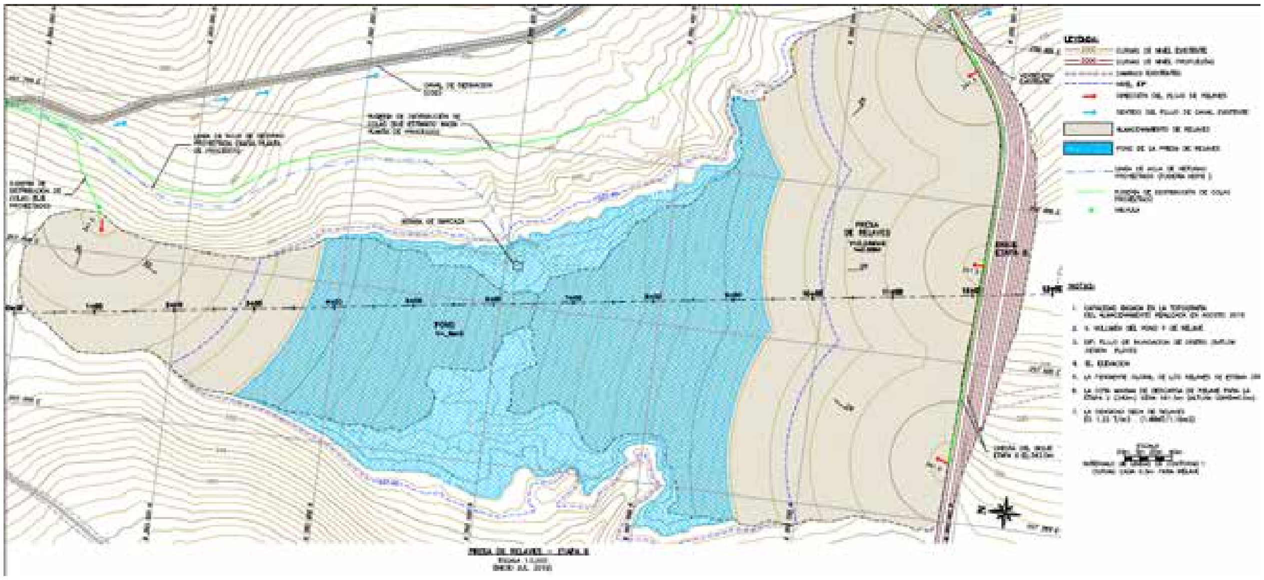
Tierra Group also carried out a revision and update of the tailings disposal plan and the water balance plan for the Project. The tailings disposal plan is used to determine future tailings storage capacity and useful lifespan of the growth stages of a given facility. This plan integrates the designed storage capacity with future disposed tailings volumes and climatological variables to estimate the storage volumes and elevations (tailings and operation pond) over time.

The current tailings arrangement considers the discharge of tailings at the upstream face of the main dam, i.e. from south to north. During the water balance review, the location of the current discharge points and the maximum level of tailings discharge were considered to be at 338.5 m, which accomodates 0.5 m of freeboard below the dam crest (339 m).

Results of the water balance, considering the above parameters, provided 6 months of storage capacity in the existing dam (from October 2016 to March 2017), with a storage capacity of 0.21 Mm^3 (0.29 Mt).

Tierra Group carried out an evaluation considering new discharge points in the northern extent of the impoundment (i.e. from north to south), which would migrate the supernatant pool to the center of the impoundment. It was found that by implementing this revised tailings deposition plan an additional six months of tailings storage can be realized in the current facility, extending its life to September 2017, as shown in Figure 18.3. The estimates were developed considering the bathymetric information supplied by Apoena (08/30/16), considering a tailings production of 51,440 t/month, as per the current mine plan, and a maximum tailings discharge level of 338.5 m.

Figure 18.3 Revised Discharge Points in the Existing Tailings Dam



18.4.2 Future Tailings Dam Raise

18.4.2.1 Previous Work

DAM was commissioned by Yamana in 2013/2014 to develop an engineering design for a TSF to store approximately 7.0 Mt of tailings. The following operational parameters formed the design basis for DAM's design:

- Annual tailings production: 1.1 Mt
- Life of mine: 7.3 years
- Total tailings production: 8 Mt
- Dry apparent density of tailings: 1.40 t/m^3
- Total tailings volume: 5.7 Mm^3
- Total pulp flow: 170.3 m^3/h
- Pulp Density: 1.48 t/m^3
- Percentage of solids (by mass): 50.05
- Density of Solids: 2.81 t/m^3
- Fresh water demand: 29.90 m^3/h

DAM performed hydrologic, hydraulic, geologic and geotechnical investigations and engineering analyses to design a conventional TSF. The proposed TSF uses locally available borrow source materials (saprolite) to construct a downstream earth fill dam in three progressively higher dam stages to contain tailings and process supernatant (water).

The DAM design includes an internal drainage system (sand and gravel) that is extended with each subsequent dam stage. Unsuitable foundation soils were to be removed below the dam footprint prior to constructing the dam. Table 18.2 shows the proposed dam construction stages (dam crest elevations) versus storage (volume and time) relationship.

TABLE 18.2 TAILINGS DAM STAGE DESIGN VOLUMES AND STORAGE CAPACITY BY DAM				
Stage	**Dam crest elevation (m)** [1]	**Elevation of the stored tails (m)**	**Total dam volume (m^3)**	**Available capacity (years)**
No. I	339	337.5	2,335,600	2.3
No. II	342	340.5	3,664,000	1.7 [2]
No. III	348	346.5	7,183,000	4.0

[1]Topographic datum updated to reflect WGS84 in 2016.
[2]Stage II accounted for only 1.7 years capacity due to the restrictive view on capital expenditure by the previous owner in 2014.

DAM's report outlines that the capital cost estimate accuracy is within +/-10% (Detailed Level) and assumes that all of the construction material will come from available borrow areas located less than 3 km from the entrance of the process plant. The total capital cost estimated by DAM for the tailings dam raise to elevation 348 m is 3.8 M Brazilian Reais along with an estimated operational cost of 318K Reais per month.

18.4.2.2 2016 Stage II TFS Engineering Design Review And Update

In October, 2016, Aura commissioned Tierra Group to perform an overall review and re-design of the Stage II tailings dam raise (crest elevation 342 m ASL) and detailed engineering package developed by DAM. This scope of work is expected to be complete by late January, 2017.

The design work is currently underway, contemplates raising the dam height 3 m and maintaining 2H: 1V upstream and downstream dam slopes. A field geotechnical investigation is defined to corroborate geotechnical parameters used in the Stage I design, and establish those for the Stage II design.

A tailings deposition plan has been developed, which prescribes adding tailings discharge points in the north and east impoundment to extend the life of the Phase II TSF to 2.3 years. Table 18.3 shows tailings storage capacity of Stages I and II.

TABLE 18.3 TIERRA GROUP'S VOLUMES AND STORAGE CAPACITY OF STAGES I AND II OF THE TAILINGS DAM					
Stage	**Dam Crest Elevation (m)**	**Tailings Discharge Elevation (m)**	**Incremental Volume of Dam (m^3)**	**Tailings Storage Cum. (Mt)**	**Remaining Capacity (Years)**
Stage I	339	338.5	230,000	1.76	1.0*
Stage II	342	341.5	80,000	2.98	2.3

**Additional discharge point at the eastern end of impoundment.*

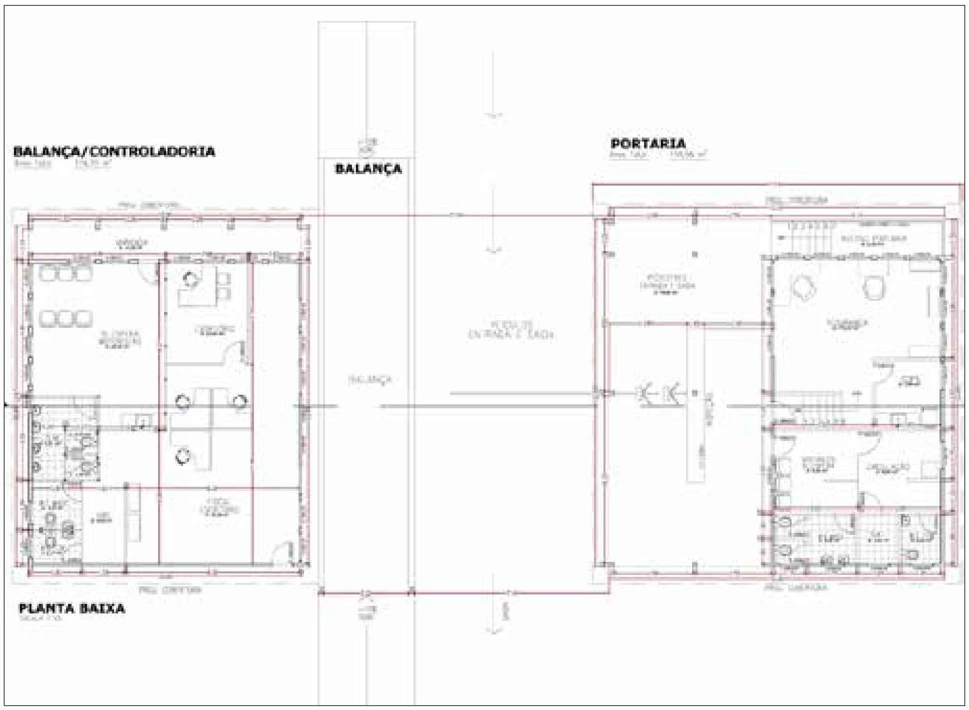
The Stage II final design will require an additional 90,000 m^3 of fill to be placed downstream of the existing dam. The resultant facility will have a footprint area approximately equal to 155,000 m^2, which is nominally 5% greater than its current footprint area. Tierra Group is expected to complete the design work in January 2017.

18.5 WORKSHOP, WAREHOUSE AND OFFICES

18.5.1 Main Gate / Guard House

Figure 18.4 shows the layout for both main entry points to the Ernesto and the Pau-a-Pique mines. Both layouts are similar and include a security gate located next to an industrial scale to control truck loads and inventory control as well as inspection of any consumable coming into the sites.

Figure 18.4 Layout of Main Entry to Ernesto and Pau-a-Pique Sites



18.5.2 Office Building

The Ernesto site is comprised of a multiple office area which is located adjacent to the processing plant. This multiple office area includes specific areas for:

- Administrative building, which includes offices for the general manager, mine and plant managers, technical services team (i.e. geology, mine and process engineering), procurement and cost controller, as well as areas for employee training and a medical office for first-aid (Figure 18.5).
- Two fully serviced conference rooms;
- Area for printers and IT-related tasks;
- Cafeteria and restaurant area (Figure 18.6);
- A change room (dry room) for all plant and mine employees, which is designed for 80 men and 30 women at a shift change, with a wall dividing the men and women change areas. The male change room includes eight showers, eight toilets and 240 lockers. The women change room includes three showers, three toilets and 30 lockers (Figure 18.7).

- Chemical and metallurgical laboratory;
- Warehouse area with a storage yard; and
- Four different washroom areas.

Figure 18.5 Layout of Ernesto's Administrative Offices

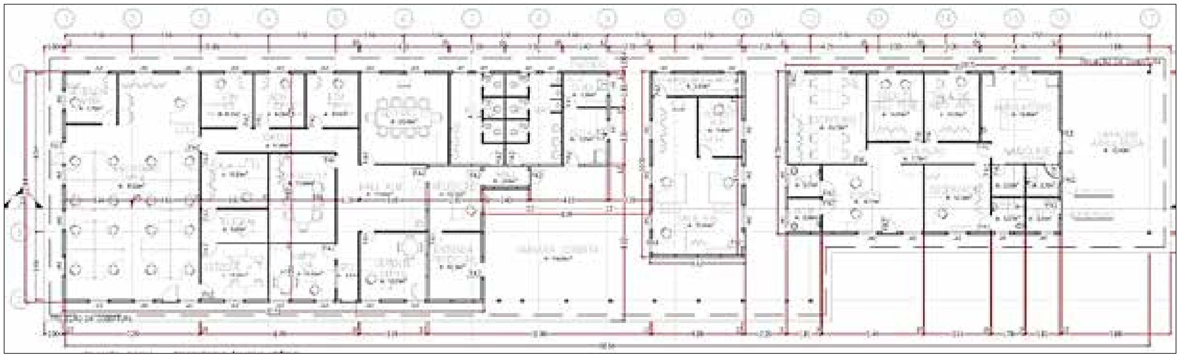


Figure 18.6 Layout of Ernesto's Cafeteria and Restaurant Area

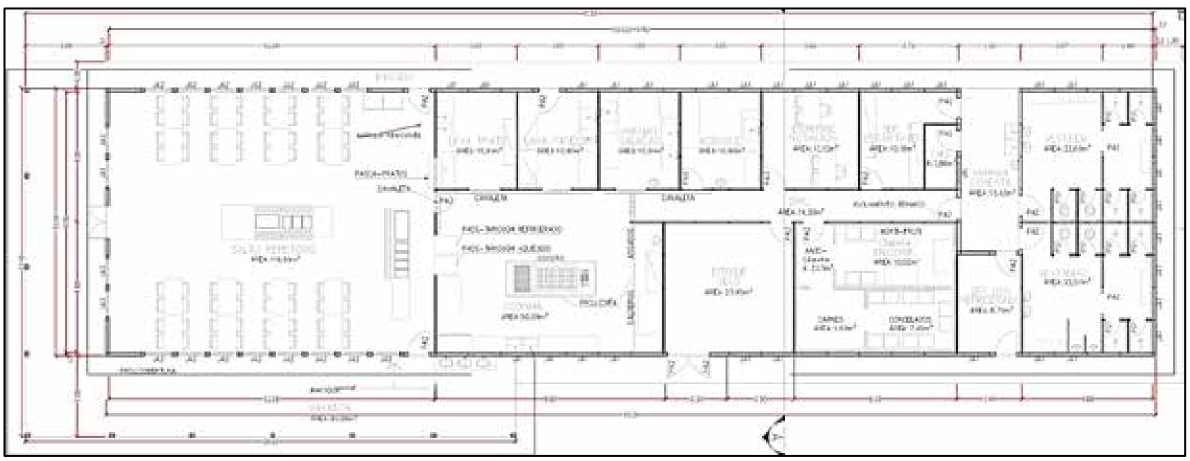
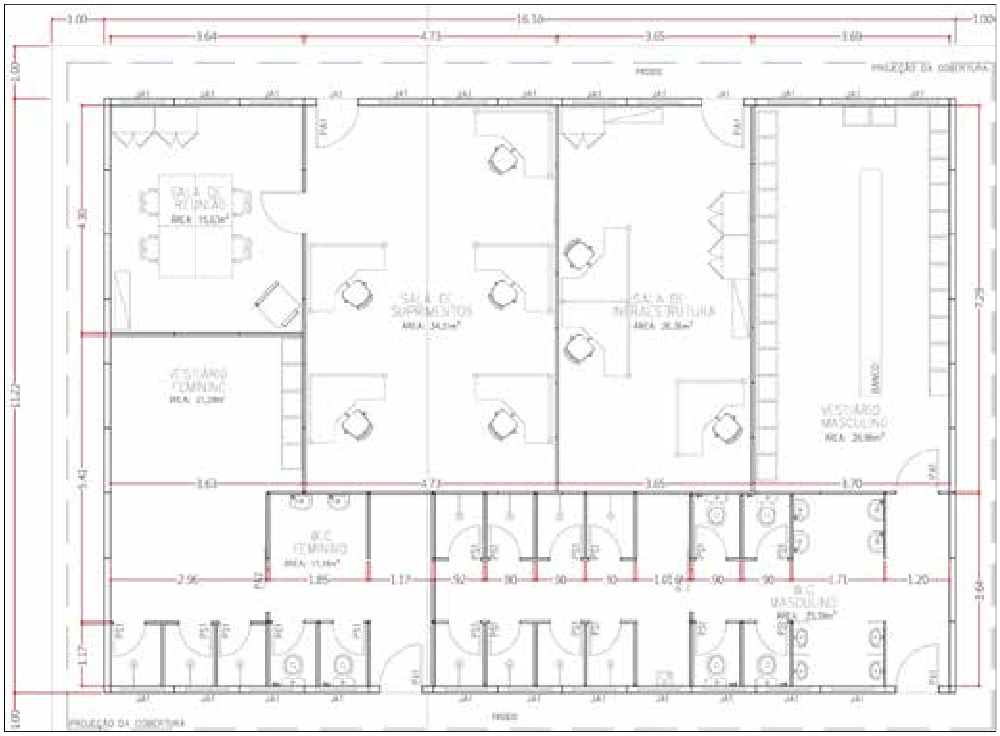


Figure 18.7 Layout of Ernesto's Change Room Area and General Services



The office area in the Pau-a-Pique site is smaller in size compared to Ernesto due to the smaller workforce assigned to work in this location. The Pau-a-Pique office area includes:

- Two main offices for technical services and management (Figure 18.8);
- One conference room;
- A storage room for all underground specialized equipment such as surveying equipment, scanners, etc.;
- Cafeteria and restaurant area (Figure 18.9);
- First-aid office;
- Offices for safety and logistics;
- Fully serviced warehouse area which is exclusive for the underground operation;
- Individual washrooms; and
- Change room area which is designed for 50 men and 20 women on a shift change. The male change room includes ten showers, five toilets and 140 lockers. The women change room includes two showers, two toilets and 20 lockers (Figure 18.10).

Figure 18.8 Layout of Pau-a-Pique's Administrative Offices

Figure 18.9 Layout of Pau-a-Pique's Cafeteria and Restaurant Area

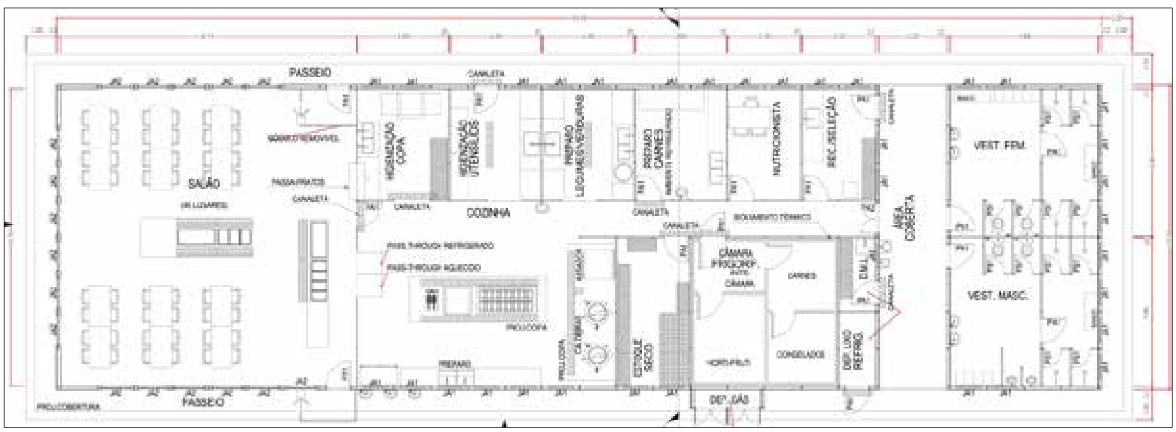
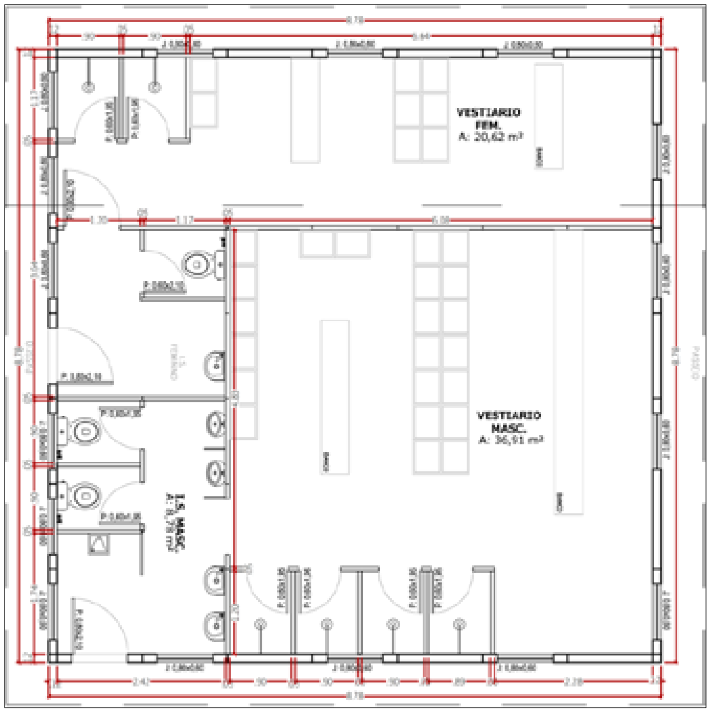


Figure 18.10 Layout of Pau-a-Pique's Change Room Area



18.5.3 Process Plant Workshop

The processing plant workshop is a fully serviced building 46 m long x 12 m wide. It includes offices for:

- Maintenance planning;
- Meeting room;
- Instrumentation/electric room;
- Assembly mechanical room;
- Area for document control;
- Washrooms; and
- Office to accommodate workshop's supervisors.

The processing plant has a tool and maintenance area for corrective work on smaller mechanical equipment (Figures 18.11 and 18.12).

The proposed Ernesto mine dry and maintenance areas will be located near the main access. These new areas will be fully serviced with modular offices and modular change rooms, similar to the ones installed at the Pau-a-Pique mine. There is an allowance assigned for these areas which was based on the historic Pau-a-Pique modular offices.

The truck service workshop and maintenance offices for the Lavrinha mine are the responsibility of the contractor engaged in the mining of Lavrinha; therefore, this is not accounted as part of the existing infrastructure.

Figure 18.11 Layout of Process Plant Maintenance Area

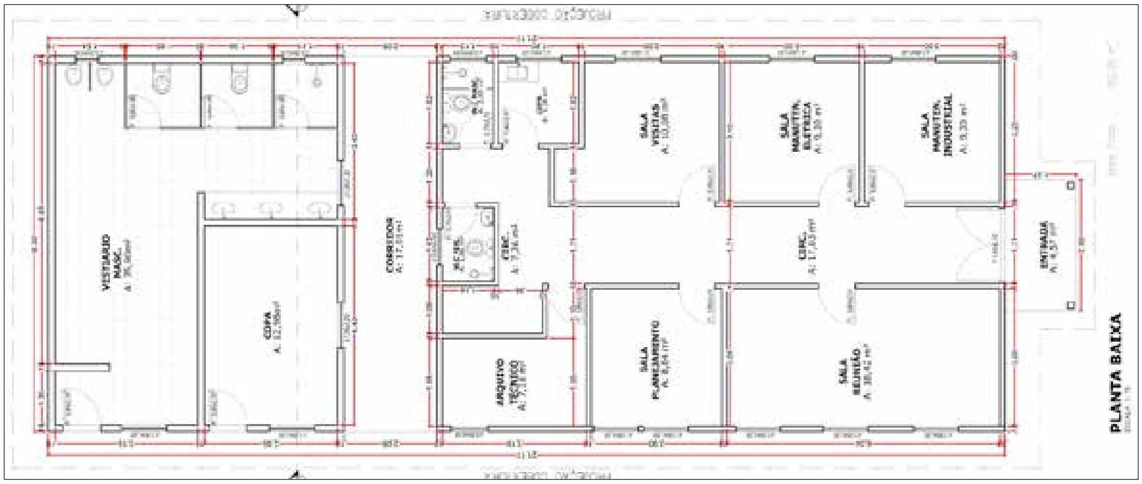
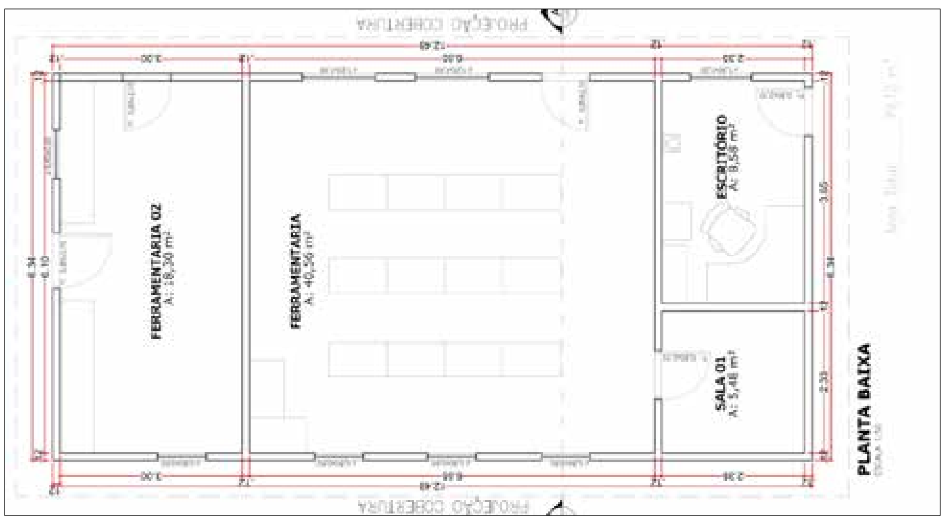


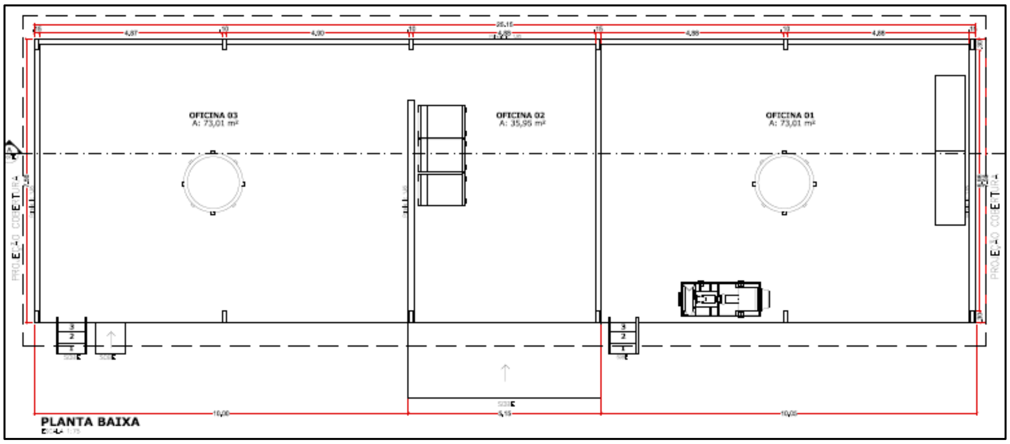
Figure 18.12 Layout for the Process Plant Tool Area



The Pau-a-Pique truck service area and maintenance workshop is 15.40 m long x 10.4 m wide. The area includes two mobile equipment service stations, maintenance offices, storage yard and a maintenance platform built with reinforced steel (Figure 18.13).

The area also includes an electrical equipment storage facility for all spare transformers, underground electrical cables, etc. as well as a fuel storage tank.

Figure 18.13 Layout of Pau-a-Pique Maintenance Area

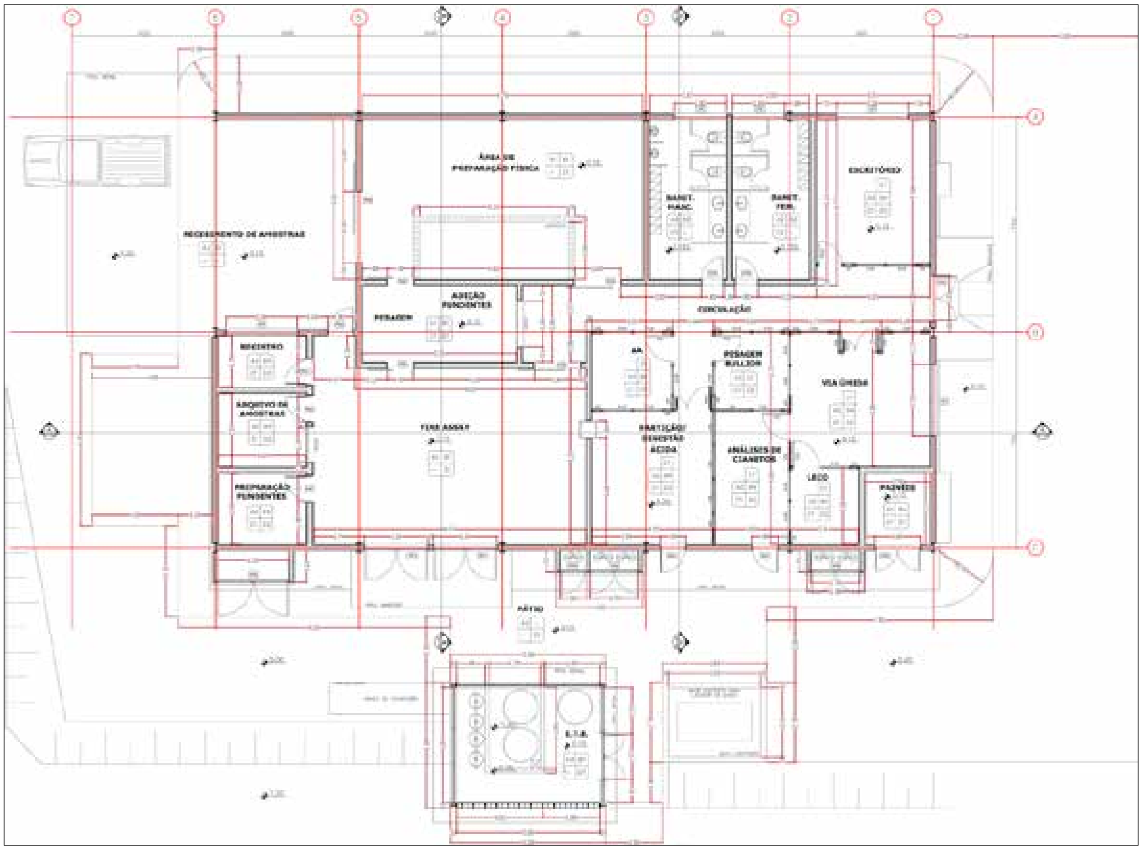


18.5.4 Site Laboratory and Plant facilities

The site laboratory layout is shown in Figure 18.14, which includes the following areas:

- Sample reception;
- Sample preparation room;
- Sample storage room;
- Physical and metallurgical laboratories;
- Fire assay;
- Chemical analysis area;
- Weighing room;
- Machine/utility area;
- Washrooms; and
- Administrative office.

Figure 18.14 Layout of the Site Laboratory



The Ernesto CIL control room area includes the processing control room with two control stations, the plant manager's office and toilets.

The processing plant reagent storage building is 30.2 m long x 16.2 m wide and was constructed with two main areas: the first area is isolated and used exclusively to store cyanide, and the second area is destined for other reagents.

18.6 AIR AND LPG SERVICES

Mining services such as ventilation, dewatering pumping, compressed air and others are described in Section 16 of this Report.

The processing plant has three cyanide reduction and carbon-in-leach air blowers which provide low-pressure air to the cyanide reduction tank and the CIL tank sparges.

There are also three air compressors that provide compressed air to the processing plant at a pressure of 7.5 bar and a nominal flow of 1,175 m^3/h. A single air dryer, fitted with pre-filters and post-filters, provides instrument-quality compressed air to the following areas or points of use:

- Plant air service points;
- CIL;
- Lubrication units; and
- Desorption control valves.

Two additional air compressors provide compressed air to all instrumentation in the processing plant at a pressure rate of 8 bar with a maximum flowrate of 873 m^3/h.

A road tanker delivers Liquified Petroleum Gas (“LPG”) to site, and offloads into two LPG storage tanks, which will supply the following points of use:

- Gold room furnace;
- Regeneration kiln; and
- Solution heater.

18.7 FUEL STATION

A fuel storage and distribution system, including tanks, pumps, washing and lubricating equipment, is available at both the Ernesto and Pau-a-Pique sites. At Ernesto the storage capacity is $30m^3$ and at Pau-a-pique it is 15 m^3.

Open pit mine equipment will be serviced twice a day by a mobile unit with a capacity of 6,000 litres.

Pau-a-Pique has its own fuel storage station located in an area near the maintenance shops. The estimated Pau-a-Pique consumption is approximately 125,000 litres of diesel per month and the consumption breakdown is as follows:

- Surface hauling trucks: 59,500 l;
- UG equipment: 51,800 l;
- Auxiliary equipment: 9,700 l; and
- Trucks: 3,600 l.

18.8 COMMUNICATIONS

The communications system for the Project is based on fiber optic, category 6 cabling and wireless network infrastructure, radio communications, telephone system and mobile telephony.

18.8.1 Shelter

The internet link, telephone system, switches, servers and the other major network equipment are located in dedicated shelters (as seen in Figure 18.15) that are near the Ernesto site’s main office and in the Pau-a-Pique site.

To provide stability and security to the services, some mechanisms are included in the shelters:

- Fire suppression systems;
- Uninterruptible power supply;
- Air conditioning with redundancy system;
- Fiber optical backbone;
- Closed circuit television system and access control; and
- Temperature and humidity control system.

Figure 18.15 Ernesto Shelter (Right) and Pau-a-Pique Shelter (Left)

18.8.2 Services

The Ernesto site has a connection to a primary Internet Service Provider (“ISP”) via fiber optic to provide the main link of 20 Megabytes per second (“Mb/s”).

A redundant system uses an available microwave link of 4 Mb/s supplied, and is maintained by a secondary ISP.

The standard phone system is based on a VoIP system connected to an E1 channel and to the network providing internal and external calls. A mobile phone system is also available at site through a signal coming from a telecommunication tower located inside the Project. This tower includes repeaters to improve the mobile network signal.

The underground mine has an emergency telephone system that provides phone lines located in strategic places within the underground mine, programmed to dial directly to security guards in case of emergency.

The radio communication system inside the Pau-a-Pique underground mine is based on laying leaky cable feeder antenna through the tunnels and access ramps to cover the areas of greater traffic of people and equipment.

Other surface areas located further away have additional towers equipped with radios and repeaters for access to communications signal.

18.8.3 Distribution

A 66 m high distribution tower is located in the Ernesto site, communicating cameras for security monitoring, radios to receive a redundancy link, and radios to provide communication with the Pau-a-Pique Project via a microwave point-to-point connection.

There is also a 30 m tower in the Ernesto area which provides mobile and radio communication across the Ernesto and Lavrinha Projects.

Pau-a-Pique has two distribution towers of 15 m and 24 m equipped with radios, antennas and cameras to provide radio and mobile communication, with point-to-point connections, internet redundancy and security monitoring.

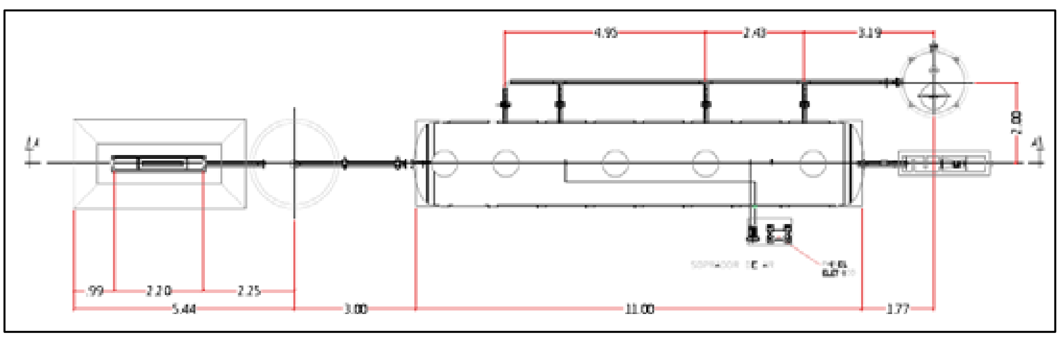
All communication links are connected to a switch core located in each shelter at both sites and in the distribution switches of each area; this connection is via fiber optic and then distributed by category 6 cabling and wireless to the buildings.

18.9 SEWAGE

The Ernesto site has a fully built sewage treatment plant (600-PK-001) (Figure 18.16). The sewage station is designed to treat 36 m^3/day of sewage biologically. The product from the sewage treatment plant goes to the tailings dam.

The Pau-a-Pique site has a smaller sewage treatment plant (621-PK-001). The sewage station is designed to treat 12 $m^{3/}$day of sewage biologically. The product of sewage treatment plant joins to the mine dewatering flow at the Pau-a-Pique site.

Figure 18.16 Site's Effluent Treatment Plant



18.10 MATERIALS DISPOSAL CENTER

At Ernesto and Pau-a-Pique sites there is a materials disposal center ("MDC"), where waste is received, classified, stored for short periods and then sent for final destination to qualified companies specialized in waste disposal (incineration, landfill or recycling). The MDCs handle all different waste classes from hazardous to inert.

When Ernesto underground starts, this building may be split / adjusted to accommodate the underground equipment workshop (currently available only at Pau-a-Pique), due to its favorable location closer to Ernesto's underground portal.

19.0 MARKET STUDIES AND CONTRACTS

19.1 GOLD PRICE

Aura does not have any forward sales or streaming gold contracts in place that are applicable to the Project, and future gold revenue will be according to spot prices on public markets.

The base case financial model for the Project utilizes a gold price of US$1,300/oz. This price remains fixed for the life of the Project. For comparison, the 48-month trailing average price for gold that existed on the effective date of this Report was approximately US$1,317/oz.

19.2 UMICORE DORÉ REFINING CONTRACT

Aura's Brazilian operating company, Apoena, has a contract with Umicore Brasil Ltda. ("Umicore") to refine its gold and silver. The contract was updated on January 1, 2016, and states costs of $R220.66 per kilogram of gold and $R33.05 per kilogram of silver for sampling, analysis and refining services.

The contract is subject to an escalator of 60% of the monthly salary (labor price) index and 40% of the wholesale price index, from the contract date of January 1, 2016.

There is a 100% gold credit and 95% silver credit applicable.

Freight and insurance to Umicore is to be paid by Apoena, and thereafter Umicore is responsible. Umicore testing and sampling results are sent to Apoena, and should be within an acceptable margin of difference of 0.05% for gold and 0.5% for silver, otherwise Apoena must notify Umicore of the difference and reanalysis will be done. Apoena can monitor the Umicore re-assay process. If a difference outside the acceptable margin still persists then the Apoena assay will be accepted as the basis for calculating refined metal.

Gold is refined within 4 days of receipt and silver within 15 days.

19.3 BRINK'S BULLION TRANSPORT CONTRACT

Apoena has a contract with Brink's - Segurança e Transporte de Valores Ltda. ("Brink's") for the shipment of up to 120 kg of doré or $R10,500,000 value per shipment. The charge is $R61,153 per shipment, plus there is a custody fee of 0.01% of the shipment value and an ad valorem tax of 0.06% of the invoice amount. The contract is dated November 13, 2016.

19.4 DINEX OPEN PIT MINING CONTRACT FOR LAVRINHA

Aura has contracted Dinex Engenharia Mineral Ltda. ("Dinex") to mine the Lavrinha open pit Deposit. The contract is based on haul distances and unit costs per tonne for waste and ore applied to the Lavrinha mine plan, plus unit costs for auxiliary equipment usage. Equipment maintenance is included in the unit costs. Table 19.1 lists the expected average unit costs by operating area.

TABLE 19.1 SUMMARY OF LOM CONTRACT MINING COSTS FOR LAVRINHA		
Operating Cost Area	**Ore (US$/t)**	**Waste (US$/t)**
Drilling	0.38	0.22
Blasting	0.40	0.30
Loading	0.40	0.31
Hauling	0.77	0.69
Aux. Equipment	0.20	0.20
Geology	0.06	0.06
Planning	0.04	0.04
G&A (Overhead)	0.06	0.06
TOTAL Mining Operating Cost	**2.31**	**1.88**

The major equipment in the fleet is specified as Volvo excavators, CAT dozers, Scania trucks and Sandvik drills. The contract term is 24 months, and is to be done by contract phase, with Phase I at 450kt/month to the end of April, 2017, and Phase II at 750kt/month to the end of mine life.

Prices have been scheduled to escalate by 20% of the Consumer Price Index and 80% of the Wholesale Price Index, from the contract date of April, 2016.

There is a 60% minimum monthly volume under the contract, i.e., if there is a stoppage for any reason 60% of the monthly contract volume shall be paid for.

If the contract is cancelled by Aura then the Company must pay for the value of the equipment, or approximately $R 16 million, approximately $4.5 million.

There are various performance specifications under the contract, including schedule, safety, productivity, and execution of Aura's mine plan and daily operating procedures.

Included in the contract is an escalating scale of penalties for lower achievement than contract target values in the various performance specifications, up to 10% of contract billing.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 INTRODUCTION

The objective of this section is to present the key environmental aspects of the development of the Project as described in this Technical Report. The descriptions found herein are based on the review of available Project information (namely the 2009 EIA and 2010 Technical Report), a review of available Project-collected data, a recent site visit and discussions with the Project team.

The Ernesto and Pau-a-Pique Deposits of the Project have existing production facilities and workings which have been in a state of temporary closure since 2014. The Lavrinha Deposit has seen near-surface garimpeiro operations and exploration over several decades.

Current activities on the Project's sites include monitoring and care and maintenance, with open pit mining of the Lavrinha Deposit and ore stockpiling recently initiated in September 2016. Section 13 of this Technical Report provides a summary of the Project's production history.

20.2 REGULATORY LANDSCAPE

Brazil is a federal republic, and its legal system is based on Civil Law tradition, characterized by codification of legal requirements. The Federal Constitution, enacted in October 1988, is the basis of the legal system.

The Federal Constitution defines mineral resources as property of the Federal Union, and ensures that states, Federal District and municipalities benefit from the exploitation of such resources. The Constitution also defines that rights to surficial land use and subsurface resources are distinct, and that the holder of mineral rights is entitled to ownership of the mined product. It also ensures that the landowner is compensated for mining activities.

In addition to the above constitutional provisions, the mining sector is regulated by the Mining Code (Decree-Law No. 227, February 28, 1967) and its Regulations (Decree No. 62934, July 2, 1968), and the Normas Reguladoras de Mineração ("NRMs"; Mining Regulation Standards), established by Ordinance of the Departamento Nacional de Produção Mineral ("DNPM"; National Department of Mineral Production). Other federal, state and municipal regulations apply to the mining sector, such as those related to taxation, the environment and land use.

The DNPM, the Ministério de Minas e Energia ("MME"; Ministry of Mines and Energy) and the federal, state and municipal environmental protection agencies are the main bodies of regulation and supervision of mining activities. The authority to tax mining related activities is defined in the Federal Constitution. There are no specific socioeconomic rules applicable to the mining industry, except for those prescribed in the environmental legislation.

20.2.1 Mining Regulations

Mining rights are granted by means of an Ordinance issued by the MME. As mentioned above, title of mineral rights does not imply ownership or possession of the surficial land above the minerals. According to Brazilian mining legislation, mineral rights titleholders do have the right to use and access areas to be explored and mined. This includes rights of way and easements, over public and private lands.

Surface rights are usually acquired by agreements between the holder of the mineral right and the landowners. Such agreements involve rental payments for occupation of the area, proportional to the economic use of the land, and compensation for damages to property.

If an agreement with a landowner cannot be reached, the holder of the mineral right can follow a legal procedure, as defined in the Mining Code, whereby the judge of the judiciary district where mineral right is located defines the amount of rent and compensation to be paid to the landowner. Besides establishing the judicial procedure, the Mining Code also defines parameters and limits for the definition of the values of rent and compensation.

The description of mine easements required for Project development and a Plano de Aproveitamento Econômico ("PAE"; Mine Development Plan), must be submitted to and approved by the DNPM in order to be granted a mining concession (Portaria de Lavra).

20.2.2 Environment Permitting

Environmental obligations are based on the polluter-pays principle, as outlined in the Federal Constitution, which states that "those who exploit mineral resources are obliged to reclaim the degraded environment, according to the technical solution as determined by the competent government agency". In the specific case of mining, the environmental licensing process covers the Project from planning, to construction, operations, and closure.

The Federal Government, the states and municipalities have joint authority to monitor companies' compliance with environmental legislation and to impose administrative sanctions such as fines, interdictions and restrictions of activities, tax incentives and benefits. The State environmental agencies are responsible for the environmental licensing of mining activities, except in the cases of Projects with interstate or major Projects which are managed at the Federal level.

The basis of the environmental legislation is defined in the Federal Constitution, in the Forest Code (Law No. 12.651, dated May 25, 2012), the Law on the National Environmental Policy (Law No. 6938, August 31 1981) and the Conselho Nacional de Meio Ambiente ("CONAMA" - National Council of the Environment) Resolutions Nos. 1/86, 9/90, 10/90 and 237/97.

As a general rule, CONAMA Resolution No. 237/97 establishes the criteria for environmental licensing of any Projects or activities intended to use environmental resources, considered effectively or potentially polluting, or likely to cause environmental degradation in any way. The resolution also defines three stages for obtaining of environmental licenses.

The environmental licensing procedures applicable to the Project are specified in CONAMA Resolution No. 9/90, which establishes the rules for environmental licensing of Projects subject to the mining concession regime. In the State of Mato Grosso, the environmental agency responsible for environmental licensing is the Environment State Secretariat – SEMA (EPA-MT), as provisioned by the State Complementary Law Nº 38 of Sept/1995, updated by Complementary Law Nº232/05, which determines the State's Environmental Code.

Environmental licensing occurs in three phases, namely:

- Licença Prévia (“LP”; Planning License): planning stage, when environmental impact studies are required. License must be by request to the relevant environmental department for Project development or expansion. This license does not authorize construction, but approves the environmental feasibility review of the Project and authorizes its location and conceptual design. It also sets the conditions to be considered during the detailed design.
- Licença de Instalação (“LI”; Construction License): this license refers to the construction stage of the Project itself, and to the implementation of environmental control projects and programs. The application for the LI must contain the Environmental Management Plan (“PCA”; Plano de Controle Ambiental), including a detailed Project description and specific action plans to mitigate and monitor environmental impacts described in the LP phase. The validity of this license is established by the activity or Project installation schedule, and is generally no greater than six years. Undertakings which require deforestation also usually require a “Vegetation Removal Authorization” (Autorização para Supressão de Vegetação).
- Licença de Operação (“LO”; Operation License): The application for the LO can only be submitted after the obtainment of the mining concession and the implementation of the environmental action plans comprised in the PCA. An inspection is conducted to verify that all technical details and requirements described in the approved design have been implemented and to verify compliance with provisions of the LP and LI. The license authorizes the start-up of the works/Project and is generally valid for 4 to 10 years.

Other licenses and permits may also be required for activities such as: water use, storage and diversion, infrastructure construction, production, transportation, storage and use of controlled materials and explosives (under the authority of the Federal Police and Brazilian Army); processing, disposal or transportation of waste; reuse, recycling and sale of waste; and wastewater discharge, among others.

For water usage and discharge, in addition to environmental licenses there are specific authorizations required from the Agência Nacional de Águas (“ANA” – National Water Agency) for water extraction, construction of pipeline and installations and for water discharge (Law No. 9984, July 17 2000 and Decree No. 3692, December 19 2000). A specific request and authorization is needed for each of those.

Another key legislation relevant to the Project is the above mentioned Forest Code (Law No. 12.561/2012). The federal code along with State Complementary Law Nº 38/95 define forested areas that are to be permanently preserved (Areas of Permanent Preservation or APP). These areas include riverbanks, springs, slopes and hill tops. Deforestation within APPs is only allowed for certain activities (including resource and mineral extraction of substances such as gold ore), and only upon approval of the proponent’s Plano de Recuperação de Área Degradada (or “PRAD”). This plan provides for both compensating and mitigation measures.

20.2.3 Socioeconomic Legislation Considerations

No specific socioeconomic rule is defined in the legislation applicable to the mining industry. In its turn, the environmental legislation sets rules for public consultation within the scope of licensing processes of mineral Projects.

An environmental impact assessment and its respective Relatório de Impacto Ambiental ("RIMA" - Environmental Impact Report) must be submitted and approved as a condition for obtaining of environmental licenses for mining Projects. The State environmental agency - in the case of the EPP Project, the Environment State Secretariat must assess such report, and may call public hearings for discussion of the RIMA and information about the Project and its environmental impacts. Also the Public Prosecutor's Office, civil entities or groups of at least 50 citizens may request the environmental agency to call such public hearings.

Public hearings must be recorded in appropriate minutes, which must contain all documents presented and produced during the hearings. The minutes and attachments, in addition to the RIMA, shall be used as the basis for the assessment and decision on the application for an environmental license (CONAMA Resolution No. 9/87).

20.2.4 Environmental Compensation Legislation

Federal Law No. 9.985, dated July 18 2000, provides that for the environmental licensing of Projects with significant impact, the Project owner has the obligation to support the implementation and maintenance of conservation units. The Law does not specify the exact meaning of the term 'support', however, it establishes a minimum of 0.5% of the total estimated costs for implementation of the Project to be allocated to support conservation units.

The amount of resources to be allocated for conservation unit support is to be set by the competent environmental agency, taking into consideration the degree of environmental impact caused by the Project. In view of the lack of clarity for the definition of the value of the compensation, the Supremo Tribunal Federal ("STF" - Federal Supreme Court) decided for 0.5% of the implementation cost (representing the highest value anticipated in Brazilian law) to be allocated for purposes of environmental compensation.

20.2.5 Health and Safety Legislation

Occupational health and safety standards that must be observed by mining companies are detailed in NRM 22. The rules establish standards for work procedures and safety conditions, emergency operations and personnel training, among others. Mining companies must also comply with or implement health and safety programs established in the Consolidação das Leis do Trabalho ("CLT"; Consolidation of Labor Laws), such as occupational health control program, environmental risks prevention program, in-house commission for loss prevention, social security professional profile and risk-management program.

20.2.6 Mine Closure Legislation

In regard to mine closure, Decree No. 97632 dated April 14 1989, which regulates the Law on the National Environmental Policy, generally establishes that a plan for reclamation of degraded areas must be a part of the EIA. In addition to that, NRM 20, specifically defines the administrative and operational procedures to be adopted in the case of "Suspension, Closure and

Resumption of Mining Operations"; while NRM 21 defines the rules for "rehabilitation of the explored, mined and impacted areas."

According to NRM 20, the Mine Closure Plan must be included in the Mine Development Plan (a requirement for the application for the mining concession, as already mentioned above), and must be updated periodically as appropriate. The prescription for periodic update of the closure plan allows some flexibility to adjust the closure plan to the reality of the operation, however, no substantial change in the closure plan originally approved by the environmental agency can be made without appropriate assessment and approval.

Under NRM 20, the suspension, closure, and resumption of mining operations must be authorized by the DNPM, which, in its analysis, shall consider the terms of the Program for Rehabilitation of Degraded Areas ("PRAD") submitted as part of the PCA. Justifications for mine closure must also be submitted to DNPM for approval along with detailed information on works performed; characterization of the remaining reserves; facilities/equipment demobilization plan; mine plan indicating mined areas, areas affected, reclaimed and to be reclaimed, disposal of organic soil, waste, ore and tailings, access roads and other civil works; a monitoring program comprising: (i) disposal and containment, (ii) slopes, (iii) groundwater, and (iv) drainage; soil, air and water pollution control plan; effluent discharge control plan; blocking of access ways and dangerous areas; definition of environmental impacts in the areas of influence; future use of the area; topographical/landscape conformation of the area; report on workers' occupational health; and physical and financial schedule of proposed closure activities.

There are no specific legislated requirements for reclamation or closure bonding for mining Projects.

20.3 EXISTING ENVIRONMENTAL AUTHORIZATIONS

The Project has a valid Operating License for mill feed and processing for Ernesto and Pau–a-Pique, issued July 2013. The same license allows mining activities at Ernesto. Such license expires July 21, 2016 and the company filed for license renewal on February 1, 2016. Under Brazilian environmental legislation, once the application for renewal of the LO is submitted at least 120 days prior to the expiration date, the license remains valid until the environmental authority decides on the application for renewal.

The Project has a valid Operating License to carry out exploration and mining activities for Pau-a-Pique. Such license expires May 20, 2017, and thus requires renewal filing prior to December 21, 2016.

Both Ernesto and Pau-a-Pique have existing Portaria de Lavra concessions from the DNPM.

Mining of Lavrinha mineralization is currently permitted under two main authorizations: i) a Guia de Utilizacao from the DNPM issued on September 9, 2016, and ii) Licença de Operação para Pesquisa Mineral (LOPM) from SEMA, the Environment State Secretariat. These authorizations allow the extraction of up to 50,000 t of mineralized material, and this threshold is slated to be achieved in late November to early December 2016. An application for extraction of an additional 250,000 t of mineralized material was submitted to the DNPM on November 23, 2016. This is expected to allow the continuance of operations until the definitive mining concessions are granted.

The decision to apply for a 250,000 t extension was taken after the Company's press release of November 22, 2016 was issued, aiming to minimize the risk of disruption of operations. As such, it was taken into consideration that the authority to issue a Guias de Utilização of up to 50,000 t is with the Superintendent of the local office of the DNPM in Mato Grosso State, and above this limit the competence is with the Director-General, at the head office of the DNPM in Brasília. In view of the limitations of resources noted at DNPM's office in Mato Grosso State, which could delay the assessment of the Project's applications (both for the Guia de Utilização and for the mining concession), it was decided to apply for a 250,000 t extension, bringing the case to the head office.

Aura requested the mining concession for Lavrinha on August 21, 2016, and the application is under review by the DNPM and expected to be granted in late 2016 or early 2017. This concession will allow mining in accordance to the company's submitted mine plan, or Plano de Aproveitamento Econômico. Aura advises that the Lavrinha's full environmental operating license (LO) will be released by SEMA once the Portaria de Lavra is issued. The LO includes permission to remove up to 54.29 ha of native vegetation to allow for construction of a waste rock storage area adjacent to Lavrinha pit. In the meantime, Lavrinha waste rock is being placed within existing footprint of the Ernesto mine disturbance.

The Project has a valid Operating License for its water intake facility on Córrego Lavrinha, supplying the mine, processing and general services needs for the Ernesto site, to a maximum withdrawal rate of 100 m^3/h.

35% of the Ernesto site's surface property held by the Company is a designated Legal Reserve, in compliance with the Forest Code's provisions pertaining to conservation of native vegetation in rural properties. Maintenance, monitoring and security of the Reserve is the responsibility of the Company.

Certificates authorizing the use of explosives and chemicals at Ernesto and explosives at Pau-a-Pique were issued by the Brazilian Army on September 29, 2016.

20.4 NATURAL SETTING

The site's general climatological, geographical and geological setting and geologic characteristics are described in Sections 5 and 7 of this Technical Report; additional information relevant to environmental effects and management are provided in the following paragraphs.

20.4.1 Soils and Rock Characterization

Soils in the Ernesto and Pau-a-Pique areas tend to be strongly leached and poorly developed, classified as lithosols and cambisols, locally referred to as saprolites. Baseline studies conducted prior to mining indicated no chemical values exceeding guidelines.

Acid rock drainage prediction studies were carried out for Ernesto and Pau-a-Pique as part of the Project's environmental impact study. The EIA describes 25 drill core samples selected for Ernesto, each consisting of an interval of mineralization along with the immediately adjacent 1 m of non-mineralized material. A total of 17 and 8 samples were taken from the Lower Trap and Intermediate Trap mineralization zones, respectively. No further documentation of drill hole origin and intervals has been located in site files and databases. Samples were analysed via the Modified Acid Base Accounting method (EPA 530 94/36). The EIA summarizes the results as indicating that 3 of the 25 samples, all from the Intermediate Trap mineralization zone, present potential for acid rock drainage.

A total of 10 samples were reported in the EIA to have been collected for Pau-a-Pique using the same methodology and analysis as for Ernesto. Results indicate that 3 of the 10 samples show potential for acid rock drainage, and a further 2 samples fall within the 'uncertain' category of acid rock drainage potential.

Partial information was also located on 5 kinetic tests carried out on Pau-a-Pique (4 samples) and Ernesto material, sometime during the mine production period of the two sites. Results for the first 36 weeks of testing show circum-neutral pH conditions in all cells, although with 1 of the cells containing material selected for its 'potential acid generation' showing some indication of increasing acidity and sulphate. No further documentation of sample origin and complete test results has been located in site files and databases.

To date, no acid rock drainage predictive studies have been carried out for the Lavrinha Deposit, nor is there any documentation of testing on tailings. Additional confirmatory test work for waste rock in all mine areas as well as for tailings is recommended.

20.4.2 Vegetation, Flora and Fauna

The Ernesto and Pau-a-Pique areas are considered to be located within the Cerrado Biome, a vast tropical and semi-humid savanna in central Brazil.

The 2009 Project EIA estimated 52% of the 6,500 ha in the area of Ernesto to be anthropogenically affected by either garimpo or grazing activities, with 21% of the area classified as seasonal semi-deciduous, and 23% as 'cerrado sensu stricto' canopy cover (orchard-like vegetation with trees about 6 m high). Just over 3% of the area was deemed as floodplain lowlands.

The EIA estimated 34% of Pau-a-Pique and surrounding area to be anthropogenically affected, 47% as 'cerrado sensu stricto', and 19% as seasonal semi-deciduous.

One plant species, Myracroduon urundeuva – locally known as the aroeira or timber tree, occurs in the Project area and throughout many parts of the country. This plant is listed as Vulnerable under Brazil's endangered species list.

The Cerrado Biome of Brazil is recognized as a species-rich habitat, and during 2008 fauna surveys a total of 150 animal species were registered in the Ernesto area, and 191 species in the Pau-a-Pique area. 'Listed' species based on the 2007 the International Union for Conservation of Nature (IUCN) list include 1 reptile with low risk of extinction, 5 mammal species listed as vulnerable, and 1 species (Alouatta guariba or brown howler monkey) is listed as critically endangered.

20.4.3 Surface Water

The Project areas are within the Amazon basin and the Rio Guaporé sub-basin. At the Ernesto and Lavrinha areas, local tributaries drain either to Córrego Lavrinha to the north or Rio do Cágado to the south. Both of these watercourses coalesce into the Rio Guaporé approximately 15 km to the northwest.

In the Pau-a-Pique area, eastward surface water flows report to the Córrego Corredor which ultimately drains to the Rio Aguapeí approximately 20 km to the southeast. Westward surface flows drain to tributaries that in turn merge with the Rio do Alegre approximately 12 km to the west of the site.

The region has a number of relatively small manmade reservoir lakes, and some springs were reported at the base of the Ernesto Deposit upstream of the existing tailings impoundment. Flow in surface drainages varies seasonally with precipitation, with higher flow rates during summer months (January to April). Lowest flows tend to be seen from June through August. Baseline water quality studies for the EPP Project were carried out over two campaigns in 2008. Surface sampling was conducted at points downstream of the Project sites, and upstream as relevant.

Surface waters in the Ernesto area of influence tend to be of neutral pH range and moderately to well-oxygenated. A wide range of alkalinity values were observed during the 2008 campaigns, possibly reflecting instances of purely precipitation-influenced chemistry (lower alkalinity) versus geologically-influenced chemistry (higher alkalinity). Sites sampled were slightly elevated in dissolved aluminum, manganese, phosphorus and sulfide. Suspended sediment load was evident at some sites where total metal concentrations were several times higher than dissolved levels. Elevated zinc was noted as occurring in at a sample site draining the west side of the site, but this value could not be substantiated upon review of the laboratory analysis reports. Mercury was not detected in any of the samples.

Surface waters in the Pau-a-Pique area of influence are similar in quality to Ernesto with the exception that pH is somewhat higher, in the neutral to slightly alkaline range.

20.4.4 Ground Water

Aquifers of relevance to the Project include both fractured, open systems that are largely connected with surface water, as well as some instances of semi-confined and confined deeper systems.

Groundwater in the southern areas of Ernesto near the existing tailings impoundment tends to occur in the 5 m to 10 m depth range in unconfined aquifers, and flowing southward consistent with the hydrographic regime. In the area of ridge where the Ernesto Deposit is located, aquifers are considered to be confined, and flowing northwards. Depth to water in the area of the Ernesto Deposit is estimated to be 20 m to 40 m below surface, with water intercepted during mining of the lower benches of the Ernesto pits.

Groundwater in the Pau-a-Pique area has been assessed only at a basic level. The area shows similar characteristics to Ernesto, with both fractured open systems and deeper semi-confined and confined aquifers. Most surface drillholes in the area of the deposit showed depth to water of greater than 30 m.

Groundwater in the Ernesto area tends to be of neutral pH range, with highly variable levels of oxygenation, and elevated in dissolved iron, manganese and aluminum.

20.4.5 Adjacent Protected Areas

As mentioned in the previous section, Aura maintains a 233.97 ha Legal Reserve along the eastern portion of the Ernesto Property. The nearest federal or state-designated conservation unit is the 120,000 ha Parque Estadual Serra de Santa Bárbara, approximately 19 km southwest of the Pau-a-Pique site. As reported in the Ausenco Technical Report for the Project, there is very basic state monitoring and management of this protected area with large portions of the land base still held privately.

20.4.6 Socioeconomics

The Ernesto and Pau-a-Pique sites are situated within the adjacent municipalities of Pontes e Lacerda and Porto Esperidião, which are two of the 23 municipalities within Mata Grosso's state planning region VII. Based on 2015 government estimates, the municipality of Pontes e Lacerda has just over 43,000 inhabitants and Porto Esperidião approximately 11,500 inhabitants.

Census data from 2007 included in the Project's EIA submission indicates that approximately 65% of Pontes e Lacerda inhabitants live within urban centres, while only 35% of Porto Esperidião's population is considered urban. In general, agriculture and livestock are main economic activities in the region along with mining of various scales and timber extraction. Pontes e Lacerda has a more developed infrastructure for healthcare, municipal water and sewage, emergency services and education.

The region also sees some conflict around ownership and use of lands, with involvement of farmers, garimpeiros, and 'landless people' (Trabalhadores Rurais Sem Terra).

There are three distinct recognized and titled indigenous lands in the region of the Project. All are within the municipality of Pontes e Lacerda and are over 25 km to the northwest of the Ernesto site. The Project's EIA does not note the existence of culturally significant sites affected by the Project.

There are no communities or permanent dwellings within the Project footprint.

20.5 EXISTING SITE CONDITIONS

The Ernesto and Pau-a-Pique facilities have been in a state of temporary closure since suspension of operations in late 2014. Project personnel and contractors provide maintenance, security, monitoring and inspections. The Ernesto and Pau-a-Pique sites were visited and reviewed on May 16 and 17, 2016. It was not possible to visit the Lavrinha area during this visit due to land transfer procedures still in progress. Aura advises that since the May 2016 site visit, agreements have been reached with the landowners in the area, allowing mining activities to proceed over the life of mine.

No evidence of potential environmental issues was observed during the May 2016 site visit, or was evident during review of information.

Only minor evidence of erosion was noted on the Ernesto waste rock pile. The rock appears to be highly weathered with minimal signs of the presence of pyrite and sulphide oxidation. There are no external waste rock piles at Pau-a-Pique apart from waste rock utilized for construction of the work platforms mainly in the portal area.

The Ernesto open pits showed no evidence of large scale instability or sulphide oxidation. Water was present in lower levels of both pits.

The tailings storage facility showed adequate freeboard and no major evidence of dam erosion. The last independent safety inspection of the facility was carried out in September 2016.

Maintenance dewatering of Pau-a-Pique underground workings was being carried out during temporary closure. Water was being pumped to a small pond near the main portal, and re-used for industrial purposes with a small amount infiltrating downstream.

In May there was minimal inventory of reagents on site, and all hazardous products appeared to be appropriately stored. There were adequate solid waste management facilities and during the site visit all wastes appeared to be appropriately stored prior to removal from site.

Domestic effluent treatment plants at both Ernesto and Pau-a-Pique were functioning, albeit with a significantly reduced influent load.

Aura was carrying out required surface and ground water quality monitoring. No compliance issues were noted.

20.6 ENVIRONMENTAL MANAGEMENT

During temporary closure the site Environment, Health and Safety program has been managed by two professional-level supervisors along with technician and labour assistance. Offices are located at the Ernesto site. During previous mine operation the group was also responsible for managing a native plant nursery on the north side of the Property. The nursery has facilities for seed collection, processing and storage, composting, and propagation of up to 60,000 plants per year (Figure 20.1).

Environmental personnel during operations will include a Health, Safety and Environment Coordinator, an Environmental Analyst, a Tailings Storage Facility Operator, one contracted waste management position, and labour support from provided from health and safety staff.

A review of monitoring data indicates that Aura is complying with the monitoring, inspection and surveillance programs stipulated in operating licenses for Ernesto and Pau-a-Pique.

Figure 20.1 Nursery Facility at Ernesto

20.7 CONSIDERATIONS FOR THE PROJECT

20.7.1 Mine Workings and Resource to be Mined

The updated mine plan includes a new area, Lavrinha, in addition to the Ernesto and Pau a Pique Deposits. Lavrinha is within 1 km of the Ernesto Deposit and has similar geology.

Additional project disturbance for Lavrinha mining and waste rock storage is estimated by Aura to be in the order of 55 ha. Much of the existing pit area has been previously affected by smaller scale mining, and there are no permanent residences in the area. It is expected that noise, dust and vibration emissions from Lavrinha operations will be similar in scale to emissions during the 2013 to 2014 open pit mining of the Ernesto Deposit.

Underground mining will utilize both cemented waste rock fill and non-cemented waste rock fill in order to optimize ore recovery. The backfill process decreases the project footprint and is also expected to reduce the potential for surface subsidence.

20.7.2 Waste Rock Management

A new waste rock storage area is required for mining of the Lavrinha Deposit, as described in section 16.2.5. Approximately 14 Mt of waste rock is expected to be generated. Aura is in the permitting phase for this waste rock storage facility, as noted in section 20.3 There is available waste rock storage space within the Ernesto mine footprint for initial mining stages of Lavrinha. As stated in section 20.4.1, while similar in geology to Ernesto, confirmatory acid rock drainage characterization is recommended for Lavrinha waste rock.

Underground development is not expected to generate waste rock for disposal at surface. Virtually all waste rock generated through the Ernesto development will be utilized as either cemented rock fill ("CRF") or non-cemented waste rock fill. Moreover, Aura advises that approximately 445kt of additional waste rock backfill is required and will be sourced from the existing Ernesto waste rock pile. The CRF and aggregate preparation plant will be located immediately adjacent to the main portal. Throughput will be relatively small (approximately 45 t/h) and is not expected to produce significant noise and dust emissions. The screened products generated from the Ernesto waste rock pile are recommended to be used for underground backfill purposes only. Additional testwork can be carried out to determine its geochemical stability and suitability for other civil works applications.

Waste rock generated through mining of Pau-a-Pique will remain underground and utilized as waste fill. Backfilling of stopes and ore development voids is slated to require approximately 44 kt of additional rock fill, which can be backhauled from the Ernesto waste rock pile as required.

20.7.3 Ore Processing

There are no significant changes to the processing circuit or reagents with respect to the previous operating period. Aura advises that they are investigating the feasibility of switching to liquid sodium cyanide in place of the briquette form previously utilized, and that any requisite permitting will be obtained prior to implementing this change.

20.7.4 Water Management

Estimated fresh water consumption during normal operation is 70.6 m^3/h, below the permitted license limit of 100 m^3/h from the existing Córrego Lavrinha pump intake. Approximately 130 m^3/h is expected to be recycled from the tailings impoundment to the process plant.

Dewatering will be continuously carried out during mining of both the Ernesto and Pau-a-Pique underground developments. Groundwater water inflows to the Ernesto workings are estimated to be between 4 and 94 m^3/h for mean dry and maximum wet season, respectively. As noted in section 16.1.9.3, excess water will be pumped to the processing plant. Pau-a-Pique dewatering will continue to discharge to a small pond adjacent to the main portal, with some of this water utilized for drilling and other industrial purposes. The balance of water infiltrates downstream, and Aura routinely monitors to confirm compliance with applicable regulations and permit requirements. An updated dewatering estimate for Pau-a-Pique is recommended in order to ensure adequate surface water management capability.

The Ernesto waste rock pile (and similarly the Lavrinha waste rock pile, once constructed) are designed to free drain to minimize buildup of internal pore pressures. The Ernesto pile toes in to a lowland area just upstream of the tailings impoundment, as shown in Figure 20.2. No visible evidence of acid rock drainage is seen in the field, nor is obvious in the downstream water quality data for 2015.

Figure 20.2 Lowland Area at Toe of Ernesto Waste Rock Pile. Tailings Impoundment in Background

Minimal dewatering requirements are anticipated for the Lavrinha pit based on the elevation of groundwater in the Ernesto pit being some 50 m below the bottom of the proposed Lavrinha pit. As noted in section 16.2.8, confirmatory hydrogeological studies are recommended during Lavrinha mining operations.

Pits and waste rock piles occur on heights of land and require no major diversion structures for water management. The tailings storage facility is flanked by two diversion ditches which direct seasonal runoff around the impoundment and downstream.

The existing sewage treatment systems will be utilized during operation. Treated domestic effluent from Ernesto washrooms and work areas is discharged to the tailings impoundment. Effluent from the kitchen/cafeteria sources is collected in holding tanks and removed from site for treatment in Pontes e Lacerdo. Treated domestic effluent at Pau a Pique is exfiltrated to the local drainage basin.

Discharges from the Ernesto site include controlled releases of excess tailings impoundment water, in order to maintain sufficient freeboard at all times. These planned releases are expected to occur on an as-required basis throughout the Project life. Aura reports that the most recent impoundment water release occurred from July 8 to August 18, 2016 and totalled 243,242 m^3.

20.7.5 Tailings Management

As described in section 18.4, the existing tailings storage facility will be utilized to store tailings and recover and recirculate process water. As in the 2013-2014 operating period, the carbon-in-leach tailings slurry will be treated in a cyanide reduction tank prior to leaving the process plant; thus the tailings impoundment pond will contain minimal levels of cyanide.

The current design of the facility allows for additional dam raises, and Aura estimates that the current configuration will provide 11 months of storage before the next dam raise is required.

The dam has an internal vertical drain system to control the structure's phreatic surface. Drainage is collected and pumped to the impoundment pond (Figure 20.3).

Figure 20.3 Seepage Collection Pond, Below Tailings Dam

The tailings facility is routinely inspected and monitored, with the most recent independent annual inspection and review carried out in September 2016, and duly submitted to the Federal Ministry of Mines (DNPM). While noting the overall condition of the facility as "Satisfactory", the inspection report recommended several maintenance measures to be carried out prior to the 2016-17 rainy season in order to maintain optimum physical and hydraulic stability of the structure. These items primarily include regrading and vegetation control measures, and Aura has carried out the measures necessary to ensure integrity of the facility until the next scheduled dam raising in 2017.

Aura has recently commissioned an independent firm to carry out an engineering review of the dam, and anticipates completion of the review in late 2016. Aura anticipates updating the tailings water balance as part of the review recommendations.

As described in section 20.4.1, some geochemical characterization via the Modified Acid Base Accounting method has been carried out on ore/waste intervals in various areas of the Ernesto and Pau a Pique Deposits. However to date the tailings material has not been evaluated, and it is recommended that confirmatory testing be carried out within the first year of operation.

20.7.6 Other Infrastructure

The Project will utilize the same administration, energy, and transportation infrastructure as in the 2013-2014 operating period.

20.7.7 Solid Waste Management

The Project will utilize the same administration solid waste management facilities and system as in the 2013-2014 operating period.

20.7.8 Employment

Direct employment estimate is approximately 302 persons, similar to the previous operating period from 2013 to 2014.

20.8 SITE CLOSURE

Site closure costs are estimated at $6.0M, with an additional $1.0M allocated for supporting studies. These costs were reviewed by the author and found to be reasonable. The cost model assumes some closure-related expenditures during the operating period for studies and closure plan updates, as well as for decommissioning of completed mine areas such as the Pau a Pique underground workings (Table 20.1).

TABLE 20.1
CLOSURE MEASURES FOR MAIN PROJECT COMPONENTS

Area	Principal Closure Elements
Pre-Closure Supporting Studies	geochemical characterization, basic design and execution plans, physical stability evaluation on structures to remain post-closure (pit, waste rock pile, tailings dam), closure plan updates
Pits	drainage control, perimeter rock berm with revegetation, revegetation of upper slopes ramps
Underground Workings	removal of equipment and ventilation, electrical, and pumping infrastructure, plugging of ventilation shafts, concrete plug at portals
Waste Rock Piles	resloping, surface drainage control, soil cover where required, revegetation
Tailings Impoundment	removal of infrastructure such as pipelines, pumps and

TABLE 20.1 CLOSURE MEASURES FOR MAIN PROJECT COMPONENTS	
Area	**Principal Closure Elements**
	the emergency spillway, regrading towards existing drainage channels, surface drainage control, soil cover placement, revegetation
Industrial Areas (process, maintenance, warehousing, electrical, portal entrances)	soil contaminant characterization and hazardous waste and product removal, decontamination of equipment as required, dismantling and/or demolition of equipment and buildings, revegetation

The closure cost estimate has site monitoring and maintenance costs allocated for the post-closure period. Surface and water quality monitoring is expected to continue for five years following cessation of operations. Maintenance on revegetation areas as well as surface drainage systems is expected to be carried out for three years following closure.

It is recommended that supporting studies and comprehensive closure plan development be initiated within the first year of operation.

21.0 CAPITAL AND OPERATING COSTS

21.1 ERNESTO CAPITAL AND OPERATING COSTS

21.1.1 Ernesto Capital Cost Estimate

Construction of the Ernesto underground Project will start after the Pau-a-Pique mine lateral development has been completed. Additionally, within the evaluation of the Ernesto underground Project, it has been assumed that all Pau-a-Pique mining equipment will be transferred to the Ernesto Project progressively, and additional equipment units needed to complete the required fleet will be purchased/leased until the LOM average 800 tpd ore production level has been reached.

As per the current mine plan and schedule, the Ernesto underground Project reaches full production approximately six months after commencement, with production mainly from ore development and primary stope extraction. This early production is possible due to the Ernesto orebody location and its close proximity to surface. The Ernesto orebody will be accessed via a twin decline concept which serves as definition drilling and main access. Later in production this arrangement will create a loop for traffic fluidity and will assist with achievement of ventilation requirements.

An equipment schedule has been generated to reflect the ramp-up and transition from Pau-a-Pique to Ernesto development and the production phase. As mentioned above, the mine development equipment (complete with equipment operators) will be transferred from Pau-a-Pique to Ernesto, including a partial technical services team and supervisors. Required maintenance and overhead personnel will be employed to support the initial development program (the 6 month pre-production period). The remaining equipment and labour force from Pau-a-Pique will be transferred to Ernesto progressively as Pau-a-Pique winds down, and Ernesto moves into production to achieve the projected workforce level and mobile equipment units required for the remaining mine life.

Pre-production capital costs are estimated at US$6.36M over a five month period. The total capital cost for Ernesto has been estimated at US$23.0M which includes capital development, sustaining capital, allocated labour, and mobile equipment capital for the duration of the mine life. The capital development portion has been estimated at US$11.5M including US$4.5M for pre-production and US$7.0M for sustaining capital required until the mine closes.

The closure cost for Ernesto underground mine has been included in the consolidated cash flow and was estimated at US$3.0M. This closure cost is not included in the US$23.0M capital cost.

Sustaining capital expenditure for the remainder of the mine life has been estimated for completion of outstanding works including the following items:

- CRF surface plant
- Office equipment and existing equipment repairs
- Road resurfacing (crushed/screen aggregates)
- Replacement of small items i.e. face pumps, fans, electrical distribution boxes.

Capital development expenditures over the LOM at Ernesto are presented in Tables 21.1 and 21.2.

Table 21.1 Capital Expenditure for Mining LOM at Ernesto	
Capital Expenditure	**Total US$M**
Capital Development Direct Cost	6.68
Indirects (Equipment, Labour, Other)	16.28
TOTAL CAPEX (development)	**22.97**

Table 21.2 Capital Development Expenditure for Mining LOM at Ernesto					
Capital Expenditure (US$M)	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Total US$M**
Capital Development Direct Cost	4.06	1.86	0.51	0.25	6.68
Indirects (Equipment, Labour, Other)	7.81	4.49	2.66	1.32	16.28
TOTAL CAPEX (development)	**11.88**	**6.35**	**3.17**	**1.57**	**22.97**

Unit development costs are listed in Table 21.3.

Table 21.3 Summary of unit Development Capital Costs at Ernesto				
Description	**Unit**	**Direct Cost**	**Labour**	**Total**
Primary Decline 4.5 m x 4.5 m	Metre	$1,440	$369	$1,808
Level Development Waste 4.5 m x 4.5 m	Metre	$1,368	$369	$1,737
HW Exploration Drive 4.5 m x 4.5 m	Metre	$1,477	$391	$1,869
Development Ore	Tonne	$23.39	$13.12	$36.51
Primary Stoping	Tonne	$41.19	$13.12	$54.31
Secondary Stoping	Tonne	$24.14	$13.12	$37.26

Note: *all development includes only associated direct costs e.g. drilling, blasting, piping, electricals, ground support, equipment operating, and CRF backfill whereas the labour portion has been allocated under the direct labour cost.*

Underground and surface equipment rental costs are presented in Tables 21.4 and 21.5.

Table 21.4 **Summary of Monthly UG Equipment Rental (included in LOM Capex) at Ernesto**			
Equipment Class	**No of Units**	**US$/ mth**	**TOTAL US$/mth**
Underground Equipment			
Jumbo 2 Boom Hydraulic	3	24,944	74,832
LHD 410*	3	0	0
LHD 307	1	17,395	17,395
ADT Volvo 30TF Truck	3	11,104	33,313
Sandvik DS311 Bolter*	2	0	0
Blasting Truck	2	2,924	5,849
Anfo Loader*	2	0	0
Scissorlift GS*	2	0	0
Scissorlift Services	2	5,498	10,997
Boom Truck - Materials	1	7,148	7,148
Flatbed Truck - Materials	1	3,666	3,666
Shotcrete - Jumbo*	1	0	0
Shotcrete Mixer	2	4,289	8,578
Mine Grader	1	3,207	3,207
Backhoe Excavator Stoping	1	4,003	4,003
Track Loader Stoping	1	4,003	4,003
Stope Buldozer	1	5,214	5,214
Stope Jammer - CRF	1	4,003	4,003
Mine Fuel/Lube Truck	2	2,327	4,654
Personnel Carrier (16 man)	2	3,160	6,320
Subtotal UG Equipment	**34**	**102,884**	**193,179**

*Note: * units do not carry any rental cost as they are to be transferred from Pau-a- Pique (owned equipment)*

Table 21.5 **Summary of Monthly Surface Equipment Rental (included in LOM Capex) at Ernesto**			
Equipment Class	**No of Units**	**US$/ mth**	**TOTAL US$/mth**
Surface Equipment			
Surface Loader Volvo L120	1	8,111	8,111
Jacklegs / stopers **	6	11,438	68,626
ERN Mine Manager Truck	1	1,406	1,406
Mine Supervisor Kubota	1	750	750
Mine Maintenance Kubota	1	750	750
ERN Tech Services Kubota	1	750	750
Commander Max DPS (4 seater) Geology**	1	24,375	24,375
Subtotal Surface and DD Equipment	**12**	**47,579**	**104,768**

*Note:** units have been included as one-time purchases due to their low cost value*

21.1.2 Ernesto Operating Costs

Operating cost estimates have been built utilizing advance rate cycles and first principle derived costs for each heading and then applied against schedule physicals including, but not limited to, the following mining consumables: drilling, blasting, services, ground support, and Aura's supplied labour rates which were verified by an independent consultant that conducted a labour study. Total LOM costs and unit costs are presented in Tables 21.6 and 21.7.

TABLE 21.6
SUMMARY OF LOM OPERATING COST ESTIMATES AT ERNESTO

Operating Cost Area	US$M	US$/ t ore
Mining	43.12	49.69
Mining Overhead	11.38	13.12
Total OPEX	**54.50**	**62.81**

TABLE 21.7
OPERATING COST ESTIMATE BREAKDOWN AT ERNESTO

Cost Area	Unit Cost US$/ t ore
UG Definition Diamond Drilling	3.83
Development Ore	7.52
Primary Stoping	14.80
Secondary Stoping	7.71
Mine Indirect Cost	13.12
Direct Mine Labour	15.84
Total Mine Operating Cost	**62.81**

Labour costs have been based on scheduled manning requirements for the operations, in line with Aura's organizational chart. Salaries and benefit structures are calculated in accordance with current prevailing salary structures in Brazil for the prescribed positions. The salary structures and labour rates are compliant with the provisions required under Brazilian law. All on-costs have been factored into the labour rates, including bonuses, allowances for vehicle and accommodation (where relevant), annual leave, health insurance and medical provisions.

Mining costs have been developed based on a schedule of rates for underground production, development and diamond drilling, and utilize first principle developed rates. Costs of other inputs for mining operations, including provision of power, water and services, are based on existing contract rates with external suppliers and estimated consumption rates.

CRF costs are presented in Table 21.8 and include preparation of the CRF at a plant on surface near the Ernestro portal. The costs do not include hauling the CRF underground and placement. The CRF costs are included in the total mine operating cost estimate of US$62.81/t for Ernesto.

TABLE 21.8 OPERATING COSTS AT SURFACE FOR CRF			
Item	**US$M**	**US$/t Ore**	**US$/m^3 Fill**
Cement	1.80	2.08	10.10
Diesel	0.13	0.15	0.72
CRF Plant Power	0.04	0.04	0.21
Labour	0.48	0.56	2.73
CRF Plant Maintenance	0.02	0.02	0.09
Mobile Equipment	0.02	0.02	0.10
Services	0.01	0.01	0.03
Expenses	0.67	0.78	3.81
Total Operating Cost	**3.17**	**3.66**	**17.79**

21.2 LAVRINHA CAPITAL AND OPERATING COSTS

21.2.1 Lavrinha Capital Costs

Mining operations at Lavrinha have been contracted to Dinex on a full-service basis. Dinex will supply all equipment and maintenance, and the costs will be included in the contract operating cost. Therefore, there are no capital costs associated with the Lavrinha Project.

21.2.2 Lavrinha Operating Costs

The Lavrinha operating costs are based on an open pit mining contract with Dinex, as presented in Section 19 of this Report. A summary of the unit costs is presented in Table 21.9. Aura has been actively mining in this area of Brazil for over half a decade utilizing mining contractors.

TABLE 21.9 SUMMARY OF LOM CONTRACT MINING COSTS FOR LAVRINHA		
Operating Cost Area	**Ore (US$/t)**	**Waste (US$/t)**
Drilling	0.38	0.22
Blasting	0.40	0.30
Loading	0.40	0.31
Hauling	0.77	0.69
Aux. Equipment	0.20	0.20
Geology	0.06	0.06
Planning	0.04	0.04
G&A (Overhead)	0.06	0.06
Total Mining Operating Cost	**2.31**	**1.88**

21.3 PAU-A-PIQUE

21.3.1 Pau-a-Pique Capital Cost Estimate

Construction of Pau-a-Pique, including the Ernesto process plant and site infrastructure, was effectively completed by the previous owner Yamana at the end of 2012. In late 2014 the mine was placed on care and maintenance by Yamana. The existing infrastructure and installations are functional and require minimal work before mining recommences.

Sustaining capital expenditure for the remainder of the mine life has been estimated for completion of outstanding work, including the following items:

- Access and development of new stopes
- Surface maintenance shop upgrades
- Equipment up-front mechanical work and associated parts
- Office equipment and existing equipment repairs
- Roads resurfacing (crushed/screen aggregate)
- Small items (i.e. face pumps, fans, distribution boxes).

A total of US$7.8M for initial and sustaining capital expenditures is required over the LOM for Pau-a-Pique (see Table 21.10).

TABLE 21.10
PAU-A-PIQUE CAPITAL EXPENDITURE FOR MINING LOM

Capital Expenditure	US$M
Preproduction	0.97
Equipment Rental	1.11
Development	5.69
TOTAL CAPEX	**7.77**

Unit costs of development are presented in Table 21.11.

TABLE 21.11
UNIT COSTS OF DEVELOPMENT

Unit Development Costs	Direct Cost US$/ m
Primary Decline 4.5 m x 4.5 m	1,785
Level Development Waste 4.3 m x 4.5 m	1,628
HW Exploration Drive 4.3 m x 4.5 m	1,364

21.3.2 Pau-a-Pique Operating Costs

Operating cost estimates have been built from first principles, utilising historical advance rates, contractual rates for haulage, consumables and Aura’s labour rates. A summary by cost area is presented in Table 21.12.

TABLE 21.12 SUMMARY OF LOM OPERATING COST ESTIMATES		
Operating Cost Area	**US$M**	**US$/ t ore**
Mining	16.55	51.72
Mining Overhead	2.00	6.21
Total Operating Cost	**18.55**	**57.93**

Labour costs have been based on scheduled manning requirements for the operations, in line with Aura's organizational chart. Salaries and benefit structures are calculated in accordance with current prevailing salary structures in Brazil for the prescribed positions. The salary structures and labour rates are compliant with the provisions required under Brazilian Tax Law. All on-costs have been factored into the labour rates, including bonuses, allowances for vehicle and accommodation (where relevant), annual leave, health insurance and medical provisions.

Mining costs have been developed based on a schedule of rates for underground production, development and diamond drilling utilising first principle developed rates. Costs of other inputs into the mining operations, including provision of power, water and services, are based on existing contract rates with external suppliers and estimated consumption rates.

21.4 PLANT AND TAILINGS DAM

The processing plant will be fed with ore from all three deposits following the LOM schedules described in Section 16 of this Report.

An ore stockpile will be established by the primary crusher area which will allow stabilization of the monthly plant throughput.

During the first 26 months of operation, the processing plant will treat an average of 55,000 tonnes of ore per month; this average throughput will be primarily from the Lavrinha open pit and partially from the Pau-a-Pique underground.

After month 27, the Ernesto underground will become the sole source of ore feed to the plant as Lavrinha and Pau-a-Pique become depleted, and this will result in a lower average monthly throughput of 21,500 tonnes per month.

The operational costs presented in the sections below outline the two operational regimes that will be experienced by the plant at 55K tonnes/month and 21.5K tonnes/month.

21.4.1 Plant and Tailings Capital Cost Estimate

The Ernesto processing facility and additional site infrastructure were fully commissioned by the previous owner in late 2012. The current condition of the plant infrastructure and installations is good and functional with only minimal maintenance work before processing recommences.

An allowance of US$4.5M has been included in the Project financial model as part of the processing plant capex for major equipment replacements and/or sustaining capital work to be executed within the processing plant.

There is also US$3.7M for the tailings dam raise over the LOM. The first 3 m raise is scheduled by Q4 2017 at a total cost of US$1.2M which was estimated by DAM in its 2014 detailed engineering report. In 2016, Tierra Group, a company specialized in tailings dam design, was commissioned by Aura to review the previous detailed engineering package developed by DAM. The details of this work are discussed in Section 18 of this Report.

21.4.2 Plant Operating Costs

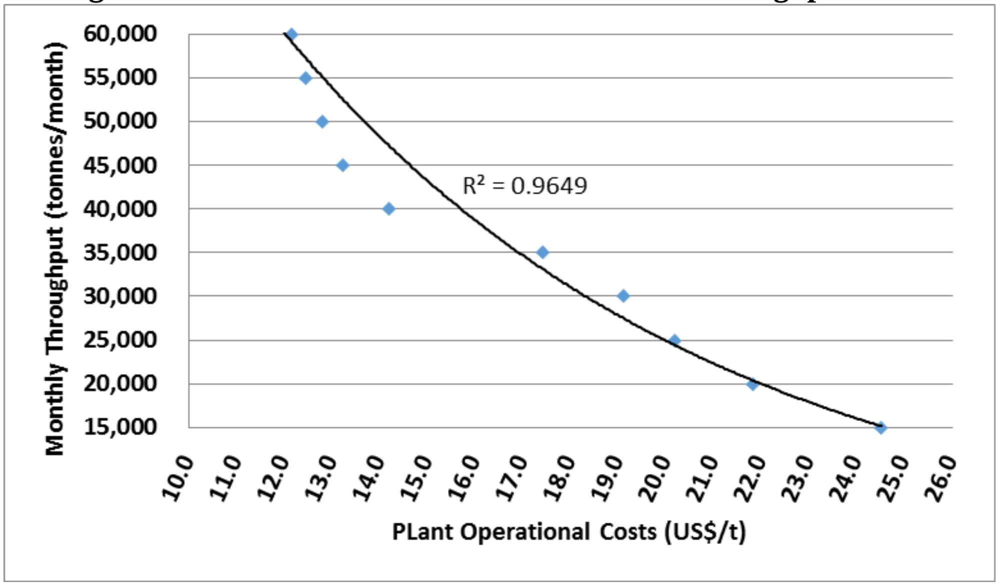
The processing costs are presented in two categories: fixed and variable costs. Fixed costs include plant labour and fixed contracts to operate the plant. Variable costs include all consumables, maintenance parts, power and other variable cost components. The processing cost for the 55 Kt/month production rate is estimated at US$12.5/t, and for the 21.5 Kt/month rate is estimated at US$21.3/t, as presented in Table 21.13.

TABLE 21.13
PROCESS PLANT OPEX BREAKDOWN

Cost Breakdown	55Kt/month (‘000 US$)	21.5Kt/month (‘000 US$)
Labour Cost	153.7	135.2
Contract Cost	39.7	26.5
Total Fixed Costs	**193.4**	**161.7**
Maintenance Cost	45.6	30.4
Consumables Cost	258.3	141.7
Power Cost	156.0	101.7
Contingency	32.7	21.8
Total Variable Costs	**492.6**	**295.6**
Total Cost (US$)	**686.0**	**457.3**
US$/t	**12.5**	**21.3**

Figure 21.1 shows the variation of processing costs at different throughputs.

Figure 21.1 Process Plant OPEX At Different Throughput Rates



21.4.2.1 Process Plant Fixed Costs

21.4.2.1.1 Process Plant Manpower

Table 21.14 shows the forecasted manpower costs for the processing plant presented between management, operations and maintenance.

The Project considers that most of the plant personnel will be sourced locally from the town of Pontes e Lacerda as local labour has mining expertise due to the other mines that operated in the area. It is also anticipated that some of the positions listed in the processing plant will be filled by Aura employees transferred from the Sao Francisco mine to the Project.

Labour costs were defined after a salary survey was conducted in early 2016 by Parametro RH, a human resources company based in Sao Paulo, Brazil. This survey provided average, maximum and minimum salaries and benefits for more than 150 employment positions based on eleven active mining companies operating in Brazil. This information was complemented by Aura's operating experience in the Mato Grosso area having operated two other gold mines.

TABLE 21.14
PROCESSING PLANT LABOUR BREAKDOWN

Area	Job Title	Schedule	# Employees		Total cost (US$)	
			21.5kt/ month	55kt/ month	21.5kt/ month	55kt/ month
Management	Plant Manager	Admin	1	1	13,818	13,818
	Maintenance Manager	Admin	1	1	4,846	4,846
	Plant Superintendent	Admin	1	1	2,138	2,138
	Administrative Support	Admin	1	1	3,237	5,444
	Plant Team Leader	5x2	1	2	9,126	9,126
	Control Room Operator	4x4	4	4	4,632	4,632
Crushing	Process Technician II	5x2	2	2	4,632	13,895
Grinding/Gravity	Process Technician II	5x2	2	6	7,828	7,828
CIL	Process Technician II	4x4	4	4	2,316	4,632
CIL	Process Technician II	5x2	1	2	4,632	9,263

Detox	Process Technician II	5x2	2	4	3,102	3,102
Smelter	Smelter I	Admin	1	1	2,320	2,320
	Smelter I	Admin	1	1	7,519	7,519
Maintenance	Planner	Admin	1	1	1,919	1,919
	Tools clerk	Admin	1	1	1,536	1,536
	Electrical Lead	Admin	1	1	3,331	3,331
	Instrumentation Technician	Admin	1	1	3,192	3,192
	Electrical Technician II	4x4	4	4	11,025	11,025
	Electrician II	4x4	4	4	9,333	9,333
	Mechanical Lead	Admin	1	1	2,756	2,756
	Mechanical Technician II	4x4	4	4	9,220	9,220
	Mechanic II	4x4	4	4	7,849	7,849
	Mechanic II	Admin	5	5	9,604	9,604
	Boiler Technician	Admin	1	1	1,921	1,921
	Lubricator	Admin	1	1	1,313	1,313
	Crane Operator	Admin	1	1	2,061	2,061
Total			**51**	**59**	**135,205**	**153,622**

21.4.2.1.2 Plant Fixed Contracts

Fixed contract costs for the plant are related to transportation, a front-end loader for operating the stockpile and other smaller contracts to support plant operation. The estimated costs for the 55K tonnes/month and 21.5K tonnes/month stages is US$39,700/month and US$26,500/month, respectively.

21.4.2.2 Process Plant Variable Costs

21.4.2.2.1 Processing Consumables

The process consumables are presented in Table 21.15 as two categories: 1) wear materials such as grinding media, liners, etc. and 2) chemical reagents for the process. All consumables are expected to be manufactured in Brazil, including sodium cyanide.

TABLE 21.15
PROCESSING CONSUMABLE COSTS

Materials	Specific Consumption (g/t)	Cost US$/t Ore	21.5Kt/month (US$/month)	55Kt/month (US$/month)
Wear Material				
Grinding Balls	550	0.74	15,992	40,911
Other wear materials	50	0.11	2,394	6,123
Crusher Liners	150	0.20	4,367	11,172
Cyclone lining	40	0.03	672	1,719
Mill Liners	337	0.46	9,798	25,065
Screens	30	0.08	1,713	4,383
Chemicals				
Sodium Cyanide	718* / 385	1.79* / 0.96	38,570*	52,906
Carbon	50	0.15	3,247	8,305

Lime	1,000	0.24	5,074	12,981
Thickener Coagulant	90	0.15	3,231	8,265
Sodium Metabisulphite	550	0.34	7,391	18,906
Copper Sulfate	250	0.65	13,874	35,492
Sodium Hydroxide	5.51	0.24	5,260	13,455
Hydrochloric Acid	5.00	0.05	1,008	2,578
Fuel			4,113	8,226
Gas (Boiler)			3,906	7,812
Leach Aid			21,070	0
Total (US$/t)			**141,691**	**258,300**

*Ernesto ore is expected to consume higher amounts of cyanide compared to Pau-a-Pique and Lavrinha ore.

Sodium cyanide is by far the most expensive consumable in the operation averaging US$2,490/t delivered to site. Cyanide costs have become more competitive in recent years leading to a potential opportunity to reduce costs.

21.4.2.2.2 Process Plant Maintenance Costs

Maintenance costs include general materials and spare parts used in the processing plant as well as small service contracts for electrical and mechanical activities.

The total maintenance costs will fluctuate between US$30,400/month and US$45,650/month depending on whether the plant is running at 21.5Kt/month or 55Kt/month, respectively.

21.4.2.2.3 Power Costs

The Project has a current power supply contract with the Mato Grosso Energy Utility Company ("ENERGISA") which is valid until the end of 2017. Under this contract, the cost per megawatt-hour ("MWh") is R$181.6 or US$56.7 at a foreign exchange rate of US$1.0:R$3.2.

The largest power consumer across the entire Project is the processing plant, for the crushing and grinding stages.

The power costs are expected to be between US$156,000 and US$101,000 per month for 55Kt/month and 21.5Kt/month, respectively.

21.5 GLOBAL G&A COSTS

The Project's operational cost includes a fixed global G&A cost which entails all related labour, consumables, and services that are used commonly by all operating mines, as shown in Table 21.16. In addition to the global G&A, each mine and the processing plant has its own local G&A costs.

Based on the mining schedule, the Project will have the Lavrinha open pit and the Pau-a-Pique underground producing at the same time for approximately 27 months and thereafter the Ernesto underground will become the sole source of ore to the plant. Based on this schedule, global G&A costs have been broken down into the two cases.

TABLE 21.16 GLOBAL G&A COSTS – TWO MINES VS. ONE MINE OPERATING		
OPERATION	**LAV + PPQ (‘000 US$)**	**ERN (‘000 US$)**
Labour Cost	1,614	1,406
Consumables Cost	123	103
Contract Cost	2,021	1,816
Others Cost	376	332
Total Cost (US$/year)	**4,134**	**3,658**

Table 21.17 shows the breakdown of the global G&A labour costs when two mine are operating at the same time.

TABLE 21.17 GLOBAL G&A LABOUR COSTS – TWO OPERATING MINES			
Area	**Job Title**	**Quantity**	**Annual Salary (‘000 US$)**
Security	Security Supervisor	1	29.9
	Safety technician	2	38.5
IT	Senior IT Engineer	1	122.9
	IT Analyst	1	34.8
Logistics	Contracts Analyst	2	69.6
	Contracts Assistant	1	23.5
Warehouse	Warehouse leader	2	61.7
	Technician	6	114.5
Infrastructure	Supervisor	1	62.2
	Technician	1	22.7
Services	Services Manager	1	75.0
Management	General Manager	1	150.0
	Junior Analyst	1	31.6
Human Resources	HR Manager	1	62.2
	HR Analyst	1	44.4
	HR Jr. Analyst	1	30.1
	Admin Assistant	1	22.7
Controller	Snr. Controller	1	74.1
	Analyst	2	51.9
Community/Projects	Project Manager	1	112.5
Safety	Snr. Safety Engineer	1	91.7
Health	Onsite Doctor	2	141.8
	Nurse	1	32.4
	Assistant	2	46.1
Environment	Environmental Eng.	1	34.8
	Tailings Specialist	1	31.9
	TOTAL (US$/year)		**1,614**

The global G&A contracts include items such as truck rental agreements for key operational personnel and management, site security, telecommunications contract (i.e. internet, phone, servers, printers), general site maintenance (i.e. vegetation clearance, air conditioning, pest control, etc.), environmental equipment, auditing costs and bus transportation for all employees from Pontes e Lacerda to site.

The global G&A consumables are mostly fuel for trucks and busses as well as IT, environmental and warehouse related items.

General liability and property insurance is based on the most recent premium and is expected to remain relatively constant at US$360,000 per year, which includes property and general liability coverage.

21.5.1 Freight and Refining Costs

Freight and refining costs for gold doré bars have been based on historical costs at the time of preparation of the LOM schedule, and are subject to market adjustment.

Gold transportation costs are accounted as US$9.44 per recovered gold ounce (gold sold) and the refining cost is estimated to be US$5.63 per ounce of payable gold (i.e. 99.99%).

21.6 PROJECT MANPOWER

Total manpower per annum is presented in Table 21.18. The labour summary does not include contractors.

TABLE 21.18
PROJECT LABOUR SUMMARY

Labour Category	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Site G&A	39	39	39	39	39	39	39
Ernesto Process Plant	59	59	59	51	51	51	51
LAV Open Pit Overheads	19	19	19	-	-	-	-
PPQ Underground	85	87	-	-	-	-	-
ERN Underground	-	-	116	183	184	183	183
TOTAL MANPOWER	**202**	**204**	**233**	**273**	**274**	**273**	**273**

22.0 ECONOMIC ANALYSIS

The economic analysis is based on the 2016 EPP LOM plan (comprised of the Lavrinha open pit, Pau-a-Pique and Ernesto underground Projects) including updated production plans, capital forecasts, manpower forecasts and operating costs for the Lavrinha, Ernesto and Pau-a-Pique Projects. The LOM plan covers the period 2016 to 2022.

Mining has been sequenced to start with open pit mining of the Lavrinha Deposit for a period of 28 months. Pre-production at the Pau-a-Pique underground mine starts one month after mining commences at Lavrinha, and lasts two months. Production mining at Pau-a-Pique is carried out for 17 months. Pre-production at Ernesto lasts five months and is scheduled to end when mining at Pau-a-Pique is completed. Production mining at Ernesto is then carried out for 43 months. The total LOM sequence is 69 months, or 5.8 years.

The LOM plan and economic analysis is based on Aura obtaining/renewing the necessary licenses and permits to re-commence mining operations in 2016 at Pau-a-Pique, and for the Ernesto underground Project around mid-2017, including the use of explosives and disposal of tailings into the existing tailings storage facilities near the Ernesto processing plant. Operational costs are based on a combination of Aura supplied costs for Lavrinha pit, processing, G&A, and P&E first principle estimates including all mining consumables, labour, maintenance, overheads, and mine services which are based on each Project mine schedule.

Capital works are currently under way to recommence mining at Pau-a-Pique, and the Lavrhina open pit has started pre-stripping with initial ore stockpiling. All work has been done in accordance with Brazilian mining law and regulations. The Ernesto underground Project is scheduled to start around mid-2018 and the initial capital expenditure is related to accessing the orebody and installing the required surface infrastructure. Sustaining capital for the Lavrinha pit and the Ernesto process plant have been estimated by Aura, with P&E providing the required capital estimates to sustain Pau-a-Pique and Ernesto underground operations over their mine life.

All costs are in 2016 US dollar nominal terms and inflation has not been considered in the cash flow analysis. Neither costs nor revenue have been escalated with any Consumer Price Index ("CPI") or other base commodities inflation. A 3.20 (BRA:USD) exchange rate has been used across all calculations in the financial model unless otherwise noted.

22.1 ECONOMIC CRITERIA

22.1.1 Revenue

- Average of 0.65Mtpa mined during the first two years of production from Lavrinha pit and Pau-a-Pique underground. Average 0.27Mtpa for the next three years from Ernesto underground, then 0.2Mtpa in the last year from Ernesto.
- Average gold production of 36,100 oz/year over the 5.8 year LOM.
- All ore mined is supplied directly to the process plant and treated with no substantial build-up of stockpile material.
- Average grade is 3.17 g/t Au at average recovery of 93% for Lavrinha and Pau-a-Pique and 86% for Ernesto.
- For the LOM, the Au price is $1,300 per ounce at 99.99% payable metal during refining
- Exchange rate for the LOM is 3.20 (BRA:USD)

- Royalty for the LOM is 2.5% with CEFEM Brazilian tax at 1.0%
- Revenue is recognised 30 days after the time of shipment of Au doré bars.

22.1.2 Costs

- Mine life from 2016 through to 2022
- Initial maximum capital cost of $18.2M (Partially funded by the Yamana Debt Facility of US$9.0M and an Aura Rights Offering in 2016 of approximately US$4.0M; including working capital and contingency)
- Capital costs of $38.9M for construction and sustaining of Pau-a-Pique and Ernesto underground Projects, Ernesto plant sustaining capital and $7.0M closure cost
- Average operating cost over the mine life of $71.36 per tonne milled
- Average production C1 cash cost per ounce of $837 for LOM.

22.2 NET PRESENT VALUE

The after-tax NPV at a 5% discount rate from 2016 through to completion of LOM for the base case is estimated at $28.5M and the IRR is estimated at 100%, with a payback of 1.2 years. Project economics are summarized in Table 22.1.

The after-tax undiscounted cash flow of the EPP Project is estimated at $36.4M over the LOM. The annual cash flow is negative in the first year and positive for the subsequent years.

TABLE 22.1
AFTER-TAX BASE CASE PROJECT ECONOMICS

Operating Statistics	**Life-Of-Mine (LOM)**
Ore Tonnes	2,298,000
Au (g/t)	3.17
Plant Recovery (%)	88.7%
Gold production (payable) oz Au	207,700
Cash cost US$/oz	837
All-in Sustaining cost US$/oz	1,064
Estimated Cash Flows	**(US$ 000's)**
Gold Revenue	269,996
Government Royalties	(2,700)
Refining and Transport	(3,130)
Net Smelter Return (NSR)	**264,167**
Mining costs	(104,766)
Processing costs	(36,783)
Total Project G&A	(22,449)
Private Royalty	(6,750)
Pre-tax Cash Earnings	**93,418**
Income taxes	(8,328)

TABLE 22.1 **AFTER-TAX BASE CASE PROJECT ECONOMICS**	
PIS/COFINS Credits[1]	8,328
After-tax Cash Earnings	**93,418**
Capital and Sustaining Capital	(38,946)
Closure Costs	(7,020)
Cash Flow to Entity	**47,452**
Debt Yamana *(Including Interest)*[2]	(11,016)
Cash Flow to Equity	**36,436**
NPV 5%	**28,517**
NPV 8%	**24,737**
NPV 10%	**22,540**
IRR	**100%**

(1) PIS/COFINS are tax credits under Brazilian Tax Regulation for exporters and those can be used to offset against income tax liabilities or refunded in cash.

(2) As previously disclosed, in order to facilitate the acquisition of the Project, the previous owner, SBMM, a company affiliated with Yamana, made available to the Company's operating entity a working capital facility of up to US$9M (the "Working Capital Facility"). The Working Capital Facility bears interest at 4% per annum on the outstanding balance. The funds advanced from the Working Capital Facility have been invested in the capital, care-and-maintenance and engineering requirements of the Project to restart the Project and to complete the NI 43-101 technical reporting. The Working Capital Facility is expected to be repaid with the initial free cash flow from the Project or will be payable in full by April 30, 2018. Should the Project not enter into production and the Company not have sufficient funds to repay the Working Capital Facility on the due date, such amount outstanding will, at the option of Yamana, be converted into common shares of the Company at a 10% discount over the 20 day VWAP of the Company's common shares based on the period prior to the due date. At no point in time may Yamana own, beneficially or otherwise, greater than 19.9% of the issued and outstanding common shares of the Company.

Table 22.2 presents a summary of the Project financial model.

TABLE 22.2 **EPP PROJECT FINANCIAL MODEL SUMMARY**								
LAV/ERN/PPQ - LOM Operating Statement								
		Y1	**Y2**	**Y3**	**Y4**	**Y5**	**Y6**	**TOTAL**
Ore Tonnes Mined	kt	623	669	306	256	257	187	2,298
Ore Tonnes Processed	kt	608	678	299	253	264	196	2,298
Processed Head Grade Au	g/t	1.78	2.13	3.69	5.58	5.31	4.37	3.17
Process Recovery	%	93	93	88	86	86	86	88.7
Gold Payable	%	99.99	99.99	99.99	99.99	99.99	99.99	99.99
Recovered Oz Au	kOz	32.4	43.2	30.6	38.7	39.0	23.7	207.7
Gold Price	US$/Oz	1,300	1,300	1,300	1,300	1,300	1,300	1,300
Sales								
Gold Oz Sold	kOz	29.1	44.6	29.9	38.2	39.6	26.4	207.7
Total Sales	**US$M**	**37.8**	**57.9**	**38.9**	**49.6**	**51.5**	**34.3**	**270.0**
Production Cost								
Mining	US$M	23.26	25.81	12.90	15.46	15.31	12.02	104.8
Processing	US$M	7.58	8.46	5.58	5.37	5.62	4.17	36.8

TABLE 22.2 EPP PROJECT FINANCIAL MODEL SUMMARY								
G&A	US$M	4.13	4.13	3.82	3.66	3.66	3.05	22.4
Total Operating Cost	US$M	34.98	38.40	22.30	24.49	24.59	19.24	164.0
Royalties & CEFEM Tax	US$M	1.32	2.03	1.36	1.74	1.80	1.20	9.4
Refining & Doré Transportation	US$M	0.44	0.67	0.45	0.58	0.60	0.40	3.1
Total Cost	**US$M**	**36.74**	**41.10**	**24.11**	**26.80**	**26.99**	**20.84**	**176.6**
Pre-Tax Cash Earnings	**US$M**	**1.05**	**16.83**	**14.75**	**22.84**	**24.48**	**13.48**	**93.4**
Tax & Credits								
Income Tax Payable	US$M	0.20	1.73	1.46	2.40	2.10	0.44	8.3
PIS & COFINS Credit Available	US$M	2.94	3.35	2.07	1.90	1.79	1.51	13.6
PIS & COFINS Credit Used	US$M	-0.20	-1.73	-1.46	-2.40	-2.10	-0.44	-8.3
After-Tax Cash Earnings	**US$M**	**1.05**	**16.83**	**14.75**	**22.84**	**24.48**	**13.48**	**93.4**
Capital & Sustaining Capital	US$M	7.89	6.26	11.08	7.50	4.95	1.27	38.9
Reclamation & Closure Cost	US$M	0.02	1.65	-	-	0.85	4.50	7.0
Yamana Loan	US$M	-	-	4.41	6.61	-	-	11.0
Net Cash Flow	**US$M**	**-6.87**	**8.92**	**-**	**8.00**	**18.67**	**7.71**	**36.4**
NPV @ 5%	**US$M**							**28.5**
IRR	**%**							**100**
Cash Cost (C1) / Oz Sold	US$/Oz	1,122	938	775	680	678	864	837
Mining Cost/Ore Mined	US$/t	37.32	38.60	42.10	60.43	59.48	64.45	45.59
Processing Cost/Tonne Processed	US$/t	12.47	12.47	18.66	21.27	21.27	21.27	16.01
G&A Cost/Tonne Processed	US$/t	6.80	6.10	12.75	14.49	13.85	15.55	9.77
Total Cost / Tonne	US$/t	56.59	57.17	73.51	96.19	94.60	101.27	71.36
All Inclusive Cost (C2) / Tonne	US$/t	69.29	68.99	109.68	125.49	117.13	132.20	96.16
All In Sustaining Cost / Oz	US$/Oz	1,300	1,083	1,072	819	793	1,092	1,064

22.3 SENSITIVITY ANALYSIS

Key economic risks for the EPP Project were examined by running sensitivity analysis on the following:

- Gold Price;
- Foreign Exchange (BRA:USD);
- Operating Costs; and
- Capital Costs.

The sensitivities and the impact on cash flows have been calculated for -15% to +15 % variations against the base case, as presented in Table 22.3. The analysis is presented graphically in Figures 22.1 and 22.2.

TABLE 22.3 SENSITIVITY ANALYSIS									
Gold Price Sensitivity After-Tax (US$M)									
US$/oz	**1,100**	**1,150**	**1,200**	**1,250**	**1,300***	**1,350**	**1,400**	**1,450**	**1,500**
NPV	-6.7	2.1	10.9	19.7	**28.5**	37.3	46.1	54.9	63.7
Net Cashflow	-5.1	5.3	15.7	26.1	**36.4**	46.8	57.2	67.6	78.0
IRR (%)	-9	10	31	59	**100**	166	288	565	1,632
NPV After Tax (US$M)									
% Change	**-15%**	**-12%**	**-8%**	**-4%**	**0%**	**4%**	**8%**	**12%**	**15%**
Capex	34.6	33.1	31.5	30.0	**28.5**	27.0	25.5	24.0	22.5
Opex	51.1	45.5	39.8	34.2	**28.5**	22.9	17.2	11.6	5.9
Net Cash Flow After Tax (US$M)									
% Change	**-15%**	**-12%**	**-8%**	**-4%**	**0%**	**4%**	**8%**	**12%**	**15%**
Capex	43.5	41.7	40.0	38.2	**36.4**	34.7	32.9	31.1	29.4
Opex	62.7	56.1	49.6	43.0	**36.4**	29.9	23.3	16.7	10.2
IRR After Tax (%)									
% Change	**-15%**	**-12%**	**-8%**	**-4%**	**0%**	**4%**	**8%**	**12%**	**15%**
Capex	146	133	121	110	**100**	91	83	76	69
Opex	1,055	435	240	150	**100**	68	46	30	17
BRA:USD Exchange Rate									
FOREX				**3.0**	**3.2***	**3.5**	**3.8**		
NPV (US$M)				18.7	**36.4**	39.3	48.4		
IRR %				54	**100**	252	969		

Note: ** represents Base Case scenario*

Figure 22.1 Project NPV Sensitivity Analysis

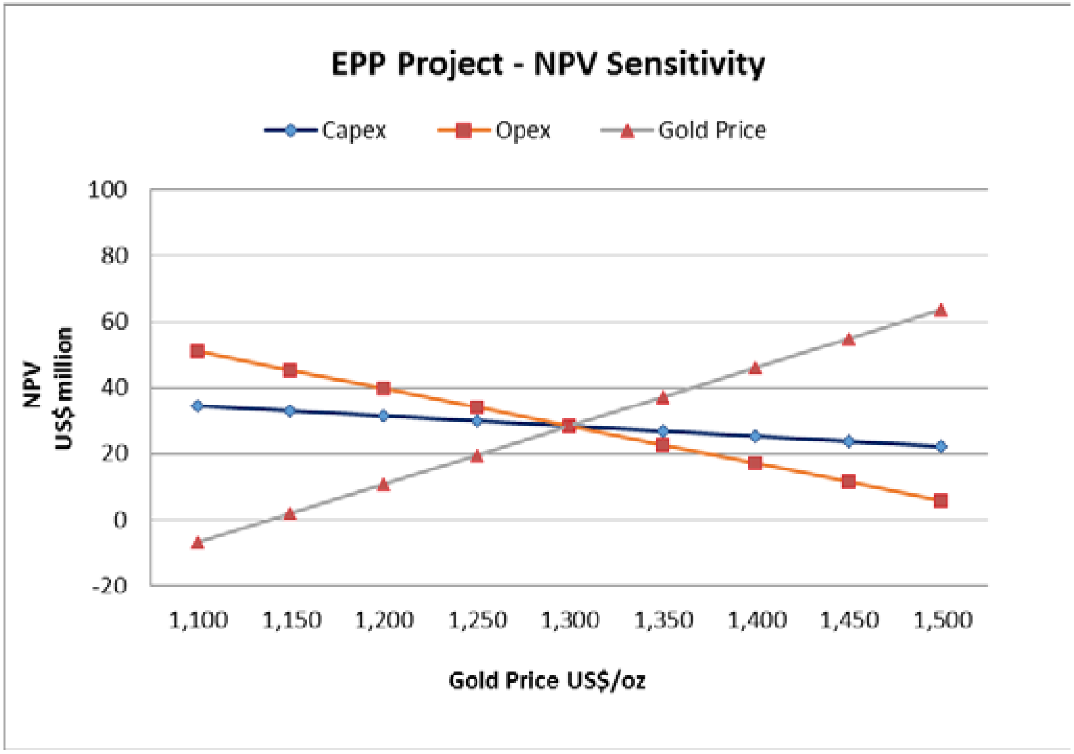
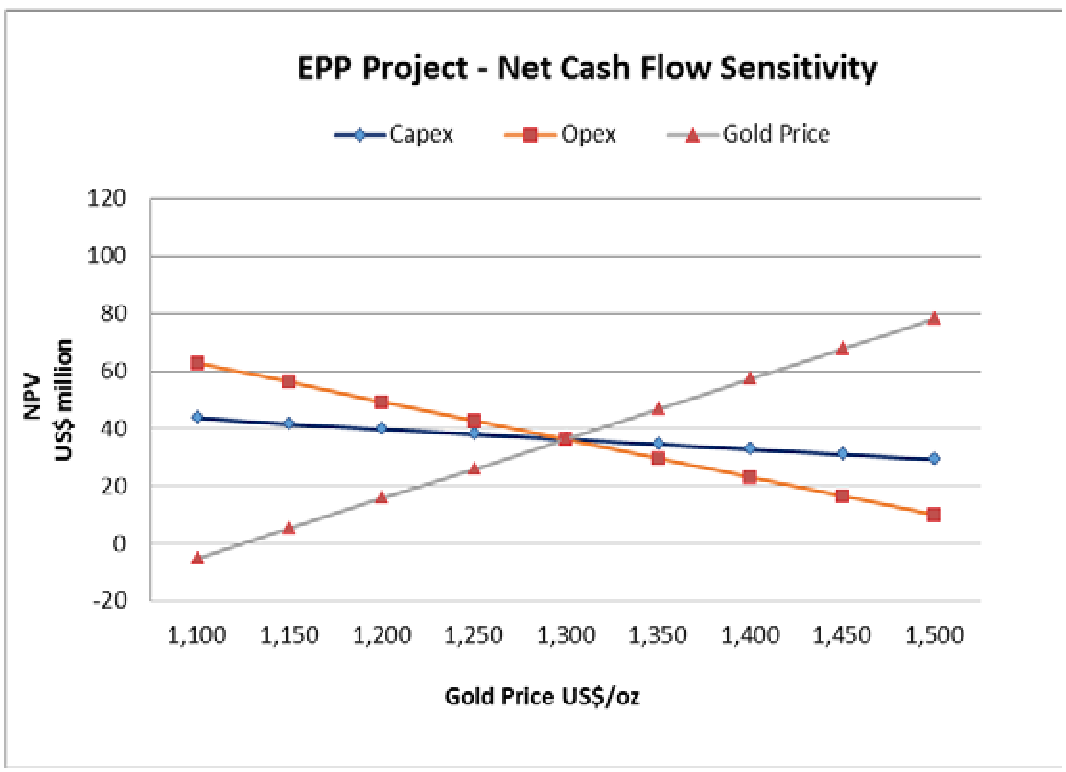


Figure 22.2 Project Net Cash Flow Sensitivity Analysis



22.4 ERNESTO HIGHER RECOVERY UPSIDE CASE

As was stated in Section 13 of this Technical Report, for the Ernesto Y3 H1 sample a complete retest was carried out, at the 106 micron grind, this being the only sample with sufficient weight remaining to allow it. The gravity recovery was down several percentage points but the intensive leach recovery increased from the previous 92.4% to 99.7% with the use of Leach Aid. This is an increase of 7.3 %. In view of this result a case can be made for increasing the other intensive leach recoveries, which could make the overall Ernesto recovery increase to 88% levels.

The Ernesto ore recovery was increased from the base case of 86% to 88% as an upside case to see the effects on overall Project economics. For the Ernesto 88% recovery case, and a US$1,300/oz gold price, the after-tax NPV at a 5% discount rate from 2016 through to completion of LOM is estimated at $31.3M and the IRR is estimated at 104%. Recovered gold over the LOM increases to 210,521 ozs compared to 207,689 ozs for the 86% recovery case.

22.5 CONSENSUS CASE

In a November 25, 2016, publication of Analyst Consensus Commodity Price Forecasts, CIBC Global Mining Group listed a gold price of approximately US$1,350/oz gold price for the the period 2017 to 2019. This price, along with a long term higher than base case foreign exchange rate of BRA:USD = 3.5:1 were input to the financial model. The after-tax NPV at a 5% discount rate from 2016 through to completion of LOM for the consensus case is estimated at $47.7M and the IRR is estimated at 497%.

23.0 ADJACENT PROPERTIES

There are no immediately adjacent properties to the Lavrinha, Ernesto and Pau-a-Pique Properties, however, Aura holds surface and mineral rights to the São Vicente and São Francisco Mines through Mineração Apoena S.A. ("Apoena"), a beneficially-owned indirect subsidiary of Aura. The São Francisco and São Vicente Mines are respectively located 68 km and 102 km northwest of the Ernesto Deposit (Figure 23.1).

Figure 23.1 Location of the São Vicente and São Francisco Mines

Source: *Google Earth 2016*

The regional geological setting for the São Vicente and São Francisco Mines is similar to the Ernesto Property and numerous other gold occurrences that comprise the Guaporé Gold Belt of central west Brazil and east central Bolivia. These gold deposits are hosted by the Aguapeí Mobile and Mafic Arc Belt. This belt is formed by a major crustal scale shear zone or break that separates the Archean Amazon Craton on the east from the Proterozoic Paragua Craton on the west. The belt extends for more than 600 km in a north-northwest direction and is characterized by a prominent mountain range composed of a 1,200 m thick sequence of Proterozoic clastic sediments known as the Aguapeí Group.

The following descriptions of Aura's properties in the Guaporé Gold Belt have been derived from previously filed technical reports. The QP's for this Report have not verified the information. The reader is cautioned that this information is not necessarily indicative of the mineralization on the Property that is the subject of this Technical Report. Furthermore, the Mineral Resources and Reserves for the properties that are described in this section have been depleted by mining operations subsequent to the September 2011 Mineral Resource and Reserve estimates that are reported below.

23.1 SÃO VICENTE MINE

The São Vicente Mine (Reid et al. 2012a) was an open pit, heap-leach operation located in the State of Mato Grosso, in the municipality of Nova Lacerda, on the eastern slope of the São Vicente Ridge. The Mine is located approximately 90 km north-northwest from the city of Pontes e Lacerda and 560 km northwest of Cuiabá, the capital of Mato Grosso State. The open pit is located at approximately 14°32’43.4” S latitude and 59°47’11.3’’ W longitude.

Reid et al. (2012a) report that gold mineralization at São Vicente has a strike length of over 1,000 m in two parallel northwest trending zones along the flanks of an anticline. The East Mineralized Zone shear strikes parallel to a metaconglomerate body close to the boundary between the Aguapeí Group rocks and the basement rocks. The gold mineralization is characterized by a combination of two main sets of quartz veins gently dipping to northeast and higher angle shear veins. The West Mineralized Zone follows the western limb of the São Vicente anticline and strikes parallel to the EMZ showing the same pattern of quartz veining as the EMZ. These two zones are situated within a larger regional area of shearing approximately 10 km long by 2 km wide and are proximal to the major regional shear zones.

The gold mineralization is associated with quartz and to a lesser extent with pyrite and with minor arsenopyrite. The quartz veins are typically millimetre to several centimetre-thick quartz veins that cut the host rocks in two prominent directions. The sub-vertical set of quartz veins are associated with mylonite shear zones and are sub-parallel to foliation in the meta-arenite host rocks. The other set of quartz veins are flat to shallow dipping and cross-cut the foliation and bedding of the host rocks. Free gold is common and is visible as fine to coarse grains, some up to 10 mm in diameter. Fine gold also occurs in sericite, sulphides and silicates.

In September 2011, Aura reported an NI 43-101 Mineral Resource estimated to be 5.97 million tonnes of Measured and Indicated Mineral Resource at an average grade of approximately 0.91 g/t Au and Inferred Resource of 0.29 million tonnes at an average grade of 0.46 g/t Au, using a 0.25 g/t Au run-of-mine dump leach ore cut-off grade and a 0.40 Au (g/t) crushed gravity leach ore cut-off grade.

The São Vicente Mine was operated as a conventional open pit supplying crushed gravity leach ore to the gravity circuit with dump leach ore going directly to the leach pad together with the higher grade crusher/gravity tails. As of September, 2011, Proven and Probable Reserves were estimated at 2.94 million tonnes at a grade of 0.84 g/t Au using the 0.25 g/t Au run-of-mine dump leach ore cut-off grade and 0.40 Au (g/t) crushed gravity leach ore cut-off grade. The mine ceased production in 2014.

23.2 SÃO FRANCISCO MINE

The São Francisco Mine (Reid et al. 2012b) is located in the western portion of Mato Grosso State in west central Brazil, close to the Bolivian frontier some 560 km west of the capital city of Cuiabá. The open pit is located at latitude 14°50’S and longitude 59°37’W. The São Francisco

Mine was an open-pit, heap leach gold mine. From 2011 to 2014, the mine has focused on a crushing-gravity gold recovery-heap leach process rather than run-of-mine heap leach. The ore contained a significant component of gravity gold, which required detailed sampling and attention to mine planning to ensure that the gravity gold was recovered prior to placement of ore on the leach pad.

The local rocks at the São Francisco Mine have been subjected to low-grade metamorphism. They consist of fine to coarse-grained meta-arenites (metamorphic sandstones), with locally reddish-coloured meta-pelites (metamorphic mudstones) and occasionally metaconglomerates (old pebble beds) of the Fortuna Formation, the basal unit of the Aguapeí Group. The rock units are folded into a series of broad folds that can be traced over several kilometres. The folds trend NNW-SSE and plunge NW. They are faulted and sheared, generally parallel to the folding, and are crosscut by fractures that strike WSW-ENE.

Reid et al. (2012b) interpreted the São Francisco Mine as a shear hosted lode gold deposit composed of narrow, 1 to 5 cm wide, quartz veins containing free gold. The veins and vein systems and stockworks both parallel and crosscut the bedding planes. Mineralization is enclosed by a steeply dipping, tabular hydrothermal alteration zone characterized by silicification with lesser sericite and chlorite. The gold occurs as free gold and frequently as coarse nuggets measuring several millimetres in diameter with the quartz, as laminations along the fracture planes, and within limonite boxwork after pyrite and arsenopyrite.

In September 2011, Aura reported an NI 43-101 Mineral Resource Estimate to be 10.92 million tonnes of Indicated Mineral Resource at an average grade of approximately 0.95 g/t Au and an Inferred Resource of 0.08 million tonnes at an average grade of 0.47 g/t Au, using a 0.23 g/t Au cutoff grade. The September 30, 2011 Mineral Reserve Estimate by Aura was 10.89 million tonnes of Probable Mineral Reserve at an average grade of 0.91 g/t Au using a 0.25 g/t Au cut-off grade.

The São Francisco Mine is still in production and operated by Aura, however, Mineral Resources and Reserves have been depleted by mining operations subsequent to the September 2011 Mineral Resource and Reserve Estimates.

24.0 OTHER RELEVANT DATA

P&E is not aware of any other data or information that is relevant to the EPP Project.

25.0 INTERPRETATIONS AND CONCLUSIONS

P&E concludes that financial modeling of the Project has determined that the Project will be economically viable and profitable. The Lavrinha Deposit is planned to be mined by open pit method, and the Pau-a-Pique and Ernesto Deposits mined by underground methods, utilizing the existing processing plant and tailings storage area, to produce gold. This Report outlines a total Project Proven and Probable Mineral Reserve Estimate of 2.3Mt at 3.17 g/t Au containing 233,600 ozs of gold. The Project has a low initial capital cost at US$18.2M since much of the site infrastructure is already in place. Overall Project economics are strong, with an after-tax NPV of US$28.5M, an after-tax IRR of 100%, and a payback of 1.2 years using the base case metal price of US$1,300/oz Au and a BRA:USD=3.2:1 foreign exchange rate. The Project mine life is planned at 5.8 years.

P&E concludes that this Report demonstrates the viability of the EPP Project as proposed, and that further development is warranted.

The following interpretations and conclusions have been drawn from this Report. The conclusions highlight items that characterize this Feasibility Study or are otherwise significant in terms of defining the Project.

25.1 PROPERTY DESCRIPTION

Title on the Property is in good order. Royalties exist on all deposits in the mine schedule.

The area to be developed represents only a fraction of the Aura land position, and several nearby exploration targets have been identified.

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

The Project’s local climate and geography allow for year-round mining. The Ernesto and Pau-a-Pique sites have existing suitable access for supply and services as well as for ore haulage, and there is adequate local skilled workforce availability in the region. There are no communities or permanent dwellings within the Project footprint.

The Ernesto Property contains a 130 tonnes per hour CIL process plant, which includes crushing, milling and tailing facilities with power supplied from the national grid via a 12 km 138 kV transmission line from Pontes e Lacerda. The Ernesto Property also contains a gate house, administration offices, core shack, explosives storage facility, and the mined-out Ernesto open pit and waste rock dump. The Lavrinha Property is contiguous to Ernesto and does not contain any infrastructure. The Pau-a-Pique Property contains an underground mine that was operated by Yamana until late in 2014, and surface facilities for administration and maintenance.

Aura has existing surface rights over most of the Project area either via direct ownership or agreements with landowners. Negotiations are in process for a remaining parcel in Lavrinha and a small portion of the Pau-a-Pique Project area. Aura is also updating the landowner agreements for resumption of ore haulage along the 47 km access between Pau-a-Pique and Ernesto; this process is well underway. While no impediments are anticipated for concluding these pending surface rights and access road use agreements, delays could stand to affect the execution of the Project.

25.3 HISTORY

The region has seen considerable exploration and mining activity over the past three centuries. Artisanal mining on the Property began in the 1980's, followed by several drill campaigns by mining companies in the 1990's.

25.4 GEOLOGICAL SETTING AND MINERALIZATION

Regional and local geology which controls mineralization is well understood.

25.5 DEPOSIT TYPES

The Ernesto-Lavrinha and Pau-a-Pique Deposits are broadly similar in host lithologies, structural style, alteration, and mineralization and all share characteristics of shear-hosted lode gold deposits.

25.6 EXPLORATION

Exploration of the Ernesto, Lavrinha and Pau-a-Pique Deposits has been comprehensive, and methodologies and practices applied are considered appropriate.

25.7 DRILLING

Exploration drilling on the Property is extensive. Drill campaigns have been carried out by previous companies since 2005. Aura drilled the Ernesto, Lavrinha and Pau-a-Pique Deposits in 2015, focussing on in-fill drilling in the mineral resource areas.

25.8 SAMPLE PREPARATION, ANALYSES AND SECURITY

It is P&E's opinion that sample preparation, security and analytical procedures for both the Ernesto and Pau-a-Pique Deposits drilling and sampling programs were adequate for the purposes of this Mineral Resource Estimate.

It is MCB's opinion that sample preparation, security and analytical procedures for the Lavrinha Deposit drilling and sampling programs were adequate for the purposes of this Mineral Resource Estimate.

25.9 DATA VERIFICATION

Based upon the evaluation of the QA/QC programs undertaken by Yamana and Aura, as well as P&E's due diligence sampling, P&E concludes that the data are of good quality for use in the Ernesto and Pau-a-Pique Mineral Resource Estimates.

For Lavrinha, MCB had the same conclusion as P&E since the Lavrinha drilling campaigns were carried out simultaneously with Ernesto, applying the same procedures and sampling protocols.

25.10 MINERAL PROCESSING AND METALLURGICAL TESTING

The EPP process plant started operation in 2013 and was operated until October, 2014, receiving feed from the Ernesto open pit and the Pau-a-Pique underground mine.

Samples of the three deposits (Ernesto, Pau-a-Pique and Lavrinha) were selected in 2016 from available core and sample coarse rejects to represent half years according to the production forecast for the Project. In the main, the core samples were sent for grinding testwork while the coarse rejects were sent for hydrometallurgical testing. SGS Lakefield, Canada, performed the grinding work, which consisted of SAG Power Index and Bond Ball Mill Work Index testwork, while SGS Geosol of Belo Horizonte, Brazil, performed the hydrometallurgical testwork, consisting of Gravity Recovery of Gold, bottle roll leach tests and settling testwork.

The grinding circuit has more than adequate capacity to handle the tonnages planned for the Project. In view of this it may be advisable to investigate whether it would be beneficial to grind finer.

The overall recoveries for the Pau-a-Pique and Lavrinha metallurgical testwork samples are very good at approximately 93%. Those for the Ernesto samples are lower than expected, at approximately 86%, even after the re-leach results are taken into account. Further work should be carried out on Ernesto material to ascertain the reasons for this. The work should investigate using finer grinds, increased cyanide levels and also the use of Leach Aid.

25.11 MINERAL RESOURCE ESTIMATES

25.11.1 Ernesto

The Mineral Resource Estimate for the Lower Trap zone at Ernesto was estimated by conventional 3D computer block modelling methods employing Dassault Systemes Geovia mining software V6.71. The Mineral Resource Estimate is based on surface diamond drilling, core sampling and gold assaying. Assaying was performed at SGS and ALS commercial laboratories in Belo Horizonte and at Yamana mine laboratories Ernesto and MFB as well as the Aura Sao Francisco lab, all in Brazil.

Gold mineralization of the Lower Trap zone at Ernesto consists largely of free gold hosted by mylonite, muscovite schist, and quartz veins accompanied by sulphides that occur along the sheared contact between meta-tonalite and meta-arenite. Mineralization is epigenetic, hydrothermal in origin and is structurally controlled. The rock foliation and mineralized contact trend NNW and have a shallow dip of approximately -25° NNE. The contact is not uniformly planar and is subject to rolling. The Intermediate Trap zones at Ernesto were mined by open pit from 2013 to 2014. Drill hole intersections of these zones are located in the meta-arenite rocks above the Lower Trap zone, however, the Intermediate Trap zones are not included in the mineral resource estimates in this Report. The Lower Trap zone has not been mined underground except by garimpeiro (illegal miners) in small workings at one site near surface. This site is outside the current resource area. The narrow widths of the Lower Trap mineralization and depth below topography all but preclude open cast mining and the Lower Trap zone is amenable only to underground mining.

The exploration drillhole database for the Lower Trap zone underground Mineral Resource Estimate area contains 329 diamond drill holes totalling 47,932.22 m. Drill hole lengths range from 9.10 m to 615.55 m. The Mineral Resource Estimate is defined by 87 drillholes.

The Mineral Resource Estimate wireframes were constructed from mineralization intersections in drill holes at a cut-off grade of 1.5 g/t Au over a minimum vertical mining width of 2.0 m. Gold price used for the resource estimate was US$1,275/oz. The 1.5 g/t Au cut-off grade was based on US$33/t for mining, US$11/t for processing, and US$10/t for G&A. Processing assumptions are 93% recovery, 99.99% for payable and $15/oz Au for doré transport and refining. Mineralization widths are commonly narrower than minimum mining width and were "bulked out" to at least the minimum width using adjacent assays.

Assay grades were capped at 40 g/t Au. Assay composites were generated for the zone intersections from the assays captured by GEMS software in the mineralized wireframes. Equal length composites were generated dynamically at a nominal 2.0 m down-hole length. This method ensures that the grade weighting is correctly applied for bulked out domain widths but results in variable composite lengths.

Two block models were created, a lithologic model for geologic interpretation and a resource block model. The X-axis of the resource block model is rotated to 95° azimuth. Resource block size is 10 m x 10 m x 2 m vertical which is suitable for selective mining and benching methods such as room and pillar, drift and fill and mechanized cut-and fill. Ordinary Kriging interpolation was carried out using multiple search distances and search ellipses oriented to the NE mineralization plunge. Inverse distance squared and nearest neighbour interpolation methods were employed for model validation.

Water immersion bulk density testing was carried out at Ernesto by Yamana for 627 core samples in 84 ER series holes and an additional 25 tests were performed during Knight Piesold geotechnical work in 2015, P&E due diligence sampling (6) in June 2015 and as a separate exercise by Aura personnel (8) carried out in February 2016. The Mineral Resource Estimate is contained almost entirely within mylonite-sericite schist (bulk density 2.62) and quartz veining (bulk density 2.62) of the Lower Trap and thus 2.62 t/m^3 was employed as the bulk density for conversion of resource volume to Mineral Resource Estimate tonnes.

The Mineral Resource Estimate was classified as Indicated and Inferred based on drill hole spacing, confidence in the assaying and geologic confidence in the zones interpretation and grade continuity.

The total Indicated Mineral Resource Estimate for a 1.5 g/t Au cut-off grade is 734,000 tonnes averaging 6.70 g/t Au (158,200 ounces gold). The total Inferred Mineral Resource Estimate for a 1.5 g/t Au cut-off grade is 308,000 tonnes averaging 6.30 g/t Au (62,400 ounces gold).

Validation of the grade interpolation and the block model was carried out by on-screen review of grades and other block model estimation parameters versus drill hole composites, by comparison of assay, composites, zone intersections and block grades, comparison to alternate ID2 and nearest neighbour interpolations, and review of the volumetrics of wireframes versus reported resources. In P&E's opinion, the Mineral Resource Estimate is reasonable and has been undertaken according to industry standard practice.

25.11.2 Lavrinha

The Lavrinha drill hole data originated from different drill campaigns that were determined by MCB to be in compliance with NI 43-101 quality control checks and data storage policies. The raw data was exported to text files for data manipulations, population statistics, geological/alteration/resource modeling and grade estimation. MCB's opinion is that the drill hole database including Au (g/t) grades and bulk density determination (t/m^3) are valid and suitable for a Mineral Resource Estimate. Lavrinha´s database stored in Access and Excel formats was provided by Aura, and totalled 165 drill holes and 20,867.41 m drilled, with a total of 20,372 samples analyzed for gold.

The dataset is comprised of three drilling campaigns with their respective objectives. The Lavrinha Deposit was linked to the Ernesto Deposit exploration strategies since it was considered to be its satellite deposit, therefore the same operational procedures, documentation and database management were applied.

The Lavrinha Deposit is located approximately 500 m west of the Ernesto Deposit with a strike of approximately 50°NE extending approximately 400 m along strike and 500 m down plunge. It is characterized by a swarm of parallel veins, sub-parallel to the plunge. Mineralized lodes occur with variable thickness up to 12 m and are distributed within a thick sericite-muscovite schist unit. The maximum depth from surface that mineralization is intersected by drill holes is approximately 150 m.

Geological interpretation and mineralized lode modeling of the Lavrinha Deposit were carried out using Micromine® software by MCB with technical support from Aura staff. The following tasks were performed:

- Surface geological mapping to define contacts, geological and structural features;
- Re-logging and data validation of all core focusing on lithology, hydrothermal alteration, sericitization and silicification;
- Interpretation of hydrothermal and geology features using two sets of cross-sections;
- Parallel plunge (azimuth N145° / Dip Vertical) – 6 m spacing;
- Perpendicular plunge (azimuth N55° / Dip 57°SE) – 25 m spacing for further fine tuning of the model; and
- Interpretation of mineralized lodes using 0.2 g/t Au envelope and geological contacts and alteration layers as hard boundaries.

The methodology used for sample compositing was to create 2 m down-hole composites, starting from the collar. Short intervals less than 0.5 m were excluded to avoid bias at the ends of drill holes intersections within the geological 3D domain.

Due to the volumetric significance of outlier grades, it was decided not to apply any capping and only reduce the influence search radius of these extreme values. The steps taken to treat these extreme values were as follows:

- Identification of the upper threshold for outliers. The threshold value was identified as Au >23.54 ppm;
- Identify the geological 3D domains that contain outlier samples within their boundaries;

- Estimate the grade of these blocks (only these blocks) with all samples (including outlier samples) with a regular grade interpolation search strategy; and
- Remove the outlier samples from the dataset (outliers samples are considered missing) and estimate the remaining blocks using regular search criteria.

A bulk density model based on rock type was used for volume to tonnes conversion with the Mineral Resource Estimate averaging 2.77 tonnes/m^3.

Geostatistics was used to populate the block model grades within geological 3D domains based on composite values that were stored in the form of constrained XYZ points. Mathematically this approach is regarded as an interpolation approach. The classification was based on three different search strategies (Measured, Indicated and Inferred Mineral Resources) with manual adjustments to remove irregularities. Each of the search strategies is based on a proportional value obtained from the technique called Quantified Kriging Neighborhood Analysis.

The Mineral Resource Estimate for the Lavrinha Deposit has been reported above a 0.5 g/t Au cut-off grade, inside an optimized pit shell with a gold price of US$1,300/oz. Mining costs were considered at US$2.44/t and US$1.89/t for mineralized material and waste haulage, respectively, plant process costs of US$10.24/t and G&A of US$3,800,000 per year as well as a process recovery of 93%. The total Measured Mineral Resource Estimate for a 0.5 g/t Au cut-off grade is 74,000 tonnes averaging 2.31 g/t Au (5,500 oz gold). The total Indicated Mineral Resource Estimate for a 0.5 g/t Au cut-off grade is 1,226,000 tonnes averaging 2.25 g/t Au (88,700 oz gold). This gives a total Measured plus Indicated Mineral Resource Estimate of 1,300,000 tonnes averaging 2.25 g/t Au (94,200 oz gold). The total Inferred Mineral Resource Estimate for a 0.5 g/t Au cut-off grade is 283,000 tonnes averaging 2.51 g/t Au (22,800 oz gold).

In MCB’s opinion, the Mineral Resource Estimate is reasonable and has been undertaken according to industry standard practice.

25.11.3 Pau-A-Pique

Pau-a-Pique gold mineralization consists largely of free gold accompanied by sulphides hosted in mylonite, muscovite schist, biotite schist, quartz veins as well as meta-tonalite and metaconglomerate that occur along and adjacent to sheared contacts between meta-tonalite and meta-conglomerate. Mineralization is epigenetic, hydrothermal in origin and is structurally controlled. There is a high “nugget” (40%) effect and high grades may be localized in areas of restricted dimensions.

The Mineral Resource Estimate for the Pau-a-Pique Deposit was estimated by conventional 3D computer block modelling methods employing Dassault Systemes Geovia mining software V6.4 and V6.71. The estimate was undertaken according to NI 43-101 standards.

CIM definitions were followed for the Mineral Resource Estimate and were based on:

- 32,554 m of surface diamond drilling and underground fan diamond drilling in 313 holes, core sampling and assaying as well as underground face channel chip sampling and assaying totalling 2,428 samples for 1,241.73 m.
- Wireframing at a 1.5 g/t Au cut-off grade over a 3 m minimum horizontal mining width.
- Ordinary kriging grade interpolation.

- Alternative estimations, using inverse distance squared and nearest neighbour methods, validate the ordinary kriging method in P&E's opinion.
- The wirefame cut-off results locally in narrow intersections of gold mineralization with grades >1.5 g/t Au on trend within the zones not meeting minimum mining width. For the generally lower grade footwall and hanging wall lenses P3 and P4 where drilling is less intensive and there is no mining history, the interpretation of mineralization continuity may be affected in that alternative interpretations of continuity are possible and confidence of the resource interpretation is reduced.

The Mineral Resource Estimate was classified as Indicated and Inferred based on drill hole spacing and data quality, channel sampling locations, confidence in the assaying and geologic confidence in the zones interpretation and grade continuity. P&E cautions that the Indicated Mineral Resources held in remnant pillars, sills and "skins" left in stopes may not all be recoverable pending engineering study.

P&E concludes:

- The geology and mineralisation of gold within the Pau-a-Pique mining lease are well understood. The geological models were appropriate to guide the Mineral Resource Estimates and have been and continue to be developed in a professional manner.
- Geology databases are professionally constructed and are sufficiently error free to support Mineral Resource Estimates.
- Data comprising the Mineral Resource Estimate at Pau-a-Pique are from a variety of drill campaigns with variable data quality.
- The quality assurance programs for all drilling campaigns are acceptable.
- The Pau-a-Pique Mineral Resource Estimate models were developed using industry accepted methods. Lithologic and structural interpretations were properly used in guiding and controlling grade interpolation. Data analyses were appropriately used to determine grade interpolation domains. Extreme grades were dealt with by capping high grades which sufficiently controlled over interpolation of such grades in all domains.
- Reasonableness of grade interpolation was reviewed by visual and statistical comparison of block model grades verses drill hole composites. Good agreement was observed.

In P&E's opinion, the Mineral Resource Estimate is reasonable and has been undertaken according to industry standard practice.

The Mineral Resource Estimate wireframes were constructed from mineralization intersections in drill holes at a cut-off grade of 1.5 g/t Au over a minimum mining width of 3.0 m. The Mineral Resource Estimate cut-off grade of 1.5 g/t Au was derived from a Au price of US$1,275 /oz, costs of US$29/t for mining, US$11/t for processing, US$10/t for G&A and US$7/t for mill feed surface transportation, at a 93% process recovery.

The total Indicated Mineral Resource Estimate for a 1.5 g/t Au cut-off grade is 519,000 tonnes averaging 4.05 g/t Au (67,600 ounces gold). The total Inferred Mineral Resource Estimate for a 1.5 g/t Au cut-off grade is 117,000 tonnes averaging 4.45 g/t Au (16,700 ounces gold).

25.11.4 Total Project Mineral Resource Estimate

The total Measured and Indicated Mineral Resource Estimate for the Project is 2,553,000 tonnes averaging 3.89 g/t Au containing 320,000 oz Au. The total Inferred Mineral Resource Estimate for the Project is 708,000 tonnes averaging 4.48 g/t Au containing 101,900 oz Au.

25.12 MINERAL RESERVE ESTIMATES

25.12.1 Ernesto

The Mineral Reserve Estimate is as of July 31, 2016, and was developed from the Mineral Resource Estimate model prepared by P&E. The Probable Mineral Reserve was derived from the Indicated Mineral Resource Estimate.

The cut-off grade (2.35 g/t Au) was based on a US$1,165/oz gold price, 93% metallurgical Au recovery, 99.99% payable, royalties and CEFEM tax totalling 3.5%, gold doré bar transport and refining costs totalling US$0.45/g Au, mine direct and mine indirect costs totalling US$62.41/t, US$10.30/t processing cost, and US$6.12/t processed for the projected share of the overall multi-mine global G&A cost that would be incurred by the proposed Ernesto underground mine Project.

The Probable Mineral Reserve Estimate for the Lower Trap zone of the Ernesto Deposit is 868,000 t at 5.03 g/t Au containing 140,000 oz gold.

25.12.2 Lavrinha

The Mineral Reserve Estimate is as of July 31, 2016, and was developed from the Mineral Resource Estimate model prepared by MCB. The Mineral Reserve Estimate was estimated at a cut-off grade of 0.48 g/t Au using an average short-term gold price of US$1,100 per ounce.

The Proven Mineral Reserve Estimate for the Lavrinha Deposit is 67,000 t at 1.85 g/t Au containing 4,000 oz gold, and the Probable Mineral Reserve Estimate is 1,043,000 t at 1.68 g/t Au containing 56,300 oz gold. The Total Proven and Probable Mineral Reserve Estimate is 1,110,000 t at 1.69 g/t Au containing 60,300 oz gold.

25.12.3 Pau-a-Pique

The Mineral Reserve Estimate is as of July 31, 2016, and was developed from the Mineral Resource Estimate model prepared by P&E. The Probable Mineral Reserve was derived from the Indicated Mineral Resource Estimate.

The cut-off grade (2.40 g/t Au) was based on a US$1,165/oz gold price, 93% metallurgical Au recovery, 99.99% payable, royalties and CEFEM tax totalling 3.5%, gold doré bar transport and refining costs totalling US$1.56/t, mine direct and mine indirect costs totalling US$58.08/t, US$12.50/t processing cost, and US$6.44/t processed for the projected share of the overall multi-mine global G&A cost that would be incurred by the proposed Pau-a-Pique underground mine Project.

The Probable Mineral Reserve Estimate for the Pau-a-Pique Deposit is 320,000 t at 3.24 g/t Au containing 33,300 oz gold.

25.12.4 Total Project Mineral Reserve Estimate

The Total Proven Mineral Reserve Estimate for the Project is 67,000 t at 1.85 g/t Au containing 4,000 oz gold. The Total Probable Mineral Reserve Estimate for the Project is 2,231,000 t at 3.20 g/t Au containing 229,600 oz gold. The Total Proven and Probable Mineral Reserve Estimate is 2,298,000 t at 3.17 g/t Au containing 233,600 oz gold.

25.13 MINING METHODS

Mining has been sequenced to start with open pit mining of the Lavrinha Deposit for a period of 28 months. Pre-production at the Pau-a-Pique underground mine starts one month after mining commences at Lavrinha, and lasts two months. Production mining at Pau-a-Pique is carried out for 17 months. Pre-production at Ernesto lasts five months and is scheduled to end when mining at Pau-a-Pique is completed. Production mining at Ernesto is then carried out for 43 months. The total LOM sequence is 69 months, or 5.8 years.

25.13.1 Ernesto

The Ernesto Deposit will be mined by a Drift and Fill method, using a combination of drifting in ore and transverse primary and secondary stopes. The orebody will be accessed by one main ramp, with a second access for definition drill access and ventilation purposes.

The definition drilling proposed jointly by Aura and P&E will be an important step in improving the understanding of HW and FW characterization and thickness and their influence on development and stoping activities at Ernesto.

The Ernesto Project will use the majority of the Pau-a-Pique Project's underground mobile equipment once Pau-a-Pique operations ceased. Additional units will be mobilized and commissioned to complete the required fleet.

The Ernesto cemented rockfill plant has been selected and sized to deliver the required backfill quantity and quality.

Secondary stoping maximum span is sensitive to the ability of mine personnel to consistently tight fill the primary stopes. Poor quality or delayed backfill will cause the main access drives to deteriorate and will make mining of the secondary stopes more difficult.

The presence of mylonite and its thickness will require re-analysis of ground support density and maximum stope span.

Ernesto's groundwater inflow estimates are sensitive to the hydraulic conductivity of the rock mass and the potential for intersections with water-bearing structures. The actual groundwater inflows could vary from the estimates, potentially requiring additional dewatering.

Ore dilution calculations were enhanced through the use of data and information provided by KP's geotechnical model and recommendations.

The current mine life of the Ernesto Project is four years.

25.13.2 Lavrinha

Aura has contracted Dinex to mine the Lavrinha open pit Deposit. The contract is based on haul distances and unit costs per tonne for waste and ore applied to the Lavrinha mine plan, plus unit costs for auxiliary equipment usage. Equipment maintenance is included in the unit costs.

The major equipment in the fleet is specified as Volvo excavators, CAT dozers, Scania trucks and Sandvik drills. The contract term is 24 months, and is to be done by contract phase, with Phase I at 450kt/month to the end of April, 2017, and Phase II at 750kt/month to the end of mine life.

25.13.3 Pau-a-Pique

Underground mining at Pau-a-Pique will be conducted by an Avoca choke blasting stoping method. Ore will be transported up the main access ramp and then along a 47 km surface road to the Ernesto process plant.

The overall mining and development strategy is believed to be suitable from a rock mechanics perspective given the expected rock mass conditions and the available historical data. The management of dilution will be critical to the success of the proposed mining approach. Dilution will need to be managed through a combination of the following factors:

- The placement of the ore drives
- Ground support practices
- Drilling and blasting practices
- Panel span
- Stand-up time of the panels

Existing mobile equipment and surface facilities on care and maintenance are in good order but will require thorough inspection and mechanical work. Costs have been allocated to address this matter.

The current mine life of Pau-a-Pique is one and a half years.

25.14 RECOVERY METHODS AND PROCESS DESIGN

The CIL processing plant is fully built, operational and serviced by all required infrastructure. The installed flowsheet is suitable to process ore from all deposits at estimated overall gold recoveries of 93% for Lavrinha and Pau-a-Pique ores and 86% for Ernesto ore.

The equipment installed in the plant is considered to be in excellent condition since the plant ran for less than two years and the preventive maintainance done during the subsequent care and maintenance period was diligent.

Reagent dossages considered in the baseline operating cost calculations have not been optimized and Aura expects to achieve lower reagent consumptions as it moves the Project into production.

Due to the soft nature and low abrasion index of the ores to be treated, the plant is expected to have longer cycles for liner changes at the crushing and grinding stages compared to operational history.

25.15 PROJECT INFRASTRUCTURE

The current infrastructure installed in the Project was designed to sustain a total production capacity of 90,000 tonnes per month. The new LOM mining sequence and more selective mining process will result in lower monthly throughputs compared to the currently installed capacity and this would enable the Project to run under two operating regimes: Lavrinha and Pau-a-Pique mines producing 55,000 tonnes/ month and then, Ernesto undeground producing stand-alone 21,500 tonnes/month. All infrastructure sourrounding the processing plant will accommodate and support future production plans.

The existing primary powerline and all electrical components (i.e. substations, etc) have been confirmed to have enough capacity to supply energy under the two operating regimes.

As outlined previously, the Ernesto Project will greatly benefit from the existing infrastructure installed at Pau-a-Pique, which will be systematically transferred to Ernesto as Pau-a-Pique operations ramp down.

The tailings dam facility will undergo a 3 m raise in 2017, which will provide additional tailings storage capacity for another 2.3 years. A final raise for the remainder of the Project will require further detailed study.

25.16 MARKET STUDIES AND CONTRACTS

The financial model is based on a gold price of US$1,300/oz. The 48-month trailing average price as of the effective date of this Technical Report was approximately US$1,317/oz. Gold revenue for the Project will be subject to spot prices.

Aura, through its wholly-owned Brazilian company Apoena, has contracts with Umicore to refine its gold and silver. It also has a contract with Brink's to transport doré.

Aura has contracted Dinex to mine the Lavrinha open pit. The contract is full service and includes providing all mining equipment, drilling, blasting, loading, hauling and maintenance.

25.17 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The Project has experienced and qualified environmental management staff and facilities in place. A review of the site, permits, and monitoring data indicate that Aura is complying with the monitoring, inspection and surveillance programs stipulated in operating licenses for Ernesto and Pau-a-Pique.

The Project has several key operating permits in hand to allow mining and processing activities to commence. The remaining permits and authorizations are in the application process, and there is reasonable certainty of obtaining these in due course. Delays in obtaining these pending approvals may in turn, delay or otherwise affect the Project, in particular, the cost-effective mining of the Lavrinha deposit.

The Project cost model provides for additional test work in 2017 for acid rock drainage studies for tailings and waste rock. If preliminary test results indicate potential for acid rock drainage and/or metal leaching, confirmatory studies and testing will be required. These findings may, in turn, indicate that additional prevention or control measures are required for waste rock and/or tailings.

25.18 CAPITAL AND OPERATING COSTS

25.18.1 Capital Costs

Initial capital for the Project is estimated at US$18.2M and is low since it is partially funded by the Yamana debt facility and since much of the Project infrastructure is already in place.

There are no material capital costs to mine Lavrinha since it is a contract operation. Pre-production costs for Pau-a-Pique are estimated at US$7.8M and for Ernesto are estimated at US$23.0M.

Plant sustaining capex is estimated at US$4.5M over the LOM, tailings dam raises are estimated at US$3.7M over the LOM, and closure costs are estimated at US$7.0M.

A contingency of US$2.3M has been included in the financial model.

25.18.2 Operating Costs

Operating costs for open pit mining at Lavrinha are based on the Dinex contract, and are estimated to average US$2.31/t ore and US$1.88/t waste over the LOM.

Operating costs for underground mining at Pau-a-Pique and Ernesto have been developed from first principles and contain known consumable unit costs, labour rates from a salary survey and rates paid during care and maintenance, existing electrical power rates, and known costs for other services. The average cost for mining at Pau-a-Pique over the LOM is estimated at US$57.93/t ore, and for Ernesto is estimated at US$62.81/t ore.

Processing costs have been developed from first principles, budgeted consumption rates, and quotations from suppliers. The processing cost for a 55 Kt/month production rate is estimated at US$12.5/t, and for a 21.5 Kt/month rate is estimated at US$21.3/t.

Global G&A costs are considered to be all labour, consumables and services that are used commonly by the mines such as general management, information technology, supply chain, human resources, HSEC and other services such as cleaning, site security, restaurant, etc.

Given that the Project will function in two defined operating regimes (the first regime with Lavrinha and Pau-a-Pique producing a combined 55,000 tonnes/month and the second regime having Ernesto producing stand-alone at 21,500 tonnes/month), the Global G&A costs have been structured as such. The annual cost for Global G&A is estimated at US$4.1M under the Lavrinha/Pau-a-Pique operation and US$3.6M for the Ernesto stand-alone operation.

25.19 FINANCIAL MODEL

The after-tax NPV at a 5% discount rate from 2016 through to completion of LOM for the base case is estimated at $28.5M and the IRR is estimated at 100%, with a payback of 1.2 years. The after-tax undiscounted cash flow of the EPP Project is estimated at $36.4M over the LOM.

The Ernesto ore recovery was increased from the base case of 86% to 88% as an upside case to see the effects on overall Project economics. For the Ernesto 88% recovery case, the after-tax NPV at a 5% discount rate from 2016 through to completion of LOM is estimated at $31.3M and the IRR is estimated at 104%. Recovered gold over the LOM increases to 210,521 ozs compared to 207,689 ozs for the 86% recovery case.

Using a consensus price forecast of US$1,350/oz gold, along with a higher than base case foreign exchange rate of BRA:USD = 3.5:1, the after-tax NPV at a 5% discount rate from 2016 through to completion of LOM is estimated at $47.7M and the IRR is estimated at 497%.

26.0 RECOMMENDATIONS

P&E specifically recommends proceeding with detailed engineering and preparations for production based on the positive economics predicted by the designs and financial evaluations contained herein.

26.1 ERNESTO

26.1.1 Mineral Resource Estimate

A number of holes were drilled that cut the Mineral Resource Estimate wireframe but were not sampled. Some holes were sampled up hole but not in the Lower Trap zone. The few holes with non-sampled explicit intervals within the zone also need to be sampled. Two holes did not reach the Lower Trap zone and should be deepened if possible. Holes in question are:

- ERN0076
- ERN0080
- ERN0084
- ERN0088
- ERN0074
- ERN0062
- ERN0078
- ERN0093
- ERN0098
- ERN0089 and relog bottom of hole
- ER059 and ER091 should be deepened by 50 m and 90 m respectively.

Several revisions of the drill hole and assay databases during the course of Mineral Resource Estimation for Ernesto indicated that the database received from Yamana had been incompletely verified and needs review and cross-referencing with original records for the drill hole and assay databases. This has been done for the Indicated Mineral Resource Estimate portion of the database but not for the fringe areas including the Inferred Resources. Down hole surveys should be thoroughly reviewed against original survey records since it is difficult to validate survey accuracy of azimuths in sub-vertical drill holes by simple on-screen reviews and routine examination. Completion of the re-logging to update the lithology database should be completed for drill holes in the Inferred Mineral Resource Estimate areas.

P&E reviewed the Yamana and Aura QAQC programs and the lab internal QAQC blanks and reference standards and in P&E's opinion, the assay database is acceptable for Mineral Resource and Mineral Reserve Estimation. Recommendation is made for all future drilling and channel sampling programs at the Project to include a more consistent approach to QC protocol for all samples to be sent for laboratory analysis. QC protocol should include the insertion of QC samples (blanks, CRMs and duplicates) in the field before batches are shipped for analysis.

Modelling of a lower grade envelop in the Inferred Mineral Resource Estimate area in the northern part of the Property is recommended to better understand geometry-continuity of the mineralized zone.

The best potential to develop additional Mineral Resource Estimates for the future lies in fill-in drilling and sampling to upgrade the Inferred Mineral Resource Estimates to Indicated Mineral Resource Estimates.

26.1.2 Mining Methods

The purpose of the planned underground definition drilling program is to provide additional information needed to finalize the level and stope designs prior to drifting in ore and stoping. The definition drilling work would be done using a just-in-time approach and as such Aura would need to efficiently carry out the associated core logging, assaying, geotechnical testing work and to timely update its geological and Mineral Resource Estimate model and revise level and stope design phase by phase. The definition drilling program would provide new information and data on the locations / elevations of the limits of geological zones including the altered mylonite zone; rock quality; folding / discontinuities that may be present between the existing surface diamond drill holes. The extent to which projected ore tonnages and grades, the mine schedule and estimated costs could be affected by the outcome of the definition drilling program and mine design finalization work is uncertain. Stope phases with greater surface diamond drilling density may be insignificantly affected by definition drilling results.

Based on the available rock mass information, enhanced ground support will likely be required when random features are encountered that can form wedges, or when zones of reduced rock mass quality are intersected. On this basis, it was considered reasonable to assume that an additional 5% of the stopes and access drives will require shotcrete.

The performance of the access drives is sensitive to the mining sequence, effective spans established and the ground support practices. Larger effective spans and increased stand-up time will increase the likelihood for instabilities, increased ground support and rehab requirements, and decreased production.

The stope span recommendations are sensitive to the ability of mine personnel to consistently tight fill the mined stopes as soon as possible after the completion of each stope. Poor quality or delayed backfill will cause the main access drives to deteriorate and will make mining of the secondary stopes more difficult.

The span and ground support recommendations are sensitive to the thickness and rock mass quality of the mylonite. If the mylonite is generally thicker or of poorer rock mass quality than currently expected, increased ground support will be required and a higher proportion of the stopes will need to be mined with a 3.5 m back span. The definition drilling proposed by Aura and P&E will be an important step in improving the understanding of this unit.

The decision to use a primary-secondary sequence instead of the pillarless retreat sequence was made after the completion of KP's work and the stability of the secondary stope pillars has not been evaluated. The secondary stope pillars are expected to be founded on saprolite in some areas. There is a risk that the pillars could fail into an adjacent open primary stope. The performance of the pillars will be dependent on the pillar geometry and rock mass quality, the detailed excavation sequencing, backfill practices, and the presence and thickness of any saprolite.

The groundwater inflow estimates are sensitive to the hydraulic conductivity of the rock mass and the potential for intersections with water-bearing structures. The actual groundwater inflows could vary from the estimates, potentially requiring additional dewatering.

It will be important to update the existing 3D lithological models, including the saprolite and mylonite models, to incorporate the results of any additional exploration drilling and/or an improved understanding of the deposit geology.

For detailed design, the domain definition, stability analyses, recommendations, and groundwater inflow estimate should be updated to account for the results of the additional site investigations and any changes to the geological models, large-scale structural interpretations and/or underground mine plan. Additional analyses are also recommended to advance the recommendations to support detailed design:

- The stability of the crown pillar has not been evaluated in detail. The crown pillar will be established late in the mine life. The proposed mining method, mine geometry and sequence limit the effect of potential instabilities associated with the crown pillar and provide an opportunity to gain experience with the deposit rock masses before the pillar is established. Additional engineering studies should be completed prior to establishing the crown pillar.
- Evaluating the required crown pillar dimensions and the stability of the secondary stope pillars, including the impact of the saprolite and further analysis of the mylonite and its influence on achievable stope dimensions and ground support following the completion of the definition drilling.
- Additional geomechanical logging should be completed to better define the spatial variation of the rock mass quality in the immediate HW of the proposed stopes, as well as the spatial variation in the distribution of the mylonite and saprolite. The definition drilling currently proposed by Aura could be used for this purpose.

An in-situ CRF strength of 0.5 MPa is recommended. Having consistent feed material that is within the required particle size distribution specification is an important consideration in ensuring that the CRF achieves the target strength and quality on a consistent basis. There is an opportunity to increase the maximum particle size to 5 inch (127 mm); however, strength test work should be completed to ensure that no loss in strength or segregation is observed.

Prior to the procurement of the CRF preparation system, it will be important to confirm the plant capacity can support the requirements of the underground mine while ensuring sufficient catch-up capacity to make up for periods of system downtime. The current placement strategy requires many steps before ultimate placement in the stopes. The actual cycle time should be confirmed so that the operating cost estimate can be as accurate as possible.

Although the CRF facility as summarized in this Techncial Report does not include a rock mechanics laboratory, a QAQC program should be put in place, using either contracted lab services or existing Aura facilities in the area, to monitor the particle size distribution of the prepared CRF aggregate, and test for the strength of the placed CRF to ensure that excessive consumption of cement does not occur.

It is recommended to investigate/negotiate a robust cement supply contract with a closer bulk transfer port. Backfill placement is directly linked to cement delivery. Discussions should be held with potential suppliers to determine cement delivery schedules. The cement system has been specified with one storage silo; an additional silo and screw conveyor can be procured as part of the vendor with no modifications made to the mixing system.

Additional confirmatory acid rock drainage test work for waste rock in all mine areas as well as for tailings is recommended. The Project financial model allows for US$50,000 for these studies in 2017.

There may be an opportunity to use fibrecrete instead of the combination of shotcrete and mesh. Depending on a number of factors, this could improve cycle time.

The span and ground support recommendations are sensitive to the thickness and rock mass quality of the mylonite. If the mylonite is generally thinner or of better rock mass quality than currently expected, less ground support will be required and a higher proportion of the stopes can be mined with the full 7 m back span.

The development of a detailed mining sequence may identify opportunities to improve expected opening performance and productivity. For example, a pillarless retreat sequence is expected to improve the performance of the access drives.

The encountered rock mass quality and observed opening performance should be documented during development of the proposed definition drill drives and the initial stages of underground mining. This represents an opportunity to refine the stope dimensioning and ground support prior to the start of production.

26.2 LAVRINHA

26.2.1 Mineral Resource Estimate

MCB recommends the following:

- Organization of the drill core in the temporary shed in Pontes e Lacerda.
- Assay drill core intervals not sampled.
- A complete review of the database information and cross-referencing with original records for the drill hole and assay databases.
- Update the surface topography files with more precision.
- Additional drilling is recommended at Lavrinha to drill off the deposit in the SW of the Property towards the adjacent valley and also at the southern end of the deposit where the density of drilling is reduced and there are some lenses that can be potentially delineated near surface.

The results of “G912-6” Geostats Standard are based on 18 assayed samples. The results indicated a slight bias in grade. It is recommended to check the certification of this standard due to the random values around the second standard deviation.

26.2.2 Mining Methods

MCB recommends that the Lavrinha waste rock storage area design be advanced to a detailed engineering level including elements such as foundation evaluations, design criteria, stability analysis, internal and surface drainage design.

26.3 PAU-A-PIQUE

26.3.1 Mineral Resource and Mineral Reserve Estimate

P&E offers the following recommendations:

- Drill hole down hole surveys should be reviewed for implausible readings and these should be removed and the resulting re-positioning of the hole toe examined for impact on the resource wireframing.
- Additional drilling is recommended for the west target zone to identify the mineral resource potential.
- A structural study is recommended to identify and model major gold-bearing shear zones in the deposit for future exploration drill targets.
- It is strongly recommended that definition drilling be carried out in the Indicated Resources contained in the NNW lower portion of main zone P2 and the foot wall lenses P3 and P4 in the SSE portion of the deposit, before their development.

The Pau-a-Pique Mineral Resource and Mineral Reserve Estimates meet requirements to be classified as an Indicated Mineral Resource Estimate and a Probable Mineral Reserve Estimate. Additional work in the future will focus on converting Mineral Resource Estimates from Indicated to Measured Mineral Estimates prior to mining by additional definition drilling. Plans for definition drilling are in place, and once mining recommences the diamond drilling will resume using three drill rigs owned by Aura with a contracted labour force.

Exploration will commence on the mining lease and surrounding exploration lease to increase the Mineral Resource base. Drilling will be conducted to the North and at depth at the Central orebody mainly to enable the continuation of main ramp development and update the mine plan if positive results are obtained. Another drilling objective will be to convert Inferred Mineral Resource Estimates into Indicated Mineral Resource Estimates.

P&E recommends that Mineral Resource and Mineral Reserve modelling work should be done twice a year. A Mineral Resource and Reserve Estimate update should be done at the beginning of the year, and a depletion update should be done at the end of year.

26.3.2 Mining Methods

P&E strongly recommends that definition drill data be available ahead of the stope extraction which subsequently must be used in the mine planning process before a particular stope is developed and mined. This will enable the mine operations to properly place the ore accesses within the stope designed boundaries and minimize stope dilution incurred during extraction, which the operation struggled with in the past.

Further geotechnical work will be required to assess the use of 100% swellex bolts versus rebar bolts for all waste development including the main ramp and reduction of wall bolt length from 2.4 m to 1.8 m due to the fact that the mine life for Pau-a-Pique’s openings will be significantly shorter than bolt life expectancy, and that ground conditions are estimated to improve at depth as observed in the geotechnical model.

It is recommended that the 220 m Elev sill pillar extraction should be investigated. Mining of this and future sill pillars should be well understood and planned as it presents upside potential to the mine cash flow.

A LOM has been prepared that produces in average 123Ktpa totalling 370Kt of ore mined from both development and stoping. The optimal method to extract the ore from underground is Avoca with choke blasting but will need further refinement and optimization. Load – haul – dump using Avoca with choke blasting is the mining method used in the current financial model and mine plan. Other options could be investigated with a decision on implementation and use expected after mining recommences, but not earlier than the second year of production.

The proposed stope dimensions are based on limited mining experience and will need to be refined during the initial mine operations. A key advantage of Avoca mining is that the panel strike length can be adjusted as mining progresses based on the observed panel performance. The following are recommended:

- Geotechnical mapping should be undertaken during the development of the undercut and overcut for each stope. The results of the mapping should be used to plan the initial panel strike lengths.
- The panel performance should be monitored using regular CMSs and possibly instrumentation. The collected data should be used to document the actual panel dimensions and dilution. The rock mass quality of the HW and FW and the time the panel remains open should also be documented.
- The panel strike length should be adjusted based on the observed stope performance during mining.
- A final panel reconciliation should be completed for each stope and the design of future panels should be updated using the data collected from each stope.
- The mine engineering department will need to include adequate ground control staff and resources to support mine development and operations.
- Numerical stress modelling is recommended to evaluate the extraction sequence and the offset between the development and the ore body. The results of the modelling can also be used to confirm some of the inputs to the Mathews Stability Graph, as well as the stope sizing and ground support recommendations.
- Additional kinematic and numerical analyses are recommended to refine and confirm the ground support recommendations. For example, numerical modelling could be used to refine the length of the cable bolts recommended in the HW and FW of the overcuts and undercuts.
- An evaluation of the stability of the raises is recommended prior to their development.

P&E recommends that significant attention must be dedicated to stope drilling and blasting practices mainly around the drill pattern, hole spacing, firing practice, energy distribution per hole and per blast, and interdepartmental accountability/responsibility for the entire process.

P&E recommends the first few stopes be treated as test stopes with a new drill pattern and spacing to be developed jointly by Aura technical services and operations management by analysing in greater detail the contributing factor to the stope over-break in the HW and FW area. It is possible that a larger hole spacing and pattern is required. This will subsequently put less explosive energy into the blast holes and possibly deliver the same fragmentation results due to the fact that the majority of the ore across the mine is classified as geotechnical type IV and type V with only the HW and FW varying and improving with depth.

The fan drilling technique tends to put higher energy at the toe of drilled holes and when blasted these holes impact the stability of stope walls differently than parallel drill holes. P&E recommends that stope drilling be undertaken using a parallel drilling technique as much as possible.

P&E recommends that the technical services department should develop, implement and closely monitor a stope extraction process. This should include, but not be limited to, stope design, drilling and blasting, stope closure, and stope reconciliation using a CMS survey technique to properly determine the over/under break of the HW and FW.

Optimization of the current mine plan, extraction sequence, and costs is recommended annually with a close look at tracking all mining costs and achieved performance in all headings from development to stope drilling, blasting and mucking of broken ore. This should be corroborated with specific cash cost reductions and ore extraction optimization KPI's. Such a program is highly recommended by P&E to be implemented immediately after mining recommences at Pau-a-Pique.

Once relevant information has been captured/obtained by the mine technical department as a result of these measures, P&E recommends that at least once a year the mine plan and budget model should be updated and presented to upper management for approval.

All mine operation documentation including, but not limited to, standard operating procedures ("SOP's"), position description, plans, process, forms, and flow charts must be updated to reflect current or intended changes to the mine operations for training and orientation purposes as well as compliance with Brazilian regulatory requirements.

Relative to mine planning, mine budgeting and cost control, mine reconciliation, ground control management plan, equipment maintenance plan, and operational KPI's, P&E recommends the establishment of RACI (responsibility, accountability, controls, and implementation) charts with clear deliverables.

26.4 PROCESSING PLANT

The grinding circuit has more than adequate capacity to handle the tonnages planned for the Project. In view of this it may be advisable to investigate whether it would be beneficial to grind finer.

The overall recoveries for the Pau-a-Pique and Lavrinha samples are very good. Those for the Ernesto samples are lower than expected even after the re-leach results are taken into account. Further work should be carried out on Ernesto material to ascertain the reasons for this. The work should investigate using finer grinds, increased cyanide levels and a trade-off study should be performed to confirm the industrial benefits of using Leach Aid in the CIL process.

Since the plant has more than enough capacity to grind finer, a series of tests should be performed to establish the optimum grind size for Ernesto ore, and then to establish the optimum leach conditions. Since Ernesto does not come on line for some years, this work can be carried out while other ores are being processed.

Although Ernesto recoveries are shown to be acceptable, there is a need to better understand the ore geometallurgically and better define other metallurgical implications in the processing plant. The following process plant recommendations are also provided:

- Continue with optimization efforts around reagent dosage, focusing on the two operating regimes outlined in the study.
- Review operating manuals to better control densities in the process, especially important for soft ores with high amounts of fines. This improvement needs to be focused at the E-Cat stage and CIL.
- Review the existing SAG mill control logic as the ore to be fed from all deposits is softer than originally expected. This logic would target the use of SAG mill speed and SAG pressure to prevent liner damage in situations where load cannot be built within the SAG mill.

26.5 INFRASTRUCTURE

Finalize the Tierra Group study, which includes a trade-off assessment of using waste rock instead of saprolite to build the next tailings storage facility raise. This study includes a better characterization of the acid generation potential testwork on the waste rock.

26.6 ENVIRONMENTAL

There have been no ARD characterization tests done on tailings or Lavrinha waste rock, and it is recommended that confirmatory acid rock drainage testwork for waste rock in all mine areas be carried out, and similarly for the tailings.

An updated dewatering estimate for Pau-a-Pique is recommended in order to ensure adequate surface water management capability.

It is recommended that supporting studies and comprehensive closure plan development be initiated within the first year of operation.

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28.0 CERTIFICATES

CERTIFICATE OF QUALIFIED PERSON

EUGENE J. PURITCH, P.ENG.

I, Eugene J. Puritch, P. Eng., residing at 44 Turtlecreek Blvd., Brampton, Ontario, L6W 3X7, do hereby certify that:

1. I am an independent mining consultant and President of P&E Mining Consultants Inc.
2. This certificate applies to the technical report titled “Feasibility Study and Technical Report on the EPP Project, Mato Grosso, Brazil” (the “Technical Report”), with an effective date of July 31, 2016.
3. I am a graduate of The Haileybury School of Mines, with a Technologist Diploma in Mining, as well as obtaining an additional year of undergraduate education in Mine Engineering at Queen’s University. In addition I have also met the Professional Engineers of Ontario Academic Requirement Committee’s Examination requirement for Bachelor’s Degree in Engineering Equivalency. I am a mining consultant currently licensed by Professional Engineers and Geoscientists New Brunswick (License No. 4778), Professional Engineers, Geoscientists Newfoundland & Labrador (License No. 5998), Association of Professional Engineers and Geoscientists Saskatchewan (License No. 16216), Ontario Association of Certified Engineering Technicians and Technologists (License No. 45252) the Professional Engineers of Ontario (License No. 100014010) and Association of Professional Engineers and Geoscientists of British Columbia (License No. 42912). I am also a member of the National Canadian Institute of Mining and Metallurgy.

   I have read the definition of “qualified person” set out in National Instrument 43-101 (“NI 43-101”) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.

   I have practiced my profession continuously since 1978. My summarized career experience is as follows:
   - Mining Technologist - H.B.M.& S. and Inco Ltd., ... 1978-1980
   - Open Pit Mine Engineer – Cassiar Asbestos/Brinco Ltd., ... 1981-1983
   - Pit Engineer/Drill & Blast Supervisor – Detour Lake Mine, ... 1984-1986
   - Self-Employed Mining Consultant – Timmins Area, ... 1987-1988
   - Mine Designer/Resource Estimator – Dynatec/CMD/Bharti, ... 1989-1995
   - Self-Employed Mining Consultant/Resource-Reserve Estimator, ... 1995-2004
   - President – P&E Mining Consultants Inc, ... 2004-Present
4. I have not visited the Property that is the subject of this report.
5. I am responsible for co-authoring Sections 14, 25 and 26 of the Technical Report along with co-authoring those sections of the Summary pertaining thereto.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have read NI 43-101 and Form 43-101F1. This Technical Report has been prepared in compliance therewith.
8. I have had no prior involvement with the Property that is the subject of this Technical Report.
9. 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.

Effective date: July 31, 2016
Signing date: January 13, 2017

{SIGNED AND SEALED}
[Eugene J. Puritch]

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Eugene J. Puritch, P.Eng.

CERTIFICATE of QUALIFIED PERSON

ANDREW BRADFIELD, P. ENG.

I, Andrew Bradfield, P. Eng., residing at 5 Patrick Drive, Erin, Ontario, N0B1T0, do hereby certify that:

1. I am an independent mining engineer contracted by P&E Mining Consultants.
2. This certificate applies to the technical report titled "Feasibility Study and Technical Report on the EPP Project, Mato Grosso, Brazil" (the "Technical Report"), with an effective date of July 31, 2016.
3. I am a graduate of Queen's University, with an honours B.Sc. degree in Mining Engineering in 1982. I have practiced my profession continuously since 1982. I am a Professional Engineer of Ontario (License No.4894507). I am also a member of the National CIM.

   I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is:

   - Various Engineering Positions – Palabora Mining Company 1982-1986
   - Mines Project Engineer – Falconbridge Limited.. 1986-1987
   - Senior Mining Engineer – William Hill Mining Consultants Limited 1987-1990
   - Self-Employed Mining Engineer.. 1990-1991
   - GM Toronto – Bharti Engineering Associates Inc. 1991-1996
   - VP Technical Services, GM of Australian Operations – William Resources Inc. 1996-1999
   - Self-Employed Mining Engineer.. 1999-2001
   - Principal Mining Engineer – SRK Consulting 2001-2003
   - COO – China Diamond Corp. 2003-2006
   - VP Operations – TVI Pacific Inc.. 2006-2008
   - COO – Avion Gold Corporation 2008-2012
   - Self-Employed Mining Engineer.. 2012-2013
   - Self-Employed Mining Engineer, COO - P&E Mining Consultants................2014-Present

4. I visited the Property that is the subject of this report, from June 18 to 23, 2015.
5. I am responsible for authoring Sections 2, 3, 19, 22 and 24 and coauthoring Sections 12, 16, 25 and 26 of the Technical Report along with those sections of the Summary pertaining thereto.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had no prior involvement with the Property that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. 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.

Effective date: July 31, 2016
Signing date: January 13, 2017

{SIGNED AND SEALED}
[Andrew Bradfield]

Andrew Bradfield, P. Eng.

CERTIFICATE of QUALIFIED PERSON

ALEXANDRU VERESEZAN, P. ENG.

I, Alexandru Veresezan of 25 Stookes Crescent, Richmond Hill, Ontario, L4E 0J4, and Senior Associate Mining Engineering with P&E Mining Consultants Inc. of Brampton, Ontario., do hereby certify that:

1. This certificate applies to the technical report titled "Feasibility Study and Technical Report on the EPP Project, Mato Grosso, Brazil" (the "Technical Report"), with an effective date of July 31, 2016.
2. I am a graduate of Mining Engineering with a Master of Engineering, Honors Underground Mining degree from the University of Petrosani in 1993. I have practiced my profession as a Mining Engineer since graduation. I am a Registered Professional Engineer in good standing with the Professional Engineers Ontario (License No. 100078587).

   I have read the definition of "qualified person" set out in 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 purpose of NI 43- 101.

   My summarized career experience is as follows:

   - P&E Mining Consultants Inc.: (Sr. Associate Mining Engineer)..................................2016-Present
   - Barrick Gold Corp, Copper Division: (Mine Manager/ Alternate GM).............................2012-2015
   - Barrick Gold Corp. (Manager, Underground Mining, Corporate Office) 2008-2012
   - Wardrop Engineering Inc.: (Sr. Mining Engineer) 2006-2008
   - Cementation SKANSKA Canada Inc.: (Mining Eng/Estimator/ Pro Cost Control) 2002-2006
   - Dynatec Corp: (Estimator / EIT) 2001-2002
   - S.C. Grandemar S.A.: (Open Pit Manager) 1999-2000
   - Cluj, Romania Various: (Junior-Senior Project Engineer) 1993-1999

3. I am responsible for co-authoring Sections 15, 16, 21, 25 and 26, of the Technical Report along with those parts of the Executive Summary pertaining thereto.
4. I am not aware of any material fact or material change with respect to the subject matter of this technical report which is not reflected in the technical report.
5. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
6. I visited the Property that is the subject of this report from May 17 to 25, 2016 and from October 24 to 29, 2016.
7. I have had no prior involvement with the Property that is the subject of this Technical Report.
8. I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43- 101F1.
9. As at the effective date of the Technical Report, 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.

Effective date: July 31, 2016
Signing date: January 13, 2017

{SIGNED AND SEALED}
[Alex Veresezan]

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Alex Veresezan, P. Eng.

CERTIFICATE of QUALIFIED PERSON

DAVID A. ORAVA, P. ENG.

I, David A. Orava, M. Eng., P. Eng., residing at 19 Boulding Drive, Aurora, Ontario, L4G 2V9, do hereby certify that:

1. I am an Associate Mining Engineer at P&E Mining Consultants Inc. and President of Orava Mine Projects Ltd.
2. This certificate applies to the technical report titled "Feasibility Study and Technical Report on the EPP Project, Mato Grosso, Brazil" (the "Technical Report"), with an effective date of July 31, 2016.
3. I am a graduate of McGill University located in Montreal, Quebec, Canada at which I earned my Bachelor Degree in Mining Engineering (B.Eng. 1979) and Masters in Engineering (Mining - Mineral Economics Option B) in 1981. I have practiced my profession continuously since graduation. I am licensed by the Professional Engineers of Ontario (License No. 34834119).

   I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

   My summarized career experience is as follows:

· Mining Engineer – Iron Ore Company of Canada.	1979-1980
· Mining Engineer – J.S Redpath Limited / J.S. Redpath Engineering.	1981-1986
· Mining Engineer & Manager Contract Development – Dynatec Mining Ltd.	1986-1990
· Vice President – Eagle Mine Contractors	1990
· Senior Mining Engineer – UMA Engineering Ltd.	1991
· General Manager - Dennis Netherton Engineering	1992-1993
· Senior Mining Engineer – SENES Consultants Ltd.	1993-2003
· President – Orava Mine Projects Ltd.	2003 to present
· Associate Mining Engineer – P&E Mining Consultants Inc.	2006 to present

4. I visited the Property that is the subject of this report on February 16-21, 2016, and May 13-20, 2016.
5. I am responsible for coauthoring Sections 15, 16, 25 and 26 of the Technical Report along with those sections of the Summary pertaining thereto.
6. I am an independent of the Issuer applying all of the tests in Section 1.5 of NI 43-101.
7. I have had no prior involvement with the Property that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and the Report has been prepared in compliance therewith.
9. 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.

Effective Date: July 31, 2016
Signing Date: January 13, 2017

{SIGNED AND SEALED}
[David Orava]
____
David Orava, M. Eng., P. Eng.

CERTIFICATE OF QUALIFIED PERSON

RICHARD SUTCLIFFE, Ph.D., P. GEO.

I, Richard Sutcliffe, Ph.D., P. Geo., residing at 100 Broadleaf Crescent, Ancaster, Ontario, do hereby certify that:

1. I am an independent geological consultant and Senior Geological Advisor at P&E Mining Consultants Inc.
2. This certificate applies to the technical report titled "Feasibility Study and Technical Report on the EPP Project, Mato Grosso, Brazil", (the "Technical Report") with an effective date of July 31, 2016.
3. I am a graduate of the University of Toronto with a Bachelor of Science degree in Geology (1977). In addition, I have a Master of Science in Geology (1980) from University of Toronto and a Ph.D. in Geology (1986) from the University of Western Ontario. I have worked as a geologist for a total of 32 years since obtaining my M.Sc.degree. I am a geological consultant currently licensed by the Association of Professional Geoscientists of Ontario (License No 852).

   I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is:

· Precambrian Geologist, Ontario Geological Survey	1980-1989
· Senior Research Geologist, Ontario Geological Survey	1989-1991
· Associate Professor of Geology, University of Western Ontario.	1990-1992
· President and CEO, URSA Major Minerals Inc.	1992-2012
· President and CEO, Patricia Mining Corp	1998-2008
· President and CEO, Auriga Gold Corp	2010-2012
· President, Pavey Ark Minerals Inc	2012-Present
· Consulting Geologist	1992-Present

4. I have not visited the Property that is the subject of this report.
5. I am responsible for authoring Sections 6, 7, 8 and 23 and coauthoring Sections 25 and 26 of the Technical Report along with those sections of the Summary pertaining thereto.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had no prior involvement with the Property that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. 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.

Effective date: July 31, 2016
Signing date: January 13, 2017

{SIGNED AND SEALED}

[Richard H. Sutcliffe]

Richard H. Sutcliffe, PhD, P Geo.

CERTIFICATE OF QUALIFIED PERSON

RICHARD E. ROUTLEDGE, P.GEO.

I, Richard E. Routledge, P.Geo., residing at 1386 Queen's Line, PO Box 335, Minden, Ontario, K0M 2K0, do hereby certify that:

1. I am an independent Consulting Geologist who has been contracted by P&E Mining Consultants Inc.
2. This certificate applies to the technical report titled "This certificate applies to the technical report titled "Feasibility Study and Technical Report on the EPP Project, Mato Grosso, Brazil" (the "Technical Report"), with an effective date of July 31, 2016.
3. I graduated with a Bachelor of Science degree, Major in Geology, from Sir George Williams (Concordia) University in 1971 and with a Masters degree in Applied Exploration Geology from McGill University in 1973. I have worked as a geologist for about 43 years since post-graduation. I am a Professional Geologist registered in the Province of Ontario (APGO No. 1354) and licensed by the Northwest Territories (NAPEGG No. L744).

   I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. My relevant experience for the purpose of the Technical Report is:

   - Independent Consulting Geologist. ....2011 – Present
   - Roscoe Postle Associates Inc., Consulting Geologist .... 1998 – 2011
   - Independent Consulting Geologist .... 1994 – 1997
   - Vice President Exploration, Greater Lenora Resources Corp. .... 1993 – 1994
   - Teck Explorations Ltd, Evaluations and Mineral Commodities Geologist. .... 1985 – 1992
   - Derry, Michener, Booth & Wahl, Exploration and Consulting Geologist. .... 1973 – 1985

4. I have visited the property that is the subject of this Technical Report on June 18-21, 2015.
5. I am responsible for co-authoring Sections 11, 12, 14, 25 and 26 of the Technical Report along with those sections of the Summary pertaining thereto.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101. I am independent of the Vendor and the Property.
7. I have had no prior involvement with the Property that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.
9. 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.

Effective date: July 31, 2016
Signing date: January 13, 2017

{SIGNED AND SEALED}
[Richard E. Routledge]

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Richard E. Routledge, P. Geo.

CERTIFICATE OF QUALIFIED PERSON

DAVID BURGA, P.GEO.

I, David Burga, P. Geo., residing at 3884 Freeman Terrace, Mississauga, Ontario, do hereby certify that:

1. I am an independent geological consultant contracted by P & E Mining Consultants Inc.
2. This certificate applies to the technical report titled "This certificate applies to the technical report titled "Feasibility Study and Technical Report on the EPP Project, Mato Grosso, Brazil" (the "Technical Report"), with an effective date of July 31, 2016.
3. I am a graduate of the University of Toronto with a Bachelor of Science degree in Geological Sciences (1997). I have worked as a geologist for a total of 12 years since obtaining my B.Sc. degree. I am a geological consultant currently licensed by the Association of Professional Geoscientists of Ontario (License No 1836).

   I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is:

   - Exploration Geologist, Cameco Gold..1997-1998
   - Field Geophysicist, Quantec Geoscience 1998-1999
   - Geological Consultant, Andeburg Consulting Ltd. 1999-2003
   - Geologist, Aeon Egmond Ltd...2003-2005
   - Project Manager, Jacques Whitford 2005-2008
   - Exploration Manager – Chile, Red Metal Resources 2008-2009
   - Consulting Geologist..2009-Present

4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring Section 4 and co-authoring Sections 9, 10, 25 and 26 of the Technical Report along with those sections of the Summary pertaining thereto.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had no prior involvement with the Property that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. 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.

Effective date: July 31, 2016
Signing date: January 13, 2017

{SIGNED AND SEALED}
[David Burga]

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David Burga, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

JARITA BARRY, P.GEO.

I, Jarita Barry, P.Geo., residing at 2485B Hwy 3A, Nelson, British Columbia, V1L 6K7, do hereby certify that:

1. I am an independent geological consultant contracted by P & E Mining Consultants Inc.
2. This certificate applies to the technical report titled "This certificate applies to the technical report titled "Feasibility Study and Technical Report on the EPP Project, Mato Grosso, Brazil" (the "Technical Report"), with an effective date of July 31, 2016.
3. I am a graduate of RMIT University of Melbourne, Victoria, Australia, with a B.Sc. in Applied Geology. I have worked as a geologist for a total of 9 years since obtaining my B.Sc. degree. I am a geological consultant currently licensed by the Association of Professional Engineers and Geoscientists of British Columbia (License No. 40875). I am also a member of the Australasian Institute of Mining and Metallurgy of Australia (Member No. 305397);

   I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is:

   - Geologist, Foran Mining Corp. 2004
   - Geologist, Aurelian Resources Inc. 2004
   - Geologist, Linear Gold Corp. 2005-2006
   - Geologist, Búscore Consulting 2006-2007
   - Consulting Geologist (AusIMM) 2008-2014
   - Consulting Geologist, P.Geo. (APEGBC/AusIMM) ....2014-Present.

4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for co-authoring Sections 11, 12, 25 and 26 of this Technical Report along with those sections of the Summary pertaining thereto.
6. I am independent of the Issuer applying all of the tests in section 1.5 of National Instrument 43-101.
7. I have had no prior involvement with the Property that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.
9. 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.

Effective date: July 31, 2016
Signing date: January 13, 2017

{SIGNED AND SEALED}
[Jarita Barry]
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Jarita Barry, P.Geo.

CERTIFICATE of QUALIFIED PERSON

BRADLEY HOWE, P. ENG.

I, Bradley Howe, P. Eng., do hereby certify that:

1. I am a mining engineer employed by Paterson & Cooke Canada Inc., with business address 1351-C Kelly Lake Road, Unit 2, Sudbury, Ontario, P3E 5P5.
2. This certificate applies to the technical report titled "Feasibility Study and Technical Report on the EPP Project, Mato Grosso, Brazil", (the "Technical Report") with an effective date of July 31, 2016.
3. I am a graduate of The University of British Columbia, with an B.A.Sc. degree in Mining Engineering in 2008. I have practiced my profession continuously since 2008. I am a Professional Engineer of Ontario (License No.100230061). I am also a member of the National CIM.

   I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is over 8 years of experience in underground mining, with a focus on backfill, for a variety of commodities with project experience nationally and internationally.
4. I did not visit the Property that is the subject of this report. I visited the laboratory in Belo Horizonte, Brazil, that performed the backfill test work. Robert Brown, P.Eng., Principal of Paterson & Cooke Canada Inc., visited the Property on June 17 to 19, 2015 in his role as Project Director and directly oversaw my work.
5. I am responsible for coauthoring Sections 16, 25 and 26 of the Technical Report along with those sections of the Summary pertaining thereto.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had no prior involvement with the Property that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. 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.

Effective date: July 31, 2016
Signing date: January 13, 2017

{SIGNED AND SEALED}
[Bradley Howe]

Bradley Howe, P. Eng.

CERTIFICATE of QUALIFIED PERSON

GRAHAM. P. HOLMES, ARSM, P.ENG

I, Graham P Holmes, P. Eng., residing at 12 Wenonah Dr. Mississauga, L5G 3W1, do hereby certify that:

1. I am a senior process engineer employed by Jacobs.
2. This certificate applies to the technical report titled "Feasibility Study and Technical Report on the EPP Project, Mato Grosso, Brazil", (the "Technical Report") with an effective date of July 31, 2016.
3. I graduated from The Royal School of Mines, Imperial College, London University with an honours B.Sc. degree in Mining Engineering with Mineral Technology Option in 1966. I have practiced my profession continuously since 1966. I am a Professional Engineer of Ontario (License No.20196507). I am also a member of the National CIM.

   I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is:

   - Various Engineering Positions –Cerro de Pasco Corp. Peru. 1966-1969
   - Plant Metallurgist and Senior Metallurgist–Brenda Mines Ltd........................... 1969-1972
   - Concentrator Superintendent –Bell Copper Division, Noranda Mines Ltd......... 1972-1979
   - Plant Manager- Andaluza de Piritas SA. Spain... 1979-1980
   - Manager Mineral Processing and Development–Sherritt Gordon Mines Ltd. 1980-1986
   - Process Engineer–Senior Process Specialist--Jacobs. ....1986-to present

4. I visited the Property that is the subject of this report, from November 23 to 26, 2016.
5. I am responsible for authoring Section 13, and coauthoring Sections 25 and 26 of the Technical Report along with those sections of the Summary pertaining thereto.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had no prior involvement with the Property that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. 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.

Effective date: July 31, 2016
Signing date: January 13, 2017

{SIGNED AND SEALED}
[Graham P Holmes]

Graham P Holmes, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

FERNANDO A. CORNEJO, P.ENG.

1. I, Fernando A. Cornejo P.Eng., do hereby certify that I have been employed by the Issuer since April 2014 and am currently Vice-President, Projects with Aura Minerals Inc.; located at 26th Floor – 161 Bay Street, Toronto, ON, M5J 2S1, Canada.
2. This certificate applies to the technical report titled “Feasibility Study and Technical Report on the EPP Project, Mato Grosso, Brazil”, (the “Technical Report”) with an effective date of July 31, 2016.
3. I graduated from Universidad Nacional de San Agustin, Arequipa, Peru with an honours B.Sc. in Chemical Engineering in 2001 and from Ecole Polytechnique de Montreal, Canada with an M.Eng. Degree in Chemical Engineering in 2005.
4. I am a Professional Engineer registered with the Professional Engineers of Ontario with License # 100170042
5. I have practiced my profession since 2001 in a range of operational, technical consulting, project management and Executive roles in Mexico, Canada and Peru. My relevant Experience for the purpose of the Technical Report is:

   - Process Engineer - BHP Billiton Base Metals (Tintaya Mine)……………..…2001-2002
   - Senior Process Eng./Technical Services – Rio Tinto Iron Ore (IOC)….….…. 2005-2007
   - Global Project Integration Manager – SGS Minerals…………………….….…2007-2011
   - Multiple Technical/Executive Engineering Roles – Jacobs Engineering…….. 2011 - 2013
   - Vice President, Project Development -Aura Minerals……………………….2013-Present

6. I have read the definition of “qualified person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a “qualified person” for the purposes of NI 43-101.
7. I am responsible for Section 17 and co-authoring Sections 18, 21, 25 and 26 of the Technical Report along with those sections of the Summary pertaining thereto.
8. I am non-independent of the Issuer applying the test in Section 1.5 of NI 43-101.
9. I have had prior involvement with the Property that is the subject of this Technical Report as I am a VP of Aura Minerals Inc.
10. I visited the EPP Project multiple times in 2015 and 2016, with a most recent visit dated October 2016.
11. I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.
12. As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report that I am responsible for, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective date: July 31, 2016
Signing date: January 13, 2017

{SIGNED AND SEALED}
[Fernando A. Cornejo]

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Fernando A. Cornejo, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

MARCELO ANTONIO BATELOCHI, AusIMM (CP)

I, MARCELO ANTONIO BATELOCHI, P. Geo., residing at Av. Raja Gabaglia, 4.943 | Sala 101, Belo Horizonte, Minas Gerais, Brazil, do hereby certify that:

1. I am an independent geological and Mineral Resources consultant contracted by MCB Consultants.
2. This certificate applies to the technical report titled "Feasibility Study and Technical Report on the EPP Project, Mato Grosso, Brazil" (the "Technical Report"), with an effective date of July 31, 2016.
3. I holds a degree in 1991, Bachelor of Honors from School of Geology at UNESP - São Paulo State University, Brazila. I have worked as a geologist and Mineral Resources and Reserves for a total of 25 years since obtaining my B.Sc. degree. I am a Member of the Australasian Institute of Mining and Metallurgy and am qualified as a Chattered Profession of Geology and Mineral Resources (Qualified to assign JORC and NI 43-101 Mineral Resource Reports).

   I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is:

   - Rio Tinto Brazil (MCR|Iron Ore – 2 years and Rio Paracatu Mineração| Gold – 10 years) 1992-2003
   - Vale (Master Geologist – Mineral Resources and Ore Reserve Specialist)................................2003-2007
   - Ferrous Resources do Brazil (General Manager of Geology and Mine Planing – Iron Ore)......2007-2013
   - Independent Consultant (Clients: Beadell Brazil, Votorantim Metais, Yamana, Ferrous, PA Gold, B&A, MCB, SAM – Sulamericana de Metais, Yamana Gold Inc.).................................2012-2016

4. I visited three times Lavinha Pit area and Ernesto facilities, during period of Jun-Sept/2015, that is the subject of this report.
5. I am responsible for co-authoring Sections 9, 10, 11, 12, 14, 15, 16, 21, 25, 26 of the Technical Report along with those sections of the Summary pertaining thereto.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had no prior involvement with the Property that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. 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.

Effective date: July 31, 2016
Signing date: January 13, 2017

{SIGNED AND SEALED}
[Marcelo Antonio Batelochi]

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Marcelo Antonio Batelochi, AusIMM (CP)

CERTIFICATE of QUALIFIED PERSON

DIANE LISTER, P.ENG.

I, Diane Lister, P.Eng., residing at 18 Michie Place, Marsh Lake, Yukon, Y0B1Y2, do hereby certify that:

1. I am an independent environmental engineer and principal of Altura Environmental Consulting, and am contracted by Aura Minerals Inc.
2. This certificate applies to the technical report titled "Feasibility Study and Technical Report on the EPP Project, Mato Grosso, Brazil", (the "Technical Report") with an effective date of July 31, 2016.
3. I am a graduate of the University of British Columbia with a Bachelor of Applied Science degree in Geological Engineering (1989), and a Master of Applied Science degree in Mining Engineering (1994). I have practiced my profession as an environmental engineer continuously for 22 years in a range of operational, technical and consulting roles since obtaining my M.A.Sc. degree. I am a Professional Engineer and member in good standing with the Association of Professional Engineers and Geoscientists of British Columbia (License #25689) and the Association of Professional Engineers of Yukon (License #1552).

   I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is:

   - Geologist / Environmental Engineer, Geddes Resources Limited 1990-1992
   - Environmental Engineer, Quintette Coal Mine, Canada, Teck Corporation 1994-1996
   - Environmental Coordinator, Brewery Creek Mine, Canada, Viceroy Minerals.. 1996-1999
   - Consulting Environmental Engineer – Cia. Minera Antamina, Perú.................. 1999-2002
   - Consulting Environmental Engineer, Altura Environmental Consulting 2002-2005
   - Environmental Manager, Gualcamayo Project, Viceroy Exploration 2006-2007
   - Consulting Environmental Engineer, Altura Environmental Consulting 2007-Present

4. I visited the Property that is the subject of this report, May 16 to 17, 2016.
5. I am responsible for authoring Sections 5 and 20 and co-authoring Sections 25 and 26 of the Technical Report along with those sections of the Summary pertaining thereto.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had no prior involvement with the Property that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. 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.

Effective date: July 31, 2016
Signing date: January 13, 2017

{SIGNED AND SEALED}
[Diane Lister]

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Diane Lister, P.Eng.

CERTIFICATE of QUALIFIED PERSON

ROBERT A. MERCER, Ph.D., P. Eng.

I, Robert A. Mercer, P. Eng., residing at 162 Silver Lady Lane, North Bay, Ontario, P1B 8G4, do hereby certify that:

1. I am employed by, and carrie.d out this assignment for: Knight Piesold Ltd, 1650 Main Street West North Bay, Ontario P1B 8G4 tel. (705) 476-2165; fax (705) 474-8095; e-mail: rmercer@knightpiesold.com
2. This certificate applies to the technical report titled "Feasibility Study and Technical Report on the EPP Project, Mato Grosso, Brazil", (the "Technical Report") with an effective date of July 31, 2016.
3. I am a graduate of Queen's University with degrees Geological Engineering, Mining Engineering and Mining Rock Mechanics (B.Sc. (1988), M.Sc. (1992) and Ph.D. (1999), respectively). I am a Professional Engineer with over 25 years of rock mechanics experience. My recent work ranges from managing geomechanical site investigation programs to providing ongoing rock mechanics support to operating underground and open pit mines. I have worked on over 100 mining and civil projects world-wide and I am a licensed Professional Engineer in Ontario (License No. 90521915), Nunavut (License No. LI 774), and Newfoundland and Labrador (License No. N7932). I am also a designated Consulting Engineer in Ontario (License No. 4065).
4. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. My relevant experience for the purpose of the Technical Report is:

· Knight Piesold Consulting (North Bay, ON) Canada, Managing; Principal	2012 - Present
· Knight Piesold Consulting (North Bay, ON) Canada, Specialist -; Engineer - Rock Mechanics	2002 - 2012
· Knight Piesold Consulting (North Bay, ON) Canada, Senior Rock; Mechanics Engineer	1999 - 2002
· Engineering Seismology Group Ltd., Rock Mechanics Research; Engineer	1994 - 1999
· Dept. of Mining Eng., Queen's University, Research and Teaching; Assistant.	1991 - 1999
· Anglo American Corp., Rock Mechanics Officer and Strata Control; Officer, South Africa	1988 – 1999

5. I visited the Property that is the subject of this report, from June 18 to 22, 2015.
6. I am responsible for co-authoring the aspects of Section 16 related to the Ernesto Deposit, 25 and 26 along with those sections of the Summary pertaining thereto.
7. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
8. I have had no prior involvement with the Property that is the subject of this Technical Report.
9. I have read NI 43-101 and Form 43-101Fl and this Technical Report has been prepared in compliance therewith.
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.

Effective date: July 31, 2016
Signing date: January 13, 2017

{SIGNED AND SEALED}
[Robert A. Mercer]

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Robert A. Mercer, Ph.D., P. Eng. Managing Principal, North Bay.

CERTIFICATE of QUALIFIED PERSON

MATTHEW L. FULLER, CPG., LEG.

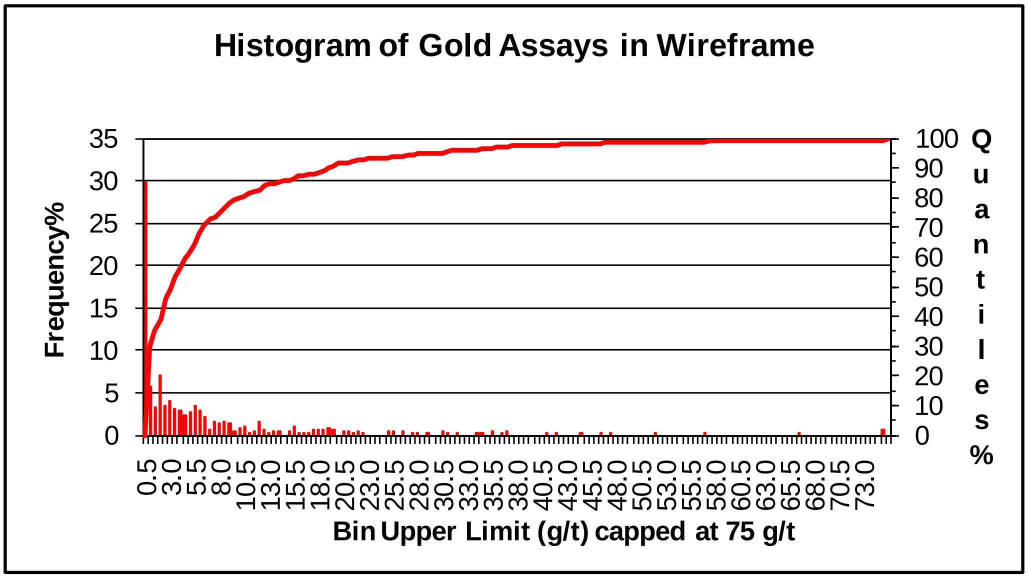
1. I, Matthew L. Fuller, CPG, LEG do hereby certify that I am the Co-Founder of Tierra Group International, Ltd. since 1 January 2012, and am currently a Principal Engineering Geologist with Tierra Group International, Ltd.; located at 1746 Cole Blvd. Suite 130, Lakewood, Colorado, 80401 USA.
2. This certificate applies to the technical report titled "Feasibility Study and Technical Report on the EPP Project, Mato Grosso, Brazil", (the "Technical Report") with an effective date of July 31, 2016.
3. I graduated from Colorado State University with a B.S. degree in Geology in 1982. I am a Certified Professional Geologist through the American Institute of Professional Geologist (# 8757), and a Licensed Engineering Geologist in Washington State (License # 2135). I have practiced my profession since 1980 in a range of technical consulting, project management and Executive roles in U.S.A. Canada, Mexico, Central and South America, Africa and Australasia. My relevant Experience for the purpose of the Technical Report is:
    - Founder & Principal Engineering Geologist – Tierra Group International, Ltd. 2012-present
    - Vice President & Principal Engineering Geologist – Tetra Tech Inc. 2007-2013
    - Founder & Principal Engineering Geologist – Vector Colorado, LLC 2003-2007
    - Mining Group Manager – Olsson Associates 2001 - 2003
    - Sr. Engineering Geologist - SRK Consulting Inc. 1999 – 2001
    - Engineering Geologist – Hydro-Triad, Ltd. 1990 - 1999
4. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I am responsible for co-authoring Sections 18, 25 and 26 of the Technical Report along with those sections of the Summary pertaining thereto.
6. I am non-independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had no prior involvement with the Property that is the subject of this Technical Report.
8. I visited the EPP Project on 25 November 2016.
9. I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.
10. As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report that I am responsible for, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

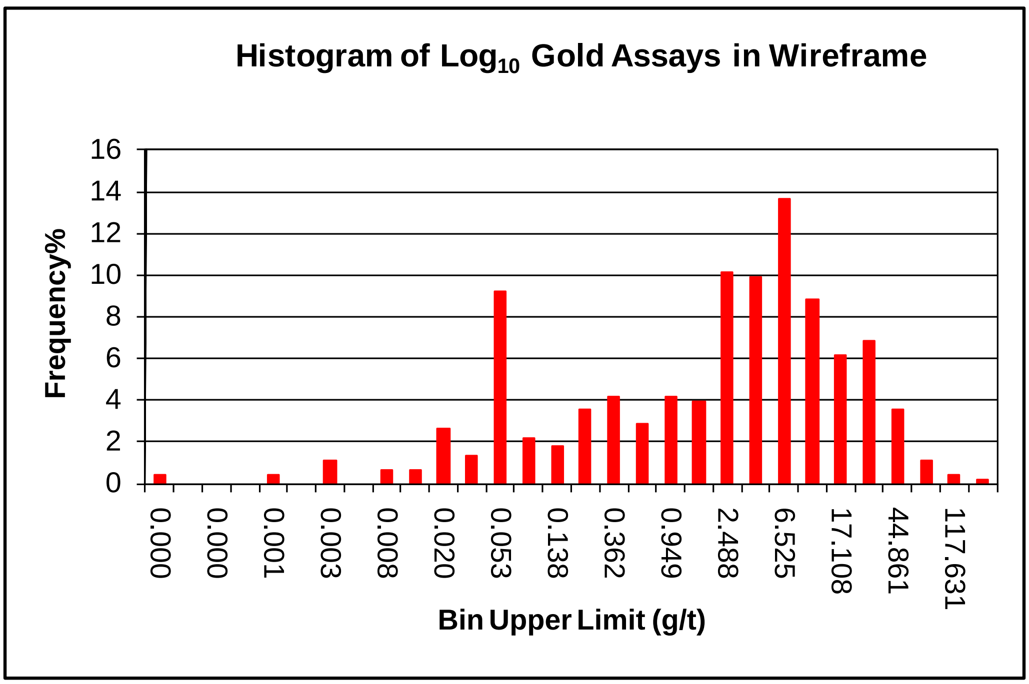
Effective date: July 31, 2016
Signing date: January 13, 2017

{SIGNED AND SEALED}
[Matthew L. Fuller]

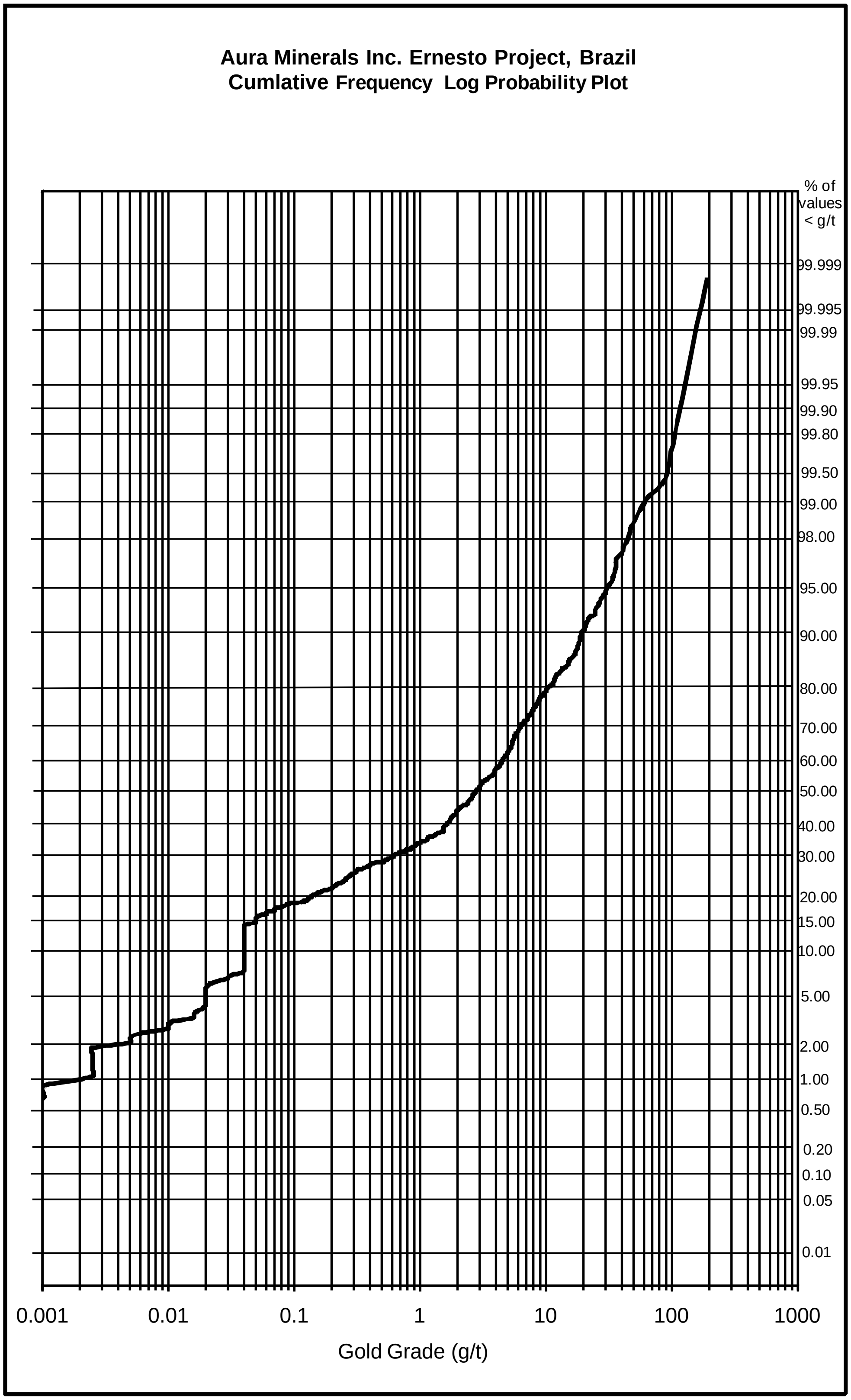
Matthew L. Fuller, CPG., LEG.

APPENDIX I. ERNESTO

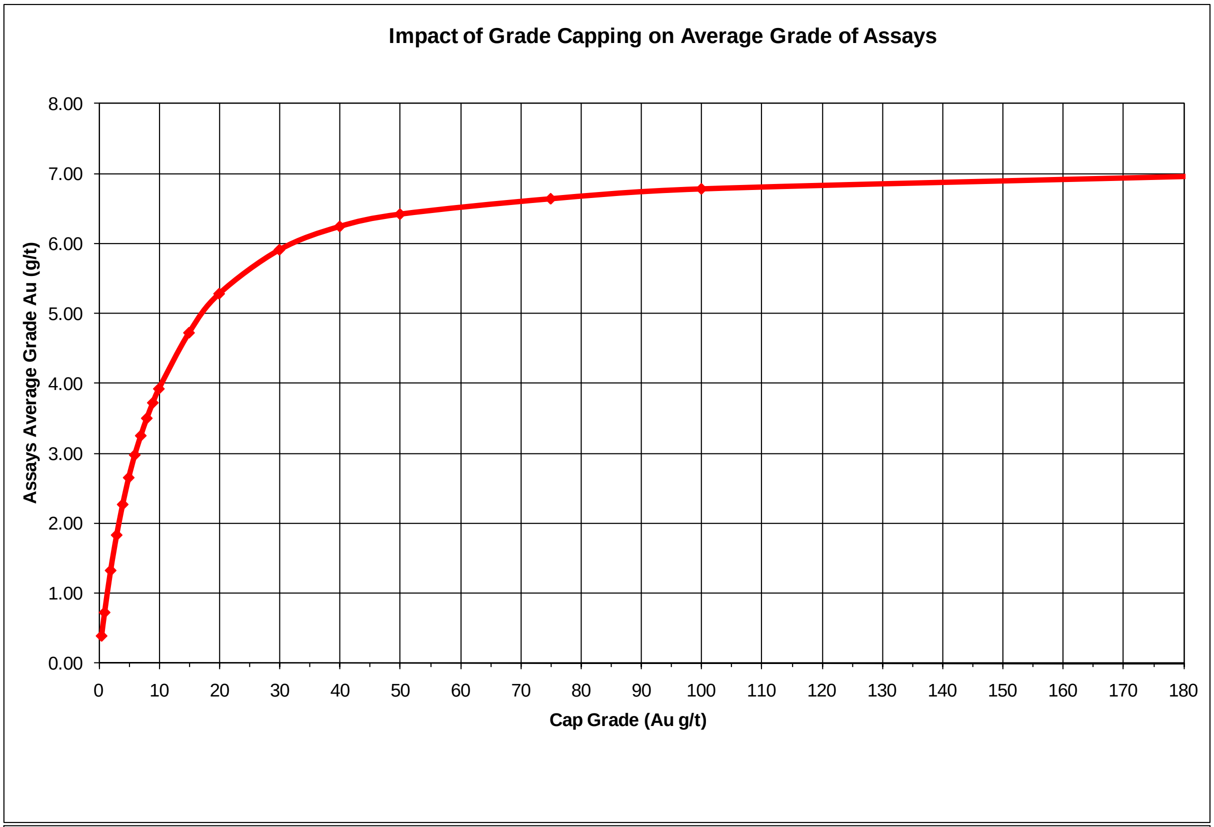

Histogram of Gold Assays in Wireframe
Frequency%
Quantiles%
Bin Upper Limit (g/t) capped at 75 g/t
0
5
10
15
20
25
30
35
0
10
20
30
40
50
60
70
80
90
100
0.5
3.0
5.5
8.0
10.5
13.0
15.5
18.0
20.5
23.0
25.5
28.0
30.5
33.0
35.5
38.0
40.5
43.0
45.5
48.0
50.5
53.0
55.5
58.0
60.5
63.0
65.5
68.0
70.5
73.0

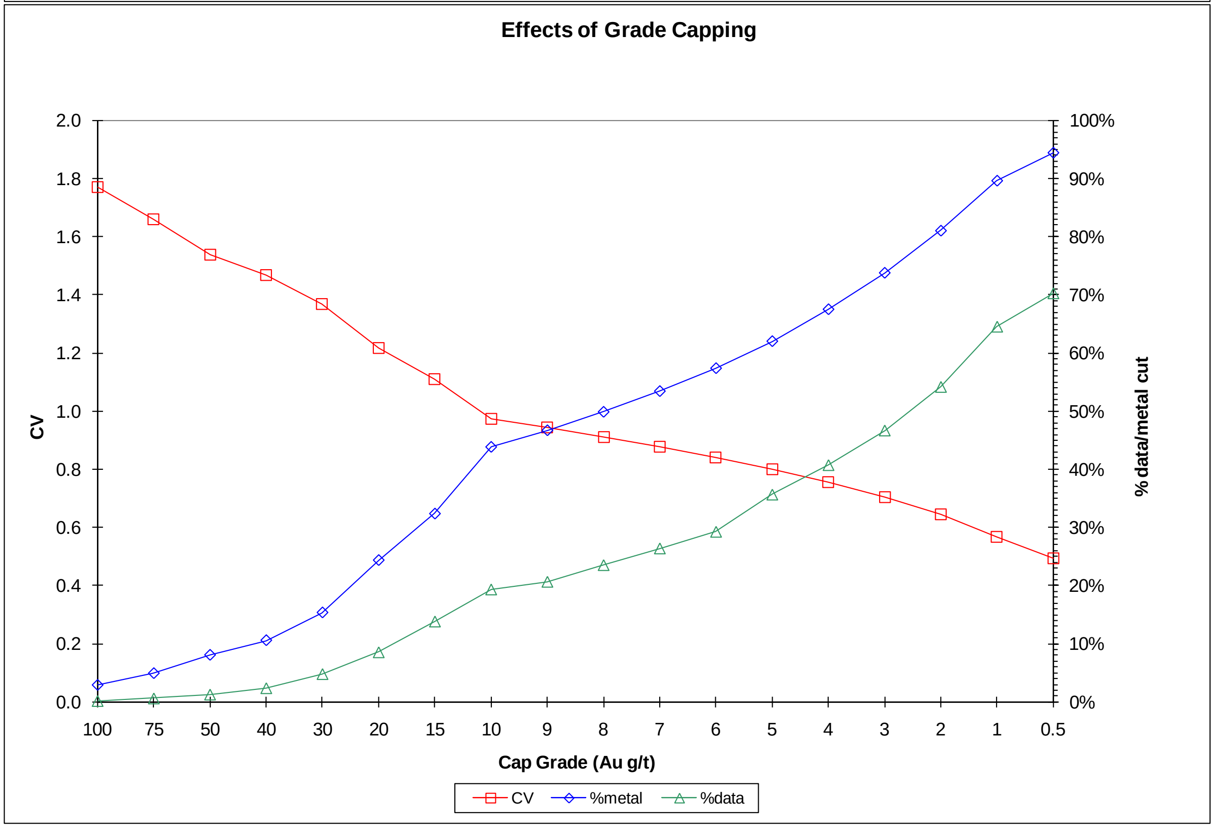

Histogram of Log10 Gold Assays in Wireframe
Frequency%
Bin Upper Limit (g/t)
0
2
4
6
8
10
12
14
16
0.000
0.000
0.001
0.003
0.008
0.020
0.053
0.138
0.362
0.949
2.488
6.525
17.108
44.861
117.631

Wireframe Assays Log-Probability Plot



Wireframe Assay Top Cuts

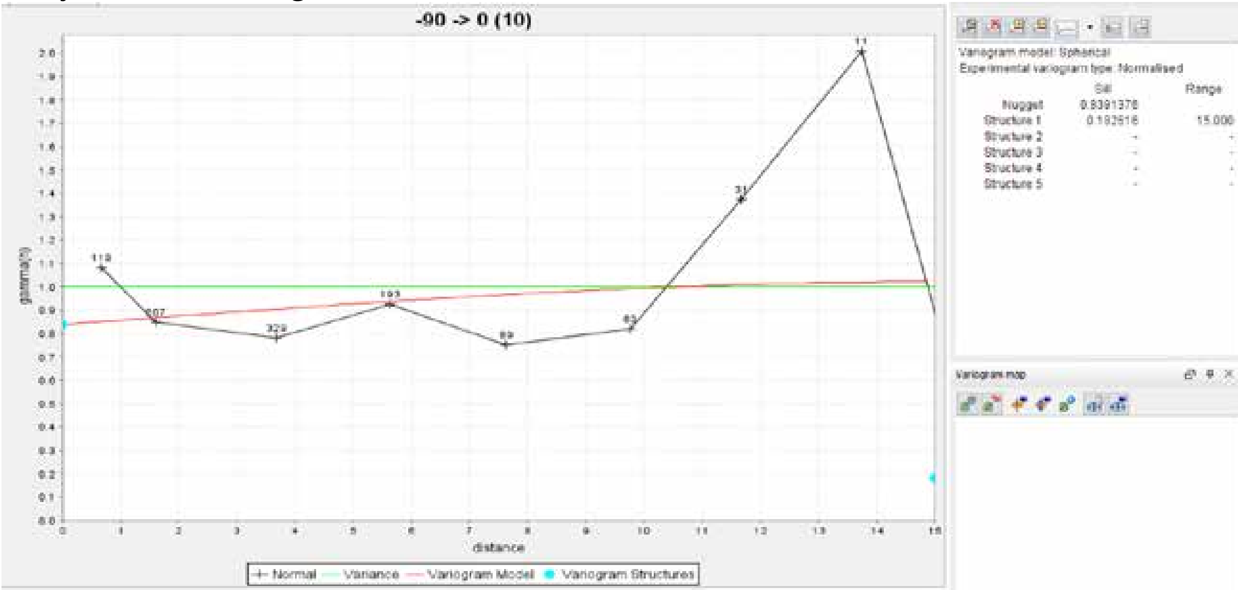


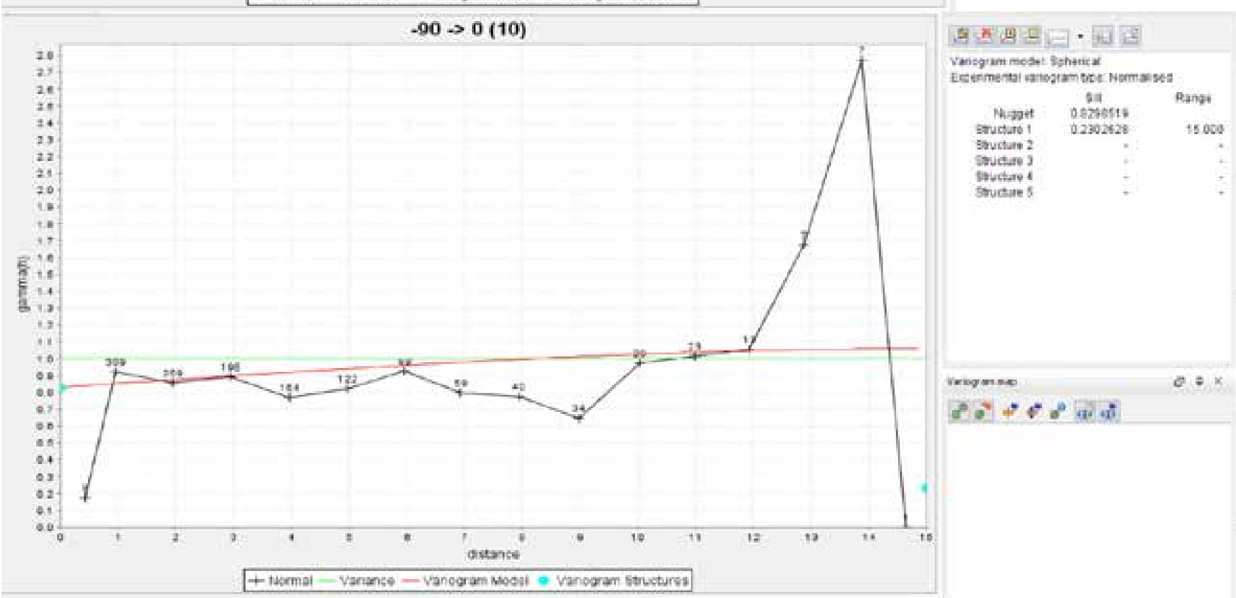


VARIOGRAPHY

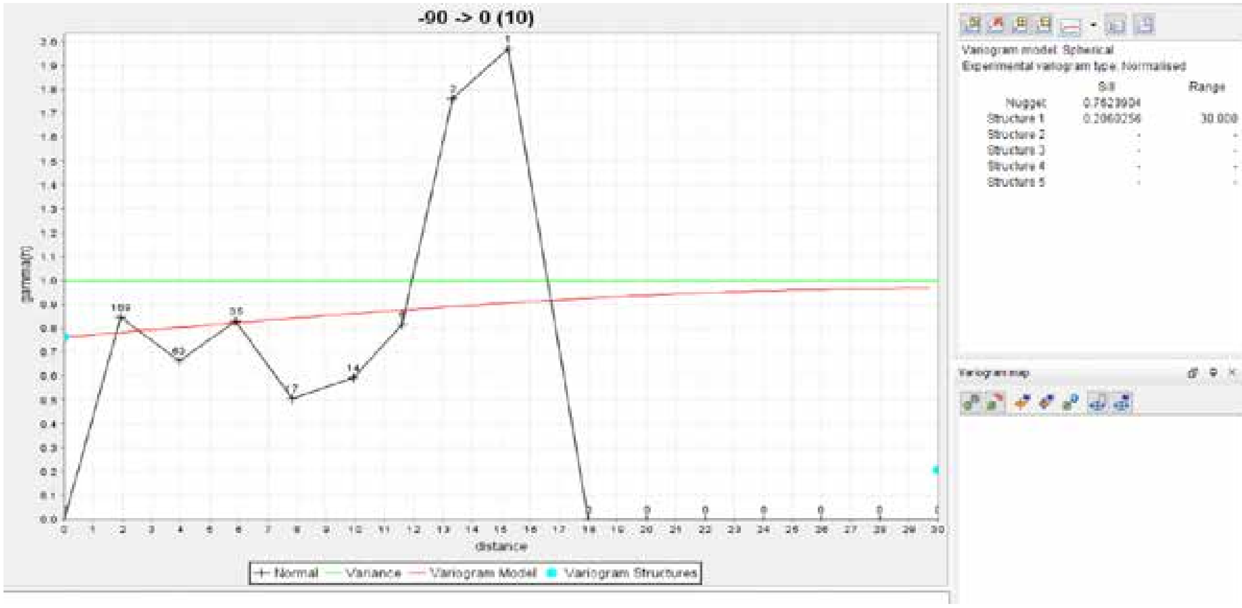
Down hole linear Semi Variograms

Assays 2 m and 1m Lags

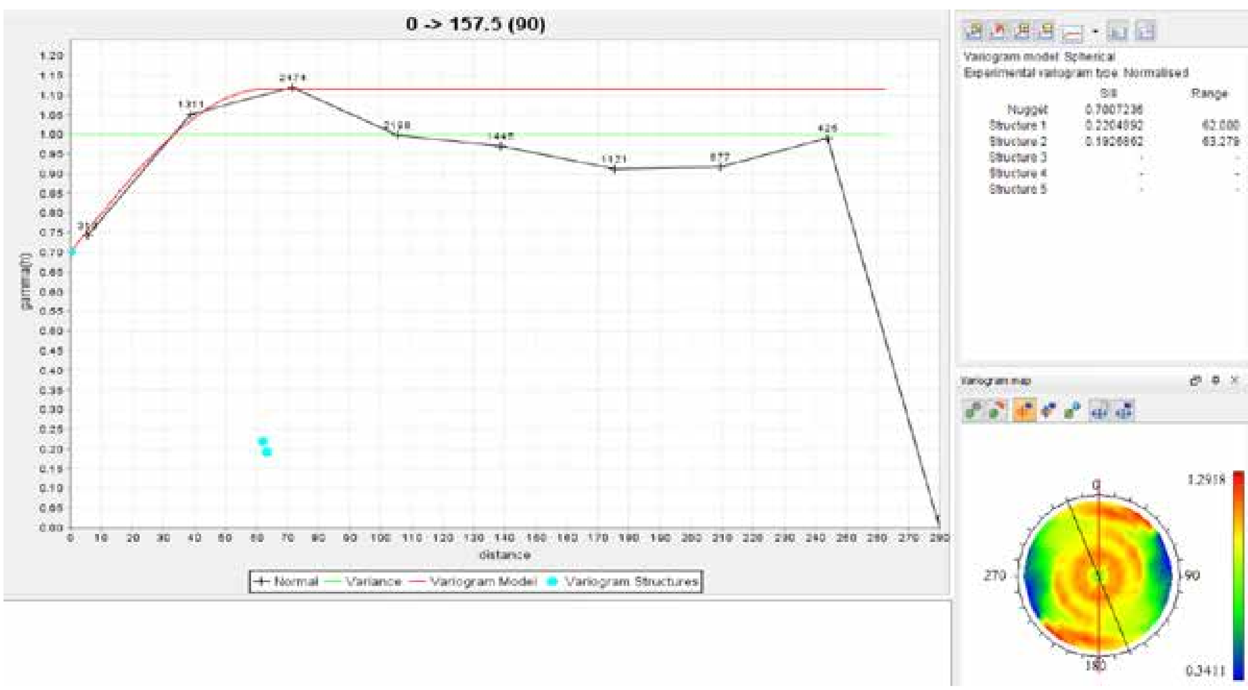


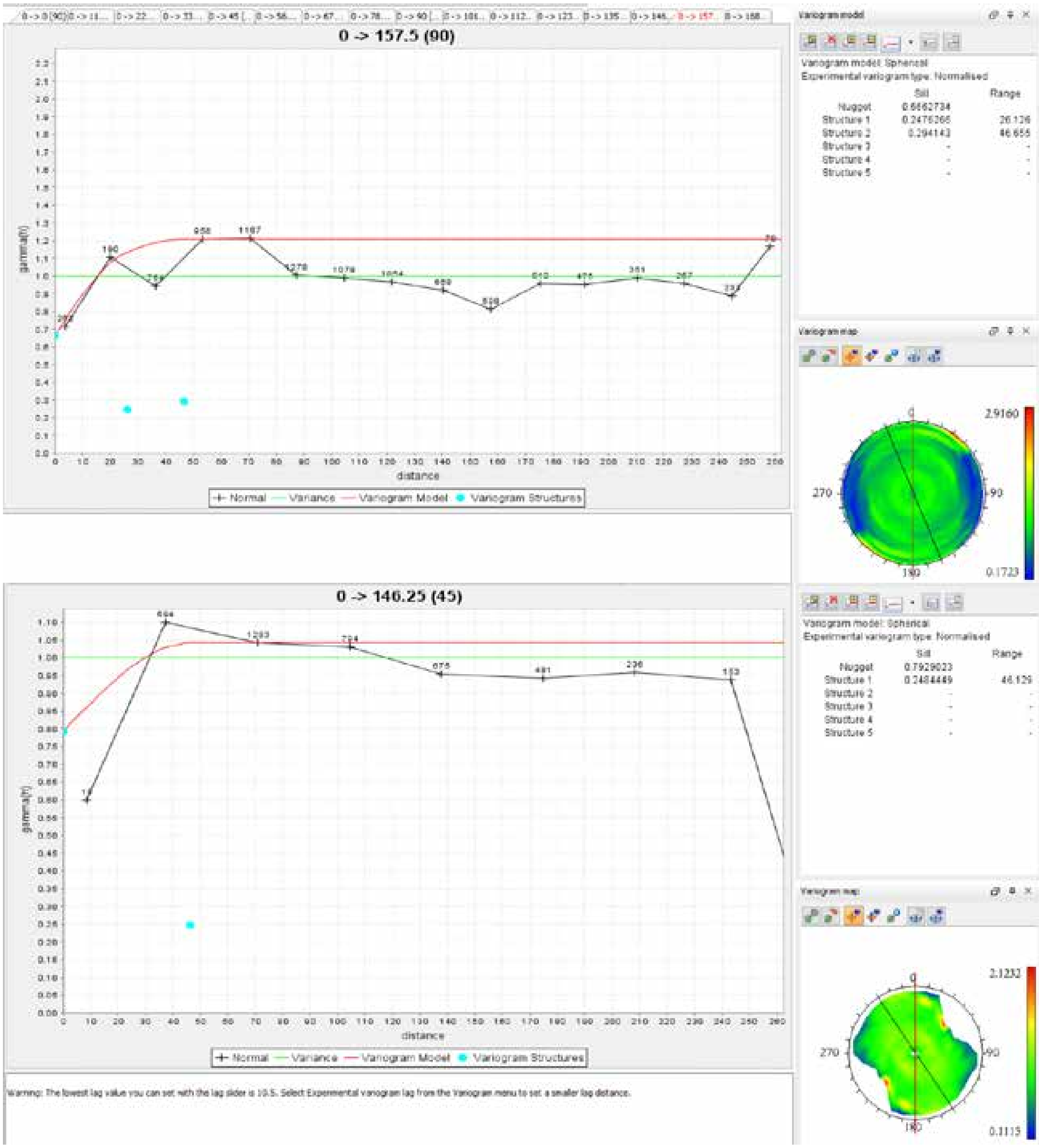


Composites Down-Hole Linear



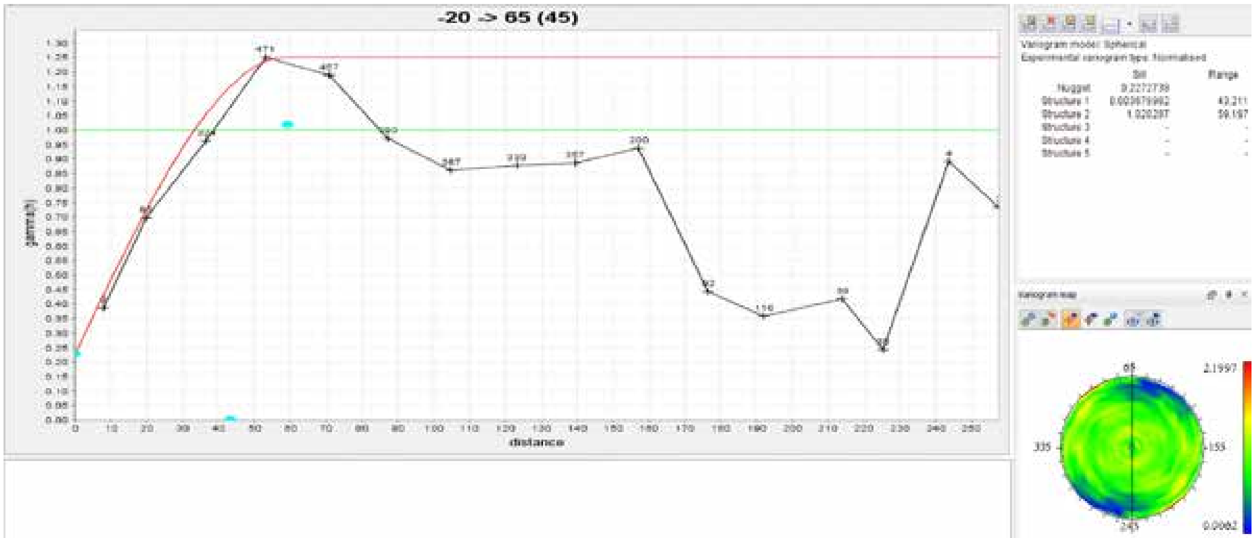
3D Semi Variograms
Strike



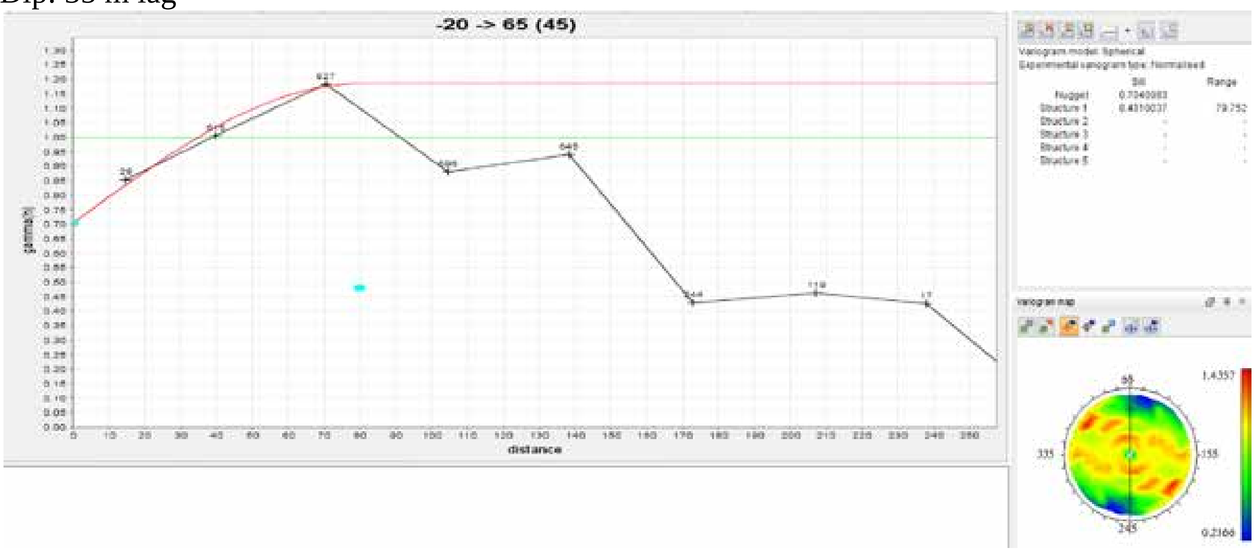


0 -> 157.5 (90)
gamma(h)
distance
Normal
Variance
Variogram Model
Variogram Structures
Variogram model
Variogram model: Spherical
Experimental variogram type: Normalised
Sill
Range
Nugget 0.6662734
Structure 1 0.2475265 26.126
Structure 2 0.294143 46.655
Structure 3 - -
Structure 4 - -
Structure 5 - -
Variogram map
2.9160
0.1723
0 -> 146.25 (45)
Variogram model: Spherical
Experimental variogram type: Normalised
Nugget 0.7929023
Structure 1 0.2484449 46.129
Structure 2 - -
Structure 3 - -
Structure 4 - -
Structure 5 - -
2.1232
0.1115
Warning: The lowest lag value you can set with the lag slider is 10.5. Select Experimental variogram lag from the Variogram menu to set a smaller lag distance.

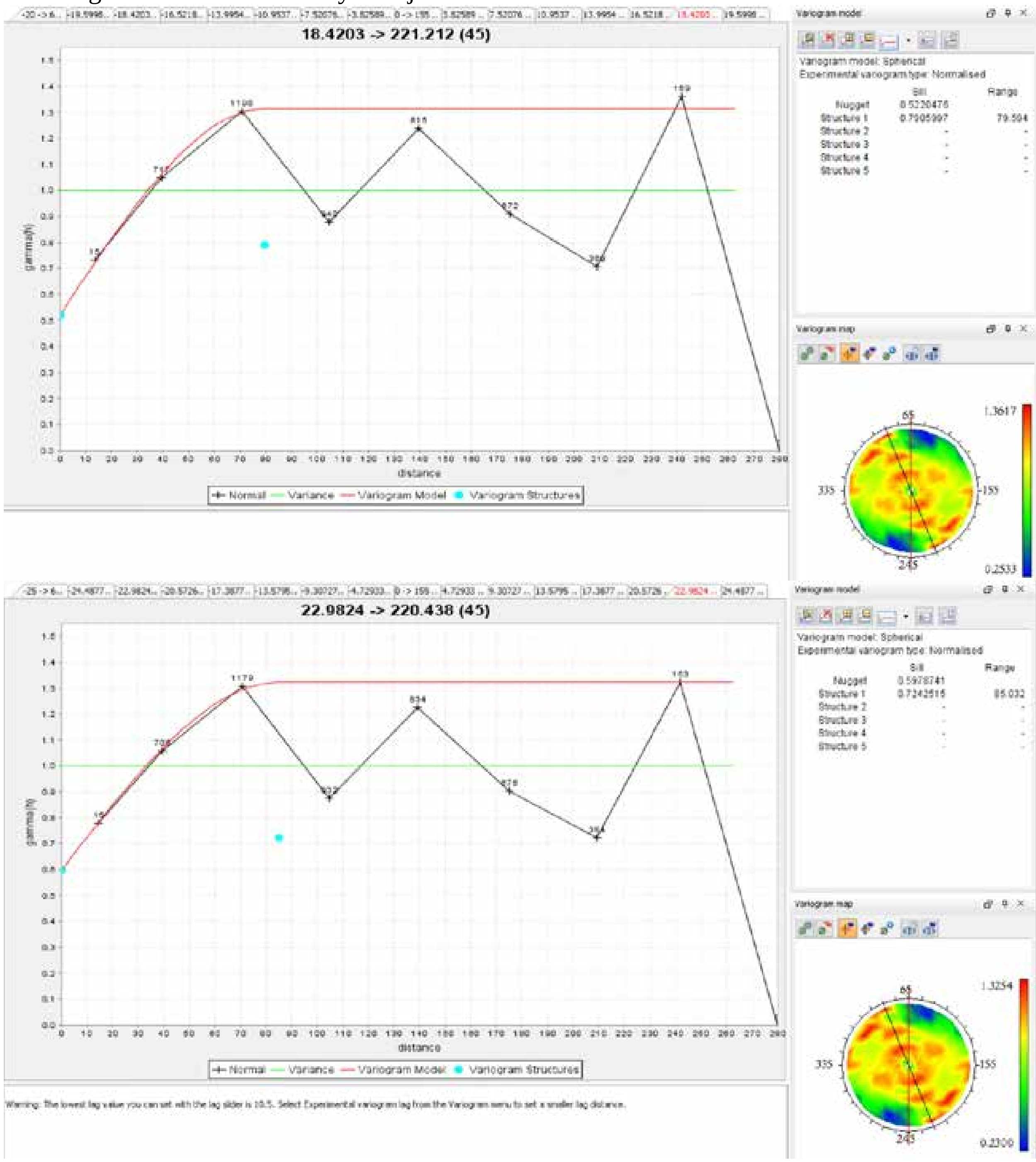
Dip: 17 m lag

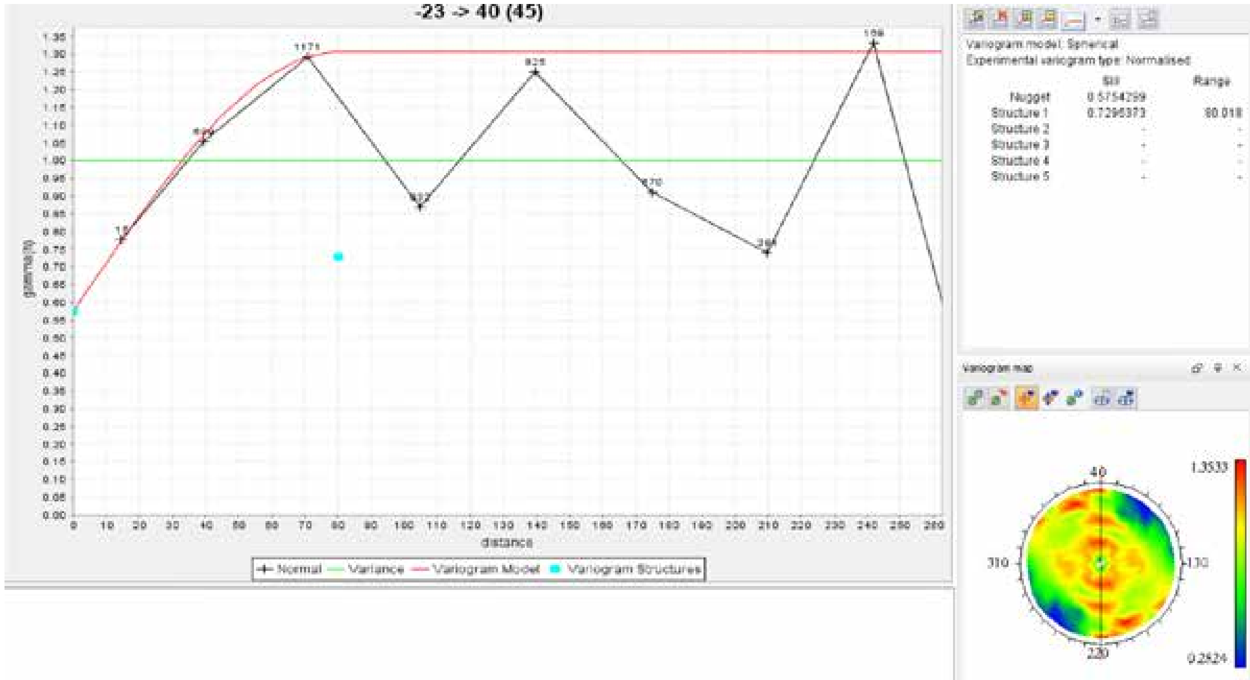


Dip: 35 m lag



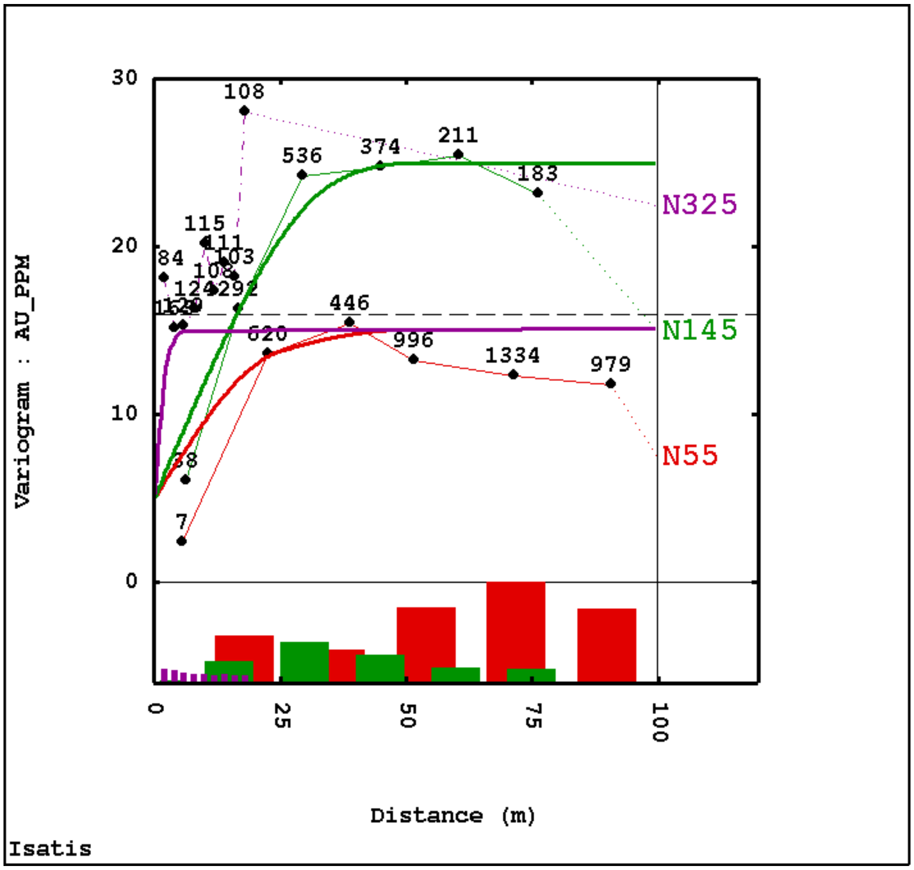
3D Plunge Maximum Continuity - Major Axis



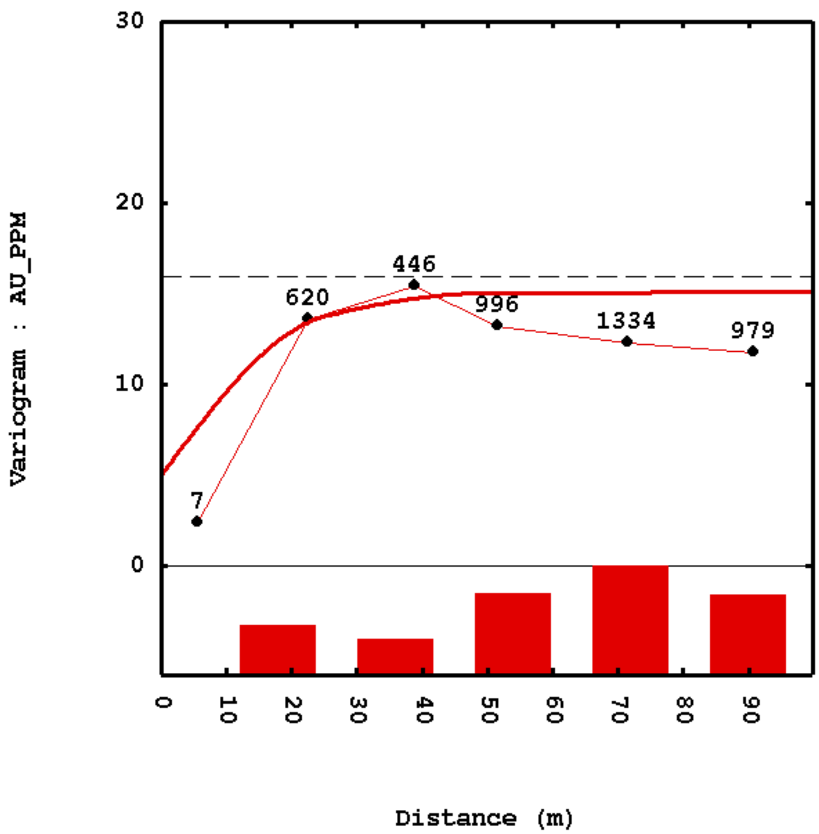


Intermediate Axis

APPENDIX II. LAVRINHA

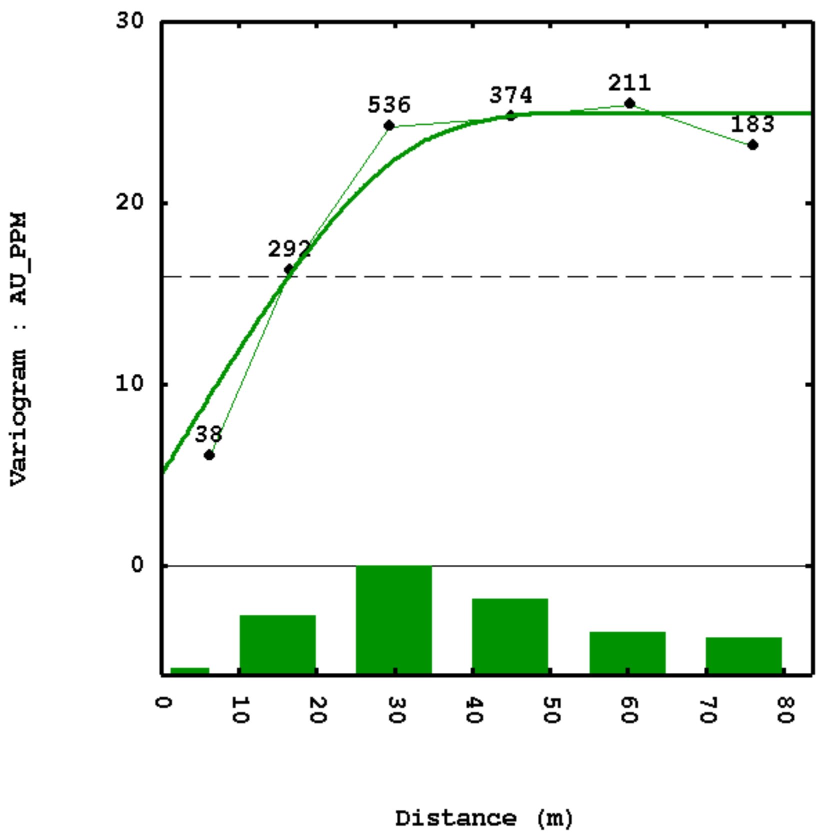


30
20
10
0
Variogram : AU_PPM
108
536
374
211
183
115
84
111
103
108
124
292
446
620
996
1334
979
38
7
N325
N145
N55
0
25
50
75
100
Distance (m)
Isatis



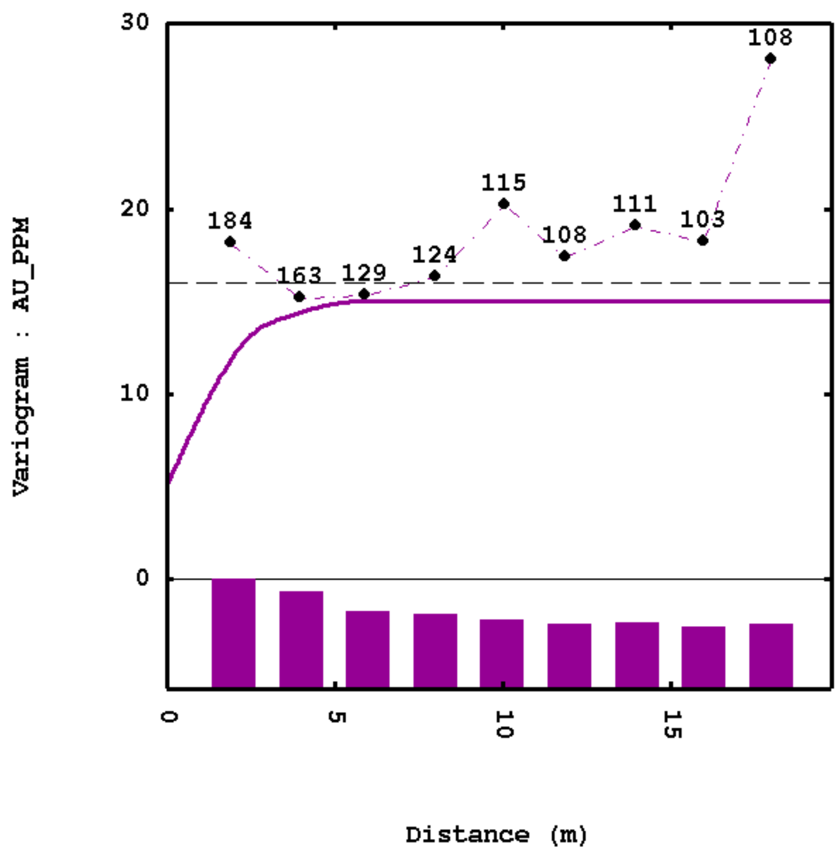
Isatis

Samples/DDH_2m(Sel_AUPPM cog 33)
- Variable #1 : AU_PPM
Variogram : in 1 direction(s)
D1 : N55
Angular tolerance = 22.50
Lag = 18.00m, Count = 6 lags, Tolerance = 50.00%
Horizontal Slicing = 18.00m
Vertical Slicing = 18.00m
Model : 4 basic structure(s)
Global rotation = Azimuth=N235.00 Dip=30.00 Pitch=-180.00 (Geologist Plane)
S1 - Nugget effect, Sill = 5
S2 - Spherical - Range = 3.00m, Sill = 6
Directional Scales = (25.00m, 35.00m, 3.00m)
S3 - Spherical - Range = 6.00m, Sill = 4
Directional Scales = (50.00m, 40.00m, 6.00m)
S4 - Spherical - Range = 50.00m, Sill = 10
Directional Scales = (9999.00m, 50.00m, 9999.00m)



Isatis

Samples/DDH_2m(Sel_AUPPM cog 33)
- Variable #1 : AU_PPM
Variogram : in 1 direction(s)
D2 : N145
Angular tolerance = 22.50
Lag = 15.00m, Count = 6 lags, Tolerance = 50.00%
Horizontal Slicing = 15.00m
Vertical Slicing = 7.50m
Model : 4 basic structure(s)
Global rotation = Azimuth=N235.00 Dip=30.00 Pitch=-180.00 (Geologist Plane)
S1 - Nugget effect, Sill = 5
S2 - Spherical - Range = 3.00m, Sill = 6
Directional Scales = (25.00m, 35.00m, 3.00m)
S3 - Spherical - Range = 6.00m, Sill = 4
Directional Scales = (50.00m, 40.00m, 6.00m)
S4 - Spherical - Range = 50.00m, Sill = 10
Directional Scales = (9999.00m, 50.00m, 9999.00m)



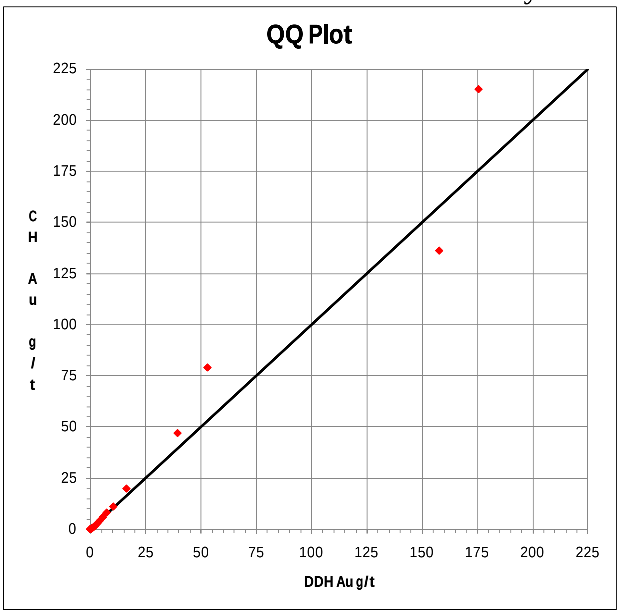
Isatis

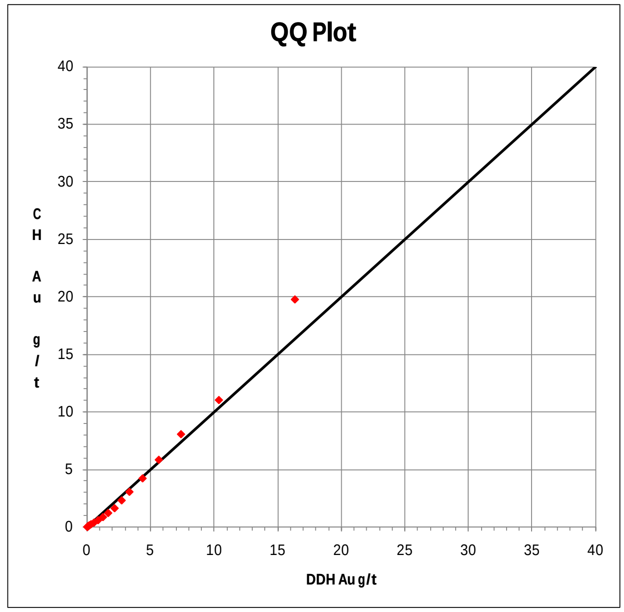
Samples/DDH_2m(Sel_AUPPM cog 33)
- Variable #1 : AU_PPM
Variogram : in 1 direction(s)
D3 : N325
    Angular tolerance = 22.50
    Lag = 2.00m, Count = 10 lags, Tolerance = 50.00%
Model : 4 basic structure(s)
Global rotation = Azimuth=N235.00 Dip=30.00 Pitch=-180.00 (Geologist Plane)
S1 - Nugget effect, Sill = 5
S2 - Spherical - Range = 3.00m, Sill = 6
    Directional Scales = (25.00m, 35.00m, 3.00m)
S3 - Spherical - Range = 6.00m, Sill = 4
    Directional Scales = (50.00m, 40.00m, 6.00m)
S4 - Spherical - Range = 50.00m, Sill = 10
    Directional Scales = (9999.00m, 50.00m, 9999.00m)

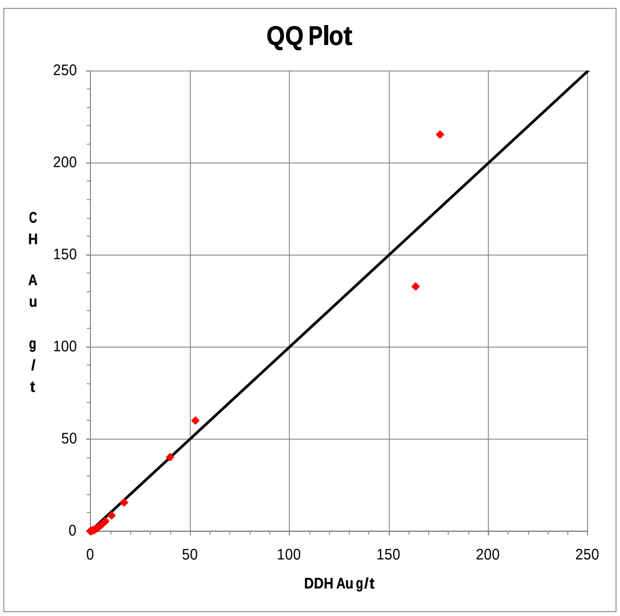
APPENDIX III. PAU-A-PIQUE

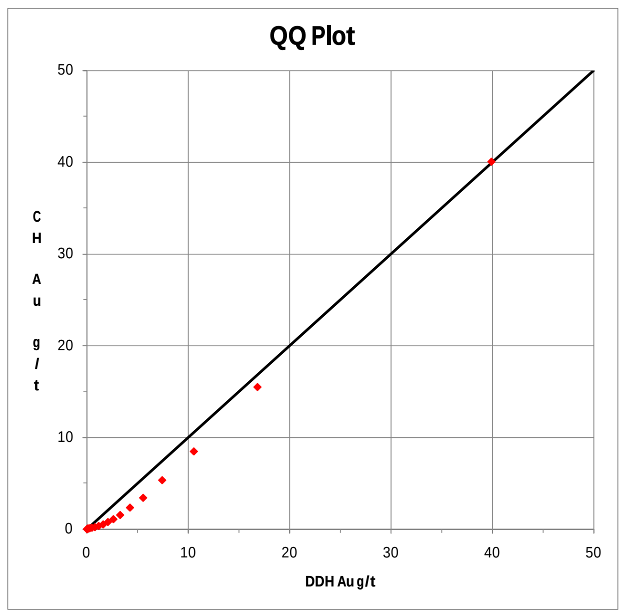
QQPlots

Main Zone DDH Core Assays versus Channel Chip Assays Constrained by 1.5 g/t Au Wireframes on Levels and Constrained by Zone Wireframe

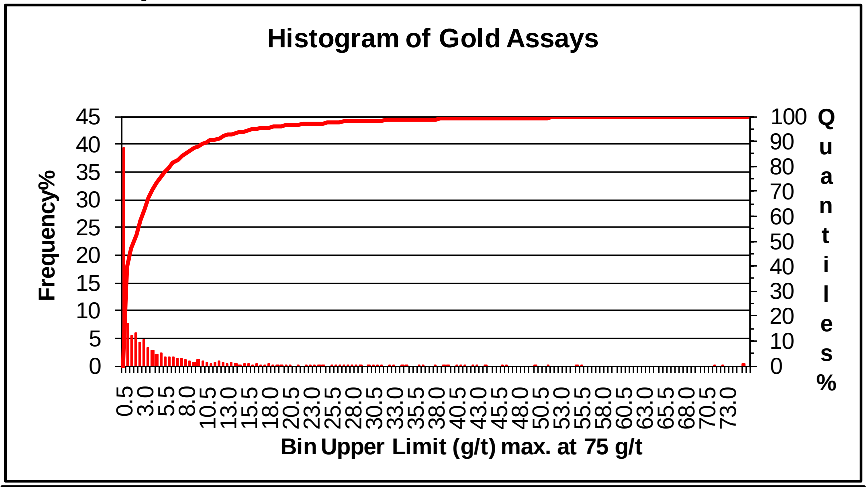


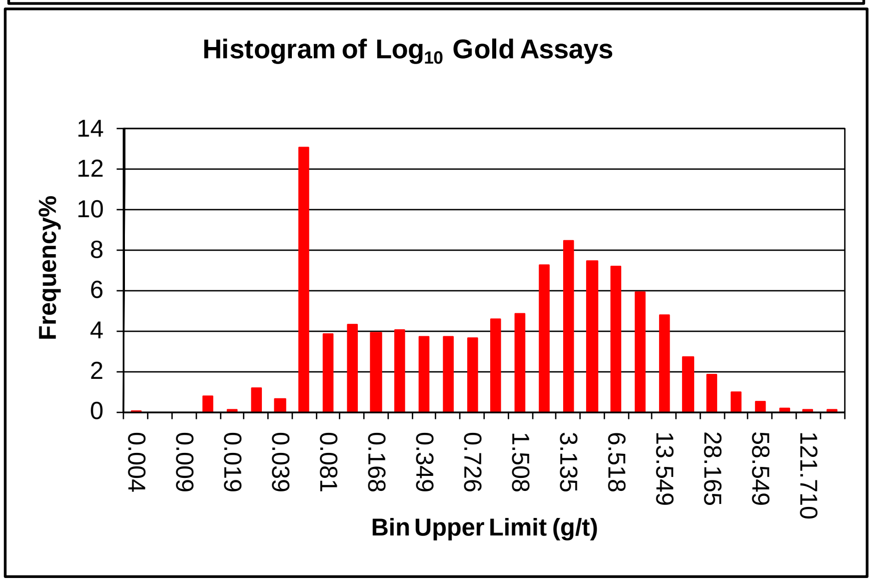




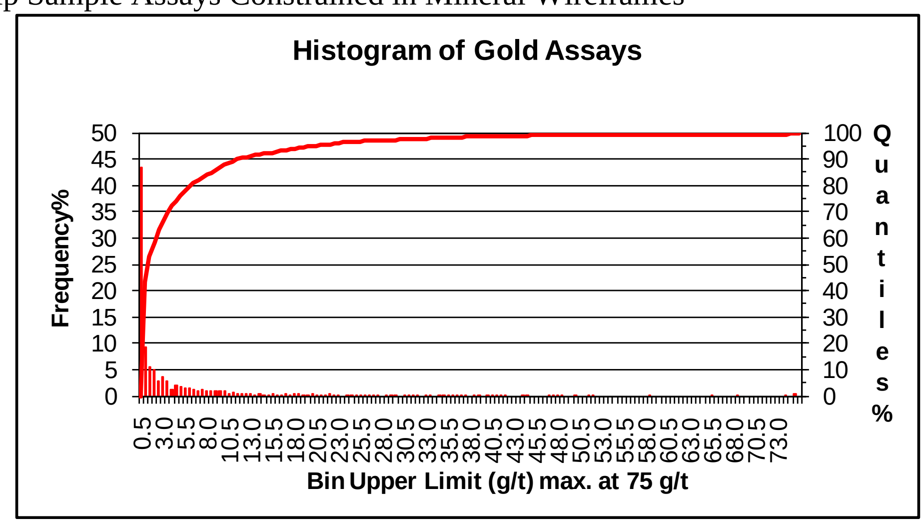


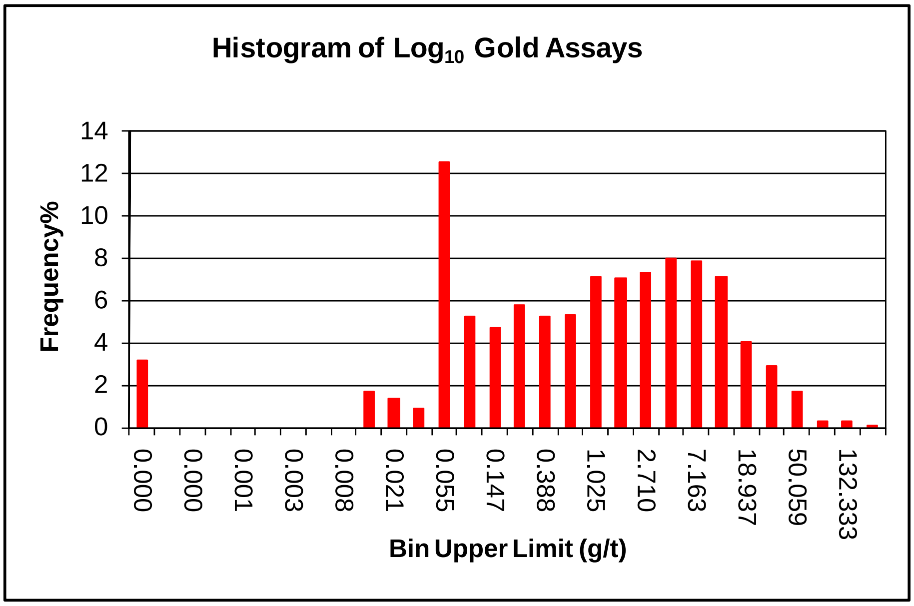
Statistics: Drill Core Assays Constrained in Mineral Wireframes



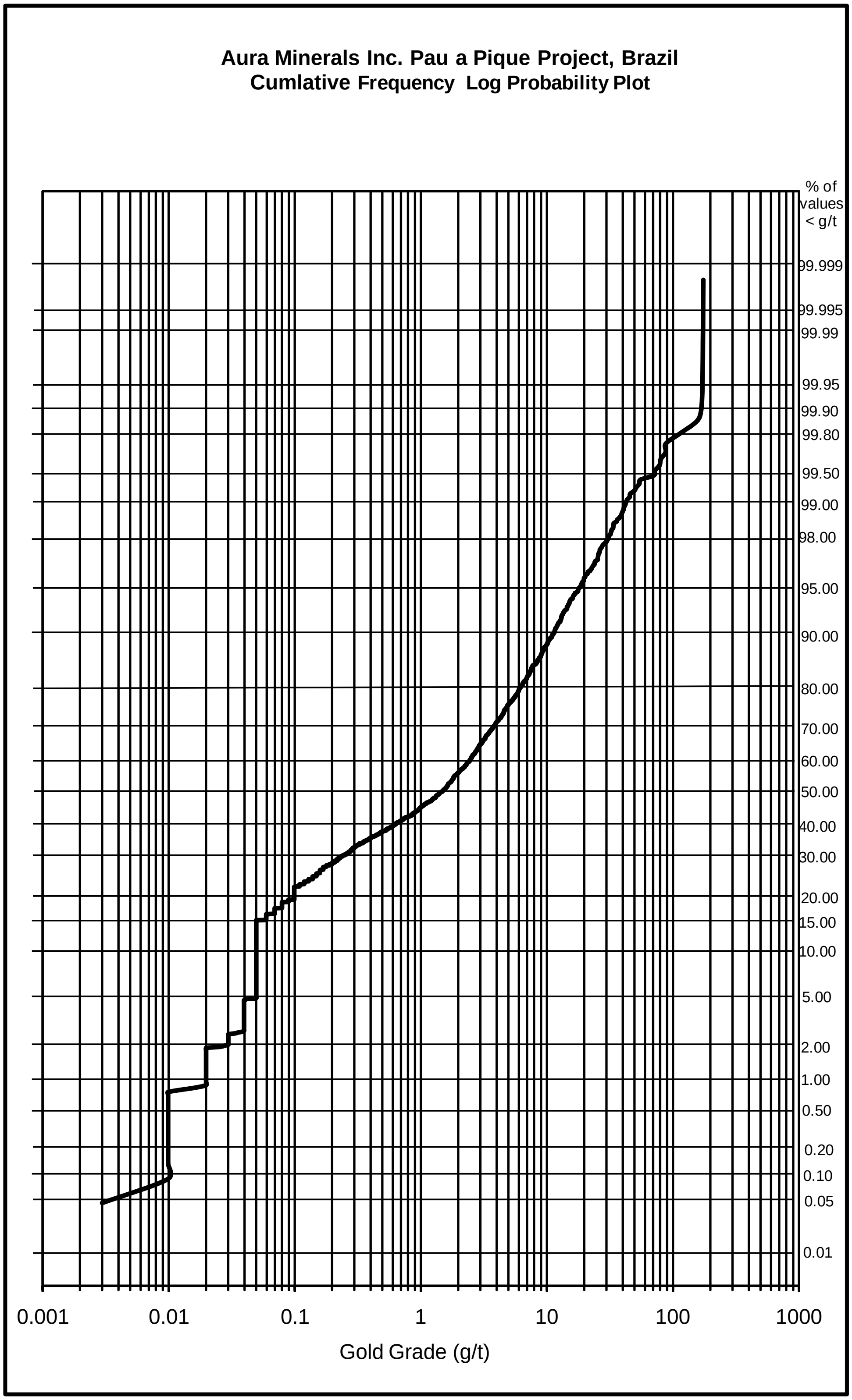


Channel Chip Sample Assays Constrained in Mineral Wireframes

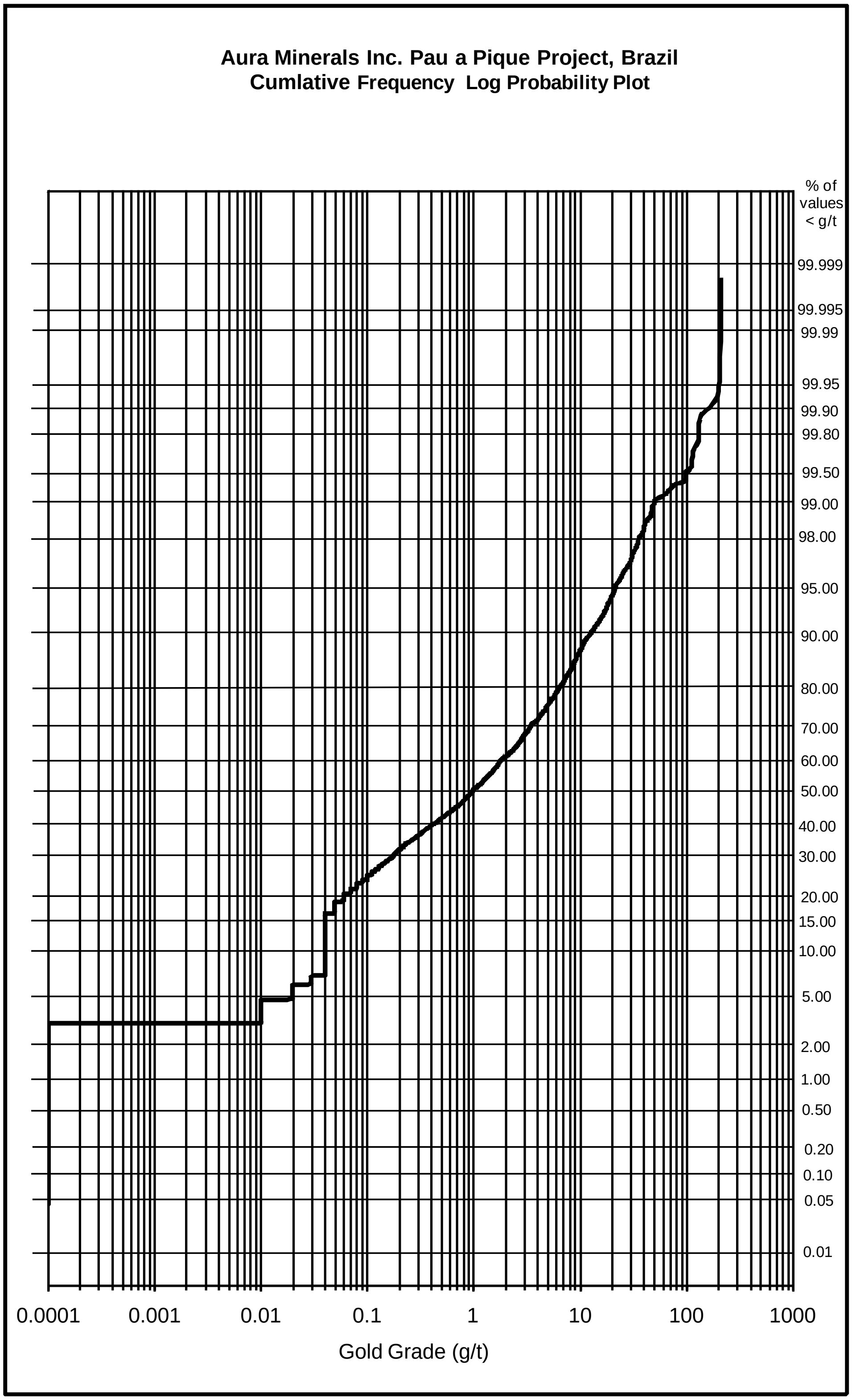



Histogram of Log10 Gold Assays
Frequency%
14
12
10
8
6
4
2
0
0.000
0.000
0.001
0.003
0.008
0.021
0.055
0.147
0.388
1.025
2.710
7.163
18.937
50.059
132.333
Bin Upper Limit (g/t)

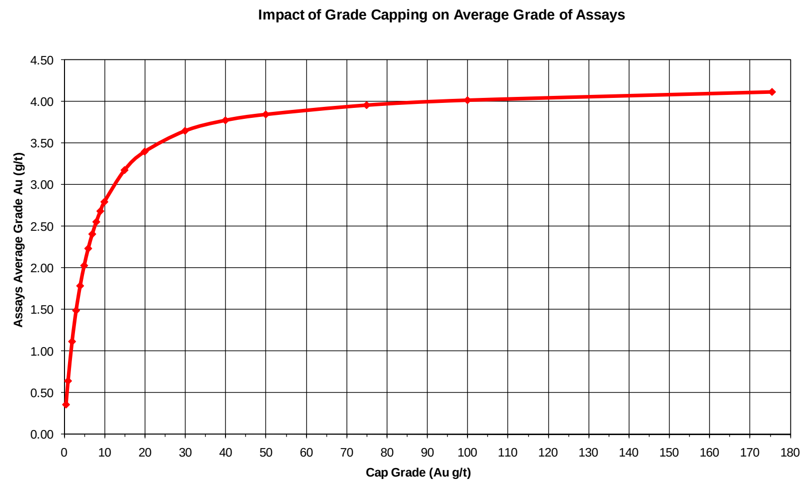
Drill Core Assay Log-Probability Plot

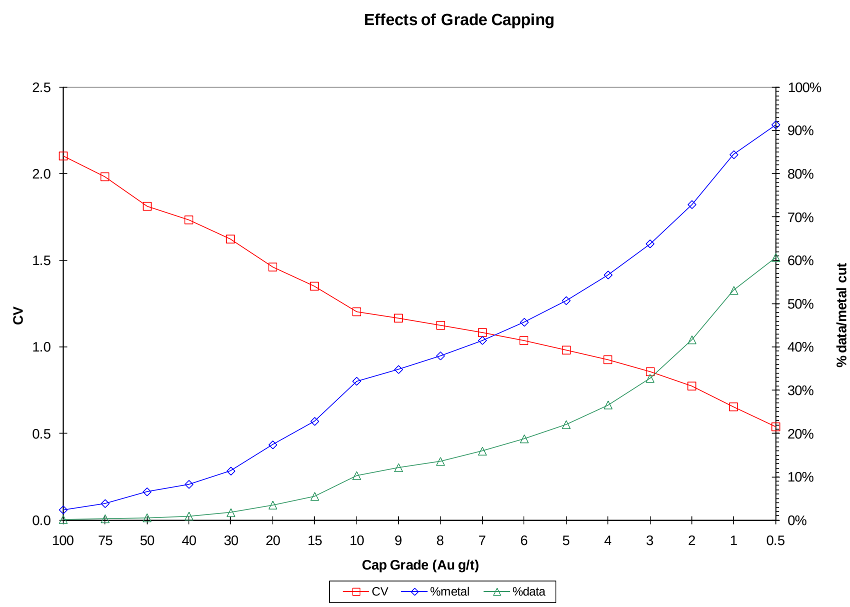


Channel Chip Sample Assay Log-Probability Plot

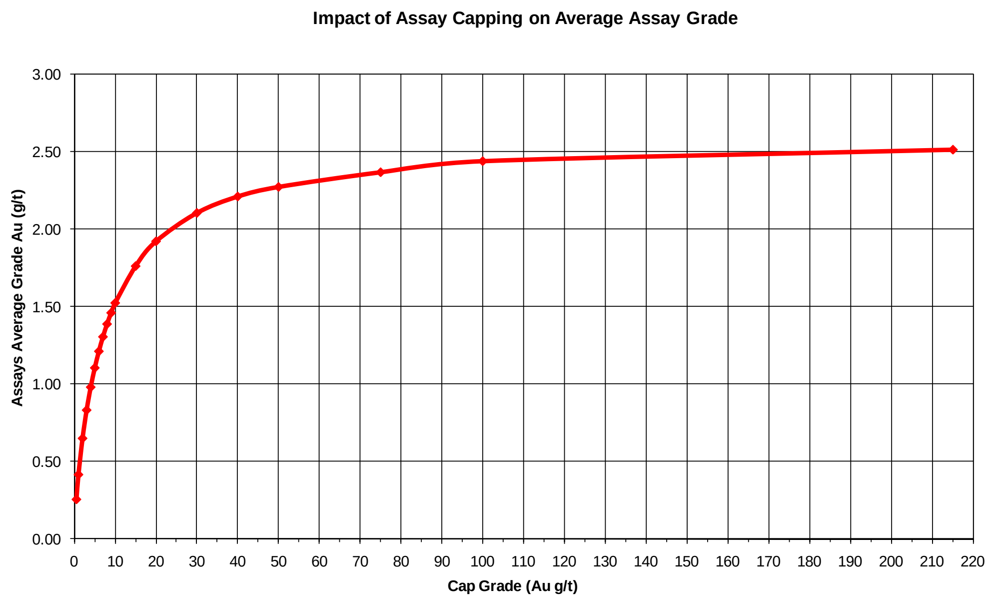


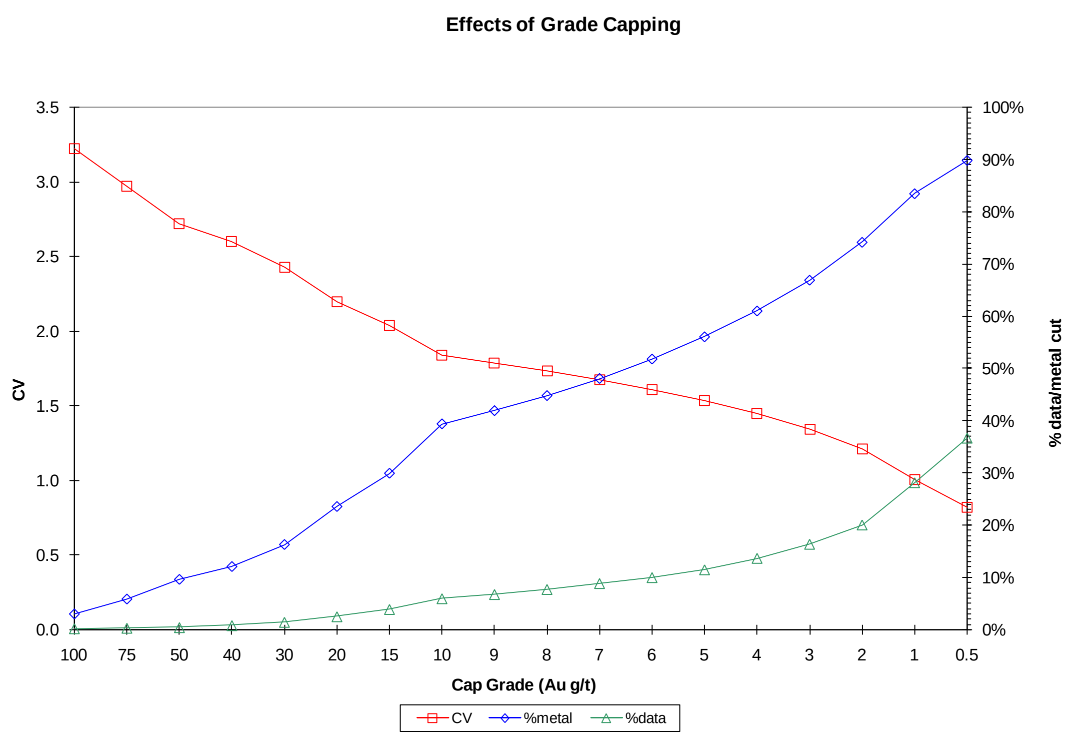
DH Assay Top Cuts





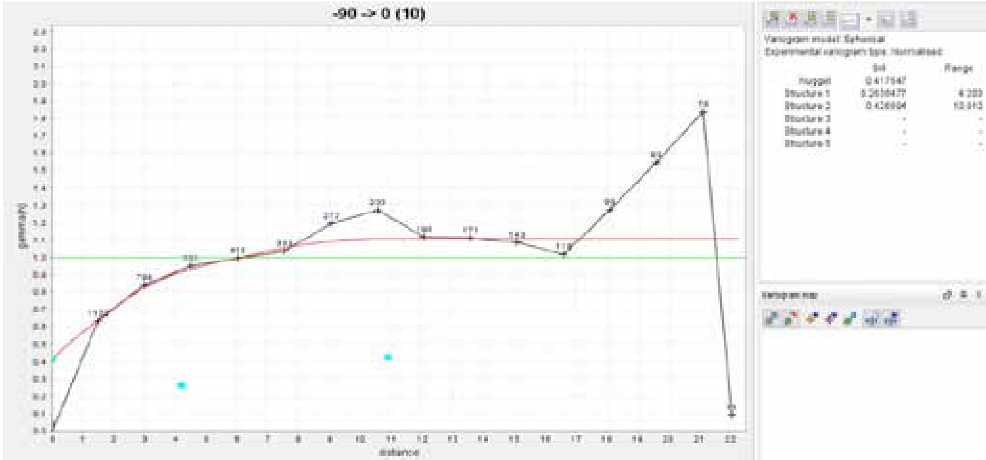
Channel Chip Sample Assay Top Cuts



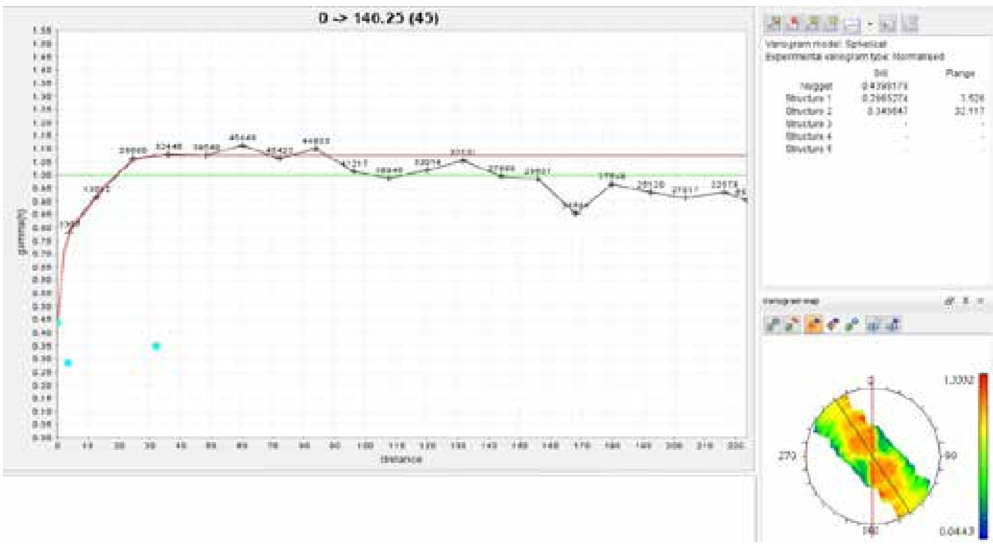


VARIOGRAPHY

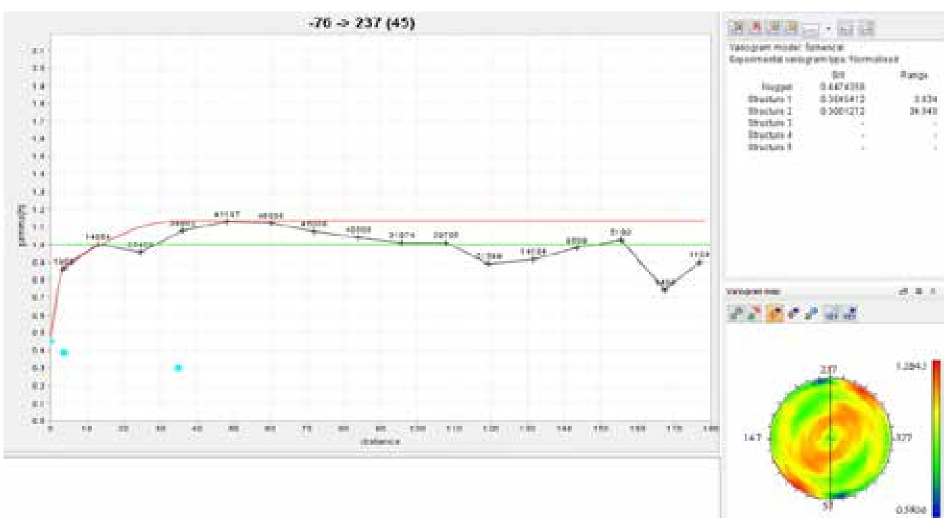
Down hole linear DDH



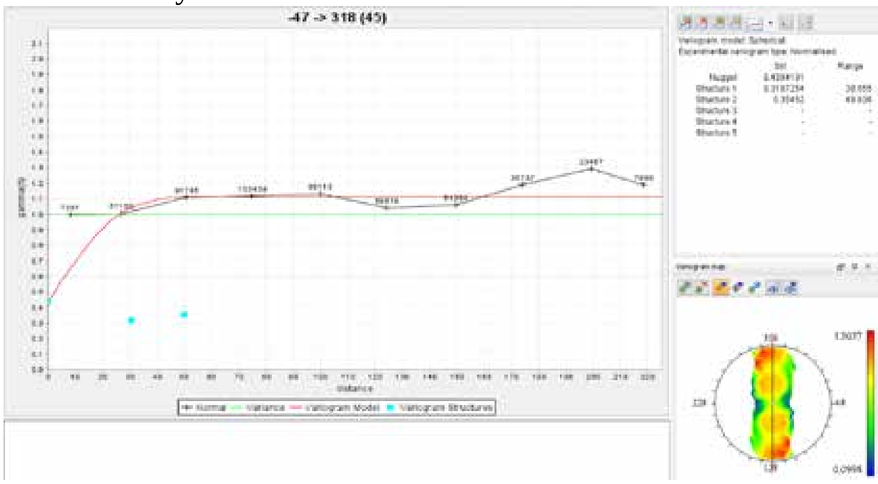
3D Strike



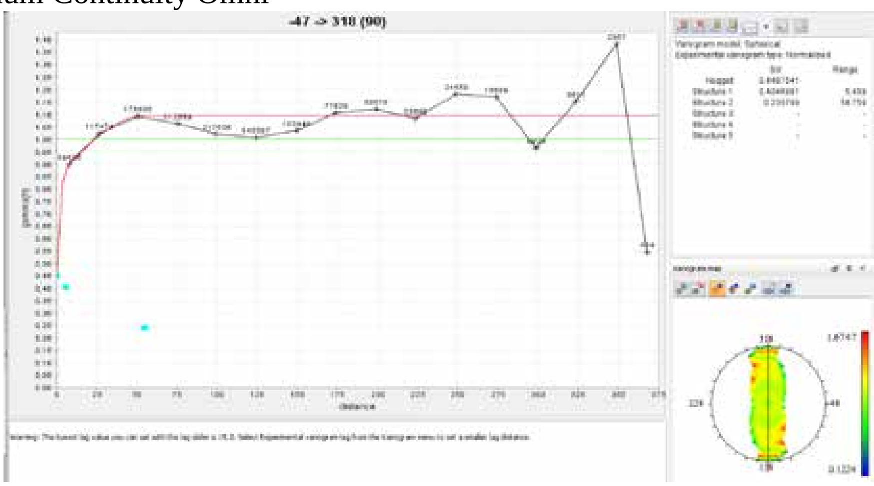
3D Dip



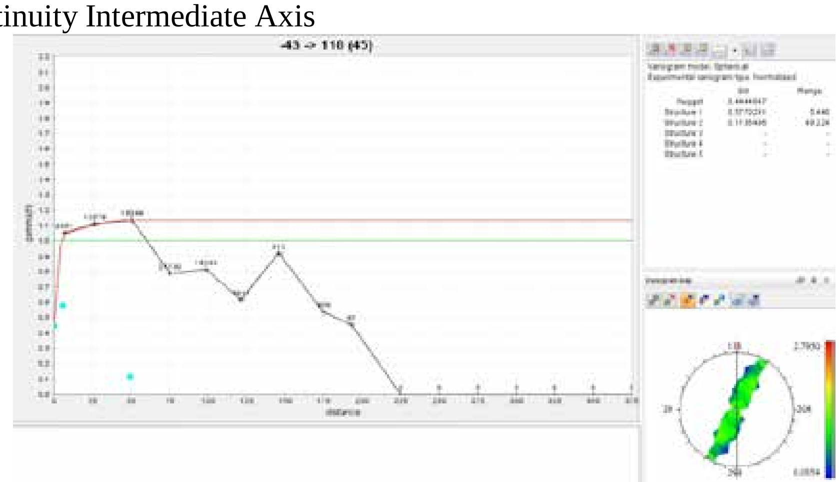
Plunge Maximum Continuity



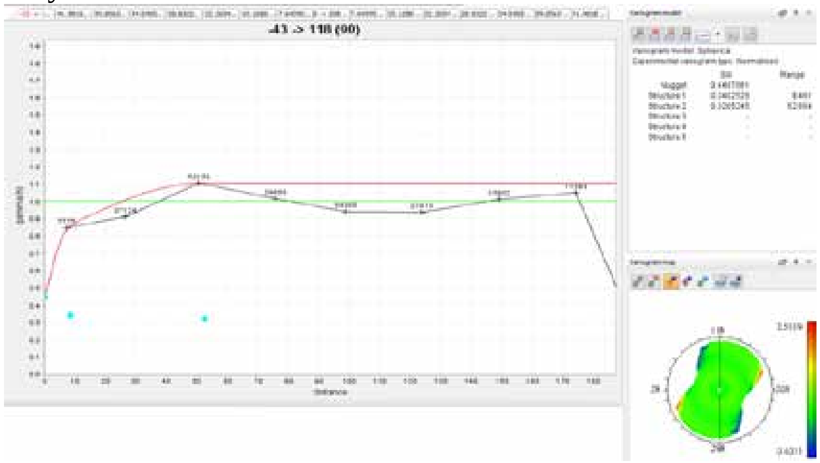
Plunge Maximum Continuity Omni



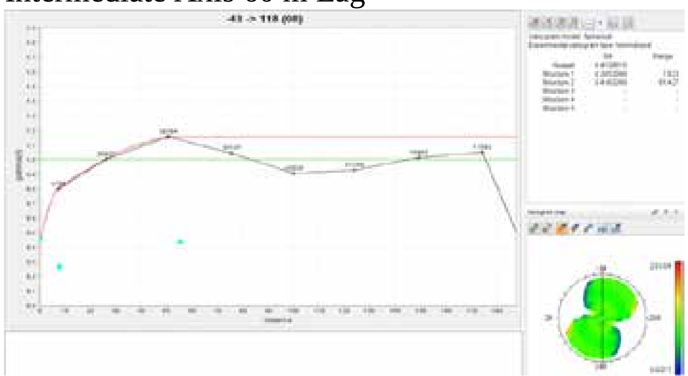
Maximum Continuity Intermediate Axis



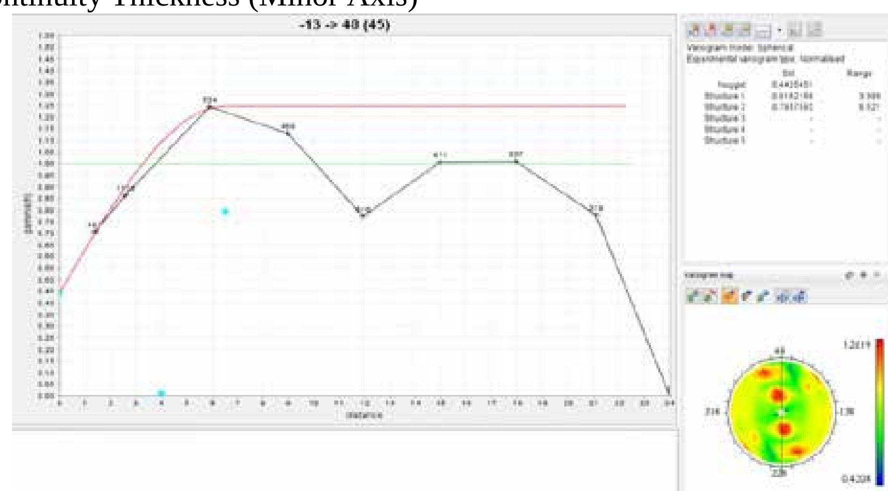
Maximum Continuity Intermediate Axis Omni



Maximum Continuity Intermediate Axis 60 m Lag



Maximum Continuity Thickness (Minor Axis)



Maximum Continuity Thickness (Minor Axis) Omni