Report Effective Date: January 31, 2018

FEASIBILITY STUDY OF THE RE-OPENING OF THE ARANZAZU MINE ZACATECAS, MEXICO

Prepared by Aura Minerals Inc.
In accordance with the requirements of National Instrument 43-101 "Standards of Disclosure for Mineral Projects" of the Canadian Securities Administrators

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1. SUMMARY

1.1 INTRODUCTION

This Technical Report has been prepared by Aura Minerals Inc. (Aura Minerals) in collaboration with engineering companies and specialized consultants and is in accordance with the requirements of National Instrument 43-101 “Standards of Disclosure for Mineral Projects” of the Canadian Securities Administrators (NI 43-101).

This Technical Report provides a Feasibility Study (FS) for the recommencement of operations at Aura Minerals’ wholly owned Aranzazu mining operation in Zacatecas State, Mexico (the Project). In January of 2015, due to the then mining and commodity price environment and other fixed costs, Aura Minerals made the decision to put the mine into care and maintenance while temporarily stopping underground development and production. Since then, a new assessment of the Mineral Resource estimate, a new Mineral Reserve estimate, the mining design and sequence, the underground geotechnical interpretations as well as metallurgy, mineralogy and tailings dam design were conducted, and are presented in this FS.

This FS presents the updated Aranzazu Mineral Resources estimates as of January 2018, prepared and validated by Farshid Ghazanfari, P.Geo. There were two previously disclosed Technical Reports for the Project, the first report dated November 20, 2011 (the 2011 Technical Report) entitled, “NI 43-101 Technical Report and Resource Estimate on the Aranzazu Property, Zacatecas State, Mexico”, prepared for Aura Minerals by William J. Lewis P.Geo., of Micon International Limited (Micon) and the second report dated September 28, 2015 (the 2015 Technical Report) entitled “Preliminary Economic Assessment of the re-opening of the Aranzazu Mine” prepared by PandE Mining and Aura Minerals in collaboration with other external consultants.

Subsequent to the 2011 Technical Report three Mineral Resource updates were completed by Aura Minerals (2013, 2014 and 2015). The mineralization domains were significantly modified after the release of the 2015 Technical Report. The Mineral Resource model was built on the improved geological understanding of grade continuity, domain orientation, and shape gained from the 2013 and 2014 Resource Models.

This FS also provides the first NI 43-101 Mineral Reserve estimate developed for the Aranzazu Mine since its opening in 2011. Mineral Reserves are expected to be recovered from the underground mine, since the prior open pits have been essentially exhausted. Geotechnical considerations and deposit dimensions of specific sections of the mine determine when transverse or longitudinal long hole stoping is to be used with the expectation of approximately 90% and 10% of the production coming from each method respectively.

After the ramp-up period is complete and without any significant expansion, the production is expected to remain similar to the average 2,600 tonnes per day (TPD) of throughput produced prior to shut down for care and maintenance in January 2015. A five-month underground development program followed by an eight-to-nine month ramp up period is expected prior to achieving the 2,597 TPD throughput in the plant.

Key Project infrastructure includes continued underground development with associated pumping and ventilation infrastructure, and the construction of a new Tailings Dam storage facility (TD5) in Year 1. Additionally, the construction of a new tailings thickener is also planned for Year 2 in order to maximize process water recovery and relieve the existing fresh water system. There is currently sufficient power to operate the mine and processing facilities, but a dedicated 6.0 km, 34.5 kV line from the national power company is planned to be built and connected to the mine.

All monetary values shown in this FS are US Dollars (US$) unless otherwise stated.

The term "Aranzazu" refers to the immediate area surrounding the open pits and underground workings of the Arroyos Azules mine, where active mining will be carried out. The term Property refers to the entire land package owned by Aura Minerals.

1.2 HISTORY, LOCATION AND OWNERSHIP

Aranzazu is located within the Municipality of Concepcion del Oro in the State of Zacatecas, Mexico near its northern border with the State of Coahuila. The Property is situated in a rugged mountainous area and is accessed either from the city of Zacatecas, located 250 km to the southwest, or from the city of Saltillo, located 112 km to the northeast in the State of Coahuila. Both Zacatecas and Saltillo have modern airports with daily flights to and from Mexico City and parts of the United States. Aranzazu lies on the western edge of the town of Concepcion del Oro, with a population of approximately 6,500 people. Most of the families have had a historic connection to mining, resulting in the availability of a semi-skilled to skilled workforce.

The mine facilities are at an elevation of approximately 2,150 masl, with the surrounding mountains reaching elevations of 3,300 masl. The area is semi-arid and moderately vegetated with acacia shrubs, scrub trees and bushes, Joshua trees and various cacti. The average high temperature in the summer is about 22°C and the average winter high is about 15°C. The average summer low temperature is about 15°C and the average winter low temperature is about 5°C.

The area receives approximately 432 mm of rain annually and annual pond evaporation is estimated at 1983 mm. The majority of the rain falls during the wet season from June through October, and the 50-year recurrence interval 24-hour storm is estimated at 93 mm. Occasionally, snow does occur in the area, but quickly melts on all but the most protected northern slopes.

The climate is mild year-round and poses no limitations to the length of the operating season. Freezing temperatures can occur overnight but quickly warm to above freezing during daylight hours.

Historical mining activities began in the district as early as 1548. In 1891, the Mazapil Copper Company of Manchester, England began mining and smelting operations that continued through to 1962. From 1962 until 2008, various companies have owned and operated the Aranzazu Mine.

After shutting down in 1992 due to low metal prices and a labour dispute, the mining operations were restarted on a limited scale in 2007 by a private Mexican company. Aura Minerals acquired 100% of the Aranzazu Mine (formerly known as the El Cobre project) in June 2008. Production was suspended in January 2009 but restarted on a limited basis in 2010, with commercial production declared effective February 1, 2011. A summary of reported Aura Minerals production is contained in Table 1-1.

Table 1-1 Summary of Aranzazu Production (2008 to 2017)

Year	Mill Feed (tonnes)	Head Grades			Conc (tonnes)
		Cu (%)	Au (g/t)	Au (g/t)	
2008	148,511	0.69	0.25	7.9	3,116
2009*	-	-	-	-	-
2010	57,211	0.51	-	-	831
2011	632,297	0.90	0.48	12.9	13,455
2012	771,774	0.85	0.50	11.9	20,671
2013	796,413	0.98	0.48	16.2	25,813
2014	861,983	0.88	0.45	14.6	26,294
2015 - 2017*	-	-	-	-	-

** Mine under care and maintenance*

Aura Minerals owns the Aranzazu Property indirectly through its 100% owned subsidiary Newington Corporation S.L. (Newington) which, in turn, holds 100% of the Aranzazu Property through its Mexican subsidiary Aranzazu Holding S.A. de C.V. (Aranzazu Holding). The 38 mineral concessions are mostly contiguous with some having been established prior to current staking regulations which vary in size, shape and orientation. The total property area is approximately 11,182 ha. All concessions are valid for 50 years, with the term extendable for concessions maintained in good standing. Mining concession duties are paid semi-annually and the yearly total for 2018 is approximately 2,303,490 Mexican pesos (MXN), which is equivalent to approximately US$128,000 at an exchange rate of 18:1.00 MXN:US$.

The previous owner Macocozac S.A. de C.V. transferred its rights to the Aranzazu Property to Aranzazu Holding in exchange for a 1.0% Net Smelter Returns royalty (NSR) on the copper production when, during any calendar month, the monthly average copper price as quoted by the London Metals Exchange (LME) equals or exceeds US$2.00/lb.

Aranzazu Holding has a creditor agreement to repay outstanding debt of US$6.5M with certain suppliers and contractors who worked with Aranzazu before the 2015 shutdown. Aranzazu Holding is to commence payment to creditors two months after receipt of payment for the first concentrate shipment that may be any time between April 2018 and no later than April 2019. The debt is to be paid to each creditor in 36 equal monthly payments, with full payment by no later than April 2023.

To the extent known, the Aranzazu Property is not subject to any other royalties, back-in rights, or other encumbrances.

One potential and ongoing issue with surface rights is that squatters have constructed homes in some areas near the edges of the town on the mineral concessions. Within the town, some portions of the water supply pipeline serving the mine were built over the decades prior to acquisition by Aura Minerals. Should the mine require access to or direct use of these lands in the future, they may be obligated to lease or purchase the surface rights to these areas.

1.3 GEOLOGY AND MINERALIZATION

In the Concepcion del Oro district, a Tertiary intrusive complex ranging in composition from quartz monzonite to granodiorite intrudes the Jurassic and Cretaceous limestone along the axial plane of the El Mascaron antiform. The intrusive complex is also localized by the regional transform fault system. Associated with the intrusive complex and its structural system and alteration regime, copper, gold and silver mineralization occurs as chimneys, mantos, stockworks and disseminations hosted in exoskarn, endoskarn, quartz monzonite, hornfels and marble. These have been over printed by post-skarn hydrothermal alteration consisting of propllitic phyllic and potassic alteration styles. Both mineralized and un-mineralized skarns are considered.

The orebody has a strike length of 1.5 km, width up to 250 m and a depth of 600 m. The orebody consists of seven mineralized domains which are BW, AA, Mexicana South and North, Glory Hole Footwall (GHFW) and Glory Hole Hangingwall (GHHW) and Cabrestante. These are multiple chimney structures dipping south east between 70 to 90 degrees.

The distribution of the various alteration phases and associated copper mineral species is variable along the strike of the main structure in the Aranzazu Mine and consists of several zones. In the BW and upper Mexicana zones, host to propylitic alteration, the copper mineralization is mostly chalcopyrite. Copper mineralization is present in phyllic and potassic alteration styles at depth and to the southeast through to the Cabrestante zone where there is chalcocite, copper sulphosalts, chalcopyrite and bornite. The trace metal assemblages also vary depending on the copper minerals present. The BW zone contains moderate amounts of arsenic, but is relatively low in antimony, bismuth, and tellurium. In areas of phyllic and potassic alteration where multiple copper mineral species are present, the amounts of arsenic, antimony, bismuth and tellurium increase. Gold mineralization occurs throughout all the alteration phases previously mentioned apart from skarn alteration. Gold grades are generally higher in the phyllic and potassic alteration assemblages, compared to the propylitic altered rocks.

1.4 EXPLORATION AND DATA MANAGEMENT

Aura Minerals has carried out core and reverse circulation (RC) drilling to upgrade the Mineral Resources in the Mexicana, Arroyos Azules and Glory Hole areas. This occurred in two phases, from August to December 2008, and April 2009 to May 2011. The total amount of drilling completed in the two phases was 108,052 m in 471 holes. Since the 2011 Technical Report, Aura Minerals has reported an additional 10,000 m in 90 holes with February 7, 2014 as the database cut-off date. Thirty of these drill holes (UAZ-51 to UAZ-81) were drilled in 2010, but the assay data missed the cut-off date for the 2011 Resource Model. An additional 4,167 m in 37 drill holes have been drilled since February 7, 2014 and this data has been included in the current Mineral Resource estimate.

Drill hole spacing in these areas is now approximately 25 m by 25 m between the 2050 m and 1850 m elevations. RC drilling was carried out in the Glory Hole-Porfido and Cabrestante areas to test near-surface mineralization that has the potential to be mined by open pit. Drill hole spacing in these areas is also approximately 25 m by 25 m for the near-surface mineralization. The deep mineralization, below 1700 m elevation, was also tested by core drilling in the AA, Glory Hole-Porfido and Cabrestante areas, but on wider drill spacing than the near-surface material.

Through its exploration drilling program, Aura Minerals has been successful in confirming and expanding upon the historical drilling, thus justifying the use of the associated data in its current 2015 Mineral Resource estimate.

The Aura Minerals drilling and Quality Assurance/Quality Control (QA/QC) programs, as well as the results from previous programs up to 2011, were previously reviewed by Micon, who concluded that the programs followed the 2010 CIM exploration best practices guidelines. A rigorous QA/QC program was conducted by Aura Minerals in November 2014 and reviewed by an independent QP, Farshid Ghazanfari P.Geo., who concluded that the new assay data contained in the March 2015 Resource followed the 2010 CIM exploration best practices guidelines. Additional QA/QC sampling of historical holes by the QP during 2017 also shows no significant bias and reaffirmed and further justified use of historical assays in resource estimate.

1.5 MINERAL RESOURCE ESTIMATE

The drill-hole database for the Mineral Resource estimate includes drilling and assaying up to February 7, 2014, the effective date of the database.

The drill holes used in the Mineral Resource estimate total 219,586 m of drilling in 1,336 drill holes. From this drilling, 87,971 samples of various lengths were collected and assayed for total copper, 76,875 samples were assayed for gold, and 80,545 samples were assayed for silver.

The block size selected for the model was 5.0 x 5.0 x 5.0 m. The search ellipsoids were oriented based on the local orientation of the geological interpretation and ranges of continuity obtained from the variography. The copper, gold, silver and arsenic grades were estimated using inverse distance squared estimation (ID2) and the 2.0 m assigned and capped composites.

The Mineral Resource estimate used a bulk density that was interpolated using ID2. A total of 3,442 density measurements recorded in the database have an average density of 2.88 t/m^3. Many sample intervals within the mineralized domains do not have specific gravity values, therefore assigned vales based on lithology types were applied to the missing intervals.

After 2011, additional arsenic, bismuth and antimony assays were added to the Mineral Resource model. However, the arsenic, bismuth and antimony are penalty elements that affect the value and saleability of the copper concentrate and, as such, their distribution within the deposit will influence mine planning and blending of the mined material in the processing facility. To assist the mining engineers in planning, the block model was updated to include the distribution of arsenic within the deposit. Arsenic grades are included in the current Mineral Resource model as well as their associated sales penalties as evidenced by the terms in Aura Minerals' recent offtake contracts.

The historical underground workings and current workings up to January 1, 2015, were removed from the resource model to account for the tonnage that had been mined to that date. Similarly, the mined topographic surface was updated to January 1, 2015, the approximate date that mining activities were suspended. Table 1-2 provides a breakdown of the sulphide only Mineral Resources by category.

Table 1-2 Mineral Resource Estimate (Sulphide Material Only)

Category	NSR Cut-off (US$/t)	Tonnes (,000s)	Cu (%)	Cu (,000s lbs)	Au (g/t)	Au (,000s oz)	Ag (g/t)	Ag (,000s oz)
Measured	45	3,923	1.71	147,823	1.05	133	17.84	2,250
Indicated	45	8,562	1.57	296,576	1.10	303	20.89	5,750
Measured and Indicated	**45**	**12,485**	**1.61**	**444,399**	**1.08**	**436**	**19.93**	**8,000**

Notes

1. The Mineral Resource estimates were prepared in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by the CIM Council on May 10, 2014, and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, adopted by CIM Council on November 23, 2003, using geostatistical and/or classical methods, plus economic and mining parameters appropriate to the deposit.
2. Mineral Resources stated at a cut-off of US$45/t NSR.
3. NSR values have been calculated using a long-term price forecast for copper (US$3.00/lb), gold (US$1,280/oz) and silver (US$18/oz)., resulting in the following formula: NSR ($/t) = (Cu% x US$39.76) + (Au g/t x US$20.95) + (Ag g/t x US$0.32).
4. NSR values are based on the proposed concentrate off take-terms dated September 2017 and the 2015 Technical Report metallurgical recoveries of 88% for copper, 59.4% for gold, 70.3% for silver and 80% for arsenic.
5. The figures only consider material classified as sulphide mineralization.
6. The figures may not add due to rounding of the numbers to reflect that they are estimates.
7. Mineral Resources are effective January 31, 2018.

The Mineral Resource estimate is based on a US$45/t NSR cut-off grade which would meet the requirements for potential economic extraction as defined by CIM standards and definitions for Mineral Resources. To meet the criteria of potential economic extraction, block model estimates were viewed in plan and section to ensure that all resources above the US$45/t NSR cut-off form a continuous mineralized zone.

The mineralization domains that underpin the Mineral Resource were created based on an NSR formula for copper, gold and silver that considered engineering and economic factors, as well as smelter and refining terms.

The narrowed NSR mineralization domains continue to follow geological continuity, lithological controls and structural orientation. The fixed NSR has decreased the number of tonnes available for mining compared to previous estimates, however, the newly constrained NSR wireframes did increase copper, gold and silver grades significantly.

No environmental, permitting, legal, title, taxation, socio-economic, marketing or political issues have been identified that would adversely affect the Mineral Resource estimates in Table 1-2.

1.6 MINERAL RESERVE ESTIMATE

The Mineral Reserve estimate presented in this Technical Report has been prepared in compliance with the "CIM Standards on Mineral Resources and Reserves – Definitions and Guidelines" as referred to in NI 43-101. The Qualified Person for the Mineral Reserve estimates section is Mr. Adam Wheeler, C. Eng.

Mineable stope shapes have been defined using an NSR value which has been calculated based on the metal prices, metallurgical recoveries and concentrate off-take terms. Stope designs are based on a break-even NSR cut-off of US$60/t ore which is calculated from the total mine operating cost (mining, processing and G&A). Stope shapes were generated using DataMine's Mine Shape Optimizer (MSO) which targeted only Measured and Indicated Mineral Resources. Final stope shapes and associated ore and waste development were designed using the Deswik CAD software. Dilution was applied in the form of planned and unplanned dilution from hanging wall and footwall end-wall along with backfill dilution where applicable. Total dilution is approximately 15%.

Table 1-3 Mineral Reserve Estimate (Sulphide Material Only)

Category	NSR Cut-off (US$/t)	Tonnes (,000s)	Cu (%)	Cu (,000s lbs)	Au (g/t)	Au (,000s oz)	Ag (g/t)	Ag (,000s oz)
Proven	60	1,872	1.70	69,973	1.08	64	18.3	1,100
Probable	60	2,770	1.74	106,439	1.23	110	19.9	1,771
Proven and Probable	**60**	**4,642**	**1.72**	**176,412**	**1.17**	**174**	**19.2**	**2,872**

Notes

1. The Mineral Reserve estimates were prepared in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by the CIM Council on May 10, 2014, and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, adopted by CIM Council on November 23, 2003, using geostatistical and/or classical methods, plus economic and mining parameters appropriate to the deposit.
2. Mineral Reserves are the economic portion of the Measured and Indicated Mineral Resources. Mineral Reserve estimates include mining dilution and mining recovery. Mining dilution and recovery factors vary with specific reserve sources and are influenced by several factors including deposit type, deposit shape and mining methods.
3. The NSR cut-off US$60/t is based on the total predicted operating cost.
4. NSR values have been calculated using a long-term price forecast for copper (US$3.00/lb), gold (US$1,280/oz) and silver (US$18/oz)., resulting in the following formula: NSR ($/t) = (Cu% x US$39.76) + (Au g/t x US$20.95) + (Ag g/t x US$0.32). NSR values are based on the proposed concentrate off take-terms (dated September 2017) and the 2015 Technical Report metallurgical recoveries of 88.0% for copper, 59.4% for gold, 70.3 % for silver and 80.0% for arsenic.
5. The stope designs targeted only Measured and Indicated Mineral Resources, but where Inferred Mineral Resources were included within mining shapes they were treated as waste with zero grade.
6. Stope dimensions were established by a geotechnical assessment performed by Call & Nicholas, Inc. in 2017.
7. Dilution was applied in the in the form of planned and unplanned dilution from hanging wall and footwall end-wall. Dilution from backfill (for secondary stopes) was also included. All dilution material was assumed at zero grades. Total dilution is approximately 15%.
8. Mining recoveries of 94% (i.e. 6% losses) and 99% (1% losses) were applied to the stopes and ore development sill cuts respectively.
9. Mineral Reserves are effective January 31, 2018.

1.7 MINE DESIGN

The Aranzazu underground mine is accessed by two portals which are near the processing plant. Main ramps, driven at 5.0 m high by 5.0 m wide are used to access the underground workings. For the re-start plan, two new ramps are designed to access the GHFW and GHHW zones. Ramps are designed at the same dimensions, with an average gradient of approximately 14.2%.

Mining will be carried out using the Long Hole Open Stope (LHOS) mining method to extract the ore. Stopes will be extracted in a transverse method (perpendicular to the strike of the orebody) using a primary/secondary stope configuration. In this mining method, haulage galleries are developed parallel to the ore zones on each production level as defined by the sub-level interval. Sub-level intervals are designed between 25 m to 30 m to minimize the amount of waste development on the sub-levels. While the majority of the stopes will be mined transverse, there are some areas that will allow the ore to be extracted in a longitudinal approach. Stopes are to be filled with cemented rockfill for primary and uncemented waste material for secondary stopes.

A geotechnical block model was developed from the drill hole geotechnical data which was used to determine the stope design widths, ground support recommendations, dilution estimates, pillar sizes and backfill strength. The model was developed to estimate the Geotechnical Material Type (GMT) both within the ore zone and the surrounding waste rock. In general, the rock quality of Aranzazu is considered fair to good, however there are some zones of poor to very poor rock mass quality, which are usually controlled by major fault zones, which have been accounted for in the mine design.

Mineral Reserves have been calculated using an NSR cut-off of US$60 per tonne. The cut-off included the estimated mining costs (both contractor and owner costs), processing costs, and general and administration costs. Costs for the contractor were based on quotations, obtained from reputable Mexican firms, for the expected development, stoping, haulage, and backfilling requirements. Aranzazu processing and G&A costs were based on historical operating cost (i.e. 2013 to 2014) adjusted for inflation and updated salary ranges.

Mineral Reserve tonnes and grades include estimates for both dilution and recovery. Dilution has been estimated to come from both planned and unplanned sources. The unplanned dilution was estimated as a function of the GMT of the stope hanging wall rock mass. In addition, secondary stopes include a factor for backfill dilution. All dilution is assumed to have zero grade. The overall dilution included in the Mineral Reserves is approximately 15%. Recovery (ore losses) has also been applied to the final Mineral Reserve tonnes. Stopes are assumed to have 94% recovery while ore development is assumed at 99% recovery.

The mine development and production schedule were developed using the Deswik® software package. The sequence for both development and production activities was developed and the appropriate rates and production targets were applied to achieve the required schedule of activities. During the first year, development will focus on the GHFW ramp and level development, as well as establishing access to the existing mining zone. Ore production will begin within the first three months from easily accessible areas; however, it will require a ramp up period of approximately 14 to 15 months to reach full production of 2,597 TPD.

Aura Minerals intends to use a mining contractor to do all the development and stoping, haulage of ore to the mill and placement of backfill. Aranzazu will provide the main services, such as ventilation, de-watering (pumping), compressed air and electrical reticulation along with technical services.

Ventilation will be provided by two 2.35 m diameter Axial Mine Fans. Fresh air will intake through the main portals and exhaust through ventilation raises. The pumping system is designed to handle 40 to 45 L/s of water. Water will be pumped in multiple stages from the bottom of the mine to a central clarifier sump. Clean water will be pumped to the surface storage ponds.

For the existing areas of the mine, electrical power will be provided from the existing sub-stations on surface feeding at 4.16 kV. A new sub-station will be installed to supply power to the new sections of the mine (GHFW and GHHW zones) at 13.6 kV. Compressed air and water will be supplied using the existing systems.

1.8 METALLURGY AND PROCESSING

Aura Minerals conducted a major test program during the fall of 2017 at ALS Laboratories in Kamloops B.C. in an attempt to find a reagent scheme which would provide a separation of enargite from the other copper bearing minerals and gold.

The test work involved first establishing that older core had not degraded over time and was thus suitable for inclusion in a composite sample for the main body of tests. There were a series of 13 rougher flotation tests and 16 cleaner flotation tests where grind size and various reagents were tested in an effort to find a suitable separation of enargite from the other copper bearing minerals and gold. A further 11 rougher flotation tests were carried out on the variability samples to determine if the finer grinding, identified as the best route to arsenic separation, was viable over the parts of the ore body represented by the variability samples and thus by interpolation in the mining plan. The programme is fully explained in Item 13 of this Technical Report.

The overall metal recoveries, estimated from the testwork results, are expected to be 88% for Copper, 69.9% for Gold, 70.0% for Silver and 83.0% for Arsenic over the life of the operation.

The plant is expected to treat an average of 2,597 TPD of ore and utilise conventional processing steps, crushing, grinding, flotation and dewatering. The plant will use existing equipment that is already installed. The one major change to the grinding circuit configuration will be the inclusion of the regrind mill as a primary grinding mill. This will enable the required tonnage to be ground to the finer flotation feed size. The regrinding mill will be fed with product from two of the primary mill discharges to achieve this.

The flotation circuit will remain largely unchanged. The only flowsheet change will be to enable the concentrate from the second of the four banks of cells to be directed either to final concentrate or to scavenger concentrate depending on concentrate grade.

Dewatering will be achieved using the existing thickener and pressure filter, which will have plates added, to accommodate the extra tonnage of concentrate expected.

The process control aspects of the plant will be upgraded to allow a more modern approach to controlling both the grinding and flotation circuits. An Online Stream Analyser (OSA) will be installed to aid the operators in controlling the flotation circuit effectively. This is described more fully in Item 17 of this Technical Report.

1.9 INFRASTRUCTURE

Conventional slurry tailings will be disposed of at a new tailings storage facility designated TD5 which is scheduled for construction in Q3 2018. TD5 Stage 1 is designed to store conventional flotation tailings slurry, which will be deposited at a rate of 2,597 TPD. This slurry will be pumped from the process plant to TD5 (distance of approximately 4.0 km). TD5 will be constructed with two zoned-earth-fill tailings dams ("primary" and "south" dams) at the eastern side of TD5. The TD5 tailings dams will be constructed by annual construction stages for the first three years of operation using downstream construction methodology.

Each stage (1A, 1B and 1C) will provide approximately one year of tailings storage. The tailings storage facility design is based on SEMARNAT regulations and Canadian Dam Association (CDA) guidelines. A conceptual level design was completed for expansions to the TD5 tailings dam to provide a total of 10.1 Mt of tailings storage. The expansion will be completed by sequential downstream raises to the tailings dam.

There are three existing tailings storage facilities: Tailings Dam No. 4 (TD4) which, with buttress construction currently underway, has an available storage capacity of 259,500 dry metric tonnes ("Dmt") and the old Tailings Dam No. 1 and No. 2 (TD1 and TD2) offers an additional short-term capacity of 306,000 Dmt of tailings which equates to a total storage capacity of 565,500 Dmt. This additional storage capacity is equivalent to around 0.6 years of full production. Aura Minerals' current plan is to build the new tailings storage facility, currently licensed, to the east of the current operation, referred to as TD5.

There is currently sufficient power to operate the mine and processing facilities, but a dedicated, 6.0 km, 34.5 kV line from the national power company is planned to be built and connected to the mine. This power line, tailings dam construction, cemented rock fill plant, and sustaining capital for both the plant and mobile equipment are all part of the capital expenditure during the early years of mine operation. All other site infrastructure remains available from the previous operating period and functional to support the project start-up.

1.10 ENVIRONMENT

Aranzazu is considered a brownfield site and mining of the existing deposits has been carried out in several campaigns since 1962, with mining activity in the district documented as early as 1548. The Project is favorably situated in a semi-arid climate with net evaporation, and is not located within any protected natural areas, priority terrestrial regions or areas of importance for wildlife conservation.

Most permits for the Aranzazu operation are either still valid from the mine's last operating period or require only minor administrative processes to re-activate. Existing water concessions from Aranzazu's wells allow withdrawal of up to 1,081,495 m^3/year, and along with mine dewatering contributions are sufficient for the Project's water needs. There are no discharges from the processing circuit planned.

The Project restart is not expected to significantly alter the local socioeconomic conditions that existed at operating levels in 2014. Direct employment is similar to 2014 levels and is not expected to increase dramatically as a result of the Project. The Project plan considers two environmental supervisors and one community relations liaison reporting to a Security, Health and Environmental Superintendent.

Mining, processing and support operations for the Project will operate within the existing infrastructure footprint. A new tailings storage facility will be constructed for the Project, Tailings Disposal No. 5 (TD5). This facility has undergone design improvements since it was first permitted in 2014. The updated design incorporates downstream dam construction methodology as well as zoned earth fill embankments with internal drainage to control the phreatic surface in the embankment and enhance stability. At closure, ponding and saturation will be minimized through grading and construction of a closure spillway to route storm water runoff from the cover system, and by maintaining surface water diversion channels. Geochemical testing campaigns in 2010 and 2017 indicate that most tailings from historic tailings facilities contain sufficient calcite and low sulphide mineral content such that production of net acidity is improbable, although tailings containing lower amounts of calcite (generally from intrusive-based ore) may generate localized acidity if deposited in isolation. All tailings tested in 2010 to the Mexico tailings standard were well below maximum permissible levels for metal leaching of waste materials. Testing in 2017 yielded similar results. However, it was noted that leachate concentrations of certain metals exceed the much lower U.S. Environmental Protection Agencies (EPA) Maximum Contaminant Level (MCL) – this is the legal threshold limit on the amount of a substance that is allowed in public water systems under the Safe Drinking Water Act. Nonetheless, tailings dam seepage will be collected and routed to geomembrane-lined seepage collection ponds and recycled back to the process plant and monitored to assess leachability under site conditions.

The TD5 Operations, Maintenance and Surveillance (OMS) Manual and Emergency Action Plan (EAP) will be developed by the Engineer of Record.

Approval of both the design update and the associated change of land use authorization for Stage 1 of TD5 are expected before the end of August 2018. Aura Minerals will be required to compile and submit design and environmental assessment documentation for the later stages of TD5 and obtain associated approvals and change of land use authorizations in order to provide sufficient tailings capacity for the Project beyond Year 3. Three existing tailings storage facilities (TD4, TD1, and TD2) offer additional storage capacity of up to 565,500 Dmt.

No new waste rock storage facilities will be required for the Project; moreover, there is potential of reducing the volume of existing waste rock piles by using the waste rock for stope backfill and for tailings dam construction. Geochemical testing campaigns in 2010 and 2017 indicate that waste rock is unlikely to be acid-generating. The material is considered suitable for structural fill, though having potential for solubilizing of some metals on contact with water.

Aura Minerals acquired 100% of the Aranzazu mine in June 2008 and with this transaction acquired ownership and responsibility for older workings including abandoned shafts, the north waste rock pile, an abandoned oxide leach site, water pumping and conveyance systems, and a series of smaller tailings impoundments (TD1 through TD4, and historic TD5). The Project cost model assumes US$6.5 M for site closure (including both existing workings and the Project to be constructed). No other environmental, regulatory, social or community factors were identified as having potential to materially affect the construction, operation and decommissioning of the Project.

1.11 OPERATIONAL COSTS (OPEX)

Table 1-4 shows the operational costs for Aranzazu estimated at US$57.66/t.

The mine will be fully contracted and managed by a small owner's management and technical services team. The underground contractor will provide equipment and operators for development and stope production. All mine consumables will be sourced directly by Aranzazu.

The processing plant considers a full workforce including plant operations, metallurgy and technical services, maintenance and safety. All costs related to consumables have been updated with new quotes from registered suppliers. General and Administrative (G&A) includes labor, services, insurance and also, the costs associated with the sale of the concentrate including the transportation to Port of Manzanillo.

Table 1-4 Estimated LOM Operational Costs

Category	Cost (US$/t)	Total LOM Cost (US$M)	Comments
Mining			
Contractor Mining	$34.74	$161.3	Direct mining costs
Owner's costs	$4.11	$19.1	Operations and technical support, power, explosives
Total Mining	**$38.86**	**$180.4**	
Total Processing	$10.91	$50.6	
General and Admin.	$6.78	$31.5	Site management, fees, administration,
Total Operating Cost	**$56.54**	**$262.5**	
Royalties	$1.11	$5.2	Landowner royalties
Total	**$57.66**	**$267.6**	**Total Operating Cost**

The operation will employ around 165 direct employees and another 150 indirect employees, and it is expected that the majority of the workforce will be local. For updated salary and benefits, Aura Minerals considered the latest salary survey provided by CAMIMEX in 2017 which outlines benchmark salaries in the Mexican Mining Sector.

1.12 CAPITAL COSTS (CAPEX)

Table 1-5 outlines the total capital expenditures required for the Project including underground mine development, tailing storage, plant refurbishment, infrastructure, closure costs and contingency are US$92.5M over the life of the mine.

Pre-production capital for initial ramp development, tailings dam construction and plant refurbishment and start-up costs are US$32.1M in the first year. Although there is mill production in the second half of Year 1, the mine is expected to reach commercial production in the first quarter of Year 2.

The LOM sustaining capital for ongoing mine development, additional tailings storage, mine equipment, plant upgrades, exploration drilling, and mine closure is US$60.4M.

Table 1-5 Total Capital Expenditure (i.e. Initial and Sustaining)

Capital Item	**Initial Capital (US$M)**	**Sustaining Capital (US$M)**
Pre-Production	$5.8	-
Underground Development	$12.2	$33.1
Tailings Dam	$6.9	$7.1
Mine Equipment	$2.2	$3.3
Plant	$2.0	$5.1
Powerline	$1.2	-
Exploration / Delineation Drilling	$0.5	$3.5
Sub-Total	**$30.8**	**$52.0**
Contingency (5%)	$1.3	$1.9
Closure Cost	$0.0	$6.5
Total	**$32.1**	**$60.4**

1.13 FINANCIAL EVALUATION

Table 1-6 shows the metal prices used in the study which were based on long-term forecasted prices for copper, gold and silver from a leading Canadian Schedule I Bank.

Table 1-6 Summary Metal Prices

Commodity Price	**Year 1**	**Year 2**	**Year 3 Onwards**	**LOM Average**
Copper (US$/lb)	$2.90	$2.95	$3.10	$3.06
Gold (US$/oz)	$1,250	$1,299	$1,301	$1,297
Silver (US$/oz)	$18.23	$19.47	$19.83	$19.62

Foreign Exchange rate was considered at 18.0:1.00 (MXN:USD) according to projections provided by two leading Canadian banks.

The financial evaluation considers an outstanding debt of US$6.5M with suppliers and contractors who worked with Aranzazu before the 2015 shutdown. This outstanding debt requires payment over a three-year period starting two months after commercial production is reached. The debt is to be paid to each creditor in 36 equal monthly payments, with full payment no later than April 2023.

Table 1-7 outlines the total cash operating cost before precious metal credits for the Project at US$389.4M or US$2.51/lb Cu (including treatment and transportation charges and royalties). The reportable cash cost after credits is US$220.3M or US$1.42/lb Cu. The All-in-Sustaining Cost is US$1.81/lb Cu.

Table 1-7 Total Cash Operating Costs Summary

LOM Total Cost Breakdown	US$M	US$/lb Cu
Smelting, Refining, Treatment and Freight*	$121.7	$0.78
Cash Operating Costs	$262.5	$1.69
Royalties	$5.2	$0.03
Reportable Cash Costs	$389.4	$2.51
Credit: Gold Revenue	-$139.3	-$0.90
Credit: Silver Revenue	-$29.8	-$0.19
Reportable Cash Costs after precious metals credits	$220.3	$1.42
Copper Produced (M lbs.)	155.2	
Total Cash Costs (payable Cu)	-	**$2.51**
Total Cash Costs (payable Cu) After Credits	-	**$1.42**
Add: Sustaining Capital**	$60.4	$0.39
Total Costs incl. Sustaining Capital	$280.7	$1.81
***Includes Royalties, contingency, all sustaining capital after Year 1 and closure costs*		
All-in Sustaining Total Cash Costs**	-	**$1.81**

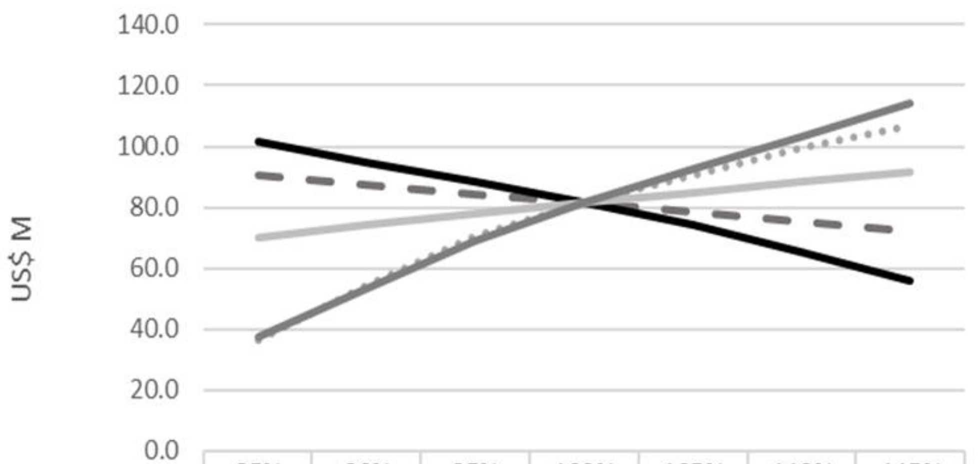
The after-tax NPV at 5.0% discount is US$81.4M and an IRR of 136.7%. The Project will produce a cash flow of US$100.6M with a payback of the initial capital in 22 months from start of production. The following Table 1-8 summarizes the overall economics of the Project:

Table 1-8 Project Economics Summary

Project Summary	**Units**	
Total Throughput	t	4,641,775
Mine Life	years	5.5
Recovered Metal in Conc.		
Cu production	t	70,416
Cu production	M lbs	155.2
Au production	oz	122,149
Ag production	oz	2,010,105
Concentrate Produced	t	306,156
Cu Concentrate Grade	%	23%
OPEX		
Total Cash Operating Costs*	US$M	$220.3
Total Cash Operating Costs*	US$/lb Cu	$1.42
**After gold, silver credits*		
AISC**	US$M	$280.7
AISC**	US$/lb Cu	$1.81
***Includes sustaining capital and closure costs*		
CAPEX		
Initial Capital	US$M	$32.1
Sustaining Capital	US$M	$60.4
Total Capital (incl. Closure)	US$M	$92.5
Financial Summary		
Total Revenue	US$M	$620.8
Net Smelter Return (NSR)	US$M	$499.0
LOM cash flow	US$M	$100.6
NPV (5.0%)	US$M	$81.4
IRR	%	137%
Payback	Months	22.0

Project Sensitivity is shown in Figure 1.1 evaluated at +/-15% range for copper and gold prices, capital and operating costs, and currency exchange. The Project is shown to be most sensitive to changes in the copper price, exchange rate and operating cost. The Project value is less sensitive to gold price and capital cost.

Figure 1.1 Project Sensitivity Results



	85%	90%	95%	100%	105%	110%	115%
Cap Ex	90.6	87.5	84.4	81.4	78.3	75.2	72.1
Op Costs	101.8	95.0	88.2	81.4	74.5	65.4	55.7
Exch Rate	36.2	54.2	70.3	81.4	90.8	99.3	107.1
Gold Price	70.1	74.4	77.9	81.4	84.9	88.3	91.8
Copper Price	37.7	53.4	69.1	81.4	92.4	103.4	114.3

1.14 CONCLUSIONS AND RECOMMENDATIONS

1.14.1 Conclusions

- The Project demonstrates economic viability with an NPV and IRR of US$81.4M and 136.7% respectively, based on the following metal prices: US$3.06/lb Cu, US$1,297/oz Au and US$19/oz Ag at FOREX of 18:1.0 MXN:US$.
- As demonstrated above, the Project exhibits attractive economics using base case price assumptions. In addition, the Project economics are insulated somewhat from any modest downward pressures in metal prices, in particular, the copper price, due to the modest capital expenditures (US$32.1M in Year 1) required to restart the Project in attaining commercial production. In addition, any further weakening of the MXN:USD exchange rate would also be beneficial on the Project's various metrics, in particular, on an NPV basis, as the expected increase in revenue from selling in US$ over LOM would outweigh the capital sensitivity in the first year.
- Based on the entirety of the Project's analysis, resuming mining operations is recommended.
- The revised Mineral Resource wireframe is more selective, and targets average higher grades compared to the previous model used by the operation in 2014.
- The new mine design improves the project economics by increasing the sub-level interval to 25 m to 30 m where possible in order to minimize development meters and reduce capital costs.

- The metallurgical program has provided further positive results in regards of metal recoveries at higher grades as well as an enhanced understanding of the arsenic distribution and its treatment at the plant level.
- Previous studies have concluded that the mineral concessions forming the Aranzazu Property has the potential for the discovery of further zones of copper-gold mineralization of similar character and grade as those exploited in the past.
- This FS has benefited greatly from the existing Aranzazu database knowledge (i.e. consumables, unit costs, etc.) as well as the years of operating experience gained by the local workforce; all this in combination with key consultants and experts in the different areas of expertise.

1.14.2 Recommendations

- The positive outcomes from the FS support the re-start of the operation with the new mining plan and processing modifications.
- The Aranzazu Mine and Property shows potential for further exploration to extend mine life.
- Further metallurgical test work is required to deal with arsenic levels in ore for Years 4 and 5; it is envisioned two potential ways of dealing with high arsenic levels: (i) by blending ore coming from these high arsenic areas with low arsenic ores and (ii) pursue a hydrometallurgical control, which is the least desirable.
- A detailed mine closure plan will be required in the next years.

The estimated cost of these recommendations is approximately US$1.0M to US$1.5M including a 10% contingency.

2. INTRODUCTION

2.1 THE ISSUER

Aura Minerals Inc. (Aura Minerals) is incorporated in the British Virgin Islands (BVI) and is a mid-tier gold and copper production company focused on the development and operation of gold and base metal projects in the Americas.

The Company's producing mines include the San Andres gold mine in Honduras and the Ernesto/Pau-a-Pique gold mine in Brazil. The company has conducted an exploration program at its Sao Francisco gold mine in Brazil to determine if a re-start of the mine is feasible. In addition, the Company has two additional gold projects in Brazil, Almas and Matupa, and one gold project in Colombia, Tolda Fria.

The common shares of Aura Minerals are traded in the Toronto Stock Exchange (TSX) under the symbol “ORA”.

The subject of this report is a Feasibility Study (FS) for the re-start of the Aranzazu Mine located in the State of Zacatecas, Mexico.

2.2 TERMS OF REFERENCE

Aura Minerals commissioned several consultants to work on different elements of the Technical Report. These consultants are listed below:

- Farshid Ghazanfari Consultancy - Geology
- Adam Wheeler, Robert Dowdell (Dowdell Mining Ltd.) – Mining, Capex, Opex, Economic Analysis
- Paul Cicchini (Call & Nicholas, Inc.) – Geotechnical Underground
- Graham Holmes (Jacobs Engineering) – Metallurgy and Processing
- Cam Scott (SRK) – Tailings Dam No. 4
- Brett Byler (Wood PLC) – Tailings Dam No. 5
- Diane Lister (Altura Environmental Consulting) – Environmental and Permitting
- Paul O’Brien (Anthem Capital) – Financial Modelling and Valuation

The study necessary to complete this FS was primarily performed during the period January 2017 to January 2018 and incorporated Qualified Person expertise from all of the consulting companies. The Technical Report Authors and co-Authors worked closely with Aura Minerals personnel during the course of the study.

The economic model prepared for the FS is based on metal prices and off-site cost assumptions advised by Aura Minerals. Each individual QP reviewed these prices and assumptions and are satisfied as to their reasonableness. The accuracy of the estimates and the inputs to the project financials are consistent with the level of study.

2.3 SOURCES OF INFORMATION

A full list of references is included in Item 27. Key sources of information used in preparation of this Technical Report include:

- The Aranzazu Property resource estimate and block model, prepared by Farshid Ghazanfari P.Geo., dated January 31, 2018
- Geotechnical modelling and other geotechnical data prepared by Call & Nicholas, Inc., which was reviewed by Adam Wheeler, Robert Dowdell and Colin Connors
- Life of Mine Plan and mining design developed by Adam Wheeler, Robert Dowdell and Colin Connors
- Aranzazu mine and concentrator key performance indicators (KPIs) and monthly operational reports
- A level increase for the existing tailings storage study prepared by SRK
- New tailings disposal facility TD5 detailed design prepared by Wood PLC (Formerly Amec Foster Wheeler)
- Metallurgical support and processing guidance provided by Jacobs Engineering
- Environmental points and other associated matters reviewed by Altura Environmental Consultants

2.4 QUALIFIED PERSONS

The names and details of the Qualified Persons who have prepared this Technical Report, or who assisted the Qualified Persons, are listed in Table 2.1. The Qualified Persons meet the requirements of independence as defined in NI 43-101 other than Mr. F. Cornejo who is the VP Projects of Aura Minerals and C. Connors who is the Director, Mining of Aura Minerals.

Table 2-1 Persons Who Prepared or Contributed to this Technical Report

Qualified Person	Position	Employer	Independent of Aura Minerals	Date of Last Site Visit	Prof. Designation	Items of Technical Report
Mr. F. Ghazanfari	Principal	Farshid Ghazanfari Consultants	Yes	January 2018	P.Geo.	Items, 7, 8, 9,10, 11, 12, 14, and parts of 25, 26
Mr. A. Wheeler	Principal Mining Engineer	Independent Consultant	Yes	September 2017	Chartered Engineer	Item 15, 16, 21, 22, and parts of 1, 25, 26
Dr. Robert Dowdell	Principal Mining Engineer	Dowdell Mining Limited	Yes	May 2017	Chartered Engineer	Item 15, 16, 21, 22, and parts of 1, 25, 26 (Co-Author)
Mr. Colin Connors	Director, Mining Aura	Aura Minerals	No	May 2018	RM-SME	Item 15, 16, 21, 22, and parts of 1, 25, 26 (Co-Author)
Mr. Paul Cicchini	President	Call & Nicholas, Inc.	Yes	March 2017	P.E.	Item 16 – Geotechnical Aspects
Mr. G. Holmes	Principal	Jacobs Engineering	Yes	September 2017	P.Eng.	Items 13, 17 and parts of 26
Mr. C. Scott	Principal	SRK Canada	Yes	May 2017	P.Eng.	Item 18 – all related to TD4, parts of 1, 25 and 26.
Mr. B. Byler	Senior Associate	Wood PLC (Formerly Amec Foster Wheeler)	Yes	June 2018	P.E.	Item 18.1, 18.3, 25.2.4.2, 25.2.4.2, 26.5.1, 26.5.3
Ms. D. Lister	Principal	Altura Environmental Consulting	Yes	August 2017	P.Eng.	Items 4, 5, 6, and 20 and parts of 1, 25, 26
Mr. F. Cornejo	VP Projects	Aura Minerals	No	January 2018	P.Eng.	Items 1, 2, 3, 27, 28 and parts of 18, 21, 22, 25, 26
Other Experts who assisted the Qualified Persons*						
Expert	**Position**	**Employer**	**Independent of Aura Minerals**	**Visited Site**	**Items of Technical Report**	
Mr. R. Barbosa	CEO	Aura Minerals	No	January 2018	Items 1, 19, 21, 22, 25, 26	
Mr. R. Goodman	VP Legal Affairs	Aura Minerals	No	January 2017	Items 1, 4, 19, 21, 22, 25, 26	
Mr. Paul O'Brien	Principal	Anthem Capital Group Inc.	Yes	January 2018	Items 22 and parts of 1 and 26	
Mr. Andrew Falls	Principal	Exen Consulting Services	Yes	NA	Item 19	
Mr. Juan Pablo Duenas	Aranzazu Finance Manager	Aura Minerals	No	January 2018	Parts of 4, 5, 6, 21, 22, 25,26	
Mr. E. Allen, Ms. J. Adelman	Consultants	Core Geoscience Services Inc.	Yes	No	Parts of 20	

* *In addition, several employees of Aura Minerals Inc. provided information and data to the qualified persons*

2.5 LIST OF ABBREVIATIONS AND ACRONYMS

AA finish – Atomic Absorption finish

ALTN - alteration

BVI – British Virgin Islands

CAMIMEX – Camara Minera de Mexico

CAPEX – Capital Costs

CIM – Canadian Institute of Mining and Metallurgy

CNA – Comision Nacional de Agua

CONABIO – National Commission for the Knowledge and Use of Biodiversity

CONAGUA – National Water Commission

CNI – Call & Nicholas, Inc.

CRF – Cemented Rock Fill

CSS – Closed Side Settings

CV – Coefficient of Variation

DWT – Dead Weight Tonnage

DAP – Delivered at Port

EAP – Economically Active Population

EBITDA – Earnings before Interest, Taxes, Depreciation, Amortization

ELOS – Equivalent Length of Slough

EPA – Environmental Protection Agencies (U.S.)

FMEA – Failure Modes and Effects Analysis

FOREX – Foreign Exchange

FOS – Factor of Safety

G&A – General and Administrative

GMT – Geomechanical Material Type

GHFW – Glory Hole Foot Wall

GHFHW – Glory Hole Hanging Wall

HDPE – High Density Polyethylene

ID^2 - Inverse Distance Squared

IDW – Inverse Distance Weighting

IMSS – Mexican Social Security Institute

INAFED – National Institute for Federalism and Municipal Development

INEGI – National Institute of Statistics, Geography and Informatics

IRR – Internal Rate of Return

IRS – Intact Rock Strength

KPI – Key Performance Indicators

LAN – National Water Law

LAU – Licencia Ambiental Unica

LGDFS – General Law on Sustainable Forest Development

LGEEPA – General Law of Ecological Balance and Environment Protection

LGVS – General Law on Wildlife

LHD – Load Haul Dump

LITHD – Lithology

LME – London Metals Exchange

LHOS – Long Hole Open Stope

LOM – Life of Mine

LPGIR – General Law for the Prevention and Integral Management of Wastes

MCL – Maximum Contiminent Level

MIA – Manifestacion de Impacto Ambiental (Environmental Assessment)

ML – Mining Law

MSO – Mine Shape Optimizer

MXN – Mexican Pesos

NN – Nearest Neighbour

NOM – Norma Oficial Mexicana – a technically-based enforceable standard

NPV – Net Present Value

NSR – Net Smelter Return

OMS – Operations, Maintenance and Surveillance

OPEX – Operational Costs

OSA – Online Stream Analyzer

OXIDE – weathering surface

PEA – Preliminary Economic Assessment

PGA – Peak Ground Acceleration

PLC – Programmable Logic Controller

PROFEPA – Procuraduría Federal de Proteccion al Ambiente - Federal Environmental Enforcement Agency

Q – Rock Mass Quality

QA – Quality Assurance

QC – Quality Control

QP – Qualified Person

RC – Reverse Circulation

RMEIA – Regulation for the Environment Impact Assessment

RMR – Rock Mass Rating

RQD – Rock Quality Designation

SEMARNAT – Secretaria de Medio Ambiente y Recursos Naturales - Secretariat of Environmental and National Resources

SMN – National Weather Service

SRM – Standard Reference Material

SSN – National Seismological Service

TCR – Total Core Recovery

TD – Tailings Disposal

TSX – Toronto Stock Exchange

USCS – Unified Soil Classification System

UG – Underground

UGWK% - Depletion Solids

UTM – Universal Transverse Mercator

2.6 UNITS OF MEASUREMENT

Unit	**Abbreviation**
American Dollar	US$
Canadian Dollar	CAD$
Capital Expenditure	CAPEX
Centigrade	℃
Centimetre	cm
Cubic metre	m^3
Day	d
Dead weight ton (imperial ton – long ton)	Dwt
Density	t/m3/g/cm3
Dry metric tonne	Dmt
Foot/feet	Ft
Gram	g
Gram/litre	g/l
Gram/tonne	g/t
Hectare	ha
Hour	H
Horse Power	HP
Kilogram	kg
Kilogram per tonne	kg/t
Kilometre	km
Kilo tonne	Kt
Kilopascal	kPa
Kilovolt	kV
Kilovolt amp	kVA
Kilowatt	KW
Kilowatt hour	kWh
Litre	L
Litre per second	L/s
metres above sea level	masl
Megawatt	MW
Mega Volt Ampere	MVa
Metre	m
Metre per hour	m/h
Metre per second	m/s
Metric tonne	t
Metric tonne per day	tpd
Metric tonne per hour	t/h
Micron	µm
Milligram	mg
Milligram per litre	mg/L
Millimetre	mm
Million	M
Million tonnes	Mt
Million tonnes per annum	Mtpa
Part per million	ppm
Percent	%

Second	s
Short ton	st
Square metres	m^2
Tonnes per day	tpd
Troy ounce	oz
Wet metric tonne	wmt
Work Index	Wi
Year	yr

3. RELIANCE ON OTHER EXPERTS

In regards to income tax payable on earnings within the economic model, which supports the key financial results reported in this Technical Report, the operating costs reported in Item 21 and the economic analysis in Item 22, the Authors have relied on information provided by Mr. Juan Pablo Duenas, Finance Manager of Aranzazu in Mexico.

In regard to legal matters, such as acquisition agreements, status of mineral titles, ownership matters, etc., as set out in Item 4 of this Technical Report, Aura Minerals has relied on information provided by Mr. Ryan Goodman, VP Legal Affairs of Aura Minerals. Aura Minerals has advised that the exploitation licences for the Aranzazu Property have been reviewed and found to be in order and that it has obtained legal opinions as to the validity of the mineral titles claimed.

In regard to Market Studies and Contracts, as set out in Item 19 of this Technical Report, Aura Minerals has relied entirely on Mr. Andrew Falls who is the Principal at Exen Consulting Services, specialized company in metal trading out of Toronto, ON, Canada.

In regard to power line construction (Item 18), Aura Minerals has relied on the experience and advise of Mr. Raul Botello who is an experienced Energy Consultant in Mexico, located in Zacatecas.

In regard to environmental aspects of this Technical Report (Item 20), Aura Minerals has relied on the information presented by Mr. Mario Aldaco, former HSE Manager of Aranzazu in Mexico.

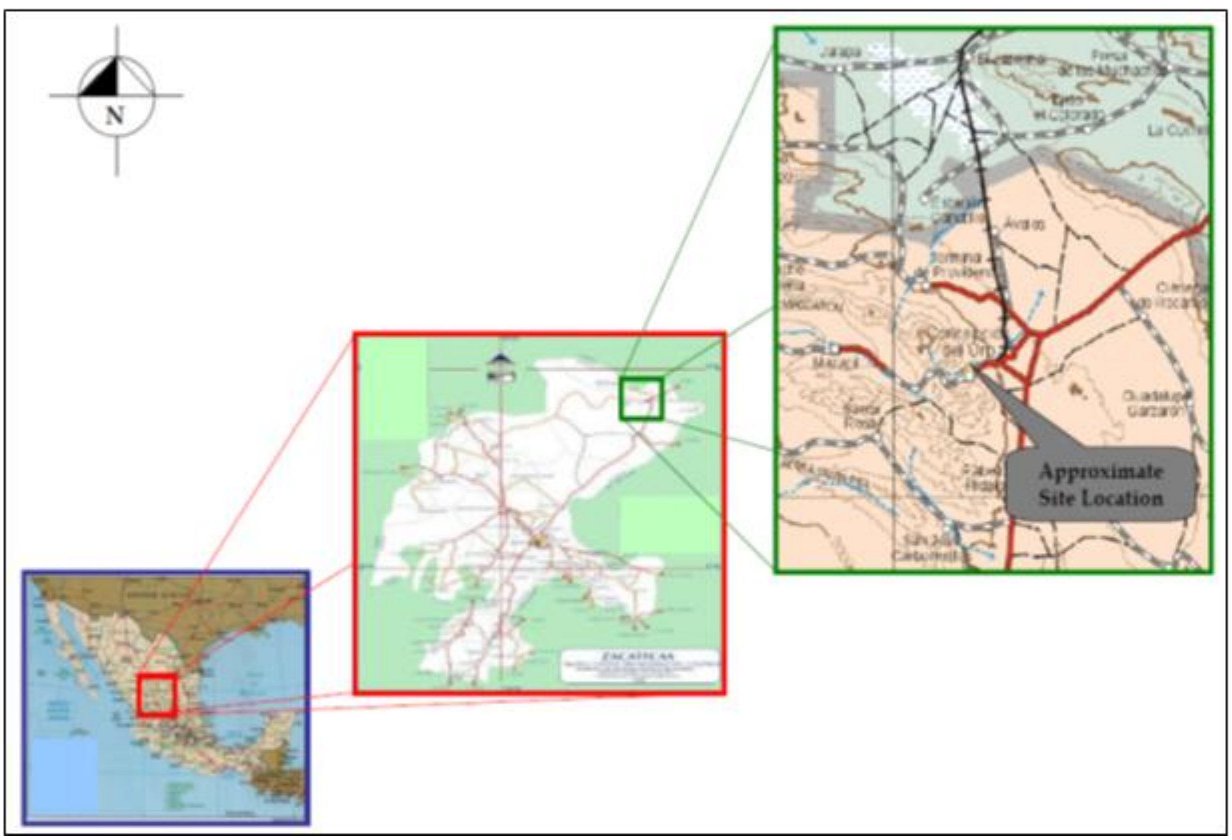
4. PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Aranzazu Mine is located at the western limits of the Municipality of Concepcion del Oro, approximately 250 km northeast of the city of Zacatecas, which is the state capital. The town of Concepcion del Oro also gives its name to the mining district which surrounds it. The Project lies within a rugged mountainous area with the higher points reaching 3,300 masl. The Project is accessed from either the city of Zacatecas, located 250 km to the southwest, or from the city of Saltillo, located 112 km to the northeast in the State of Coahuila.

The Arroyos Azules open pit and mill facilities are located at approximate UTM coordinates of 253,600 East and 2,724,150 North in zone 14 WGS 84, or 101°26' West longitude and 24°37' North latitude, and are located on the INEGI Concepcion del Oro G-14C-62, 50,000 scale map. Figure 4.1 shows the overall Project location.

Figure 4.1 Overall Project Location



4.2 OWNERSHIP AND PROPERTY DESCRIPTION

4.2.1 Ownership, Rights and Encumbrances

Aura Minerals has a 100% effective control over the Aranzazu Property through its wholly-owned subsidiary Newington Corporation S.L (Newington) and its wholly-owned Mexican subsidiary, Aranzazu Holding S.A. de C.V. (Aranzazu Holding).

Pursuant to the definitive acquisition agreement dated June 3 2008, Aura Minerals purchased the Newington shares from Clapham Luxembourg S. A. L. (Clapham) through the payment to Clapham of US$57.5M in cash and the issuance of 9,295,117 common shares of Aura Minerals then valued at US$12.5M.

Prior to Aura Minerals' purchase of the Aranzazu Property, Macocozac S.A. de C.V. (Macocozac) controlled the surface rights covering all concession areas and owned 100% of the Property. Under Macocozac, the Property was not subject to any royalties, back-in rights, or other encumbrances. Pursuant to an agreement dated May 7, 2008, Macocozac transferred its rights to the Aranzazu Property to Aranzazu Holding in exchange for a 1% Net Smelter Returns royalty (NSR) on the copper production when, during any calendar month, the monthly average copper price as quoted by the London Metals Exchange (LME) equals or exceeds US$2.00/lb.

The previous operator acquired the right to explore and develop the Aranzazu Mine by signing an option agreement with the previous owner, Macocozac, on December 15, 2006. This agreement expired on March 31, 2008, with no interest having been earned.

As stated in ***Aura Minerals Consolidated Financial Statements for the years ended December 31, 2015 and 2014,*** Aranzazu Holding filed for administrative proceedings under the Mexican Commercial Bankruptcy Law in 2015. On December 16, 2016 the Second District Court of Coahuila issued a resolution approving the agreement reached with the required majority of its creditors, the 'Convenio de Reestructura y Pago a Acreedores' (the "Convenio")[1]. The Court ruled that the agreements per the Convenio are final for the subscribing creditors and establishes payment methods and payment schedules to those creditors. Aranzazu Holding is to commence payment to creditors two months after receipt of payment for the first concentrate shipment that may be any time between April 2018 and no later than April 2019. The debt is to be paid to each creditor in 36 equal monthly payments, with full payment by no later than April 2023. A default in payment may be court-enforced. For further details, please see Items 21 and 22, Capital and Operation Costs, and Economic Analysis.

On March 8, 2018 Aranzazu Holding entered into a US$20M loan facility (the "Facility") and an off-take agreement (the "Off-Take Agreement") with Louis Dreyfus Company Metals ("LDC Metals") for the re-start of operations and copper concentrates to be produced from its Aranzazu. The Facility is guaranteed by Aura Minerals and its interests in Aranzazu and the San Andres mine pursuant to a pledge agreement of the shares of Aranzazu Holding and Aura's Honduran subsidiary. The Off-Take Agreement covers 100% of the copper copper concentrates to be produced from Aranzazu. For further details, please see Items 21 and 22, Capital and Operation Costs, and Economic Analysis.

To the extent known, the Aranzazu Property is not subject to any other royalties, back-in rights, or other encumbrances.

4.2.2 Mineral and Surface Tenure

Prior to December 21, 2005, exploration concessions were granted for a period of six years in Mexico and could be converted to exploitation concessions thereafter. However, as of December 21 2005 (by means of an amendment made on April 28 2005 to the Mexican mining law), there is now only one type of mining concession. Therefore, as of that date, there is no distinction between exploration and exploitation concessions on all new titles granted. All mineral concessions are now granted for a 50-year period and are extendable provided that the application is made within the five-year period prior to the expiry of the concession and that the concessions are kept in good standing. For the concessions to remain in good standing, a biannual fee must be paid to the Mexican government and a report must be filed in May of each year that covers the work performed on the Property between January and December of the preceding year.

Ownership or possession over the land surface of mining concessions are separately endowed rights; when the concession holder does not have surface rights to access the lands where the mining concession is located, the holder can directly negotiate the use of land for mining activities with the owners of the surface rights. In the case that no agreements are reached for the use of the surface, mining concessionaries are entitled to start a procedure contemplated in the Mining Law to obtain the following:

- an expropriation
- a temporary occupation; or
- an easement

Aura Minerals notes that it controls surface rights covering all mineral concession areas, rights that were transferred to Aranzazu Holdings during the mine’s purchase from the vendor Macocozac S.A. de C.V. The extent and continuity of surface rights were not independently verified during the review for this Technical Report.

A number of the mineral concessions which form the Aranzazu Property were established prior to the current mineral concession staking regulations and consist of irregular shapes and orientations. See Figure 4.2 for a Mineral Concession map of the Aranzazu Property and Table 4-1 for relevant information regarding the individual exploitation concessions. The 38 mineral concessions are mostly contiguous and vary in size, for a total property area of approximately 11,182 ha. The extent and continuity of mineral rights were not independently re-verified during the review for this Technical Report. Mining concession duties are paid semi-annually and the yearly total for 2018 is approximately 2,303,490 Mexican pesos (MXN), which is equivalent to approximately US$128,000 at an exchange rate of 18:1.00 MXN:US$.

One potential and ongoing issue with surface rights is that squatters have constructed homes in some areas near the edges of the town on the mineral concessions. Within the town, some portions of the water supply pipeline serving the mine were built over during the decades prior to acquisition by Aura Minerals. Should the mine require access to or direct use of these lands in the future, they may be obligated to lease or purchase the surface rights to these areas.

Figure 4.2 Aranzazu Property Mineral Concessions Map

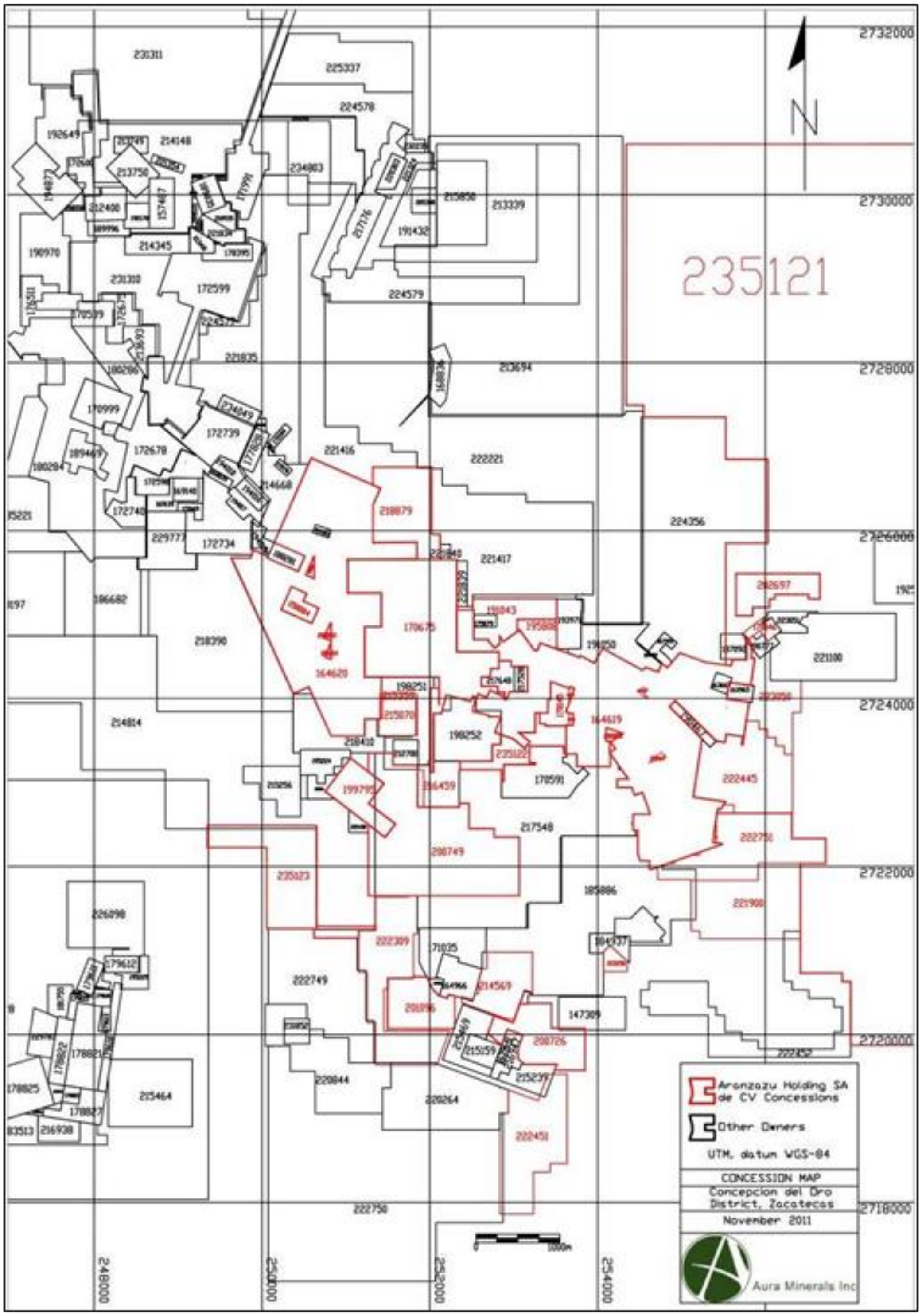


Table 4-1 Summary of the Exploitation Concession Information for the Aranzazu Property

Name	Number	Date Granted (50 year term)	Area (ha)	Annual Duty (MXN) 2018	Annual Duty ($US) 2018[1]
Demasías de la Nueva Guillotina	195664	Sep 14 1992	1.3799	$ 438.00	$24.33
Macocozac I	164619	May 22 1979	411.8367	130,044.00	$7,224.67
La Lotería	170675	Jun 11 1982	189.0107	59,684.00	$3,315.78
Demasías del Carmen y La Santa Fe	195667	Sep 14 1992	0.8583	274.00	$15.22
India Mexicana	170046	Mar 15 1982	6.6762	2,110.00	$117.22
Macocozac II	164620	Mar 22 1979	329.114	103,924.00	$5,773.56
El Pinacate	194636	May 7 1992	0.6545	208.00	$11.56
La Descuidada	178145	Jul 11 1986	12.8125	4,048.00	$224.89
La Guillotina	186014	Dec 14 1989	0.7614	242.00	$13.44
El Descuido	191043	Apr 29 1991	33.6944	10,642.00	$591.22
Ampliación el Descuido	195808	Sep 22 1992	13.1851	4,166.00	$231.44
Ampliación de la Descuidada	196542	Jul 23 1993	0.6449	206.00	$11.44
Los Nuevos Pinitos	200084	Jul 30 1994	10	3,160.00	$175.56
Reyna del Cobre	200085	Jun 30 1994	2.4559	778.00	$43.22
El Hueco	200086	Jun 30 1994	0.6728	214.00	$11.89
Anexas de La Guillotina	200083	Jun 30 1994	0.9939	316.00	$17.56
La Esperanza	199795	May 25 1994	33	10,422.00	$579.00
San Antonio	201096	Nov 14 1994	42.5314	13,432.00	$746.22
La Guadalupana	200726	Sep 26 1994	54.6271	17,252.00	$958.44
Nuevo Aranzazu	218879	Jan 23 2003	68.4636	21,620.00	$1,201.11
La Negra	200749	Sep 26 1994	195.8328	61,838.00	$3,435.44
La Conchita	202697	Dec 15 1995	33.8745	10,698.00	$594.33
La Apuesta	235121	Oct 8 2009	8,873.37	1,592,062.00	$88,447.89
La Apuesta Fracc.1	235122	Oct 8 2009	29.0054	5,206.00	$289.22
La Apuesta Fracc.2	235123	Oct 8 2009	99.5616	17,866.00	$992.56
La Laja 3	222309	Jun 24 2004	101.2162	31,962.00	$1,775.67
San Francisco 1	214569	Oct 1 2001	43.6438	13,782.00	$765.67
La Laja 5	222451	Jul 8 2004	95.8407	30,264.00	$1,681.33
La Cara	215250	Feb 13 2002	6.5896	2,082.00	$115.67
El Eden	215070	Feb 6 2002	18.9561	5,988.00	$332.67

Name	Number	Date Expires (50 Year term)	Area (ha)	Annual Fee (MXN) 2018	Annual Fee ($US) 2018[1]
El Eden	216459	May 16 2002	23.1092	7,298.0	$405.44
El Eden	215359	Feb 18 2002	2.7047	856.0	$47.56
Arco Iris	222445	Jul 8 2004	95.6715	30,212.0	$1,678.44
Arco Iris 1	221900	Apr 6 2004	99.9371	31,558.0	$1,753.22
Arco Iris 2	222751	Aug 26 2004	94.753	29,922.0	$1,662.33
Arco Iris 3	223050	Oct 6 2004	44.6094	14,088.0	$782.67
Arco Iris 3 Fraccion 1	223051	Oct 6 2004	9.6548	3,050.0	$169.44
La Laja 5	222452	Jul 8 2004	100	31,578.0	$1,754.33
Totals			11,181.70	$ 2,303,490.0	$127,971.67

Note 1: Assumes an exchange rate of 18 Mexican pesos per 1 United States Dollar
Table provided by Aura Minerals Inc.

4.3 LICENCES, PERMITS AND ENVIRONMENT

Up to suspension of operations in January 2015, Aura Minerals had acquired and maintained the required licences for operation of the mine and supporting activities including explosives handling and use, hazardous waste management, and water use. Under its water concession Aranzazu Holding is authorized to annually withdraw up to 1,081,495 m^3 of water (approximately 123 m^3/h) from its two wells located near El Salero approximately 15 km from the Property.

Since suspension of operations in January 2015, some requirements of the existing permits and licences may have been altered and it is recommended that Aura Minerals review the licences to ensure maintenance of good standing. Certain modifications and additional authorizations have been identified for the updated Project and are in process; Item 20 provides information on existing and Project-specific requirements for environmental permitting.

As discussed in Item 20, Aranzazu is a brownfield project with a centuries-long history of mining. Aura Minerals has made significant effort to inventory the various legacy workings and assign appropriate closure costs. It is recommended that during the next mine operations period, work continue on closure plan development including with review and regular update of these costs.

4.4 OTHER FACTORS AND RISKS TO ACCESS, TITLE OR RIGHT/ABILITY TO PERFORM WORK ON PROPERTY

To the extent known, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the Property.

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

5.1 ACCESSIBILITY AND LOCAL RESOURCES

The Aranzazu Property is readily accessible from the city of Zacatecas, capital of the Mexican State of Zacatecas, via paved roads. Access is primarily gained by taking Mexican State Highway 45 from Zacatecas to Fresnillo. After about 17 km, a turn-off leads to Highway 54, which connects Zacatecas with the industrial centres of Monterrey and Saltillo, to the northeast. The turn-off to Concepcion del Oro is located approximately 230 km from the junction between Highways 45 and 54. Zacatecas is an old colonial city and an important mining centre. The city has approximately 120,000 inhabitants and hosts an international airport.

The city of Saltillo, capital of the State of Coahuila, is the closest population and manufacturing centre, with a population of approximately 800,000 people. The city supports a strong automotive industry and is known as "Little Detroit". The automotive parts support industry, as well as other light and heavy manufacturing in the steel, ceramic and plastic sectors, is well developed. Saltillo is located 110 km northeast of Concepcion del Oro on Highway 54. Monterrey, in Nuevo León state and the third largest city in Mexico, is located 80 km east of Saltillo on Highway 40, and is an industrial metropolis and business hub of more than 4 million people. Saltillo and Monterrey are host to international airports with daily flights to the U.S. and other parts of Mexico
.

The Aranzazu Property lies 5 km from Highway 54. The local road which connects the highway to the mine area is a paved road which provides the primary access to Concepcion del Oro and Mazapil, including access to the Frisco-Tayahua Salaverna and San Marcos mines, approximately 6 km further west, and the Goldcorp Penasquito project approximately 20 km west of the Property.

The Property area lies on the western edge of the town of Concepcion del Oro. The cobblestone road that connects Concepcion del Oro with the small town of Mazapil crosses the concession area. Concepcion del Oro is a town of approximately 6,500 people, with approximately 12,900 inhabitants in the entire Municipal area. Most of the families have had a historic connection to mining, resulting in the availability of a semi-skilled to skilled workforce.

There are modest services in Concepcion del Oro including several small hotels, gas stations and restaurants, small stores and groceries. In general, most of the mine supplies come from the surrounding regional centres of Saltillo and Monterrey.

5.2 PHYSIOGRAPHY AND CLIMATE

The mine area is located in rugged mountains, the highest of which is at an elevation of about 3,300 masl. The mine facilities are at an elevation of about 2,150 masl. The area is semi-arid and moderately vegetated with acacia shrubs, scrub trees and bushes, Joshua trees and various cacti. The average high temperature in the summer is about 22°C and the average winter high is about 15°C. The average summer low temperature is about 15°C and the average winter low temperature is about 5°C.

The area receives approximately 432 mm of rain annually and annual pond evaporation is estimated at 1983 mm. The majority of the rain falls during the wet season from June through October, and the 50-year recurrence interval 24-hour storm is estimated at 93 mm. Occasional snow does occur in the area, but quickly melts on all but the most protected northern slopes.

The climate is mild year round and poses no limitations to the length of the operating season. Freezing temperatures can occur overnight but quickly warm to above freezing during daylight hours.

Figure 5.1 View Looking East, showing Aranzazu Processing Plant, Administration Area, Security Pit, and Historic Workings

Figure 5.2 View looking southwest, showing historic tailings deposits and the town of Concepcion del Oro. The processing plant and mine area are located at the base of the hills in the distance.

5.3 INFRASTRUCTURE

The Aranzazu Property is a brownfields project with existing water and power infrastructure in place to support resumption of mine operations.

The current fresh water supply for the processing plant relies on a 14 km pipe line from the Project's well field on the plains to the east of the operation (El Salero Pump Station 1, UTM coordinates of 265,600 East and 2,724,550 North in zone 14 WGS 84, or 101°18'55" West longitude and 24°37'03" North latitude). Three booster pumps are located along the line: Pump Station 2, on the plains, Pump Station 3 on the edge of the town and Pump Station 4 in the centre of town at the old smelter site that is owned and operated by a third party. The current authorized maximum annual water use is 1,081,495 m^3 (approximately 123 m^3/h). Items 17.3.1 and 18.11 provide further information on the fresh water makeup system and the assumptions used in evaluating the Project.

Power to the site is currently supplied via a 33 kVA power line which is shared with Concepcion del Oro. Aura Minerals has a contracted capacity of 4.5 MW. As described in Item 18.10, the Project includes new, exclusive, 6-km medium tension power line to supply up to 8 MW to Aranzazu.

The Project includes constructing the starter dam phases of a new tailings storage facility, TD5, yielding 2.81 Mt or 1,870,000 m^3 capacity. Tailings from selected areas of the site's legacy impoundments (1 through 4) will be utilized for dam construction in conjunction with suitable rock for buttress and drainage systems. Aura Minerals has submitted supporting documents to update the existing approvals for TD5 construction and has applied to change the current land use of the TD5 footprint area. Further detail on the tailings impoundment is provided in Item 18.1.

During operation tailings return water from the decant system and underdrains at the tailings storage facility will be conveyed to Pump Station 3 to combine with the fresh water supply. From there the water will be pumped to the processing plant via Pump Station 4.

Waste rock from previous years of operation is stored in piles both external to and within the open pit. A substantial quantity has also been placed in mined out stopes in the underground mine. The Project assumes that a portion of waste rock currently stored on surface can be utilized for backfilling future stopes, and no additional surface storage areas are required.

6. HISTORY

The district has a very long mining history with activities documented as early as 1548. Extensive detail of the mine's history and prior ownership is provided in Aura Minerals 2008 Technical Report for the Property (*Technical Report and Audit of the Preliminary Resource Estimate on the Aranzazu Property, Zacatecas, Mexico, July 15, 2008*).

In 1891, the Mazapil Copper Company of Manchester, England began mining and smelting operations that continued through to 1962. From 1962 until 2008, various companies have owned and operated the Aranzazu Mine. After shutting down in 1992 due to low metal prices and a labour dispute, the mining operations were restarted on a limited scale in 2007 by a private Mexican company.

The 2008 Technical Report for Aranzazu provides production estimates from 1962 to 1992.

Aura Minerals acquired 100% of the Aranzazu Mine (formerly known as the El Cobre project) in June 2008. Production was suspended in January 2009 but restarted on a limited basis in 2010, with commercial production declared effective February 1 2011. As of January 2015, Aranzazu has been in temporary suspension and all capital projects, including underground development work, have also been deferred. A summary of reported Aura Minerals production is contained in Table 6-1.

Table 6-1 Summary of Aura Production for the Aranzazu Mine

Year	Period	Mill Production (dry tonnes)	Mill Head Grade			Concentrate Shipped (dry tonnes)
			Copper (%)	Gold (g/t)	Silver (g/t)	
2008	June to December	148,511	0.69	0.25	7.92	3,116
2009	January to December	0	----	----	----	0
2010	January to December	57,211	0.51	----	----	831
2011	January to December	632,297	0.90	0.48	12.89	13,455
2012	January to December	773,900	0.85	0.50	11.98	20,983
2013	January to December	796,413	0.98	0.48	16.21	25,815
2014	January to December	941,461	0.96	0.45	15.53	27,731
2015*	January to March	84,932	0.52	0.31	9.01	2,359
2016		0	----	----	----	0
2017		0	----	----	----	0
2018		0	----	----	----	0

**Mine in Care and Maintenance as of January 2015.*

7. GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The Concepcion del Oro area contains Jurassic to Cretaceous limestone, siltstone, and shale intruded by Tertiary intrusive rocks (Figure 7.1). A thick sequence of limestone is represented by several formations in the area (Figure 7.2).

The Jurassic age rocks are represented by the Zuloaga limestone. The Zuloaga limestone is grey, has a greywacke to mudstone texture and in places contains black chert nodules. The estimated thickness of the formation is 400 m. The Zuloaga limestone is the main sedimentary rock host of the mineralization at Concepcion del Oro. Conformably overlying the Zuloaga is the upper Jurassic La Caja Formation of siltstone and inter-bedded limestone. The La Caja Formation is divided into four distinct units. The basal Unit A is a shale and black limestone. La Caja Unit B is a clayey limestone with distinctive ammonite and pelecypod fossils. Unit C is a cherty phosphorite, and Unit D is a calcareous siltstone with chert beds and nodules. The estimated thickness of the La Caja Formation is 60 m.

The Cretaceous sedimentary rocks consist of shales and limestones and include the Taraises limestone/shale, the Cupido limestone, the La Pena limestone and the Cuesta del Cura limestone in the Concepcion del Oro area. The Cuesta del Cura limestone is the youngest member of the Lower Cretaceous rocks. Upper Cretaceous limestone and shale of the Indidura Formation and shale of the Caracol and Parras Formations overlie the Cuesta del Cura limestone to the north of Concepcion del Oro. The total thickness of the Cretaceous sedimentary rocks is variable and ranges between 2,000 m and 2,540 m.

Two main types of intrusive rocks are present in the Concepcion del Oro area. One is a biotite bearing phase of a quartz monzonite to granodiorite rock. The second intrusive type contains hornblende as the predominant mafic, rock forming mineral. In the area, these intrusive rocks are intruded into the axis of an antiform that constitutes the crest of the range. The age of the intrusive rocks at Concepcion del Oro is approximately 41 Ma based on K-Ar age dating.

The Concepcion del Oro region is located within the strongly folded Eastern Fold Belt (Sierra Madre Oriental). The Upper Cretaceous-Early Tertiary Laramide orogeny with NE oriented compression folded the Jurassic-Cretaceous sedimentary sequence against the Coahuila Peninsula. The Concepcion del Oro-Providencia-Mazapil region is located in the curvature (hinge) zone where the folds of the Sierra Madre Oriental change strike from NW-SE to E-W. This change in orientation also leads to NE to E-W strike-slip faults which in combination with dilational axial zones in anticlines lead to the emplacement of intrusives, dikes, skarn and mineralized bodies within the local regional structural makeup (Figure 7.3).

The El Mascaron mountain range that hosts the Concepcion del Oro district, is crudely antiformal in nature with the crest of the antiform marked by Jurassic age rocks and the limbs composed of Cretaceous age rocks. Two major transform fault systems are mapped in the El Mascaron range. The major fault system strikes northwest and displays dextral strike slip movement. Northeast striking transform faults are interpreted to be antithetic with respect to the northwest trending major structures. The northeast trending structures exhibit sinestral strike slip movement. Synthetic splays are common on the northwest structures. Normal faults disrupt the strike slip fabric; normal faults trend generally north and east.

The structural system influenced the emplacement and contact of the Concepcion del Oro intrusive complex with the country rock. In the area from Aranzazu to Cabrestante, the contact, in places, generally follows the major northwest striking structure. In the area from Cabrestante to Cata Arroyo, the intrusive-country rock contact is somewhat controlled by the northeast trending antithetic structure that is coincident with the intrusive-country rock contact.

Figure 7.1 Regional Geology Map for the Concepcion Del Oro Mining District

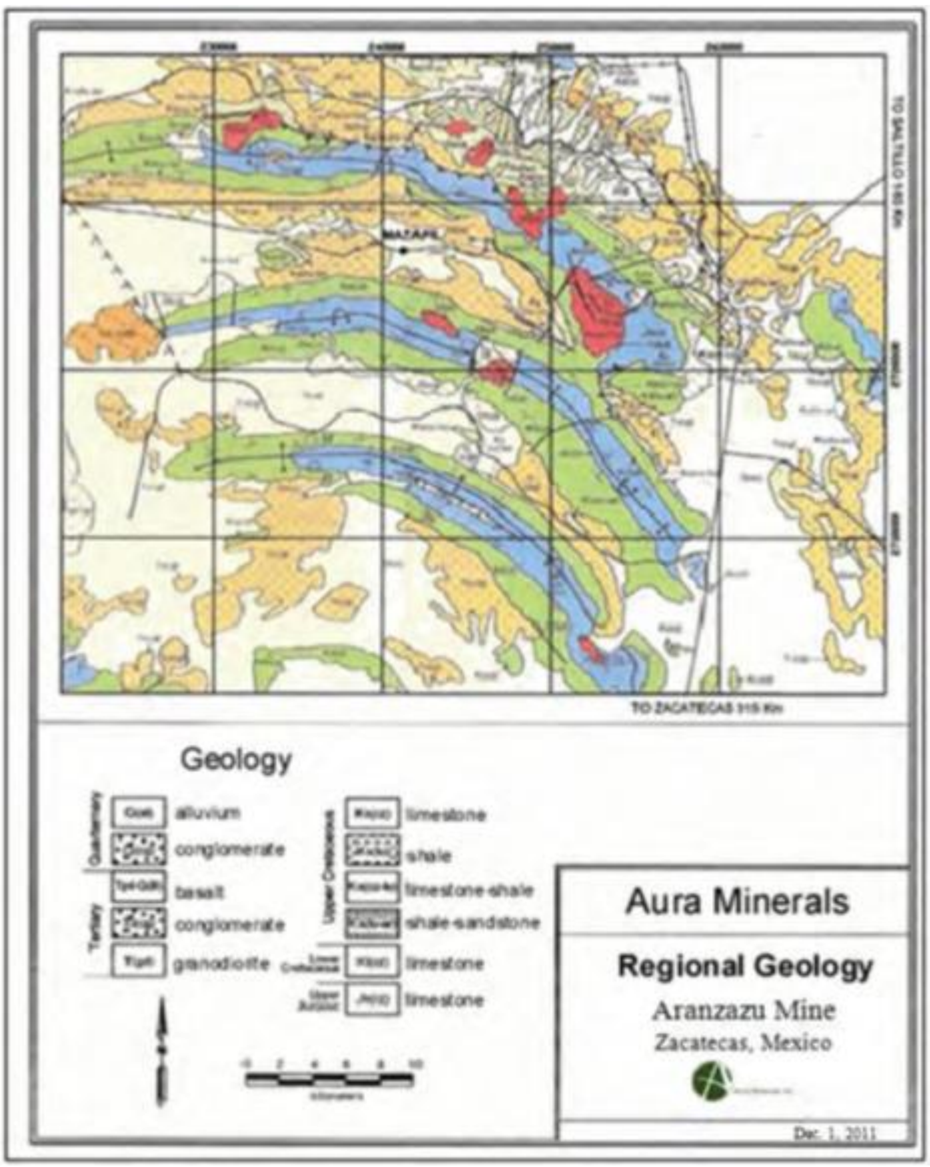


Figure 7.2 Generalized Stratigraphic Column in the Concepcion Del Oro Mining District

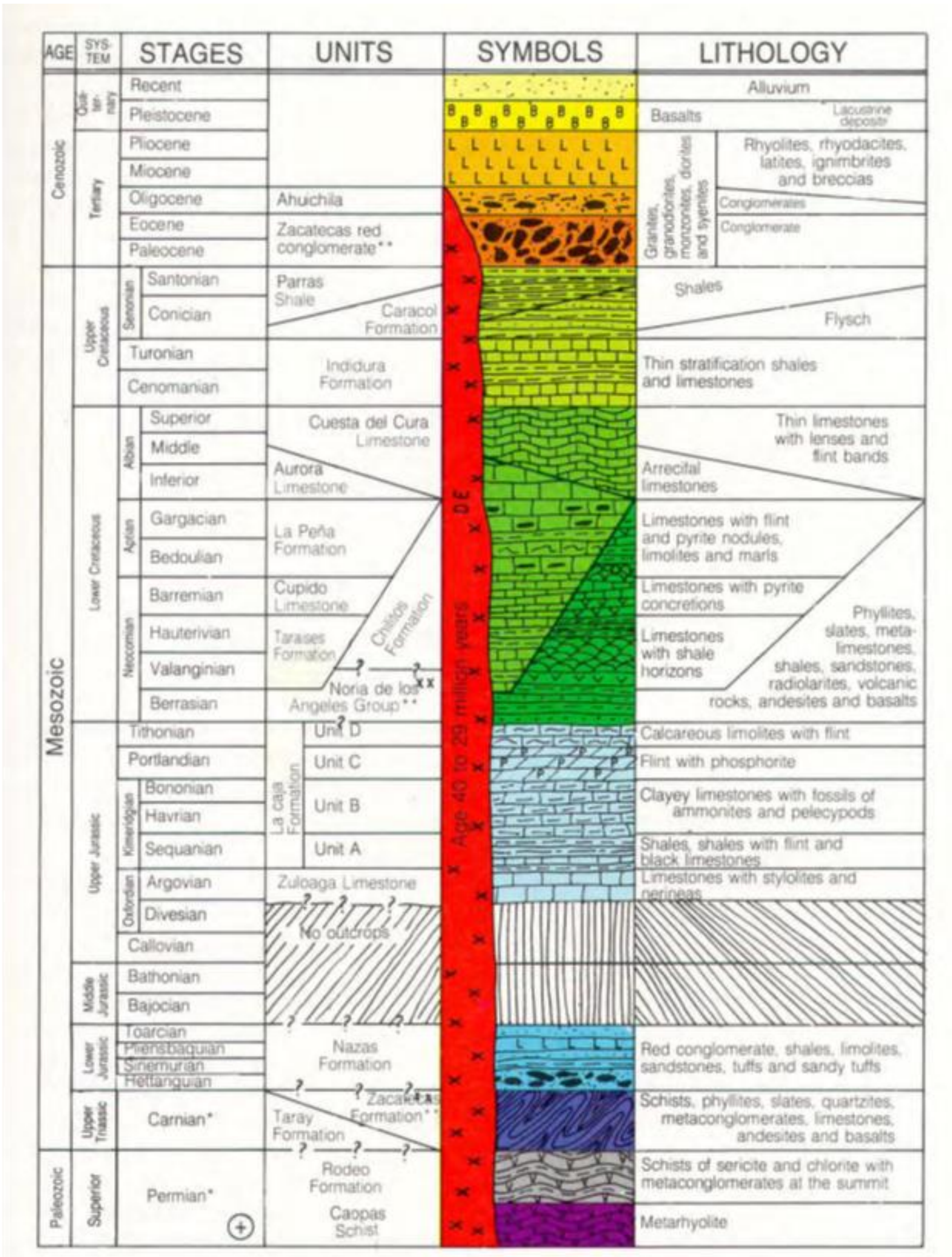
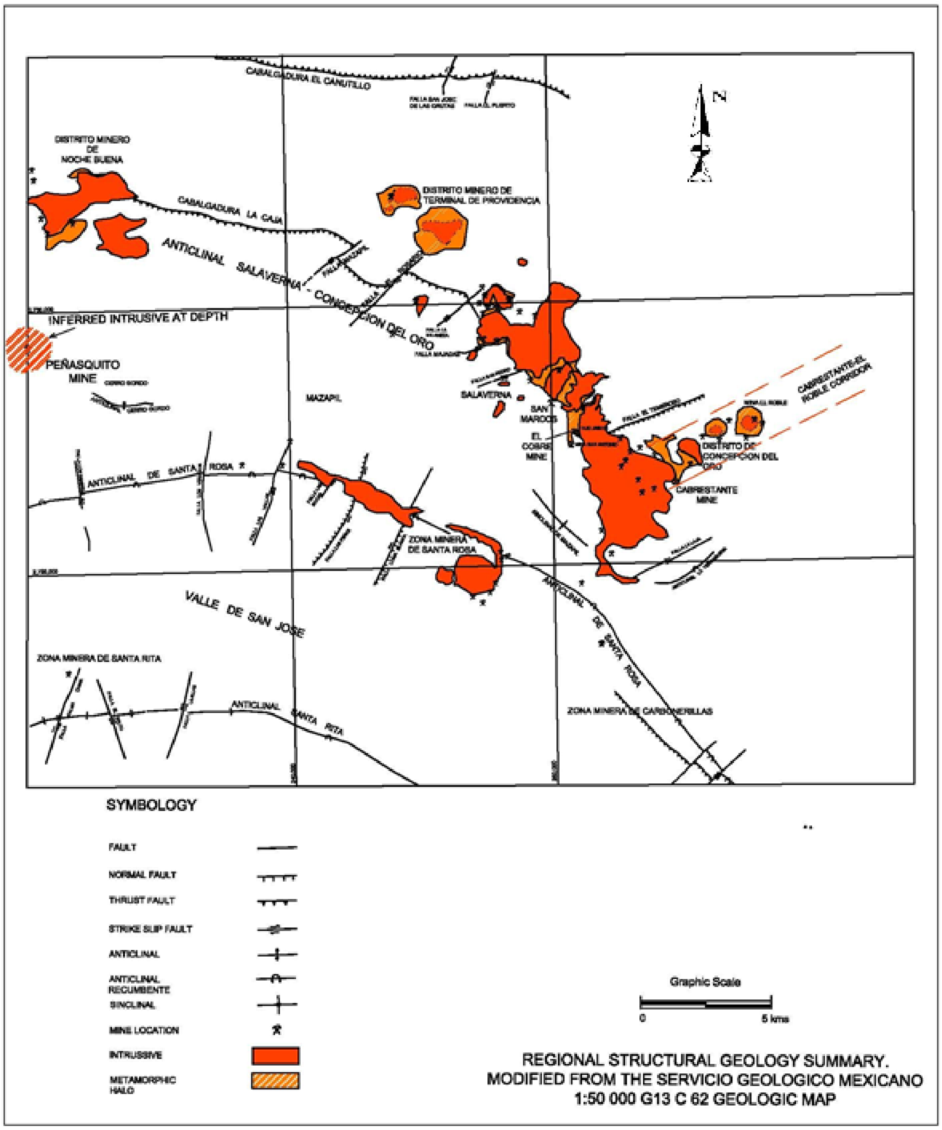


Figure 7.3 Structural framework of Concepcion Del Oro and its relationship Tertiary Intrusive



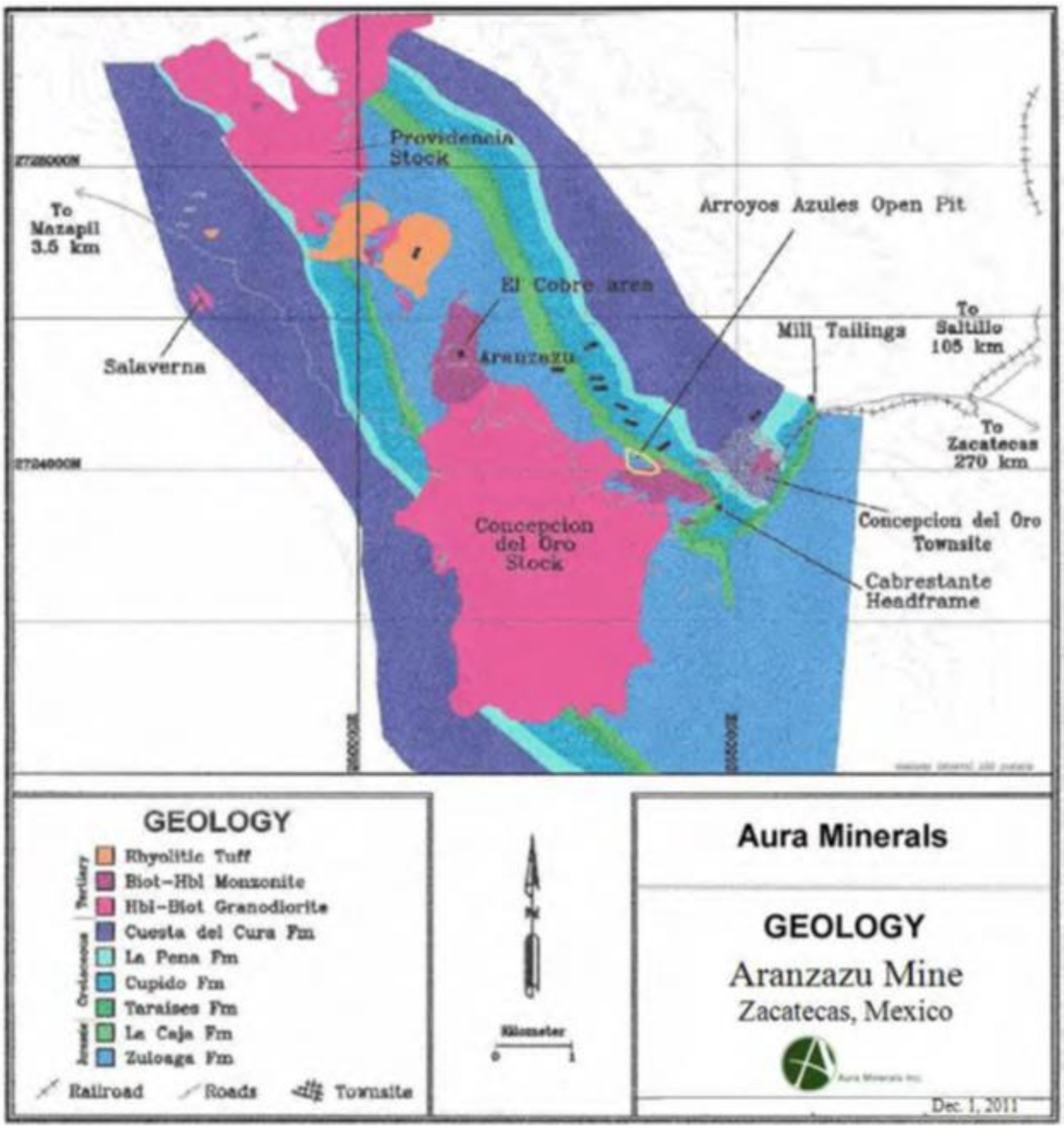
7.2 PROPERTY GEOLOGY

A Tertiary intrusive complex ranging from diorite to granodiorite intrudes the Jurassic and Cretaceous limestones along the axial plane of the El Mascaron antiform. (Figure 7.4).

The intrusive complex is also localized by the regional transform fault system. Associated with the intrusive complex, the structural system and alteration regime, copper, gold and silver mineralization is present as chimneys, mantos, stockworks and disseminations. The mineralization is hosted in exoskarn, endoskarn, quartz monzonite, hornfels and marble that have been overprinted by post-skarn hydrothermal alteration consisting of propylitic, phyllic and potassic alteration styles. Both mineralized and un-mineralized skarns are noted. Early stage skarn alteration is dominated by garnet and the opaque assemblage includes abundant magnetite and pyrite. Retrograde alteration is minimal and consists of epidote and quartz after garnet. Where the skarns are un-mineralized, the lack of hydrothermal alteration is evident. In the Arroyos Azules open pit zone, the mineralized skarns occur adjacent to portions of the intrusive complex with intense quartz vein stockwork and sericite. The approximate strike length of the mineralization at Arroyos Azules is 1.5 km. An area of mineralized skarn also occurs 2.5 km northwest of the Arroyos Azules in the Aranzazu area. The Aranzazu copper, gold and silver mineralization occurs along the contact of a quartz monzonite intrusive that has intruded into a sequence of Mesozoic limestones and siltstones.

Sediments in the Property area are thrusted and folded into well-defined anticlines and synclines. Cretaceous compression in the Concepcion del Oro area was north directed and resulted in east-west anticlinal axes. These axes are often breached by deep-seated structures which form the main conduits for the emplacement of Cretaceous to Tertiary intrusive rocks. The thrust and fold belt was subsequently refolded into arcuate belts which resulted in the development of major cross faults. These later faults cut the axial plane faults of the earlier folding event. The fault intersections focused igneous and hydrothermal activity. The anticline and syncline belts are doubly plunging, forming a dome of sedimentary rocks intruded in the axial planes and cross faults. The copper-gold systems occur on the anticline structures and systems similar to Peñasquito occur on the synclinal structures or on the limbs of the folds. The Concepcion del Oro stock is emplaced in the anticline axial plane. This portion of the anticline plunges to the east and forms a stratigraphic trap for hydrothermal fluids.

Figure 7.4 Aranzazu Property Geology Map



7.3 ALTERATION AND MINERALIZATION

Alteration in the Aranzazu intrusive consists of quartz-sericite-illite or phyllic alteration. Several other areas of mineralized skarn, or mineralized veins and chimneys, occur throughout the area and have not received systematic exploration.

The distribution of the various alteration and mineralization phases is variable along the strike of the Arroyos Azules structure. From the northwest moving southeast on strike, the BW zone formed as vertical chimney and is marked by propylitic alteration. The alteration mineral assemblage includes pyrite, chlorite, epidote and calcite. The next zone southeast is known as the Mexicana. In the upper portions of the Mexicana mineral zone, the alteration is predominately propylitic; below the 2,047 m elevation, the propylitic alteration transitions into phyllic alteration. The phyllic alteration is defined as quartz, sercite-illite and pyritic bearing rock. The next mineralized zone southeast from Mexicana is the Arroyos Azules (AA) zone. This zone has been mined in the past by open pit and currently, at the time of this report an active pit was in operation in this zone (AA pit) for oxide materials. Alteration at the Arroyos Azules (AA) and Glory Hole Porfido (GHP) zones is predominately phyllic alteration with areas of potassic alteration. The potassic alteration indicator mineralogy is secondary biotite and potassium feldspar. Potassic alteration increases at depth and can be observed in Glory Hole Porfido zone (FW zone) especially within intrusive and endoskarn. Alteration Further southeast from the Glory Hole zone to Cabrestante and ZCC, the mineralization is hosted in primarily phyllic alteration. Towards Cabrestante and ZCC zone, the gold grade within the deposit generally increases.

In the Glory Hole Porfido/Cabrestante area the mineralized bodies are irregular to tabular shape, are generally oriented NNW and occur at or close to the limestone-intrusive contacts. The mineralized bodies persist a considerable distance into the eastern intrusive apophysis or “tongue” (GHP-FW orebody), or into the hanging wall limestones. The kinematic indicators exhibit evidence the Cabrestante dilatational zones were formed by a dextral component of movement along the overall WNW trend of the intrusive-limestone contact. The Cabrestante area is also the starting point for the NE trending Cabrestante-Diamante-El Roble structural corridor where the stratigraphic displacements indicate the presence of a complex pattern of strike-slip faults associated with strong bending of the anticlinal axis.

The mineralization in Aranzazu has a strike length of 1.5 km, a width of up to 250 m and depth extents up to 600 m. The Aura Minerals’ exploration and definition drilling shows that mineralization is closing off at BW, Mexicana and AA but open at depth in Glory Hole Porfido zone. The deepest drilling conducted to date was carried out by Aura Minerals indicated the presence of strong copper mineralization at an elevation of 1,500 m, or 600 m below surface.

Copper, gold and silver mineralization occurs in a porphyry alteration assemblage related to the intrusion of an igneous rock complex consisting of quartz monzonitic to granodioritic rocks. The igneous rocks have intruded a sequence of Jurassic to Cretaceous limestones and siltstones. The sedimentary rocks have undergone contact metasomatism and skarnification and now represent a group of rocks ranging from endoskarn and exoskarn, proximal to the intrusive, to marble, distal to the intrusive. Hornfels is present and reflects the contact metasomatic alteration of clastic rocks. Porphyry alteration consisting of propylitic, phyllic and potassic alteration overprints the skarn, quartz monzonite and, in some areas, marble and hornfels. The porphyry alteration is the mineralizing event that deposited the suite of metals that comprise the mineral deposits of interest.

The copper mineral species present in the different zones vary depending on the alteration style that is prevalent. In the BW zone, host to propylitic alteration, the copper mineralization is mostly chalcopyrite. Copper mineralization present in phyllic and potassic alteration styles is chalcocite, copper sulphosalts, chalcopyrite and bornite. The trace metal assemblages also vary depending on the copper minerals present. The BW zone contains moderate amounts of arsenic, but is relatively low in antimony, bismuth and tellurium. In areas of phyllic and potassic alteration where multiple copper mineral species are present, the amounts of arsenic, antimony, bismuth and tellurium increase.

Other phases of mineralization are evident in the Concepcion Del Oro area. Zinc-lead-silver deposits are noted as discrete, relatively small chimney and manto deposits. The zinc-lead-silver mineralization phase is present as both distal deposits relative to the Arroyos Azules copper mineralization and as late stage mineralization that postdates and cross cuts the Arroyos Azules copper mineralization.

Some skarn and intrusive zones contain high concentrations of molybdenum. Often garnet skarn contains several percent of coarse grained molybdenum and it is also common in veins which cut skarn and intrusive. Molybdenum occurs within weakly propylitized endoskarn and exoskarn, though it is not clear from geological relationships where the molybdenum mineralization occurs paragenetically relative to the copper and zinc-lead-silver pulses of mineralization. Molybdenum grades do not return assays that would be considered economic.

Gold mineralization occurs throughout all of the alteration phases previously mentioned with the exception of skarn alteration. Gold grades are generally higher in the phyllic and potassic alteration assemblages compared to propylitic altered rocks.

8. DEPOSIT TYPES

The Concepcion Del Oro district represents a porphyry copper deposit that has been eroded down to a relatively deep level, as evidenced by the outcropping skarn alteration. Skarn alteration is barren unless the skarn is overprinted by the later porphyry style alteration marked by propylitic, phyllic and potassic alteration types. Trace metal geochemistry supports the classification of Concepcion del Oro as a porphyry deposit. Strongly anomalous bismuth, tellurium, arsenic and antimony are present and are typical trace metal assemblages associated with porphyry copper deposits. This trace metal assemblage is most anomalous in areas of phyllic and potassic alteration.

The Aranzazu porphyry stock is adjacent to a non-mineralized granodiorite. The quartz monzonite porphyry is potentially mineralized porphyry copper which covers an area of approximately 1 km^2. Quartz-sericite veining is in part stockwork, but the stock margins are dominated by parallel veins indicating a strong flow vector of hydrothermal fluids into the skarn zones. The Aranzazu stock consists mainly of granodiorite to quartz monzonite and monzonites of porphyroblastic to equigranular textures. Biotite and hornblende typically make up 2.0% to 5.0% of the intrusive phases, and magnetite is commonly present in the 1.0% to 2.0% range. The borders of the stock exhibit other younger phases such as quartz-feldspar porphyries, medium grained biotite-rich granodiorites, and aphanitic dikes of andesitic composition. These younger phases could have a closer relationship to the magmatic chambers responsible for metasomatism and mineralization since they are observed in proximity to mineralized skarns and zones of alteration. In the intrusive border zone where the GHP-Cabrestante orebodies are located pre-mineral quartz-feldspar porphyries occur as pre-mineral dikes between skarn orebodies.

Porphyry alteration classifications are defined on the basis that certain indicator minerals are present. Propylitic alteration has indicator minerals consisting of chlorite, epidote, calcite and pyrite. Phyllic alteration is defined by the mineral assemblage of sericite, quartz vein stockworks and pyrite; potassic alteration is marked by secondary biotite and hydrothermal potassium feldspar. Different species of copper bearing minerals are somewhat restricted to the various alteration styles. Propylitic alteration has chalcopyrite as the main copper bearing mineral. Both phyllic and potassic alteration are host to a wider range of copper minerals including chalcopyrite, chalcocite, tennantite, and a suite of copper sulphosalts.

A characteristic feature of many porphyry systems is the regular pattern of zonal metal distribution-both vertically from deep levels proximal to the composite porphyry stock and laterally from centrally isolated pottasic alteration to marginal propylitic zones.

In Aranzazu, geochemical zonation at the orebody scale shows the typical distribution of proximal copper and molybdenum in the deeper feeder zones, and zinc, lead and silver towards the distal peripheries, both longitudinally and laterally. Relatively high arsenic levels are consistent throughout the district and are particularly high in the peripheral zone like Caberstante and ZCC zones SE of the Aranzazu Property.

The assays of Pb-Zn-Ag define a clear peripheral zonation surrounding the copper orebodies of the district. Southeast of the Cabrestante shaft there are numerous small workings that were developed in small pockets of high Ag-Pb-Zn which seem to reflect lateral zonation from the Cabrestante copper-rich zone. The north and south flanks of the Cabrestante-El Diamante-El Roble corridor NE of Concepcion del Oro town has numerous showings along small occurrences (mantos, veins, dike contacts) which exhibit gossan mineralization with high values of Zn-Pb-Ag-As. These small occurrences could reflect the periphery of possible copper orebodies closer to the intrusive contacts under cover in the corridor.

Multi-elemental zoning in El Cobre mine is based only on the new sampling program carried out in 2013 (Albinson and et.al). The results show that Ag, Pb, Zn, and As increase strongly in the peripheral narrow structures of the Conejos zone.

Zonation of zinc with respect to copper, is clearly evident in the Palomas orebodies which represents the lateral zonation to the west of the copper-rich Mexicana and BW orebodies. It is also possible the zinc-rich Palomas zone could have an underlying copper-rich zone.

Molybdenum generally reports assays of interesting grade in the active Aranzazu Mine. The geochemical zonation of molybdenum in the BW to Cabrestante orebodies shows values in excess of 500 ppm in the intermediate and deep portions of the deep GHP-FW orebody and the NW side of the Mexicana orebody. It is possible the deeper portions of the copper-rich orebodies average enough grade of molybdenum to justify economic recovery of this metal.

Worldwide, porphyry copper deposits are important sources of copper, molybdenum, zinc, lead, gold and silver are often important by-products. Important porphyry copper deposits include the Battle Mountain Complex, Nevada, U.S.A., Bingham Canyon, Utah, U.S.A., Highland Valley, British Columbia, Canada and the Grasberg Complex, Irian Jaya, Indonesia.

9. EXPLORATION

9.1 INTRODUCTION

Historical drilling was discussed in Item 6 of this Technical Report and was extensively discussed in the 2011 Technical Report.

For simplicity and point of reference, the term historical in this report is referring to all exploration activities before 2008 which were undertaken by previous owners.

9.2 2010 TO 2011 EXPLORATION PROGRAM

This section was adopted directly from previous 2011 Technical Report.

"*The 2010 to 2011 exploration program was comprised primarily of drilling, with some further work conducted to bring the historical data into the database.*

From July 16, 2010 to April 2011, Aura Minerals drilled a further 39,176 m of core drilling in 126 holes and 17,489 m of RC drilling in 98 holes to better define the Mexicana, Arroyos Azules and Glory Hole-Porfido zones. The total drilling completed by Aura Minerals by the end of Phase 2, including Phase 1 drilling, is 108,052 m in 471 holes as of May 20 2011, the effective date of the drill hole database used for the September 1, 2011 Mineral Resource estimate (effective date).

Core and RC drilling was carried out to upgrade the mineral resources in the Mexicana, Arroyos Azules and Glory Hole areas. RC drilling was carried out in the Glory Hole-Porfido and Cabrestante areas to test near surface mineralization that has the potential to be mined by open pit. The deep mineralization, below 1700 m elevation, was also tested by core drilling in the Arroyos Azules, Glory Hole-Porfido and Cabrestante Areas.

The bulk of this drilling was directed at the Arroyos Azules structure from the Mexicana zone southeast to the Tiro Azules zone. The drilling was planned to reduce the drill spacing along strike and down dip and to conduct drilling proximal to existing historic drilling for comparative and check purposes. The results of the drilling by Aura Minerals, in conjunction with historic and Zacoro drilling form the database for the updated resource estimate discussed in this report.

Aura Minerals has outlined an exploration program for the Aranzazu Property based on a number of priority exploration targets that remain relatively untested. The exploration targets demonstrate the possibility that further areas of economic mineralization may occur either laterally along strike of the existing mineralized zones or at depth below the previously mined areas."

9.3 2011 TO 2014 EXPLORATION PROGRAM

From 2011 to 2014 no exploration drilling was done and any drilling completed was primarily for the purpose of stope definition and delineation of ore bodies below the mining levels. Definition drilling during this period continued for BW, AA, Mexicana and Glory Hole Porfidio zones.

During 2013, district-wide field work, carried out in the Aranzazu-Concepcion del Oro district by Exploraciones del Altiplano, SA de CV on a contract basis, in the claim block of Aranzazu, identified three principal exploration targets within the Aranzazu claim blocks. The work consisted of mapping, sampling and semi-detailed review of the historic maps and reports present in the files of the mine. The objective was to evaluate and prioritize the district wide exploration potential, according to an estimate of the size and grade of the targets within the claim block.

An important part of the report discusses the geologic analogies that the Aranzazu claim block has with nearby deposits, and benefits from the knowledge gained from important modern discoveries in the adjacent Tayahua claim block to the west, namely the discovery by Kennecott and Minera Frisco of the blind large skarn deposit (+100 M tons) associated with the cupula of the Salaverna intrusive.

10. DRILLING

10.1 INTRODUCTION

Based on NI43-101 guidelines all of the drilling before February 2001 is considered to be historical without consideration of drill hole data validity. Since only limited number of core boxes prior to 2008 were available in the Aura Minerals core shack facility, re-sampling , re-logging and further validation of these cores was not possible. Therefore, it is noted that all previous drilling before 2008 has been considered to be historical drilling and is not discussed here even though the results of this drilling was used by previous owners in successive resource estimation of the Aranzazu mine and also, by Micon in their 2011 resource model.

The prior to 2008 drill hole data was used for the current model which is presented in this Technical Report. The historical drilling data has been validated on many occasions by Aura Minerals and independent consultants (Micon 2011). In the opinion of the QP there is no reason to believe that previous validations of historical data were biased.

Aura Minerals has conducted additional drilling for the purpose of exploration and resource definition since 2008. The 2010-2011 drilling campaign is summarized here from the previous 2015 Technical Report.

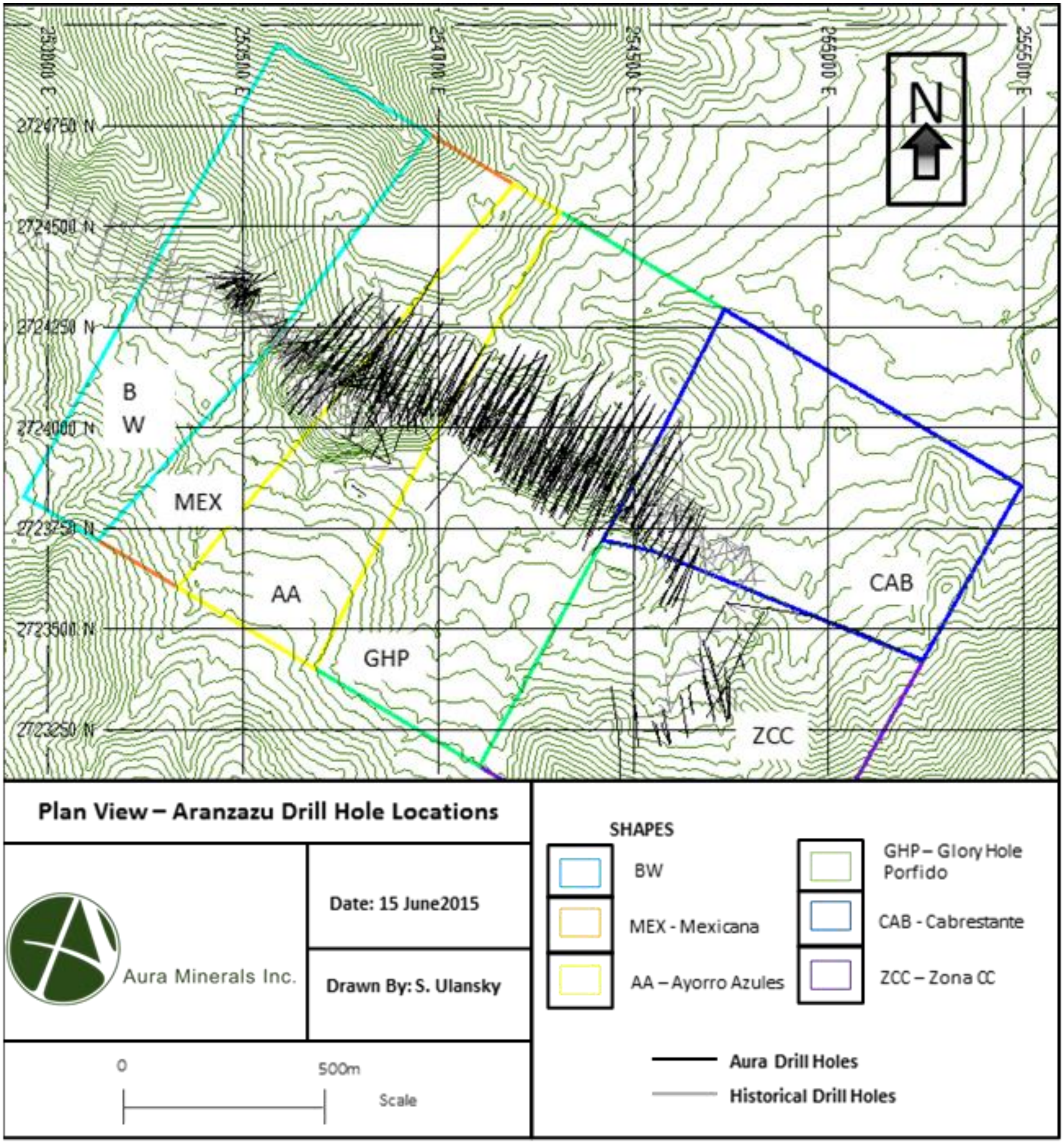
Since 2011 drilling was limited to definition drilling in stopes to upgrade the current resource estimate in a few selected zones. Aura Minerals did not undertake any major exploration activities to further delineate the resources at depth. The definition drilling campaign was conducted in two campaigns during 2012 to 2013 and 2014. In 2017, a limited drilling campaign was conducted to capture some geotechnical parameters. Table 10-1 shows the amount of drilling per campaign up to 2014; the associated results were stored in a geological database for the purpose of resource estimation showing number of samples and average length.

Figure 10.1 shows the drill hole locations and includes historical drilling on the Aranzazu Property.

Table 10-1 Drill campaign information and related information in database (period ending 2014)

						Samples		
	Years	Holes	Length		Average Length / Hole	Cu	Au	Ag
			m	*%*	*m*			
Mazapil Copper	1961-1962	3	339	0.1%	113	92	74	75
Asarco Mexicana	1969-1973	110	9,181	4.0%	83	1,698	32	106
Macocozac	1978-2007	473	35,532	15.7%	75	13,018	3,087	7,076
Zacoro	2007	160	42,611	18.8%	266	10,360	10,366	9,962
Aura	2008-2014	648	139,319	61.4%	215	65,164	65,163	65,163
Total		1394	226,982					

Figure 10.1 Plan Showing Aura Minerals Drill Hole Locations



10.2 2010 TO 2011 AURA MINERALS DRILLING PROGRAM

This section was adopted and summarized from previous 2011 Technical Report.

"From July 16, 2010 to April, 2011, Aura Minerals drilled a further 39,176 m of core drilling in 126 holes and 17,489 m of RC drilling in 98 holes to better define the Mexicana, Arroyos Azules and Glory Hole-Porfido zones. The total drilling completed by Aura Minerals by the end of Phase 2, including Phase 1 drilling, is 108,052 m in 471 holes.*

Core and RC drilling was carried out to upgrade the mineral resources in the Mexicana, Arroyos Azules and Glory Hole areas. Drill hole spacing in this area is approximately 25 m by 25 m, between the 2,050 m and 1,850 m elevation. RC drilling was carried out in the Glory Hole-Porfido and Cabrestante areas to test near surface mineralization that has the potential to be mined by open pit. Drill hole spacing in this area is also approximately 25 m by 25 m for the near surface mineralization. The deep mineralization, below the 1,700 m elevation, was also tested by core drilling in the Arroyos Azules, Glory Hole-Porfido and Cabrestante areas but on a wider drill spacing than the near surface material.

In general, the mineralization strikes to the northwest and dips steeply to the northeast with a strike length of approximately 2,200 m. The true width of the mineralization varies from drill hole to drill hole and is dependent not only on the width of the mineralization but also on the angle at which the drill hole intersected the mineralization. At this time, Aura Minerals has not estimated the true width of the mineralization.

Drilling activity on the Property was suspended on May 12, 2011, with the last hole drilled being underground core hole UAZ-081. The last drill hole included in the database as of May 20, 2011 is UAZ-050, which means there are 30 drill holes that have not been included in the September 1, 2011 resource estimate (effective date), as assay results were pending at that time."

**Arroyos Azules zone in the current report is called AA zone*

Holes UAZ-51 and above, although drilled in late 2011, were not included in the database used by Micon to prepare their resource estimate (Micon 2011). Therefore, the Author of this Item reviewed the results of those holes and were now incorporated in the current Technical Report and estimation presented in this section. These holes were collared from surface and target deeper parts of the ore body.

Table 10-2 summarizes the significant intercepts for holes which were drilled in 2011. The mineralized intervals and corresponding assay results are also presented.

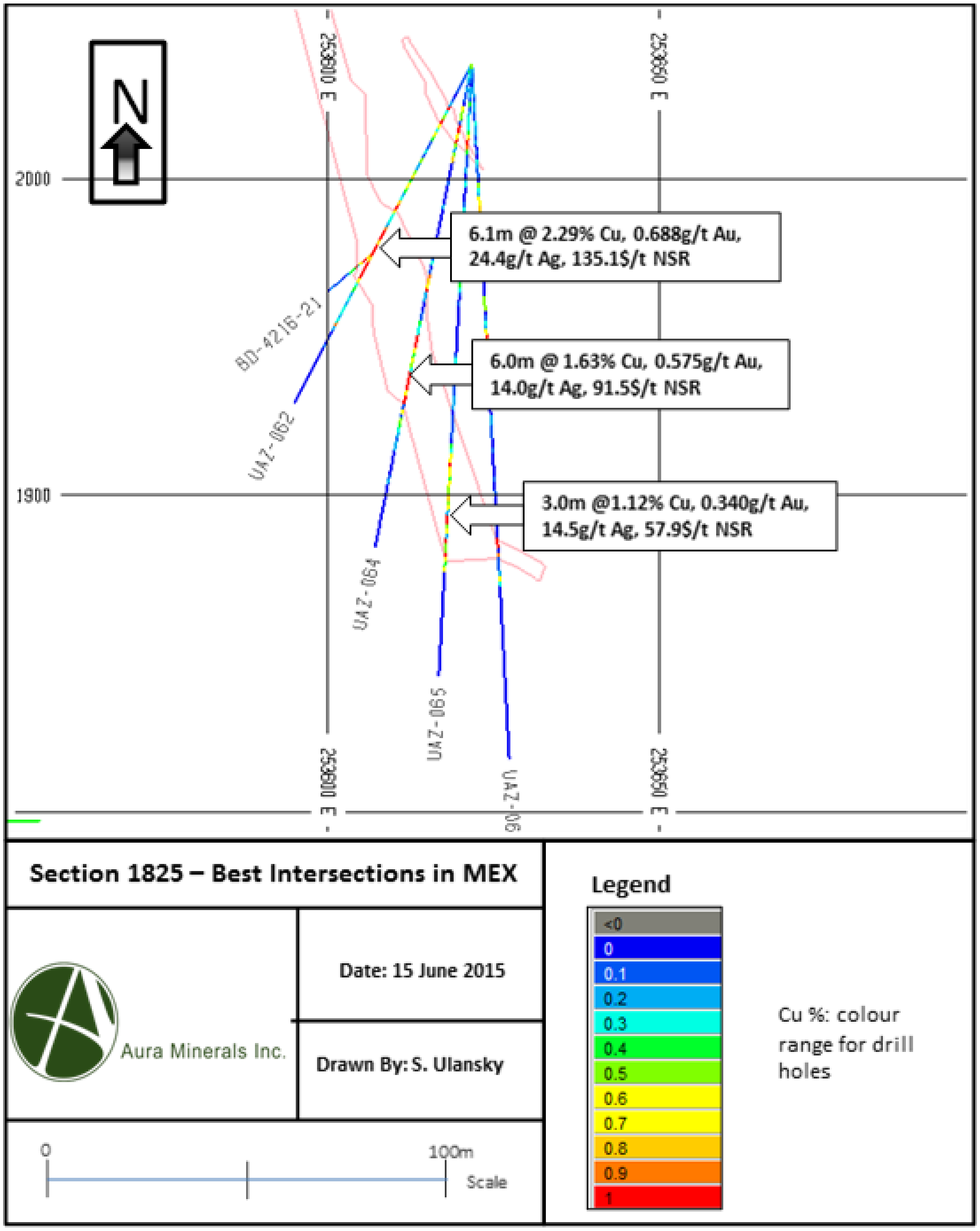
Figure 10.2 illustrates northwest looking drill hole cross-sections that shows 2011 drill hole results which were not included in the 2011 Micon resource estimates (UAZ series). The interpreted wireframes based on NSR values are also shown for references (see Item 14). The red wireframe delineates the Mexicana North zone. As shown in the figure, four drill holes, UAZ-62, UAZ-64, UAZ-65 and UAZ-66 are grading more than 0.5% copper and have NSR values above US$45. The mineralized intercepts for these drill holes are also shown in Table 10-2. The intervals displayed are core-length intervals and not true width. These holes extended the Mexicana zone to greater depth.

Table 10-2 Selected Significant Intercepts from Surface Core Drilling Results (period ending 2011)

Drill Hole ID	Mineralized Interval (m)			Assay Results		
	From	To	Interval	Cu (%)	Au (g/t)	Ag (g/t)
UAZ-051	42	45	3	1.51	0.91	62.8
UAZ-051	51	55.5	4.5	1.4	0.333	33.7
UAZ-051	69	72	3	1.23	0.193	25
UAZ-051	88.5	94.5	6	1.8	0.425	31.2
UAZ-052	108	118.5	10.5	2.76	0.821	26.4
UAZ-052	135	145.5	10.5	2.5	0.391	25.9
UAZ-053	49.5	52.5	3	1	0.27	11.4
UAZ-053	175.5	178.5	3	1.92	0.505	13.7
UAZ-053	180	186	6	3.71	0.843	21.1
UAZ-055	24	27	3	1.32	0.283	14.7
UAZ-055	57	66	9	4.09	1.013	80.2
UAZ-055	67.5	70.5	3	1.1	0.55	30.3
UAZ-055	87	93	6	1.55	0.243	31.9
UAZ-056	36	40.5	4.5	0.87	0.758	47.8
UAZ-056	124.5	127.5	3	0.98	0.503	16.6
UAZ-057	22.5	25.5	3	1.07	0.24	12.7
UAZ-058	49.5	52.5	3	1.4	0.21	14.3
UAZ-058	123	126	3	1.78	0.548	19.8
UAZ-058	144	147	3	1.21	0.323	12
UAZ-058	156	160.5	4.5	0.74	1.252	5.6
UAZ-060	24	28.5	4.5	1.11	0.26	13
UAZ-060	85.5	91.5	6	1.57	0.484	12.7
UAZ-060	99	102	3	1.28	0.443	16.3
UAZ-061	55.5	60	4.5	2.68	0.607	33.5
UAZ-061	72	75	3	1.56	0.425	11.1
UAZ-062	57.7	62.2	4.6	1.2	0.353	23.4
UAZ-062	63.8	68.3	4.6	1.39	0.495	18.4
UAZ-062	69.9	76	6.1	2.29	0.688	24.4
UAZ-064	19.5	22.5	3	1.62	0.18	20.5
UAZ-064	87	91.5	4.5	1.49	0.462	32.9
UAZ-064	99	105	6	1.63	0.575	14
UAZ-064	106.5	109.5	3	1.32	0.363	11.9
UAZ-065	142.5	145.5	3	1.12	0.34	14.5
UAZ-066	150	153	3	1.67	0.548	29.3
UAZ-068	39	42	3	8.67	1.675	133.6
UAZ-068	129	132	3	3.42	3	19.1

UAZ-069	43.5	46.5	3	6.79	1.903	90.6
UAZ-069	66	69	3	2.38	0.905	37.1
UAZ-069	127.5	130.5	3	11.9	2.405	82.3
UAZ-069	132	135	3	4.58	2.71	25.6
UAZ-069	144	150	6	2.84	1.194	11.7
UAZ-070	45	49.5	4.5	5.73	1.317	81
UAZ-070	51	64.5	13.5	2.87	0.706	28.8
UAZ-070	150	154.5	4.5	3.59	0.755	21.7
UAZ-070	157.5	160.5	3	3.44	0.735	11.6
UAZ-071	66	69	3	1.44	0.36	17.8
UAZ-071	147	151.5	4.5	4.94	2.18	41.9
UAZ-076	49.5	52.5	3	1.46	0.248	14.8
UAZ-076	55.5	58.5	3	1.43	0.313	11.5
UAZ-076	99	102	3	1.17	0.225	7.1
UAZ-076	186	193.5	7.5	2.46	0.885	49.9
UAZ-077	93	96	3	1.37	0.455	8.8
UAZ-077	97.5	102	4.5	2	0.438	14.1
UAZ-079	60	66	6	1.54	0.539	23.4
UAZ-080	84	93	41.9	1.18	0.323	16.7
UAZ-080	96	105	9	8.71	1.7	126.9
UAZ-080	130.5	133.5	3	1.17	1.005	18.5
UAZ-081	43.5	46.5	3	1.54	0.235	16.9

Figure 10.2 Section 1825, MEX Area Showing the 2011 DDH Significant Intersections



10.3 2012 TO 2014 AURA MINERALS DRILLING PROGRAM

The goal for this drilling was to delineate resources for production and mine planning purposes and to also upgrade resource categories. After the 2011 drilling campaign was concluded in May 2011, further drilling was suspended until August 2012. Since 2012 all of the drilling activities were carried out underground from stopes and levels. Table 10-3 shows all the holes which were drilled during the 2012 to 2014 definition drilling programs.

The majority of drilling activities during 2012 were focused on delineation of the BW zone. In 2013, definition drilling was carried out in BW, AA and Mexicana zones. In 2014 Glory Hole Hanging Wall zone (GHH) was also included.

Significant intersections are presented in Table 10-4. The intercepts are interval lengths, as true widths have not been determined. A copper cut-off grade of 0.5% and a minimum length of 3.0 m have been used to define the significant intercepts.

Figure 10.3 displays a number of Aura Minerals' 2012 to 2014 drill holes that have intersected copper-gold-silver mineralization in the BW zone on the Aranzazu Property. The drill hole traces are coded by copper grade. Nine UG drill holes have intersected high grade copper-gold mineralization as shown by the intercepts in Figure 10.3. From these nine holes, significant intercepts from four selected holes to be shown on the section. These intercepts are listed in Table 10-3. The lengths shown in Figure 10.3 are intercept lengths and not the true width of the mineralization.

The 2012 to 2014 drilling was conducted from underground with goal of delineating of each mineralized zone, for pre-production, as well as to upgrade the resource.

Table 10-3 Aura Minerals UG Drill Hole Collar Locations (period 2012 to 2014)

Hole ID	Year Drilled	East (meters)	North (meters)	Elevation (meters)	Depth (meters)	Azimuth (°)	Dip (°)
BD-4375-20	2012	253514.52	2724347.82	1996.43	57.85	203.4	-17.31
BD-4385-10	2012	253502.36	2724375.23	1996.93	71.15	220.3	-18.56
BD-4385-11	2012	253502.36	2724375.22	1996.02	69.75	239.8	-46.01
BD-4385-12	2012	253502.37	2724375.21	1997.08	63.15	240.0	-16.42
BD-4385-13	2012	253501.99	2724376.09	1996.49	67.15	249.5	-38.00
BD-4385-14	2012	253501.96	2724376.11	1997.26	69.90	250.1	-15.00
BD-4385-15	2012	253503.05	2724374.42	1998.26	77.95	248.1	9.00
BD-4385-16	2012	253504.09	2724373.69	1998.60	69.40	200.2	10.00
BD-4385-17	2012	253503.59	2724374.01	1998.45	71.45	223.1	8.00
BD-4385-7	2012	253505.15	2724373.06	1996.43	51.60	207.0	-16.09
BD-4385-8	2012	253505.14	2724373.03	1996.79	75.00	200.5	-34.40
BD-4385-9	2012	253504.09	2724373.72	1996.02	66.15	215.5	-47.76
BD-4160-53	2013	253854.09	2724153.03	2003.32	120.25	154.6	-21.71
BD-4160-54	2013	253853.24	2724153.20	2004.03	91.75	166.9	-11.25

BD-4160-55	2013	253853.16	2724153.36	2003.54	118.65	167.4	-20.50
BD-4216-21	2013	253667.48	2724213.37	2002.02	83.65	265.0	-27.00
BD-4216-22	2013	253669.23	2724212.04	2002.05	82.35	240.0	-21.00
BD-4216-23	2013	253669.34	2724214.34	2001.81	29.30	213.2	-19.00
BD-4216-25	2013	253672.19	2724213.77	2002.00	93.30	211.0	-17.00
BD-4216-27	2013	253672.26	2724213.87	2001.98	124.45	227.0	-41.00
BD-4365-28	2013	253543.83	2724346.95	1975.47	82.50	207.0	-30.00
BD-4365-29	2013	253543.82	2724346.96	1975.55	53.05	188.2	-50.17
BD-4365-30	2013	253543.49	2724346.63	1975.45	96.65	220.0	-28.00
BD-4365-31	2013	253543.73	2724346.74	1977.00	82.65	232.0	-29.00
BD-4365-32	2013	253543.73	2724346.74	1977.00	82.80	231.3	13.09
BD-4365-33	2013	253543.73	2724346.74	1977.00	83.30	231.1	-40.06
BD-4365-34	2013	253543.73	2724346.74	1977.00	81.90	244.5	-28.52
BD-4365-35	2013	253543.73	2724346.74	1977.00	87.20	238.5	-37.30
BD-4365-36	2013	253543.73	2724346.74	1977.00	91.25	250.2	-35.97
BD-4380-37	2013	253511.37	2724368.28	1976.32	59.60	272.9	-16.12
BD-4380-38	2013	253512.24	2724368.26	1976.16	121.25	272.9	-28.37
BD-4380-39	2013	253512.09	2724368.30	1976.05	78.45	272.9	-39.49
BD-4380-40	2013	253510.42	2724366.91	1976.36	80.05	253.9	-10.42
BD-4380-41	2013	253511.89	2724367.32	1976.14	80.55	253.9	-23.00
BD-4380-42	2013	253512.26	2724367.56	1976.12	86.80	253.9	-35.28
BD-4380-43	2013	253511.50	2724365.97	1976.18	60.30	232.4	-13.77
BD-4380-44	2013	253512.28	2724366.43	1976.07	70.35	232.4	-29.49
BD-4380-45	2013	253513.07	2724366.90	1976.09	82.05	232.4	-43.30
BD-4380-46	2013	253513.99	2724366.33	1976.56	80.25	205.4	-14.13
BD-4380-47	2013	253514.29	2724366.96	1976.67	85.95	205.4	-30.16
BD-4380-48	2013	253514.32	2724366.58	1976.07	90.95	205.4	-44.08
BD-4380-49	2013	253523.12	2724380.79	1975.98	78.20	296.9	-54.03
BD-4380-50	2013	253523.07	2724380.10	1976.51	107.05	274.9	-41.32
BD-4380-51	2013	253523.33	2724378.58	1976.04	82.00	232.2	-52.17
BD-4380-52	2013	253523.99	2724377.99	1975.99	106.85	197.1	-45.76
C-1955-56	2013	253545.57	2724382.41	1954.72	121.80	221.1	-29.40
C-1955-57	2013	253545.54	2724382.39	1955.06	101.40	221.4	-20.63
C-1955-58	2013	253545.96	2724382.87	1954.61	102.85	223.1	-38.76
C-1955-59	2013	253546.26	2724383.19	1955.32	121.50	222.5	-61.52
C-1955-60	2014	253546.37	2724383.41	1955.30	122.55	225.5	-79.16
C-2004-61	2014	253852.53	2724155.15	2004.82	115.40	180.8	-35.69
C-2004-62	2014	253852.46	2724153.88	2003.36	153.75	180.1	-50.62

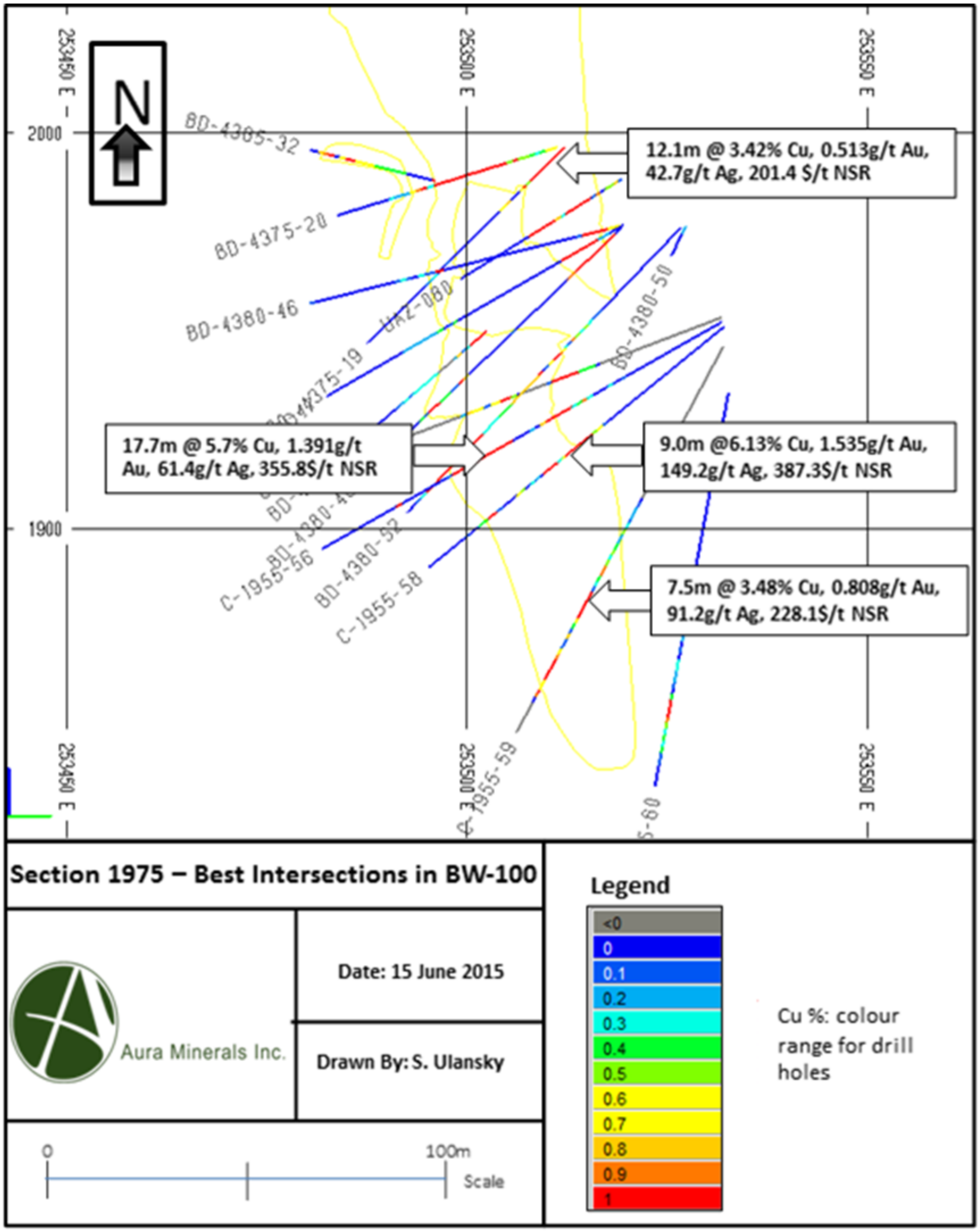
Table 10-4 Significant Intercepts from UG Core Drilling Results (period 2012 to 2014)

Drill Hole ID	Mineralized Interval (m)			Assay Results		
	From	To	Interval	Cu (%)	Au (g/t)	Ag (g/t)
BD-4365-1	0	3	3	1.93	1.29	28.5
BD-4365-1	7.5	15	7.5	3.98	0.818	52
BD-4365-2	3	6	3	3.14	0.834	46.5
BD-4365-2	16.1	29.8	13.7	1.4	0.501	26.7
BD-4365-2	38.1	43.6	5.5	2.05	0.785	29.7
BD-4365-3	4.5	7.5	3	1.25	0.237	9.5
BD-4365-4	1.5	8.7	7.2	2.49	0.724	37
BD-4365-4	30.5	35	4.5	1.9	0.477	31.7
BD-4365-6	23.7	26.7	3	1.59	0.234	9
BD-4365-6	31.2	34.8	3.7	2.32	0.541	31
BD-4385-11	49.6	54.1	4.5	0.96	0.435	5
BD-4385-12	39.2	42.2	3	1.76	0.213	11
BD-4375-18	11.5	24.1	12.6	2.37	0.479	33.8
BD-4375-18	39.1	42.1	3	2.45	3.831	31.5
BD-4375-18	58.6	61.6	3	1.87	0.457	42.5
BD-4375-19	0	12.1	12.1	3.42	0.513	42.7
BD-4375-20	13.2	32.7	19.5	5.87	1.597	134.7
BD-4375-20	34.2	37.2	3	1.2	0.481	21
BD-4216-21	52.3	56.8	4.5	1.09	0.344	23
BD-4216-21	59.8	62.8	3	1.31	0.519	25
BD-4216-27	15.2	18.2	3	1.14	0.361	5
BD-4216-27	96.2	102.2	6	1.57	0.513	9.3
BD-4365-30	34	40	6	2.5	0.446	44.8
BD-4365-31	24.5	32	7.5	3.57	0.861	68
BD-4365-31	36.5	66.7	30.2	3.36	0.909	76.3
BD-4365-32	26.8	37.3	10.5	2.59	0.594	42.4
BD-4365-32	43.3	49.5	6.2	1	0.346	30
BD-4365-32	56.6	65.6	9	0.48	3.374	9
BD-4365-32	70.6	73.6	3	1.74	0.681	26
BD-4365-32	76.6	79.6	3	1.38	0.713	22
BD-4365-33	24.7	27.7	3	1.3	0.429	31
BD-4365-33	29.2	57.7	28.5	3.49	0.878	78.1
BD-4365-33	59.1	65.1	6	1.74	0.434	33.5
BD-4365-33	70	73.8	3.8	2.57	0.47	57.3
BD-4365-34	22.3	41.2	18.9	2.21	0.603	44.4
BD-4365-35	25.3	28.3	3	1.37	0.499	29.5
BD-4365-35	31.3	50.8	19.5	4.04	1.357	93.3
BD-4365-36	26.5	46.6	20.1	2.49	0.604	46.3
BD-4365-36	64.4	69.5	5.1	1.24	1.045	15.1
BD-4380-38	53.4	56.4	3	0.86	1.489	16
BD-4380-42	46.6	49.6	3	1.1	0.424	2.5

BD-4380-45	5.9	9.2	3.3	3.78	0.994	59.8
BD-4380-46	3.1	9.8	6.7	5.51	1.3	111.2
BD-4380-47	7	17.7	10.7	2.79	0.573	29.3
BD-4380-48	2.5	16	13.5	2.43	0.403	31.6
BD-4380-48	63.4	67.5	4.1	3.57	0.868	45.6
BD-4380-50	69.3	73.4	4.1	0.33	8.244	58.9
BD-4380-52	42.5	46.6	4.1	2.52	0.546	48.5
BD-4380-52	72.6	78.6	6	3.89	0.473	33
BD-4380-52	84	93.8	9.8	4.45	1.027	47
BD-4160-53	80.8	94.5	13.8	6.37	1.465	53.3
BD-4160-53	104.9	108.1	3.2	3.67	0.952	45.8
BD-4160-54	72.7	80.7	8	4.61	0.295	24.5
BD-4160-55	84.7	89.2	4.5	3.78	0.543	47.3
BD-4160-55	96.7	105	8.3	4.87	2.105	64.6
C-1955-56	48.8	58.2	9.4	3.37	0.669	57.8
C-1955-56	66.3	84	17.7	5.7	1.391	61.4
C-1955-57	43.6	50.9	7.4	2.13	0.37	34.7
C-1955-57	62	65	3	5.71	0.643	112.5
C-1955-58	48.8	57.8	9	6.13	1.535	149.2
C-1955-58	60.2	65.1	4.9	2.26	0.505	52
C-1955-58	71.1	74.1	3	0.67	1.687	79
C-1955-58	80.3	83.4	3.1	14.5	2.926	133
C-1955-59	81.1	88.6	7.5	3.48	0.808	91.2
C-1955-59	99.1	106.6	7.5	2.21	0.759	40.2
C-1955-60	100.4	106.4	6	1.31	0.106	13.5
C-2004-61	65.9	70.4	4.5	3.14	0.671	21.3
C-2004-61	73.4	90.6	17.2	3.42	0.711	30.8
C-2004-61	107	111.4	4.3	6.71	1.483	73.7
C-2004-62	63.9	66.9	3	2.56	0.607	17
C-2004-62	69.9	92.4	22.5	2.97	0.505	27
C-2004-62	107.5	111.1	3.6	7.79	0.898	72.5
C-2004-62	125.3	128.4	3.1	10.6	2.298	96.4

(1) True width has yet to be determined.
(2) Using a copper cut-off grade of 0.5%.

Figure 10.3 Section 1975, BW Area Showing the UG Significant Intersections (2012 to 2014)



10.4 2017 GEOTECHNICAL DRILLING PROGRAM

In 2017, Aura Minerals conducted a geotechnical drilling program for the purpose of preparing the FS and underground mine plan. A total of 1403.6m were drilled in seven holes (Table 10-5). Aura Minerals contracted Call & Nicholas, Inc. from Tucson, Arizona to supervise the drilling program and prepare a geotechnical report which would be based on results of this drilling program.

The drill core was logged, primarily to collect geotechnical parameters including Rock Quality Data (RQD) and then logged again for geology and sampled. A limited number of samples were selected for geotechnical tests and then core samples were halved and sent to the assay laboratory.

The 2017 geotechnical drill program was conducted from underground (200 level) and focused specifically on the Glory Hole Zones, both the Footwall and Hanging Wall zones (GHP and GHH). An additional exploration hole was also drilled (from 1965 level) below the current resources to investigate continuation of the Glory Hole Zone, down plunge. This hole intersected mineralization within the skarn and peripheral quartz porphyry intrusive.

Significant intersections from the 2017 drill program are presented in Table 10-6. The intercepts are interval lengths, as true widths have not been determined. A copper cut-off grade of 0.5% and a minimum length of 3.0 m have been used to define the significant intercepts.

Figure 10.4 displays two of the 2017 Aura Minerals' drill holes that have intersected copper-gold-silver mineralization in the Glory Hole Porfido (GHP) zone. The drill hole traces are coded by copper grade.

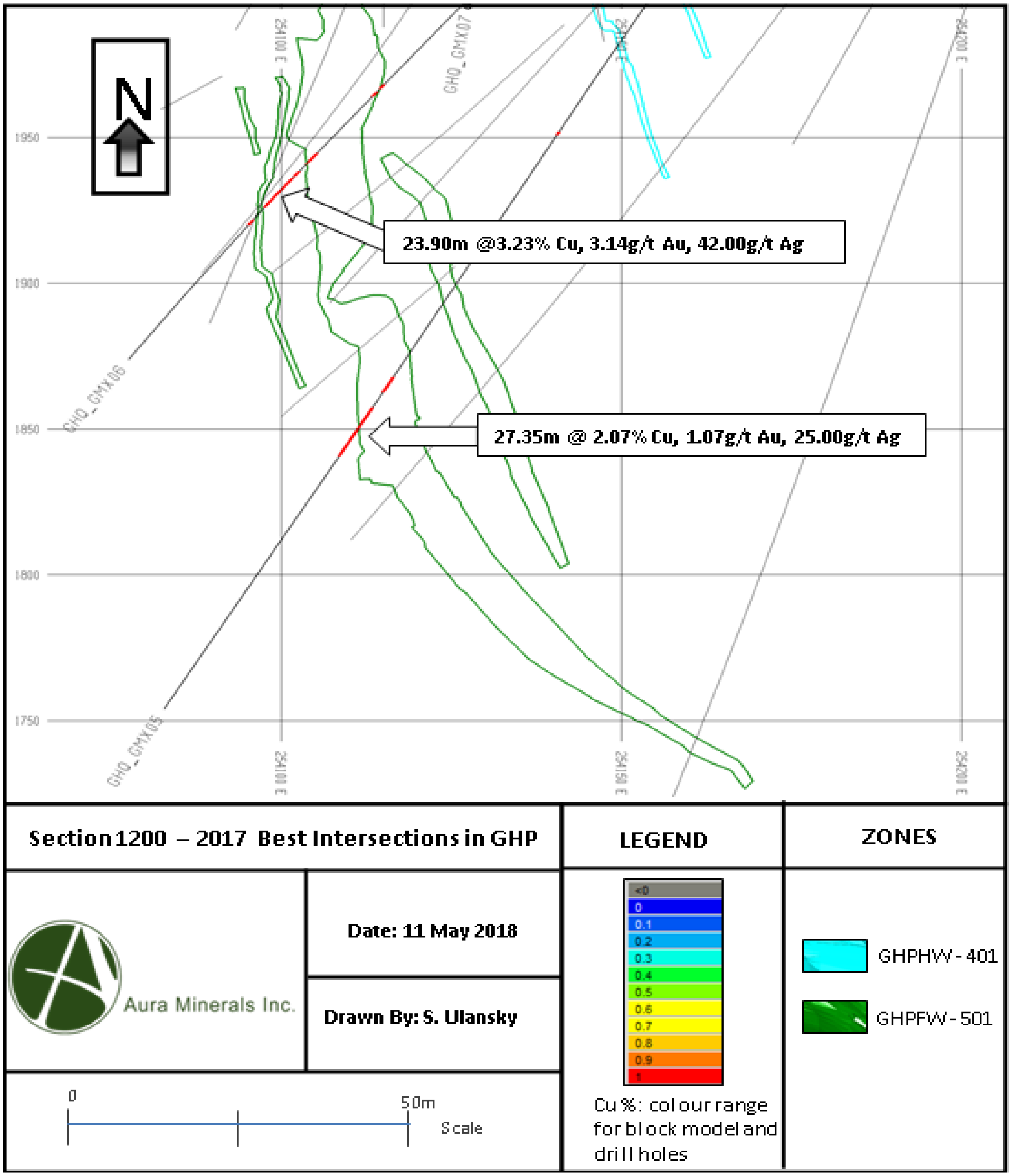
Table 10-5 Geotechnical UG Drill Hole Collar Locations (2017)

Drill Hole ID	Collar					
	North	East	Elevation	Azimuth	Dip	Length
GHP_GMX01	2,723,907	254,508	2,002	217	-25	71
GHP_GMX02	2,723,929	254,434	2,002	201	-48	251
GHP_GMX03	2,723,950	254,384	2,002	203	-41	186
GHP_GMX04	2,723,977	254,340	2,001	200	-70	315
GHP_GMX05	2,724,015	254,252	2,012	202	-59	231
GHP_GMX06	2,724,033	254,127	1,999	204	-51	150
GHP_GMX07	2,724,036	254,126	1,999	278	-64	200

Table 10-6 Selected Significant Intercepts (2017)

Drill Hole ID	Mineralized Interval (m)			Assay Results		
	From	To	Interval	Cu (%)	Au (g/t)	Ag (g/t)
D-1963-100	434.2	448	13.8	1.05	0.94	17.1
D-1963-100	457.2	459.8	4.15	2.51	2.54	32.1
GHP-GMX 01	26.75	35.6	4.3	0.85	0.80	21.3
GHP-GMX 02	183.9	209.45	25.6	1.47	1.29	18.6
GHP-GMX 02	218.55	227.6	9.1	1.33	0.67	13.1
GHP-GMX 03	11.65	16.2	4.55	1.86	1.05	31.0
GHP-GMX 03	124.4	140.8	16.4	3.38	3.28	45.6
GHP-GMX 04	55.55	68.2	12.7	3.24	1.64	67.5
GHP-GMX 04	72.65	80.35	7.7	1.03	0.48	12.6
GHP-GMX 04	199.55	248.1	58.6	1.75	1.05	21.8
GHP-GMX 05	167.2	173.4	6.2	1.65	0.95	29.4
GHP-GMX 05	179.45	198.95	19.5	2.44	1.12	25.5
GHP-GMX 06	69.9	94.1	28.4	3.06	2.98	39.7
GHP-GMX 07	160.3	175.45	15.2	1.17	1.12	13.9

Figure 10-4 Section 1975, Glory Hole Porfido (GHP) Area Showing the UG Significant Intersections (2012 to 2014)



11. SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 INTRODUCTION

Aura Minerals has implemented its own sample preparation, analysis and security protocols for the Aranzazu Property. Methods and protocols were previously discussed in the 2011 Technical Report but will be discussed here to reflect any changes since the last 2011 Technical Report was published.

11.2 AURA MINERALS SAMPLING METHOD AND APPROCH

11.2.1 Core Drilling

Core samples are placed into plastic core boxes, labelled with the drill hole and box numbers. The core boxes are collected on a regular basis from the UG drill rig and transported to the core shack facility by Aura Minerals’ employees, to conduct logging and splitting. The core boxes are labelled with the start and end of the interval for that box and then photographed. Geotechnical logging is carried out, including measurements of total core recovery (TCR), rock quality designation (RQD), rock mass quality (Q), rock mass rating (RMR) and intact rock strength (IRS).

In general, the core recovery is good, averaging 96%. There are some areas that show core recovery less than 90%. The RQD averages 76%, indicating that somewhat less than one-quarter of the recovered drill core is broken into pieces less than 2 1/2 times the core diameter. Table 11-1 shows a summary of the average core recovery and RQD by lithology code. The hornfels (Hfl) lithology has the poorest average RQD at 66.6%, with the skarn lithologies (XSk) and (ESk) having average RQDs of 75.3% and 77.1%, respectively. Thus, on average, the hornfels and skarn lithologies are more broken than the marble/limestone and porphyry lithologies.

Table 11-1 Average Core Recovery and RQD for Aura Mineral Drill holes by Lithology Code**

Lithology	Lithology Code	Lithology ID	Number of Data	Drill Hole Recovery (%)	Number of Data	Rock Quality (RQD %)
Breccia	Brxx	1	38	*	38	*
Clastic Sedimentary	Css	3	10	*	10	*
Limestone	Lms	4	1,408	96.2	1,407	81.9
Marble	Mbl	5	5,560	95.1	5,556	78.6
Hornfels	Hfl	6	6,601	96.3	6,550	66.6
Exoskarn	Xsk	7	7,268	94.1	7,256	75.3
Endoskarn	Esk	8	9,217	96.6	9,191	77.1
Plagioclase, biotite, qtz +/- hornblende	PBQA	9	4,441	96.3	4,441	79.3
Porphyritic texture - plagioclase, biotite, K-feldspar, quartz	PPBF	10	4,811	96.8	4,802	77.7
Total			**9,306**	**95.9**	**39,203**	**75.7**

*There is insufficient data to include RQD and Recovery for Brxx and Css
**Data provided by Aura Minerals Inc.

Drill core was split using a diamond saw blade. Half the core was placed in a numbered bag along with corresponding sample number for shipment to the lab. The remaining half of the core was placed back into the core box and retained at a location off-site in a secure core shed facility. The core pans were cleaned between each split sample to avoid any contaminations from previous samples. A geologist responsible for logging was monitoring the core cutting to ensure that the samples were correctly taken.

Samples are typically 1.5 m in length with a minimum sample length of 0.1 m. The fixed 1.5 m length samples are not locally adjusted for lithology or alteration. However, visually barren hanging wall rock is left un-sampled.

A core duplicate sample is taken by quartering one half of the core and putting each quarter into different numbered bags which are then sealed. The remaining quarter core is put back into the box. The labelled and sealed bags are put into large sacks that are sealed and the sacks are sent to the preparation laboratory in Zacatecas using secure transport.

Aura Minerals systematically inserted certified standard samples, sample duplicates and blank samples in a non-sequential order in all batches of drill core samples sent for assay.

The significant core drilling intersections is included in Item 10 for 2012-2014 drilling.

11.2.2 Density Measurements

This section describes the methods that the exploration department at the Aranzazu mine uses for measuring bulk density:

Samples that represent both ore and waste material are selected from the drill hole. A piece of core approximately 10cm in length is selected and its depth and dry weight are recorded. The core is then sealed in lacquer to ensure that the rock is impermeable to water. The core is submerged in water and the weight is recorded. The calculation used is:

$$\text{Density (g/cm}^3\text{)} = \frac{\text{dry sample}}{\text{(dry sample – wet sample)}}$$

It was recommended that Aura Minerals add additional steps after measuring wet density by placing samples in an oven for 12 hours to calculate moisture content in future bulk density measurements.

11.2.3 QP Observations During the 2014 Site Visit

During December of 2014, Farshid Ghazanfari, P.Geo. Geological QP, the Author of this report observed core logging and sampling practice at the Aranzazu Mine. The QP was satisfied with implemented procedures for core handling and sampling. In the Author's opinion, the level of information that was collected during logging is for interpretation and resource estimation practices. However, additional attention and time needs to be spent on logging of structural signatures in the core including measuring and recording of fault zones. The core shack facility, which is located off-site, is secure and is monitored by security personal after working hours. However, the core shack facility seems to have reached its maximum capacity and for any future drilling, the company needs to expand or build a new core shack.

Many selected core boxes from 2012 to 2014 drilling campaign were observed and the core logs were reviewed by the author during the site visit. Based on this observation the author believes that core recovery is usually better than 98% due to extra care implemented in sampling and logging. The author has no reason to believe that samples are not representative, nor are there any indications of sampling bias.

Based upon both the Author's observations during the site visits and close collaboration of the Author with Aura Minerals geologists, it can be concluded that Aura Minerals' sampling method and approach follows the CIM exploration best practices guidelines issued in 2000.

11.3 AURA MINERALS SAMPLE PREPARATION, ANALYSIS AND SECURITY PROTOCOLS

Aura Minerals has used two certified commercial laboratories to assay the Aranzazu drill cores for copper, gold and silver. Since 2012 drill hole samples were shipped to the SGS lab in Durango, Mexico as the primary lab for assaying:

SGS de Mexico, S.A. de C.V.,
Antimonio #121
Cd. Industrial

Durango, Durango, México C.P. 34208

The half core samples are weighed, dried and then crushed to 75.0% passing 2.0 mm, split to 250 g and pulverized to 85.0% passing 75 microns. Gold assays were determined by fire assay with an AA finish; over limit assays were determined by fire assay with a gravimetric finish. Silver assays were determined by three acid digestions with an AA finish; over limit assays were determined by fire assay with a gravimetric finish. All samples were analyzed by inductively coupled plasma mass spectrometry for multi-element analysis.

The assay protocol for SGS lab as follows:

ICP40B: 32 elements by four-acid digestion/ICP-AES finish (recommended for Ag, Cu, Zn, Fe, Mo, Pb, As, Bi, Sb)

- If Ag >100 g/t then use AAS21E (2g, 3-acid digest, AAS finish)
- If Ag >300 g/t then use FAG313 (FA/ Gravimetric finish)
- If As >1% then use ICP90Q (ore grade sodium peroxide fusion, ICP-AES)
- ICP14B: 2% check on Hg ONLY

Table 11-2 ICP40B-detection limits

ICP40B

32 ELEMENTS BY FOUR-ACID DIGESTION / ICP-AES

	LIMITS				LIMITS				LIMITS		
Ag	2ppm	-	10ppm	Fe	0.01%	-	15%	Sb	5ppm	-	1%
Al	0.01%	-	15%	K	0.01%	-	15%	Sc	0.5ppm	-	1%
As	3ppm	-	1%	La	0.5ppm	-	1%	Sn	10ppm	-	1%
Ba	1ppm	-	1%	Li	1ppm	-	1%	Sr	0.5ppm	-	1%
Be	0.5ppm	-	0.25%	Mg	0.01%	-	15%	Ti	0.01%	-	15%
Bi	5ppm	-	1%	Mn	2ppm	-	1%	V	2ppm	-	1%
Ca	0.01%	-	15%	Mo	1ppm	-	1%	W	10ppm	-	1%
Cd	1ppm	-	1%	Na	0.01%	-	15%	Y	0.5ppm	-	1%
Cr	1ppm	-	1%	Ni	1ppm	-	1%	Zn	1ppm	-	1%
Co	1ppm	-	1%	P	0.01%	-	15%	Zr	0.5ppm	-	1%
Cu	0.5ppm	-	1%	Pb	2ppm	-	1%				

The following elements can be added to above package:
Ce, Hf, Nb, Rb, Se, Te, Th, U

FAA313: 30 g fire assay, AAS finish Au (reporting limits 0.01 to 10.0 g/t)

- If Au >10.0 g/t then use FAG303 (FA/Gravimetric finish)

ICP90Q: Sodium peroxide fusion/ICP-AES finish (recommended for <u>ALL Cu >1.0%</u> and <u>ALL Fe > 15.0%</u> and <u>ONLY 2.0% for As <1.0%)</u>

Table 11-3 ICP90A-detection limits

ICP90A

SODIUM PEROXIDE FUSION / ICP-AES

	LIMITS		LIMITS		LIMITS
Al	0.01% - 25%	K	0.1% - 25%	Sc	5 ppm - 5%
As	30 ppm - 10%	La	10 ppm - 10%	Sn	50 ppm - 5%
Ba	10 ppm - 10%	Li	10 ppm - 10%	Sr	10 ppm - 1%
Be	5 ppm - 2.5%	Mg	0.01% - 30%	Ti	0.01% - 25%
Ca	0.1% - 35%	Mn	10 ppm - 10%	V	10 ppm - 5%
Cd	10 ppm - 5%	Mo	10 ppm - 10%	W	50 ppm - 5%
Cr	10 ppm - 10%	Ni	10 ppm - 10%	Y	5 ppm - 5%
Co	10 ppm - 10%	P	0.01% - 25%	Zn	10 ppm - 10%
Cu	10 ppm - 10%	Pb	20 ppm - 10%		
Fe	0.01% - 30%	Sb	50 ppm - 10%		

AAS71D: Cu oxide method, citric acid leach, AAS (recommended for CuO; samples sent to Vancouver)

Crush/Coarse duplicates are inserted at the first stage of crushing of the split drill core at a rate of approximately 2.0% of the total samples (= DUP in assay certificate).
Pulp duplicates are inserted at the final pulverizing stage at a rate of approximately 6.0% of the total samples (= REP in assay certificate).

The assay lab method and detection limits for primary lab are summarized in Table 11-4

Table 11-4 Assaying methods and Lab QA/AC (Primary Laboratory-SGS) Summary

Element	SGS Code	Digestion	Finish	Reporting limits	Overlimits				QAQC Check
					SGS Code	Digest	Finish	Reporting Limits	
Ag	ICP40B	4 acid	ICP-AES	2-100 ppm	AAS21E	3-acid	AAS	100-300 ppm	10% pulps to ALS
					FAG313	Fire assay	Gravimetry	>300 ppm	10% pulps to ALS
Au	FAA313	Fire assay	AAS	0.01-10 ppm	FAG303	Fire assay	Gravimetry	>10 ppm	10% pulps to ALS
As	ICP40B	4 acid	ICP-AES	35-10,000 ppm	ICP90Q	SPF	ICP-AES	1-20%	2% using method ICP90Q (for As <1%); 10% pulps to ALS
Bi	ICP40B	4 acid	ICP-AES	30-10,000 ppm	The lab will inform if there are samples >1% and then appropriate overlimit methods will be selected				10% pulps to ALS
Cu	ICP40B	4 acid	ICP-AES	2-10,000 ppm	ICP90Q	SPF	ICP-AES	>1%	10% pulps to ALS
Fe	ICP40B	4 acid	ICP-AES	0.05-15,000 ppm	ICP90Q	SPF	ICP-AES	>15%	10% pulps to ALS
Hg	ICP14B	Aqua regia	ICP-AES	5-10,000 ppm	-				-
Mo	ICP40B	4 acid	ICP-AES	2.5-10,000 ppm	The lab will inform if there are samples >1% and then appropriate overlimit methods will be selected				10% pulps to ALS
Pb	ICP40B	4 acid	ICP-AES	10-10,000 ppm	The lab will inform if there are samples >1% and then appropriate overlimit methods will be selected				10% pulps to ALS
Sb	ICP40B	4 acid	ICP-AES	10-10,000 ppm	The lab will inform if there are samples >1% and then appropriate overlimit methods will be selected				10% pulps to ALS
Zn	ICP40B	4 acid	ICP-AES	10-10,000 ppm	The lab will inform if there are samples >1% and then appropriate overlimit methods will be selected				10% pulps to ALS
CuO	AAS71D	Citric acid leach	AAS	>0.002	-				-

*SPF: Sodium peroxide fusion

Since 2012 drill hole samples were shipped to the ALS lab in Zacatecas in Mexico as a secondary quality control laboratory for checking assays and quality control:

Transito Pesado S/N Bodega #100,200,300 y 400
Col Lomas de la Isabelica
Zacatecas, Zac. 98099

The assay protocol for ALS lab is as follows:

PREP-31: Crush to 70.0% less than 2mm, riffle split off 250 g, pulverize split to better than 85.0% passing 75 microns.

ME-ICP61: 48 elements by four-acid ICP-AES and ICP-MS for Ag, As, Bi, Cu, Fe, Mo, Pb, Sb, Zn

- If Ag >100 ppm then use ME-GRA21 (FA/Gravimetry)
- If Cu, Mo, Pb, Zn >9,000 ppm then use (+)-OG62
- If As, Bi, Sb exceed 10,000 ppm - Reprocessed with an appropriate overlimit method

CU OG62: Four-acid digestion with ICP-AES finish overlimit method for Cu, Mo, Pb and Zn. Use only when elements are >9,000 ppm (creates a buffer around 1.0% just in case SGS results are slightly above 1.0% and ALS results are slightly below 1.0%)

AU-AA25: fire assay fusion and Atomic Absorption Spectroscopy (AAS) finish

- If Au >100 g/t then use Au-GRA21 (Fire assay/ Gravimetry)

ME-GRA21: Fire assay and gravimetric finish (30g nominal sample weight) when Ag >100 ppm

Au-GRA21: Fire assay and gravimetric finish (30g nominal sample weight) when Au >100 ppm

CU-AA05: 5.0% sulphuric acid leach and Atomic Absorption Spectroscopy (AAS) finish

Chain of custody of drill samples is maintained throughout the process with the use of numbered seal tags for closing sample bags and third-party professional transportation of samples to the laboratories. Each stage of sample handling is recorded in log sheets and receipts obtained from each party involved.

Table 11-5 Assaying methods and lab QA/QC (Secondary Laboratory-ALS) Summary

Element	ALS Code	Digestion	Finish	Reporting limits	Overlimits				Comments
					ALS Code	Digest	Finish	Reporting Limits	
Ag	ME-ICP61	4 acid	ICP-AES	0.5-100 ppm	ME-GRA21	Fire assay	Gravimetry	100-10,000 ppm	-
Au	Au-AA25	Fire assay	AAS	0.01-100 ppm	Au-GRA21	Fire assay	Gravimetry	100-1,000 ppm	-
As	ME-ICP61	4 acid	ICP-AES	5-10,000 ppm	As-OG62	4 acid	ICP-AES	1-30%	Note that the overlimit method will be effective from >9,000ppm
Bi	ME-ICP61	4 acid	ICP-AES	2-10,000 ppm	Bi-OG62	4 acid	ICP-AES	1-30%	Note that the overlimit method will be effective from >9,000ppm
Cu	ME-ICP61	4 acid	ICP-AES	1-10,000 ppm	Cu-OG62	4 acid	ICP-AES	1-40%	Note that the overlimit method will be effective from >9,000 ppm
Fe	ME-ICP61	4 acid	ICP-AES	0.01-50%	-	-	-	-	-
Mo	ME-ICP61	4 acid	ICP-AES	1-10,000 ppm	Mo-OG62	4 acid	ICP-AES	1-10%	Note that the overlimit method will be effective from >9,000 ppm
Pb	ME-ICP61	4 acid	ICP-AES	2-10,000 ppm	Pb-OG62	4 acid	ICP-AES	1-20%	Note that the overlimit method will be effective from >9,000 ppm
Sb	ME-ICP61	4 acid	ICP-AES	5-10,000 ppm	-	-	-	-	-
Zn	ME-ICP61	4 acid	ICP-AES	2-10,000	Zn-OG62	4 acid	ICP-AES	1-30%	Note that the overlimit method will be effective from >9,000 ppm
CuO	Cu-AA05	5% sulphuric acid leach	AAS	0.001-10%	-	-	-	-	-

11.4 AURA MINERALS QUALITY ASSURANCE/ QUALITY CONTROL (QA/QC)

Field (core) duplicates are inserted by geologists randomly at the Aranzazu exploration site and account for approximately 6.0% of the total samples sent to the laboratory. Field duplicates are known as ‘duplicates’ or ‘TS’ in the exploration database. The field duplicates test how systematic and unbiased the sampling is done thereby testing the precision of the core cutters.

Coarse (crushed) duplicates are inserted by the lab at the first stage of crushing of the split drill core at a rate of approximately 2.0% of the total samples. The crush duplicate code in the assay certificates is 'DUP'. The coarse duplicates test how systematic and unbiased the sampling is done at the crushing stage thereby testing the precision of the sample preparation.

Pulp duplicates are inserted by the lab at the final pulverizing stage at a rate of approximately 6.0% of the total samples. The pulp duplicate code in the assay certificates is 'REP'. The pulp duplicates test how systematic and unbiased the sampling is at the pulp stage thereby testing the precision of the laboratory analytical equipment.

Blank material is collected by hand samples in the mine from visually barren limestone, marble and intrusive rock. The blank samples are crushed and homogenized and then packed into individual bags. The blank samples using this method were found to be too high in Cu grade, therefore, the samples will soon be collected from a nearby barren limestone quarry. The blank samples from the barren limestone quarry will first be tested in the production laboratory and if determined to be <20 ppm for Cu, will be used as the blank material in the exploration sample stream.

Quality control samples were inserted into the sample stream and account for approximately 15.0% of the total samples sent to the laboratory, with the breakdown being 4% core duplicates, 3% blanks, 4.0% standard reference material, and 5.0% core duplicates that are sent to ALS Laboratories.

This section will only discuss the QA/QC program for the 2011 to 2014 period, since the previous Technical Reports covered the QA/QC for the earlier exploration periods.

11.4.1 Standard Reference Materials

Aura Minerals has used three commercial SRMs (Standard Reference Materials) to assess the accuracy and precision of the drill hole assays. All of the SRMs were purchased from CDN Resource laboratories Ltd, of Vancouver, Canada. Table 11-6 is a list of the SRMs used at Aranzazu.

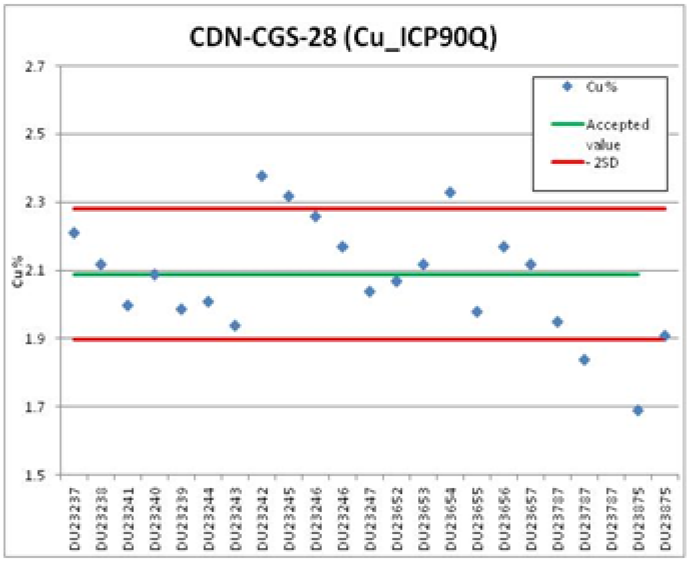
Table 11-6 Certified Standard Reference Material Information

Reference Material ID	Copper Value (%)		Gold Value (g/t)		Silver Value (g/t)	
	Accepted assay	2 SD's	Accepted assay	2 SD's	Accepted assay	2 SD's
CDN-CM-13	0.786	+/- 0.036	0.740	+/- 0.094	-	-
CDN-CGS-28	2.089	+/- 0.096	0.727	+/- 0.076	-	-
CND-CM-24	0.365	+/- 0.020	0.521	+/- 0.056	4.1	0.4

SRMs at the Aranzazu exploration site are inserted into the sample stream in prepackaged pouches each weighing 75 grams. The three standards account for approximately 4.0% of the total samples sent to the laboratory.

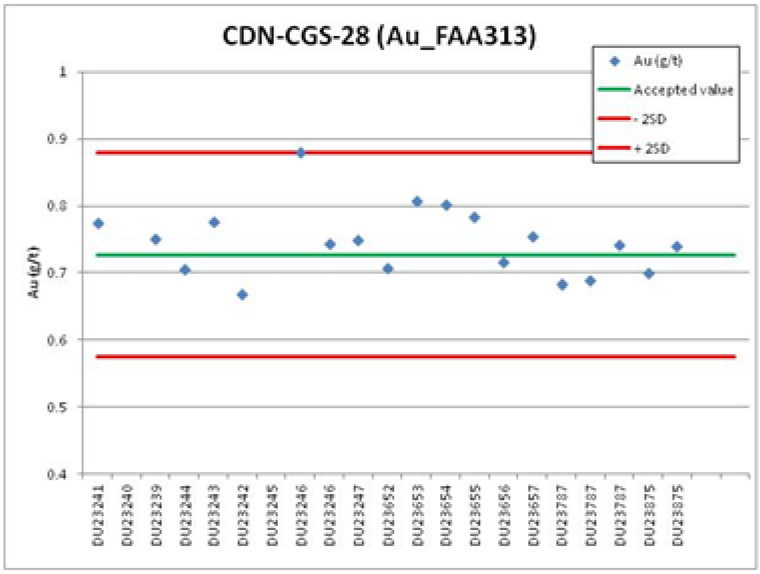
The certified method ICP90Q is the preferred overlimit method for Cu >1.0% and Fe >15.0%. Method ICP90Q was tested against standard CDN-CGS-28 and the results fall within the accepted tolerance limits and appear unbiased (Figure 11.1). These results demonstrate that SGS's method ICP90Q is reliable and accurate.

Figure 11.1 Control Charts of SRM CDN-CGS-28 for Cu using method ICP90Q



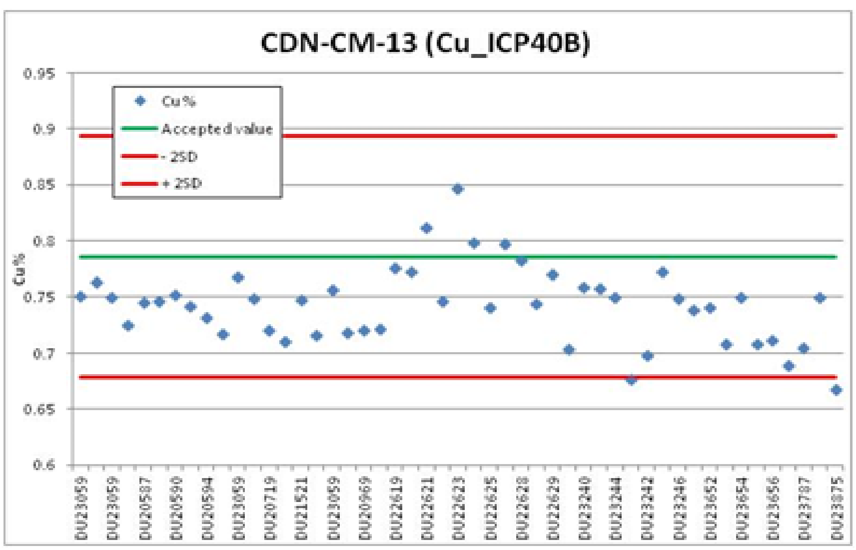
The certified method FAA313 is the preferred method of analysis for Au. Method FAA313 was tested against standard CDN-CGS-28 and the results all fall within the accepted tolerance limits and appears unbiased (Figure 11.2). These results demonstrate that SGS's method FAA313 is reliable and accurate.

Figure 11.2 Control Charts of SRM CDN-CGS-28 for Au using method FAA313



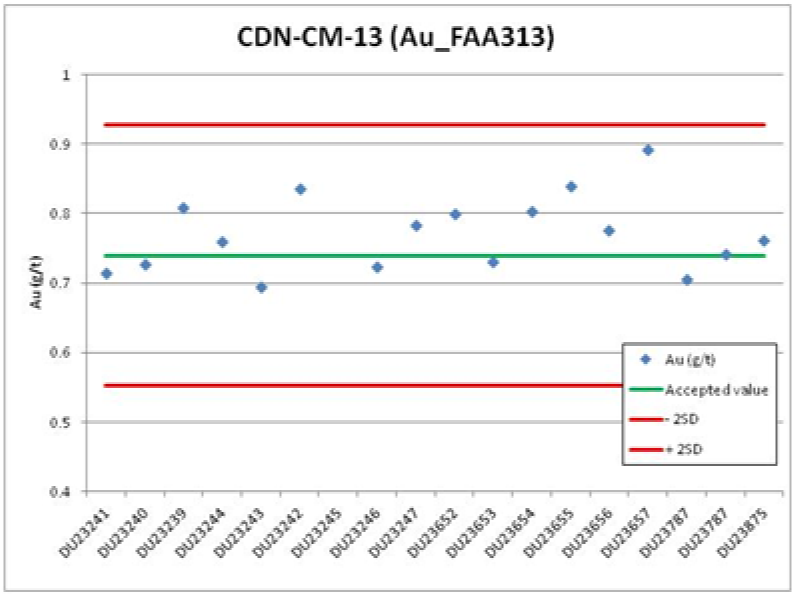
The results for copper using standard CDN-CM-13 and SGS method ICP40B fall within the accepted tolerance limits, although a bias towards the low grade values exists for the majority of the samples (Figure 11.3). The certified reference material CDN-CGS-13 is certified for a 4-acid digests, which correlates with SGS method ICP40B which also uses a 4-acid digest. These results should be equivalent because the same digest and finish methods have been used, but since they are not, the precision of ICP40B was questioned. Aura Minerals has examined the internal QAQC at SGS for ICP40B and the results are deemed acceptable.

Figure 11.3 Control Charts of SRM CDN-CM-13 for Cu using method ICP40B



The SRM CDN-CM-13 test performed very well using method FAA313 as all the samples occur within the accepted tolerance limits and the samples are unbiased (Figure 11.4).

Figure 11.4 Control Charts of SRM CDN-CM-13 for Au using method FAA313



In November 2013 an additional SRM (CDN-CM-24) was included into the sample stream. The new standard tests the low to medium grade Au and Cu range as the other two standards test the medium to high grade Au and Cu range. Silver is also certified in CDN-CM-24; the only standard that validates Ag at a low-grade range.

All of the Au samples using method FAA313 fall within the accepted tolerance limits and are spread evenly about the accepted value (Figure 11.5). The Ag results are similarly evenly distributed and contained within the accepted tolerance limits (Figure 11.6). There are, however, a few outlier values that exceed the upper tolerance limits. These results are not of a concern since the lower detection limit for Ag is <2 g/t, whereas the sensitivity of CDN-CM-24 is to 0.1 g/t. The differing of decimal places close to the lower detection limit of method ICP40B for Ag accounts for the outlier values. Copper samples using method ICP40B fall within the accepted tolerance limits (Figure 11.7).

Figure 11.5 Control Charts of SRM CDN-CM-24 for Au using method FAA313

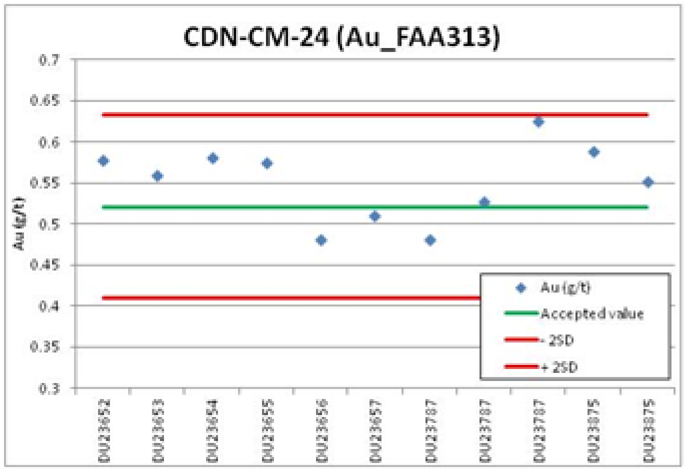


Figure 11.6 Control Charts of SRM CDN-CM-24 for Ag using method ICP40B

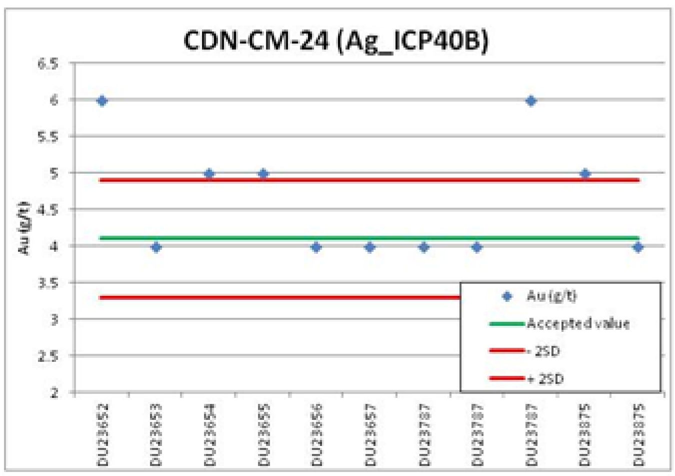
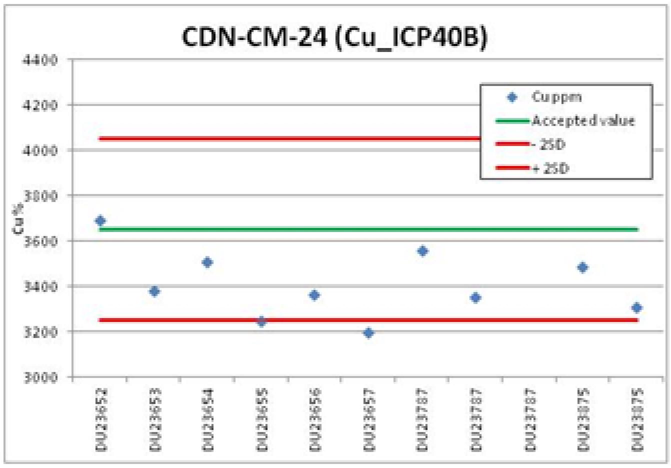


Figure 11.7 Control Charts of SRM CDN-CM-24 for Cu using method ICP40B



11.4.2 ALS Global Independent Analysis of Pulp Samples

Thirty (30) standard pulps were sent to ALS in January 2014. The reasons for processing these results are twofold: (i) to validate the precision and accuracy of the methods and results at ALS with standard reference material, and (ii) to cross-check SGS's method ICP40B with the equivalent method at ALS because ICP40B was seen to have a low bias in the 2013 assay certificates.

The precision and accuracy of the assay results at ALS Global were found to be good. Figures 11-8 to 11-9 are control charts for the standard CDN-CGS-28. Both copper and gold results using methods Cu_OG62 and Au_AA25, respectively, fall within the accepted tolerance limits and are not biased. The copper method OG62 is done with a 4-acid digest and ICP-AES finish, which is equivalent to the certified methods for copper in CDN-CGS-28. The gold method AA25 is done with a fire assay digest and AAS finish, which is equivalent to the certified methods for Au in CDN-CGS-28.

Figure 11.8 Control Charts of SRM CDN-CGS-28 for Cu using over limit method OG62

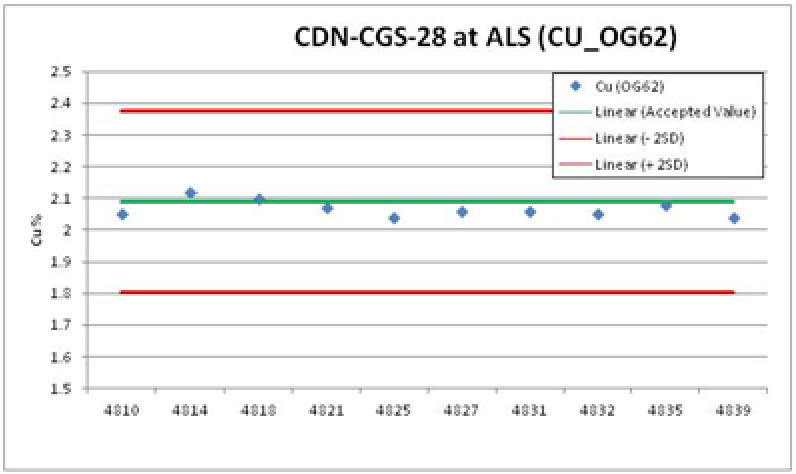
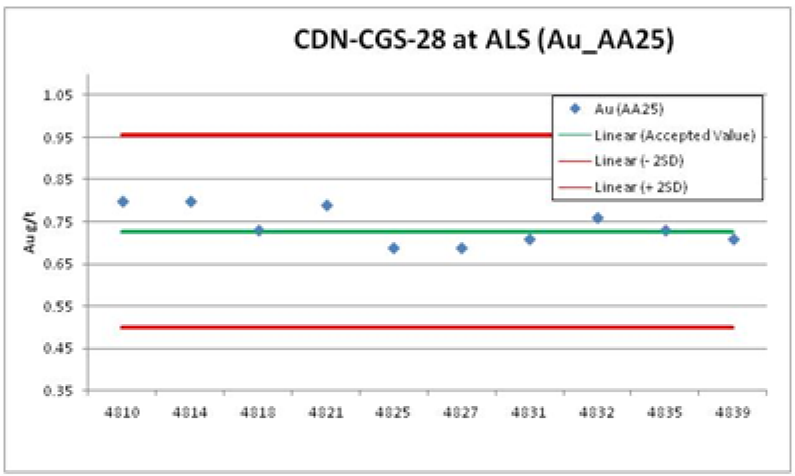


Figure 11.9 Control Charts of SRM CDN-CGS-28 for Au using method Au_AA25



Figures 11-10 and 11-11 are control charts for the standard CDN-CM-13. Both copper and gold results using methods Cu_ME-ICP61 and Au_AA25, respectively, fall within the accepted tolerance limits. The gold results are not biased, whereas the copper results are slightly biased towards the low grade values. The copper method ME-ICP61 is done with a 4-acid digest and ICP-AES finish, which is equivalent to the certified methods for copper in CDN-CM-13. The gold method AA25 is done with a fire assay digest and AAS finish, which is equivalent to the certified methods for Au in CDN-CM-13.

Figure 11.10 Control Charts of SRM CDN-CM-143 for Cu using method ME-ICP61

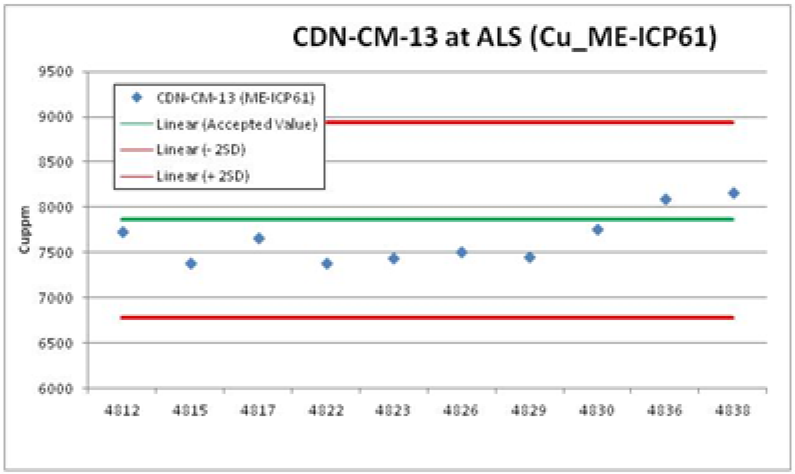
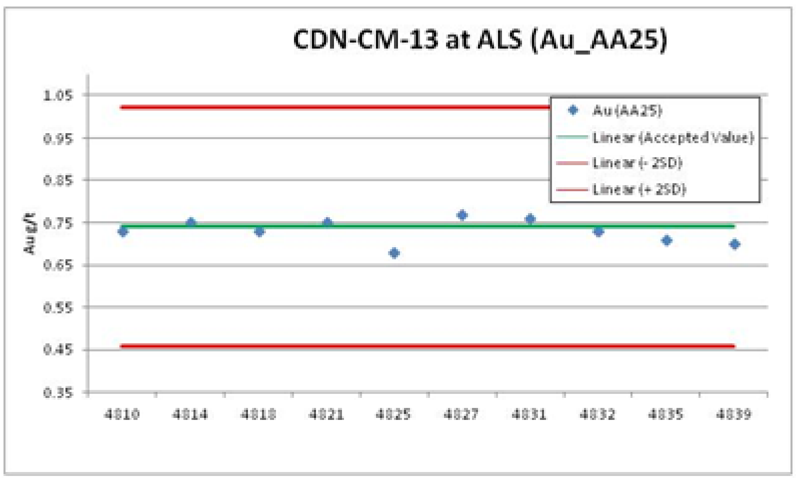


Figure 11.11 Control Charts of SRM CDN-CGS-28 for Au using method Au_AA25



Figures 11.12, 11.13 and 11.14 are control charts for the standard CDN-CM-24. Copper, gold and silver results using methods Cu_ME-ICP61, Au_AA25 and Ag_ME-ICP61, respectively, that show the results fall within the accepted tolerance limits. The gold and silver results are not biased, whereas the copper results are slightly biased towards the low grade values. The copper method ME-ICP61 is done with a 4-acid digest and ICP-AES finish, which is equivalent to the certified methods for copper in CDN-CM-24. The gold method AA25 is done with a fire assay digest and AAS finish, which is equivalent to the certified methods for Au in CDN-CM-24. The silver method ME-ICP61 is done with a 4-acid digest and ICP-AES finish, which is equivalent to the certified method or silver in CDN-CM-24. The control charts validate the methods used at ALS for copper, gold and silver.

Figure 11.12 Control Charts of SRM CDN-CM-24 for Cu using method Cu_ME-ICP61

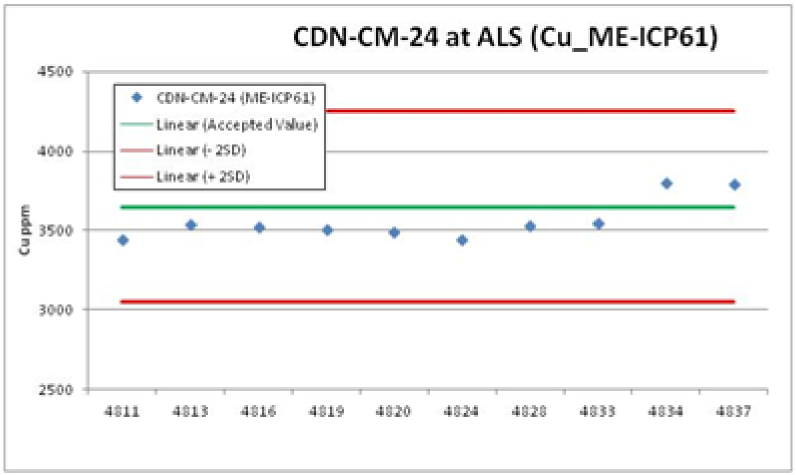


Figure 11.13 Control Charts of SRM CDN-CM-24 for Au using method Au_ME-AA25

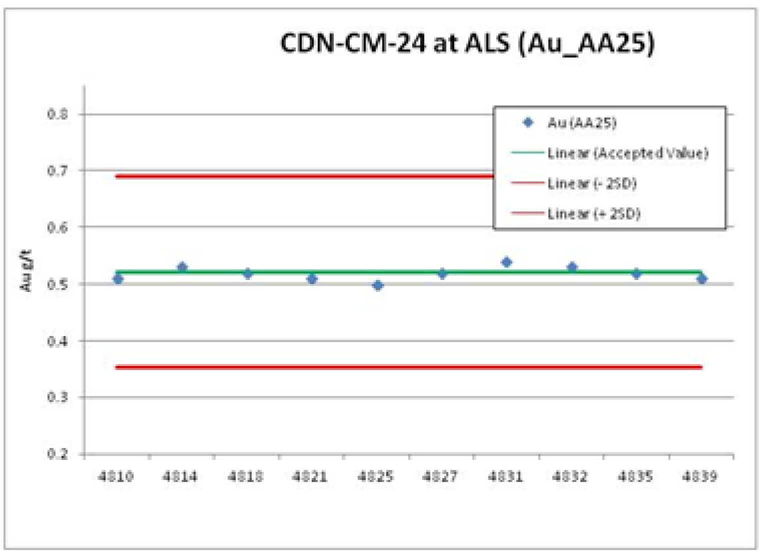
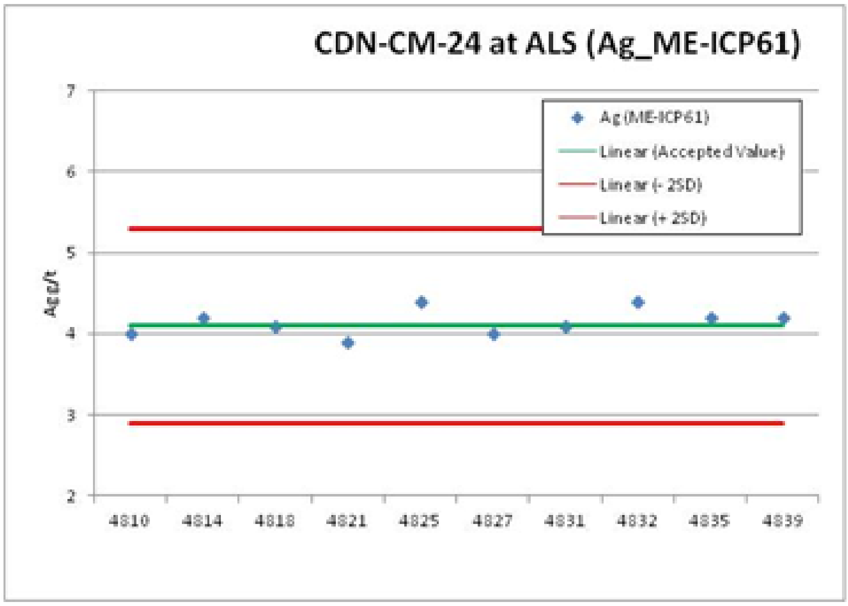


Figure 11.14 Control Charts of SRM CDN-CM-24 for Ag using method Ag ME-ICP61



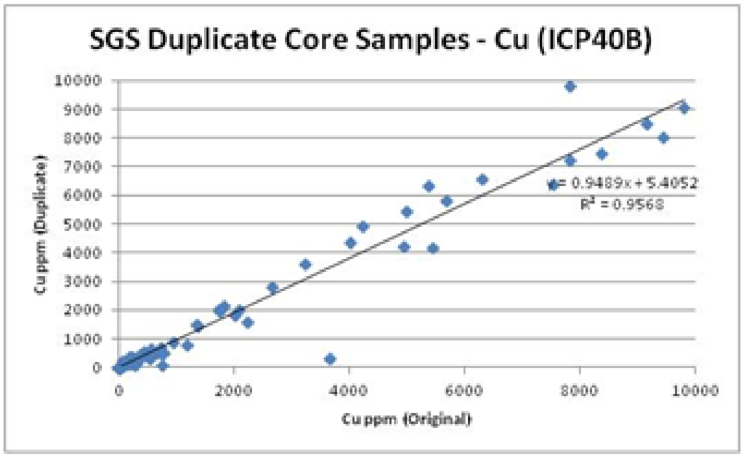
11.4.3 Core Duplicate Samples

Duplicate core samples are taken by splitting half the core and putting each quarter into separate labelled bags which are then sealed. The remaining core is put back into the box. The field duplicates test how systematic and unbiased the sampling is done thereby testing the precision of the core cutters.

Two methods have been used to analyse copper. The majority of the samples are tested using ICP40B which have an upper detection limit of 10,000 ppm. The copper results in excess of 10,000 ppm are tested with an overlimit method (ICP90Q). Figure 11.15 displays scatter plots of copper core duplicate samples for the 2012 to 2013 drilling.

The degree of correlation between the original and duplicate samples is acceptable with an R^2 value of 0.9568. The mean grades of copper using method ICP40B are 1,355 ppm and 1,291 ppm Cu for the original sample and duplicate, respectively. These grades compare fairly well, but as expected, there is some variability between original and duplicate results due to the natural variability of the mineralization.

Figure 11.15 Scatter plot of copper results for core duplicates using SGS method ICP40B



From early November 2013 copper overlimit samples were processed with method ICP90Q rather than ICP41Q. Figure 11.16 shows the resultant duplicate data set. There are only eight duplicate pairs to evaluate which decreases the likelihood of achieving meaningful statistical results. The average values are 2.501% and 2.359% for the sample and duplicate, respectively. The R^2 value is 0.8868, which is a relatively low but an acceptable result. The duplicate core samples in high grade drill core are more likely to be variable due to the 'vein-like' and irregular nature of mineralization in a skarn deposit.

Figure 11.16 Scatter plot of copper results for core duplicates using SGS method ICP90Q

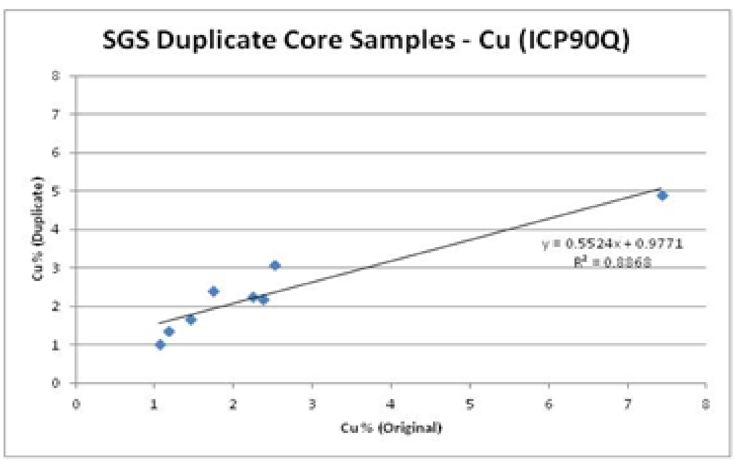


Figure 11.17 shows the duplicate data set for Ag using method ICP40B. The average values are 22.68 g/t and 24.32 g/t for the original and duplicate sample, respectively. The R^2 value is 0.8983, which is a relatively low correlation coefficient, but an acceptable value for core duplicates. With the omission of the two high grade Ag values the R^2 value improved to 0.9028. The high-grade Ag values are more variable in a piece of drill core due to the irregular nature of mineralization in a skarn deposit, thereby reducing the R^2 value significantly. High grade pulp duplicates are more likely to be similar since the rock has been homogenized several times which would reduce the natural irregularity. Figure 11.18 shows duplicate data set for Au using method FAA313, which shows a good degree of correlation (R^2 =0.9978) between the sample and duplicate pairs.

Figure 11.17 Scatter plot of silver results for core duplicates using SGS method ICP40B

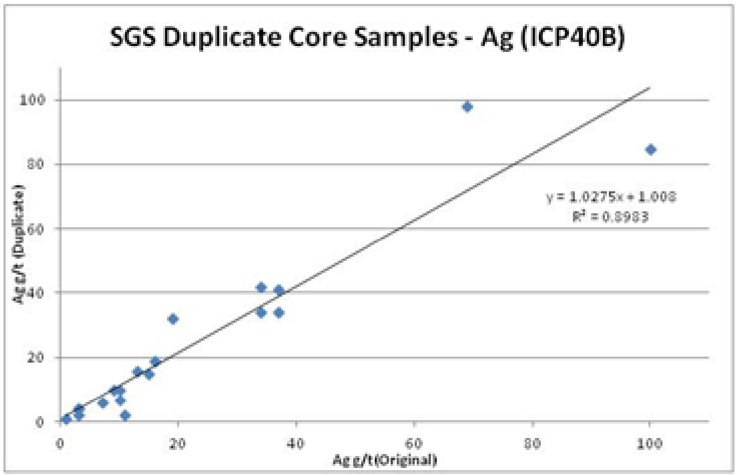
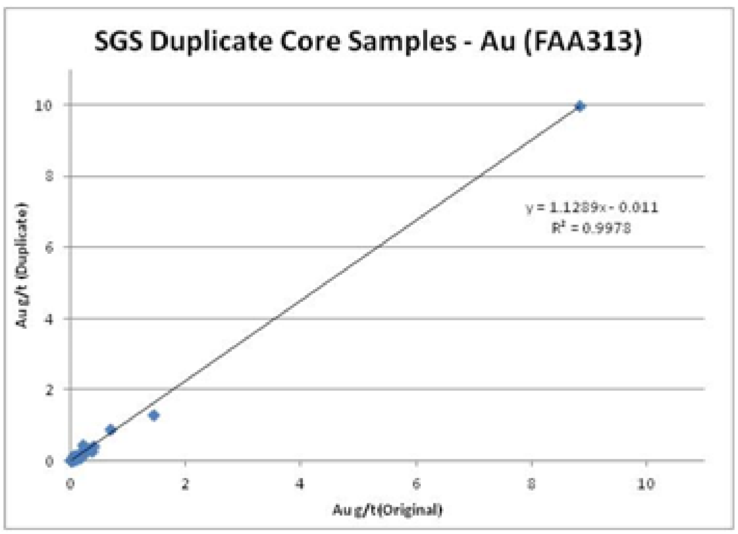


Figure 11.18 Scatter plot of gold results for core duplicates using SGS method FAA313



The mineralization at Aranzazu is non-uniform and irregular, which makes good correlation between core duplicates and their samples difficult. Gold has also been detected in free form occasionally. The absolute mean paired relative difference (AMPRD) of the core duplicate samples using a 30% tolerance limit was calculated for Cu, Au and Ag. Of the Cu pairs using method ICP40B, 74% passed, 94% of the Cu pairs using overlimit method ICP41Q passed, 75% of the Cu pairs using the overlimit method ICP90Q passed, 74% of the Ag pairs using method ICP40B passed and, 61% of the Au pairs using method FAA313 passed.

11.4.4 Crushed Duplicate Samples

Crushed or coarse duplicate samples are inserted by the lab at the first stage of crushing of the split drill core at rate of approximately 2.0% of the total samples. The crushed duplicate code in the assay certificates is DUP. The coarse duplicates test how systematic and unbiased the sampling is done at the crushing stage thereby testing the precision of the sample preparation.

Figure 11.19 displays scatter plots of gold crushed duplicate samples for the 2012 to 2013 drilling campaign using method Au_FAA313 g/t. There is good agreement between the original gold assay value and the crush duplicates with an R^2 value of 0.9992, which show that the sample preparation stage at SGS is accurate, precise and unbiased.

Figure 11.19 Scatter plot of gold results for crush duplicates using SGS method FAA313

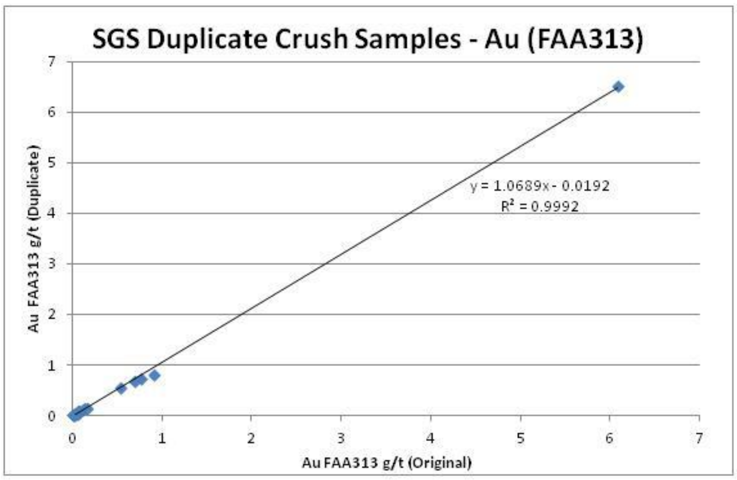
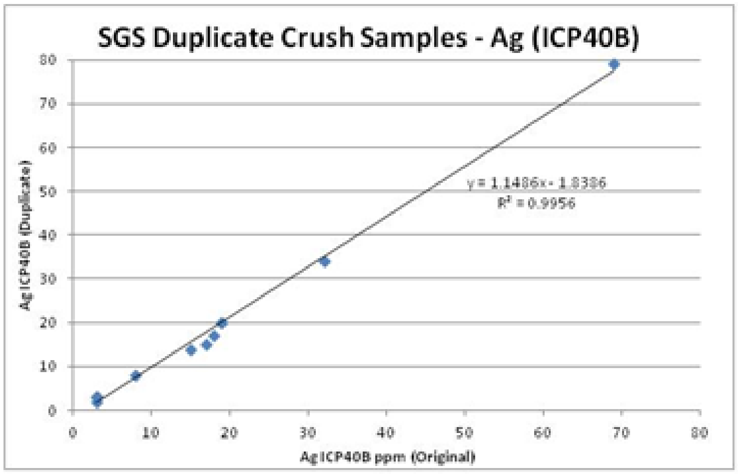


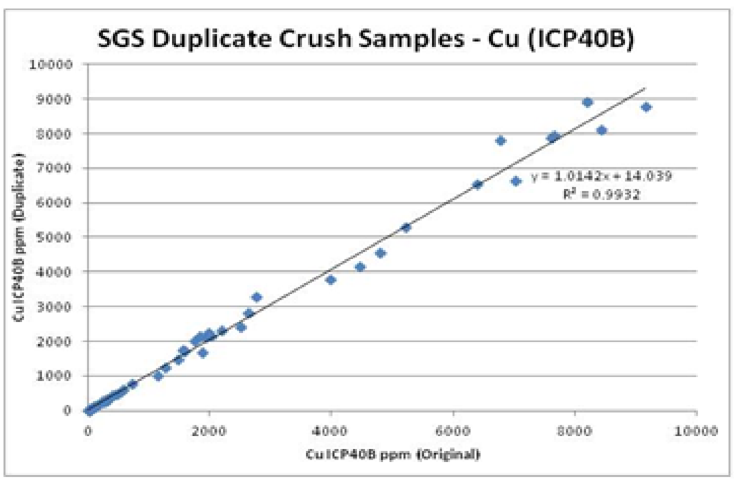
Figure 11.20 displays scatter plots of silver crushed duplicate samples for the 2012 to 2013 drilling campaign using method Ag_ICP40B g/t. There is good agreement between the original silver assay value and the crushed duplicates with an R^2 value of 0.9956, which show that the sample preparation stage at SGS is accurate, precise and unbiased.

Figure 11.20 Scatter plot of silver results for crush duplicates using SGS method ICP40B



There is good agreement between the original copper assay value and the crush duplicates with an R^2 value of 0.9932, which show that the sample preparation stage at SGS is accurate, precise and unbiased. Figure 11.21 displays scatter plots of copper crush duplicate samples for the 2012 to 2013 drilling campaign using method Cu_ICP40B_ppm.

Figure 11.21 Scatter plot of copper results for crush duplicates using SGS method ICP40B

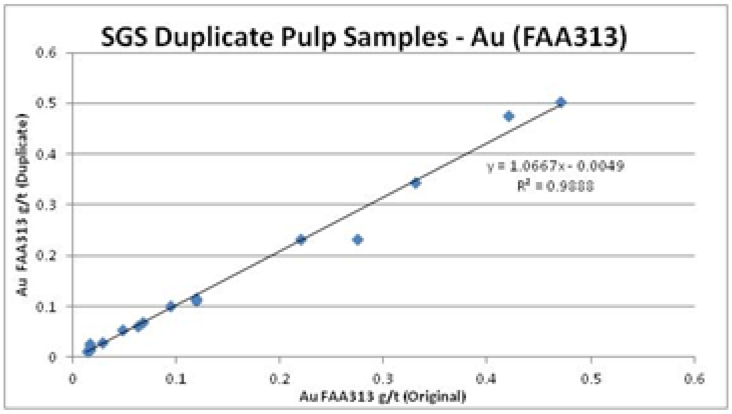


11.4.5 Pulp Duplicate Samples

Pulp duplicate samples are inserted by the lab at the final pulverizing stage at rate of approximately 8.0% of the total samples. The pulp duplicate code in the assay certificates is REP. The pulp duplicates test how systematic and unbiased the sampling is at the pulp stage thereby testing the precision of the analytical equipment.

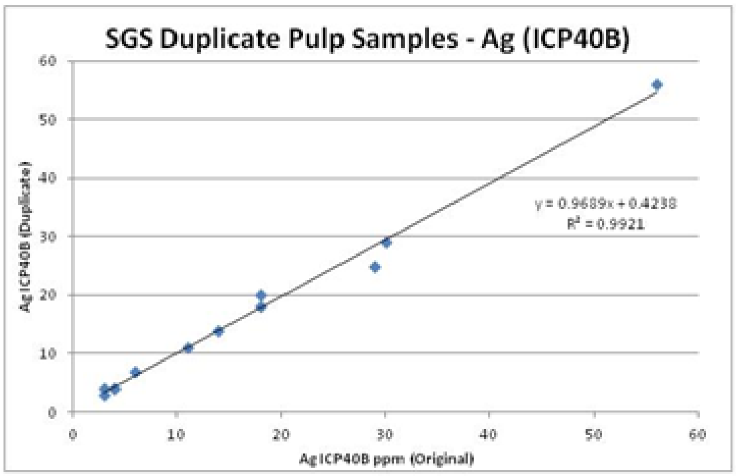
There is good agreement between the original gold assay value and the pulp duplicates with an R^2 value of 0.9888, which show that the analytical stage at SGS is accurate, precise and unbiased. Figure 11.22 displays scatter plots of gold pulp duplicate samples for the 2012 to 2013 drilling campaign using method Au_FAA313_g/t.

Figure 11.22 Scatter plot of gold results for pulp duplicates using SGS method FAA313



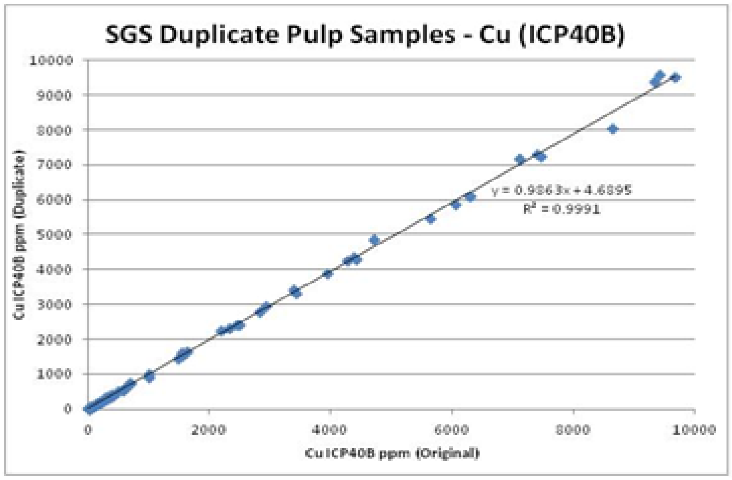
There is good agreement between the original silver assay value and the crush duplicates with an R^2 value of 0.9921, which show that the sample preparation stage at SGS is accurate, precise and unbiased. Figure 11.23 displays scatter plots of silver pulp duplicate samples for the 2012 to 2013 drilling campaign using method Ag ICP40B g/t.

Figure 11.23 Scatter plot of silver results for crush duplicates using SGS method ICP40B



There is good agreement between the original copper assay value and the pulp duplicates with an R^2 value of 0.9991, which show that the analytical stage at SGS is accurate, precise and unbiased. Figure 11.24 displays scatter plots of copper pulp duplicate samples for the 2012 to 2013 drilling campaign using method Cu ICP40B ppm.

Figure 11.24 Scatter plot of copper results for pulp duplicates using SGS method ICP40B



11.4.6 Blank Samples

Three percent (3%) of the total samples sent to the lab are inserted as blank material. The blank material is collected from limestone, marble and intrusive lithologies from hand samples that are believed to be barren. The samples are ground and homogenized several times and packed into individual bags.
The acceptable maximum values were established at 10% above the detection limits, except for Cu which was evaluated differently (Table 11-7). The acceptable maximum value for Cu was quantitatively determined by calculating the 90th percentile within WCODE 10 of the May 2013 resource model (limestone and marble lithologies) and 20 (intrusive lithologies). The combined value was 20 ppm which was set as the acceptable maximum value for Cu.

Table 11-7 Acceptable maximum values for Ag, Au and Cu in the blank samples

ELEMENT	METHOD	UNIT	DETECTION LIMIT	ACCEPTABLE MAX
Ag	FAG323	g/t	<5	5.5
Au	FAG323	g/t	<0.01	0.011
Cu	ICP40B	ppm	<2	20

The sample blanks for Ag and Au used the method FAA313 and ICP40B, respectively, all resulted in assays below their respective detection limits, which confirm that the blank samples are being prepared and inserted effectively for Au and Ag.

Thirty-nine percent (39%) of the sample blanks for Cu using method ICP40B occurred above the acceptable maximum value of 20 ppm (Figure 11.25). The highest value occurs at 297 ppm, 115% higher than the acceptable maximum value. The blank collection stage at the Aranzazu exploration site has been revised so that mineralized material is not selected and introduced into the sample stream, and blank samples are now collected from a nearby barren limestone quarry.

The quality control of SGS Laboratories was also questioned because there is the possibility that blank samples within a sequence of high grade assay values are contaminated by the samples above. The assay preparation and analytical process for cleaning was also examined and found to be acceptable.

The SGS internal blank quality control samples were reviewed and many of the samples occurred below the tolerance limit but there are a few isolated high values (Figure 11.26). Knowing this, all of the assay certificates from 2013 were looked at specifically to examine the Cu grade immediately above the position of the blank samples. The outcome is that all of the blank samples that occur greater than the tolerance limit of 20 ppm occur immediately below or within medium to high grade sample, which means that the samples below are being contaminated by the samples above. The lab was informed of the outcome and responded that they have an exception rule for when a blank (prep and method blank) failure can be ignored: if the prep or method blank concentrations are insignificant compared to the sample concentrations (i.e. 100 times less than the average value of the samples within a batch) the failure can be ignored and no repeats will be required because the concentration of the analysed elements in the blank will make no significant difference to the sample concentration. This explanation explains the isolated highs in relation to the high-grade samples.

Figure 11.25 Scatter plot of copper results for blank samples using method ICP40B

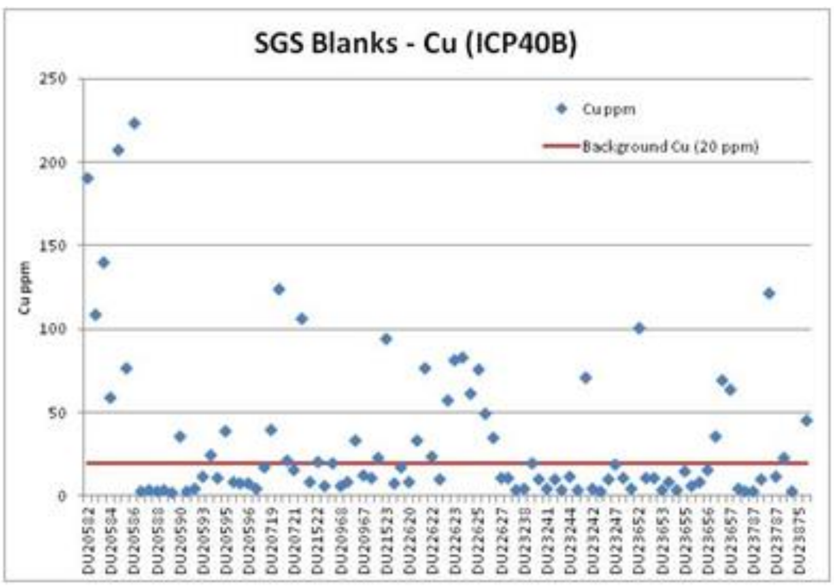
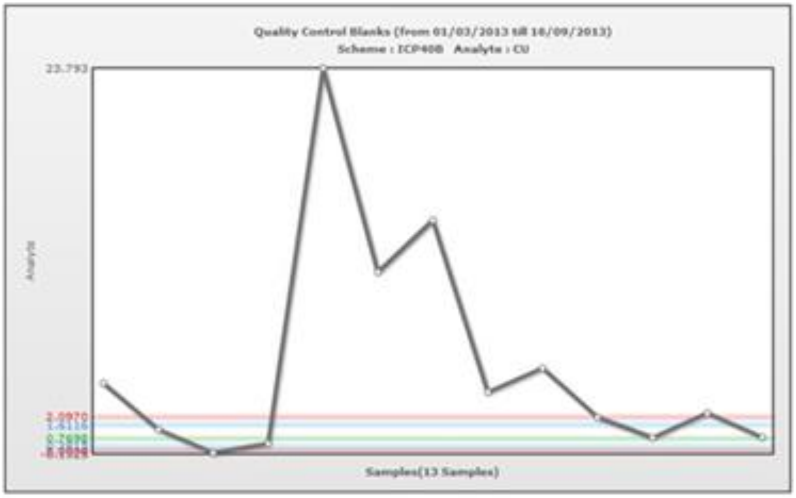


Figure 11.26 SGS internal quality control blanks during the Janurary 1 to September 16, 2013



11.4.7 Intra-laboratory Check Assays

As a further validation of the reliability of SGS Laboratories assays, duplicate core samples were submitted to ALS Global in Zacatecas, Mexico at an amount of approximately 5.0% of the total samples.

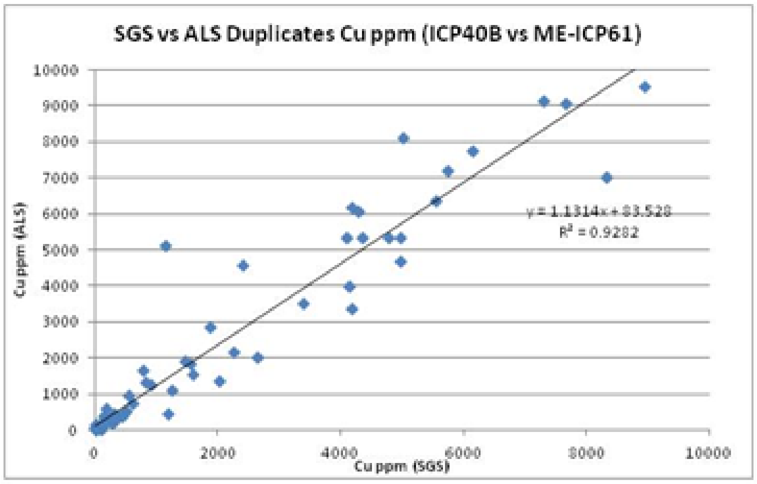
The assay methods selected at ALS are equivalent to the assay methods in use at SGS Laboratories. Table 11-8 is a comparison of the method used at both labs.

Table 11-8 SGS versus ALS assay methods

Element	Lab	Method Code	Digest	Finish
Cu <1%	SGS	ICP40B	4-acid	ICP-AES
	ALS	ME-MS61	4-acid	ICP-AES
Cu >1%	SGS	ICP41Q (discontinued)	4-acid	ICP-AES
	SGS	ICP90Q (used from October 2013)	Sodium peroxide fusion	ICP-AES
	ALS	Cu-OG62	4-acid	ICP-AES
Au	SGS	FAG323 (discontinued)	FA	AA
	SGS	FAA313 (used from October 2013)	FA	AA
	ALS	MEGRA21 (discontinued)	FA	Gravimetry
	ALS	Au_AA25 (used from October 2013)	FA	AAS
Ag	SGS	FAG323 (discontinued)	FA	Gravimetry
	SGS	ICP40B	4-acid	ICP-AES
	ALS	MEGRA21	FA	Gravimetry

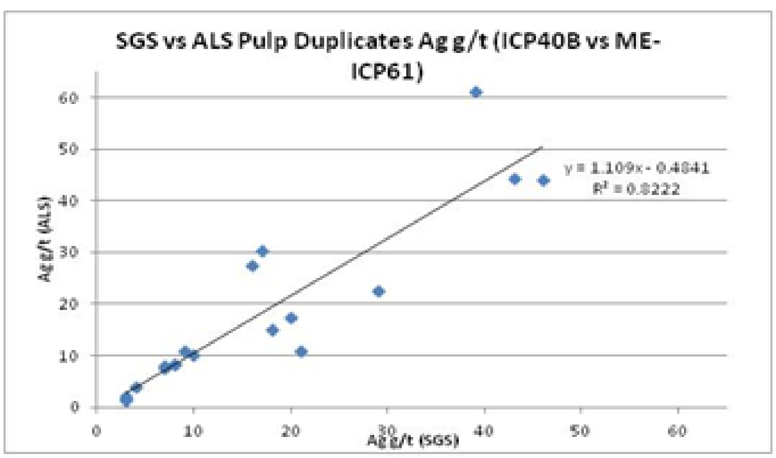
The results for copper using method ICP40B at SGS and MEMS61 at ALS show that there is a good comparison between two laboratory assay results (Figure 11.27). Four outliers were removed that grossly influenced the correlation coefficient. The mean of the SGS copper assays is 1369 ppm Cu and the mean ALS assay is 1633 ppm, an 18% absolute mean paired relative difference, which is an acceptable difference for core duplicates. The R^2 value for the pair of results is 0.9282.

Figure 11.27 Scatter plot of copper results between two labs duplicates using methods ICP40B at SGS and MEMS61 at ALS for the 2013 exploration campaign



The results for Ag using methods ICP40B at SGS and ME-ICP61 at ALS were reasonable with an R^2 value of 0.8222. There are three outlier values that significantly lower the correlation coefficient, and with the removal of these outliers the correlation coefficient rebounds to above 0.9.

Figure 11.28 Scatter plot of silver results for the Intra- lab duplicates using method ICP40B at SGS and MEMS61 at ALS for the 2013/2014 exploration campaign.

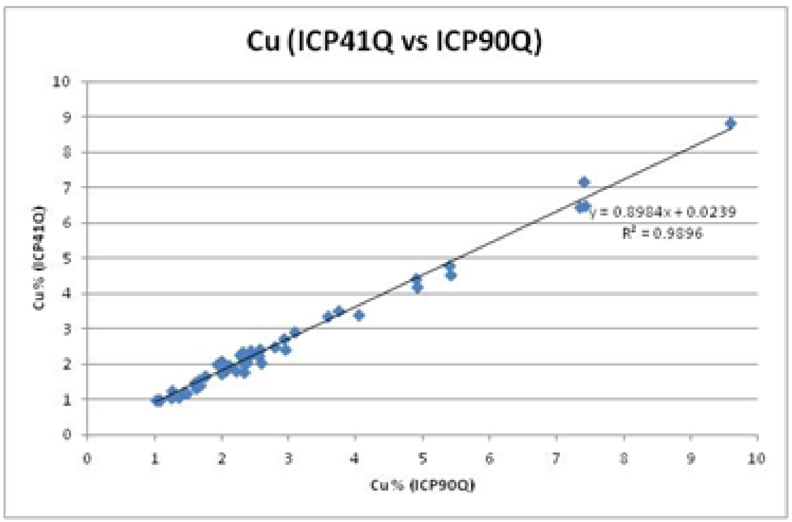


11.4.8 Validation of SGS Method ICP41Q Using ICP90Q

In the middle of September 2013 SGS Laboratories informed Aura Minerals that their method ICP41Q was validated but not certified. In order to validate the samples from 2013, 20% of the samples were re-tested using method ICP90Q which is a certified method. In addition, nine assay certificates that were currently in process at SGS Laboratories were tested with both assay methods. ICP41Q is a near total digestion method using 4-acids, whereas ICP90Q is a total digestion method using sodium peroxide fusion as the digestion method which means that the Cu recovery will be slightly higher for the fusion method.

Fifty-two (52) samples from 18 assay certificates were re-run for Cu using both method ICP90Q (validated and certified) and ICP41Q (unvalidated and uncertified) Cu. The paired results show good correlation between the methods with an R^2 value of 0.9896 (Figure 11.29). The lower grade bias using method ICP41Q was verified as the average using this method was 2.56% Cu, whereas the average using method ICP90Q is 2.82% Cu, a 9.0% variance. The copper assay results are consistently higher using method ICP90Q, but since method ICP90Q is a total digest method using sodium peroxide fusion, the results are as expected. Going forward method ICP90Q will be used for the exploration drill samples, which will be approximately 9.0% higher in grade than previously recorded.

Figure 11.29 Scatter plot of copper results >1% using method ICP90Q and ICP41Q



11.4.9 Conclusions and Recommendations

The Aranzazu deposit contains deleterious elements such as Zn, As, Bi and Sb that Aura Minerals is penalized for when the concentrate reaches the smelter. The amount of sulphide elements can be determined using an aqua-regia digest, however, silicate Zn, As, Bi and Sb minerals can only be liberated in a four acid digest.

A comparison between aqua-regia and 4-acid assay results confirms that the deposit hosts small amounts of silicate Zn, As, Bi and Sb species. To account for the total amount of Zn, As, Bi and Sb the samples should be processed in a four acid digestion medium. There is a chance that a small percentage of the volatile elements (As, Bi, Sb) volatilize in the 4-acid digest, which will ultimately lower the total amount of deleterious elements. To monitor for this potential underestimation, 2.0% of the As samples will be processed using sodium-peroxide fusion which is a process that avoids the loss of volatile elements.

The 2012 to 2013 QAQC results from SGS demonstrate that the commercial laboratory has produced reliable assays, sufficient to be used for resource estimation purposes. To improve the QAQC procedure several recommendations have been suggested by Aura Minerals and the QP for future drilling programs:

- Many of the blank samples occur above the 90th percentile of the non-mineralized limestone, marble and intrusive lithologies. This means that the lithologies selected on site are not suitable as blank material as they contain weak mineralization. Rock that occurs well outside the deposit from a neighbouring limestone quarry needs to be collected and crushed. Ideally 10% of the blanks inserted into the exploration sample stream should be tested at the production lab to verify that they are in fact non-mineralized.
- A third standard that represents low grade Cu and Au needs to be inserted as the two standards currently being used are medium to high in grade. Additionally, an Ag standard needs to be introduced. The standard to be used is CDN-CM-24 which is certified for 0.365% Cu ± 0.020%, 0.521 g/t Au ± 0.056 g/t and 4.1 g/t Ag ± 0.4 g/t. This standard has arrived on site and will be used from the beginning of October 2013.
- A fourth standard need to be introduced for high grades especially for gold and silver.
- Crush (coarse) duplicates need to be inserted by SGS at the first stage of crushing of the split drill core and assayed at the same laboratory, in this case SGS. An insertion rate of approximately one in every thirty-five samples is sufficient for this control sample. This request has been included into Aura Minerals' standard requisition as of the October 1, 2013.
- Pulp duplicates need to be inserted by SGS at the final pulverizing stage and assayed at the same laboratory, in this case SGS. An insertion rate of approximately one in every thirty-five samples is sufficient for this control sample. This request has been included into Aura Minerals' standard requisition as of October 1, 2013.
- Insert 1.0% pulp duplicates at the secondary laboratory
- Five (5) percent of the duplicates (from field, crush and pulp) should be sent to an independent laboratory and analyzed using equivalent methods to the primary laboratory. These results will be used as a duplicate check against bias. The QP preference is for this laboratory to be in Canada if it is possible.
- Standards need to be sent directly to ALS from SGS in pulp form.

12. DATA VERIFICATION

12.1 GENERAL

The Geological QP for the Project (Farshid Ghazanfari, P.Geo.) visited the Aranzazu Property during December 2014 for a total of ten days. Since the definition drilling, development and production ceased in February 2015, the above visit is still sufficiently current for the purposes of NI 43-101.

During the site visit, the QP inspected the surface exploration facilities and conducted a brief underground tour of the mine. The core shack was visited, and the logging, sampling, and sample shipment procedures were reviewed. Density sampling and testing was also reviewed.
Figures 12.1 through 12.6 are various views of the UG drilling set up, core storage and logging facilities which were observed by the QP during the site visit.

In addition, some of underground drill setups were visited (Figure 12.4). The collar locations of surface drilling were not found when driving around the Property due to increasing mining activity in recent years. However, in a previous 2011 Technical Report, they spotted and verified some of historical surface drilling.

During the site visit, the QP discussed the definition drilling program and the QA/QC procedures to ensure that delineation of resources meet CIM guidelines and that the results obtained were meaningful and pertinent.

Drill logs were reviewed for several selected holes and they were found to adequately describe the RQD, recovery, geological units, structure and mineralogy seen in the core. Thus, the logging conforms to the generally accepted industry standards currently in effect. Aura Minerals photographs the core for the entire drill hole and these photographs are part of the long-term drilling record.

The core boxes are stored off the Property in a building close to the village of Concepcion del Oro in a secure facility which has a twenty four hour security detail assigned to it seven days a week.

Figure 12.1 Aura Minerals' Core storage and Logging Facilities in Concepcion del Oro

Figure 12.2 Aura Minerals' core shack technician measuring bulk density (December 2014)

Figure 12.3 UG drilling set up during active definition drilling (December 2014)

Figure 12.4 Core recoveries from UG drilling (December 2014)

Figure 12.5 UG drill hole collars from 2014 drilling program

Figure 12.6 Blasting and stripping in active Security Pit (December 2014)

12.2 DATA VERIFICATION FOR PURPOSE OF RESOURCE ESTIMATION

The Geological QP for the Project (Farshid Ghazanfari, P.Geo.) visited Aura Minerals' Vancouver office in November 2014 prior to the site visit and spent a week working closely with the Aura Minerals resource geologist reviewing different aspects of resource estimation practices. During the following months until March 2015 (cutoff date for this Technical Report), the QP guided Aura Minerals' resource geologist to implement a new 3D modelling approach based on NSR for current resources and reviewed and revised previous parameters based on this new approach.

Since the previous Mineral Resource estimate, Aura Minerals has incorporated a number of changes into its resource estimation methodology, including:

- Addition of 4969.25 m of drilling in 59 holes.
- Geological interpretation on 126 sections in three different grid set
- Separation of the mineralization into zones that better reflect local changes in strike and dip of the mineralization.
- Introducing a NSR value as a major element into the database for 3D modelling and cut- off grade calculations
- Density model using interpolation of density data controlled by lithological domains

These additions are discussed in this Technical Report and the QP believes that the changes incorporated into the current Mineral Resource estimate will assist in further understanding the extent of the mineralization, as well as assisting in optimizing the operational planning which Aura Minerals will use during the re-opening of the Aranzazu mine.

The status of historical drilling data was discussed, and further validation was performed by the Aura database manager to make sure all historical data in the database are validated against hard copies of logging data.

Due to the higher number of missing gold and silver assays in the historical (Macocozac and Zacoro) database, it was decided to assign background default values to control the influence of the gold and silver samples in the block model (this approach is explained extensively in Item 14.3.1 and 14.3.2 of this Report). Aura Minerals has been able to confirm and in-fill the gold and silver data for portions of the model with its exploration drilling program and this was the basis upon which it was decided to include gold and silver in the previous estimates. Aura Minerals has continued to assay for gold and silver among other elements and has included both gold and silver in previous resource estimates.

Since the introduction of the NSR model will increase the role of gold and silver in estimation and mineable resources, interpretation of 3D models was solely based on available and verified Cu, Au and Ag assay data and not on assigned values. Using the same thought process, these assigned values haven't been used for capping, geostatistical analysis and compositing of assay data. They have been incorporated after all interpretations and geostatistical analysis was done to fill in the missing values and unsampled intervals.

For all other elements, including Mo, Pb, and Zn, Bi, Sb and oxide Cu, the information in the database is minimal and only available for the work which has been conducted by Aura Minerals. These elements are not reported in the resource estimates. At the present time, the database information is such that a Mineral Resource estimate for these elements would not currently meet CIM standards and therefore, would not be reportable under NI 43-101 guidelines or regulations.

Following interpolation of known sample intervals, using simple interpolation method (Item 14.4.2), a set of regression formula (see Item 14) was established to calculate grade for blocks with no As values based on Cu grades. Since As is a penalty element for concentrate sales terms, knowing the As grade is beneficial for Aura Minerals in their review of the operational parameters used for mine planning purposes and cut-off grade calculation.

Discussions regarding the categorization of the resources were held and it was concluded that, in general, in areas which rely on the use of the historical Macocozac or Zacoro data, the resources should be given a lower classification, based on the confidence which can be assigned to such data.

12.3 INDEPENDENT SAMPLING BY QP

Farshid Ghazanfari, geological QP and author of this Technical Report completed independent sampling of half-split drill core and selected pulps during the site visit in December 2014. In total, the author collected 30 samples of half-split core and 99 pulps from the Aura Minerals core shack facility. The half-split cores and pulps sent to the ALS Global laboratory in Zacatecas, Mexico to be analyzed for full suite of elements. The sampling and security of sample bags were monitored by the author to make sure they had not been tampered or contaminated. The detail of ALS assay methods for each element is explained in Item 11 of this Technical Report.

The results of half-split cores are shown in Table 12-1 against the original assay data which was recorded in the database from the same intervals and analyzed in the SGS lab in Durango, Mexico. The majority of the sampling carried out on available core boxes were drilled by Aura Minerals. Unfortunately, historical core boxes were not available in Aura Minerals' Core shack facility for sampling. The only available historical hole in the list is 53900-1 drill hole which was drilled by Zocoro. The results confirm the presence of high grade copper mineralization in this hole. In general, half-split core results return acceptable range of variance from original values in the assay database. A few higher than usual anomalies which were observed especially in hole C-2004-62 are probably reflecting the variability of high grade skarn in selected samples and difference between half-split core results and half core original assay values.

The ALS results however seem to show a bit higher values for copper and gold with an average variance of approximately 27% for copper and 39% for gold. Also, the current method which was used for silver is not suitable for samples with higher than 100 ppm silver content. Therefore, original high grade silver values (>100 ppm) from SGS cannot be directly used for comparison.

In general, the results of independent half-split core sampling confirmed mineralization in major mineralized zones and in accordance with the Aranzazu database for selected intervals. Besides differences between half-split core and half core samples, in general SGS results stay in lower spectrum and a more conservative side and therefore it is the Author's opinion that they are not biased and are in line with what could be expected from observed mineralization in cores.

Table 12-1 QP independent samples during the site visit (half-split core)

	QP Independent Samples (ALS Lab)						Aura Minerals Sample (SGS Lab)			
Hole-ID	Sample Number	From	To	Cu (%)	Au (g/t)	Ag (g/t)	Sample Number	Cu (%)	Au (g/t)	Ag (g/t)
AZC-076	3504	93.00	94.50	1.00	0.57	31.8	58487	0.85	0.60	27.5
AZC-076	3505	195.00	196.50	4.01	3.80	43.8	58561	3.51	4.10	32.5
AZC-119	3506	384.00	385.50	2.14	1.00	75.0	86764	1.76	0.78	62.9
UAZ-039	3507	96.00	97.50	0.81	0.25	19.5	96080	0.97	0.23	22.0
UAZ-039	3508	99.00	100.50	2.00	0.33	43.3	96083	1.71	0.26	41.3
BD-4385-15	3509	38.25	39.40	1.33	0.25	17.7	716	1.19	0.26	11.0
53900-1	3510	67.00	71.70	1.52	0.68	18.6	53900-1-7	2.37	0.99	21.5
UAZ-057	3511	103.50	105.00	0.04	0.01	<0.5	107723	0.00	0.01	0.3
UAZ-057	3512	132.00	133.50	5.46	1.01	58.5	107747	5.45	4.30	76.3
C-2004-62	3513	109.55	111.05	0.99	0.26	10.8	3668	1.33	0.29	13.0
C-2004-62	3514	111.05	112.55	0.93	0.20	7.6	3669	0.72	0.37	7.0
C-2004-62	3515	112.55	114.05	0.37	0.13	4.3	3670	0.46	0.14	6.0
AZC-182	3516	424.50	426.00	1.93	0.65	20.7	116243	1.95	1.00	26.6
AZC-182	3517	430.50	432.00	0.95	0.39	10.1	116248	0.72	0.39	9.2
C-2004-62	3518	127.35	128.35	16.25	2.87	>100	3684	18.40	3.80	157.0
C-2004-62	3519	128.35	129.35	1.39	0.26	15.3	3685	0.21	0.03	3.00
C-2004-62	3520	63.85	65.35	2.03	0.67	13.1	3626	2.96	0.80	17.0
C-2004-62	3521	66.85	68.35	0.76	0.13	7.3	3627	2.15	0.41	17.0
C-2004-62	3522	68.35	69.85	0.32	0.13	3.8	3628	0.86	0.20	12.0
C-2004-62	3523	69.85	71.35	2.07	1.11	15.8	3629	0.48	0.18	7.0
C-2004-62	3524	71.35	72.85	2.63	0.45	30.3	3631	2.06	0.38	15.0
C-2004-62	3525	75.85	77.35	3.12	0.59	34.8	3632	3.66	0.72	43.0
C-2004-62	3526	77.35	78.85	3.60	0.56	53.2	3634	1.76	0.23	18.0
C-2004-62	3527	78.85	80.35	3.11	0.33	39.7	3635	2.46	0.49	33.0
C-2004-62	3528	80.35	81.85	2.81	0.53	27.3	3636	2.69	0.39	31.0
C-2004-62	3529	81.85	83.35	1.35	0.44	6.6	3638	6.18	0.41	67.0
C-2004-62	3530	83.35	84.85	2.48	0.66	16.8	3639	4.29	0.58	43.0
C-2004-62	3531	84.85	86.35	1.64	0.27	13.2	3640	2.76	0.57	21.0
C-2004-62	3532	86.35	87.85	3.90	0.85	28.3	3641	2.03	0.43	15.0
C-2004-62	3533	87.85	89.35	2.23	0.41	19.2	3642	3.20	0.58	21.0

Of 99 duplicate pulp samples which were submitted for analysis in ALS, 98 samples returned with values. One sample didn't return with any value probably due to an error in the laboratory which is unknown to the QP. The rest of 98 samples are used for data verification purposes.

Figures 12.7 to 12.11 show results of assaying for all of 98 duplicate pulp samples versus their original values in the database.

Figure 12.7 shows results for copper in 98 samples assayed at SGS and ALS. Cu grade is unbiased between two labs. The scatter diagram shows that results are positively correlated between the two labs.

Since the number of very high-grade copper values (Cu %> 5) is limited in selected intervals, another scatter diagram was generated (Figure 12.8) to show resultant assays without influence of high grade values. In this case results are strongly and positively correlated between the two labs.

There is a total of two samples on each side of the trend line which show a higher standard deviation from parity. Their deviations are larger than +/- 10 acceptable ranges. It might be because of possible difference of analytical methods at the time of analysis between two labs or an unknown error in any of the labs. As (+/-) 10% relative deviation is considered reasonable precision for pulp duplicates, these results are deemed to be satisfactory for resource estimation purposes.

Figure 12.9 shows results for gold in 98 samples assayed at SGS and ALS. The scatter diagram shows that results are positively correlated between the two labs. However, there is one sample (from hole AZC-024) that shows abnormally high value (18.25 g/t) with a possible unknown error in the ALS lab.

Since the number of high grade gold values (Au > 5 ppm) is limited in selected intervals, another scatter diagram was generated (Figure 12.10) to show resultant assays without influence of high grade values. In this case results are also positively correlated between the two labs.

A set of samples on each side of trend line which show a higher standard deviation from the average. These samples fall beyond an acceptable 10% range that is deemed to be an acceptable comparison. However, it seems that there is a bias in ALS results towards the high-grade spectrum of grade values and SGS results are slightly bias towards lower spectrum of grade.

Figure 12.11 shows results for silver in 88 samples assayed at SGS and ALS. The scatter diagram was generated for only 88 samples out of 98 samples since ALS detection limit for Ag was 100 ppm with current method of analysis. The scatter diagram shows that results are positively correlated between the two labs with slight bias toward higher grade silver in ALS. As the +/-10% relative deviation is considered to have reasonable precision for pulp duplicates, these results are deemed to be satisfactory for resource estimation purposes.

For higher grade silver values, it seems SGS lab is more reliable than ALS. The ALS method apparently cannot properly analyze samples above its detection limits (>100 ppm). Therefore, it is recommended that either ALS change the method for silver or Aura Minerals re-submit the sample with higher silver values to a tertiary lab in Canada for further validations.

Figure 12.7 Comparison of original pulp assays (SGS) versus QP pulp samples (ALS) for Cu (%) All Umpire Samples

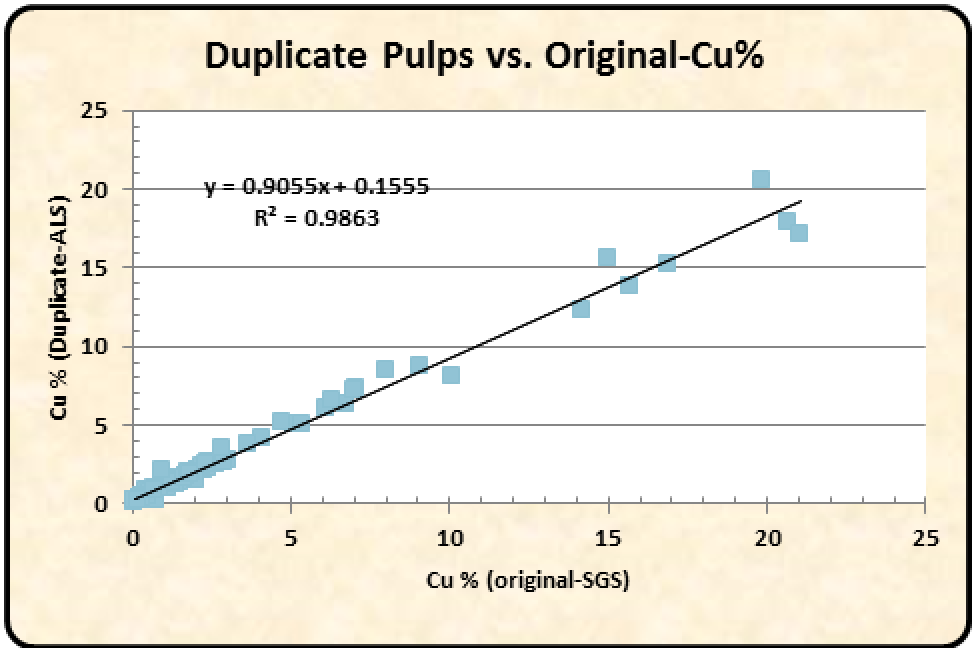


Figure 12.8 Comparison of Original Pulp Assays (SGS) versus QP Pulp Samples (ALS) for Cu < 5% samples

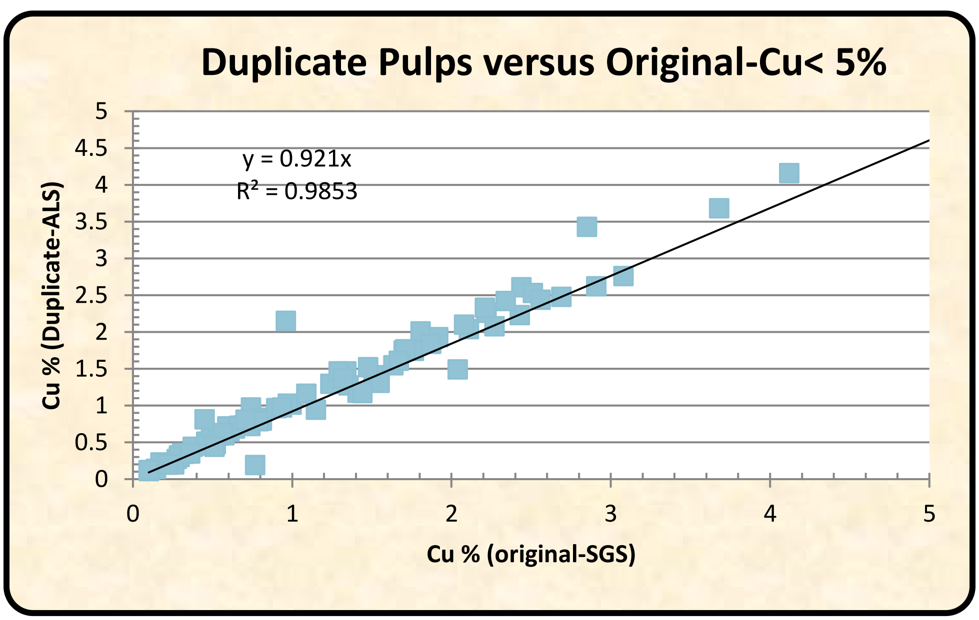


Figure 12.9 Comparison of Original Pulp Assays (SGS) versus QP Pulp Samples (ALS) for Au (ppm) All Umpire Samples

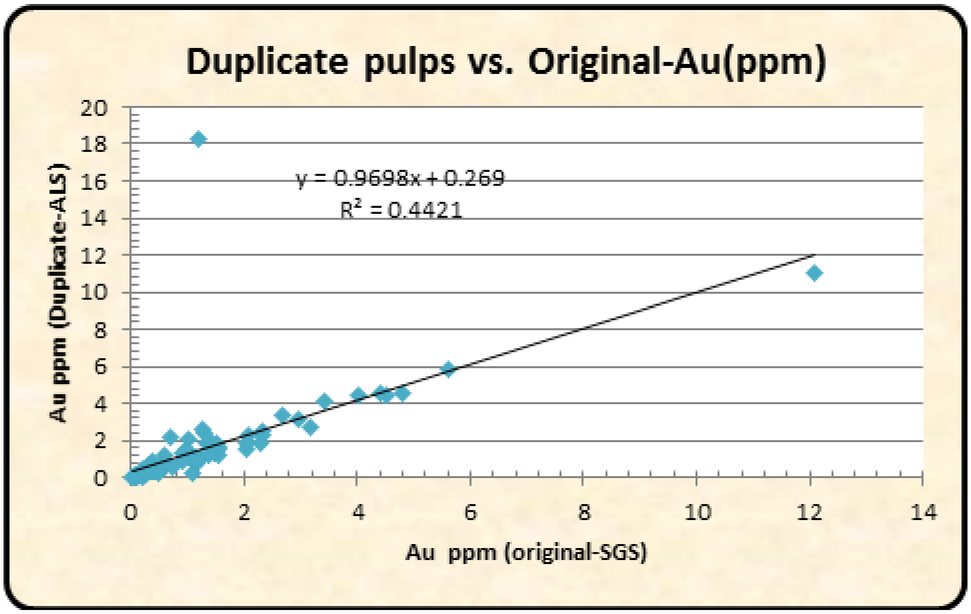


Figure 12.10 Comparison of Original Pulp Assays (SGS) versus QP Pulp Samples (ALS) for Au < 5 ppm

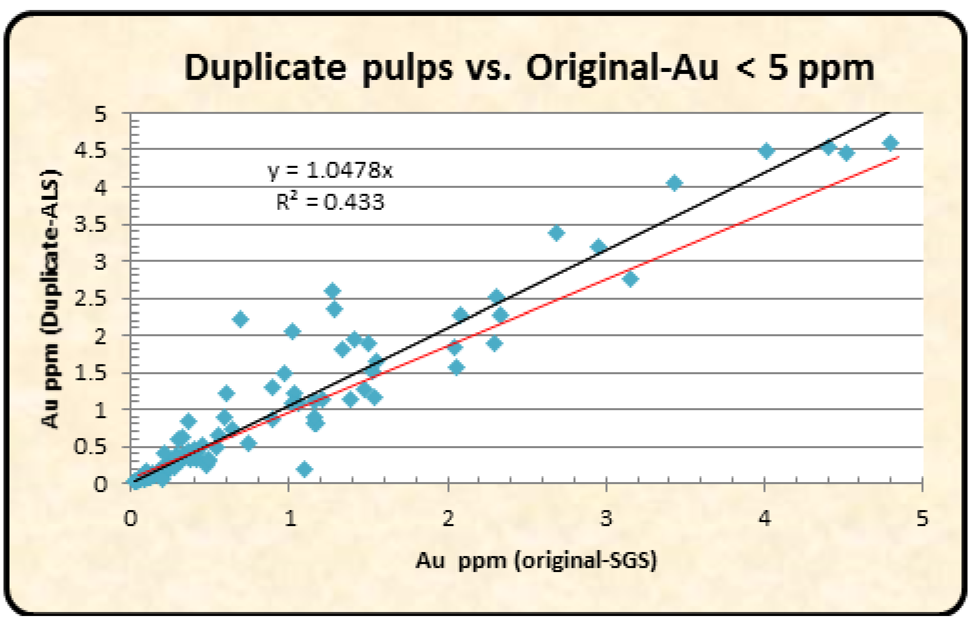
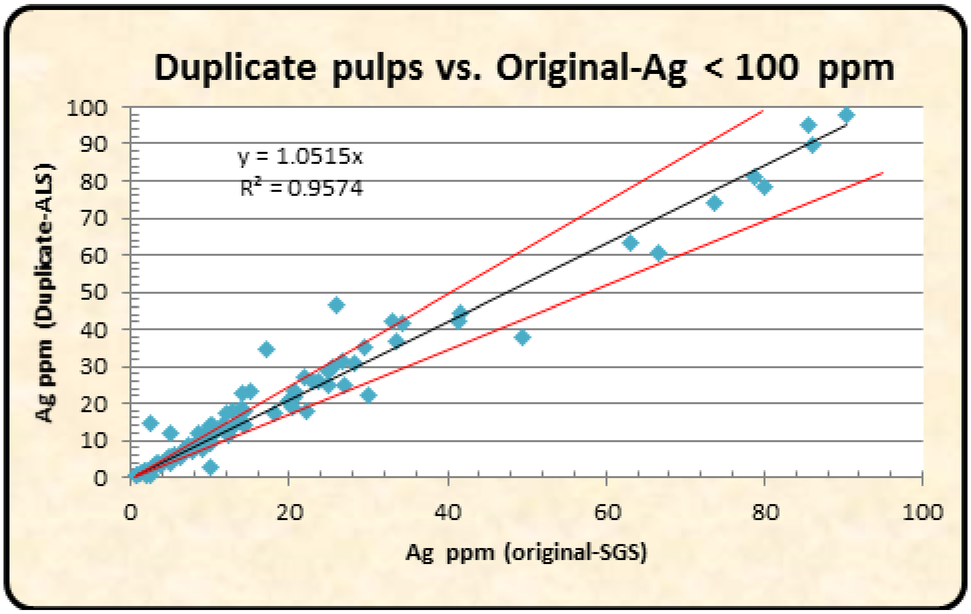


Figure 12.11 Comparison of Original Pulp Assays (SGS) versus QP Pulp Samples (ALS) for Ag < 100 ppm



12.4 ADDITIONAL QA/QC MEASURES FOR PURPOSE OF FEASIBILITY STUDY

Additional QA/QC measures were recommended in a previous NI43-101 Report by the author of this Technical Report to further validate historical drilling in the Aranzazu database. In 2017 a set of selected intervals were sawed into quarter core from existing half core using available cores in core shack facility. The collected samples were then sent to the SGS lab in Durango, Mexico for a full suite of analysis by ICP-Aqua Regia method. The list of holes and sampled intervals are presented in Table 12-2 along with sample number, original assays and re-assays analysis from the SGS lab. The Figures 12.12 to 12.14 show the scatter plot for these selected intervals. The result of these duplicate samples shows the new assays, in most of cases, are in line with historical results and there is no significant bias exist in historical drill hole data.

Table 12-2 Historical Holes Assayed Intervals versus Quarter Core Re-Assayed in 2017

	Re-assayed Samples (SGS Lab)						Historical Assay Results		
Hole-ID	Sample Number	From	To	Cu (%)	Au (g/t)	Ag (g/t)	Cu (%)	Au (g/t)	Ag (g/t)
23700-3	1502	98	99.3	0.55	0.91	3	1.37	1.88	7.2
23700-3	1509	128	130	1.00	1.64	10	1.22	2.1	11.6
23700-3	1513	138	140	0.61	1.12	7	1.18	2.26	10.1
23700-3	1503	102	104	0.88	0.54	6	1.17	0.75	8.8
23700-3	1504	104	106	1.29	1.88	13	1.00	1.58	14.7
23700-3	1501	91	92	0.67	1.69	10	0.80	2.71	11.7
23700-3	1512	132	134	0.94	1.25	8	0.80	1.44	8.9
23700-3	1505	108	110.25	0.68	2.95	6	0.69	2.69	8.4
23700-3	1511	130	132	0.90	1.98	11	0.63	1.98	8.6
23700-3	1507	124	126	0.42	0.92	5	0.63	1.43	6.3
53450-2	1516	97.89	99.48	0.48	0.69	27	1.31	1.35	12.3
53450-2	1517	60	61.43	0.84	1.45	53	0.58	1.44	29.5
53625-3	1530	141.3	143.3	15.50	2.58	83	20.30	3.39	81.8
53625-3	1526	138.68	140	2.85	0.45	14	3.07	0.41	10.2
53625-3	1528	140	141.3	0.98	0.17	4	2.86	0.45	8.2
53625-3	1525	118	120	4.43	0.54	28	1.85	0.41	10
53625-3	1524	106	108	1.23	0.16	12	1.42	0.31	8.3
53625-3	1520	62	64	0.88	0.07	11	1.40	0.23	18
53625-3	1522	70	72	1.54	0.23	12	1.13	0.27	7
53625-3	1518	22	24	0.87	0.16	8	1.12	0.48	5.3
53700-1	1534	62	64	5.02	1.171	30	6.01	1.68	37.1
53700-1	1539	98	100	5.22	1.321	56	5.60	1.25	59
53700-1	1537	94	96	5.64	1.156	66	5.40	0.89	110
53700-1	1535	92	94	2.49	0.383	32	3.23	0.31	35.8
53700-1	1532	32	34	4.92	0.377	18	2.80	0.34	24.1
53700-1	1541	106	108	1.71	1.115	15	1.83	0.58	13.6
53700-1	1542	108	110	1.87	0.513	13	1.56	0.41	5.7
53700-1	1544	116	118	1.98	0.536	14	1.45	0.24	8.2
53700-1	1533	60	62	0.77	0.099	6	1.38	0.93	5.2
53700-3	1569	68	70	4.44	2.44	24	3.68	0.69	20.2

	Re-assayed Samples (SGS Lab)						Historical Assay Results		
Hole-ID	**Sample Number**	**From**	**To**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**
53700-3	1575	176	178	3.22	0.34	27	3.60	0.48	44.3
53700-3	1559	54	56	1.16	0.43	6	2.21	0.41	9.6
53700-3	1577	212	214	3.08	0.15	27	2.03	0.27	23.6
53700-3	1560	56	58	1.69	0.26	11	1.91	0.21	13.3
53700-3	1550	16.76	17.81	0.24	0.05	2	1.87	0.69	17.4
53700-3	1548	12	14.08	0.37	0.04	3	1.80	0.21	13.4
53700-3	1549	15.13	16.76	3.56	0.16	21	1.77	0.02	13.6
53700-3	1553	24	26	2.92	0.22	26	1.74	0.31	18.9
53700-3	1551	20	22	0.23	0.04	2	1.70	0.89	13
53700-3	1547	2	6	1.28	0.14	13	1.61	0.41	13.3
53700-3	1568	66	68	0.75	0.31	5	1.55	1.58	13.3
53700-3	1573	158.45	160	2.14	1.11	8	1.47	0.21	7.7
53700-3	1566	62	64	1.74	0.34	10	1.46	0.34	9.6
53700-3	1557	46	48	0.74	0.05	4	1.43	0.14	8.1
53700-3	1567	64	66	0.74	0.25	4	1.35	0.14	5.6
53700-3	1571	104	106	1.72	0.37	18	1.24	0.85	14
53700-3	1563	60	62	5.48	0.89	34	1.22	0.27	9.1
53700-3	1555	44	46	0.23	0.11	7	1.22	0.62	6.7
53700-3	1576	178	180	1.17	0.29	12	1.15	0.13	13
53700-3	1558	48	50	1.64	0.41	17	0.86	0.55	9
53800-1	1584	14	16	1.50	0.67	13	2.78	0.86	10
53800-1	1582	12	14	2.10	0.73	15	2.36	0.72	20.3
53800-1	1595	32	33.45	1.61	0.22	13	2.04	0.41	5.1
53800-1	1579	4	6	1.87	0.25	17	2.00	0.34	3.1
53800-1	1580	6	8	1.19	0.23	10	1.84	0.48	5.2
53800-1	1586	16	18	1.30	2.14	14	1.81	2.61	10.3
53800-1	1581	10	12	1.44	2.18	12	1.68	1.1	9.6
53800-1	1597	40	42	0.71	0.32	8	1.31	0.1	9.7
53800-1	1593	30	32	1.11	0.35	8	1.09	0.33	10
53800-1	1588	18	20	1.09	0.72	10	0.98	0.31	13.9

	Re-assayed Samples (SGS Lab)						Historical Assay Results		
Hole-ID	Sample Number	From	To	Cu (%)	Au (g/t)	Ag (g/t)	Cu (%)	Au (g/t)	Ag (g/t)
53800-1	1591	28	30	1.36	1.20	14	0.93	0.48	1.7
53800-1	1590	24	26	0.56	0.41	6	0.91	1.47	6.6
53800-1	1589	22	24	0.97	0.49	9	0.86	0.86	6.7
53800-4	1607	40.86	42	3.29	4.32	29	3.45	3.15	40
53800-4	1609	44	46	2.36	2.51	27	2.71	0.86	30.4
53800-4	1618	56	58	2.40	0.71	20	2.70	0.75	19.6
53800-4	1608	42	44	2.55	2.63	24	2.56	1.27	23
53800-4	1602	36	38.25	1.87	3.14	16	2.40	0.51	13.1
53800-4	1613	48	50	1.71	1.58	22	2.36	2.67	22.7
53800-4	1599	32.15	34	2.36	1.66	19	2.36	0.62	19.2
53800-4	1616	52	54	2.42	1.75	23	2.34	0.82	13.1
53800-4	1611	46	48	2.12	0.86	29	2.30	1.2	31
53800-4	1620	58	60	2.19	1.50	18	2.26	0.41	13.2
53800-4	1606	40	40.86	2.19	1.89	21	2.20	1.92	16.3
53800-4	1615	50	52	2.86	0.95	32	2.16	1.1	17.9
53800-4	1598	29.1	32.15	2.01	2.65	17	1.98	1.51	12.2
53800-4	1600	34	36	1.50	0.68	11	1.77	0.31	8.8
53800-4	1604	38.25	40	2.75	2.49	26	1.66	1.51	20.4
53850-3	1622	316	318	4.41	2.25	99	8.09	6.03	158.3
53850-3	1623	318	320	4.22	1.58	96	2.59	1.1	65.3
53850-3	1625	322	323.4	0.82	0.33	17	1.56	0.85	43.4
53875-2	1660	72	74	2.89	6.43	10	3.86	1.37	23.4
53875-2	1661	74	76	1.07	1.40	13	3.62	0.96	55.2
53875-2	1665	88	90	2.51	3.81	17	3.45	4.18	23.3
53875-2	1632	30	32	2.29	1.40	14	2.85	1.51	14.8
53875-2	1649	56	58	1.93	2.44	16	2.64	18.51	18.8
53875-2	1667	90	92	2.09	2.86	16	2.61	2.54	18.3
53875-2	1629	21	23	2.30	1.61	16	2.57	1.3	18.8
53875-2	1654	66	68	2.07	2.08	14	2.56	1.37	17.3
53875-2	1669	92	94	2.59	1.49	20	2.43	2.47	22.7

	Re-assayed Samples (SGS Lab)						Historical Assay Results		
Hole-ID	Sample Number	From	To	Cu (%)	Au (g/t)	Ag (g/t)	Cu (%)	Au (g/t)	Ag (g/t)
53875-2	1676	104	106	2.83	1.07	24	2.41	2.61	27.6
53875-2	1674	100	102	1.91	1.71	19	2.39	1.03	21.1
53875-2	1671	94	96	2.60	1.30	20	2.36	1.37	19
53875-2	1635	34	36	2.34	1.20	17	2.35	1.39	20.5
53875-2	1672	96	98	1.97	1.93	17	2.34	1.1	21
53875-2	1652	62	64	2.26	2.24	17	2.34	2.13	19.6
53875-2	1673	98	100	1.64	1.08	16	2.24	2.74	18.6
53875-2	1653	64	66	1.68	0.89	11	2.16	1.51	13.3
53875-2	1641	44	46	2.70	1.03	22	2.15	0.82	20.9
53875-2	1651	60	62	1.67	0.90	16	2.11	1.58	16.2
53875-2	1662	76	78	1.58	1.18	18	2.06	14.4	22.5
53875-2	1633	32	34	2.48	0.80	13	2.06	0.82	10.5
53875-2	1658	70	72	1.83	0.80	19	2.01	0.41	13.9
53875-2	1647	54	56	1.88	1.75	15	1.93	1.1	14.8
53875-2	1645	52	54	1.66	1.71	18	1.90	1.71	20
53875-2	1656	68	70	1.85	1.34	15	1.87	0.75	11.6
53875-2	1640	42	44	1.62	1.52	13	1.79	0.75	14.5
53875-2	1650	58	60	1.34	2.42	12	1.76	6.86	15.6
53875-2	1643	48	50	1.47	2.12	15	1.57	1.71	16
53875-2	1639	40	42	2.13	2.17	17	1.57	1.03	13.6
53875-2	1678	106	108	1.06	0.50	10	1.55	0.55	13.4
53875-2	1642	46	48	1.61	1.58	14	1.44	3.29	14.9
53875-2	1637	36	38	1.17	1.24	9	1.37	2.06	10.9
53875-2	1628	19	21	1.28	0.64	12	1.34	0.82	8.8
53875-2	1664	86	88	1.36	1.01	15	1.30	0.62	10.6
53875-2	1631	29	30	1.22	0.87	9	1.18	1.1	9.5
53875-2	1663	84	86	0.98	1.11	10	1.08	0.72	11.6
53875-2	1630	27	29	1.03	0.97	10	0.88	0.69	5.6
54375-1	1686	444	446	1.76	1.84	22	2.89	3.75	35.2
54375-1	1684	442	444	3.51	2.85	41	2.72	2.57	24.1

	Re-assayed Samples (SGS Lab)						Historical Assay Results		
Hole-ID	**Sample Number**	**From**	**To**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**
54375-1	1688	446	448	2.06	3.05	28	2.35	2.56	30.3
54375-1	1682	441.16	442	5.37	1.72	43	1.83	2.09	19.1
54375-1	1689	448	450	1.54	1.91	14	1.07	1.81	13.6
54375-1	1681	440	441.2	1.99	1.40	22	1.06	0.78	9.6
54375-1	1680	430	432	0.78	0.74	10	0.63	1.14	6.9

Figure 12.12 Comparison of Cu% from historical assays in database versus quarter core duplicate (re-assayed samples)

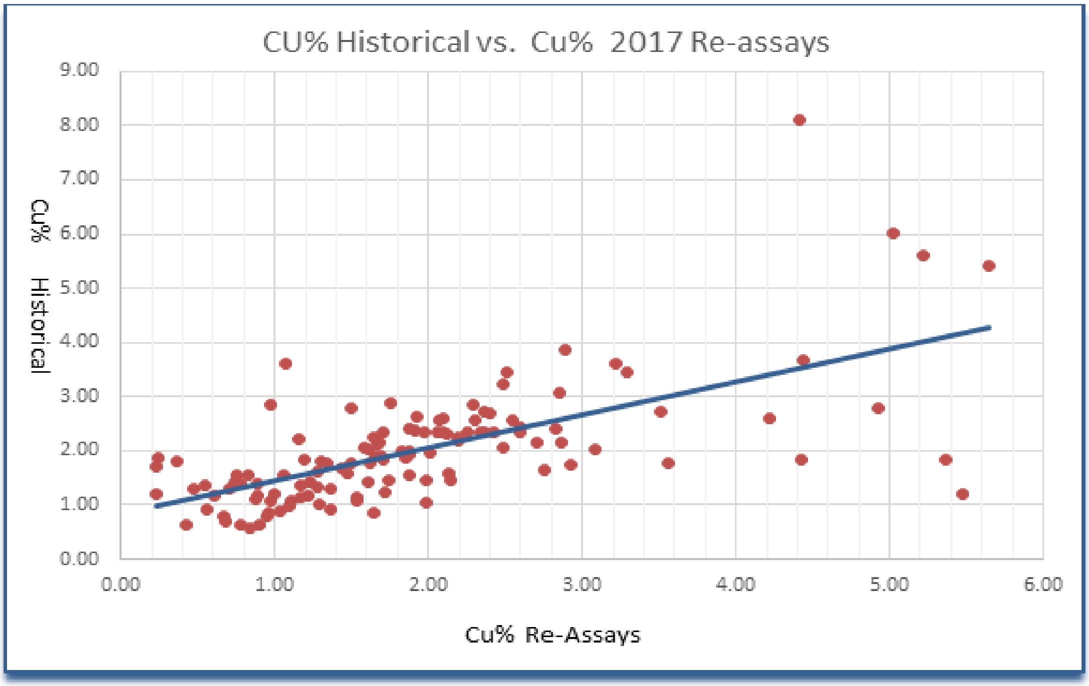


Figure 12.13 Comparison of Au from historical assays in database versus quarter core duplicate (re-assayed samples)

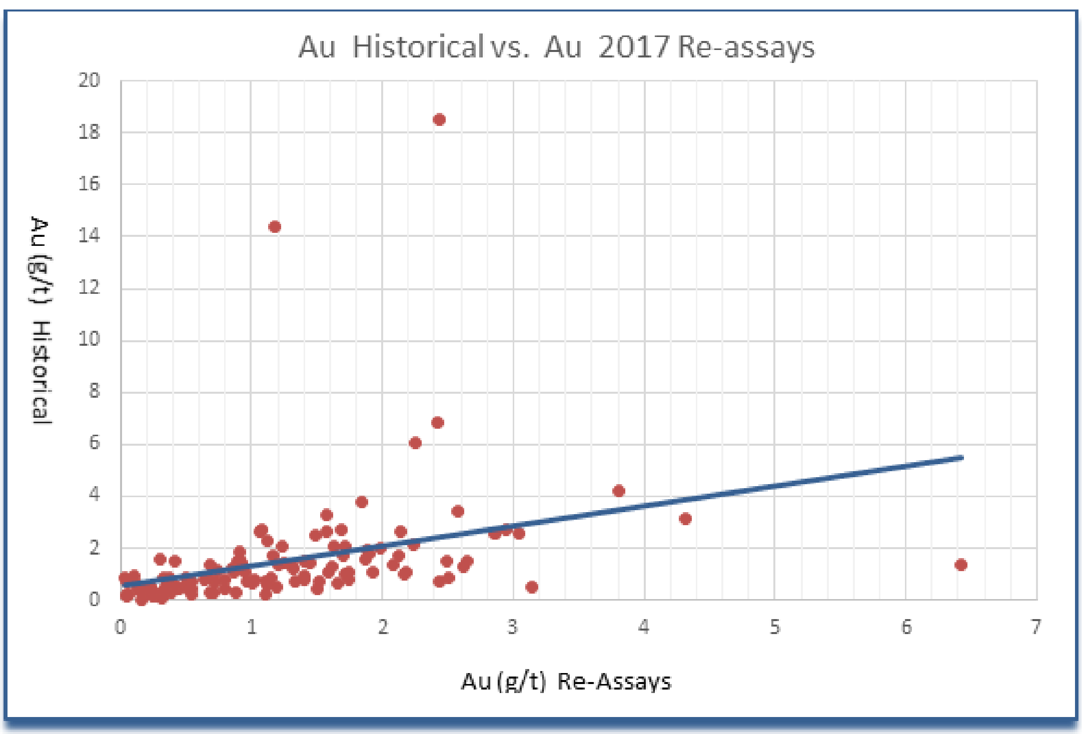
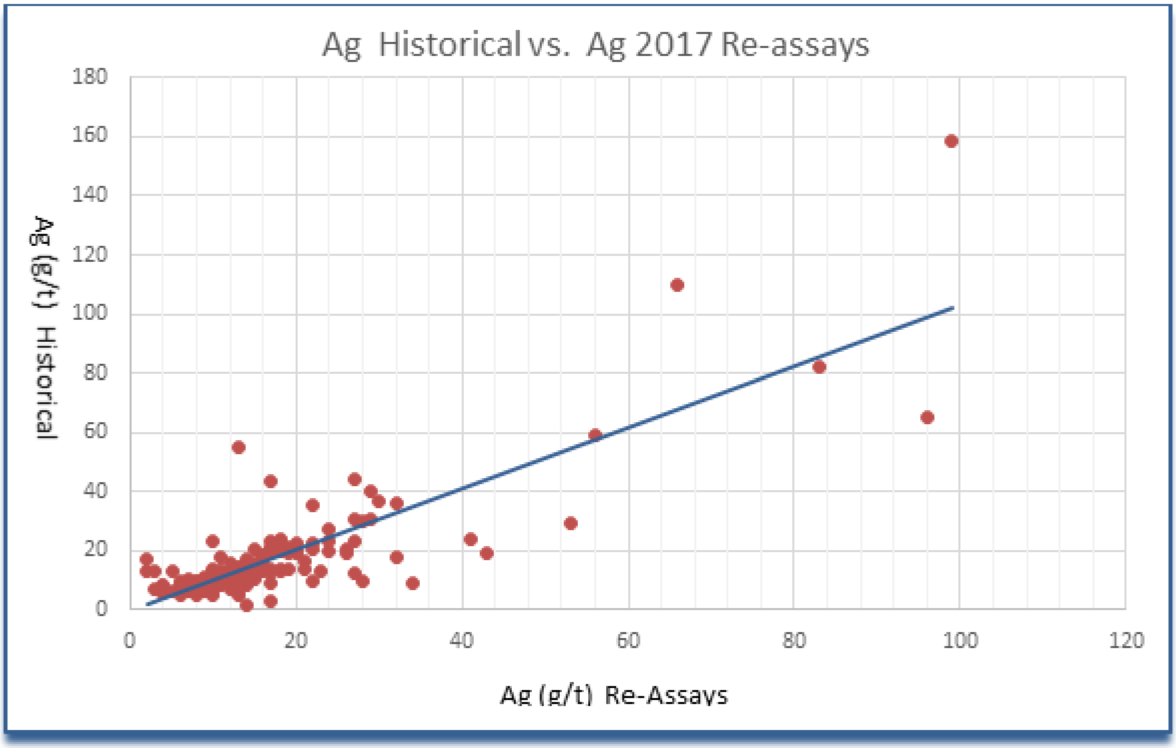


Figure 12.14 Comparison of Ag from historical assays in database versus ¼ core duplicate (re-assayed samples)



12.5 CONCLUSION

The data verification completed by the QP has led to confidence in the Aura Minerals' database which was compiled previously by Aura Minerals. The Author believes that the database is suitable for use in the resource estimation reported herein. In the Author's opinion there is no bias in nature of data and complied database.

As discussed in Item 11, the Author recommends that Aura Minerals insert a set of high grade copper, gold and silver standards into stream of samples for further validation and verification of high grades intervals. The range of assays and its matrix need to be selected in such a way that they represent high grade mineralization for all three elements (if possible) and compatible with type of deposit.

Also, the results of pulp duplicates showed that the ALS lab may have some problem assaying high grade silver (>100 ppm). Since SGS was the primary lab for analysis, this is not major issue for resource estimation, but this needs to be addressed and mitigated if it is possible.

It is recommended that Aura Minerals designate a tertiary lab for additional validation purposes especially for high grade Cu, Au and Ag samples in future drilling programs to verify original data versus pulp duplicates.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

The Aranzazu mine and process plant have been owned and operated by Aura Minerals since 2008. At that time only ore from the open pit was processed but later transitioning to an underground operation in 2011 through to 2014. Limited resources remain in the open pit to this day.

The mine and plant were operating at the time of acquisition but shut down in January 2009 and restarted in 2010 with commercial operation being declared in February 2011. The operation was again put into care and maintenance early in 2015 due to low metal prices and has remained in this state until the present. There was some metallurgical testing done in 2015 consisting of Bond work index and laboratory flotation studies on drill core samples which showed that, with some modification to the circuit, the processing rate could be increased to 3090 TPD while the copper and gold recoveries in the flotation testing reported an average recovery for samples representing Years 1 and 2 of the then projected operation at 84.0 % copper, 59.4 % gold, 70.3% silver and 37.9% arsenic.

13.2 PREVIOUS OPERATION

The production data for the years 2011 to 2014 are shown in Table 13-1 The data indicates a steady ramp up in throughput and copper recovery over the four years of operation while copper and gold grades in ore were essentially unchanged over that period. It should be noted that the copper grade of ore processed was less than half the grade of ore projected to be produced from the mine in the future while gold grades are expected to be three to five times higher. This will result in higher copper recovery and in much higher gold recovery. This is also true of the arsenic grades in the ore and in the concentrate. Arsenic grades averaged 0.103% As over the last three years of operation while arsenic recovery averaged 32.2%. In future production, the arsenic in the ore is expected to vary from the levels observed in 2012 to 2014 to up to 0.3% arsenic.

Table 13-1 Summary of Results from Operations at Aranzazu 2011 to 2014

Beneficiation Plant - Milling		2011	2012	2013	2014
Dry Feed to the Mill	tonnes	631,472	773,872	796,413	941,461
days operated in year		364	356	356	364
average tonnes per day		1735	2174	2237	2586
Crushed Ore Moisture	%	5.03	5.78	4.94	4.95
Copper Grade	**%**	**0.868**	**0.859**	**0.984**	**0.863**
Copper Grade (Oxide)	%	0.324	0.111	0.101	0.103
Copper Grade (Sulfide)	%	0.544	0.748	0.881	0.759
Gold Grade	**g/t**	**0.477**	**0.477**	**0.461**	**0.426**
Silver Grade	g/t	12.19	12.00	16.08	14.17
Arsenic Grade	%	na	0.13	0.10	0.09
Average Operation	h/day	22.14	22.64	22.85	23.70
Availability	%	93.55%	96.66%	95.73%	98.06%
Utilization	%	73.44%	84.68%	87.79%	97.07%
Total Productivity*	%	33.55%	93.40%	94.73%	108.84%
Beneficiation Plant - Flotation					
Concentrate Production	dry tonnes	13,152	20,681	25,827	28,101
Concentrate Moisture	%	11.76	11.16	10.25	10.53
Conc. Copper Grade	%	25.59	24.34	23.85	23.06
Conc. Gold Grade	g/t	13.63	10.12	8.32	8.16
Conc. Silver Grade	g/t	300.05	248.50	256.55	257.5
Conc. Arsenic Grade	%	na	1.53	0.924	0.969
Total Copper Recovery	**%**	**61.41**	**75.75**	**78.61**	**79.76**
Copper as Oxide Recovery	%	12.02	29.19	53.35	41.64
Copper as Sulfide Recovery	%	76.47	80.15	80.70	82.45
Gold Recovery	**%**	**59.48**	**56.66**	**58.45**	**57.14**
Silver Recovery	%	51.25	55.34	51.73	50.96
Arsenic Recovery	%	na	32.30	30.89	33.56
Contained Metal Production					
Copper in Concentrate	tonnes	3367.61	5024.28	6154.96	6466.94
Gold in Concentrate	kg	170.69	220.31	214.34	228.66
Silver in Concentrate	kg	3877.12	5127.76	6619.64	7222.81

13.3 METALLURGICAL TESTING

13.3.1 Background

The ore body in the Aranzazu deposit consists of skarn copper/gold/pyrite lenses spread over many distinct ore zones. Only the Glory Hole, AA, Mexicana South and BW zones with a small contribution from the open pit will be mined in the envisaged operation. The main copper mineral is chalcopyrite with significant (up to 15%) bornite and chalcocite. Deeper ore zones have substantial amounts of enargite (Cu_3AsS_4) and very minor tennantite, $Cu_6[Cu_4(Fe,Zn)_2]As_4S_{13}$. While the ore has generally high pyrite (20% to 50%), the pyrite is thought to have been laid down in a separate event from the copper minerals and gold and, as a result, there is very little locking of the values with pyrite. As the ore will be produced from the deeper zones of the deposit, the arsenic will become an increasing challenge since the enargite concentrates with other copper sulfide minerals and can cause the concentrate to exceed 3.0% arsenic in grade at which time penalties become prohibitive. Penalties for arsenic content become very significant when the concentrate grade exceeds 2.0% arsenic so controlling the arsenic grade of the concentrate will be a requirement for the operation.

The critical parameter in the ore is the copper to arsenic ratio. If this ratio is less than 7.7 the concentrate produced will almost certainly have an arsenic content in excess of 3.0%. Therefore, spikes in the Cu/As ratio in ore will need to be avoided in the mine plan. A further consequence of the arsenic content is that the copper grade of the concentrate has to be kept as close as possible to the lower marketable limit (23% Cu) in order to maximize the dilution of the arsenic.

13.3.2 Test Program

Aura Minerals conducted a major test program during the fall of 2017 at ALS Laboratories in Kamloops B.C. in an attempt to find a reagent scheme which would provide a separation of enargite from the other copper bearing minerals and gold. The test work involved first establishing that older core had not degraded over time and was thus suitable for inclusion in a composite sample for the main body of tests. There were a series of 13 rougher flotation tests and 16 cleaner flotation tests where grind size and various reagents were tested in an effort to find a suitable separation of enargite from the other copper bearing minerals and gold. A further 11 rougher flotation tests were carried out on the variability samples to determine if finer grinding, identified as the best route to arsenic separation, was viable over the parts of the ore body represented by the variability samples and thus by interpolation, over the parts of the ore body in the mining plan.

13.3.3 Sample Selection

The sample requirement was for about 200 kg of ore that would be representative of the Glory Hole ore body plus variability samples of 40 kg mass each that would be representative of the various domains within the Glory Hole and the other much smaller surrounding ore bodies (AA, Mexicana South and BW). The latter will provide 15% of the ore. The 200 kg sample was needed to provide a general composite for use in the arsenic rejection study and to optimize conditions for the rougher and scavenger flotation. As there was not sufficient core available in the 2017 drilling to fulfill these requirements, it was necessary to include core from the previous drilling. To ensure that the metallurgical performance of the older core had not been compromised several pairs of holes from recent and old drilling were also selected for comparison testing.

Both historical (2008 et seq.) and 2017 drill holes, the latter drilled mainly for geotechnical reasons, have been stored securely in the Aranzazu core shack facility in the town of Concepcion Del Oro. In general, the core recovery has been good, averaging 96%. However, there were some areas usually in the hornfels lithological zone that showed core recovery less than 90%.

Metallurgical sample preparation included splitting half drill core using a diamond saw blade into ¼ core and the quarters were placed in secure buckets with appropriate labels for their content including hole number, sample intervals and prepared for shipment to the ALS laboratory. The remaining quarter of the core was placed back into the core box for retention. The core pans were cleaned between each split of samples to avoid contamination from previous samples. A geologist, responsible for logging, monitored the core cutting to ensure that the samples were correctly taken.

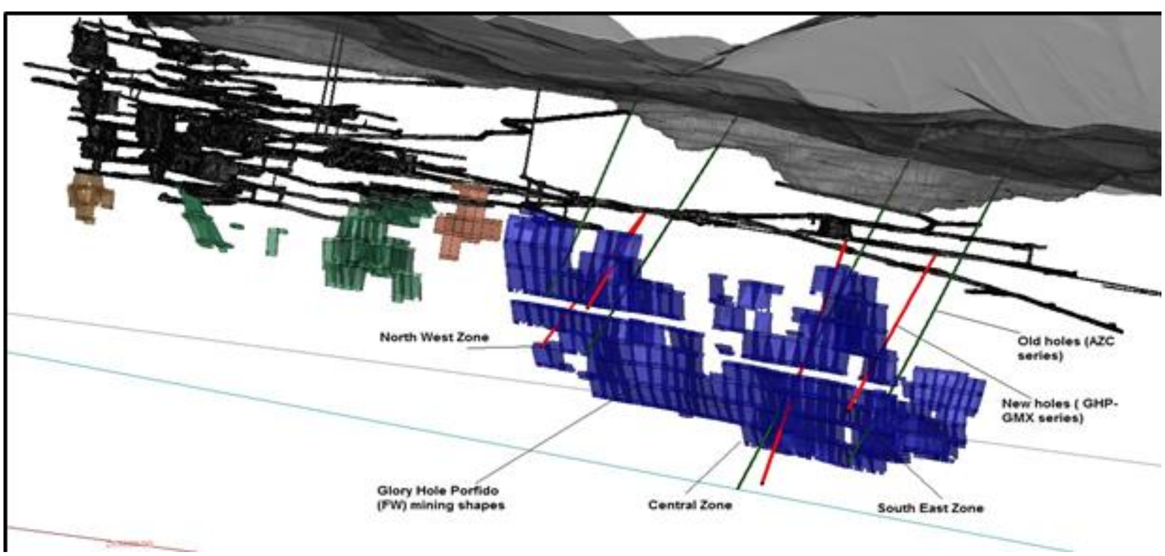
Drill core intervals from all zones in the Aranzazu mining plan were evaluated for their suitability for inclusion in the metallurgical samples based on their grade and location within the ore body. As there was insufficient core available from the recent drilling to carry out the entire program, the initial study was carried out to compare the flotation properties of the older core with that of that of the recent core. The four pairs of samples shown in Table 13-2 below, from old and new core that were spatially close and in the same lithology, were tested using grind and reagents that had been employed in the plant at shutdown in 2015 in laboratory flotation tests.

Table 13-2 Paired Samples of Old and Recent Core for Tests of Aging Effect on Flotation

Zone	Hole-ID	From	To	Length (m)	Location
GHP	GHP-GMX-07	152	162	10	NW corner
GHP	AZC-075	242	252	10	NW corner
GHP	GHP-GMX-04	220	230	10	Central zone
GHP	AZC-182	410	420	10	Central zone
GHP	GHP-GMX-06	80	92	12	NW corner
GHP	AZC-141	350	362	12	NW corner
GHP	GHP-GMX-02	180	192	12	SE Corner
GHP	AZC-210	420	431	11	SE Corner

Once it was clear that older core was acceptable for use, a composite was prepared that is representative of the ore that will be mined at Aranzazu in the next three years from the Glory Hole. This will represent 85% of the ore to be mined in the projected five years of operation.

Figure 13.1 Glory Hole Zone Showing Stopes and Location where Sample Core was Extracted



Samples were taken as per Figure 13.1 and selected for variability testing and for grinding testwork from each of the planned mining areas. These samples were selected from available core from proposed mining stopes including BW, Mexicana, and AA and Glory Hole Porfido zones and designed to be as spatially representative as possible within those areas. The samples selected can be seen in the following table, Table 13-3.

The lithology is limestone, marble, Exo-skarn and hornfels in the hanging wall zone and Endo-Skarn and granitic intrusive in the footwall.

Table 13-3 Variability Samples selected

Zone	Hole-ID	From	To	Length (m)
BW	UAZ-045	106	123	17
BW	UAZ-038	100	117	17
Mexicana South	AZC-102	300	316	16
Mexicana South	AZC-062	270	286	16
Mexicana South	AZC-016	278	288	10
AA	AZC-028	340	352	12
AA	AZC-029	320	330	10
AA	AZC-040	346	364	18
GHP	AZC-088	340	352	12
GHP	AZC-204	320	330	10
GHP	AZC-190	346	364	18
GHP	AZC-157	350	364	14
GHP	AZC-173	273	288	15
GHP	AZC-147	350	362	12
GHP	AZC-139	328	350	22
GHP	AZC-103	203	220	17
GHP	AZC-093	288	302	14

A general composite was selected from the remaining drill core in the Glory Hole Porfido zone, which was designed to be used for the bulk of the testwork to be performed. This selection can be seen in Table 13-4 below.

Table 13-4 Core Used in the General Composite Sample

Zone	Hole-ID	From	To	Length(m)
GHP	AZC--108	275	294	19
GHP	AZC-132	336	344	8
GHP	AZC-142	360	370	10
GHP	AZC-117	327	338	11
GHP	AZC-121	351	357	6
GHP	AZC-176	318	344	26
GHP	GHP-GMX-03	130	148	18
GHP	AZC-213	366	386	20
GHP	AZC-112	285	300	15

13.4 MINERALOGY

The composite was subjected to QEM-Scan analysis to confirm that the mineralogy of the copper bearing minerals was the expected mixture of chalcopyrite ($CuFeS_2$) bornite, Cu_3FeS_2 chalcocite (Cu_2S), enargite (Cu_3AsS_4) and to determine the liberation size for the enargite binaries. The elemental and overall mineralogical compositions are shown in Table 13-5. This indicates the relatively high pyrite content of the ore with iron oxide (principally magnetite), garnet and quartz as the non sulphide gangue components.

It was shown that adequate liberation should occur at a top size of about 53 microns as shown in Figure 13.2. It was noted that approximately 25% of the copper was contained in enargite so that any separation of the arsenic bearing component of the ore would result in a significant loss of copper unless a separate high arsenic copper concentrate could be produced.

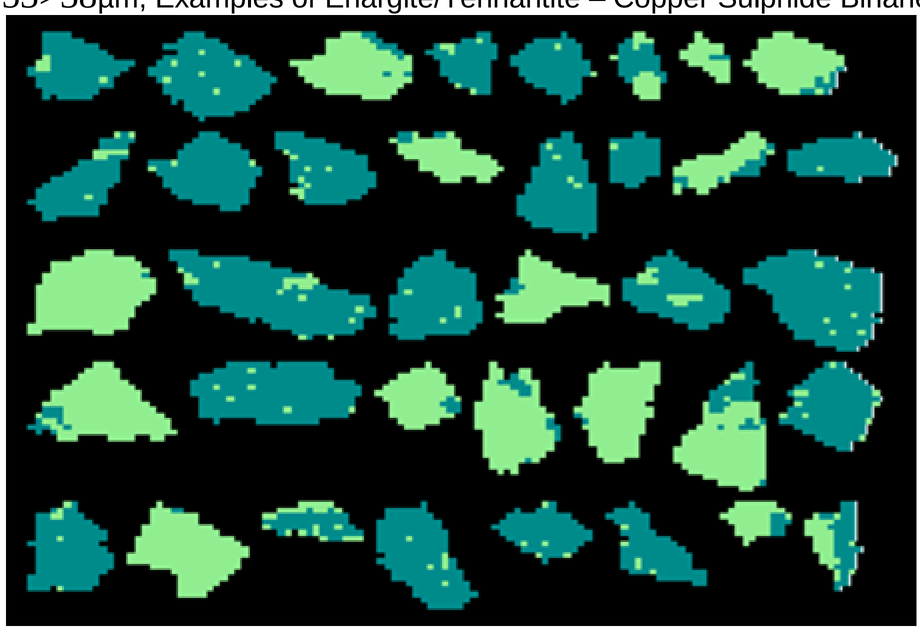
Table 13-5 Assay and QEM-Scan Data – General Composite

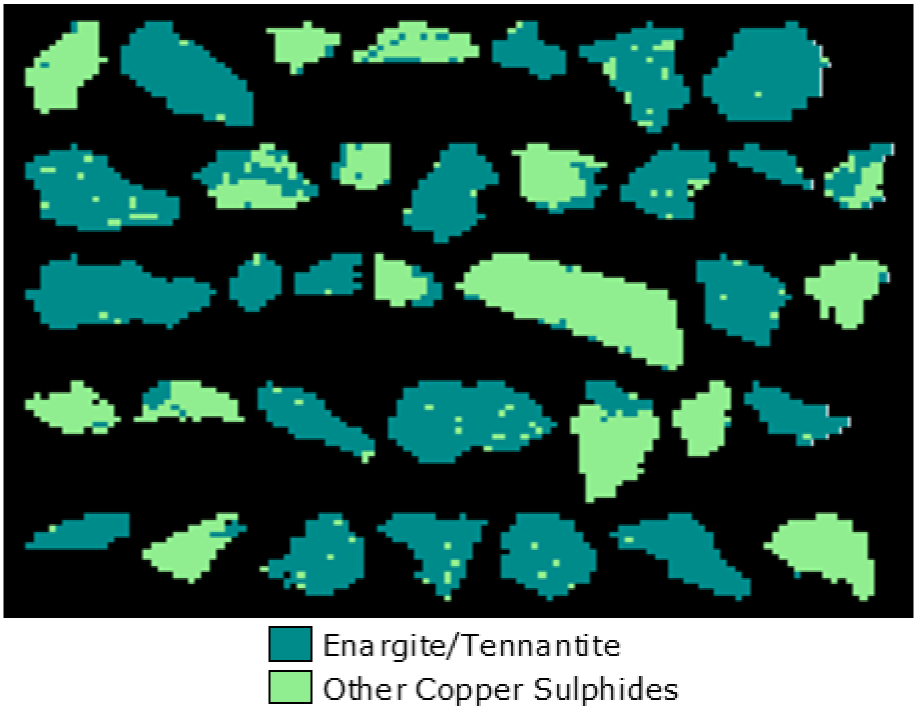
Element Assayed - percent or g/t					
Cu	Cu(ox)	Fe	S	As	Au
1.77	0.18	26.8	19.5	0.23	2.16

Chalcopyrite		%	3.3
Bornite		%	0.2
Chalcocite/Covellite		%	0.2
Enargite/Tennantite		%	1.0
Pyrite		%	32.9
Sphalerite		%	0.3
Molybdenite		%	0.1
Iron Oxides		%	16.2
Garnet		%	21.3
Quartz		%	11.2
Other Non-Sulphide Gangue		%	13.3
CuS Liberation		%	49
Primary Grind SIzIng P80		µm	207
%Cu within Enargite			25

Figure 13.2 QEM-Scan False Colour Images of -53+38 Fraction of Sulphides and Arsenides in the General Composite

<53>38µm, Examples of Enargite/Tennantite – Copper Sulphide Binaries

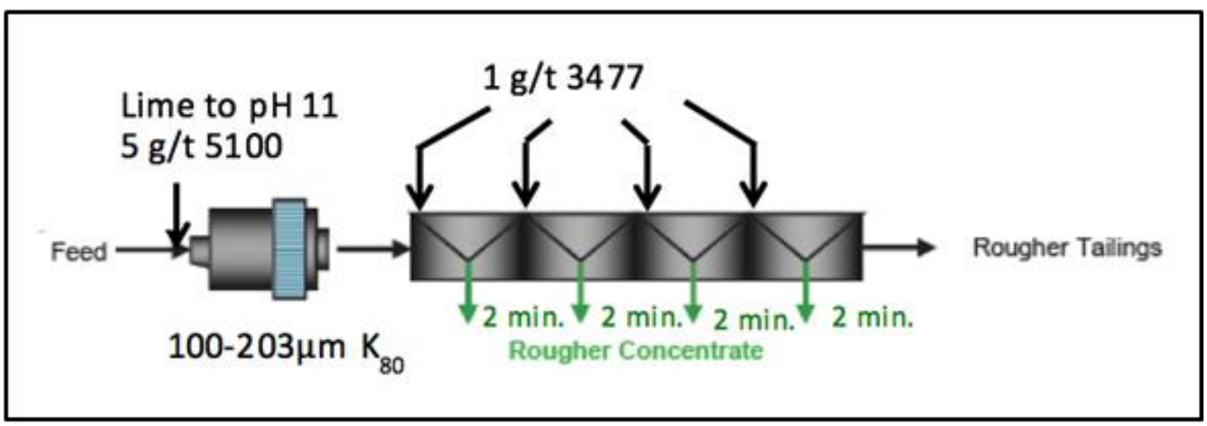




13.5 FLOTATION TESTING OF OLD AND RECENT DRILLED CORE

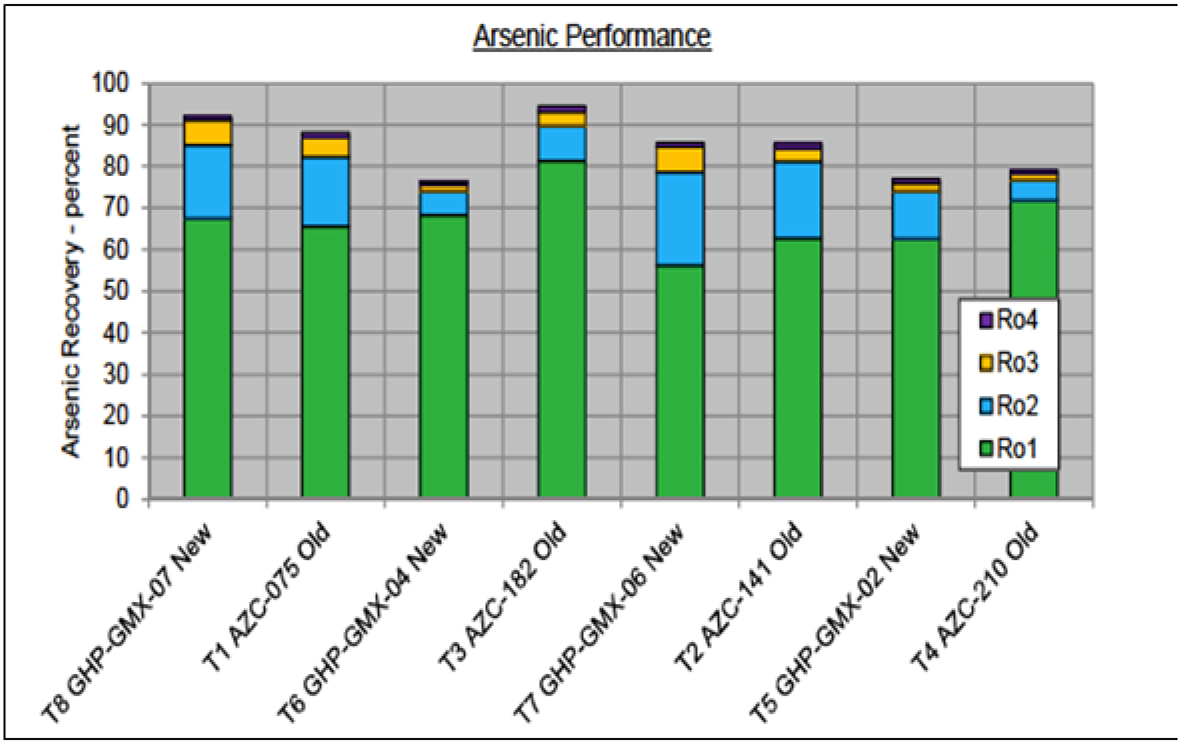
The flowsheet used in the test work, shown in Figure 13.3 below, which, as noted above, was that used for testing when the plant was operating up until early 2015 with the grind p80 being 200 microns for this initial work. Cytec 5100 (thionocarbamate) and Cytec 3477 (dithiophosphate) are used as collectors for copper bearing minerals because of their selectivity over pyrite.

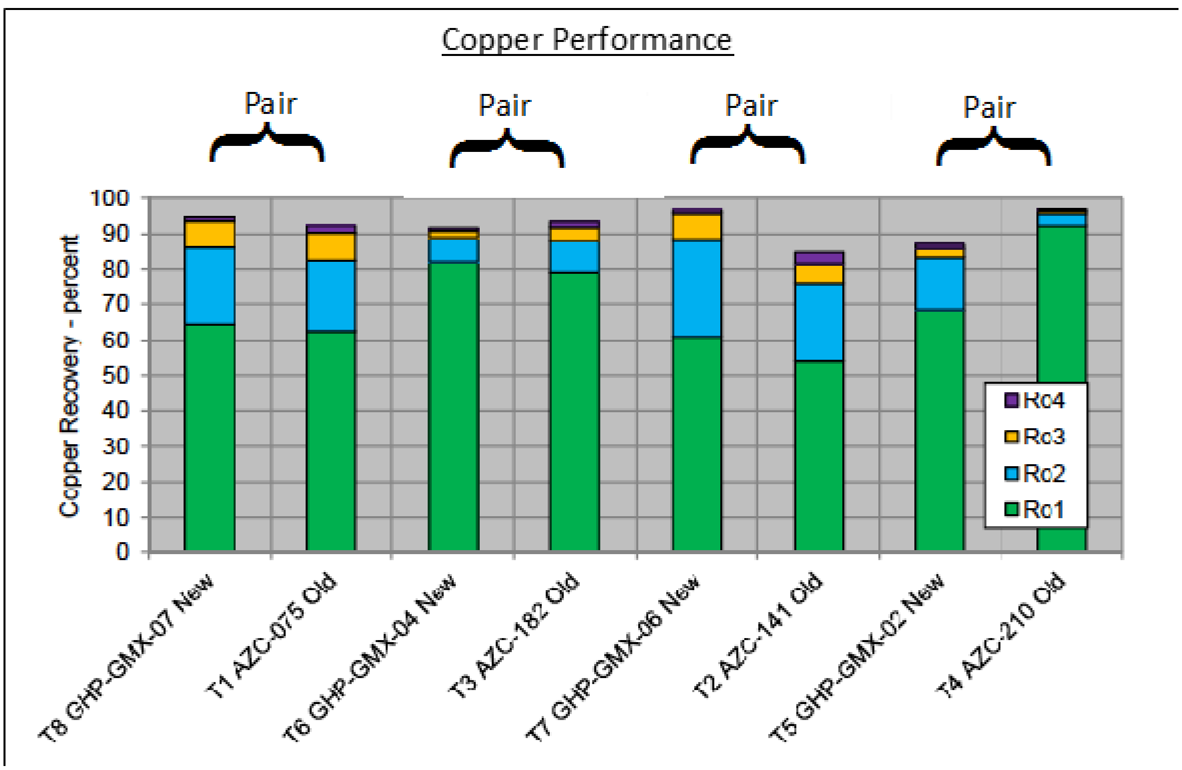
Figure 13.3 Flowsheet Used in Rougher Flotation Testing



The results shown in Figure 13.4 indicate that all sample pairs from the recent and earlier core gave similar recovery of both copper and arsenic. Any differences would be attributable to differences in the head grade rather than to the condition of the core.

Figure 13.4 Rougher Recoveries of Copper and Arsenic from Samples Paired in Location and Lithology





13.6 ROUGHER FLOTATION STUDIES WITH GENERAL COMPOSITE

Table 13-6 shows the results of various tests, done using the general composite, with the flowsheet shown in Figure 13.3. These tests examined the effect of a finer primary grind and changes in collector type and concentration on the rate of flotation of copper sulfides compared to that of enargite (as indicated by the arsenic content.)

Table 13-6 Test Conditions and Summary Results of Rougher Flotation Study

Test Objective	Test #	Collector to mill	Prim. Grind k80 (um)	Collector to Roughers	Conc. Grade after 4 min. flotation		Rec'y after 4 min flotation (%)				Cu/As (Conc)
					Cu (%)	As%	Cu	Au	As	Cu-As	
Baseline	10	5100	207	3477	21.5	2.65	82.7	61.7	80.2	2.5	8.09
Starvation collector	11	-	207	3477	19.9*	2.32*	77.6*	71.6*	78.6*	-1.0	8.57
Finer primary grind	12	5100	148	3477	22.1	2.54	86.9	71.2	82.2	4.6	8.72
Lower collector dosage	20		148	SIPX/SEX	20.5	3.52	67.4	56.8	71.3	-3.9	5.81
100um K80 flot feed	27		103	SIPX/SEX	23.0	2.83	89.0	71.9	82.0	7.1	8.12
54um K80 flot feed	28		54	SIPX/SEX	23.2	2.43	89.6	75.2	79.6	10.0	9.54
5100/3477 collector	37	5100	103	3477	22.5	2.56	89.8	78.3	83.0	6.7	8.79

* after 10 minutes flotation

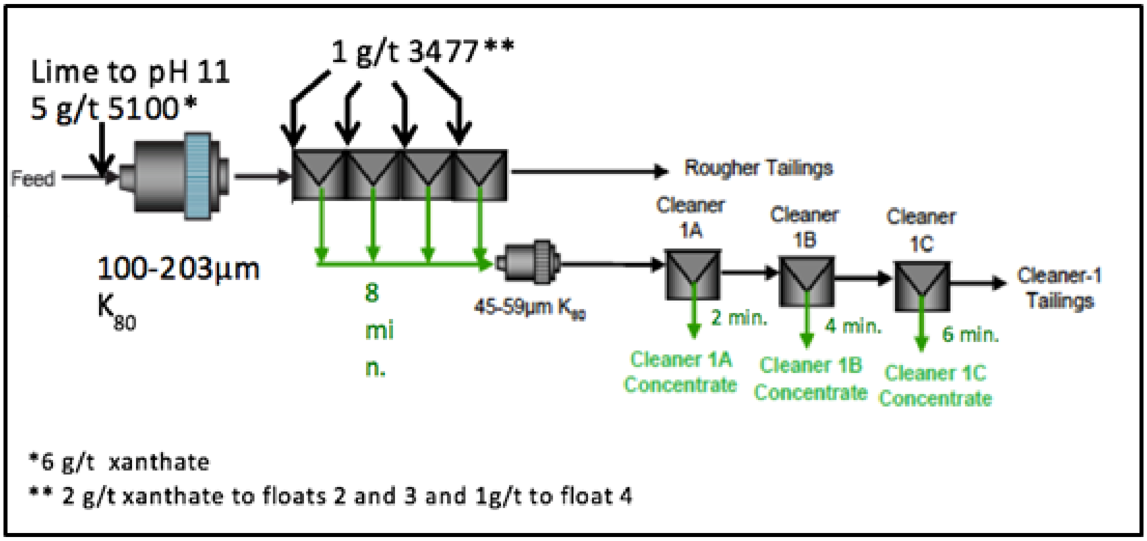
As can be noted in Table 13-6, Test 11, reducing the collector dosage had a detrimental effect in that the arsenic bearing mineral recovery was not reduced as much as for the non-arsenic bearing copper minerals. Gold recovery benefited from the much longer flotation time used in Test 11. A finer primary grind, however, was clearly beneficial to selectivity for copper sulfides over the copper sulfarsenides with the effect increasing as the primary grind was made even finer. Comparison of Test 27 which had a primary grind p80 of 103 microns where sodium isopropyl and ethyl xanthates (SIPX/SEX) (50% by weight mixture) were the collectors with Test 37 which had the same grind but 5100 and 3477 collector, shows that there was only a 0.4% decrease in the selectivity for copper over arsenic but there was an increase in the overall copper recovery and a dramatic increase of 6.4% in the gold recovery clearly making the 5100/3477 the preferred collector combination.

A few tests were conducted with addition of peroxide to the primary grind following the indication of some selectivity being generated in the cleaner tests, but the rougher tests showed little or no selectivity for copper sulfides over enargite

13.7 CLEANER FLOTATION STUDIES

The flowsheet shown below in Figure 13.5 was used for the cleaner tests. The results are summarized in Table 13-7.

Figure 13.5 Flowsheet for Cleaner Testing



The addition of the depressant to conditioning in Tests 17, and 18 had minimal effect on the copper recovery compared to arsenic recovery. An increase in selectivity was observed when hydrogen peroxide (Test 16) was added but the change was marginal. Sodium sulfide added using the REDOX potential as a measure of when sufficient reagent had been added also increased the difference in recovery (Test 19) but overall copper recovery was reduced and the test was not considered sufficiently promising to continue working with the sulfide at that time. It is of interest to note that the combination of diethylene triamine (DETA) in combination with sulfur dioxide yielding species, which has been patented as an arsenic depressant, in this case (Test 18), appears to have activated the enargite and increased copper recovery.

The tests from #23 on were designed to optimize the amount of peroxide addition with the reagent added to the regrind mill to ensure adequate conditioning. From Test 31 500 g/t of peroxide with SIPX/SEX collector gave the best arsenic depression but at a cost of approximately 2.0 % copper recovery. It was expected that a decrease of the peroxide addition to 200 g/t (Test 32) at a finer regrind would give better copper recovery while still providing adequate arsenic rejection but the recovery was in fact lower than at the coarser regrind with the higher dose of peroxide. Test 36 where peroxide was added in both primary and regrind mills gave adequate arsenic rejection but poor overall copper recovery.

A further cleaner test (Test 35) – with material balance shown in Table 13-8, using sodium sulfide as the depressant but with a much higher collector addition (30g/t) than in Test 19, showed considerable potential for there to be a route to a copper/arsenic separation. In this test, the B concentrate had an arsenic assay of 7.45% equivalent to an enargite content of 39%. The risks in implementing such a separation were seen to be too high at this stage of process development and the line of investigation has therefore been deferred until the plant is in operation.

Table 13-7 Results of Cleaner Tests on the General Composite Sample with Arsenic Depressants

Test #	Collector		Regrind (microns)	pH (cleaner)	Arsenic Depressant		Depressant addition point	Overall Rec'y to Clnr Conc (%)		
	Rougher	Cleaner			Compound	dosage (g/t)		Cu	Au	Cu-As
13	5100/3477	5100/3477	46	10.7 to 10.2	none	-	-	87.1	56.7	4.8
16	5100/3477	5100/3477	52	10	Peroxide	85	conditioning	89.1	59.1	6.3
17	5100/3477	5100/3477	52	10	Na2(ClO)3	43	conditioning	89.1	60.7	4.8
18	5100/3477	5100/3477	52	10	DETA/SO2	17-May	conditioning	90.1	58.3	4.5
19	5100/3477	SIPX	55	11.6	NaS2	220	conditioning	87.5	62.8	6.6
23	SEX/SIPX	SEX/SIPX	47	10	Peroxide	340	regrind	87.6	58.3	7.7
25	SEX/SIPX	SEX/SIPX	45	10	Peroxide	1000	regrind	85	66.5	9.6
29	5100/3477	5100/3477	50	10	Peroxide	85	regrind	89.4	75	6.5
31	SEX/SIPX	SEX/SIPX	44	10	Peroxide	500	regrind	87.8	62.6	10.7
32	SEX/SIPX	SEX/SIPX	38	10	Peroxide	200	regrind	86.5	57	8.1
34	SEX/SIPX	SEX/SIPX	41	10	none	-	-	88.6	57.7	7.1
35	SEX/SIPX	SEX/SIPX*	48	10.9 to 10	NaS2	662	conditioning	90.1	70.7	6.9
36**	SEX/SIPX	SEX/SIPX	36	10	Peroxide	400	regrind	81.7	74	9.3

* Dosage increased to 15 g/t of both SEX and SIPX, ** Test 36 also had 400 g/t of peroxide added to the rougher grind.

Table 13-8 Material balance of Test 35 Sodium Sulphide depressant- with 30 g/t SIPX/SEX

Product	Weight	Assay - percent or g/tonne						Distribution - percent						
	%	Cu	Fe	S	Au	Ag	As	Cu	Fe	S	Au	Ag	As	Cu/As
Copper Con A	0.3	26.3	18.8	27.7	5.1	224	1.31	3.9	0.2	0.4	0.7	2.8	1.8	20.0
Copper Con B	0.7	44.2	10.2	26.2	24.6	474	7.45	15.2	0.2	0.9	8.0	13.6	23.2	5.9
Copper Con C	6.9	18.7	35.1	40.3	15.4	144	1.63	67.2	8.1	14.0	51.9	43.1	52.9	11.5
Copper Con D	0.7	10.6	32.2	33.3	9.0	284	1.68	3.7	0.7	1.1	2.9	8.3	5.3	6.3
Copper 1st Clnr Tail	2.1	1.83	20.8	3.40	6.9	56	0.24	2.0	1.5	0.4	7.2	5.2	2.4	7.6
Copper Ro Tail	89.3	0.17	30.0	18.5	0.67	7	0.03	7.9	89.3	83.2	29.3	27.1	14.4	5.0
Feed	100.0	1.92	30.0	19.9	2.04	23	0.21	100	100	100	100	100	100	

13.8 CONCLUSION FROM ROUGHER AND CLEANER TESTWORK

It was concluded that none of the reagent schemes tested would be effective in the Aranzazu mill either because of insufficient selectivity or because of the difficulty and cost of implementation of the scheme into the operation. It was further concluded that the lowest risk and most cost-effective procedure to increase selectivity for copper over arsenic would be to provide a finer primary grind.

13.9 LOCKED CYCLE TEST OF GENERAL COMPOSITE

A locked cycle test was performed on the general composite sample (Table 13-4) with the conditions shown in Table 13-9. Note that the reagent combination used was Cytec 5100 and Cytec 3477 that was proven to be best in the rougher tests reported above.

Table 13-9 Conditions used in Locked Cycle Test

FEED: 5 x 2 kg of General Composite ore ground to a nominal 148µm K_{80}.

Stage	Reagents Added g/tonne				Time (minutes)			pH	Redox	Air Flow (L/min)
	Lime	5100	3477	MIBC	Grind	Cond.	Float			
Primary Grind	2000	4			9			11.3	45	
COPPER CIRCUIT:										
Rougher 1	-		1	44		1	2	11.3	36	8
Rougher 2	-		1	-		1	2	11.2	34	8
Rougher Scavenger 1	-		1	-		1	2	11.2	24	8
Rougher Scavenger 2	-		1	-		1	2	11.2	32	8

Flotation Data	Rougher
Flotation Machine:	Denver
Cell Size in liters:	4.4
Aspiration:	Air
Water Type:	Fresh
Impeller Speed in rpm:	1100

Grinding Data	Primary Grind
Mill:	M5-Mild
Charge/Material:	20kg-Mild
Water:	1000ml

The results of the locked cycle test are given in Table 13-10 below which shows the final two cycles and cycles 4 and 5 combined. From these results, it can be seen that a concentrate grading 21% Cu, 14 g/t Au and 216 g/t Ag can be produced with recoveries in excess of 90% for copper and 69% for gold and silver. This recovery is downgraded to 88% for copper to allow for the slightly higher concentrate grade for ease of sale.

Table 13-10 Results of Locked Cycle Test

Product	Weight	Weight	Assay - percent or g/tonne						Distribution - percent					
	%	g	Cu	Fe	S	Au	Ag	As	Cu	Fe	S	Au	Ag	As
CYCLE 4														
Flotation Feed	100.0	1965	1.99	27.4	19.3	1.74	26	0.22	100	100	100	100	100	100
Copper Ro Con	8.8	172.7	20.9	23.7	26.6	13.4	216	2.15	92.2	7.6	12.1	67.6	72.2	85.9
Copper Ro Scav Tail	91.2	1792	0.17	27.8	18.6	0.62	8	0.03	7.8	92.4	87.9	32.4	27.8	14.1
CYCLE 5														
Flotation Feed	100.0	1962	1.99	27.5	19.2	1.81	25	0.22	100	100	100	100	100	100
Copper Ro Con	8.5	167.4	21.5	24.0	26.6	15.2	216	2.20	92.2	7.5	11.8	71.7	74.2	85.4
Copper Ro Scav Tail	91.5	1795	0.17	27.8	18.5	0.56	7	0.04	7.8	92.5	88.2	28.3	25.8	14.6
CYCLES 4 and 5														
Flotation Feed	100.0	3927	1.99	27.5	19.2	1.78	26	0.22	100	100	100	100	100	100
Copper Ro Con	8.7	340.1	21.2	23.8	26.6	14.3	216	2.17	92.2	7.5	12.0	69.7	73.2	85.7
Copper Ro Scav Tail	91.3	3587	0.17	27.8	18.5	0.59	7	0.03	7.8	92.5	88.0	30.3	26.8	14.3

13.10 TESTING OF FINER PRIMARY GRIND ON VARIABILITY SAMPLES

Following the conclusion noted above, all of the variability samples (Table 13-3) were tested using a grind p80 of ~150 microns. In addition, the high arsenic samples from the Glory Hole Porfido (GHP) were tested at a P80 of 100 microns to examine the extent to which the finer grind would improve the separation. The Cytec 5100/Cytec 3477 collector combination was used because of the better recovery of gold seen compared to the mixed xanthate suite. The results are shown in Table 13-11 below:

Table 13-11 Variability Sample Test Results at a Primary Grind p80 150 to 100 microns

Sample	Head grade			Pri Grind	Ro Conc. Grade 1-2		Recovery to Cu Ro Con. 1-2 (%)			
	Cu(%)	As(%)	Cu/As	um K80	%Cu	%As	Cu	Au	As	Cu-As
GHP-Pillar Center	2.36	0.35	6.73	159	23.9	3.45	92.5	76.9	88.8	3.6
GHP-Pillar Center 100um	2.36	0.35	6.73	100	24.2	3.45	93.9	86.7	90.2	3.7
GHP-Pillar East	2.41	0.31	7.67	157	23.1	2.98	86.0	66.5	85.2	0.9
GHP-Pillar East 100um	2.41	0.31	7.67	93	22.5	2.70	95.7	79.7	90.3	5.4
GHP Pillar West	2.35	0.18	13.09	155	22.9	1.52	91.7	69.0	88.1	3.6
GHP Pillar West 100um	2.35	0.18	13.09	95	22.9	1.55	92.2	79.6	81.7	10.5
AA Composite	1.82	0.25	7.33	160	23.7	2.86	86.2	62.0	76.3	9.9
Mexicana South	2.89	0.21	13.47	163	29.9	1.32	94.8	90.4	56.3	38.4
BW Comp	2.39	0.01	161	149	27.1	0.02	97.5	80.2	13.1	84.4

All of the variability samples have a higher copper feed assay than the general composite samples with the exception of the AA composite. The Pillar Center and Pillar East samples have a much higher arsenic feed grade than the general composite. The finer grind on the GHP samples (to ~100 microns) in all cases gave somewhat better copper recovery and dramatically improved gold recovery compared to the 150 micron grind. As expected from the head grade Cu/As ratio being below 7.7, the GHP-Pillar Center and GHP-Pillar West samples produced concentrates with more than 3.0% arsenic.

It became clear that the critical Cu/As ratio of 7.7 in feed identified in tests on the general composite (which was comprised of samples from the GHP) only applies to the GHP sample as the AA Composite although having a feed Cu/As below this ratio, produced a concentrate grading 2.86% As. The Mexicana South sample similarly gave an arsenic recovery at 56.3%, very much lower than observed in any of the GHP samples and therefore a low arsenic grade in the concentrate. Further testing will be required to determine whether similar arsenic recoveries are obtained across these ore zones which would result in a different critical Cu/As ratio for these ores.

13.11 GOLD RECOVERY TO BE USED FOR FINANCIAL MODEL

Because the head grade for gold in the composite at 1.74g/t was substantially higher than the expected mining grades over the LOM (1.17 g/t), it was perceived that there needed to be a determination of the effect of feed grade on recovery and, therefore, the gold recoveries obtained in previous operations from 2011 to 2014 and from the latest testwork, including the composite and the variability samples were plotted against the head grades. The result, as shown in Figure 13.6 while showing a relatively low Index of Determination at 0.25, do suggest a modest increase in recovery with grade and this graph has been used to calculate the gold recoveries to be expected for the head grades expected to be delivered in the mine plan. These calculated recoveries have also been used in the financial model and are shown in Table 13-12 and Figure 13.6.

Figure 13.6 Gold Recovery vs Head Grade

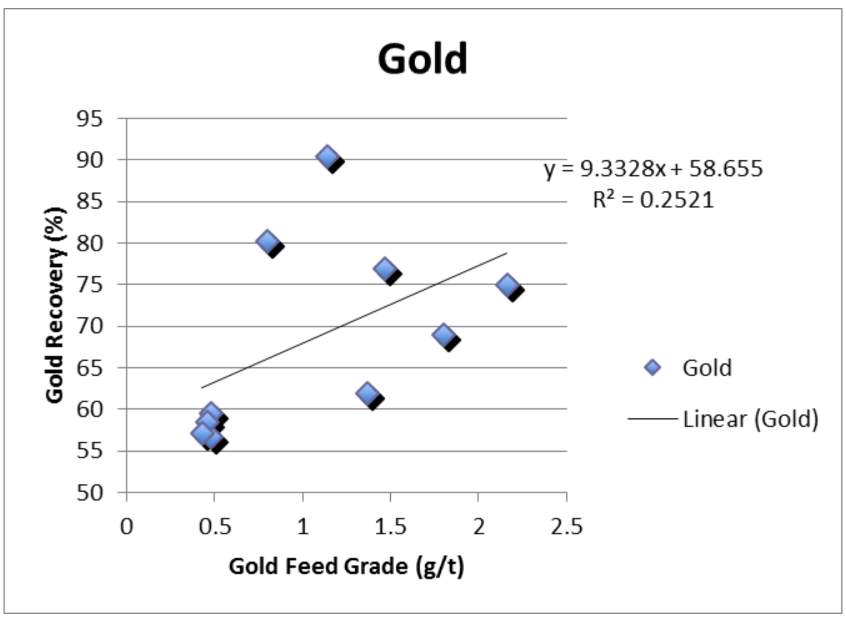


Table 13-12 Gold Grades and Recoveries used in the Financial Model

Year	Au Grade	Au Rec.%
1	0.88	66.87
2	1.42	71.91
3	1.33	71.07
4	0.98	67.80
5	0.97	67.71
6	1.35	71.25

13.12 GRINDABILITY STUDIES

When it became clear that a grind finer than that achieved in the previous operation of the plant (p80 ~207 microns) would be required, the general composite and the variability samples ball mill Bond work index was measured using an 80% passing screen size of 172 microns which would be appropriate for a ball mill grinding to a P80 of 150 microns. An even finer grind was chosen to determine how much extra grinding power would be required to get to an even finer grind (p80 of 100 microns) using an 80% passing size of 125 microns in the Bond test. The results, shown in Table 13-13, indicate that the Glory Hole ore does become more difficult to grind at finer sizes.

Table 13-13 Grindability of Variability Samples

Sample	Bond WI @172um (closing Screen)	Bond WI @125um (closing Screen)
	(kwh/tonne)	
GH Pillar West	10.5	11.4
GH Pillar Centre	10.2	11.7
GH Pillar East	9.3	11.0
BW Composite	7.5	
Mexicana South	9.6	

Using the average work index for the Glory Hole Samples and making the Bond correction for single stage ball mills, it has been calculated that the three primary mills and the regrind mill can grind the ore to a p80 of 135 microns which would be expected to give recoveries intermediate between those obtained at 150 microns and those at 100 microns in the laboratory tests. The grinding rate is assumed to be 116.7 tph with a circuit utilization of 92% operating for 365 days per year and would result in the processing of 940,240 tonnes/year which is required in Years 4 and 5 of the mine plan.

13.13 OVERALL CONCLUSIONS

A reagent to affect a separation of arsenic bearing minerals from copper sulfides was not identified.

It was found that finer primary grinding of the ore increases the overall recovery of the copper sulfides and gold significantly but does not increase the arsenic recovery to the same extent. It has been determined in open circuit tests that, for the general composite, grinding of the ore to 80% passing 150 microns compared to the previous 207 microns will increase the recovery of copper and gold in the concentrate from 82% to 88% and 62% to 70% respectively while only increasing the arsenic recovery from 80% to 82%. It was further observed that an even finer primary grind to a p80 of ~100 microns increased the copper and gold recoveries to 90% and 78% while As recovery increased only to 83%. The composite sample used in these tests was made from ore from the Glory Hole Profido zones only which represent 85% of the ore body scheduled to be mined in the next three years. Testing of variability samples showed that for the very high arsenic ores (0.3% As or higher) in this deposit, the finer grind will increase recovery of pay metals but also results in very high arsenic recovery and the differential (Cu recovery – As recovery) becomes smaller as the head grade increases. Grinding calculations indicate that the present primary circuit, with the addition of the regrind mill, can grind the ore to a p80 of 135 microns at the maximum required throughout so the expected recoveries will be intermediate between those observed with the 150 and 100-micron grinds.

A locked cycle test using a grind size (P80) of approximately 150 microns was performed and resulted in a concentrate grading 21% Cu, 14 g/t Au and 216 g/t Ag with recoveries in excess of 90% for copper and 69% for gold and silver. This recovery is downgraded to 88% for copper to allow for the slightly higher concentrate grade for ease of sale.

14. ARANZAZU MINERAL RESOURCE ESTIMATE

The current Aranzazu Mineral Resource estimate was prepared by Aura Minerals and audited by Farshid Ghazanfari, P.Geo. (a Qualified Person as defined by NI 43-101 with respect to the Mineral Resource estimates contained in this Technical Report). The Mineral Resource estimate was completed in March 2015.

The estimates were made from a 3-D block model utilizing commercial mine planning software (MineSight® 9.5) using Inverse Distance Weighting (IDW) interpolation. The Project limits are based on the UTM coordinate system, with the model rotated so that the 5.0 m x 5.0 m x 5.0 m sized blocks are aligned with the strike of the deposit.

The geological interpretation considers the Aranzazu deposit to be a copper porphyry deposit hosted primarily in skarn altered rock. This deposit contains copper, gold and silver mineralization as well as other minor uneconomic metals such as molybdenum, zinc, lead, arsenic, bismuth and antimony. The Mineral Resource estimate considers copper, gold and silver grades only; however, arsenic was also considered because it is a deleterious element that may alter the economic criteria for metal extraction.

The previous Mineral Resource estimate and Technical Report were completed in November 2011 by Micon International Limited. Since that time, a number of significant changes have been incorporated into the current resource estimation methodology, including:

- Addition of 60 drills holes for a total of 4,969.25 m
- Re-interpretation of the lithological, structural and alteration models
- Geological maps on 28 level plans
- Separation of the mineralization into zones that better reflect local changes in strike and dip of the mineralization as well as honouring grade continuity of the deposits
- Interpolation of arsenic

The mineralization solids were redone to encompass mineralization at an NSR cut-off of $45/t using the formula: NSR ($/t) = (Cu% x $56.32) + (Au g/t x $22.15) + (Ag g/t x $0.35) - $17.71. Detail of NSR cut off calculation which used for above formula was presented in the 2015 Technical Report. Since that time, the NSR cut off calculation was updated based on new terms, but the wireframe remained the same. A detail of new cut off calculation is presented in Item 14-10. The changes to the geological model by using the new approach defined by a set NSR cut-off grades have produced nine mineralization domains.

The Mineral Resource estimate has been generated from exploration drill hole assays capped and composited to 2.0 m nominal lengths, using the updated mineralization domains that better define the spatial distribution of copper, gold and silver mineralization at a defined NSR cut-off grade. Mineralization domain boundaries were considered as hard boundaries. The estimation strategy used three passes, with the search ellipsoid size and orientation based on the anisotropy determined by copper variography. The Mineral Resource estimates have been classified by their proximity to data locations and relative data reliability. Mineral Resources were tabulated for sulphide-only mineralization below the December 2014 topographic surface to account for previously mined material. The Mineral Resource estimates are reported in accordance with the May, 2014 CIM standards on Mineral Resources and Reserves, as required by NI 43-101.

14.1 DRILL HOLE DATABASE

The Aranzazu drill hole database is maintained as an acQuire® database by the database geologist (Gleidson Santos) in Brasilia, Brazil. It was built from the Zacoro Gemcom database which was based on Macocozac's paper database that was double keyed into an electronic database and then checked log by log to confirm data quality. Zacoro converted the drill hole collars from a local mine grid into UTM coordinates, using the transformation provided in Table 14-1. Aura Minerals' database maintains all coordinates in the UTM system. Aura Minerals has also verified the drill hole database against the original drill logs and assay certificates and this was done in March 2013 as part of the validation program.

Table 14-1 Drill hole Collar Conversion Factors from Local to UTM Coordinates

Local Grid			Local Conversion Add 10,000		UTM Factor		UTM Conversion Add UTM Factor	
	Easting	**Northing**	**Easting**	**Northing**	**Easting**	**Northing**	**UTM E.**	**UTM N.**
Local Grid 1	8865	10442			244990	2713570	253855	2724012
Local Grid 2	-415	669	9585	10669	244990	2713570	254575	2724239

Table reproduced from the 2007 Zacoro Technical Report.

The drill hole database for this Mineral Resource estimate includes drilling and assaying up to February 8, 2014, the effective date of the database. A complete review of the database was done prior to submitting the final database, which included modifying drill hole collars according to re-surveyed underground drive locations, omitting drill holes that could not be validated for location and including assay data that was previously missing.

The full data set used in the Mineral Resource estimate totals 219,586.2 m of drilling in 1,336 drill holes. From this drilling, a total of 87,970 samples of various lengths were collected and assayed for total copper, 76,874 samples were assayed for gold, 80,544 samples were assayed for silver, and 18,094 samples were assayed for arsenic. Figure 14.1 displays the location of the Aura Minerals drill holes as thick black lines, with reference to the historical mining areas.

The drill hole database also contains lithological descriptions that identify both lithology and alteration. Table 14-2 lists the important lithology codes with a simple description of the lithology.

Table 14-3 summarizes the number of sample assays obtained by each of the companies that has drilled the Property. There are more copper assays than gold or silver assays because historical drill hole samples were not always assayed for precious metals. The two most common sample lengths are 2.0 m and 1.5 m, with an average of 1.62 m. Table 14-4 presents summary statistics of all the assays included in the database by lithology. The exoskarn (XSk) and the endoskarn (ESk) have the highest average grades for copper, gold and silver and thus, form the basis of the mineralization interpretation. The lithology code Breccia in skarn (BRXX) has particularly high grades as well but very few samples have been taken to derive meaningful statistics at this stage. Minor localized occurrences of high grade mineralization occur within the intrusion and hornfels lithologies. Table 14-4 also presents summary statistics for other metals including molybdenum, lead, zinc, arsenic, bismuth, antimony, iron and soluble copper.

Figure 14.1 Aranzazu Mine Area and Drill hole Location

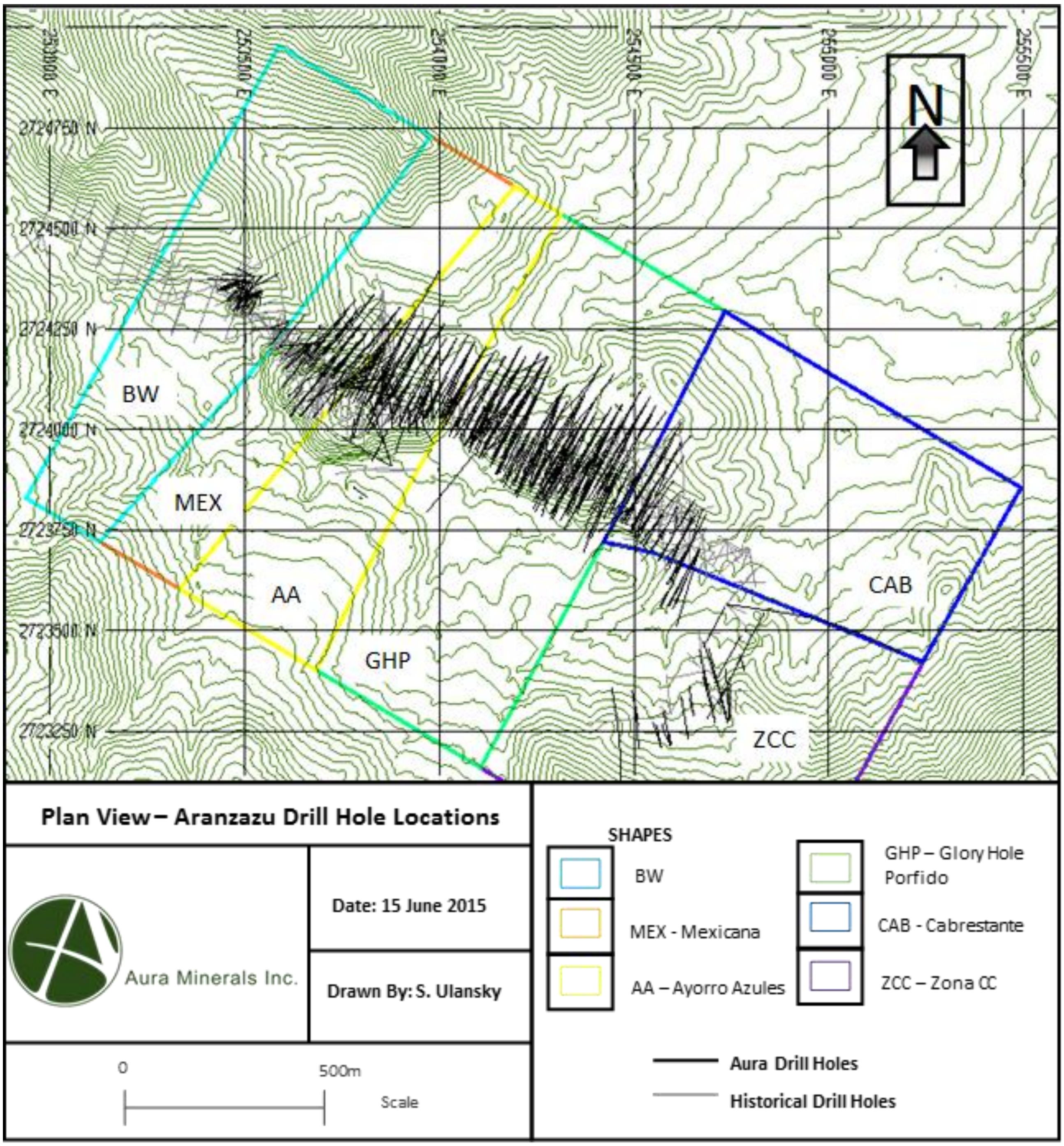


Table 14-2 Major Lithology Codes and Descriptions

Lithology Code	Integer Code (LITHD)	Description
Brxx	1	Breccia in skarn
Css	3	Clastic sedimentary sequences
Lms	4	Greyish or white limestone
Mbl	5	Fine-grained marble or recrystallized limestone
Hfl	6	Hornfels, fine-grained, brittle and concoidal fracture
XSk	7	Exoskarn, skarn developed in sedimentary rock protolith
ESk	8	Endoskarn, skarn developed in intrusive rocks
PBQA	9	Plagioclase, biotite, qtz +/- hornblende
PPBF	10	Porphyrytic texture - plagioclase, biotite, K-feldspar, quartz

Table 14-3 Summary of the Number of Sample Assays in Database Listed by Company

Company	No.of Holes	No. of Samples	Cu (%)	Au (gt)	Ag (gt)	Mo (ppm)	Zn (%)	Pb (%)	Fe (%)	As (ppm)	Bi (ppm)	Sb (ppm)	CuO (%)
Aura Minerals	609	63,657	63,639	63,638	63,638	63,639	63,639	63,638	12,517	15,568	15,566	15,566	8,646
Asarco Mexicana	100	1,517	1,516	32	106	21	34	-	-	-	-	-	19
Macocozac	473	12,940	12,686	3,087	7,076	232	207	150	35	150	150	150	190
Mazapil Copper	3	98	92	74	75	-	94	81	-	-	-	-	-
Zacoro	151	10,045	10,037	10,043	9,649	2,375	2,446	2,446	2,375	2,375	2,375	2,375	-
Totals	**1,336**	**88,257**	**87,970**	**76,874**	**80,544**	**66,267**	**66,420**	**66,315**	**14,927**	**18,093**	**18,091**	**18,091**	**8,855**

Table 14-4 Summary of the Statistics for all of the Drill hole Database Assays by Lithology

Lithology Code	Integer Code	No. Cu Data	Mean Cu (%)	Std. Dev. Cu (%)	No. Au Data	Mean Au (g/t)	Std. Dev. Au (g/t)	No. Ag Data	Mean Ag (g/t)	Std. Dev. Ag (g/t)
BRXX	N/A	38	1.90	2.09	38	0.536	0.465	38	29.50	44.37
INFL	2	598	0.32	0.50	585	0.208	0.407	596	4.19	7.17
CSS	3	26	0.25	0.24	26	2.005	5.745	26	5.06	4.38
LMS	4	2713	0.11	0.46	2585	0.079	0.275	2687	2.83	9.10
MBL	5	11660	0.10	0.43	11467	0.126	1.187	11505	2.75	9.45
HFL	6	9821	0.13	0.24	9806	0.112	0.261	9754	2.40	5.78
XSK	7	22377	0.63	1.27	14296	0.338	1.039	17243	9.60	22.37
ESK	8	19770	0.59	1.16	17996	0.430	1.166	18448	8.25	16.41
PBQA	9	10539	0.10	0.25	9823	0.054	0.197	9993	1.68	4.23
PPBF	10	10178	0.08	0.25	10040	0.050	0.219	10040	1.09	3.78
Lithology Code	**Integer Code**	**No. Mo Data**	**Mean Mo (ppm)**	**Std. Dev. Mo (ppm)**	**No. Pb Data**	**Mean Pb (%)**	**Std. Dev Pb (%)**	**No. Zn Data**	**Mean Zn (%)**	**Std. Dev. Zn (%)**
BRXX	N/A	38	10.39	34.30	38	0.013	0.018	38	0.250	0.314
INFL	2	547	56.11	185.20	547	0.009	0.026	547	0.128	0.172
CSS	3	26	60.38	92.06	26	0.006	0.003	26	0.160	0.171
LMS	4	2350	46.61	102.55	2353	0.012	0.078	2354	0.096	0.287

MBL	5	10447	10.76	105.77	10492	0.010	0.120	10495	0.112	0.451
HFL	6	8709	92.71	312.97	8715	0.006	0.064	8715	0.102	0.273
XSK	7	10534	216.90	2152.95	10535	0.005	0.042	10550	0.100	0.631
ESK	8	15738	252.28	2055.81	15718	0.008	0.076	15783	0.185	0.964
PBQA	9	7941	38.99	134.56	7953	0.002	0.008	7966	0.021	0.136
PPBF	10	9855	48.15	164.69	9856	0.002	0.017	9862	0.024	0.334
Lithology Code	**Integer Code**	**No. As Data**	**Mean As (ppm)**	**Std. Dev. As (ppm)**	**No. Bi Data**	**Mean Bi (ppm)**	**Std. Dev. Bi (ppm)**	**No. Sb Data**	**Mean Sb (ppm)**	**Std. Dev. Sb (ppm)**
BRXX	N/A	38	488.4	621.5	38	487.1	839.8	38	33.0	32.8
INFL	2	122	498.3	508.5	122	101.5	180.3	122	30.3	31.0
CSS	3	13	370.9	416.5	13	115.7	57.2	13	15.0	10.1
LMS	4	584	478.7	1651.8	584	49.1	93.0	584	24.4	109.1
MBL	5	2263	257.9	843.1	2263	83.7	287.9	2263	23.7	73.0
HFL	6	1100	481.4	1150.5	1099	114.2	334.7	1099	28.8	72.0
XSK	7	4609	532.7	947.3	4609	173.2	383.0	4609	33.4	87.8
ESK	8	6357	839.2	1361.0	6357	230.9	463.3	6357	52.9	104.7
PBQA	9	1939	160.9	554.4	1938	32.2	122.3	1938	14.3	38.3
PPBF	10	1037	185.6	575.0	1037	43.7	140.9	1037	14.3	30.1
Lithology Code	**Integer Code**	**No. Ox. Cu Data**	**Mean Ox. Cu (%)**	**Std. Dev. Ox. Cu (%)**	**No. Fe Data**	**Mean Fe (%)**	**Std. Dev. Fe (%)**			
BRXX	N/A	38	0.086	0.119	38	12.3	10.2			
INFL	2	148	0.220	0.402	98	7.5	3.0			
CSS	3	18	0.122	0.158	0	0.0	n/a			
LMS	4	628	0.018	0.055	234	3.3	3.5			
MBL	5	2772	0.036	0.212	2011	2.3	4.2			
HFL	6	239	0.072	0.077	946	5.0	3.7			
XSK	7	1627	0.130	0.283	3182	11.9	6.1			
ESK	8	1206	0.104	0.240	5829	12.3	7.3			
PBQA	9	1076	0.010	0.027	1787	4.1	3.0			
PPBF	10	1091	0.013	0.029	783	4.7	3.2			

14.2 GEOLOGICAL MODEL AND DOMAIN INTERPRETATIONS

The Aranzazu geological model was constructed using drill hole lithological and structural data; core photos were used to assist with the interpretation. The geological interpretation was created on sections and plans, which were then used to define three-dimensional wireframes using MineSight® 9.50 mining software. A fully completed lithology model was done for all rock types. The lithologies have been simplified into four groups, limestone-marble (code 10), hornfels (20), skarn (30) and intrusive (40) and have been coded into a LITHD field in the block model. Figure 14.2 displays the block model codes for the level plan at 2,000 m elevation.

The interpretation of the Aranzazu mineralization was done using drill holes colour-coded by NSR and lithology. The mineralization wireframes were grouped to form mineralization shapes that were named after the historical mining areas and coded with a domain code ranging from 100 to 702, in 9 separate domains. Figure 14.3 displays the locations of the various mineralized domain wireframes and Table 14-5 lists the domain codes, wireframe names and the number of blocks assigned to the various domain codes in the block model. Figure 14.4 is a clipped 3D view of domain 100-BW showing the shape of the mineralization domain in relation to the level geology maps and underground workings.

Domain mineralized zones were defined using the following assumptions:

- A nominal $45/t shell applying the formula to Cu, Au and Ag: NSR ($/t) = (Cu% x $56.32) + (Au g/t x $22.15) + (Ag g/t x $0.35) - $17.71
- Geological continuity defined by spatial location, structural orientations and lithological controls.
- Mineralization continuity and grade distribution.

The mineralization solids were initially created on 126 sections displaying drill holes colour-coded by NSR and lithology. The cross-sections were cut at 6.25 m, 12.5 m and 25 m intervals depending on the availability of drill hole data in each zone. Three different grid orientations were used to generate the mineralized wireframe solids: 000° in Zones 300 and 702, 060° in Zone 600 and, 030° in the remaining zones (Figure 14.5). In addition to the cross-sections, level plans on 10m intervals were used to validate the geological continuity.

There are 16 sections at 6.25m and 12.5m intervals interpreted for BW, 36 sections at 12.5m intervals in Mexicana, 11 sections at 12.5 m intervals in AA, 47 sections at 25m intervals in GHP and ZCC, 10 sections at 25m intervals in CAB and 6 sections at 25m in ZCC.

The mineralization domains were then digitized on both cross-section and level plans ensuring that the mineralization strings were snapped to the drill holes. The mineralization interpretation was extruded between half and one full section spacing for closure of the solid, which was then used to code the block model.

Figure 14.2 Location of the Lithological Contacts on Level Plan 2,000 m Elevation

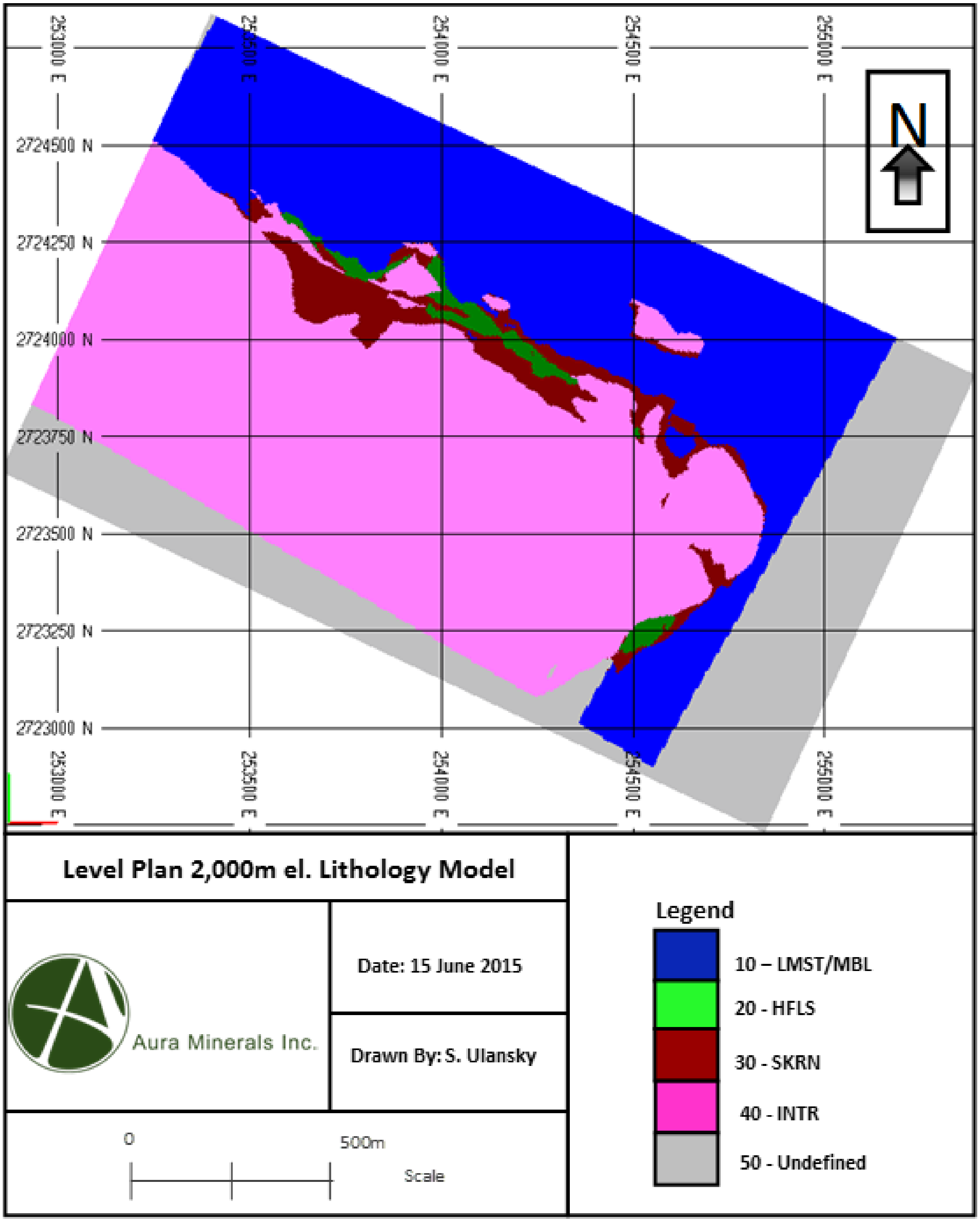


Figure 14.3 Locations of the Various Mineralized Zone Wireframes

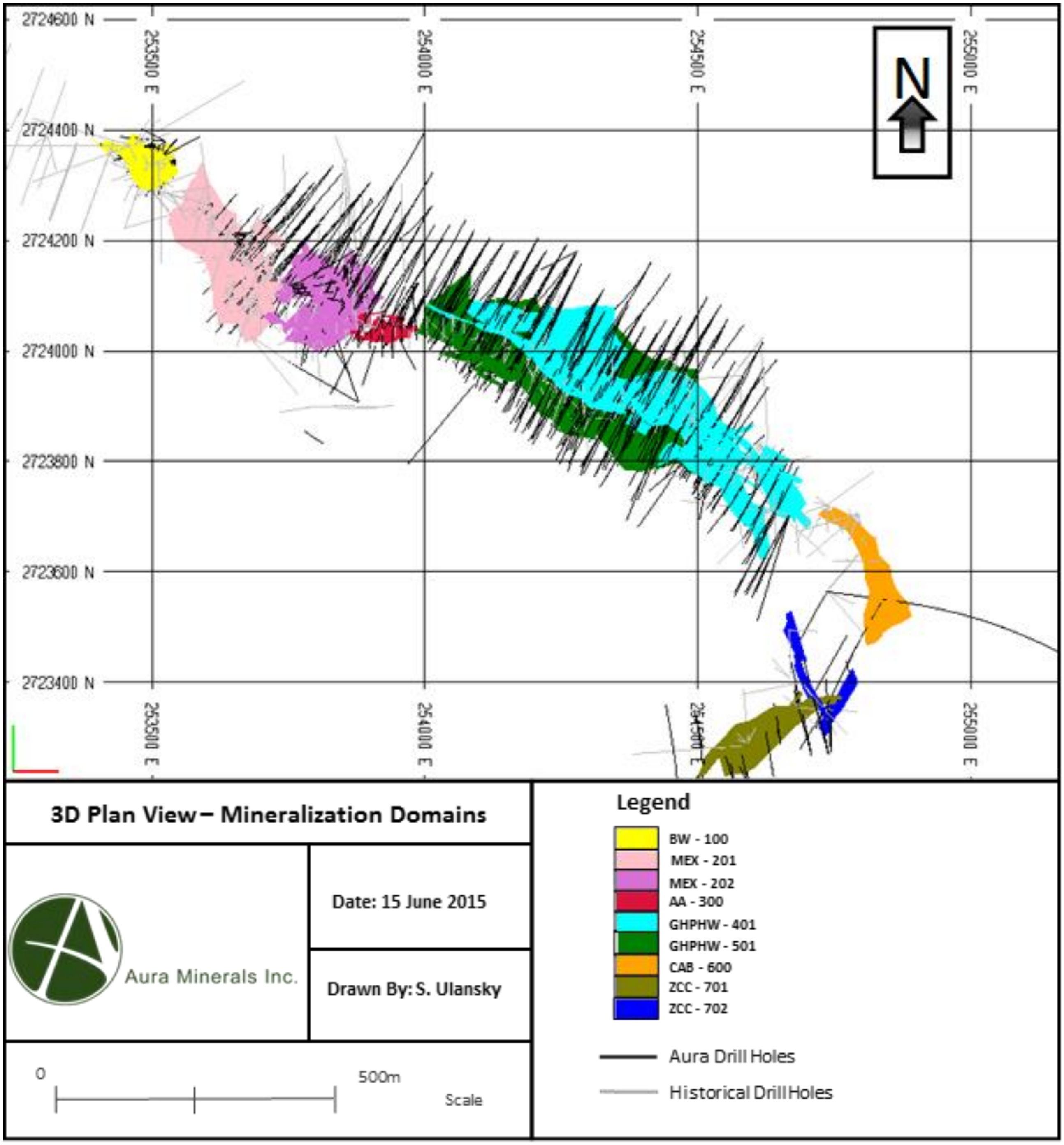


Figure 14.4 3D Plan View of Domain 100-BW Showing the Geology Level Maps and Underground Workings

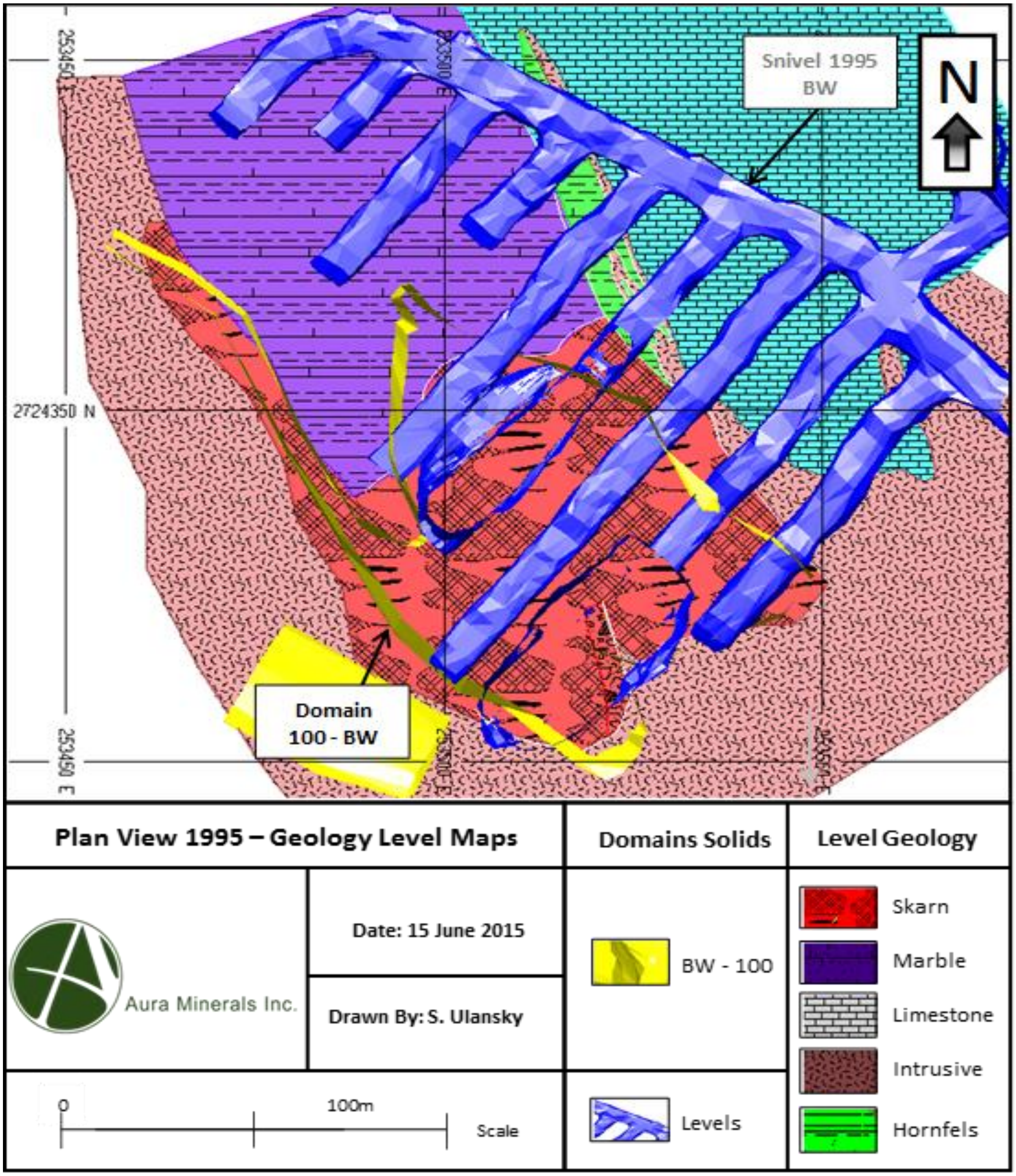


Figure 14.5 Aranzazu Mine Displaying the Aranzazu Grids

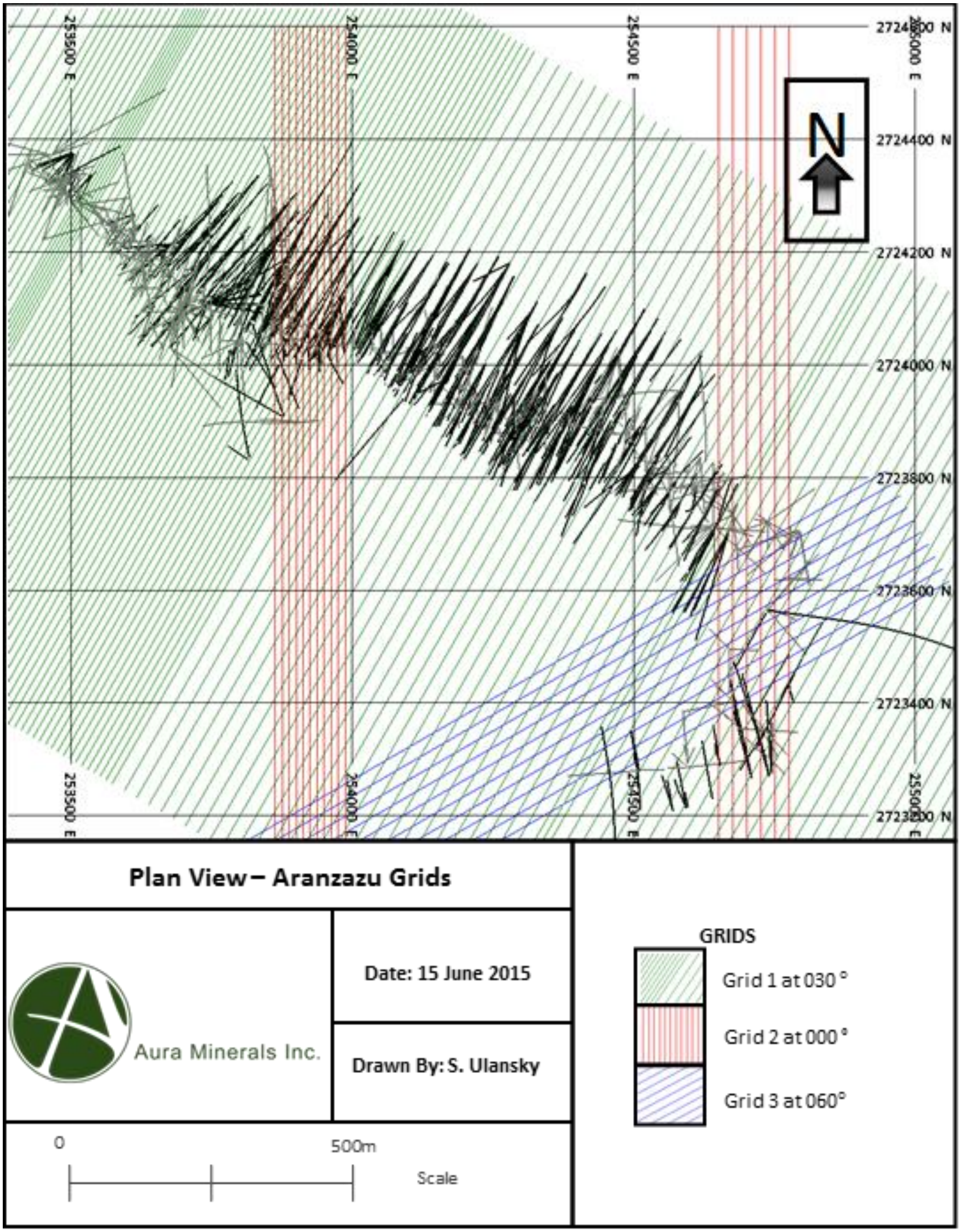


Table 14-5 Summary of Domain Codes and Wireframes

Domain	Wireframe Names	Colour	Volume of Wireframe (m^3)	No. of Blocks	Block % of Total
100 - BW	BW	yellow	580,582	7,082	5%
201 - MXN	Mexicana	pink	1,532,608	20,484	15%
202 - MXS	Mexicana	purple	1,324,081	16,142	12%
300 - AA	Arroyo Azules (AA)	red	146,546	2,548	2%
401 - GHPHW	Glory-hole Porfido Hanging Wall	cyan	1,961,437	32,830	24%
501 - GHPF	Glory-hole Porfido Footwall	forest green	3,354,974	43,154	32%
600 - CAB	Cabrestante	orange	345,511	6,444	5%
701 - CCa	Zona CC	olive	238,425	4,809	4%
702 - CCb		blue	93,661	2,459	2%
Total			**9,577,823**	**135,952**	**100%**

14.2.1 Oxide Surface

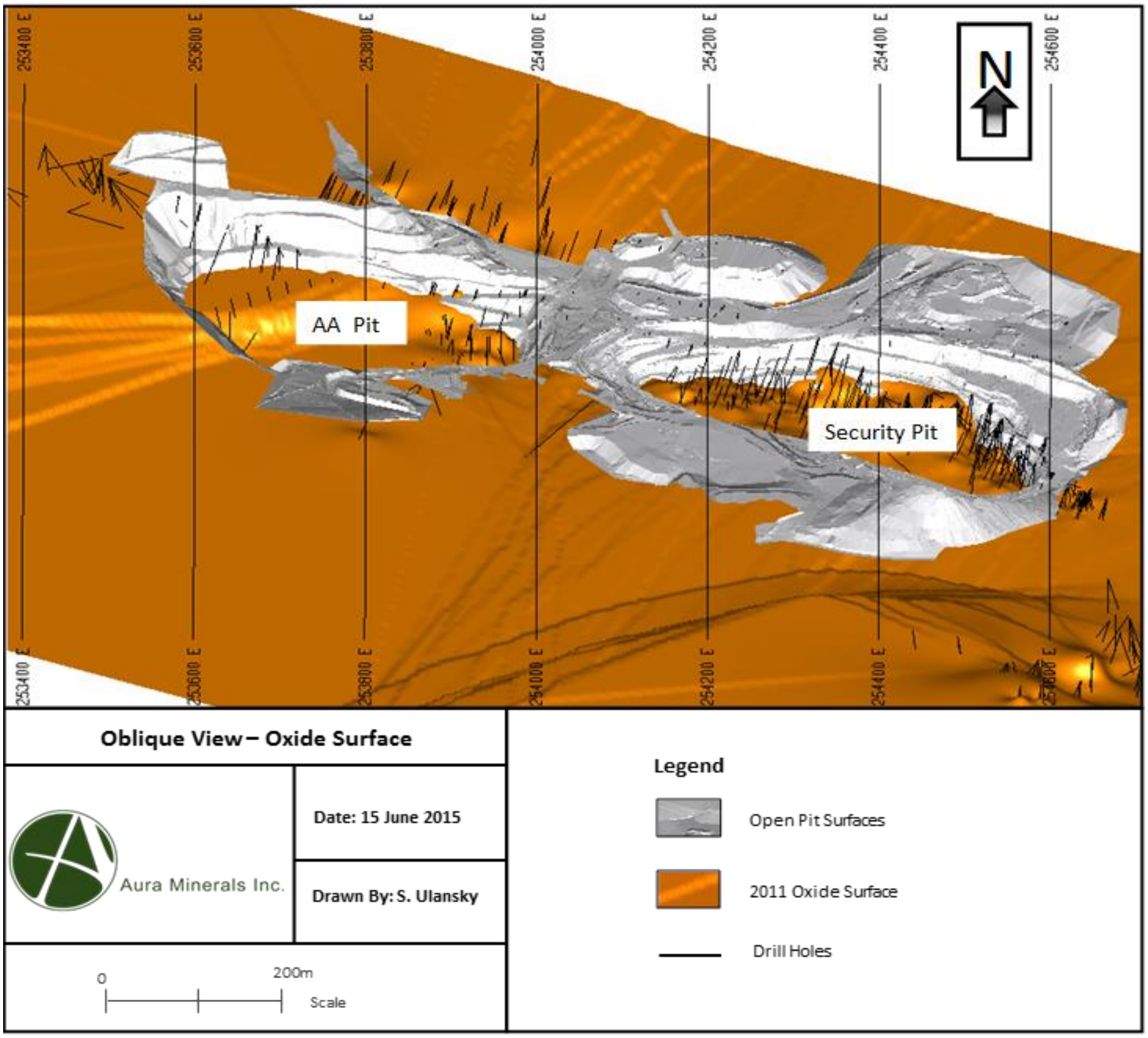
An oxide surface was modelled for the Mexicana, Arroyos Azules, Glory Hole-Porfido and Cabrestante areas, as the copper-gold-silver mineralization is situated close to the surface and this material could potentially be mined by open pit. This surface was modelled to separate material that is considered oxidized (having oxide copper minerals) from the material that is considered sulphide (limited or no oxide copper minerals). The oxidized material is thought to have poor metallurgical recovery and would not be processed by the mill.

The oxide surface was modelled based on geological observations of the abundance of hematite, goethite, limonite, malachite and azurite and the limited assays of acid-soluble copper and the acid soluble-total copper ratio. As there are only a limited number of acid soluble copper analyses, approximately 5,407 results, the geological observations were the main source of information to define the oxide-sulphide boundary. This boundary was determined by identifying the change between as-logged oxide minerals and as-logged sulphide minerals. The location of the oxide/sulphide boundary was then compared with the acid soluble copper-total copper ratio. The drill hole data were coded as either oxide or sulphide material, with the oxide material given a code of “1” and the sulphide mineralization given a code of “2” and blocks above topography given a code of “999”. This information is contained in the block model variable “OXIDE”.

A 3-D gridded surface was interpolated using inverse distance weighting to the third power, with a large search radius (1,200m) and minimum and maximum of eight closest data points. The large search radius was chosen to produce a smooth surface over the entire model area, while the limited number of data points and the third power for inverse distance weighting maintained local features. Figure 14.6

is an oblique view that shows the coded drill holes and the oxide surface, with the completed AA Pit and in progress Security Pit for reference.

Figure 14.6 Oxide Surface Interpolated from As-Logged Geological Data



14.3 EXPLORATORY DATA ANALYSIS

14.3.1 Reassigning Long Un-Sampled Intervals in Ore Zones to the Waste Zone

Exploratory data analysis was completed using Snowden's Supervisor® v8.2 software package. Statistical and graphical summaries of the copper, gold and silver assays were produced to understand the distribution of the metals in the Aranzazu deposit. Both the original sample lengths and composites were used in this analysis.

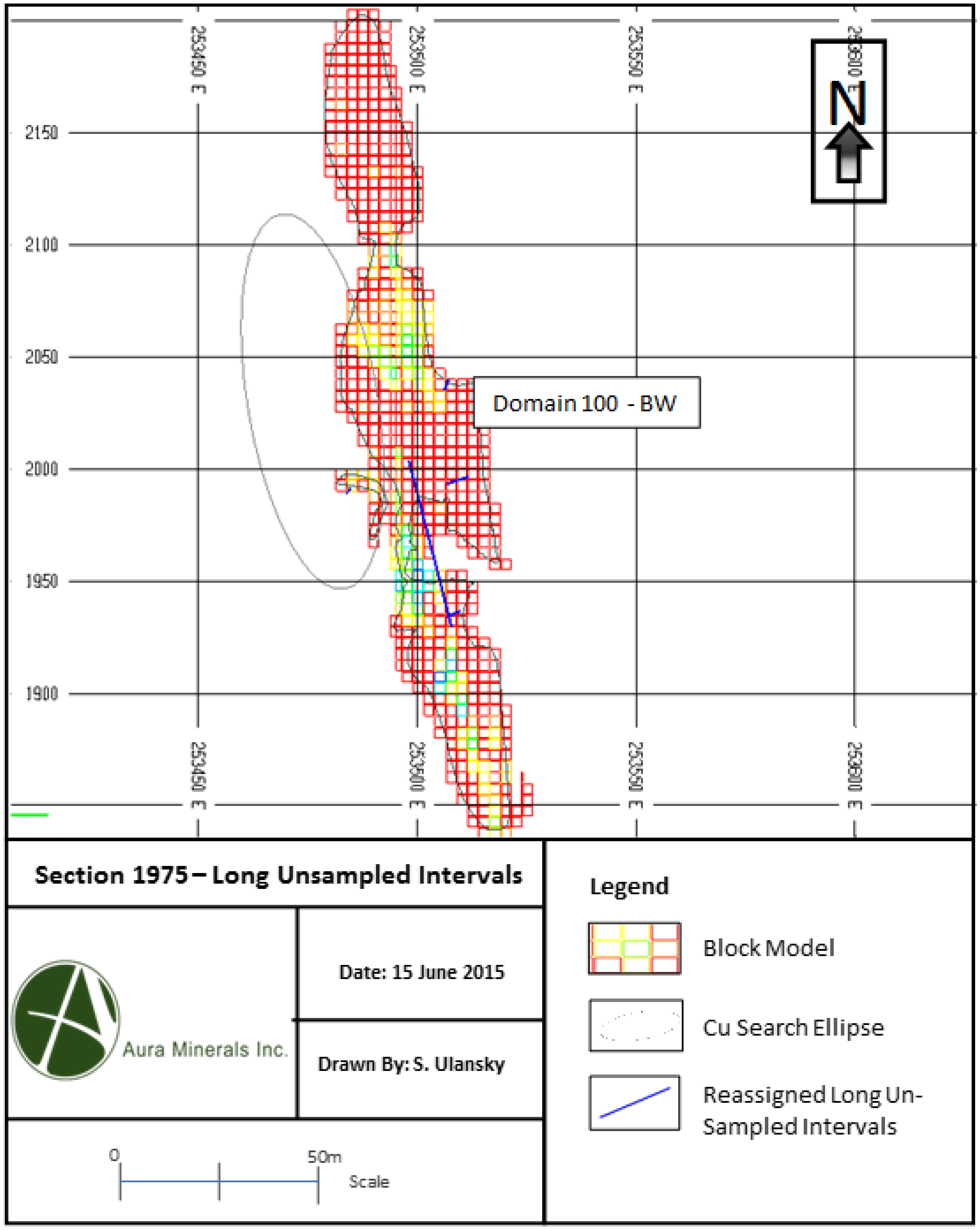
The raw data was coded as either one of nine mineralization domains, or as waste, and those long un-sampled intervals that were included into the mineralization solids due to transecting the mineralization-waste boundary, were removed manually, and reassigned into the waste zone. This step ensured that ore zone blocks in the block model were only estimated using composites coded within the ore zone solids. Figure 14.7 show an example of the long un-sampled intervals that were reassigned from domain 100 to the waste zone. In doing so, the block model used composites closer to the block estimate thereby increasing the reliability of the estimate.

Table 14-6 displays a summary of raw copper, gold, silver and NSR grades and CV for each mineralization domain.

Table 14-6 Raw Data Summary Statistics for Copper, Gold, Silver and NSR by Domain

Domain	Cu (%)			Au (g/t)			Ag (g/t)			NSR ($/t)		
	# Samples	Mean	CV	# Samples	Mean	CV	# Samples	Mean	CV	# Samples	Mean	CV
100	1980	1.62	1.36	1375	0.395	1.625	1877	25.20	1.37	1718	102.8	1.4
201	3513	0.86	1.41	1997	0.267	1.633	2205	11.05	1.49	2461	55.3	1.5
202	3166	1.38	1.43	2748	0.623	1.768	2749	12.64	1.60	2278	109.9	1.3
300	298	1.61	0.90	265	1.206	0.889	265	16.99	0.75	271	113.4	0.8
401	3123	0.92	1.29	2794	1.354	2.523	2861	17.42	1.39	2584	82.4	1.4
501	2901	1.68	1.28	2231	1.457	1.286	2420	21.37	1.31	2612	120.8	1.3
600	332	1.80	1.32	161	1.593	1.606	276	47.78	2.09	289	133.1	1.5
701	152	0.79	1.32	152	0.446	1.507	152	18.50	1.32	95	76.4	1.0
702	180	0.76	2.09	180	0.777	3.150	180	16.08	2.85	86	116.1	1.3

Figure 14.7 Long Unsampled Intervals at Margin of Domain 100 Boundary Reassigned to the Waste Zone



14.3.2 Adjustments of Un-Sampled Intervals

Some of the historical drill holes have been selectively sampled within the ore zones, likely based on visual sulphide mineralization. This leaves gaps in the assay data for these drill holes. To ensure that high-grade copper assays are not projected into areas that have not been sampled, these un-sampled intervals were assigned default copper grades based on the median copper grade of the lithology within each mineralized domain using raw un-composited assay data. Missing gold and silver assays within each mineralized domain were also adjusted according to the median grade of the lithology unit. This adjustment considered all un-sampled intervals that have a defined lithology code and mineralized domain code.

Figure 14.8 is an example of the un-sampled intervals in Mexicana that have been assigned default grades based on median lithology codes. Mexicana is inherently variable in grade across short intervals, and to honour this variability in the block model, the decision to assign grades was deemed warranted.

A total of 465 intervals were assigned default copper grades, 4,208 intervals were assigned default gold grades and 3,126 intervals were assigned default silver grades. Table 14-7 presents a summary, by major lithology code, of the number of intervals assigned default grades for copper, gold and silver.

Table 14-8 displays summary statistics of assays for zones 100 to 703, before and after the adjustment of un-sampled intervals with the default values. For all metals, the average grade has decreased as a result of the adjustment. The average copper grade decreases from 1.258 to 1.231%, a decrease of approximately 2%. The average gold grade decreases significantly, from 0.890 to 0.705 g/t, a decrease of approximately 21%. The average grade of silver also decreases from 17.795 to 15.757 g/t, a decrease of approximately 11%.

The decreases in the average grade of gold and silver confirm that block model estimates for areas with drill holes containing un-sampled intervals will not be overly influenced by high- grade intercepts from the surrounding drill holes. For those areas in the block model where the drill holes have been sampled, these adjusted intervals will only have a small influence on the block estimates, because of the limit on the number of samples taken from a single drill hole.

Figure 14.8 Assigned Intervals in Mexicana in Relation to the Block Model

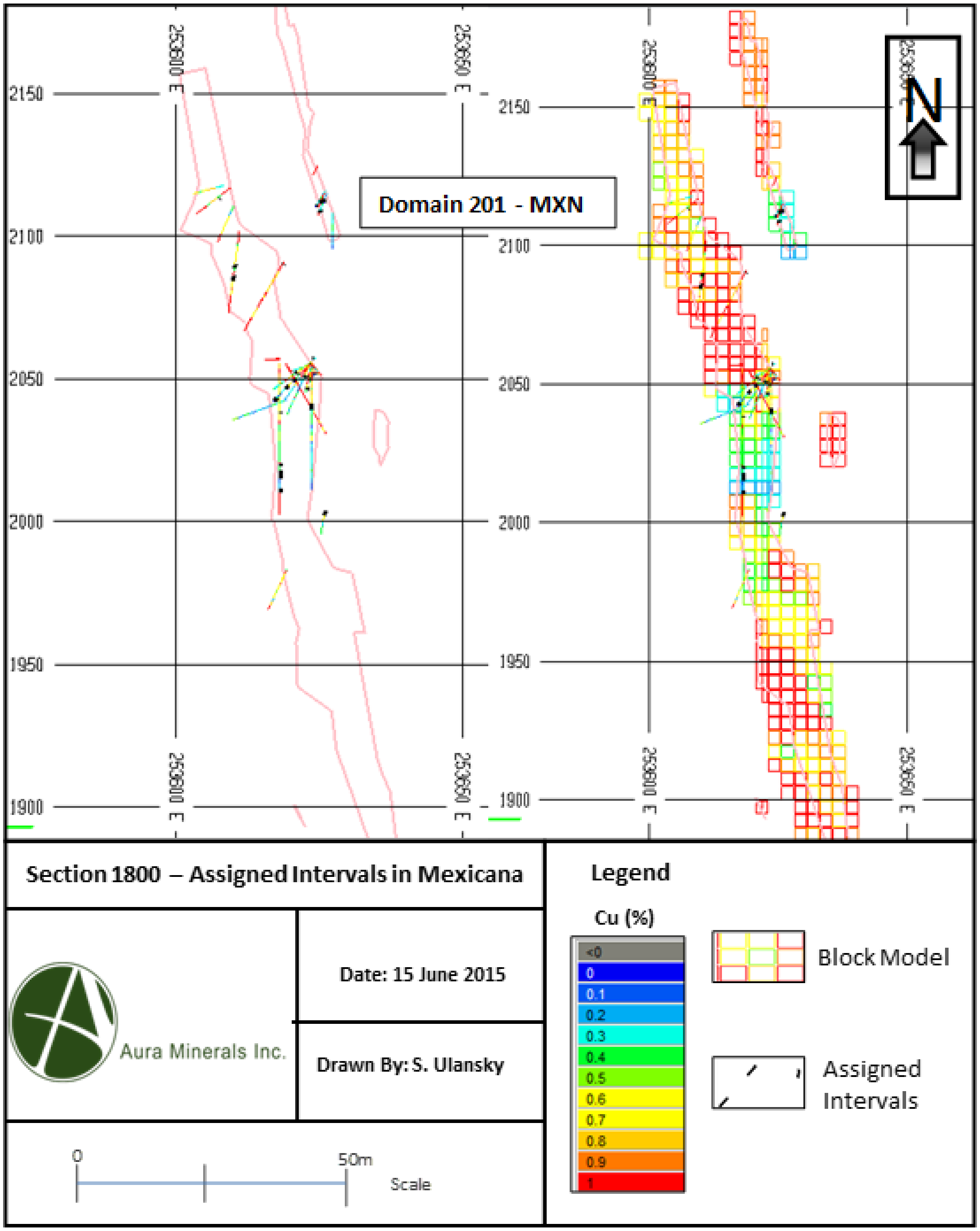


Table 14-7 Default Copper, Gold and Silver Grades Assigned to Un-Sampled Intervals

Lithology Type	Total Intervals	Type	Default Copper Grade (%)	No. of Adjusted Copper Intervals	Default Gold Grade (g/t)	No. of Adjusted Gold Intervals	Default Silver Grade (g/t)	No. of Adjusted Silver Intervals
CSS	5	median	0.410	0	0.429	0	8.4	0
BRXX	33	median	2.010	0	0.539	0	23.0	0
LMS	124	median	0.232	4	0.150	23	4.9	9
MBL	868	median	0.129	34	0.105	70	3.1	62
HFL	281	median	0.214	7	0.135	12	3.2	13
XSK	7,724	median	0.400	224	0.171	3,090	8.0	2,292
ESK	6,237	median	0.426	91	0.260	840	6.6	583
PBQA	420	median	0.188	40	0.075	91	2.5	85
PPBF	273	median	0.203	14	0.080	30	2.3	30
NR, INFL, OS, GAP	133	Default	0.010	51	0.010	52	0.01	52
Total No. of Adjusted Intervals				**465**		**4,208**		**3,126**

Note: CSS and BRXX had no unsampled intervals to fill and therefore no adjustments were made to these lithology types.
Note: a default value of 0.01 was applied to INFL, OS, NR and GAP.

Table 14-8
Summary Statistics for Assays for Domains 100 to 703 Before and After Default Adjustment

Metals	Before Adjustment				After Adjustment			
	No. of Data	Mean	Std Dev	CV	No. of Data	Mean	Std Dev	CV
Copper (%)	15,670	1.258	1.692	1.354	16,136	1.231	1.674	1.369
Gold (g/t)	11,928	0.890	1.655	1.809	16,136	0.705	1.396	1.828
Silver (g/t)	13,010	17.795	25.855	1.452	16,136	15.757	23.166	1.444

Note: Top cuts have not been applied yet.

14.3.3 Compositing

The drill hole samples were composited to 2.0 m regular lengths that honoured the geological mineralization boundaries. The composites were then coded with domain codes and the dominant lithology code was assigned to the composite. Grade capping was done after compositing because the original sample lengths are too variable in the historical drill holes.

The un-assigned composited assay data were used for analysis so that choice of the defaults would not influence the summary statistics. Figures 14.9 to 14.17 display histograms and scatter plots of the unassigned composited assays for copper, gold and silver in domains 100, 201, 202, 300, 401, 501, 600, 701, 702. Domain 703 had too few data points for reliable statistical results. The histograms show that the distributions for copper, gold and silver are positively skewed and have a lognormal shape. This shape is common for this type of mineral deposit.

There is minimal difference in the means of the composite data versus the raw assay data; however, the CV is greatly reduced over all the domains. Individual domains have highly variable CV's resulting from their data and grade distributions.

Side-by-side boxplots for copper, gold and silver 2.0 m un-assigned composite grades are shown in Figures 14.18 to 14.23. Boxplots provide a visual presentation of the distribution. The lower and upper quartiles are shown by the box, the median as the dark blue line in the box and the mean as the dark red line. The minimum and maximum grades are shown by the lines extending from the box.

The side-by-side boxplots show that there are differences in the copper, gold and silver distributions between the domains. This supports the decision to treat the domains separately for outlier analysis, variography and grade estimation.

Figure 14.9 Histograms and Scatter Plots of the 2.0 m Composites in Domain 100

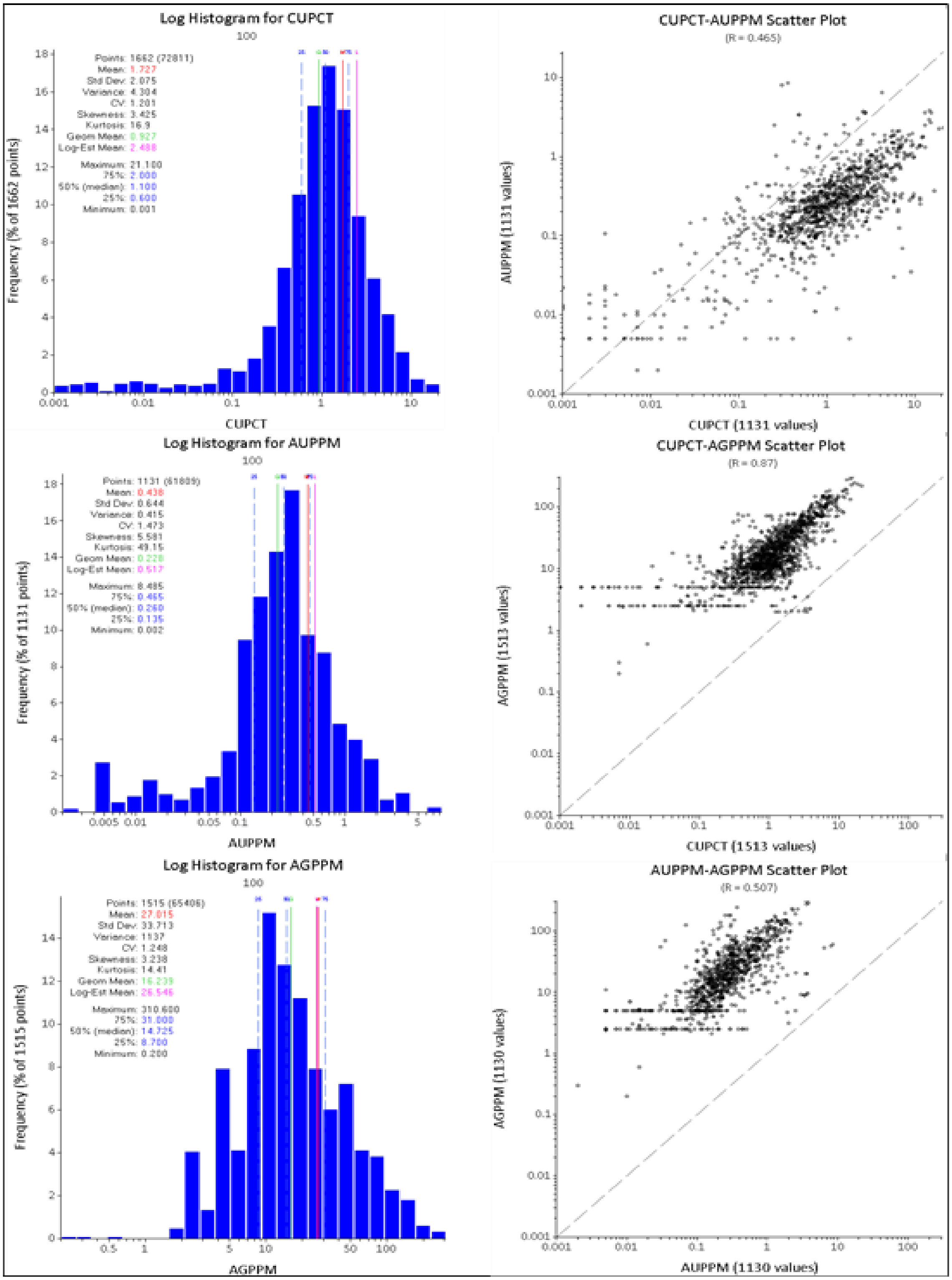


Figure 14.10 Histograms and Scatter Plots of the 2.0 m Composites in Domain 201

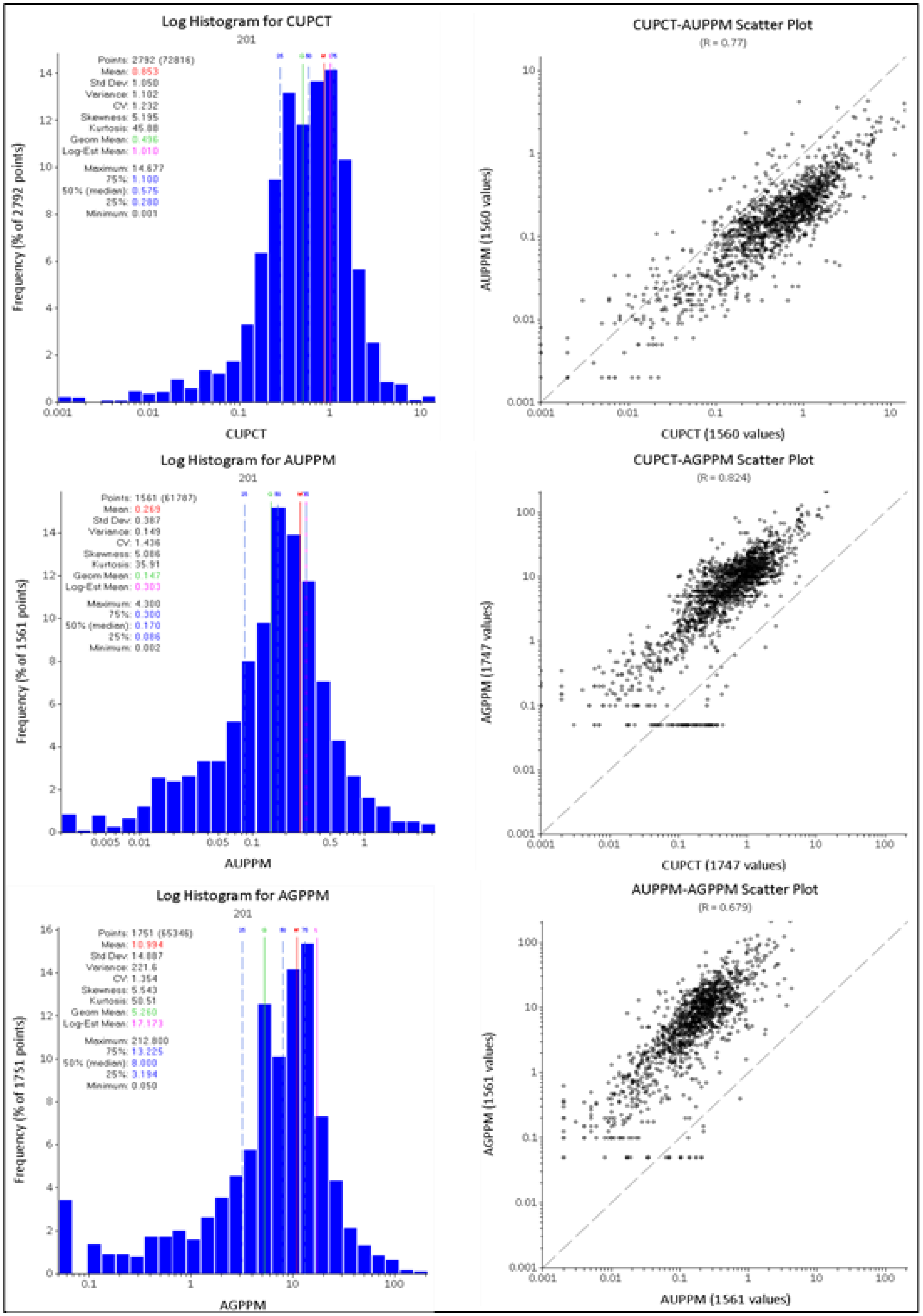


Figure 14.11 Histograms and Scatter Plots of the 2.0 m Composites in Domain 202

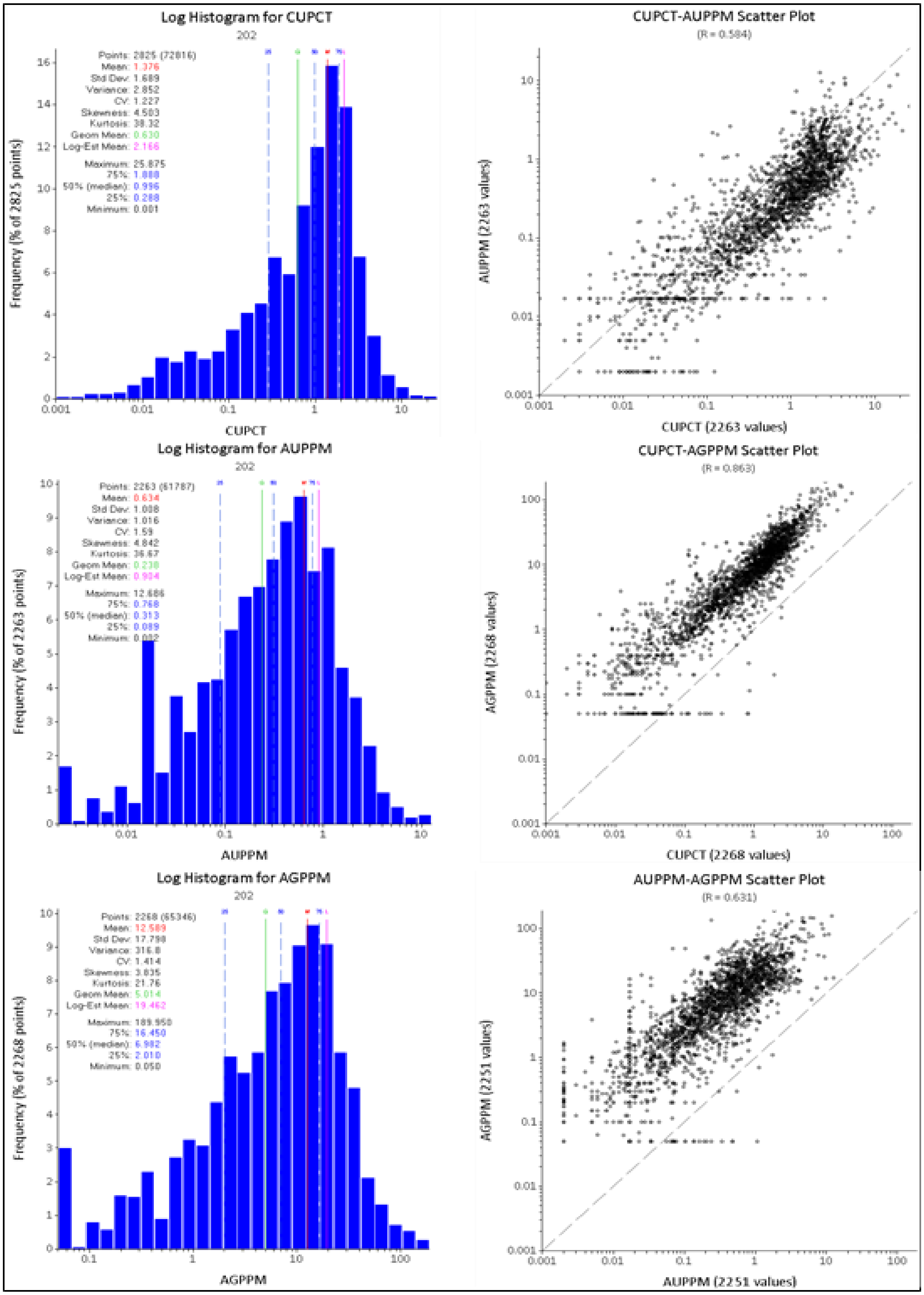


Figure 14.12 Histograms and Scatter Plots of the 2.0 m Composites in Domain 300

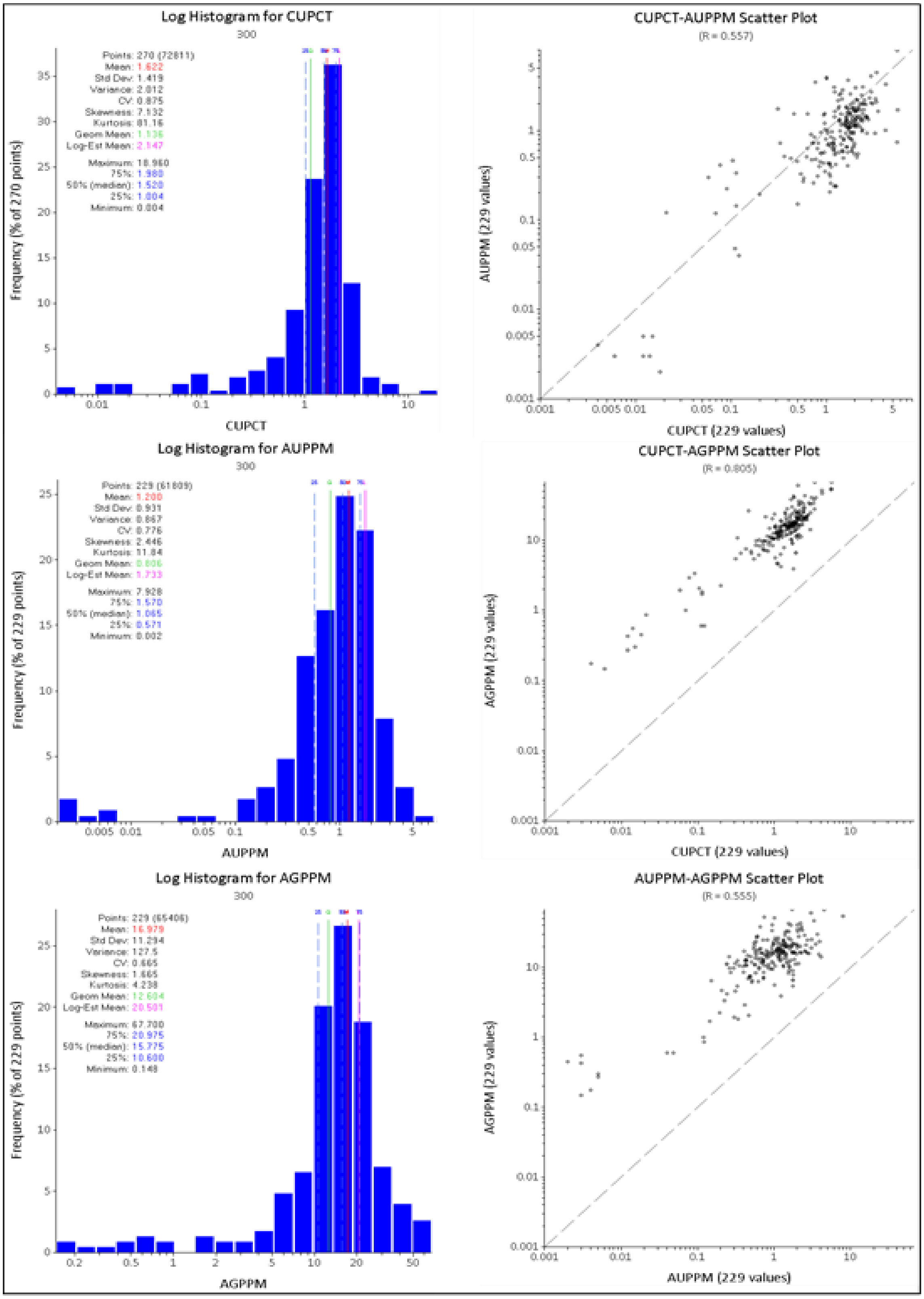


Figure 14.13 Histograms and Scatter Plots of the 2.0 m Composites in Domain 401

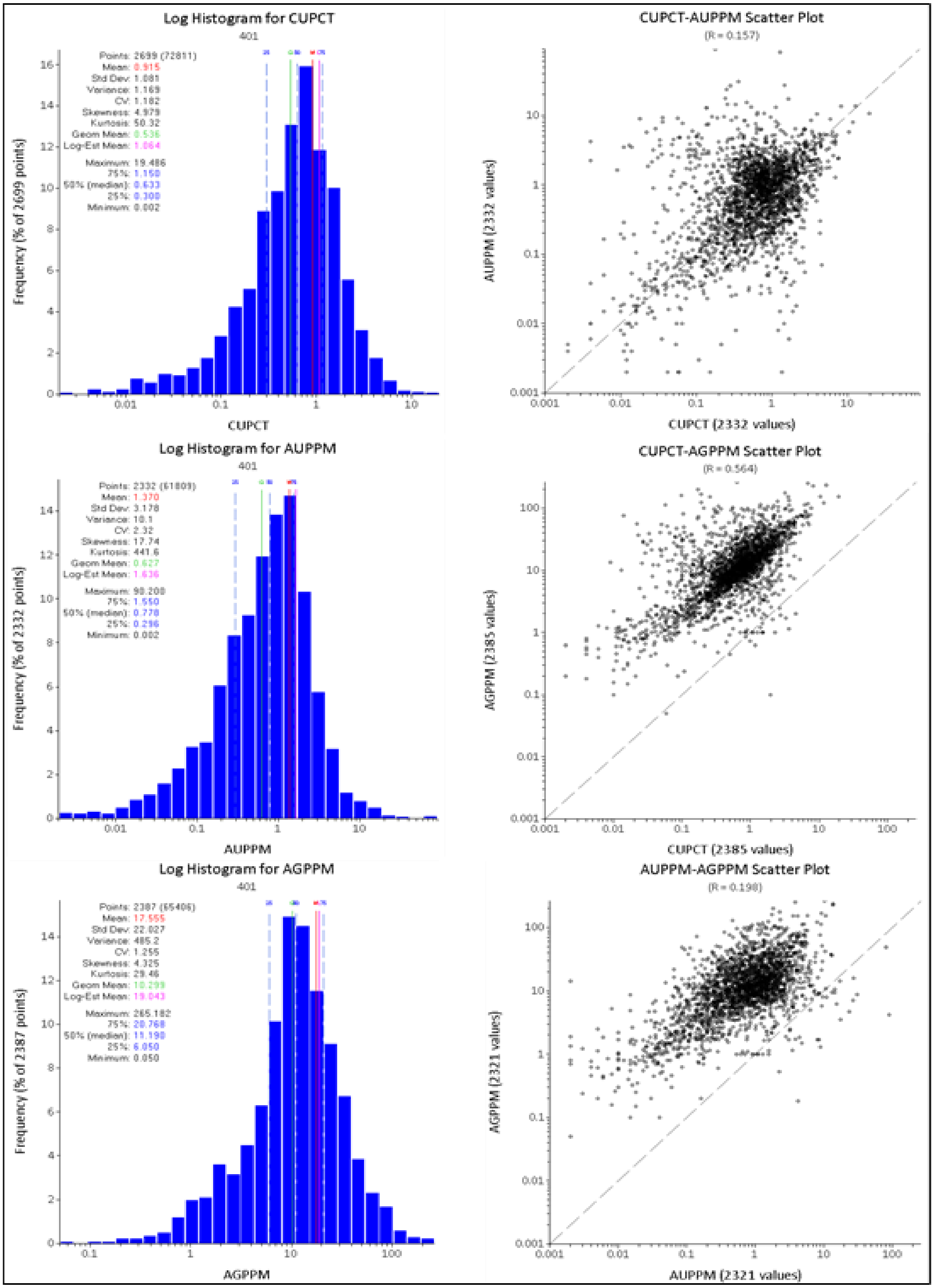


Figure 14.14 Histograms and Scatter Plots of the 2.0 m Composites in Domain 501

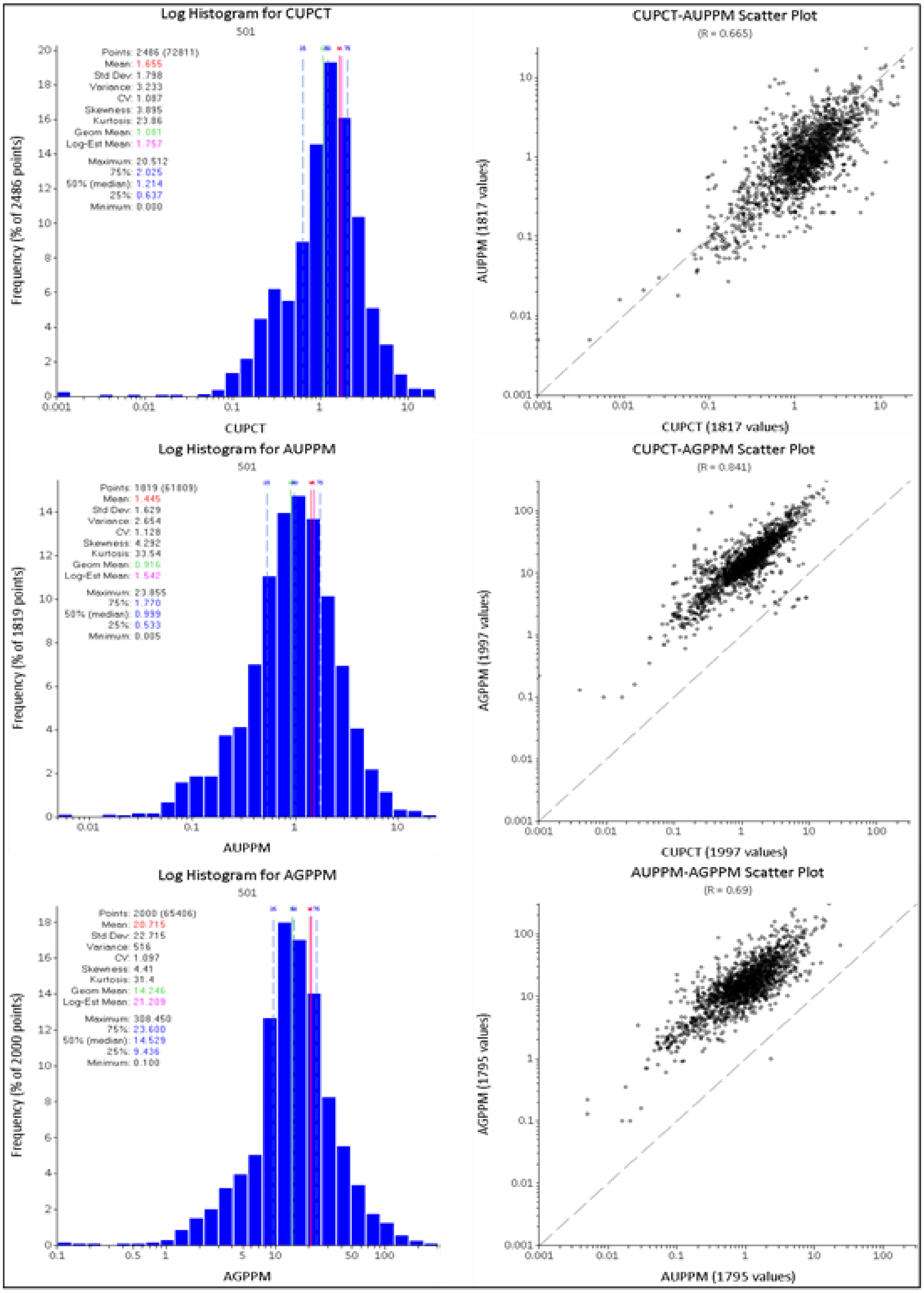


Figure 14.15 Histograms and Scatter Plots of the 2.0 m Composites in Domain 600

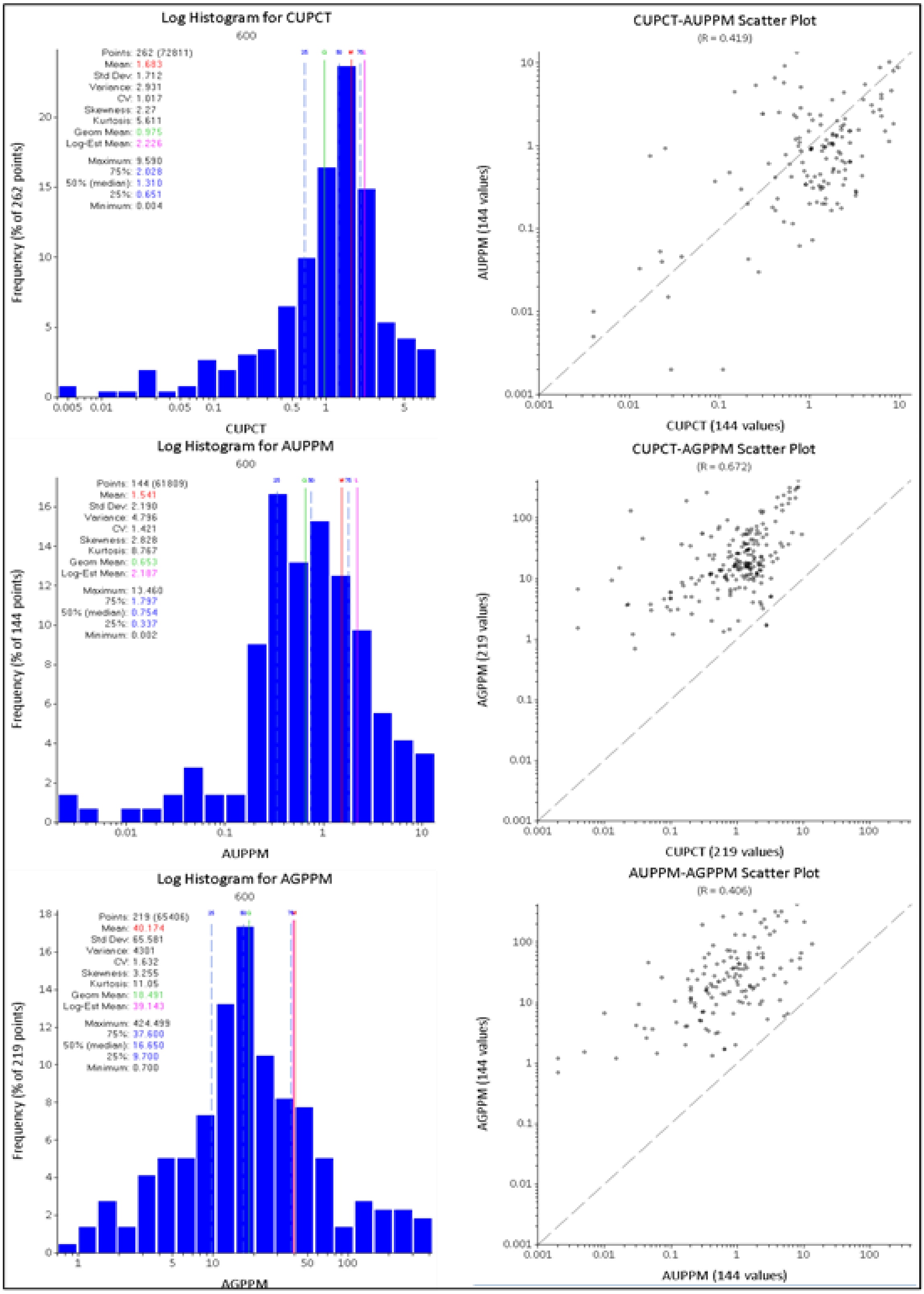


Figure 14.16 Histograms and Scatter Plots of the 2.0 m Composites in Domain 701

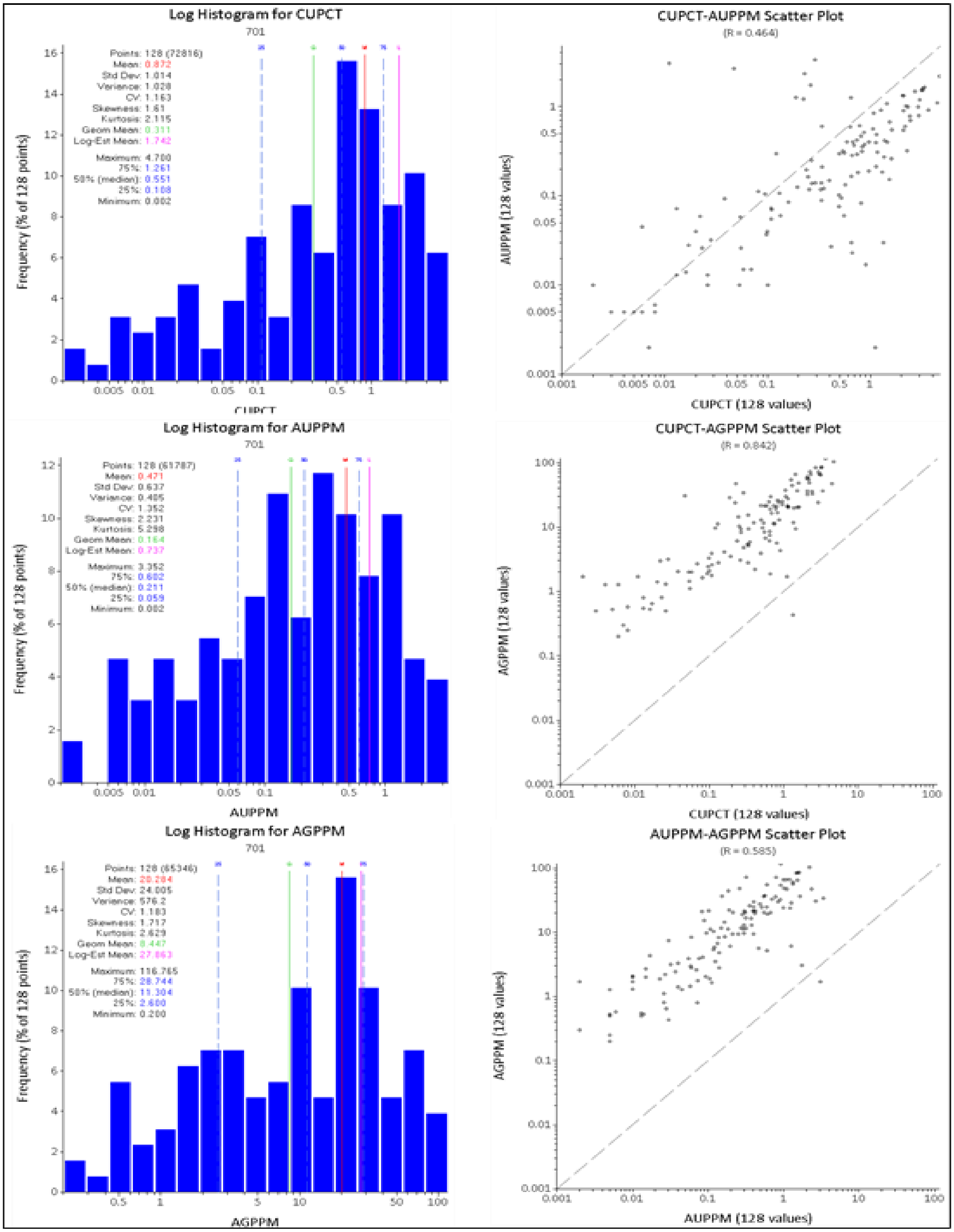


Figure 14.17 Histograms and Scatter Plots of the 2.0 m Composites in Domain 702

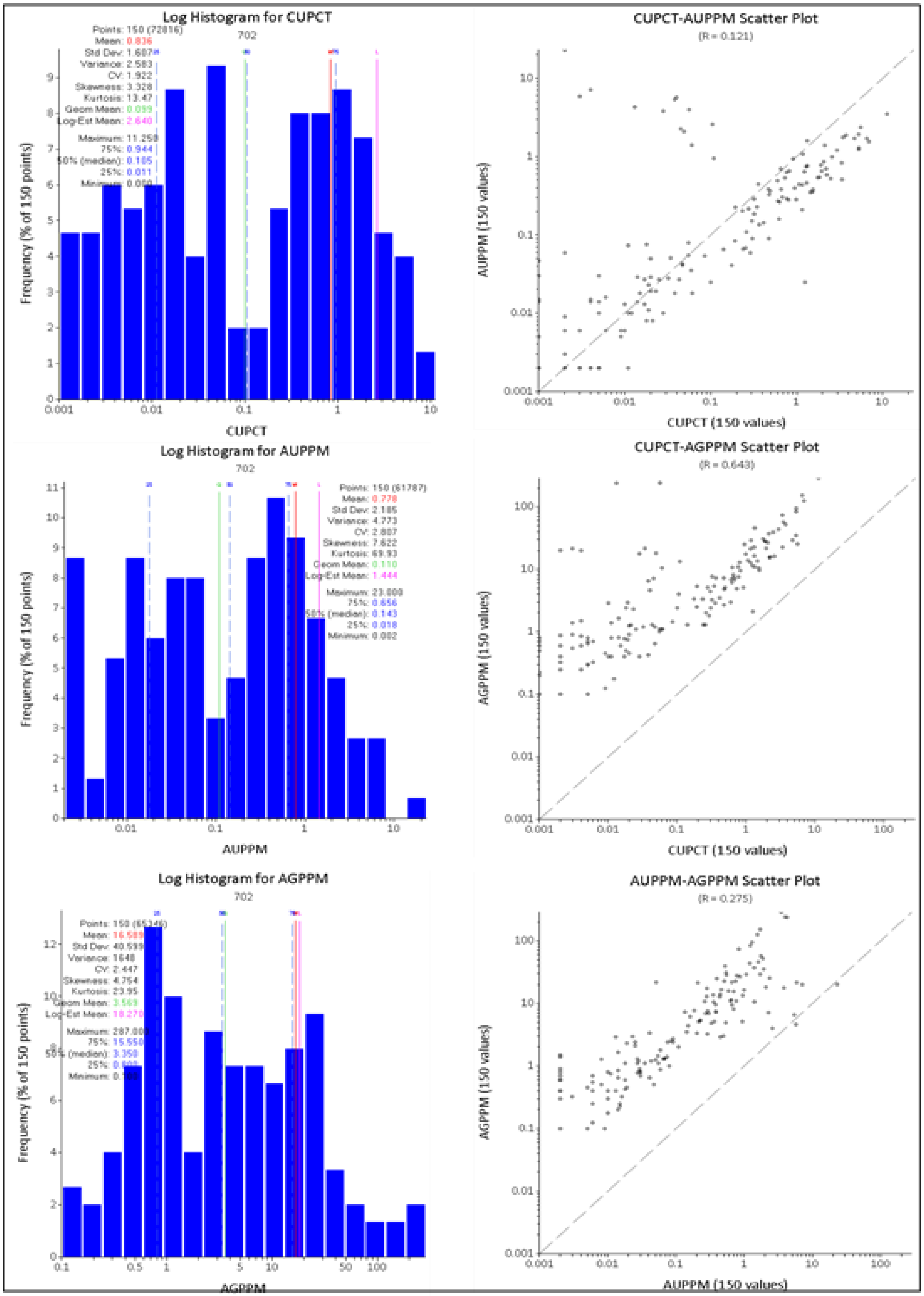


Figure 14.18 Side-by-Side Boxplots of 2.0 m Copper Composites (Domains 100, 300, 401, 501, 600)

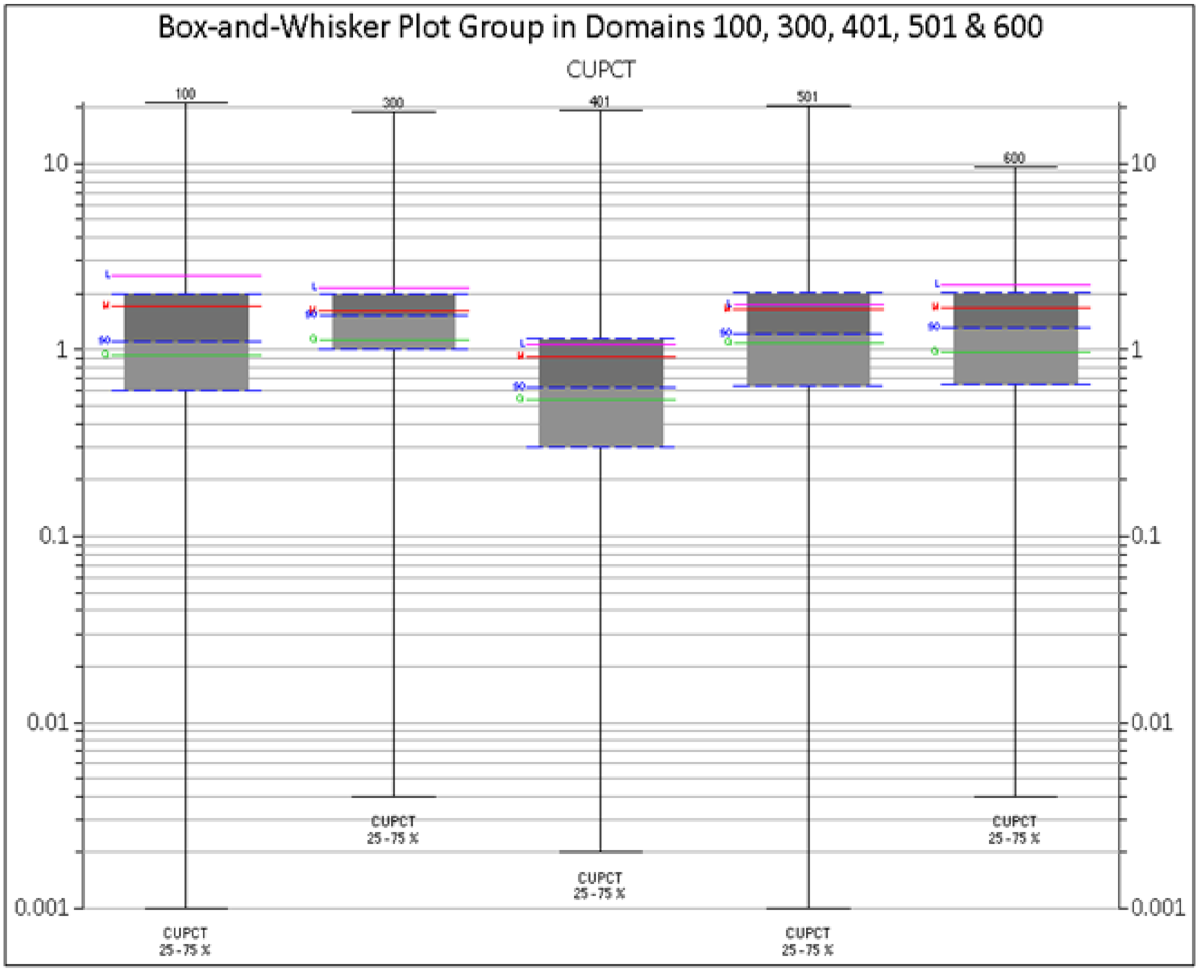


Figure 14.19 Side-by-Side Boxplots of 2.0 m Copper Composites (Domains 201, 202, 701, 702, 703)

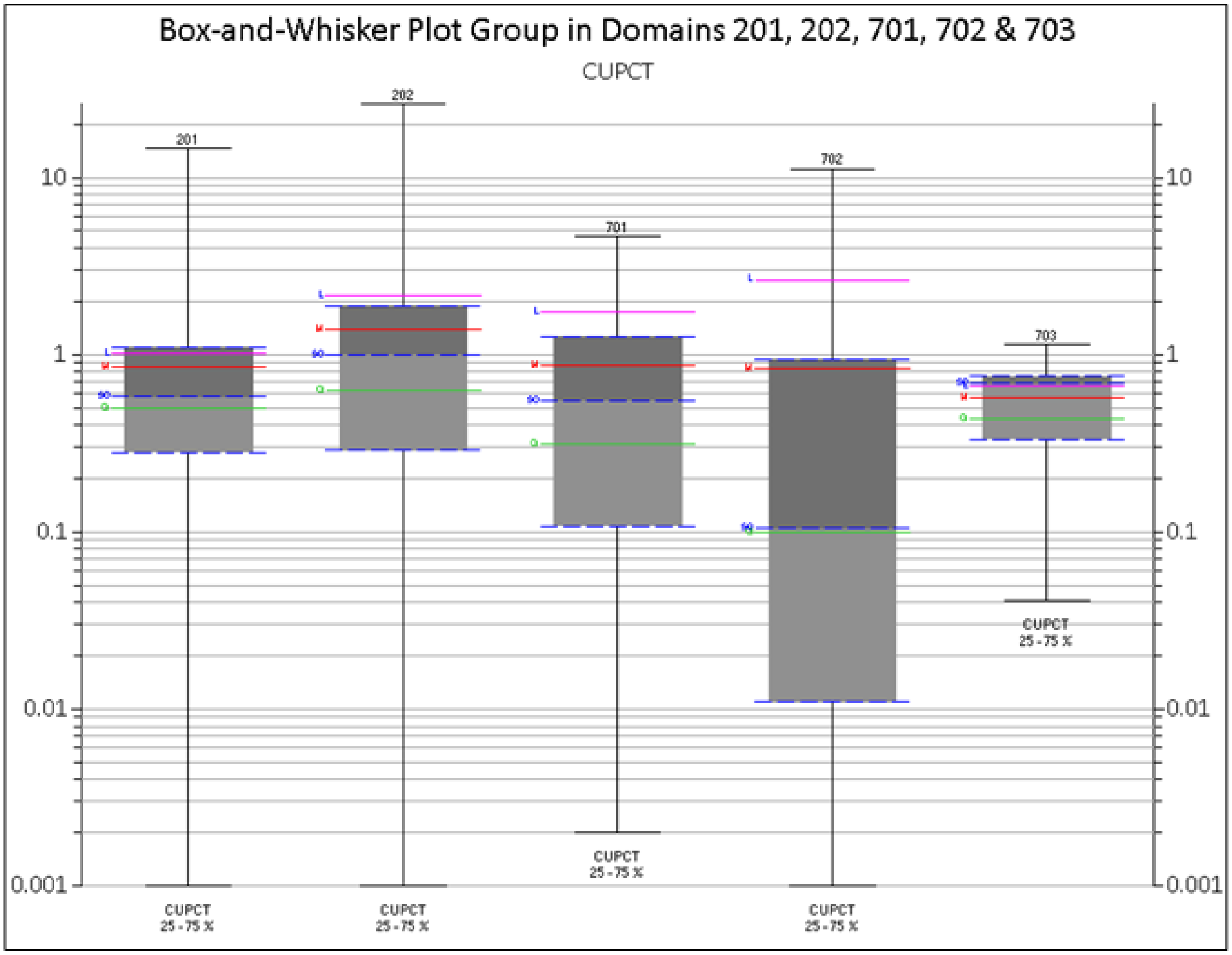


Figure 14.20 Side-by-Side Boxplots of 2.0 m Gold Composites (Domains 100, 300, 401, 501, 600)

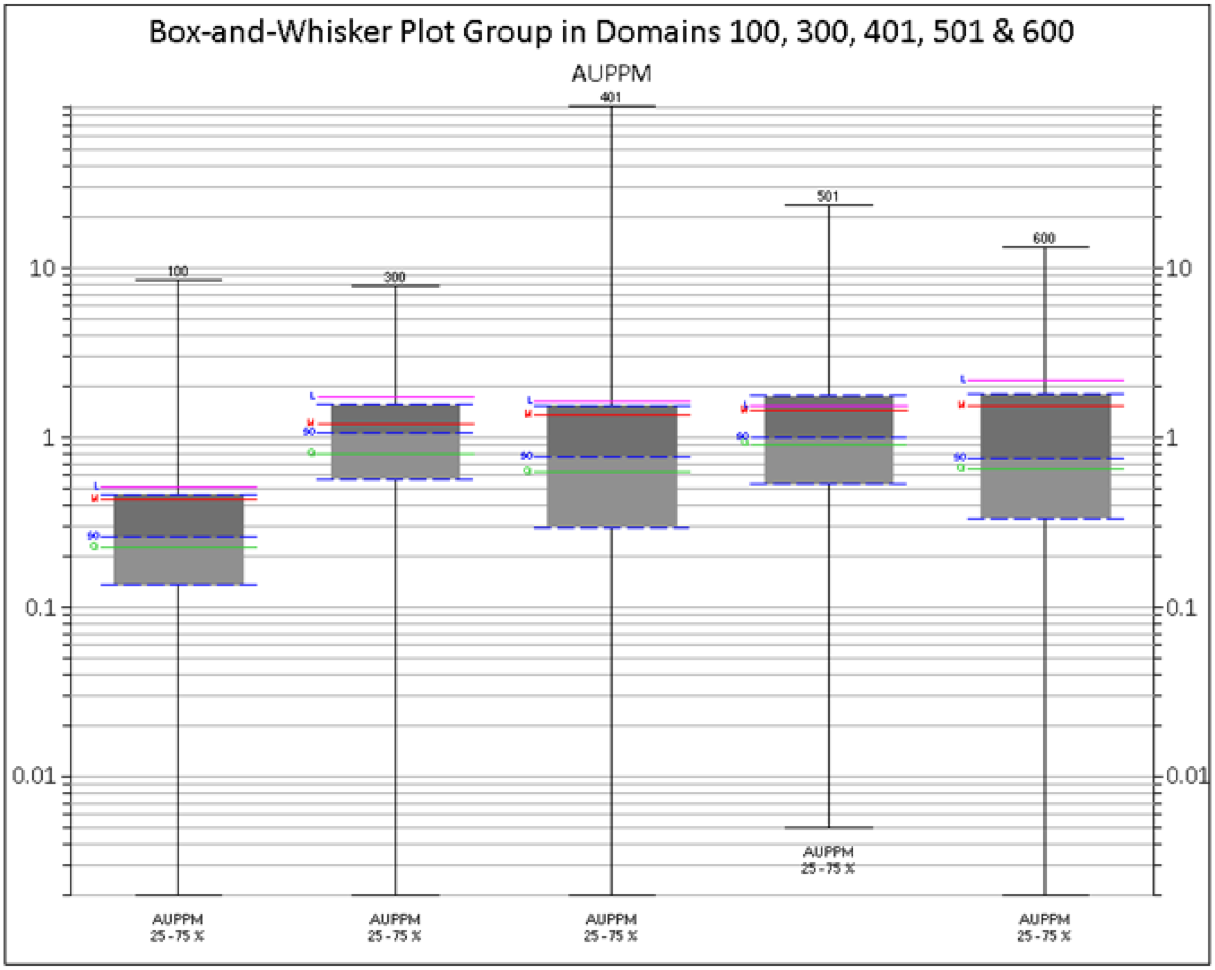


Figure 14.21 Side-by-Side Boxplots of 2.0 m Gold Composites (Domains 201, 202, 701, 702, 703)

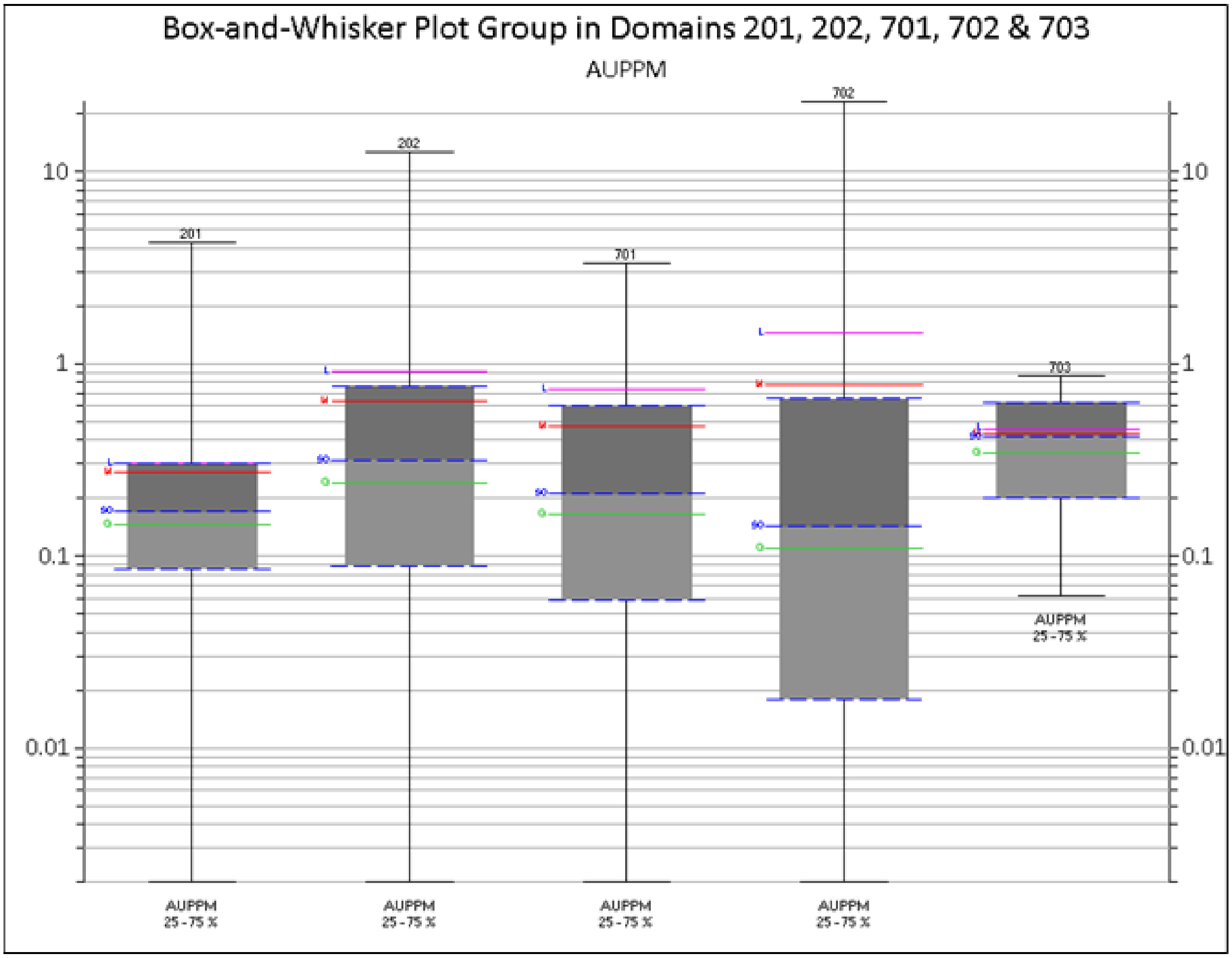


Figure 14.22 Side-by-Side Boxplots of 2.0 m Silver Composites (Domains 100, 300, 401, 501, 600)

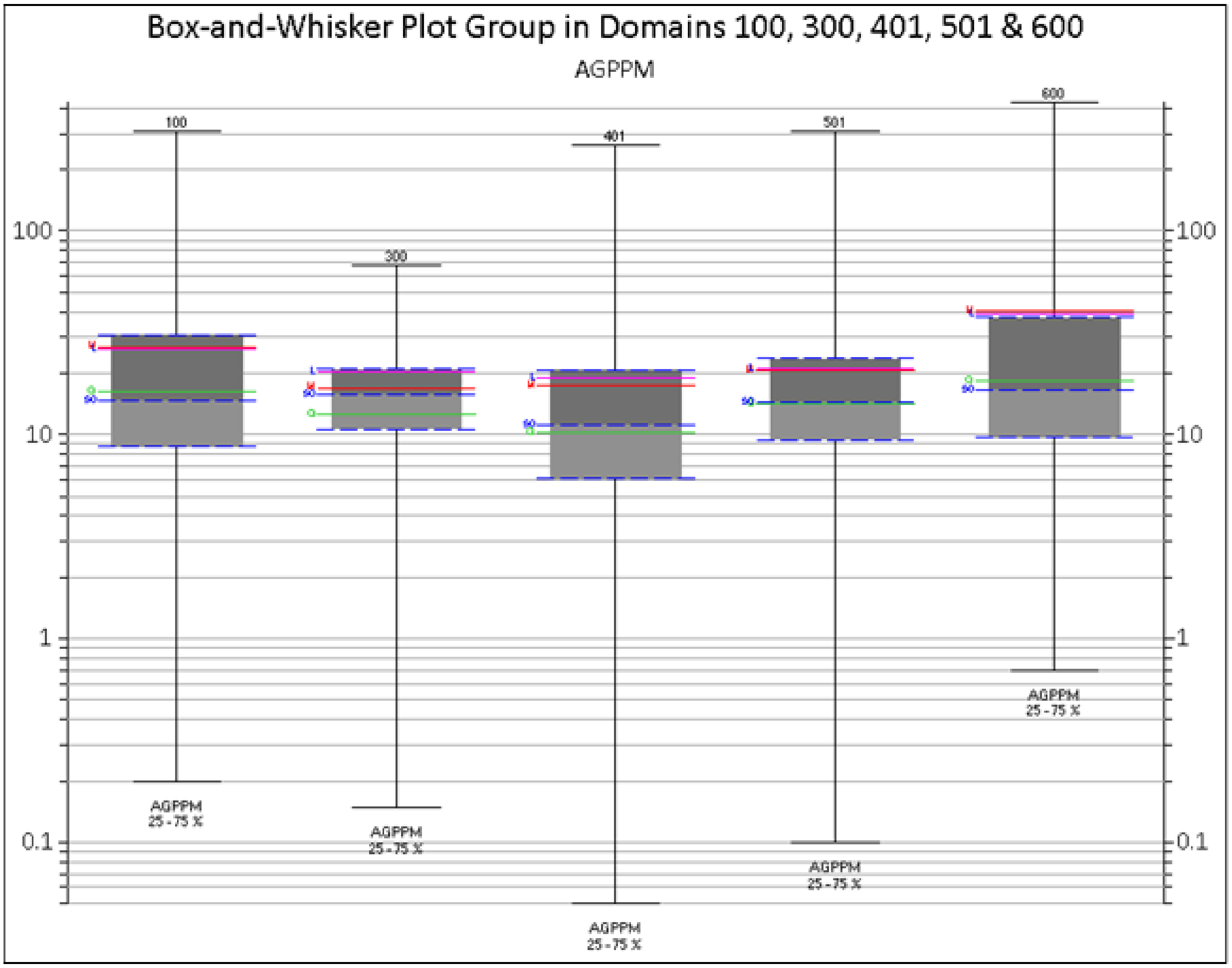
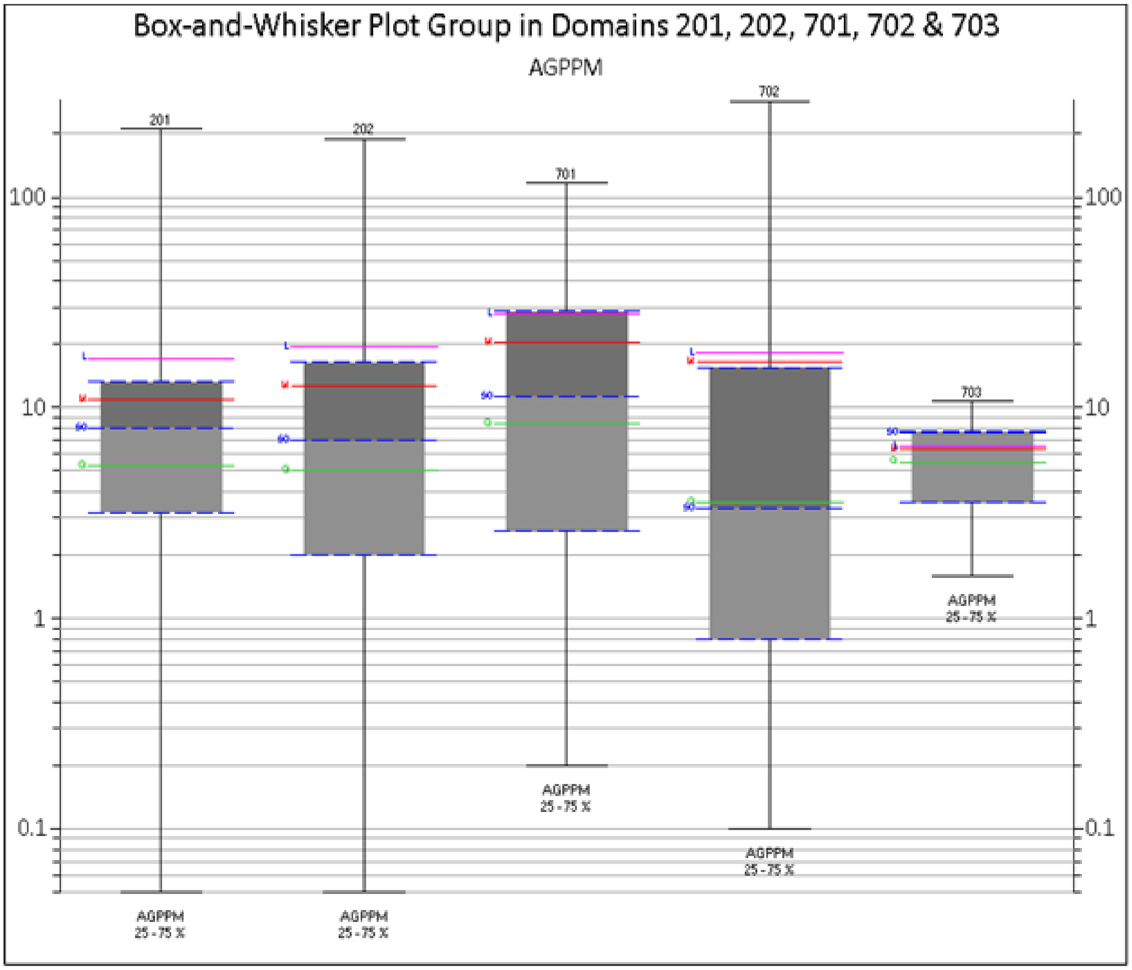


Figure 14.23 Side-by-Side Boxplots of 2.0 m Silver Composites (Domains 201, 202, 701, 702 and 703)



14.3.4 Evaluation of Extreme Grades

Statistical and graphical summaries, including histograms, log-probability plots, and coefficient of variation plots, were reviewed in order to identify the existence of anomalous outlier grades of copper, gold and silver in the composite database. Potential outliers, which are identified through the various statistical and graphical summaries, were then plotted on section and compared to the surrounding data.

The outliers identified were dealt with by top-cutting the 2 m composite grades to limit their influence during grade interpolation. The top cut values have been derived from the 2m un-assigned composite data, so that the choice of defaults would not influence the summary statistics. A total of 31 copper, 22 gold and 23 silver composites were capped. Table 14-9 shows the cap grade thresholds for copper, Table 14-10 shows the cap grade thresholds for gold and Table 14-11 shows the cap grade thresholds for silver.

Overall, the moderate CV's for the uncapped composites indicate that the grade distribution of the composites is not strongly affected by high-grade outliers; however, minimal capping values were applied to the majority of the domains.

Table 14-9 2.0 m Composite Cap Grades for Copper

Domain Name	Domain Code	Mean Cu (%)	CV*	Cap Cu Grade (%)	Capped Cu Mean (%)	Capped CV	No. of Assays Capped
BW	100	1.73	1.20	16	1.72	1.18	3
Mexicana	201	0.85	1.23	10	0.85	1.16	7
	202	1.38	1.23	14	1.37	1.16	7
AA	300	1.62	0.88	5.6	1.57	0.62	1
GHP HW	401	0.92	1.18	12	0.91	1.14	2
GHP FW	501	1.66	1.09	15	1.65	1.06	7
Cabrestante	600	1.68	1.02	8.5	1.68	1.01	4
Zone CC	701	No top cuts applied					
	702						
	703						
Total Capped							**31**

Table 14-10 2.0 m Composite Cap Grades for Gold

Domain Name	Domain	Mean Au (g/t)	CV*	Cap Au Grade (g/t)	Capped Au Mean (g/t)	Capped CV	No. of Assays Capped
BW	100	0.44	1.47	4	0.43	1.30	3
Mexicana	201	0.27	1.44	No top cuts applied			
	202	0.63	1.59	11	0.63	1.57	2
AA	300	1.20	0.78	4	1.18	0.70	3
GHP HW	401	1.37	2.32	23	1.31	1.53	6
GHP FW	501	1.45	1.13	13	1.44	1.07	5
Cabrestante	600	1.54	1.42	9	1.50	1.34	3
Zone CC	701	No top cuts applied					
	702						
	703						
Total Capped							**22**

Table 14-11 2.0 m Composite Cap Grades for Silver

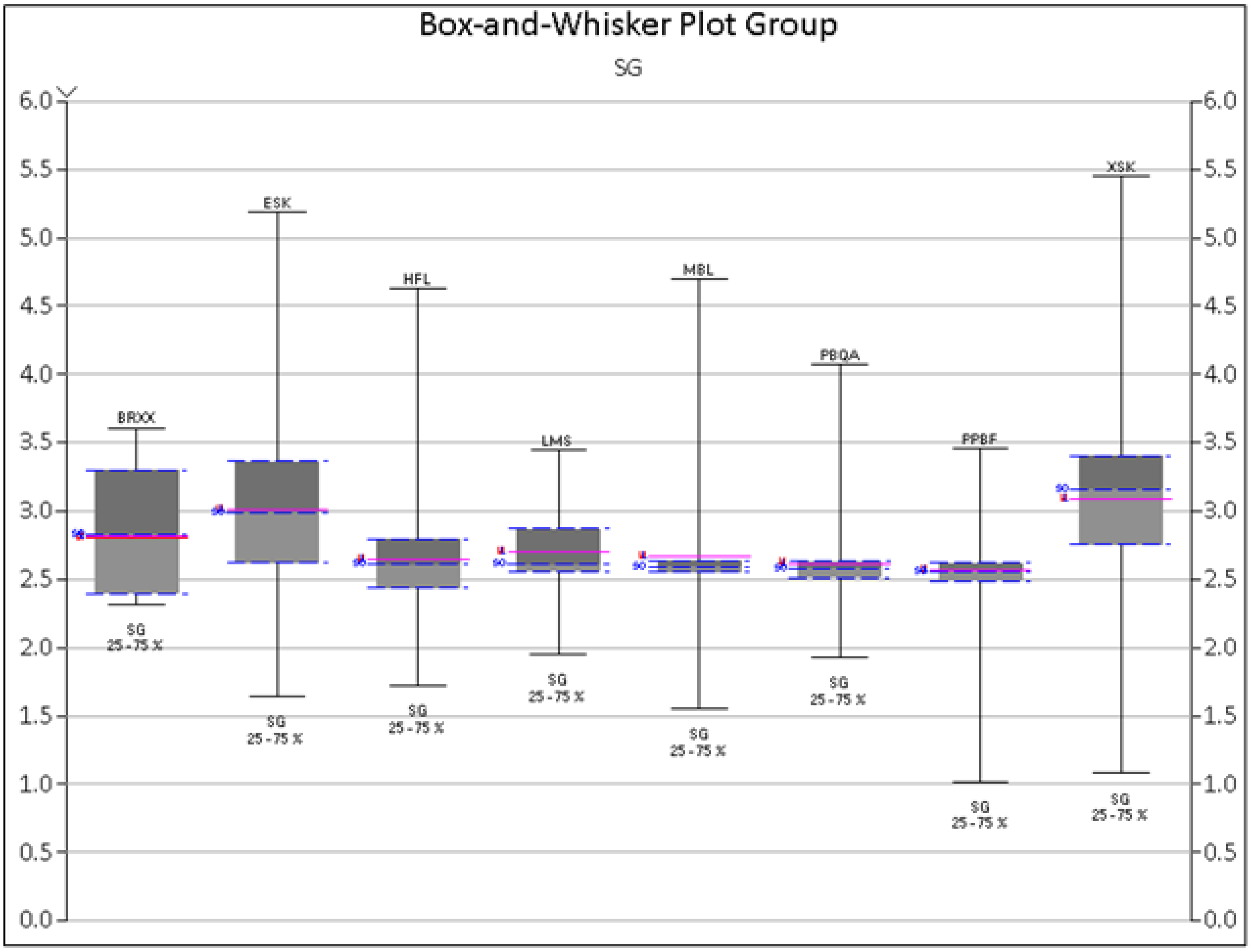
Domain Name	Domain Code	Mean Ag (g/t)	CV*	Cap Ag Grade (g/t)	Capped Ag Mean (g/t)	Capped CV	No. of Assays Capped
BW	100	27.02	1.25	240	26.91	1.22	4
Mexicana	201	10.99	1.35	100	10.82	1.22	9
	202	12.59	1.41	No top cuts applied			
AA	300	16.98	0.67	No top cuts applied			
GHP HW	401	17.56	1.26	200	17.48	1.22	5
GHP FW	501	20.72	1.10	250	20.68	1.08	2
Cabrestante	600	40.17	1.63	320	39.66	1.59	3
Zone CC	701	No top cuts applied					
	702						
	703						
Total Capped							**23**

14.3.5 Density Data

A total of 3,442 density measurements are recorded in the database that average 2.88 t/m^3. The drill core density measurements were separated by lithology for statistical analysis. Figure 14.24 displays side-by-side box plots and summary statistics of the density measurements by lithology type. There does appear to be an increase in average density with an increase in average copper grade, as the skarn lithologies (XSK and ESK) have higher average copper grades than do the limestone, marble, hornfels and intrusive lithologies.

A large number of sample intervals within the mineralized domains do not have specific gravity values and therefore, assigned values were applied. An assigned density value was chosen for each lithological unit where specific gravity measurements were missing, using either the mean or weighted average of the mean and median values.

Figure 14.24 Side-by-Side Boxplots of Density Measurements by Lithology



The assigned density values were then applied by lithology to all the missing specific gravity locations (Table 14-12). Then the density data were composited to 2.0 m regular composites using the same methodology that was used for the copper grades. The 2.0 m composite data were then used to construct a block model estimate of density using an inverse distance squared methodology. A statistical analysis of the density composite data was carried out to assess possible outlier or unreliable data. No adjustments were made to the density measurements other than to assign default grades to those drill holes that were missing density measurements.

Table 14-12 Lithology Density Default Values for 2.0 m Composites

Lithology Code	Integer Code	Type	Density (t/m3)
CSS	3	default	2.700
LMS	4	weighted average of the mean and medians combined	2.730
MBL	5	weighted average of the mean and medians combined	2.730
HFL	6	mean	2.715
XSK	7	mean	3.114
ESK	8	mean	3.050
PBQA	9	weighted average of the mean and medians combined	2.640
PPBF	10	weighted average of the mean and medians combined	2.640

14.4 GRADE MODEL AND INTERPOLATION PLAN

The block model limits were defined using UTM coordinates and the block size selected for the model was 5.0 m x 5.0 m x 5.0 m. This block size is also the same as that used in previous resource estimates. The model was rotated so that the north axis of the model is approximately parallel with the strike of the mineralization, 295°. For MineSight® software, this is a rotation of 25°. The block model definition is given in Table 14-13.

Table 14-13 Block Model Definition

	Minimum (m)	Maximum (m)	Distance (m)	Block Size (m)	No. Blocks
East	0	2200	2200	5	440
North	0	1300	1300	5	260
Elevation	1400	2400	1000	5	200

The protocols used to estimate the blocks in the block model have been designed using a philosophy of restricting the number of composites for local estimation. Restricting the number of composites helps to produce estimates that reflect the grade-tonnage distribution of a selective mining unit (SMU). While local estimates based on few composites can be conditionally biased, using this restricted search approach can produce a more reliable estimate of the global grade-tonnage curve. Figure 14.25 displays the block model limits with copper grades.

The block model variable DOMAN was coded using the domain code 3D wireframe solids along with LITHD (lithology), ALTN (alteration), OXIDE (weathering surface), and UGWK% (depletion solids). As well, the DOMAN code 3D wireframes were used to code the percentage of a block that was inside these wireframes. These partial blocks were coded into the block model item ORE%. The block model was also coded for the mined surface (item TOPO%), such that a block completely below the mined surface was given a value of 100%, while a block completely above the topography surface was given a value of zero percent.

The search ellipsoids were oriented based on the local orientation of the geological interpretation and ranges of continuity obtained from the variography. Figure 14.26 displays a plan view of the search ellipsoids showing how the directions of continuity change by domain code.

14.4.1 Copper, Gold, Silver and Arsenic Grade Interpolation

The copper, gold, silver and arsenic grades were estimated using Inverse distance squared estimation (ID^2) and the 2.0 m assigned and capped composites. Each domain was estimated using only the composites tagged with those domain codes, i.e. hard boundaries. Copper search parameters were used for gold, silver and arsenic grade estimates.

Down-the-hole copper correlograms were initially computed to determine the variogram range and principal direction of grade continuity for each of the nine mineralization domains. The resulting experimental correlograms were modelled using a nugget effect and two spherical structures. Copper search parameters were designated after correlograms for copper were modelled, and these parameters were used in the anisotropic ID^2 estimate.

The search strategy used a nested three pass approach for the search ellipsoids oriented in the direction of maximum continuity. The first pass used a large search ellipsoid expanded to in-fill the mineralized wireframes. The second pass search radius was then chosen to be approximately the range of the correlogram model, and the third search was small with a distance of approximately 80% of the correlogram model. These passes were run in sequence from the largest to smallest search ellipsoid, so as to not overwrite the estimates with the least amount of confidence.

The search strategy required a minimum of four to six and a maximum of 10 to 30 2.0 m composites to make a block estimate, with a maximum of two composites per drill hole, and a maximum of four from any single quadrant. This requires that at least two drill holes are used in a block estimate for estimation passes one, two and three. Table 14-14 contains a detailed list of the search strategy used for ID^2 copper, gold, silver and arsenic grade estimation.

Upon completion of the estimate the block model was checked for missing blocks. All missing IDCU, IDAU and IDAG estimates were re-assigned to 0.01 % or g/t as they fell outside of the wide spaced search criteria.

Table 14-15 provides a list of the number of blocks in each domain and number of blocks for which grade was estimated. Approximately 99.7% of the mineralized domain wireframes have copper, gold and silver block grade estimates. The remaining 0.3% of total blocks were not estimated because they did not fulfil the block model estimation criteria of using greater than one drill hole per block estimate. A total of 358 blocks from domain 401 were assigned a default value of 0.01 % or g/t for copper, gold and silver (Figure 14.27).

Figure 14.25 Plan View of the Block Model Area

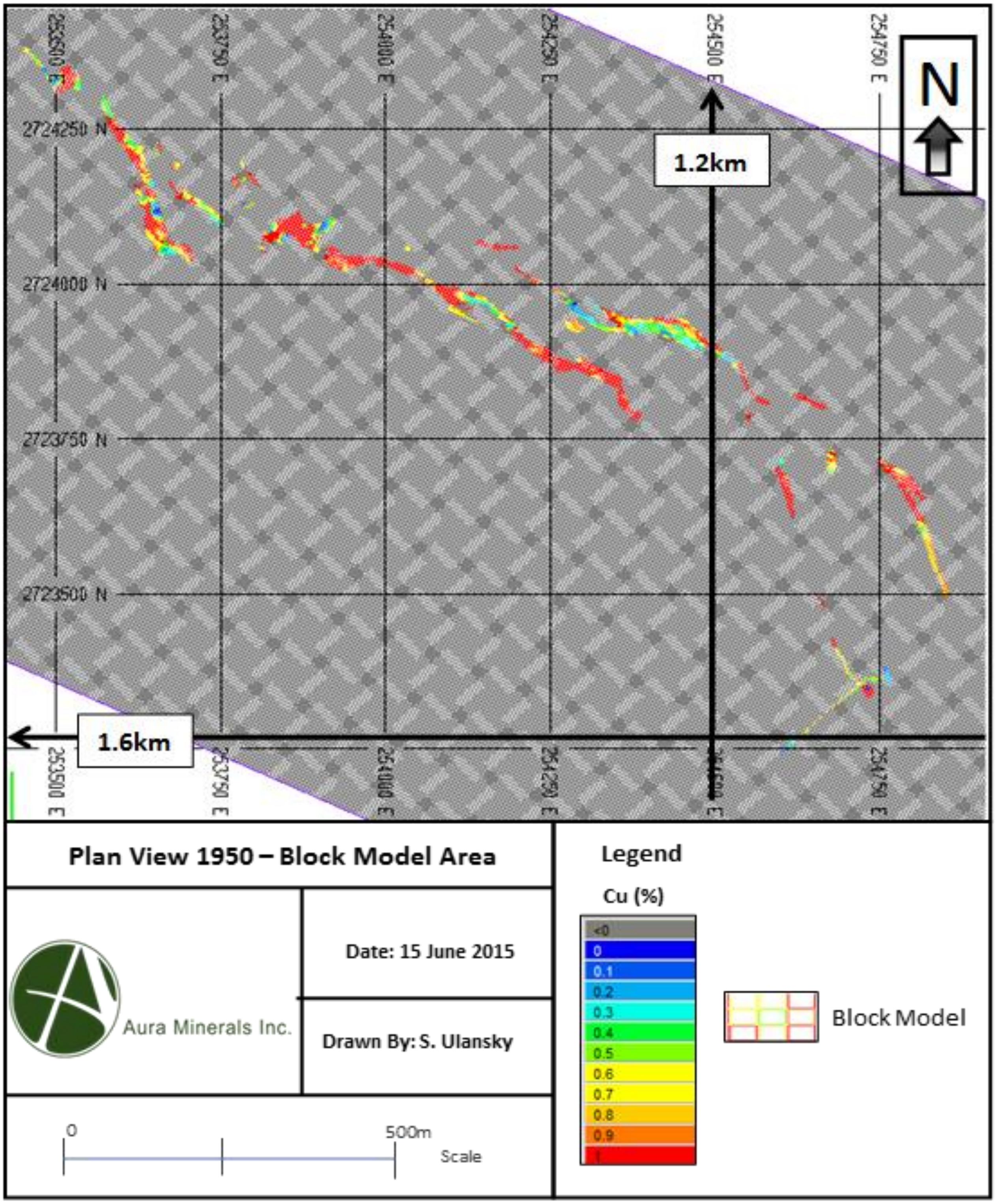


Figure 14.26 Plan View of the Copper Search Ellipsoids by Domain Code 100 to 702

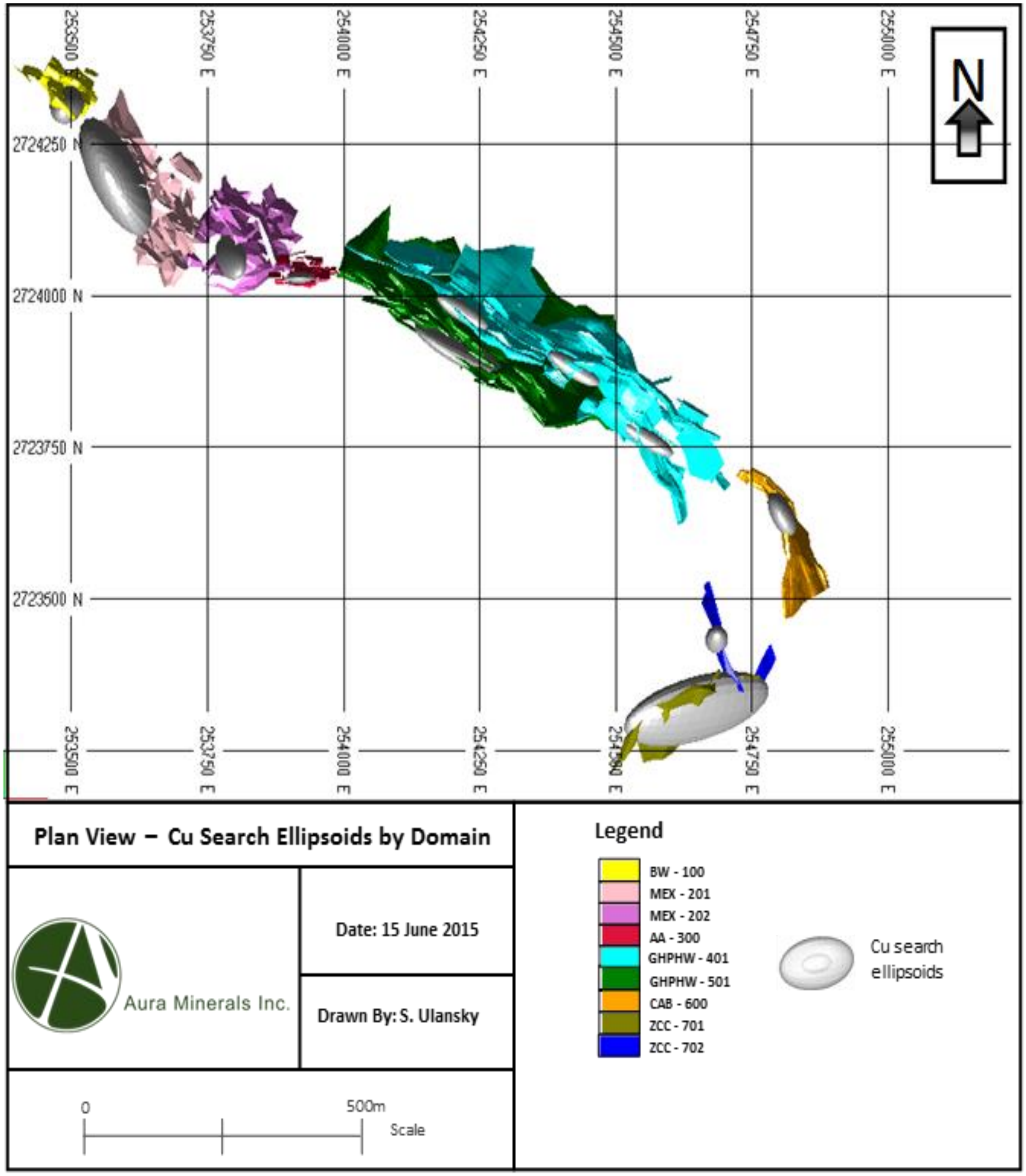


Table 14-14 Search Strategy for Inverse Distance Copper, Gold, Silver and Arsenic by Domain Code 100 to 702

Domain Code	Pass No.	Radius 1 (m)	Radius 2 (m)	Radius 3 (m)	ROT1 (Z)	ROT2 (X)	ROT3 (Y)	Min. Cmps	Max. Cmps	Max per DH	Max per Quad.	Max No. Empty Quad.
BW-100	1	260	90	90	-145	80	-180	4	10	2	4	3
BW-100	2	85	30	30	-145	80	-180	4	20	2	4	3
BW-100	3	50	22	20	-145	80	-180	6	30	2	4	3
MEX-201	1	270	210	160	-120	85	0	4	10	2	4	3
MEX-201	2	90	70	30	-120	85	0	4	20	2	4	3
MEX-201	3	30	30	20	-120	85	0	6	30	2	4	3
MEX-202	1	160	60	44	-32.7	-67.7	25.5	4	10	2	4	3
MEX-202	2	80	30	22	-32.7	-67.7	25.5	4	20	2	4	3
MEX-202	3	54	25	16	-32.7	-67.7	25.5	6	30	2	4	3
AA-300	1	195	90	30	0	90	-180	4	10	2	4	3
AA-300	2	65	30	10	0	90	-180	4	20	2	4	3
AA-300	3	17	10	4	0	90	-180	6	30	2	4	3
GHPHW-401	1	165	70	45	119	-14.5	-105.5	4	10	2	4	3
GHPHW-401	2	55	23	15	119	-14.5	-105.5	4	20	2	4	3
GHPHW-401	3	30	13	7	119	-14.5	-105.5	6	30	2	4	3
GHPFW-501	1	300	220	100	111.7	-39.3	-103	4	10	2	4	3
GHPFW-501	2	60	60	20	111.7	-39.3	-103	4	20	2	4	3
GHPFW-501	3	30	30	9	111.7	-39.3	-103	6	30	2	4	3
CAB-600	1	150	120	60	-120	85	0	4	10	2	4	3
CAB-600	2	50	40	20	-120	85	0	4	20	2	4	3
CAB-600	3	40	20	10	-120	85	0	6	30	2	4	3
ZCC-701	1	220	120	50	-100.5	35.6	58.7	4	10	2	4	3
ZCC-701	2	110	60	25	-100.5	35.6	58.7	4	20	2	4	3
ZCC-701	3	50	25	10	-100.5	35.6	58.7	6	30	2	4	3
ZCC-702	1	240	120	100	-176.3	-78.8	-116.7	4	10	2	4	3
ZCC-702	2	60	40	25	-176.3	-78.8	-116.7	4	20	2	4	3
ZCC-702	3	30	20	12	-176.3	-78.8	-116.7	6	30	2	4	3

All rotations are in degrees and follow the left hand rule in the order Z, X, Y. Dips are positive up and negative down.

Table 14-15 Count of the Number of Combined Copper, Gold and Silver Block Grade Estimates

Domain	No. of Blocks	% of Total	No. of Est. Blocks	% Estimated
100	7082	5%	7082	100.0%
201	20484	15%	20484	100.0%
202	16142	12%	16137	100.0%
300	2548	2%	2548	100.0%
401	32830	24%	32472	98.9%
501	43154	32%	43154	100.0%
600	6444	5%	6444	100.0%
701	4809	4%	4809	100.0%
702	2459	2%	2459	100.0%
100-702	**135952**	100%	**135589**	99.7%

Table 14-16 shows summary statistics for combined block model grade estimates for all the domain codes. Estimates for domain codes 100 to 702 were estimated using inverse distance squared. For domains 100 to 702 there are 135,952 block estimates with an average grade of 1.31% Cu, 0.92 g/t Au, 17.40 g/t Ag and $85.00/t NSR. Note that all model statistics were reported exclusive of a topographic surface.

Figure 14.27 Long Section View of Domain 401 Showing the Unestimated Blocks in the Inverse Distance Model

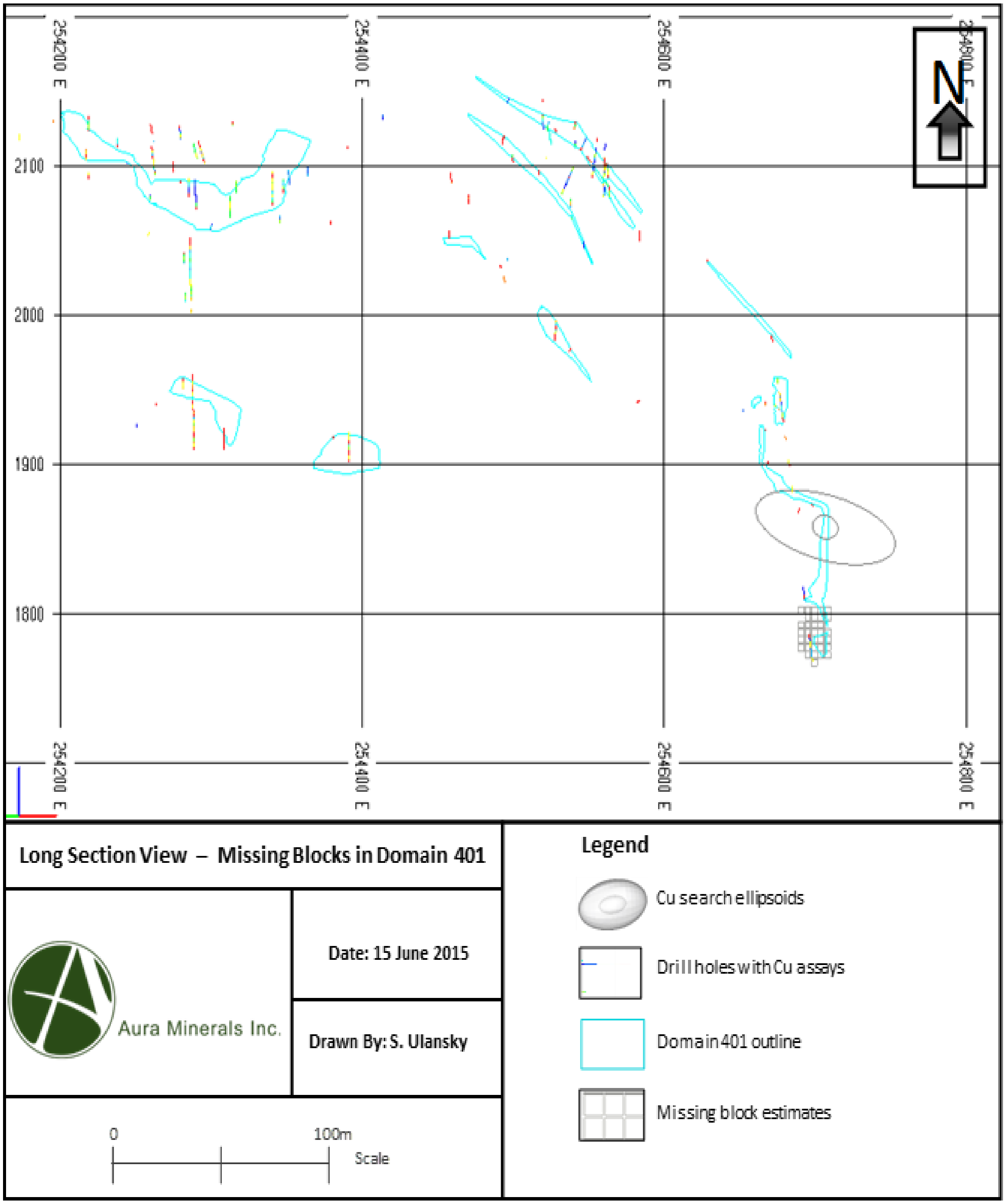


Table 14-16 Summary Statistics for Combined Block Model Estimates of Cu, Au, Ag and NSR

Domain	Cu (%)			Au (g/t)			Au (g/t)			NSR ($/t)		
	No. of Est.	Mean	CV	No. of Est.	Mean	CV	No. of Est.	Mean	CV	No. of Est.	Mean	CV
100	7,082	1.51	0.76	7,082	0.402	0.967	7,082	23.23	0.78	6,968	85.77	0.84
201	20,484	1.03	0.65	20,484	0.267	0.797	20,484	10.73	0.71	19,685	52.49	0.82
202	16,142	1.24	0.71	16,142	0.488	0.938	16,142	10.61	0.83	15,312	70.76	0.82
300	2,548	1.46	0.31	2,548	1.043	0.499	2,548	14.44	0.37	2,532	93.52	0.34
401	32,830	1.06	0.70	32,830	1.121	0.999	32,830	19.43	0.78	32,070	75.65	0.71
501	43,154	1.69	0.55	43,154	1.333	0.624	43,154	19.44	0.58	43,135	113.57	0.61
600	6,444	1.48	0.52	6,444	0.983	0.949	6,444	27.96	0.92	6,379	98.48	0.50
701	4,809	0.69	0.68	4,809	0.458	0.917	4,809	17.26	0.66	4,241	43.11	0.70
702	2,459	0.87	1.00	2,459	1.244	2.274	2,459	13.45	1.26	2,006	80.05	0.93
100 - 702	**135,952**	**1.31**	**0.64**	**135,952**	**0.918**	**0.849**	**135,95**	**17.40**	**0.71**	**132,328**	**85.00**	**0.70**

14.4.2 Multielement Interpolation: Arsenic

Block model estimates were also made for arsenic because it is considered a penalty element. The estimation procedure used inverse distance squared (ID^2), capped 2.0 m composites and a three-pass search strategy, similar to that used for copper, gold and silver.

As there are fewer samples analyzed for the minor metals, the estimation procedure produced fewer block estimates than for copper, gold and silver. Estimation of arsenic was done to allow for different blending scenarios to manage the penalty element within specific tolerances. Table 14-17 summarize the unassigned statistics for the arsenic ID^2 block estimates. The estimation parameters and ranges for arsenic are the same as the estimation parameters and ranges defined for inverse distance copper.

Table 14-17 Summary Statistics for Block Model Estimates of As (without assigned blocks)

Domain	As (ppm)		
	No. of Ests	Mean	CV
100	4,706	509.1	0.86
201	20,484	390.4	0.54
202	16,035	1,088.7	0.90
300	2,470	1,717.3	0.47
401	28,986	1,234.7	0.88
501	37,657	1,802.5	0.61
600	4,483	1,912.9	0.80
701	4,809	1,260.8	0.87
702	2,372	439.1	2.06
100 to 702	**122,002**	**1,241.2**	**0.75**

Following the ID^2 estimation of arsenic, blocks that have a copper estimate but do not have estimates of arsenic were assigned default values of arsenic using regression equations developed in Excel. The regression equations for the arsenic default values consider the best relationship with copper. This was done to ensure that every block that has a copper, gold and silver estimate, would also have an estimate of arsenic.

Table 14-18 shows a list of the equations used to predict the values of arsenic. Table 14-19 display summary statistics for the block model estimates that include those predicted using the regression equations. The number of assigned blocks used for arsenic has been tabulated in Table 14-20. In general, the means of the block model estimates, which include the regression prediction grades, are similar to the means of the direct ID2 estimates. There is a 2.0% increase in arsenic grades in the final block model versus the unassigned arsenic model and a 4.0% decrease in CV.

Table 14-18 Equations used to fill in un-estimated blocks for Arsenic

Metal	Domains	Constraints	Equation
As	100	Cu ≤1.5 and As <0	(Asppm) = -334.55 * (Cu%)^2 + 836.86 * (Cu%) – 2.0981
		Cu >1.5 and As <0	(Asppm) = 38.497 * (Cu%) + 576.53
	202	Cu ≤2.0 and As <0	(Asppm) = -757.14 * (Cu%) + 638.72
		Cu > 2.0 and As <0	(Asppm) = 2000
	300	Cu ≤ 1.8 and As <0	(Asppm) = -247.71 * (Cu%)^2 + 1187.6 * (Cu%) + 536.55
		Cu >1.8 and As<0	(Asppm) = 2000
	401	Cu ≤1.3 and As <0	(Asppm) = -764.83 * (Cu%)^2 + 1716.1 * (Cu%) + 360.54
		Cu >1.3 and As <0	(Asppm) = -169.63 * (Cu%)^2 + 1362 * (Cu%) – 482.16
	501	Cu ≤3.8 and As <0	(Asppm) = 542.95 * (Cu%) + 950.11
		Cu >3.8 and As <0	(Asppm) = 2500
	600	Cu ≤2.0 and As <0	(Asppm) = 267.61 * (Cu%) ^2 + 878.4 * (Cu%) + 350.02
		Cu >2.0 and As <0	(Asppm) = 2000
	702	Cu ≤1.75 and As <0	(Asppm) = 148.04 * (Cu%)^2 – 351.4 * (Cu%) + 554.78
		Cu >1.75 and As <0	(Asppm) = 500

Table 14-19 Summary Statistics for Block Model Estimates of As (with Assigned Grades)

Domain	As (ppm)		
	No. of Ests	Mean	CV
100	7,082	514.9	0.70
201	20,484	390.4	0.54
202	16,142	1091.0	0.89
300	2,548	1714.5	0.46
401	32,830	1212.1	0.85
501	43,154	1818.1	0.57
600	6,444	1949.3	0.67
701	4,809	1260.8	0.87
702	2,459	437.5	2.03
100 to 702	**135,952**	**1262.0**	**0.72**

Table 14-20 The number of Adjusted Multielement blocks using Regression Formulas

DOMAIN	No. of Blks
100	2,376
201	0
202	107
300	78
401	3,844
501	5,497
600	1,961
701	0
702	87
100 to 702	**13,950**

14.4.3 Density

A density model was interpolated using inverse distance squared weighting (ID2), 2.0 m composites of density and a one pass search strategy oriented in the directions of continuity established for copper.

Where no density measurements were taken an assigned density, value was given to those drill hole intervals that had a valid lithology code within the mineralized domains. Either the mean or the weighted average of the mean and medians combined, were used (Table 14-21). This approach was used to better model the relationship between lithology type and density because the skarn lithologies typically have higher density measurements. A study was done to test the relationship between specific gravity and copper but no significant relationships were found, therefore the assigned SG values based on lithology type were retained.

Table 14-21 Lithology Density Default Values

Lithology Code	Integer Code	Type	Mean	Median	Density (t/m^3)
INFL	2	N/A		2.7	
CSS	3	N/A		2.7	
LMS	4	weighted average of the mean and medians combined	2.897	2.99	2.73
MBL	5	weighted average of the mean and medians combined	2.756	2.59	2.73
HFL	6	mean	2.715	2.58	2.715
XSK	7	mean	3.114	3.19	3.114
ESK	8	mean	3.05	3.05	3.05
PBQA	9	weighted average of the mean and medians combined	2.735	2.59	2.64
PPBF	10	weighted average of the mean and medians combined	2.631	2.59	2.64
OS	11	N/A		N/A	
NR and GAP	12	N/A		N/A	

** For lithologies: INFL, OS, NR and GAP a default value of 2.7 t/m3 was applied
** There were no missing values for BRXX and CSS

The interpolation of density values for each zone used search ellipsoid orientations based on the directions of continuity determined from copper grades. A minimum of three and maximum of six data were required to make a block estimate, with a maximum of three data from any one drill hole. This allows a single drill hole to make a block model density estimate. Only composite lengths between the ranges of 1.0 m to 3.0 m in length were estimated.

A list of the search strategy and ellipsoid rotation angles used for the estimation of density is provided in Table 14-22. The weighted average density of 135,952 blocks for mineralized domains (100 to 702) is 2.95 t/m^3 and compares well with average of the 38,122-composite data, which is 2.95 t/m^3 (Table 14-23).

Table 14-22 Search Strategy and Ellipsoid Rotation Angles for Density Estimation

Domain Code	Pass No.	Radius 1 (m)	Radius 2 (m)	Radius 3 (m)	ROT1 (Z)	ROT2 (X)	ROT3 (Y)	Min. Cmps	Max. Cmps	Max per DH
BW-100	1	260	90	90	-145	80	-180	3	6	3
MEX-201	1	270	210	120	-120	85	0	3	6	3
MEX-202	1	160	60	44	-32.7	-67.7	25.5	3	6	3
AA-300	1	195	90	30	0	90	-180	3	6	3
GHPHW-401	1	165	70	45	119	-14.5	-105.5	3	6	3
GHPFW-501	1	300	220	100	111.7	-39.3	-103	3	6	3
CAB-600	1	200	160	80	-120	85	0	3	6	3
ZCC-701	1	220	120	50	-100.5	35.6	58.7	3	6	3
ZCC-702	1	240	120	100	155.8	-39.8	-83.5	3	6	3

All rotations are in degrees and follow the left hand rule in the order Z, X, Y. Dips are positive up and negative down

Table 14-23 Summary Statistics for Density Block Estimates and Composite Data

Domain	Density ID2 Estimate			Density 2m Composite Data		
	No. of Blks	Mean	CV	No. of Comps	Mean	CV
100	7,082	2.96	0.05	3,726	2.88	0.07
201	20,484	3.05	0.03	7,908	3.06	0.04
202	16,142	3.00	0.05	5,581	2.99	0.06
300	2,548	2.90	0.05	900	2.85	0.08
401	32,830	2.91	0.05	9,314	2.89	0.07
501	43,154	2.93	0.05	7,946	2.95	0.06
600	6,444	2.93	0.05	1,419	2.94	0.07
701	4,809	2.78	0.04	844	2.75	0.04
702	2,459	2.84	0.05	483	2.78	0.07
100 - 702	**135,952**	**2.95**	**0.05**	**38,121**	**2.95**	**0.06**

14.5 MODEL VALIDATION

14.5.1 Model Check for Bias

A check for global bias was carried out by comparing the average metal estimates between a nearest neighbor (NN) estimate, and the average ID^2 estimates for copper, gold, and silver. Table 14-24 contains the summary statistics for the NN and ID^2 estimates. The comparison between the NN and ID^2 estimates for copper, gold and silver has 4.0% or less variance between the two models, which is an acceptable difference. Based on this analysis the copper, gold and silver block model estimates are globally unbiased.

Table 14-24 Summary Statistics for Copper, Gold, Silver and Arsenic Block Model Estimates

Metal	No. of blocks	NN Model		No. of blocks	ID^2 Model		% Variance
		Mean	CV		Mean	CV	
Cu (%)	135,594	1.37	1.05	135,952	1.31	0.64	-4%
Au (g/t)	135,594	0.96	1.33	135,952	0.92	0.85	-4%
Ag (g/t)	135,594	17.91	1.14	135,952	17.40	0.71	-3%

14.5.2 Visual Validation

Detailed visual inspections of the copper, gold, silver and arsenic block estimates were conducted in both plan and section to ensure that the interpolation results honoured the geological boundaries and the drill hole data. This validation included confirmation of the proper coding of blocks for each of the domains and the distribution of block grade estimates relative to the capped 2.0 m drill hole composites, to ensure that the drill hole data were properly represented in the model.

Figure 14.28 displays a cross-section at 925 N (12.5 m Section Grid) looking northwest, displaying the Glory Hole-Porfido Footwall mineralization boundary and the block grades colour coded by copper grade. The 2.0 m capped drill hole composites compare reasonably well with the copper grade blocks.

Figure 14.29 displays a cross-section at 1,000 N (25 mm Aranzazu Grid) looking northwest, displaying the Glory Hole-Porfido Footwall boundary and the block grades colour coded by copper grade. The 2.0 m capped drill hole composites are also shown using the same colour scheme. The block copper grade estimates match the drill hole copper intercepts reasonably well.

Based on the examination of plans and sections, and an interrogation of selected block grades, the estimates can be explained as a function of the surrounding composites, the search ellipse ranges and orientations used, as well as the applied kriging plan (Figure 14.30).

Figure 14.28 Section 925 (25 m Aranzazu Grid) Displaying ID2 Copper Block Estimates and 2.0 m Capped Copper Composite Grades

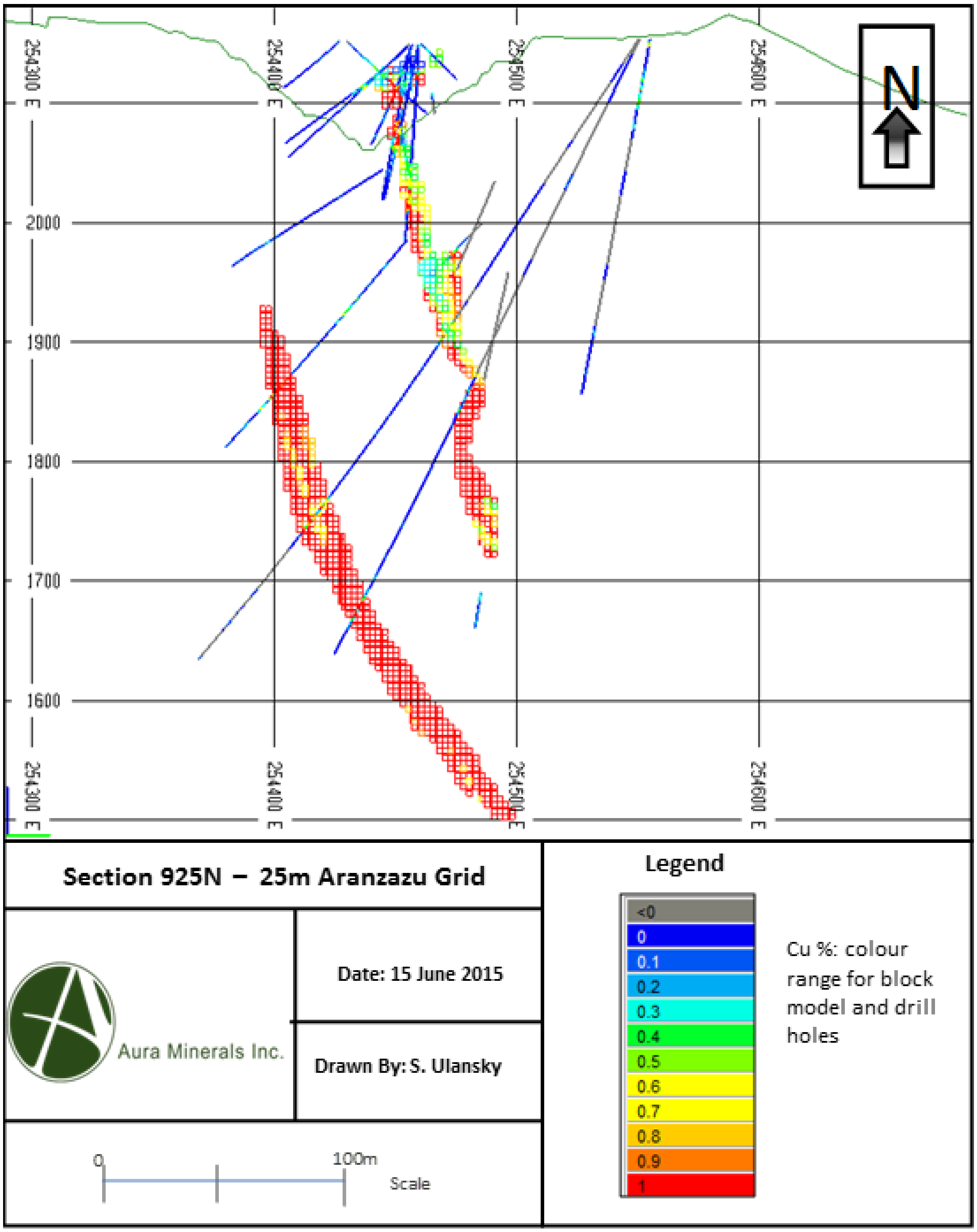


Figure 14.29 Section 1,000 N (25 m Aranzazu Grid) Displaying Glory Hole–Porfido ID² Copper Block Estimates and 2.0 m Capped Copper Composite Grades

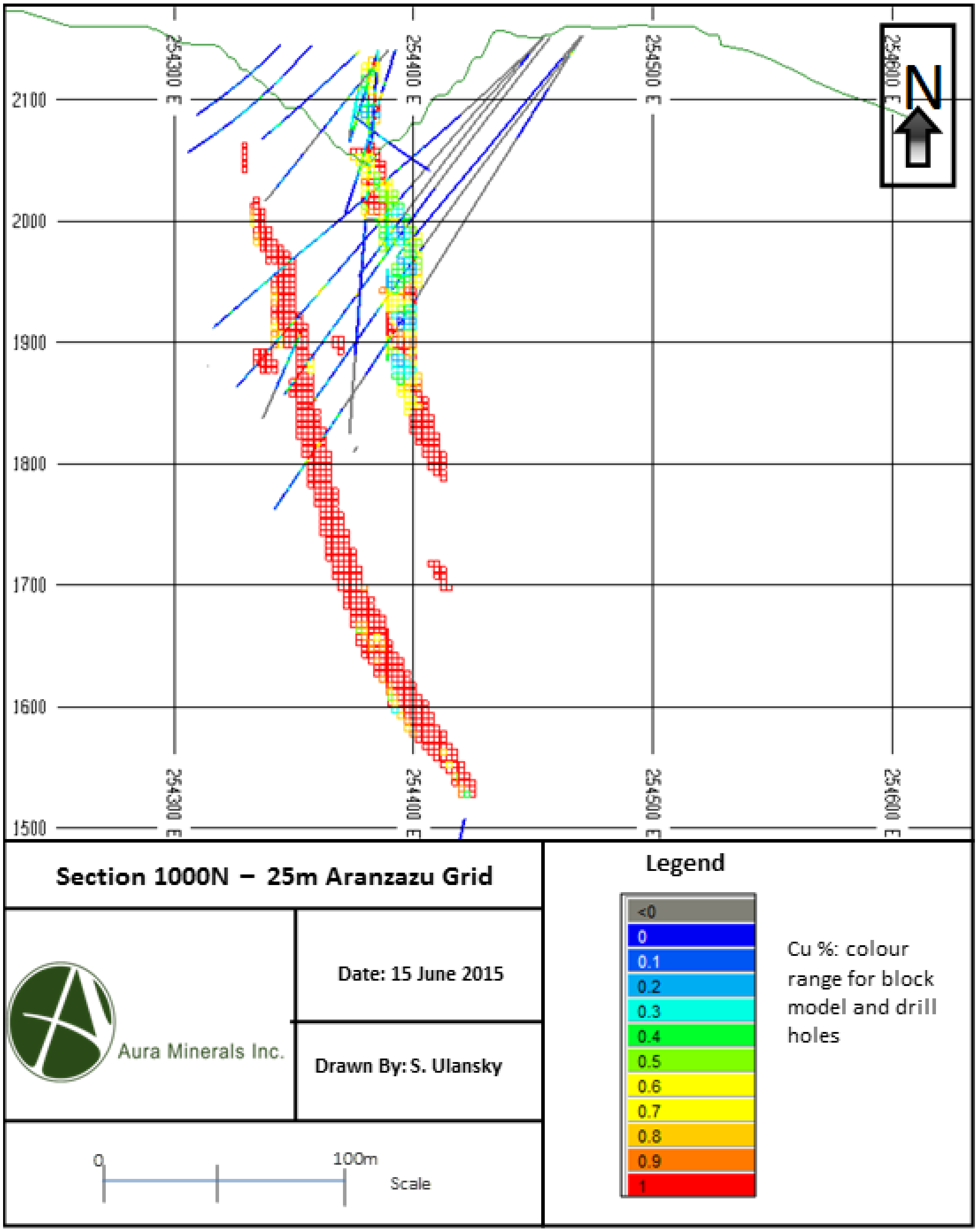
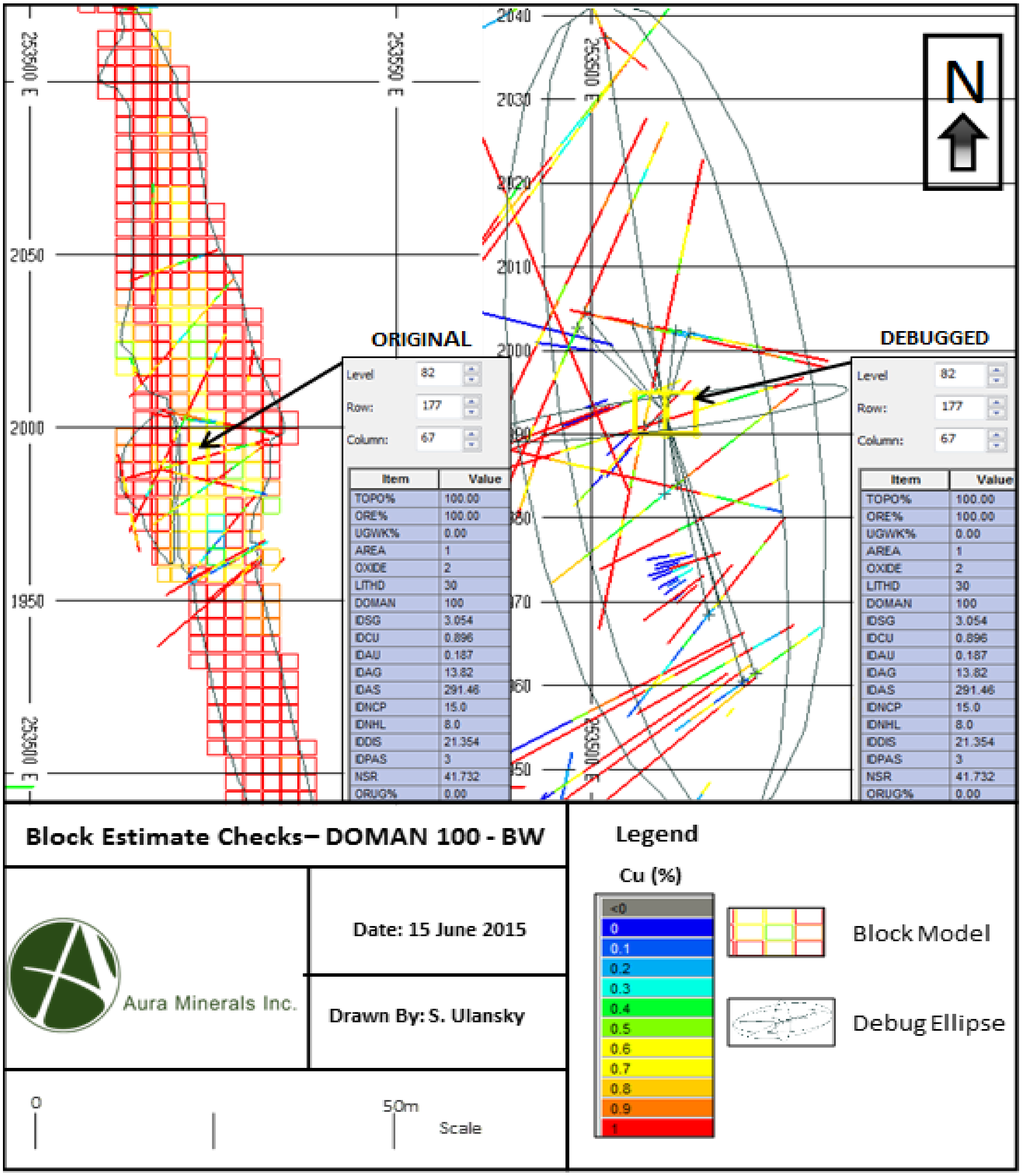


Figure 14.30 Block Validation of Estimation Parameters in Domain 100 – BW



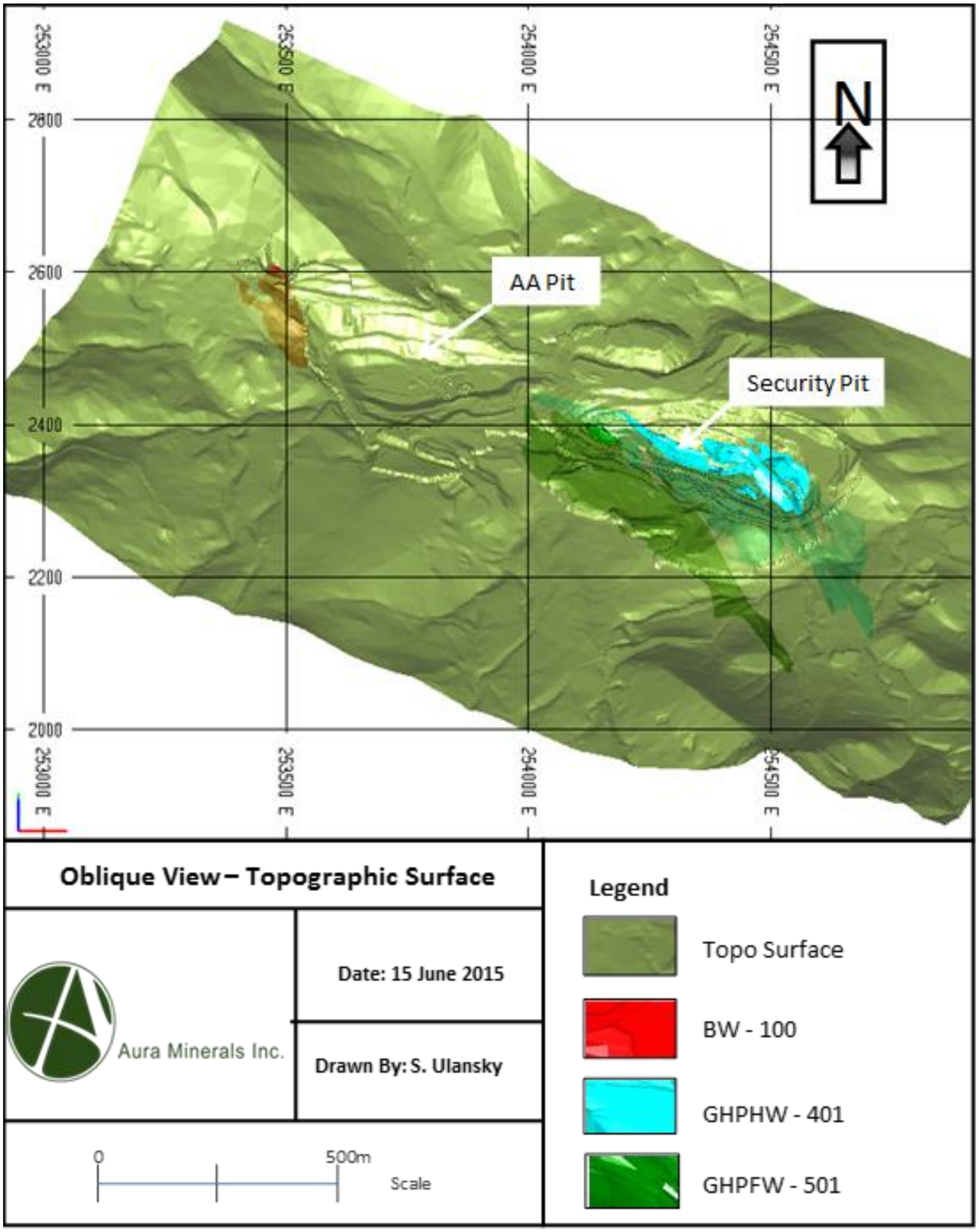
14.6 TOPOGRAPHY AND UNDERGROUND WORKINGS

The surface topography is based on contours interpreted from satellite imagery, which have been frequently updated with pit expansions. The latest mined topographic surface is current to 30 November 2014.

The mined topographic surface was used to code the block model into a field called TOPO%. A block completely above the surface topography was coded as 0%, while blocks that were intersected by the surface were proportionally adjusted to remove the percentage of the block that lies above the surface. These partial blocks were coded with percentages ranging from greater than 0% to less than 100%. A block completely below the surface was coded as 100%.

Figure 14.31 is an oblique view of the topography showing the AA (Arroyos Azules) Pit to the northwest and the Security Pit to the southeast. The domains 100, 401 and 501 have been included for reference.

Figure 14.31 Oblique View of the Mined Topographic Surface

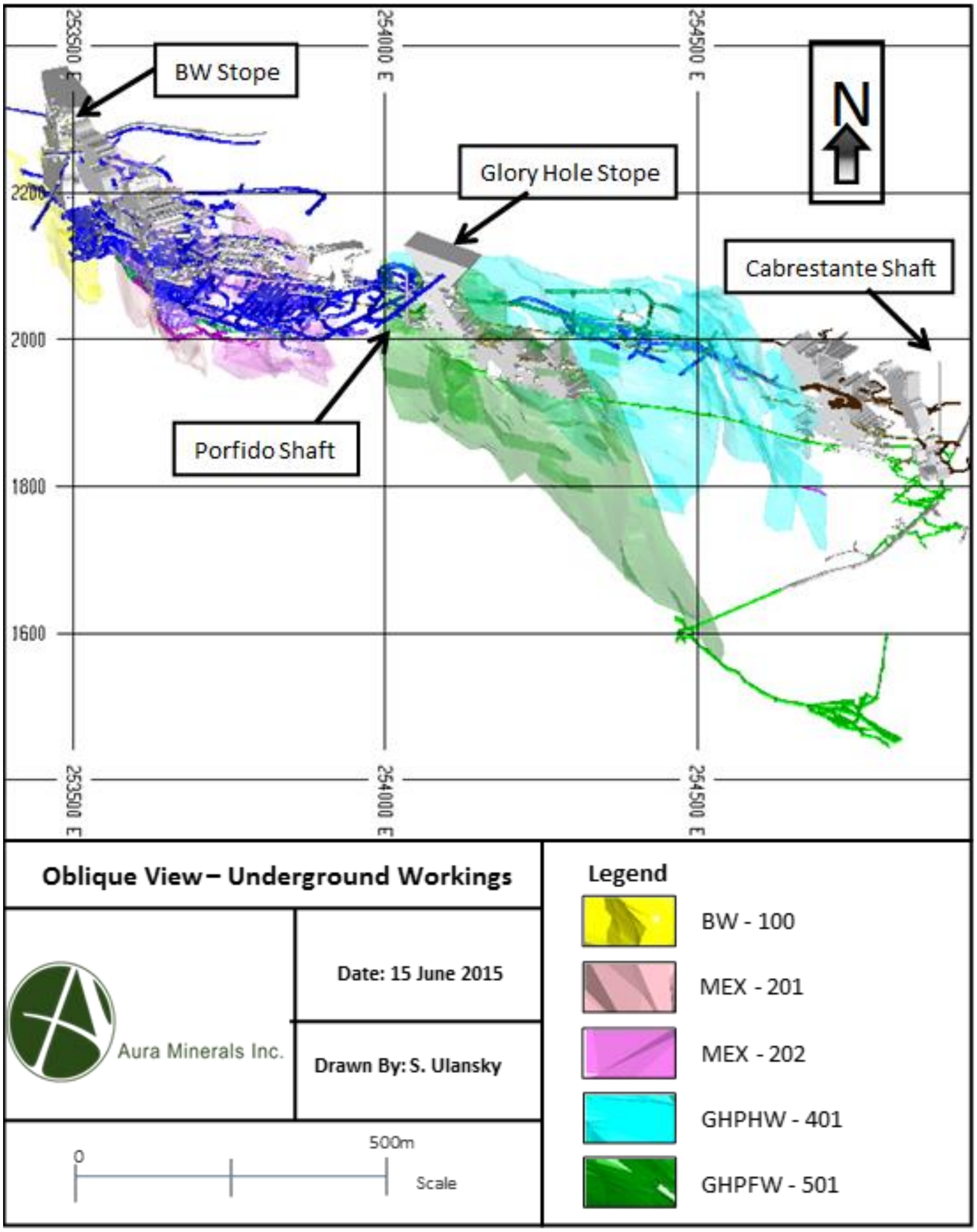


The latest underground workings solids are current as of end of the year of 2015. The historical workings were digitized as outlines and then extruded to build wireframe solids. As well, additional stopes were digitized from the historical longitudinal section and added to list of historical workings. The accuracy and completeness of the outlines of the underground excavations is unknown and, thus, only estimates of the actual mined volumes are possible.

A total of 183 wireframes of the underground workings have been digitized, which gives a volume of approximately 4.8 million m^3 or approximately 13.7 Mt assuming an average density of 2.86 t/m^3. These wireframes were used to remove the percentage of a block that had been mined. A block completely mined is coded as 0% rock, while a block not mined is left coded as 100% rock.

Figure 14.32 is an oblique view of the underground workings solids current to January 1, 2015. The primary areas of mining include the BW, Mexicana on the northwest of the open pit and the Glory Hole and Cabrestante to the southeast of the open pit.

Figure 14.32 Oblique View of the Underground Workings



14.7 RESOURCE CLASSIFICATION

The Mineral Resources for the Aranzazu deposit have been classified in accordance with the CIM definitions and standards for Mineral Resources and Mineral Reserves (CIM, 2014). The classification parameters consider the proximity and number of composite data.

Three criteria were broadly used to determine the classification of each block model estimate: the number of drill holes used to make an estimate, the distance from drill hole data, and the minimum and maximum number of composites used to estimate a block.

The classification item (CLASS) in the block model was populated sequentially as follows: (1) the number of drill holes use to make an estimate was initially used to define the Inferred category; all blocks estimated by at least one drill hole were allocated to the Inferred category, and all other blocks were left as unassigned in the CLASS category, (2) all Inferred blocks within DOMAN 201 to 702 were assigned into the Indicated category, and (3) all blocks with IDPAS=3 were assigned into the Measured category.

The field IDPAS is the maximum distance that composite data can be accepted and used in a block estimate. For IDPAS=3 this search distance is approximately ¾ of gamma in the variogram model for copper. IDPAS=2 is the range of the search ellipsoid which roughly defines the Indicated category, and IDPAS=1 is the maximum allowable range, which roughly defines the Inferred category. In addition, there are a set number of minimum and maximum composites to be used in a block estimate in order for a pass number to be valid (Table 14-25).

Table 14-25 Resource Classification Criteria for Domains 100 to 702

Category	Pass No. (IDPAS)	Approximate Distance (m)	Min No. Comps	Max. No. Comps	No. of DH
Measured	3	< 25m	6	30	3
Indicated	2	≥25 and <60	4	20	2
Inferred	1	No limit	4	10	1

The Mineral Resource classification, using the criteria listed in Table 14-25, was coded into the block model and then adjusted so that an isolated lens of Measured or Indicated material could be re-coded as either Indicated or Inferred. Figure 14.33 is an offset plan view showing the Indicated and Inferred solids that were used for re-coding and adjusting the classification item. In addition to adjustments noted above, Measured material has been restricted to Domains 202, 300, 401 and 501, which are the domains that have more closely spaced drill holes. Indicated material has been restricted to Domains 100, 201, 202, 300, 401 and 501, again based on the spacing of drill holes. The Inferred material category has been assigned to block model estimates in Domains 100, 201, 202, 401, 501, 600, 701 and 702, and also for estimates below the 1,750m elevation.

To assess the proximity to, and quantity of, composite data use to make a "typical" block model estimate, Table 14-26 shows the average number of drill holes, the average number of composites, the mean average distance to composites, the mean distance to the nearest composite, and the mean number of informed quadrants.

Table 14-26 Summary Statistics of Block Model Attributes used for Classification

Category	No. of Blks	Mean No. of Holes	Mean No. of Composites	Mean Distance to Composites (m)	Mean Nearest Composite Distance (m)	Mean No. of Informed Quadrants
Measured	26,426	5	9	25.5	14.1	3.1
Indicated	75,125	5	9	38.5	20.6	2.9
Inferred	34,219	4	8	86.1	44.0	2.7
All	135,590	4	9	48.0	25.2	2.9

For block model grade estimates that are classified as Measured, the average block model grade was estimated using nine composites from five holes from three quadrants, with the nearest sample being 14 m away and the average distance of the nine composites used to estimate the block being 25.5 m. Figure 14.34 is a sectional view of GHP showing all three classification categories in the block model and their spatial relationship to each other. Figure 14.35 is a composite longitudinal section with all classification blocks and underground workings projected to a common plane.

To summarize, the Aranzazu Mine Mineral Resources have been classified in accordance with the CIM definitions and standards for Mineral Resources and Mineral Reserves (CIM, 2014), such that:

- Block estimates with drill hole data within a distance of 25m and a minimum of six composites from three drill holes are considered material that has *confirmed geological grade continuity* and can be classified as Measured Mineral Resources.
- Block estimates with drill hole data within a distance of more than 25m but less than 60m and a minimum of four composites from two drill holes have *data spaced closely enough for the geological and grade continuity to be reasonably assumed* and can be classified as Indicated Mineral Resources.
- Block estimates that do not meet the criteria of Measured or Indicated Mineral Resources but have been interpolated and lie inside the mineralized zones are deemed to have sufficient drill hole *data for geological and grade continuity to be reasonably assumed, but not verified*, and are classified as an Inferred Mineral Resource.

Mineral Resources must also meet a requirement for *"reasonable prospects of economic extraction"*. The Aranzazu deposit has been mined as recently as 2015, with production coming from the BW and Mexicana areas. To meet the criterion of potential economic extraction, block model estimates were viewed in plan and section to ensure that all estimates above the $45/t NSR cut-off form a continuous mass that is reasonably close to the historical underground workings.

Figure 14.33 Plan View of the Indicated and Inferred Solids used for Adjusting the Resource Classification

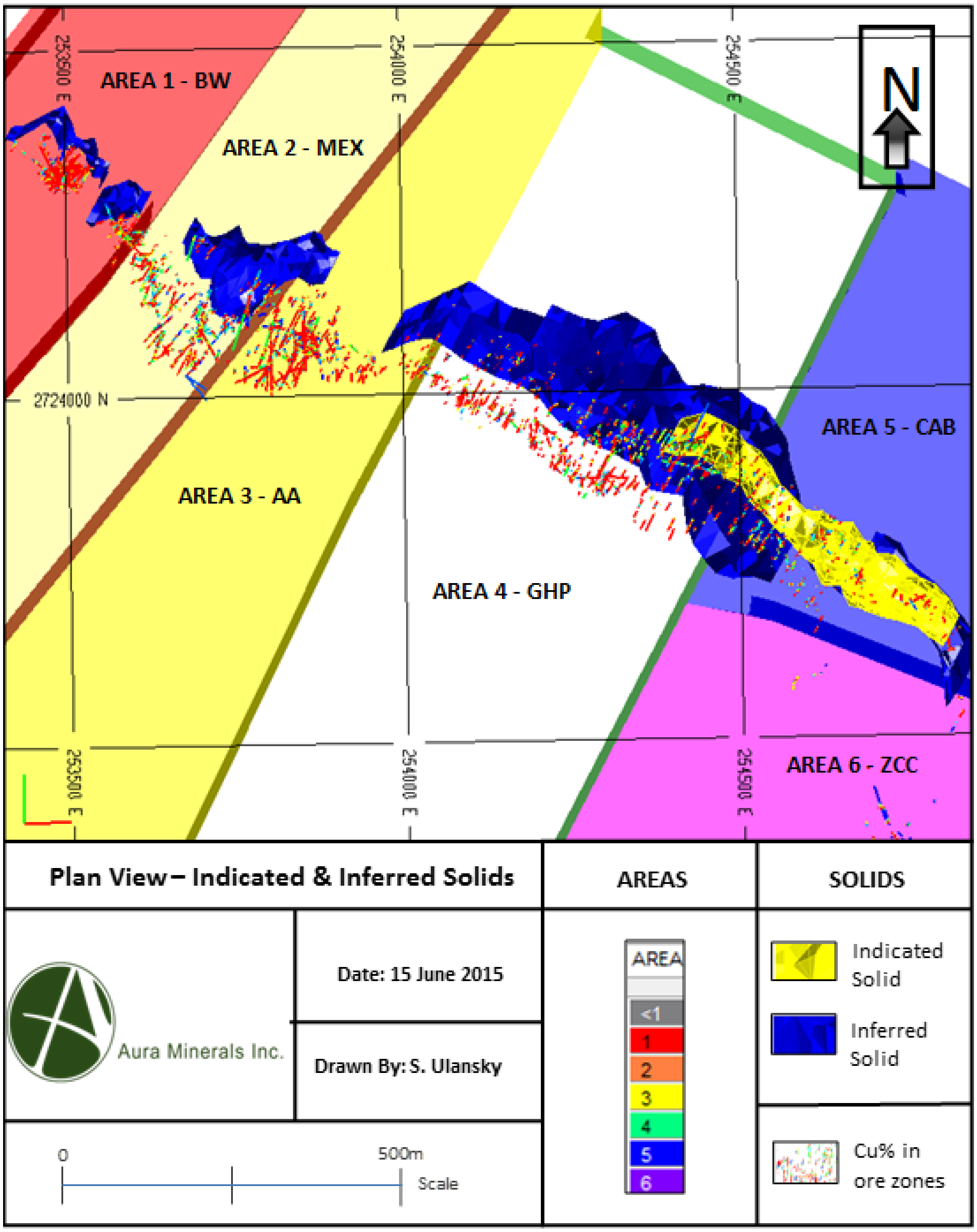


Figure 14.34 Sectional View of GHP Showing the Three Classification Categories

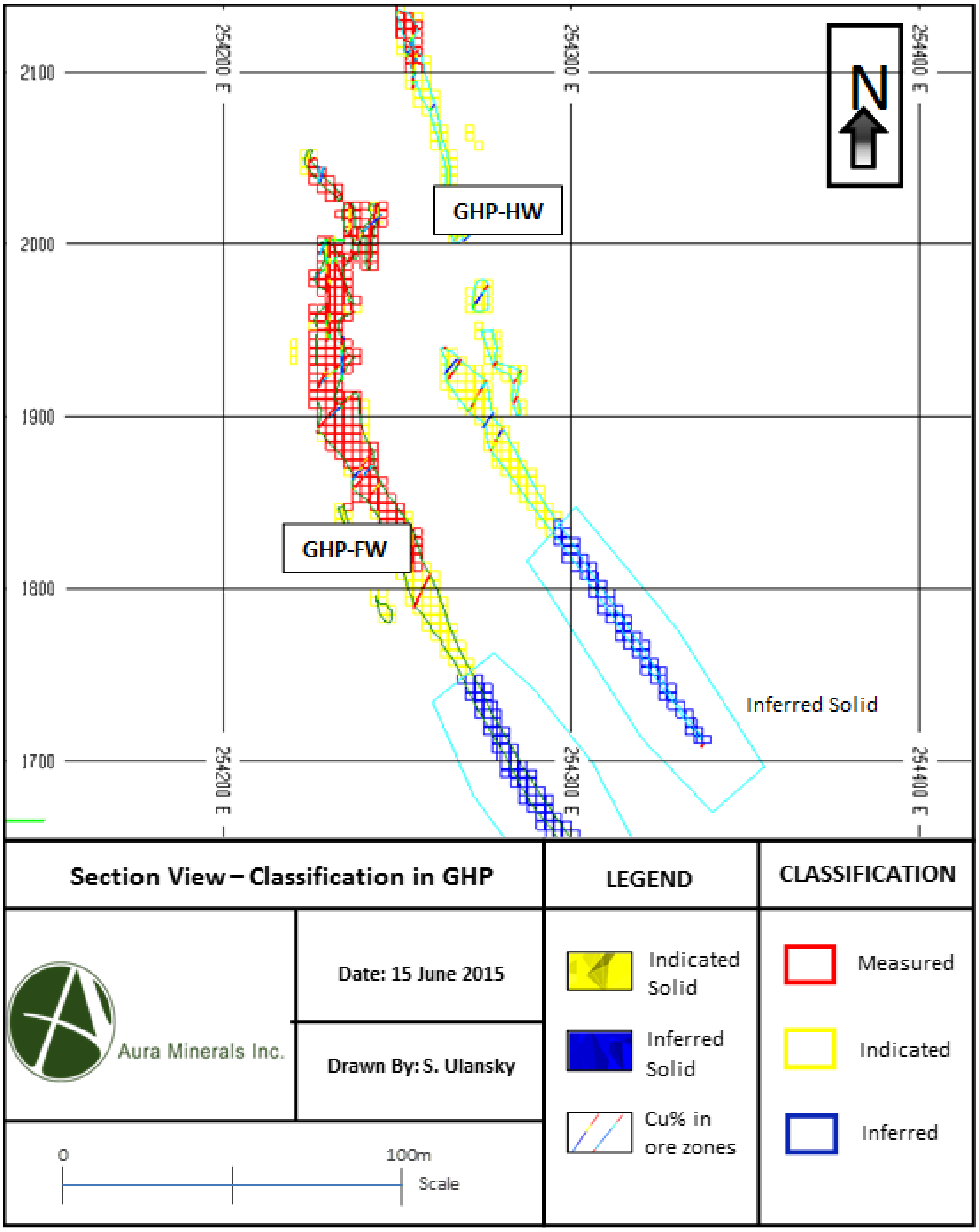
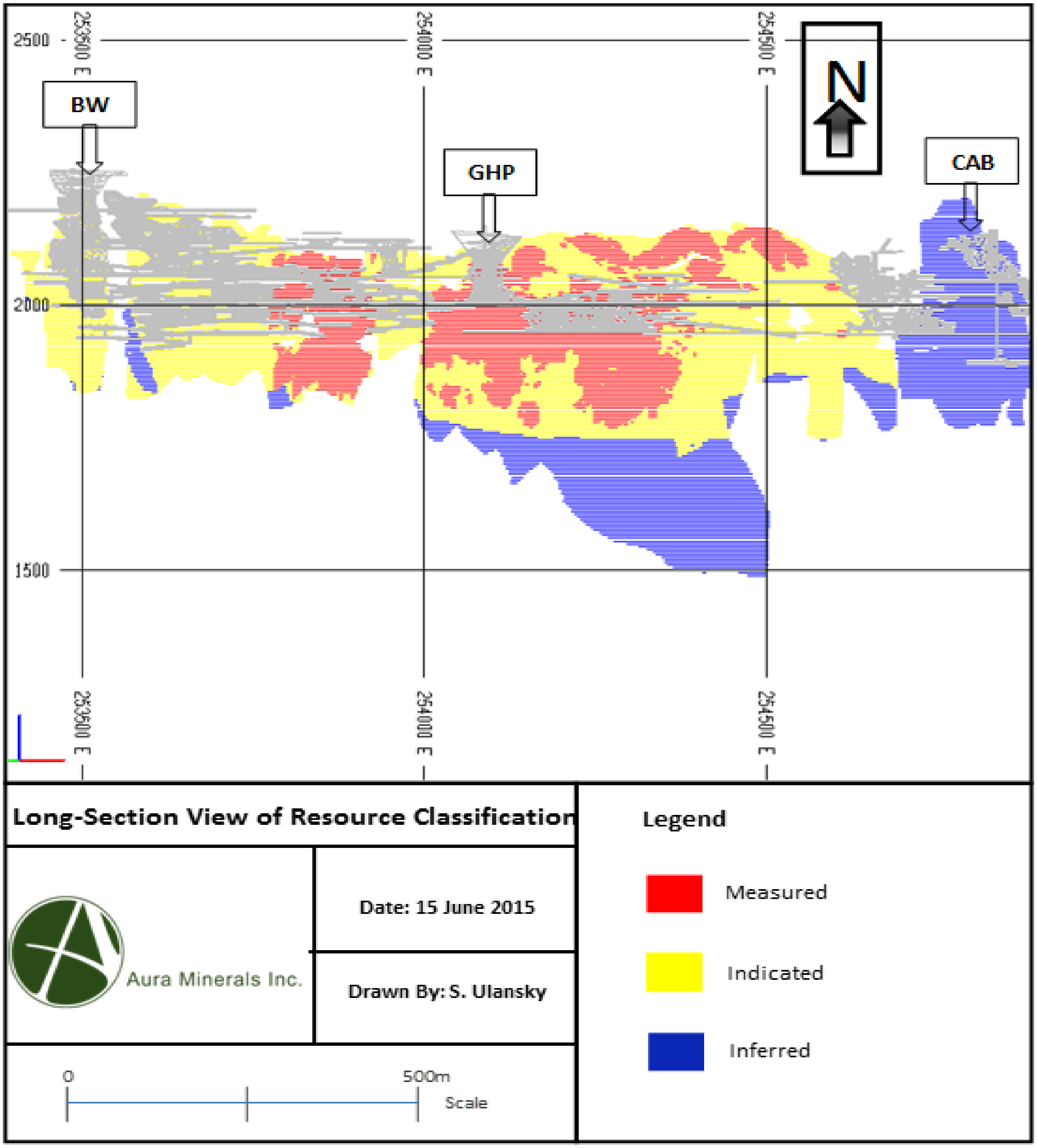


Figure 14.35 Longitudinal View Looking Northeast Displaying the Resource Classification



14.8 MINERAL RESOURCE ESTIMATE SUMMARY

The Mineral Resource estimate for the Aranzazu Mine includes estimates for copper, gold and silver. While estimates have been made for arsenic this estimate is based on a limited quantity of assay data and as such are not considered reliable enough to report as part of the Aranzazu Mineral Resources. The Mineral Resource estimates have been adjusted to account for material mined by the historical workings, and the current Aura Minerals underground and open-pit mine workings as of end of the year (EOY) of 2015.

Table 14-27 presents the March 2015 Mineral Resource estimate for sulphide mineralization summarized by category at a $45/t NSR cut-off. The Aranzazu Measured and Indicated Mineral Resources for sulphide material as reported herein for the March 2015 estimate are 12.02 Mt grading 1.65% copper, 1.10 g/t gold and 20.25 g/t silver at a $45/t NSR cut-off. The estimates were derived using a combination of ore%, underground weight% and topo% so that the resources reflect the total contained metal that remains left to be extracted.

Sensitivity tables and graphs were generated for Measured and Indicated categories combined, and Inferred categories, for both cumulative copper and cumulative NSR cut-off values. Table 14-28 and Figure 14.36 present the sensitivity results for Measured and Indicated categories at cumulative copper cut-off grades. Table 14-29 and Figure 14.37 present the sensitivity results for the Inferred category at cumulative copper cut-off grades. Table 14-30 and Figure 14.38 present the sensitivity results for Measured and Indicated categories at cumulative NSR cut-off grades, and Table 14-31 and Figure 14.39 are the results for the Inferred category at cumulative NSR cut-off grades.

There are no environmental, permitting, legal, title, taxation, socio-economic, marketing or political issues exist which would adversely affect the Mineral Resources estimated above. The Mineral Resource estimate does not take into account dilution and mining losses and all blocks within the model are based on assaying or geological parameters and not mining parameters.

Table 14-27 Measured, Indicated and Inferred Mineral Resource Estimate (Copper, Gold and Silver for Sulphide Mineralization)

Category	NSR Cut-off ($/t)	Tonnes (,000s)	Cu (%)	Cu (,000s lbs)	Au (g/t)	Au (,000s oz)	Ag (g/t)	Ag (,000s oz)
Measured	45	3,923	1.71	147,823	1.05	133	17.84	2,250
Indicated	45	8,562	1.57	296,576	1.10	303	20.89	5,750
Measured and Indicated	**45**	**12,485**	**1.61**	**444,399**	**1.08**	**436**	**19.93**	**8,000**

Notes

1. The Mineral Resource estimates were prepared in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by the CIM Council on May 10, 2014, and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, adopted by CIM Council on November 23, 2003, using geostatistical and/or classical methods, plus economic and mining parameters appropriate to the deposit.
2. Mineral Resources stated at a cut-off of US$45/t NSR.

3. NSR values have been calculated using a long-term price forecast for copper (US$3.00/lb), gold (US$1,280/oz.) and silver (US$18/oz)., resulting in the following formula: NSR ($/t) = (Cu% x $39.76) + (Au g/t x $20.95) + (Ag g/t x $0.32).
4. NSR values are based on the proposed concentrate off take-terms dated September 2017 and the 2015 Technical Report metallurgical recoveries of 88% for copper, 59.4% for gold, 70.3% for silver and 80% for arsenic.
5. Mineral Resources are inclusive of Mineral Reserves.
6. The figures only consider material classified as sulphide mineralization.
7. The figures may not add due to rounding of the numbers to reflect that they are estimates.
8. Mineral Resources are effective January 31, 2018.

Table 14-28 Sensitivity Table for Measured and Indicated Categories: Cumulative Copper Cut-off

CU COFG	Volume m³ x 1000	Density (t/ m³)	Tonnage t x 1000	Cu (%)	Au (g/t)	Ag (g/t)	NSR ($/t)	As (ppm)
>= 0.0	17,457.0	2.98	17,457	1.31	0.853	16.33	74.87	1,240
>= 0.05	17,428.0	2.98	17,428	1.31	0.854	16.35	74.99	1,241
>= 0.10	17,356.0	2.98	17,356	1.32	0.858	16.41	75.27	1,245
>= 0.15	17,265.0	2.98	17,265	1.32	0.861	16.48	75.61	1,249
>= 0.20	17,129.0	2.98	17,129	1.33	0.866	16.58	76.11	1,255
>= 0.25	16,943.0	2.98	16,943	1.35	0.871	16.66	76.73	1,263
>= 0.30	16,709.0	2.98	16,709	1.36	0.878	16.76	77.50	1,273
>= 0.35	16,422.0	2.98	16,422	1.38	0.887	16.92	78.45	1,284
>= 0.40	16,064.0	2.98	16,064	1.40	0.898	17.12	79.62	1,297
>= 0.45	15,671.0	2.98	15,671	1.43	0.909	17.34	80.90	1,311
>= 0.50	15,251.0	2.98	15,251	1.45	0.921	17.59	82.27	1,326
>= 0.55	14,799.0	2.98	14,799	1.48	0.933	17.85	83.72	1,342
>= 0.60	14,310.0	2.98	14,310	1.51	0.946	18.14	85.30	1,358
>= 0.65	13,753.0	2.98	13,753	1.55	0.960	18.47	87.13	1,377
>= 0.70	13,230.0	2.98	13,230	1.58	0.974	18.78	88.88	1,395
>= 0.75	12,694.0	2.98	12,694	1.62	0.988	19.11	90.72	1,416
>= 0.80	12,164.0	2.98	12,164	1.65	1.003	19.45	92.59	1,436
>= 0.85	11,625.0	2.98	11,625	1.69	1.019	19.81	94.57	1,459
>= 0.90	11,081.0	2.98	11,081	1.73	1.037	20.19	96.66	1,481
>= 0.95	10,538.0	2.98	10,538	1.77	1.059	20.60	98.88	1,506
>= 1.00	10,001.0	2.98	10,001	1.82	1.081	21.03	101.19	1,532

Figure 14.36 Sensitivity Results for Measured and Indicated Categories (Copper Cut-off)

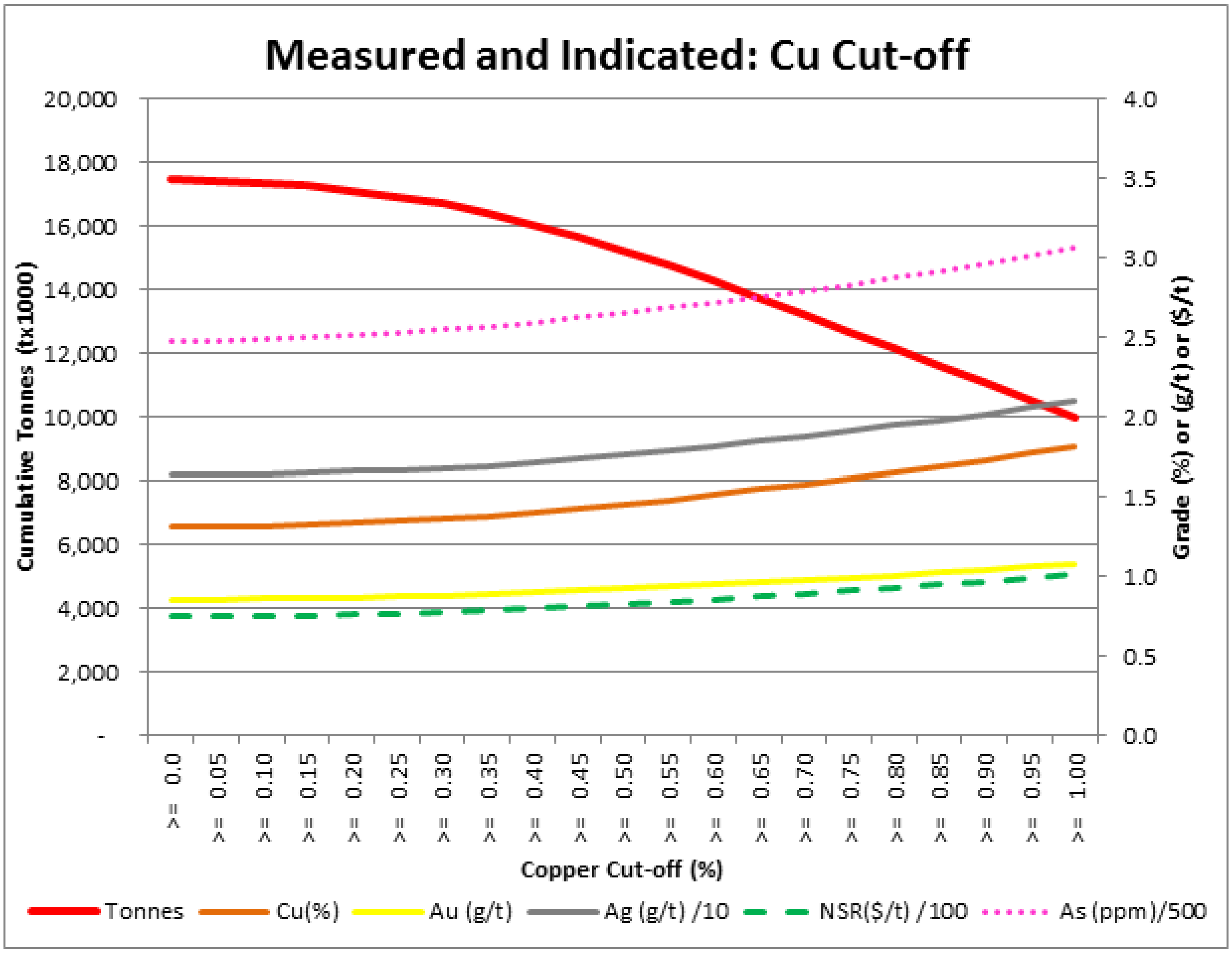


Table 14-29 Sensitivity Table for Inferred Category: Cumulative Copper Cut-off

CU COFG	Volume m^3 x 1000	Density (t/ m^3)	Tonnage t x 1000	Cu (%)	Au (g/t)	Ag (g/t)	NSR ($/t)	As (ppm)
>= 0.0	2,242	2.89	6,467	1.61	1.157	21.40	94.64	1,793
>= 0.05	2,225	2.89	6,419	1.62	1.139	21.42	94.73	1,800
>= 0.10	2,205	2.89	6,363	1.63	1.139	21.53	95.30	1,806
>= 0.15	2,188	2.89	6,316	1.65	1.143	21.65	95.86	1,812
>= 0.20	2,170	2.89	6,266	1.66	1.148	21.77	96.47	1,818
>= 0.25	2,153	2.90	6,216	1.67	1.154	21.85	97.08	1,825
>= 0.30	2,137	2.90	6,173	1.68	1.160	21.92	97.60	1,830
>= 0.35	2,119	2.90	6,123	1.69	1.164	21.95	98.14	1,836
>= 0.40	2,104	2.90	6,080	1.70	1.169	22.01	98.63	1,841
>= 0.45	2,087	2.90	6,033	1.71	1.174	22.08	99.15	1,846
>= 0.50	2,067	2.90	5,978	1.72	1.180	22.17	99.77	1,854
>= 0.55	2,045	2.90	5,916	1.73	1.186	22.24	100.41	1,860
>= 0.60	2,023	2.90	5,852	1.75	1.190	22.29	101.00	1,869
>= 0.65	1,997	2.90	5,780	1.76	1.193	22.37	101.65	1,879
>= 0.70	1,977	2.90	5,722	1.77	1.196	22.42	102.17	1,888
>= 0.75	1,955	2.90	5,660	1.78	1.199	22.48	102.70	1,897
>= 0.80	1,927	2.90	5,580	1.80	1.205	22.57	103.42	1,908
>= 0.85	1,893	2.90	5,483	1.81	1.212	22.72	104.31	1,923
>= 0.90	1,846	2.90	5,349	1.84	1.221	22.85	105.47	1,944
>= 0.95	1,782	2.91	5,167	1.87	1.235	23.08	107.10	1,971
>= 1.00	1,710	2.91	4,960	1.91	1.251	23.34	109.00	2,006

Figure 14.37 Sensitivity Results for the Inferred Category (Copper Cut-off)

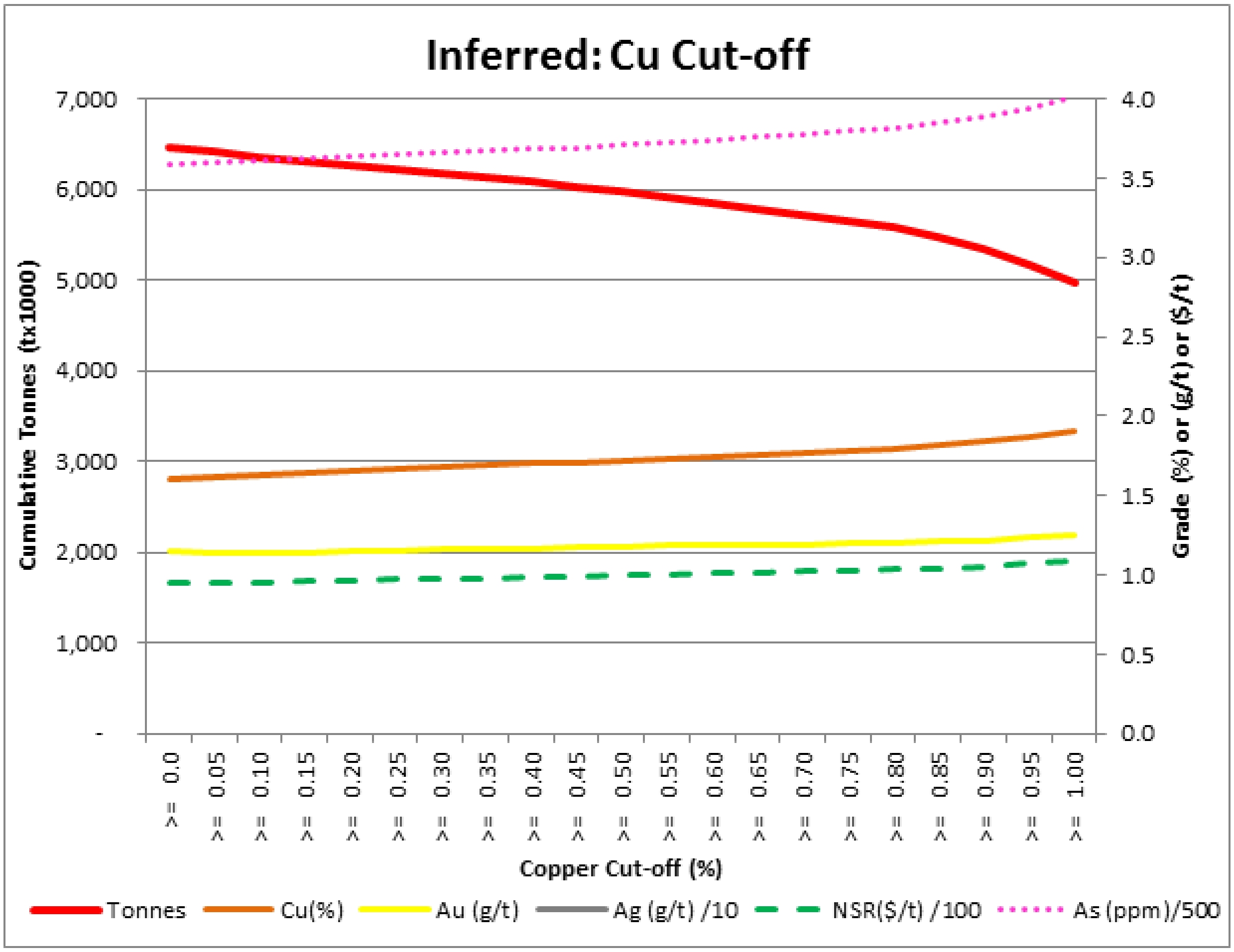


Table 14-30 Sensitivity Table for Measured and Indicated Categories: Cumulative NSR Cut-off

NSR COFG	Volume m^3 x 1000	Density (t/ m^3)	Tonnage t x 1000	Cu (%)	Au (g/t)	Ag (g/t)	NSR ($/t)	As (ppm)
>= 0	5,871	2.98	17,457	1.31	0.853	16.33	74.87	1,240
>= 5	5,844	2.98	17,374	1.32	0.857	16.40	75.21	1,244
>= 10	5,788	2.98	17,206	1.33	0.865	16.54	75.87	1,252
>= 15	5,692	2.98	16,920	1.35	0.878	16.77	76.94	1,265
>= 20	5,541	2.98	16,464	1.37	0.897	17.09	78.58	1,283
>= 25	5,342	2.98	15,864	1.41	0.923	17.52	80.69	1,305
>= 30	5,103	2.97	15,145	1.45	0.955	18.02	83.21	1,331
>= 35	4,831	2.97	14,326	1.50	0.993	18.60	86.11	1,362
>= 40	4,538	2.97	13,445	1.56	1.036	19.23	89.30	1,398
>= 45	4,216	2.97	12,485	1.61	1.084	19.93	92.89	1,440
>= 50	3,891	2.97	11,516	1.68	1.136	20.69	96.71	1,485
>= 55	3,566	2.97	10,551	1.74	1.191	21.47	100.75	1,528
>= 60	3,229	2.97	9,554	1.82	1.250	22.25	105.27	1,570
>= 65	2,904	2.97	8,592	1.90	1.314	23.07	110.06	1,608
>= 70	2,608	2.97	7,720	1.98	1.377	23.86	114.87	1,648
>= 75	2,327	2.97	6,888	2.07	1.441	24.82	119.99	1,681
>= 80	2,073	2.97	6,133	2.16	1.509	25.83	125.22	1,718
>= 85	1,844	2.97	5,455	2.25	1.580	26.84	130.54	1,763
>= 90	1,636	2.96	4,840	2.34	1.653	27.91	136.02	1,794
>= 95	1,453	2.96	4,298	2.43	1.723	29.01	141.51	1,825
>= 100	1,286	2.9627	3,801	2.53	1.797	30.18	147.28	1,851

Figure 14.38 Sensitivity Results for the Measured and Indicated Categories (NSR Cut-off)

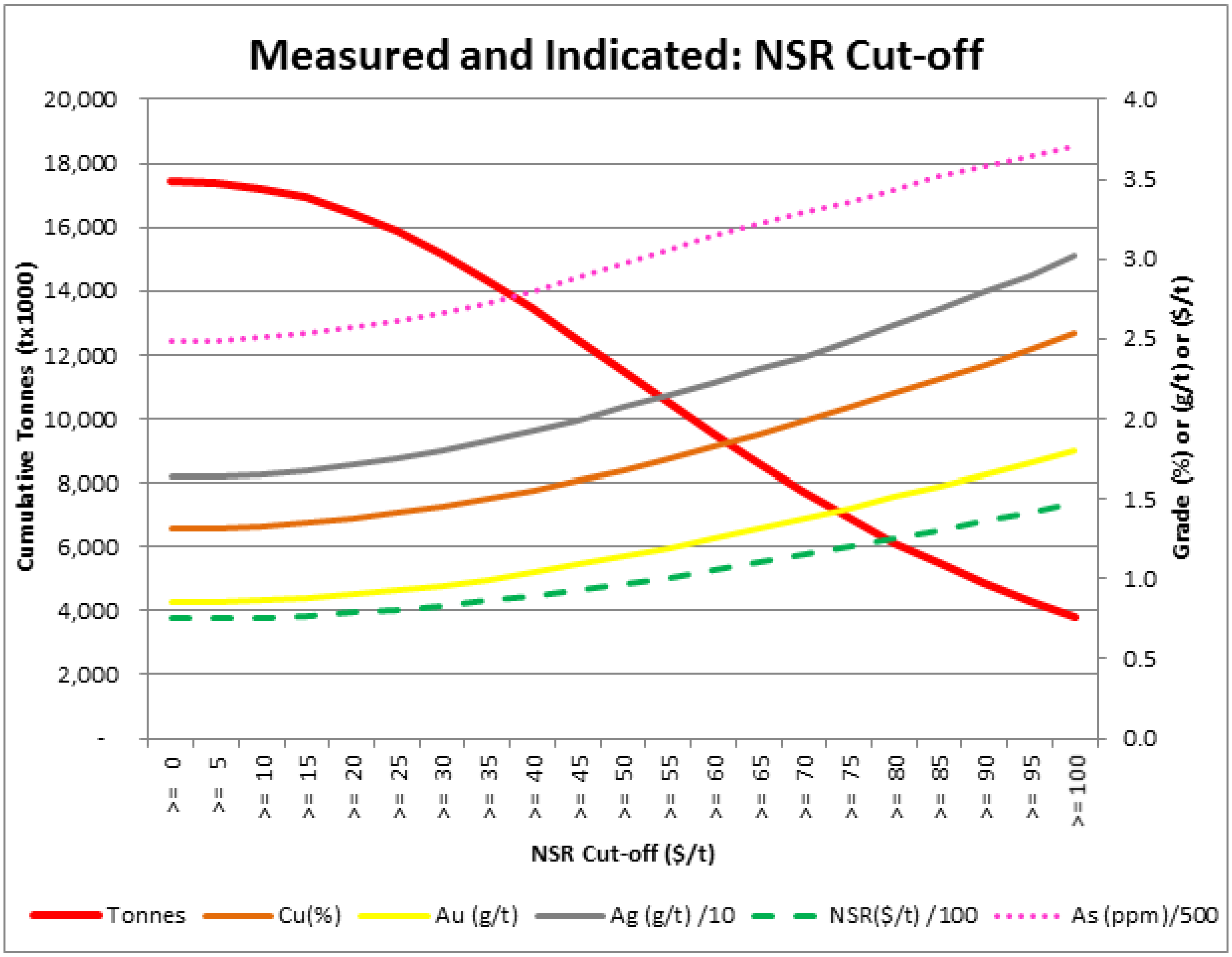
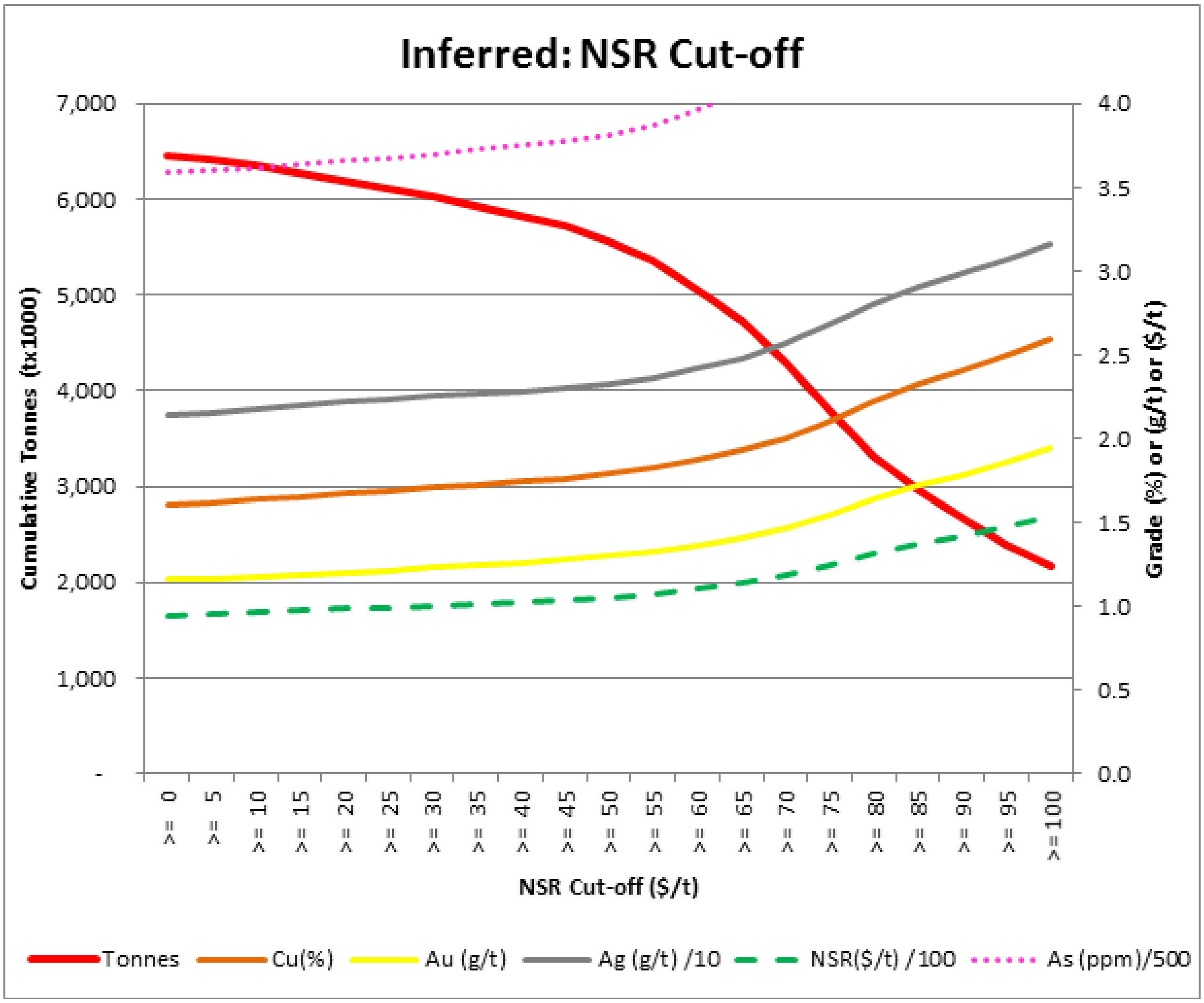


Table 14-31 Sensitivity Table for the Inferred Category: Cumulative NSR Cut-off

NSR COFG	Volume m^3 x 1000	Density ($t/\ m^3$)	Tonnage t x 1000	Cu (%)	Au (g/t)	Ag (g/t)	NSR ($/t)	As (ppm)
>= 0	2,242	2.89	6,467	1.61	1.157	21.40	94.64	1,793
>= 5	2,227	2.89	6,425	1.62	1.165	21.53	95.23	1,799
>= 10	2,203	2.89	6,357	1.64	1.176	21.73	96.17	1,807
>= 15	2,174	2.89	6,274	1.65	1.190	21.95	97.27	1,820
>= 20	2,144	2.89	6,192	1.67	1.203	22.16	98.34	1,829
>= 25	2,117	2.90	6,114	1.69	1.215	22.35	99.29	1,838
>= 30	2,084	2.90	6,023	1.71	1.230	22.54	100.38	1,849
>= 35	2,049	2.90	5,926	1.73	1.245	22.69	101.49	1,864
>= 40	2,015	2.90	5,828	1.74	1.260	22.83	102.57	1,879
>= 45	1,979	2.90	5,727	1.76	1.276	23.00	103.62	1,890
>= 50	1,923	2.90	5,568	1.79	1.299	23.28	105.22	1,908
>= 55	1,849	2.91	5,361	1.82	1.326	23.64	107.25	1,934
>= 60	1,742	2.91	5,058	1.87	1.365	24.12	110.22	1,981
>= 65	1,626	2.91	4,729	1.93	1.408	24.75	113.55	2,032
>= 70	1,477	2.92	4,300	2.00	1.469	25.71	118.13	2,090
>= 75	1,304	2.92	3,803	2.10	1.550	26.84	124.09	2,149
>= 80	1,135	2.93	3,315	2.22	1.637	28.06	130.97	2,201
>= 85	1,013	2.93	2,957	2.32	1.716	29.01	136.85	2,267
>= 90	916	2.93	2,675	2.41	1.784	29.87	142.05	2,322
>= 95	820	2.93	2,395	2.50	1.865	30.64	147.86	2,394
>= 100	742	2.92	2,167	2.59	1.945	31.60	153.17	2,419

Figure 14.39 Sensitivity Results for the Inferred Category (NSR Cut-off)



14.9 ARANZAZU NSR CUT-OFF VALUE CALCULATION

To determine the NSR value of a given ore block within the model, a set of coefficients has been developed to convert the metal grades of the copper (Cu%), gold (Au g/t) and silver (Ag g/t) within a block into the value of the produced concentrate. The coefficient-model considers the commodity prices, the metallurgical recovery and the terms of the concentrate offtake agreement.

The commodity prices used in the NSR value estimation are based on long term price forecast for copper (3.00 US $/Lb.), gold (1280 US$/oz.) and silver (18 US$/oz). The metallurgical recoveries (defined by the 2015 Technical Report) used are 88% for copper, 59.4% for gold, 70.3% for silver and 80% for arsenic.

The concentrate off-take terms considered all treatment and refining charges, penalties and allowances and transportation costs. Details of the coefficient-model parameters which were incorporated in the

NSR formula is shown in table 16-11 of this report. The coefficient equation used for determination of Block NSR value Is shown below.

NSR ($/t) = (Cu% x $39.76) + (Au g/t x $20.95) + (Ag g/t x $0.32).

An NSR cut-off of $60/tonne was used for estimation of the Mineral Reserves and is based on the determination of the total operating cost per tonne of ore (including mining, processing and general administration costs as shown in Table 16.10 of the Technical Report). For the estimation of the Mineral Resources, an NSR cut-off of $45/tonne was selected to ensure the Mineral Reserve would be fully contained within the Mineral Resources and to approximate the extents of the lower grade mineralization.

15. MINERAL RESERVES

The Mineral Reserve estimate presented in this Technical Report has been prepared in compliance with the "CIM Standards on Mineral Resources and Reserves – Definitions and Guidelines" as referred to in NI 43-101 and Form 43-101F and in force as of the effective date of this Technical Report, which is January 31, 2018
The Qualified Person accepting the professional responsibility for the Mineral Reserve estimates section is Mr. Adam Wheeler, C.Eng. Mr. Wheeler was involved in the initial underground mine planning work and subsequently provided support to the mine planning work that was performed by Stantec engineers under the direction of Colin Connors, Director of Mining for Aura Minerals.
Mr. Wheeler has reviewed the mine planning work that forms the basis of this Mineral Reserve statement and acknowledges the validity of the work.

15.1 MINERAL RESERVE ESTIMATE METHODOLOGY

The following summarizes the main steps in calculating the Minerals Reserves. Details of the mine design and mine plan development are provided in Item 16. which also provides further detail on the NSR, cut-off, stope optimization and dilution and recovery.

15.1.1 Determination of NSR Value

In order to determine the NSR value of a given ore block within the model, a set of coefficients was developed to convert the value of the copper (Cu%), gold (Au g/t) and silver (Ag g/t) grades within a block into the value of the produced concentrate. This set of coefficients was based on:

- October 2017 proposed concentrate off-take terms.
- Process recoveries of 88.0% for copper, 59.4% for gold, 70.3% for silver and 80.0% for arsenic.
- Copper price of US$3.00 per pound, gold price of US$1,280 per ounces and silver price of US$18.00 per ounce.

The coefficients used to calculate the NSR value were 39.76 for copper, 20.95 for gold and 0.32 for silver.

15.1.2 Cut-Off Grade

The NSR cut-off of US$60 per tonne of ore was based on the estimated site operating costs for contract mining, owner's costs for mining, processing and site administration costs. Total operating costs were calculated at US$ 59.08 as shown in Table 15-1.

Table 15-1 Summary of Operating Costs used for Cut-off Grade (US$/t)

Revised 2017 estimate			Notes
Contractor Mining Cost	$	36.65	includes labour, materials, equipment
Salaries	$	1.58	Aranzazu staff
Major Consumables	$	2.15	explosives, power, maintenance suppplies
Services	$	0.50	outside services
Total Mine	$	40.89	
Plant	$	11.22	Labor & Salaries, Maintenance, Power, Reagents, Services
G&A	$	6.97	Salaries site services, sales & tax, fees, insurance
Total	**$**	**59.08**	

15.1.3 Stope Optimization and Design

- The stope designs targeted only Measured and Indicated Mineral Resources, but where Inferred Mineral Resources were included within mining shapes they were treated as waste with zero metal grades.
- Stope dimensions were established by a geotechnical assessment performed by CNI in 2017.
- Stope shapes were generated using DataMine's Mine Shape Optimizer (MSO) based on the above defined parameters and the US$60 per tonne cut-off grade. Final stope shapes and associated ore and waste development were designed using the Deswik CAD software.
- Remote stopes were evaluated against the development requirement to test the economics and areas where revenue did not cover development and operating costs were removed.

15.1.4 Dilution and Recovery

- Dilution was applied in the in the form of planned and unplanned dilution from hanging wall and footwall end-wall. Dilution from backfill (for secondary stopes) was also included. All dilution material was assumed at zero grades. Total dilution is approximately 15%.
- A mining recovery of 94% (i.e. 6% losses) and 99% (i.e. 1% losses) to the stopes and ore development sill cuts respectively.
- Final diluted stopes were evaluated against the US$60 NSR cut-off to confirm inclusion in the Mineral Reserve inventory. Stopes falling below the cut-off were removed from the mine plan along with associated waste development.

15.1.5 Mineral Reserves Summary

Table 15-2 Proven and Probable Mineral Reserve Estimate for Aranzazu

Category	NSR Cut-off ($/t)	Tonnes (,000)	Cu (%)	Cu (,000 lbs)	Au (g/t)	Au (,000 oz)	Ag (g/t)	Ag (,0000 oz)
Proven	60	1,872	1.70	69,973	1.08	64	18.3	1,100
Probable	60	2,770	1.74	106,439	1.23	110	19.9	1,771
Proven and Probable	**60**	**4,642**	**1.72**	**176,412**	**1.17**	**174**	**19.2**	**2,872**

Notes

1. The Mineral Reserve estimates were prepared in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by the CIM Council on May 10, 2014, and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, adopted by CIM Council on November 23, 2003, using geostatistical and/or classical methods, plus economic and mining parameters appropriate to the deposit.
2. Mineral Reserves are the economic portion of the Measured and Indicated Mineral Resources. Mineral Reserve estimates include mining dilution and mining recovery. Mining dilution and recovery factors vary with specific reserve sources and are influenced by several factors including deposit type, deposit shape and mining methods.
3. The NSR cut-off US$60/t is based on the total predicted operating cost.
4. NSR values have been calculated using a long-term price forecast for copper (3.00 US$/lb), gold (1,280 US$/oz.) and silver (18 US$/oz)., resulting in the following formula: NSR ($/t) = (Cu% x $39.76) + (Au g/t x $20.95) + (Ag g/t x $0.32). NSR values are based on the proposed concentrate off take-terms (dated September 2017) and the 2015 Technical Report metallurgical recoveries of 88.0% for copper, 59.4% for gold, 70.3 % for silver and 80.0% for arsenic.
5. The stope designs targeted only Measured and Indicated Mineral Resources, but where Inferred Mineral Resources were included within mining shapes they were treated as waste with zero grade.
6. Stope dimensions were established by a geotechnical assessment performed by Call & Nicholas, Inc. in 2017.
7. Dilution was applied in the in the form of planned and unplanned dilution from hanging wall and footwall end-wall. Dilution from backfill (for secondary stopes) was also included. All dilution material was assumed at zero grades. Total dilution is approximately 15%.
8. Mining recoveries of 94% (i.e. 6% losses) and 99% (1% losses) were applied to the stopes and ore development sill cuts respectively.
9. Mineral Reserves are effective January 31, 2018.

16. MINING METHOD

The Aranzazu underground mine is accessed by two main ramps located within the AA and Security open pits. The ramps access the upper elevations of the Mexicana, BW and AA zones. Each ramp is approximately 5m high x 5m wide at a gradient of approximately 15%. There are several old shafts that are remaining from historic mining activity however these are not being considered for use in the re-start plan. The feasibility design envisions extending the ramp development to approximately the 1750 elevation to extract the Mineral Reserve.

Underground mine production considers a daily production rate of 2,597 TPD using a combination of longitudinal and transverse long hole mining methods for the majority of the deposit. The principal method of mining chosen is long hole open stoping (LHOS) with delayed backfill. The majority of the stopes will be mined transverse primary and secondary configuration. primary stopes will be filled with cemented rock fills (CRF) and the secondary stopes with uncemented waste rock. The stopes in the LOM plan are designed at both 15m wide and 10m wide with sub-level intervals ranging between 20 - 30m in an effort to reduce the amount of waste development on the sub-levels. In areas where the orebody width was deemed too narrow for effective transverse stopes (<10m), longitudinal mining is planned.

Long hole open stopping requires development of both a top and bottom cut excavation to access the stopes. In the case of transverse stoping, these are mined and supported to full stope width. Vertical holes are then drilled from the top horizon to the lower horizon (a length of approximately 20-25 m), loaded with explosives and blasted. At the bottom horizon, the blasted ore is removed using remotely operated Load-Haul-Dump (LHD) units. The LHDs then transfer the broken ore to haul trucks for haulage to the mill. Backfill (cemented or uncemented) will then be dumped into the stope to fill the void and provide a bottom horizon for the stope above.

A geotechnical block model was developed from the drill hole geotechnical data which was used to determine the stope design widths, ground support recommendations, dilution estimates, pillar sizes and backfill strength. The model was developed to estimate the Geotechnical Material Type (GMT) both within the ore zone and the surrounding waste rock. In general, the rock quality of Aranzazu is considered fair to good, however there are some zones of poor to very poor rock mass quality, usually controlled by major fault zones, which have been accounted for in the mine design.

Mineral Reserves have been calculated using an NSR cut-off of US$60 per tonne. The cut-off included the estimated mining costs (both contractor and owner costs), processing costs, and general and administration costs. Costs for the contractor were based on quotations, obtained from reputable local firms, for the expected development, stopping, haulage, and backfilling requirements. Aranzazu processing and G&A costs were based on historical operating cost (i.e. 2013-2014) adjusted for inflation and updated salary ranges.

As reported elsewhere, the majority of the mining reserves are contained in the GHFW zone, thus the main focus of development is in this zone. A 10m sill pillar has been designed to divide this zone into an upper and lower mining block which will facilitate mining of the block independently. The LOM does not contemplate mining these sill pillars, but it may be possible if a strong CRF can be placed in the bottom level stopes.

It is intended to use a mining contractor to do all of the development and stoping, haulage of ore to the mill and placement of backfill. Aranzazu will provide the main services, such as ventilation, de-watering (pumping), compressed air and electrical reticulation

16.1 OPERATIONAL HISTORY

Historical mining activities began in the district as early as 1548. In 1891, the Mazapil Copper Company of Manchester, England began mining and smelting operations that continued through to 1962. From 1962 until 2008, various companies have owned and operated the Aranzazu Mine. After shutting down in 1992 due to low metal prices and a labor dispute, the mining operations were restarted on a limited scale in 2007 by a private Mexican company. Aura Minerals acquired 100% of the Aranzazu Mine (formerly known as the El Cobre project) in June 2008. Production was suspended in January 2009 but restarted on a limited basis in 2010, with commercial production declared effective February 1, 2011. The mine was again closed under care and maintenance in December 2014, due to low copper prices. During 2014, approximately 69% of the mill feed came from the underground mine, thus averaging about 1,800 TPD, with total mill feed averaging about 2,700 TPD.

A summary of reported Aranzazu production is contained in Table 16-1

Table 16-1 Summary of Aranzazu Production (2008 – 2017)

Year	Mill Feed (tonnes)	Cu (%)	Au (g/t)	Au (g/t)	Conc (tonnes)
2008	148,511	0.69	0.25	7.9	3,116
2009 *	-	-	-	-	-
2010	57,211	0.51	-	-	831
2011	632,297	0.90	0.48	12.9	13,455
2012	771,774	0.85	0.50	11.9	20,671
2013	796,413	0.98	0.48	16.2	25,813
2014	861,983	0.88	0.45	14.6	26,294
2015 - 2017*	-	-	-	-	-

** Mine under care and maintenance*

Since 2014, Aranzazu has been subjected to a detailed feasibility study with the emphasis on understanding the geotechnical character of the mine. Call & Nicholas, Inc. (CNI) have completed their studies, including drilling of geotechnical holes focused on the GHFW zone, which in turn has led to the creation of a geotechnical block model. This has improved the understanding of the ground conditions and allowed planning to locate development in the more competent rocks.

Another different practice that has been introduced is to work with NSR values, ensuring that every designed stope makes an operating profit. Formerly %Cu equivalent grades were used which didn't really convey the sense of economic reality.

Although mine stope optimization software has been used to create this LOM, in practice each stope will be individually designed/dimensioned to maximize value.

16.2 GEOTECHNICAL

16.2.1 Geotechnical Setting

The Aranzazu Underground deposit is located in the Sierra Madre Oriental geologic terrain, which is characterized by Jurassic and Cretaceous carbonates and clastic sediments intruded by Laramide-age magmatism and northeastern-oriented compression related folding (Sedlock, 1993). The Aranzazu deposit consists of near vertical northeasterly dipping tabular skarn ore bodies striking approximately 900 m with a width of 40 m, and a height of 300 m. The deposit is divided into 5 primary zones: 1) Glory Hole – Hanging Wall (GHHW), 2) Glory Hole – Footwall (GHFW), 3) Mexicana Sur (MXS), 4) Mexicana Norte (MXN), and 5) BW Zone (BW). Transverse faults have been interpreted by Aura Minerals as offsetting the mining targets.

Main Geotechnical Domains. The geotechnical domains of the Aranzazu deposit are as follows:

- Skarn – Massive sulfide ore body consisting of endoskarn to exoskarn alteration proximal to intrusive bodies.
- Hornfels –Metasomatic clastic rocks with variable amount of carbonates present, often intermixed with porphyritic intrusions and massive sulfide skarns.
- Marble – Skarn altered, thickly bedded Cretaceous age limestones. The Marble is stratigraphically younger than Hornfels rock types and forms the northern-most hanging wall of the deposit.
- Intrusive – Modelled intrusive solid was observed as two (2) separate intrusive occurrences:
    - Granodiorite forming southern-most footwall rocks referred to as Intrusive and,
    - Quartz monzonite porphyry (locally referred to as Porphido Norte) which occurs interior to the intrusive and marble boundaries and referred to as Porphyry in the report.
- Fault Zones - Generally, there are two main structure systems modelled in the Aranzazu deposit. A sub-parallel bedding and a series of transverse, cross-cutting structures.

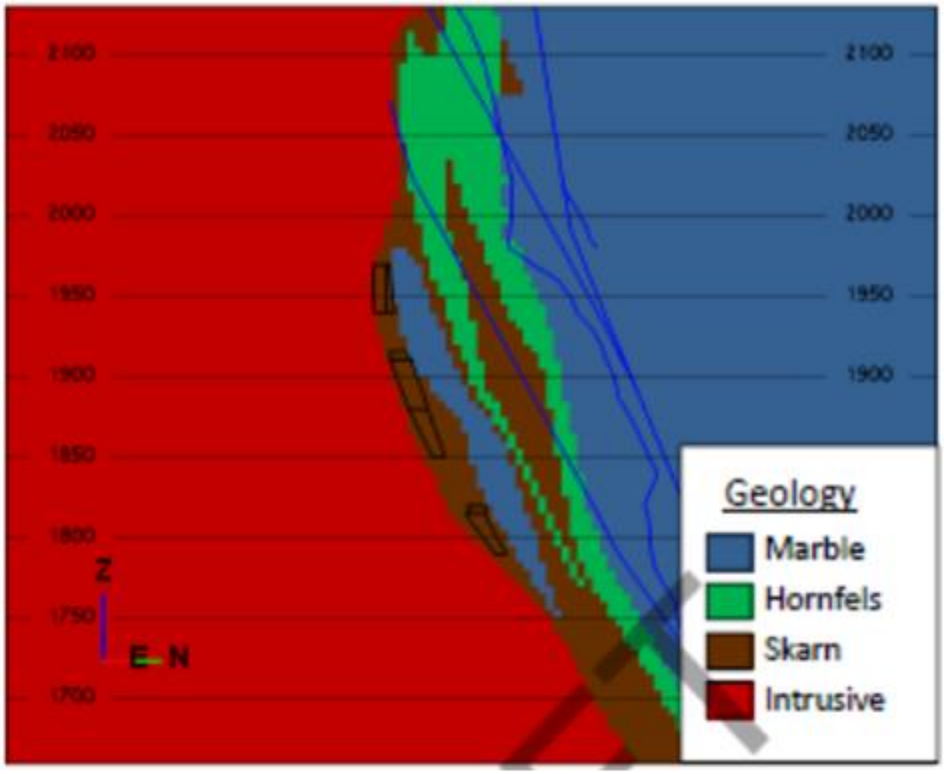
16.2.2 Geotechnical Investigation

Call & Nicholas, Inc. (CNI) performed a feasibility-level geomechanical evaluation during 2017. The purpose of this study was to generate design guidelines for:

1. Stable mining dimensions and ground support recommendations
2. General pillar design and sequencing criteria
3. Backfill requirements
4. Hydrogeological infiltration estimates

CNI's recommendations are based on the results of their stability and ground support analyses which is supported by geotechnical data from historic drill core and underground mapping data.

Figure 16.1 Geological Section Showing Main Geotechnical Domains



16.2.3 Geotechnical Data Collection and Analysis

Between May and July 2017, CNI engineers and geologists conducted a field investigation which included geotechnical core drilling and sample collection and underground inspection to support the geomechanical evaluation.

A geotechnical core drilling program consisting of 7 holes was conducted to obtain geological, geomechanical, and rock fabric data to support the evaluation. Drill hole locations were selected by CNI to target the lithologies within the hanging wall, footwall, and ore zone of the Glory Hole Footwall deposit. Core orientation was performed on 5 of the 7 geotechnical holes to determine true fracture orientations of the drill core to be used for further evaluation.

Geomechanical data has been collected for 410 drill holes totaling approximately 91,000 m throughout the Aranzazu deposit. To validate the entirety of Aranzazu's drill hole database, CNI performed a validation exercise whereby specified intervals of historical drilling were re-logged for the Q' parameters for comparison purposes. The validation exercise indicated that for the purposes of the analyses, there was very little difference in the logging performed by CNI and Aranzazu personnel.

The results of this validation work indicated that while there were some anomalies within the historic database in terms of the accuracy of the recorded data, the database shows a good correlation to observed conditions (based on drill core) and that it is effective in identifying the poor rock zones within the mining area, Consequently, all drill hole data from Aranzazu was accepted and utilized in the evaluation

16.2.4 Rock Strength Determination

Samples collected from the geotechnical core holes were sent for testing at CNI's geomechanics laboratory located in Tucson, Arizona. The purpose of the laboratory testing was to determine strength parameters for use in pillar and excavation stability analyses. Laboratory testing was conducted to ASTM standards and included small-scale direct-shear, uniaxial compression, triaxial compression, and Brazilian disk tension testing. Rock Mass Strengths were estimated from the laboratory test work as presented in Table 16-2

Table 16-2 Results of Laboratory Testing

Aranzazu Mine, Aura Minerals Inc., 2017

Rock Type	Intact Shear Strength			Fracture Shear Strength		Estimated Rock-Mass Shear Strength				
	Uniaxial Compressive Strength (MPa)	Φ (deg)	C (MPa)	Φ (deg)	C (kPa)	Median RQD (%)	Crf	Uniaxial Compressive Strength (MPa)	Φ (deg)	C (MPa)
Skarns	68.9	52.7	11.6	27.4	55.2	80	0.5	10.6	41.8	2.4
Hornfels	53.4	29.5	15.6	23.5	59.4	74	0.5	8.7	26.2	2.7
Intrusive	69.0	42.8	15.1	25.2	44.9	82	0.5	12.4	34.7	3.2
Marble	50.0	44.9	10.4	29.3	24.2	87	0.5	10.5	38.3	2.5

16.2.5 Geotechnical Block Model

Based on the geotechnical data collection and analysis, a geotechnical block model was generated as a tool for predicting rock quality based on Barton's Q'. For each core interval in the database, Q'-values were calculated based on the recorded parameters of RQD, Joint set number (J_n), joint roughness (J_r) and joint alteration (J_a). Q' values were then grouped into six Geomechanical Material Type categories termed (GMT 1) through (GMT 6). The Table below provides the GMT categories and Q' values.

Table 16-3 Description of GMT Categories

Aranzazu Mine, Aura Minerals Inc., 2017

GMT	Description	Q'
1	Intensely Fractured and Altered	< 0.6
2	Intensely Fractured	0.6 - 1.0
3	Highly Fractured	1.0 - 2.0
4	Moderate to Highly Fractured	2.0 - 4.0
5	Widely Spaced Fractured	4.0 - 10.0
6	Blocky	> 10.0

A 3D block model was generated in MineSight® based on model extents and limits provided by Aura Minerals. Rock type wireframe solids provided by Aura Minerals' exploration department were coded to the block model. Block model items were initialized to store the RQD, Jr, Ja, and Jn values, number of composites used, number of holes used, and average distance for all composites used. The spatial distributive nature of Jr, Ja, and Jn were assumed to be similar to RQD. Therefore, CNI defined modelling parameters (i.e. variography, composite selection, etc.) based on RQD.

The modelling process included the following tasks:

- Construct drill hole database, and perform data quality control
- Determine geotechnical domains using visual, statistical, and modelling methods
- Develop estimated Jn values based on RQD correlation
- Determine search parameters from domain variography
- Estimate RQD, Jr, Ja, and Jn for each block
- Generate a Q' block model based on interpolated RQD, Jr, Ja, and Jn values

Two techniques were evaluated for the geotechnical block model estimation: nearest neighbor and inverse distance weighting (IDW). A three-pass estimation technique was used for this study. The initial pass used a search radius equivalent to the variogram range and does not include the fault boundary. The second pass was constrained to within the fault sub-domain. The final pass used an indicator to preserve the low-end RQD distributions within the Fault domain. All estimation passes were restricted by geotechnical domains and required that composites have the same geotechnical domain code as the block being estimated. Each estimation technique was visually and statistically evaluated to determine which method best preserved the drill hole data distribution. The IDW estimation technique raised to the second power was considered the best for the Aranzazu dataset. Components of Q' were estimated in the block model with this methodology, using the search parameters defined by the RQD variography.

16.2.6 Determination of Stope Dimensions

CNI used the Mathews Stability Graph Method to evaluate stope dimensions. This method is an empirical design tool based on case histories from Canadian underground mines, which typically have good to very good quality rock. While the rock quality at Aranzazu is typically of fair to good quality, this approach is still considered appropriate.

The Stability Graph method accounts for the key factors influencing open stope design, including rock mass strength and structure, stresses surrounding the opening, and the shape and orientation of the stope.

The method is based on two calculated factors: *N'* (modified stability number) and *S* (hydraulic radius) which are plotted on a logarithmic-base stability graph. The graph indicates for given N and S values, the potential stability condition of a stope surface.

The stability number (*N'*) is comprised of the following components:

$$N' = Q' * A * B * C$$

Where: *Q'* = Modified Q Tunneling Quality Index
A = Rock stress Factor
B = Joint Orientation Factor
C = Gravity adjustment factor

The hydraulic radius (*S*) is calculated as follows:

$$S = (\text{Area of stope face } (meters^2)) / (\text{perimeter of stope face } (meters))$$

The majority of the mining at Aranzazu is anticipated to take place in the transverse mining direction although it envisioned that longitudinal mining will also take place. Consequently, stope stability will be controlled by the stope width and the stope height, in which the stope end wall is exposed to the hanging wall or footwall rock mass. CNI carried out this analysis for stope heights of both 20 m and 30 m; at each GMT category, using their relative N' values, the side walls and end walls were plotted to the "stable without support line", and the back was plotted to the "stable with support line".

The following tables (16-4 and 16-5) show the results of the Mathew's Stope Stability assessment for the 20 m and 30 m heights respectively.

Table 16-4 Summary of Stope Stability Analysis (20 m height)

Aranzazu Mine, Aura Minerals, Inc., 2017

Q - Prime	GMT	Transverse		Longitudinal	
		Width (m) [1]	Length (m) [2]	Width (m) [1]	Length (m) [2]
< 0.6	1	Not Stope-able (Requires Widths Less Than 5m)		Not Stope-able (Requires Widths Less Than 5m)	
0.6 - 1.0	2	6	7	7	6
1.0 - 2.0	3	8	10	10	8
2.0 - 4.0	4	12	14.5	14.5	12
4.0 - 6.0	5	19	24	24	19
6.0 - 8.0		26	32	32	26
8.0 - 10.0		28	34	34	28
> 10.0	6	28	36	36	28

[1] Dimension Controlled by Stability of the Hanging Wall
[2] Dimension Controlled by Stability of the Side Wall

Table 16-5 Summary of Stope Stability Analysis (30 m height)

Aranzazu Mine, Aura Minerals, Inc., 2017

Q - Prime	GMT	Transverse		Longitudinal	
		Width (m) [1]	Length (m) [2]	Width (m) [1]	Length (m) [2]
< 0.6	1	Not Stope-able (Requires Widths Less Than 5m)		Not Stope-able (Requires Widths Less Than 5m)	
0.6 - 1.0	2	5.5	6.5	6.5	5.5
1.0 - 2.0	3	7	8	8	7
2.0 - 4.0	4	10	11.5	11.5	10
4.0 - 6.0	5	15	17	17	15
6.0 - 8.0		18	20	21.5	18
8.0 - 10.0		21.5	20	26.5	20
> 10.0	6	25	20	31	20

[1] Dimension Controlled by Stability of the Hanging Wall
[2] Dimension Controlled by Stability of the Side Wall

The initial interpretation of the results was used to define the stope design criteria that would provide for maximum stope wall stability and minimize dilution. The maximum stope width allowable (at the different stope heights) was constrained by the GMT value of the hanging wall rock mass and that exceeding this constraint would result in excessive failure of the hanging wall. In order to minimize sub-level waste development, Aura Minerals had committed to implementing the 30 m sub-level interval for the mine design. By applying the GMT constraint on the widths, there was the potential of having stopes of different widths within the same zone. This was seen as difficult to manage operationally and not considered an option. As a result, a width of 10 m was selected for the stope designs. The consequence of this was that at this width, only stopes with hanging wall or footwalls with GMT 4 and above would be considered stable and included in the design; stopes having a hanging wall/footwall rockmass of GMT less than 4 were not considered stable and were not designed. This was seen as having a significant negative impact the reserve inventory because it was automatically excluding stopes in this category regardless of their NSR value.

Aura Minerals proposed an alternative methodology for incorporating the GMT classification into the stope designs. Rather than constrain the design process by the GMT to determine if a stope should be mined, the approach would be to use the GMT value of a stope hanging wall or footwall surface to estimate the amount of overbreak/slough (dilution) that would be expected as dilution. The mine plan would then use the diluted NSR value to determine the viability of mining the stope. The stope width criteria were also evaluated in this context and it was determined that for the GHFW zone, a 15 m width was reasonable for the LHOS mining method.

The following section summarizes the GMT-based dilution assessment.

16.2.7 Stope Overbreak Estimates

CNI have provided overbreak estimates for mining stopes of 10, 12.5, and 15 m wide in GMT categories 2- 6 based on the Equivalent Length of Slough (ELOS) method. The ELOS chart is an extension of the stability graph developed by Mathews, using empirical evidence to estimate the amount of overbreak for different ground conditions at varying hydraulic radii. CNI have delineated 2 types of overbreak that are expected to occur during stope extraction:

1. Equivalent length of estimated overbreak – which is breakage beyond the blast line
2. Equivalent length of additional slough – which is a nominal 50% of the maximum depth of collapse due to poor rock quality

Overall, the mining of stopes at these widths with GMT categories of 2 – 4, will incur some amount of undesirable overbreak due to the insufficient ground quality to maintain stability. GMT values of 5-6, indicating good to very-good rock would result in minimal overbreak.

CNI recommend stopes are designed to anticipate the initial estimated overbreak (i.e. include this estimated distance in the design of the stope blast hole configuration). By accepting this initial overbreak, the total amount of unplanned dilution can be minimized to the only the slough component. This approach will be evaluated by Aura Minerals on an economic basis. However, it is important to note that the total amount of additional slough is difficult to estimate and further dilution from what CNI have provided will be likely in some cases.

Tables 16-6 to 16-8 summarize the results of the overbreak assessment at the different stope widths.

Table 16-6 Over break and Slough Estimates (10 m width)

Aranzazu Mine, Aura Minerals, Inc., 2017

GMT	Q - Prime	Empirical Estimation of Overbreak & Slough For Unsupported Hangingwall		
		Stope Width (m)	*Equivalent Length of Estimated Overbreak (m)	**Equivalent Length of Additional Slough (m)
1	< 0.6	Not Stope-able		
2	0.6 - 1.0	10.0	2.0	1.5
3	1.0 - 2.0	10.0	2.0	0.5
4	2.0 - 4.0	10.0	1.0	< 0.5
5	4.0 - 10.0	10.0	< 0.5	< 0.5
6	> 10.0	10.0	< 0.5	< 0.5

*Breakage Beyond the Blast Line
** Nominal 50% of the Maximum Depth of Collapse

In the case of stopes with GMT value of 1, CNI recommended that stopes not be planned. A rock with GMT of 1 (Q’ < 0.6) is classified as very poor. It was envisioned that a stope wall with this rock mass quality would not merely experience slough as with the other GMT categories but rather major caving which could impact the stability of both the adjacent stopes on either side but the stope above as well. For this reason, stopes with either hanging wall or footwall of GMT <2 have been excluded from the mine plan however there is an opportunity to bring these stopes back into the mine plan by employing alternative stope designs (hanging wall "scab" pillars) to lower the risk of hanging wall caving.

Table 16-7 Overbreak and Slough Estimates (12.5 m width)

Aranzazu Mine, Aura Minerals, Inc., 2017

GMT	Q - Prime	Empirical Estimation of Overbreak & Slough For Unsupported Hangingwall		
		Stope Width (m)	*Equivalent Length of Estimated Overbreak (m)	**Equivalent Length of Additional Slough (m)
1	< 0.6	Not Stope-able		
2	0.6 - 1.0	12.5	2.0	2.0
3	1.0 - 2.0	12.5	2.0	0.5
4	2.0 - 4.0	12.5	1.0	0.5
5	4.0 - 10.0	12.5	< 0.5	< 0.5
6	> 10.0	12.5	< 0.5	< 0.5

*Breakage Beyond the Blast Line
** Nominal 50% of the Maximum Depth of Collapse

Table 16-8 Overbreak and Slough Estimates (15 m width)

Aranzazu Mine, Aura Minerals, Inc., 2017

GMT	Q - Prime	Empirical Estimation of Overbreak & Slough For Unsupported Hangingwall		
		Stope Width (m)	*Equivalent Length of Estimated Overbreak (m)	**Equivalent Length of Additional Slough (m)
1	< 0.6	Not Stope-able		
2	0.6 - 1.0	15.0	2.0	3.0
3	1.0 - 2.0	15.0	2.0	1.0
4	2.0 - 4.0	15.0	1.0	0.5
5	4.0 - 10.0	15.0	< 0.5	< 0.5
6	> 10.0	15.0	< 0.5	< 0.5

*Breakage Beyond the Blast Line
** Nominal 50% of the Maximum Depth of Collapse

A hanging wall "scab" pillar is small pillar (most likely comprised of Skarn ore material) that would remain on the hanging wall. The scab pillar will be of a nominal 2 m thickness, established fully within the rock that is GMT 4 or greater.

16.2.8 Pillar Design Analysis

CNI conducted a pillar assessment based on 3 scenarios:

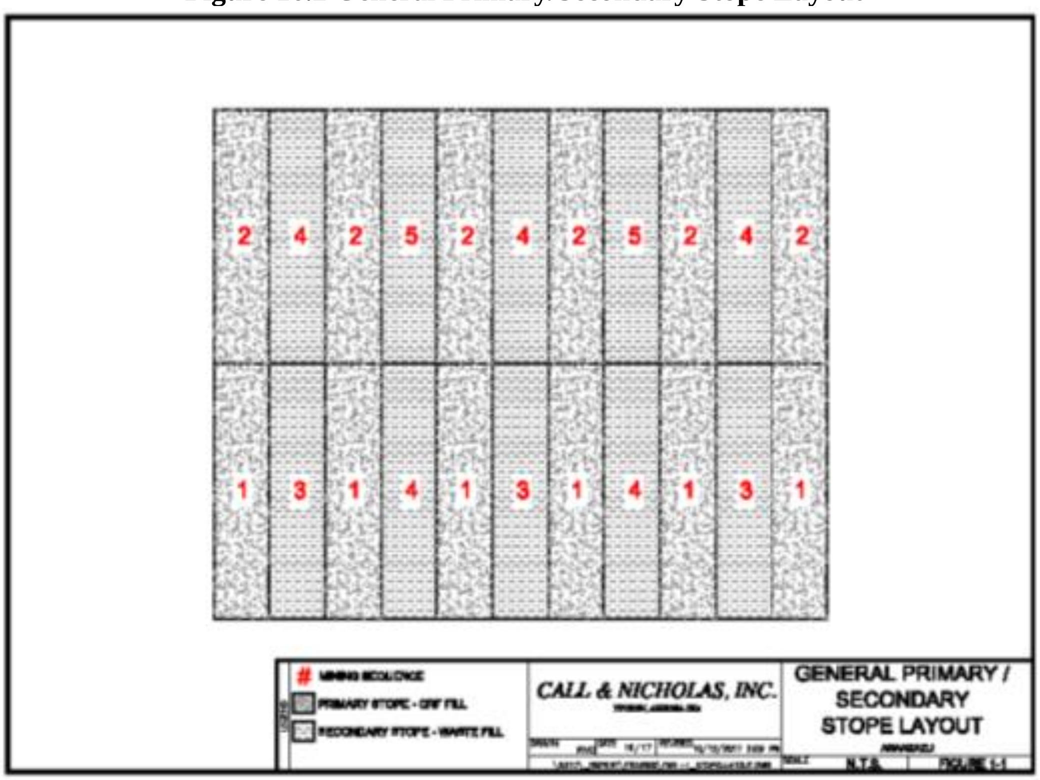
- The rib pillars between open stopes (secondary stopes)
- The access pillars between crosscuts to the stopes from the haulage galleries
- The sill pillar between upper and lower mining blocks in GHFW zone

Rib Pillars:
Aranzazu will be mined using a primary/secondary stoping sequence. As part of this mining method, primary stopes will be mined and then filled with cemented rock backfill (CRF), leaving a full width rock pillar between CRF filled primary stopes. The backfilled stopes will become the side walls of subsequent secondary stopes mined between the primary stopes. The secondary stopes can be filled with uncemented rock or run-of-mine waste (uneconomic rock). Typically, two primary stopes will be mined (and filled) in vertical alignment before a secondary stope is mined between the two lowermost primary stopes.

Using this sequence, rock pillars will be established between primary stopes when two primary stopes are being extracted on the same elevation at the same time. If two open stopes are mined simultaneously with only a single stope pillar between them, this rib pillar must sustain a nominal amount of load which is shed onto it during the mining of the surrounding stopes. Initially, CNI modelled the case of 30 m high pillars with a 10 m width to determine if the pillar would remain stable. The results indicated that the stope rib pillars will remain stable when pillars have an RQD value of more than 55. Based on CNI's ground quality block model, less than 10% of the stopes at Aranzazu have an average RQD of less than 55 indicating that there is low potential for pillar failure.

Aura Minerals will manage this risk though the planning process by trying to avoid this scenario where possible and to conduct ongoing geotechnical evaluation of the stope block during the development and mining process. Figure 16.2 illustrates the recommended primary / secondary stope extraction sequence. Subsequent to the completion of this assessment, Aura Minerals modified the mine design to increase stope widths in the GHFW zone to 15 m which further increases stability of the pillars.

Figure 16.2 General Primary/Secondary Stope Layout



Access Pillars between Stopes and the Haulage Gallery
Stopes will be accessed from the haulage gallery (levels) using cross cuts. Pillars will be left between these cross cuts, the haulage gallery, and the stoping areas. As the stopes are mined out, the vertical stress load will be redistributed to either the footwall or hanging wall. Because these access pillars will be within the footwall where this load will be shed, they should be designed to manage the increased stress load.

Pillar modelling was performed with intrusive (footwall) RQD values ranging between 20 and 90 percent. Pillars were assumed to be 4.5 m in height, 10.5 m wide, and 15 m long (assumes that the haulage gallery will be offset a minimum of 15 m from the stopes). This corresponds to the design configuration where the secondary crosscut is driven from the existing primary stope crosscuts. The results indicate that the access pillars should remain stable when they have RQD values more than 40 whereas the RQD distribution for the intrusive indicates that less than 10% of the rock mass has an RQD less than 40. Aura Minerals plans to add additional support to the pillars in areas where the model indicates low RQD.

The above scenarios were developed when all stopes were 10m wide. With the increase to 15 m wide stopes in the GHFW zone, it was determined the combined primary/secondary stope crosscut design was not practical. In this case, the stope cross cuts will be driven along the centerline of the stope resulting in pillar of approximately 10.5 m wide (based on crosscut of 4.5 m wide) which is still considered stable under the above analysis.

GHFW Zone Sill Pillar
Sill pillars are horizontal pillars often used to isolate mining blocks within the same orebody. By leaving a sill pillar, mining can take place in the upper elevations while development is ongoing to establish mining at the lower elevations. The advantage is that it provides early start to the production before all capital development and infrastructure has been put into place.

The mine design for Aranzazu incorporates a 10 m thick sill pillar in the GHFW. In the west side of the mining zone, the sill pillar is located between the 1870 and 1880 levels and between the 1850 and 1840 elevations on the east side. This split sill pillar design was implemented to accelerate the development and stope mining of the upper west block of the GHFW zone (i.e. stopes could be mining from the 1880 elevation instead of the 1850 elevation as previously designed).

CNI used Carter's crown pillar analysis to evaluate the design thickness of a sill pillar. This method calculates the factor of safety based on the size of the void shape (mined stopes), the Q' value of the rock mass and the sill pillar thickness.

Based on CNI's rock quality model, the sill pillar will be comprised of the following GMT categories:

- GMT 4 – 10%
- GMT 5 – 47%
- GMT 6 – 43%

CNI's evaluations indicate that the 10-meter sill pillar would remain stable when composed of GMT category 6 material. In cases where a substantial amount of the planned sill pillar is less than GMT 6, the 10-meter thickness may still be adequate because the sill pillar evaluation does not consider the additional support provided by the installed ground support. As-built sill pillar thickness less than 10 m is not recommended. Where ground conditions are worse than anticipated, additional support will be required to maintain stability.

16.2.9 Ground Support Recommendations

The following section detail the ground support requirements for both development and production headings. Ground support requirements were evaluated using the ground reinforcement chart based on the tunneling quality index Q' developed by Grimstad and Barton. CNI used the chart as a general guideline to estimate ground support requirements at each GMT category using the lower bounds of Q' from each GMT category.

Table 16-9 provides a summary of the ground support recommendation provided by CNI for all development. Specific conditions for development are provided below.

Main Access Development
The main access development is intended to be permanent infrastructure (open for durations in excess of a year. Therefore, fully grouted resin rebar bolts are recommended over a friction-type bolt. Friction type bolts, such as Swellex or split sets, are susceptible to corrosion in environments which are rich in sulfide mineralization

Stope (Sill-Cut) Development
Span widths in the stopes will range from 10 m to 15 m when developed to the full stope width. Cables bolts have been included in the support recommendations to provide effective long support at these widths. Cable bolt length and spacing have been recommended based on the following rules of thumb:

- Cable bolt length will be half the drift width (10-meter drift width = 5-meter cable length)
- Cable bolt spacing will be half the cable length (5-meter cable length = 2.5-meter cable spacing)

Table 16-9 Ground Support Recommendations

Cat	GMT	Rating	Q	Rock Bolts			Surface Support		Comment
				Length (m)	Type [1]	Spacing (m)	Mesh	Shotcrete	
I	6	Good - Very Good	>10	2.4m	Resin Grouted Rebar	1.8m x 1.8m	None	None	Bolting required in back only (1 row below back line). Spot bolting only on walls.
II	5	Fair	4 > Q > 10	2.4m	Resin Grouted Rebar	1.5m x 1.5m	6-gauge 100mm WW	None	Bolts to within 1.5m above floor. Mesh along back and row below back line
III	3-4	Poor	1 > Q > 4	2.4m	Resin Grouted Rebar	1.25m x 1.25m	6-gauge 100mm WW	None	Bolts and Mesh to within 1.5m above floor.
IV	1-2	Very Poor	Q < 1	2.4m	Resin Grouted Rebar	1.0m x 1.0m	6-gauge 100mm WW	75 – 100 mm	Bolts and mesh to within 1.0m above floor. Shotcrete thickness to completely cover mesh. Additional support req'd in Q' <0.6

Notes:
1. *All permanent excavations (expected life > 1 year) are to be supported with resin-grouted rebar (ramps, haulage galleries). Swellex bolts can be used in non-permanent excavations (expected life < 1 year)*
2. *Bolts and Mesh to within 1.25m above the floor required in lower Q' areas associated with Fault Zone*
3. *In GMT 1 (Q' < 0.6) – installation of lattice girders on 1.5m spacing will be required*
4. *WW – Welded Wire mesh*

Swellex, or friction type bolts, may be used in the stoping headings because these drifts are not expected to be open for a long-term duration before they are mined through.

Shotcrete

Shotcrete has been recommended in development areas which might be open for an extended period of time. The shotcrete should include fibers to provide tensile strength capacity. The thickness of the shotcrete decreases as the ground quality improves. In areas of extremely poor ground (Q' < 0.06 / GMT 1), advance should include in-cycle shotcrete (20 cm thickness), and spilling may be required to pre-support the face.

16.2.10 Backfill Requirement

To achieve nearly full ore recovery of mineable resources, Cemented Rock Fill (CRF) will be used to fill primary stopes following their extraction. These CRF stopes will become the sidewalls and pillars during the subsequent mining of secondary stopes. The strength of the CRF pillar must be adequate to sustain the overburden load of another CRF pillar. The strength of the CRF pillar is a function of the

CRF cohesion and friction angle. A 36 degree friction angle has been assumed for all analysis, which is a standard friction angle for concrete. The cohesion was varied to meet the safety criteria of 1.5. A safety factor of 1.5 was used to account for spatial variation in the backfill quality when dumped into an open stope from a height of 25 m.
Minimum backfill strengths, their corresponding cement contents, and aggregate criteria are detailed below.

- The CRF should achieve a minimum 2.75 MPa (400 psi) unconfined compressive strength (UCS).
    - CNI estimates a 5 percent Portland cement binder requirement.
- The water should be of potable quality.
- The source aggregate will be unaltered and sulfide-free and have a UCS strength greater than or equal to 40 MPa.
- The aggregate should be screened so that the material used is less than 4 inches (10 cm) but not less than 0.375 inches (1.25 cm). To achieve this:
    - First screen the 2 inch (5 cm) passing material
    - Then screen out the 0.5 inch (1.25 cm) passing material

Pillars composed of CRF with the criteria listed above are expected to remain stable provided that no more than one additional CRF-filled primary stope is stacked atop the active CRF pillar. If two CRF-filled stopes are stacked atop the active CRF pillar, a higher-strength CRF mixture of approximately 8 MPa strength (~8-10 percent Portland cement content) would be recommended.

16.3 HYDROGEOLOGY

The purpose of the hydrogeologic analysis presented here is to provide Aura Minerals with estimated pumping requirements for the underground mine. Two separate but related analyses were conducted: (1) a groundwater recharge estimate using precipitation, land cover, topography, and evapotranspiration data, and (2) a mine inflow analysis using calculated groundwater recharge, hydraulic conductivity, hydraulic gradient, and underground mine footprint.

Groundwater recharge was estimated using a surface water balance method in which infiltration was calculated as the difference between precipitation and direct runoff. Infiltration was used as an input to a soil-water balance in the root zone. Groundwater recharge was then calculated as the portion of infiltration beyond the soil storage capacity minus the actual evapotranspiration.

Recent measured volumes of infiltrated water from 2015 to 2017 suggest a current infiltration rate between 10 and 20 liters per second. Estimated infiltration rates based upon regional precipitation and calculated recharge range up to 45 L/s as shown in Figure 16.3. As mining depth increases and localized depressurization continues, the hydraulic gradient relative to the regional water table will increase and may intensify the rate of infiltration. Expected annual variation in precipitation and its effect on infiltration should also be considered. Estimates of these effects are also described in Figure 16.3.

In addition to the steady-state ground water inflows, significant short-lived infiltration events may occur when saturated fracture and fault zones are intercepted. Figure 16.4 outlines a range of estimated peak inflow rates based upon estimated fault zone geometry and hydraulic conductivity.
Specific considerations regarding peak inflows include:

- The range of peak expected flow rates from a fault zone is approximately 5 to 35 L/s.
- The flow duration is estimated to range between 30 and 60 days for a 12-meter-wide fault zone.

Based upon these calculations, CNI recommends the maintenance and installation of pumping system comprised of two pumps, each capable of handling 45-50 L/s capacity to accommodate for:

- Capture of groundwater inflow from the estimated maximum 99th percentile average inflow rate,
- Capture of groundwater inflow from a highly permeable fault for a short duration.
- Maintain redundancy to address pump failure.

Figure 16.3 Estimated Steady State Groundwater Inflow

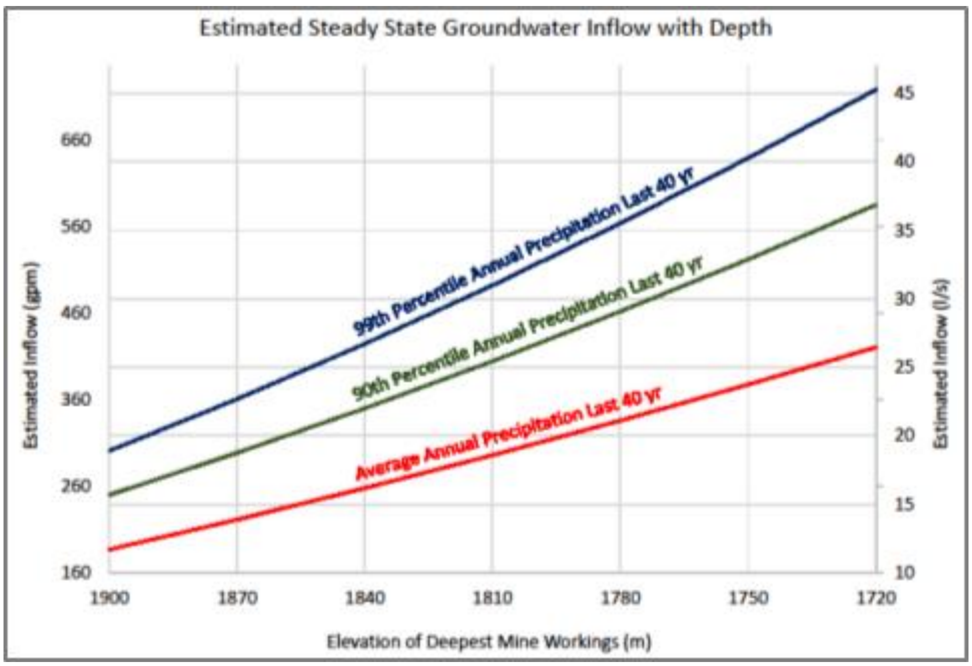
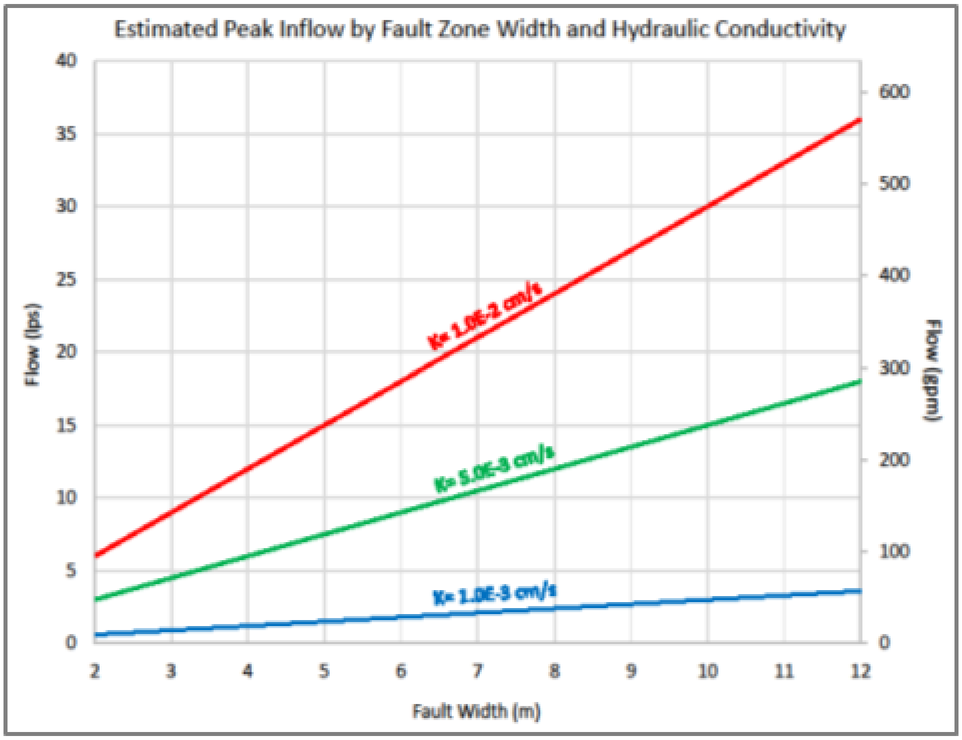


Figure 16.4 Estimated Peak Inflows

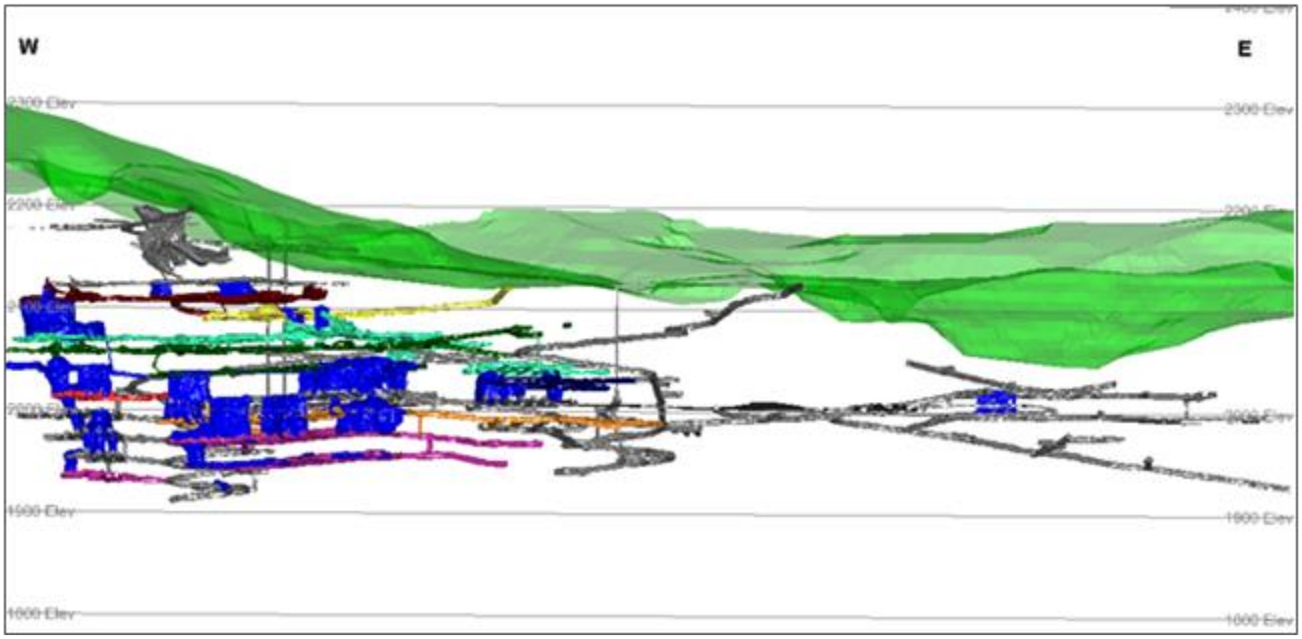


16.4 DESCRIPTION OF MINE

Extensive mining operations have been carried out within the Aranzazu Mine property. There are two main open pits – the western AA pit and the eastern Security pit. The underground workings extend over a strike length of 1980 m, although the principal concentration of stoped areas extends over 800 m. The deepest underground level goes down to 1910mRL, which is at a depth of approximately 330 m below surface. Generally, the existing level interval is 20 m.

The principal mine access is via a portal from the west end of the Security pit and a portal located in the North wall of the AA pit. In addition, there are also three shaft connections used for ventilation purposes.
Figure 16.5 shows a longitudinal section of the mine showing portal access, ventilation raises and main mining zones.

Figure 16.5 Aranzazu Underground Longitudinal View



16.5 MINING METHOD

The Aranzazu mining operation is planned to be accomplished using modern mechanized mining equipment and techniques. As the mine is already accessed through a series of portals and inclined ramps, the development strategy will be to utilize this existing infrastructure to provide primary access to the planned mining areas. The ore zone extends approximately 1,100 m along strike and extends from the 2000mRL to the 1940mRL elevation (a vertical distance of 260 m). The ore zones to be mined are steeply dipping, with thicknesses from 5 m to 30 m.

Mining will be carried out using the Long Hole Open Stope (LHOS) mining method with delayed backfill to extract the ore. Stopes will be extracted in a transverse method (perpendicular to the strike of the orebody) using a primary/secondary stope configuration. In this mining method, main level haulage galleries are established from the main ramps parallel to the ore zones on each production level as defined by the sub-level interval. From the haulage gallery, the ore zone is accessed by crosscuts at intervals specified by the designed stope width. Excavations are made in the ore at the top and the bottom of the designed stope (defined as topcut and bottomcut). Holes are drilled and blasted, and the broken muck is removed from the stope through the bottom access. When the ore has been removed from the stope, the stope is backfilled using cemented rockfill (CRF). The first primary stope will be filled up to the base level of the topcut excavation, allowing the primary stope above to be mined. Following the completion of the primary stopes on either side, the secondary stopes can be extracted (see Figure 16.2)

While the majority of the stopes will be mined transversely, there are some areas that will require the ore to be extracted in a longitudinal approach. This is planned where the orebody is less that 10m wide. In this method, a slot is developed at the opposite end from the access and mining retreats back to the common access. Where two or more stopes occur side-by-side (a maximum of 4), then these stopes may be mined in a progressive sequence. Crosscuts are only required at the extreme ends of these groups, and so both the intermediate primary crosscuts and secondary crosscuts for these areas are not required. The advantages of the longitudinal approach are that they reduce the amount of access development as compared to transverse mining and that uncemented rockfill can be used for most of the stopes which will result in lower mining costs. The disadvantages to this approach are that is not suited to wider parts of the ore zone as it is more restrictive to the number of active stopes and limit production. A further disadvantage is that if unstable ground conditions occur in the hanging wall rock mass (i.e. side walls), there is a higher potential to lose ore reserves and incur additional costs to re-activate the stope.

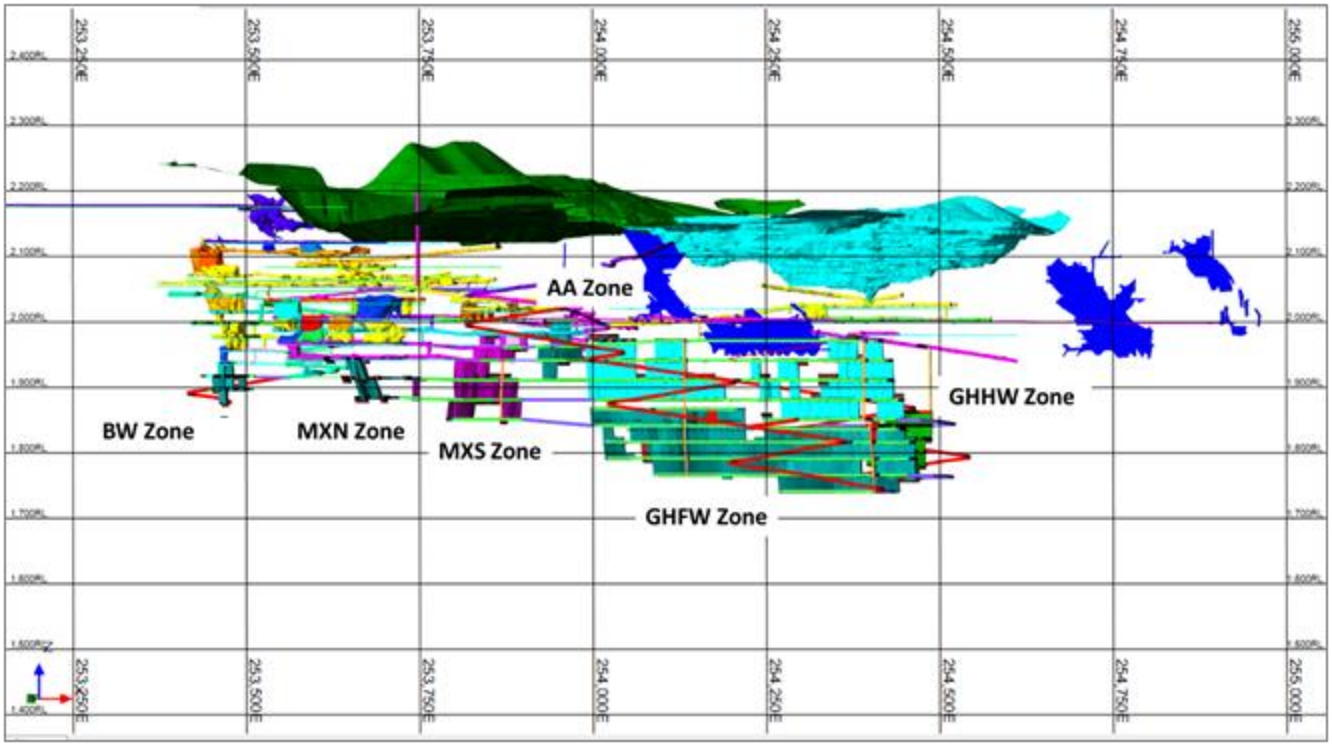
Longitudinal stopes account for less than 10% of the total stopes mined in the LOM plan however there are opportunities to increase this during the operational design phase. In addition to the ore that is planned to be extracted through either the transverse or longitudinal stoping method (including the associated top and bottom sill cuts), ore will also be extracted from benches or backstopes:

- Benching. Benching will occur where there are small patches of economic ore beneath the main planned transverses stopes, that do not justify full stopes being developed. For these areas, a bench cut can be made from the bottom sill cut, and ore extracted down to a maximum depth of 5 m beneath the sill base level.
- Backstopes: Backstopes will be taken where there is small portion of ore above the planned topcut sill of a transverse stope; a small excavation is planned to extract this ore. This will be mined using an uphole drilling and blasting sequence. Because these excavations cannot be backfilled, they can only be extracted on secondary or isolated stope panels where no further mining is planned above.

The Aranzazu deposit is composed of 6 main mining zones as shown in Figure 16.6:

1. Glory Hole Porfido FW (GHFW) Zone
2. Glory Hole Porfido HW (GHHW) Zone
3. Mexicana Sur (MXS) Zone
4. AA Zone
5. Mexican Norte (MXN) Zone
6. BW Zone

Figure 16.6 Underground Mining Zones



16.6 STOPE DESIGN

The following section describes the stope design process used to develop the mine plan. Preliminary stope shapes were generated in the DataMine® software using the Mineable Shape Optimizer (MSO) module. These preliminary stope shapes were defined by geotechnical and geometry constraints using an NSR cut-off grade of US$60 per tonne.

16.6.1 Geotechnical Recommendations

As previously described in Item 16-2, a geotechnical block model was developed by CNI to be used in the stope design process. The key inputs from the geotechnical block model to the stope design process are the maximum stope width and length for given level interval and GMT category, for a stable stope configuration and the estimate of overbreak and additional slough, for different stope dimensions and GMT categories.

The approach taken applying these geotechnical recommendations to the stope design methodology in the current study may be summarized as follows, considering the general overall geotechnical considerations in each zone.

- In GHFW zone, a panel width of 15 m was selected, along with a maximum level interval of 30 m
- In all other zones, a panel width of 10 m was applied
- In the MXN and MXS zones, a level interval of 30 m was applied
- In the GPHW, AA and BW zones, a level interval of 20 m was applied

- For optimization of LHOS stopes in the AA, BW, and MXN zones, the general approach was to strive for stable stopes with minimum overbreak, and consequently in GMT categories 4 or higher
- For optimization of LHOS stopes in the GHFW, GHHW and MXS zones, GMT limitations were not imposed, but in the subsequent design process, and controlled by the GMT categories of each stope's wall material, the amount of resulting dilution was determined from CNI's overbreak tables, and the resultant diluted grade assessed economically

16.6.2 Stope Optimization Process

All LHOS stopes were initial laid out using a stope optimization approach. The DataMine software module Mineable Shape Optimizer (MSO) was used to generate stope outlines. Primarily the optimization runs were focused on producing stope wireframe envelopes of full level height, at 20 m or 30 m depending on the region of the mine. If a full level stope could not be justified, then an additional secondary optimization was made for smaller sub-stopes, down to approximately 5 m in height. This additional secondary pass enabled small bench and overcuts to be laid out beneath and above the main LHOS stope blocks. Stope shapes were generated based on an NSR cut-off of US$60 per tonne which accounted for the mining, process and G&A costs. Table 16-10 summarizes the cost components used in the cut-off grade estimate.

Table 16-10 Summary of Operating Cost

Cost item (2017 Estimate)	**Cost ($US/tonne)**	**Comments**
Mining		
Contractor Mining	$36.65	Direct mining costs per tonne ore
Owner's costs	$4.24	Operations and technical support, power, explosives
Total Mining	**$40.89**	
Total Processing	$11.22	Plant Operating Costs
General and Administration	$6.97	Site management, fees, administration,
Total Operating Cost	**$59.08**	

In order to determine the NSR value of a given ore block within the model, a set of coefficients was developed to convert the value of the copper (Cu%), gold (Au g/t) and silver (Ag g/t) grades within a block into the value of the produced concentrate. This set of coefficients was based on the proposed concentrate purchase agreement available at the time of this study. The coefficient-model considers the commodity prices, smelter charges and penalties and transportation costs. Table 16-11 below summarize the calculation of the NSR coefficients used in the study.

The overall stope optimization process involved the following steps:

1. Import of the geotechnical block model:
2. Flagging of Unmineable Zones due to low GMT value: Based on a level height of 30 m and a required stope (along strike) panel width of 10m, GMT values need to be 4 or

higher, to achieve stable open stopes. Therefore, blocks with GMT values of 1, 2 or 3 (i.e. weaker material) was flagged as unmineable. This procedure was only applied to AA, BW, MXN zones.

3. Combine with Resource Model. The flagged geotechnical data was then combined with the resource block model. Additional areas were flagged as unmineable (25 m crown pillar underneath the open pits, and a 15 m unrecoverable sill pillar underneath previously mined stopes).
4. NSR Update: NSR values were updated from the primary grade values, based on the coefficients summarised in Table 16-11. NSR values inside blocks with an Inferred resource category were set to zero.
5. Zone Orientation: The complete combined block model was then divided into 6 separate block models, according to general orebody strike orientation. This was to facilitate generation of optimised stopes that are broadly oriented logically with each different zone.
6. Stope Optimisation: These prepared block models were then used to complete stope optimisations using the MSO process, using parameters summarised in Table 16-12. Owing to the different local level intervals required, as well as the existence of the different historical levels, reference level controls were applied individually for different zones. The optimisations results included tonnage/grade results by stope, key 3D perimeters for each stope, and wireframe models for each stope.

Table 16-11 Summary of NSR Coefficients

Calculation of Net Smelter Return (NSR) based on Proposed Off-Take Terms
Sep-17

Forecast Metal Prices

Copper Cu	3.00	$/lb
Gold Au	1,280	US $/oz
Silver Ag	18	US $/oz

Metalurgical Balance (based on average LOM grades)

		Assay				Recoveries			
	tonnes	**% Cu**	**g/t Au**	**g/t Ag**	**% As**	**Cu**	**Au**	**Ag**	**As**
Millfeed	100.00	1.73	1.20	20.18	0.16	100.0%	100.0%	100.0%	100.0%
Cu Cons	6.62	23.00	10.73	214.30	1.96	88.0%	59.4%	70.3%	80.0%

Concentrate Ratio *15.11*

Payable Metals

	% payable	Minimum Deduction		Payable Metal		**Revenue /t conc**
Copper Cu	96.50%	1.1	%	0.22	tonnes Cu	$ 1,448
Gold Au	100.00%	1.5	grams	0.30	oz Au	$ 380
Silver Ag	90.00%	50	grams	5.2884	oz Ag	$ 95
TOTAL REVENUE						**$ 1,923**

Treatment Charge	$ 135.00
Freight, Insurance etc	$ 53.82
Refining Charges	$ 69.30
TOTAL CHARGES	**$ 258.13**

Penalties

TOTAL	**$ 155.32**

Net Smelter Return	*per tonne concentrate*	**$ 1,509.86**
	per tonne ore	**$ 99.94**

Metal Factors

% Cu	**Cu factor**	**g/t Au**	**Au factor**	**g/t Ag**	**Ag factor**		**NSR/t ore**
1.73	**39.7573**	1.20	**20.9450**	20.18	**0.3175**		**$ 99.94**

Table 16-12 Stope Optimization Parameters

	Units	AA, BW, MXN	MXS	GHFW	GHHW	Notes
Cut-Off NSR	$US/t	60	60	60	60	
Section Spacing (Stope Length X)	m	10	10	15	10	Panel width
Minimum width (Y)	m	5	5	5	5	Minimum width of longitudinal stopes
Minimum width (Y)	m	100	100	100	100	Complete stopes from f/w to h/w contacts
Level Spacing (Stope Height Z)	m	30, 20	30, 20	25, 30	20	
Stope Slice Interval m	m	5	5	5	5	
Transverse Panel Rotation Angle	°	0-60	0-60	0-60	0-60	Variable By Zone
Waste pillar width minimum m	m	5	5	5	5	Intermittent pillars between different transverse ore portions
Near wall dilution	m	0.5	0.5	0.5	0	
Far wall dilution	m	0	0	0	0.5	
Minimum dip angle	°	45	45	45	45	Stope walls can be inclined +/- 45°
Maximum dip angle	°	135	135	135	135	Stope walls can be inclined +/- 45°
Maximum strike angle	°	20	20	20	20	Max strike angle possible
Maximum strike angle change	°	10	10	10	10	Max strike angle difference between top and bottom edges
Discretisation interval	m	4 x 4	4 x 4	4 x 4	4 x 4	Discretization of intersected blocks for evaluation
Max waste fraction	m	50%	50%	50%	50%	
Max side:length ratio		3	3	3	3	
Sub-Shape Vertical Splitting	m	5	5	5	5	

16.6.3 Detailed Stope Designs

Following completion of the stope optimization process, additional interactive adjustments were made to generate the final set of stope shapes.

- Remote stopes were evaluated against the development requirement to test the economics – areas where revenue did not cover development and operating costs were removed.
- Small sub-stopes that did not fall immediately above or below the level were removed.
- In cases where there were clusters of optimized sub-stopes within the same parent stope panel area, a manual design was carried out to determine if an economic stope could be generated.

The final stopes shapes were evaluated visually to confirm their validity and then passed to the ore development design and scheduling process. Stopes within the AA, BW and MXN zones were designed in DataMine. Stopes from the GHFW, GHHW and MXS zones were designed in the Deswik software. Stope top and bottom sill cuts – were designed at 5m high and the full-width of the stope were then created based on the footprint of the stope at a given elevation. Stope shapes were then cut against the sill cut shape to define the mineable stope and sill cut volumes. In areas where the MSO shapes did not extend above the level elevation, top sill cuts were created manually.

16.6.4 Dilution and Recovery

Dilution has been accounted for in the mine design through the application of several design factors.
Planned Dilution: the stope shapes generated from the MSO process have been adjusted by adding a 0.5 m “skin” to the end walls of the stope. This reflects the planned dilution that would likely be incurred during the operational design phase. Note that for stopes in the MXN, BW and AA zones, this skin was added to both the footwall and hanging wall end-wall. For stopes in the MXS, GHFW and GHHW zones, this skin was added to the footwall end wall only; the dilution for the hanging wall end-wall will be added as part of the GMT-controlled dilution factor.

Unplanned (GMT) Dilution: stopes in the MXS, GHFW and GHHW zones had additional dilution applied in the form of a GMT-controlled dilution factor. For these zones, which were considered to be most impacted by the rock mass quality of the hanging wall, unplanned dilution was considered to occur as "slough" from the hanging wall waste rock and is directly linked to the GMT value of the rock mass along the boundary of the stope. The various GMT-controlled dilution factors have been summarized in Item 16.2 (Stope Overbreak Estimates).

Stope Configuration Dilution: additional dilution from the stope side walls has been introduced depending on the configuration of the stope. Primary stopes are assumed to have zero side-wall dilution due to overbreak/slough coming from the secondary stopes and is considered as ore. Secondary stopes are assumed to have dilution coming for the CRF pillars on either side. For End-of-line and isolated stopes dilution due to overbreak/slough is assumed as waste.

Stope top and bottom cut sill will experience some level of dilution. The factor applied represent the accuracy of the geological model and the effectiveness of the ore control process during development. The assumption is that ore recovery is considered a high priority during the development of the top and bottom sill cuts and that this will translate into a higher dilution factor.

Ore recovery (mining losses) will be impacted by the effectiveness of the drilling and blasting process as well as the mucking process. Ore losses will come in the form of un-blasted material remaining on the end walls and/or blasted ore not be mucked from a stope or sill cut. Typically, in ore development, there is much greater control over these two aspects than in stopes, resulting in a higher recovery factor.

Table 16-13 summarizes the dilution and mining recovery parameters used in the mine planning. All dilution factors (except the planned dilution) are applied within the Deswik scheduling software based on the appropriate criteria of the stope (i.e. GMT of the stope end wall). Following the application of the above described dilution factors, the total estimated dilution in the LOM production plan is 14.6%. All dilution is assumed with zero grade for Cu, Au, and Ag.

Table 16-13 Summary of Dilution and Recovery Parameters

	Description	Factor	Notes
Dilution	F/W end-wall	0.5 m	Applied to all zones
	H/W end-wall	0.5 m	Applied to MXN, BW, AA Zones
	Primary Stope	0.0%	Assume secondaries either side
	Secondary Stope	5.0%	Based on 25cm either side in backfill
	End of Line Stope	2.5%	Based on 25cm on 1 side waste
	Isolated Stope	5.0%	Based on 25cm on 2 sides waste
	Top/Bottom Sill Cuts	8.5%	Based on 75cm at start and end
Recovery	Top/Bottom Sill Cuts	1%	
	SLOS Stopes	94%	
Notes	Dilution from H/W end-wall based on GMT for MXS, GHFW and GHHW		
	Dilution assumed at zero grade		
	Backfill density = 2.0 t/m^3		

16.7 MINE DEVELOPMENT

Development has been laid out so as to enable access to the designed mining areas, and to allow effective truck haulage from these stoping areas to surface. The main ramps are designed to be accessed from within the existing ramp development system.

Main access to the underground mine will use the portal located at the west end of the Security pit (2119mRL). A secondary access will use the portal located within the North wall of the AA pit (2180mRL).

16.7.1 Ramp Development

There are three (3) main ramps that have been included in the mine plan:

- GHFW Ramp
- GHHW Ramp
- BW Ramp

The GHFW Ramp will be developed from the existing Santa Barbara/3988 Ramp starting at the 2025mRL elevation. This ramp descends in stages in the footwall side of the ore zone, to advance each 30 m level access point progressively further to the east. In this way the access points approximately follow the general plunge of the GHFW orebody. Access drifts are driven from the main ramp at approximately 25-30-meter sub-levels to gain entry to the level haulage galleries.

To access the GHHW (HW) orebody, a separate ramp has been designed to start from the bottom of the existing 3988 Ramp (1949mRL elevation). This ramp advances down the hanging wall side of the GHPHW orebody, allowing for level access points at approximate 20 m sub-level intervals. Two level connections have been designed between the GHFW and GHHW development systems to the south-east at elevation of approximately 1845mRL and 1765mRL. These level access drifts are driven to allow for more efficient movement of men and equipment between the stoping areas, as well as provide an alternate means of emergency egress.

A third ramp has been designed to access the smaller BW zone located at the West end of the orebody. The ramp will be developed from the bottom of the Rampa Concepcion (1914mRL).

All ramps are designed to be 5.0 m high by 5.0 m wide (with a semi-arched back) and are driven at approximately 14.3% inclination (15% on straight sections and 12% on curved sections). Within each ramp system, subsidiary development such as muck bays, passing bays, pump stations and service bays will be included but were not designed. An allowance of 15% has been added to the centerline distance in the mine schedule to account for this additional development.

For the AA, MXS and MXN zones, planned development and access on the upper levels has been designed so as to connect with the existing workings. Below the existing workings, the haulage galleries on the new lower levels started off in the GHPFW zone have been advanced westwards so as to allow access to lower stope blocks in the AA, MXS and MXN zones.
Figures 16-7 and 16-8 show the main development designs.

Figure 16.7 3D View of All Mine Development

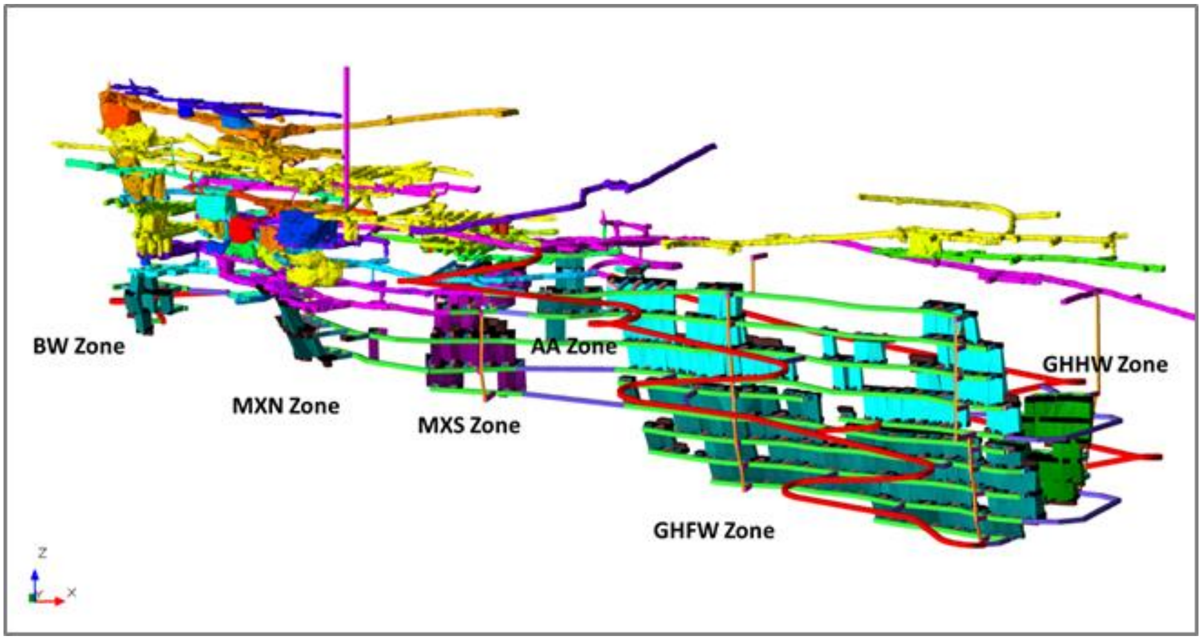
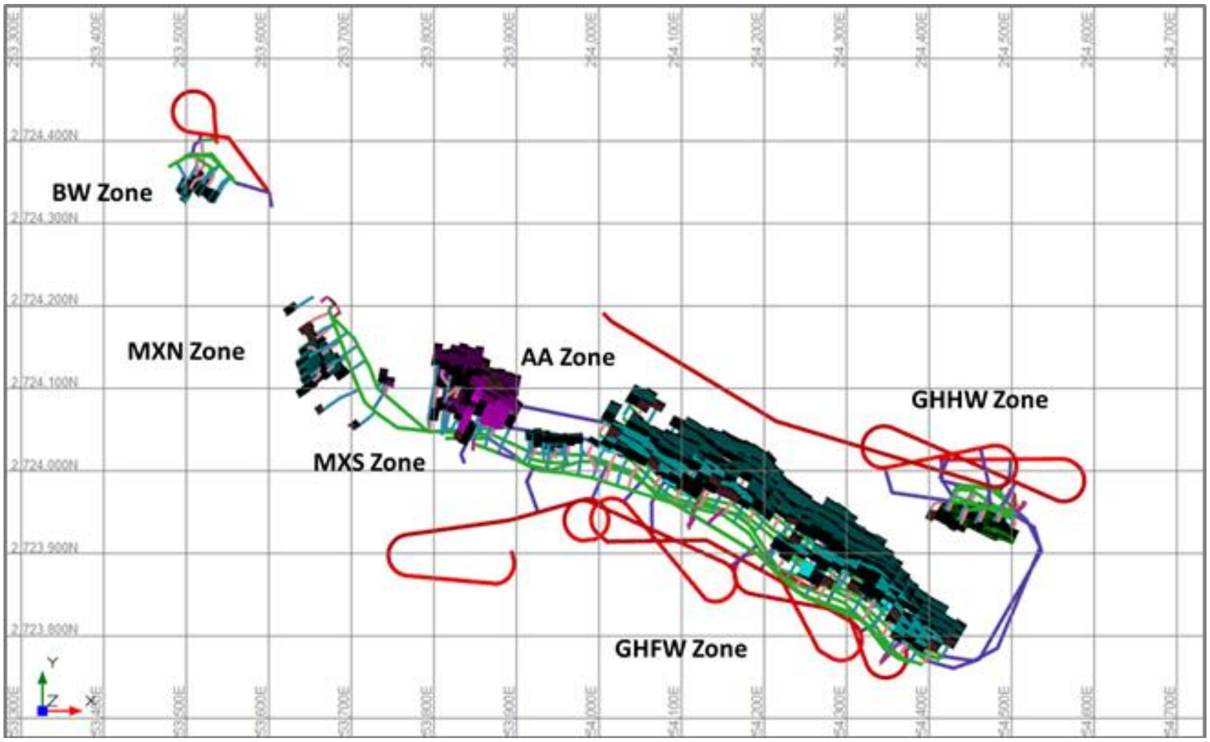


Figure 16.8 Plan of All Mine Development



16.7.2 Level Development

In all zones the same basic design methodology has been applied for the design of the necessary haulage galleries, level accesses and stope access crosscuts. This design approach can be summarized as follows:

- **Level Accesses.** The level access ramps connect from the main ramp to the access the haulage gallery at each sub-level elevation. Drifts are developed at 4.5 m high by 4.5 m wide with varying gradients depending on the location with respect to the main ramp. They were designed to provide a minimum rock pillar between the ramps and the haulage galleries of 20 m.
- **Haulage Galleries.** Haulage galleries are designed to be approximately parallel to the ore zones, so as to provide a minimum rock pillar between the stope blocks and the drift of 15 m. In areas with sharp changes in orebody orientation, the haulage galleries were smoothed out so as to allow practical access and development. This gave average distances between haulage galleries and stoping blocks of 18-20m. Haulage galleries are designed at 4.5 m high by 4.5 m wide to accommodate the installation of mine services and to provide effective clearance for mobile equipment. Development will be driven at a slight gradient (~1%) to allow for water to drain to a central location on the level.
- **Primary and Secondary Cross-cuts:** The design of the primary and secondary stope cross-cuts is different depending on the width of the stope being accessed. Cross-cuts are 4.5 m high by 4.5 m wide to accommodate the largest size loader expected to be utilized for stope mucking.
    - *10 m Wide stopes***:** Primary stope access crosscuts were designed so as to be aligned along the edge of each primary panel boundary. These crosscuts are to be used to access the primary

stope preparation sill cuts. Cross-cuts are 4.5 m high by 4.5 m wide to accommodate the largest size loader expected to be utilized for stope mucking. Secondary stope accesses are designed to be developed starting back 6-8 m from the orebody contact and will run parallel to the primary stope cross-cut. This will require the placement of "Jammed" backfill in the primary stope cross-cut following the backfilling of the primary stope to limit the roof span and limit the risk of ground instability in the secondary stope accesses
    - *15 m wide stopes:* For cross-cut development in the GHFW zone, primary and secondary stopes will be developed separately (as discussed in Item 16.2 - Pillar Stability Analysis). They are to be driven along the centerline of the stopes so as to provide effective coverage to the entire width of the stope during mucking.

- **Ventilation Access**: Access drift are developed from the Haulage Gallery to access the ventilation raise network required to ventilate the production areas. Ventilation accesses are typically designed at a length of 10-12 m with space to accommodate access to both the up and down direction raises. They are designed to be 4.0m high by 4.0 m wide.
- **Muck Bays/Backfill Bays:** Muck bays are required on each level for the intermediate storage of ore/waste. Muck bays are to be driven 5.0 m high x 4.5 m wide with a length of 16 m to allow for sufficient storage volume. In addition, the back height at the intersection of the muck bays with the haulage gallery will be increased to 6.5 m to allow for effective loading of the haul trucks. Backfill bays are also required on the levels to allow for temporary storage and mixing of the CRF material before it is transferred to the stopes.

Figures 16-9 and 16-10 illustrate the typical design configuration of a sub-level elevation. Muck bays and Backfill Bays were not designed however they were accounted for in the mine plan with a development factor of 10% applied to the haulage galleries. Note that no development factor was applied to the cross-cut meters.

A summary of the planned waste development can be found in Table 16-14.

Figure 16.9 Plan View of Typical Level Development Configuration (GHFW Zone)

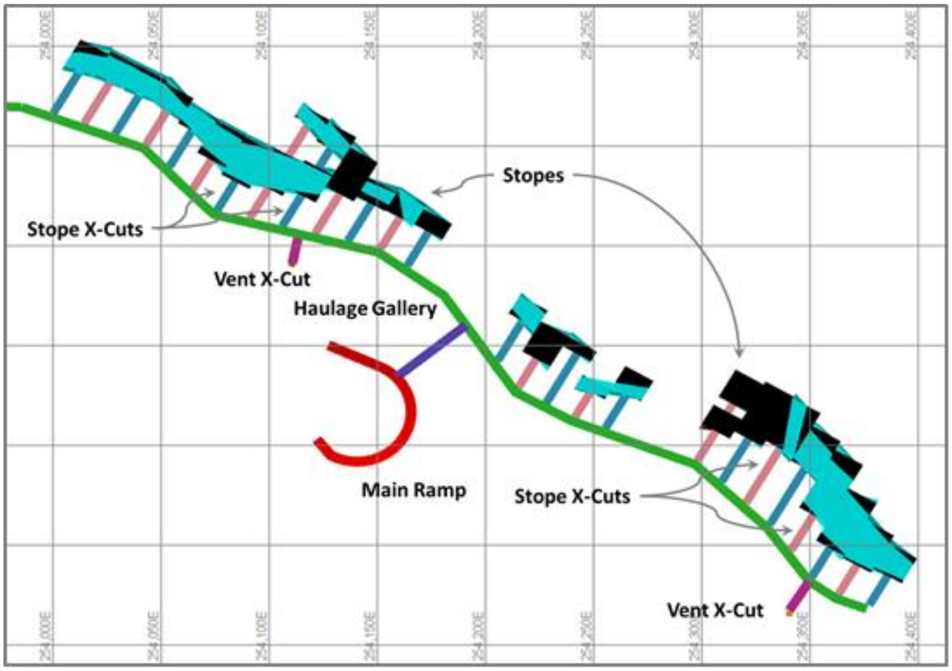


Figure 16.10 Plan View of Typical Level Development Configuration (MXS/MXN Zone)

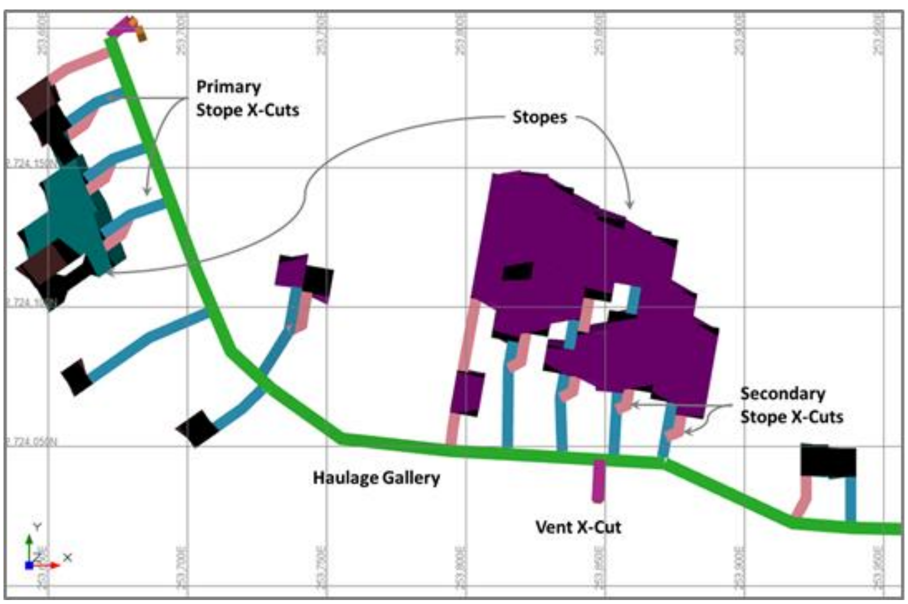


Table 16-14 Summary of Development Meters

Ramps and Level Development		Height (m)	Width (m)	Length (m)
Ramp Development	*m*	5.0	5.0	4,268
Level Access	*m*	4.5	4.5	2,067
Haulage Galleries	*m*	4.5	4.5	6,852
Stope Cross-Cuts	*m*	4.5	4.5	8,196
Vent Access Drift	*m*	4.0	4.5	605
Total Ramp & Level Development	***m***			**21,988**
Vertical Development				
Ventilation Raises	*m*	3.0	3.0	782
Total Development	***m***			**22,771**

16.8 MINING SCHEDULE

The mine development and production schedule were developed by Stantec engineers (under the direction of Aura Minerals personnel) using the Deswik® software package. Development and stope designs were imported from DataMine and adjusted based on revised design criteria for stope configuration. The sequence for both development and production activities was developed and the appropriate rates and production targets were applied to achieve the required schedule of activities. The schedule reports key results (tonnes, grades, development meters) to be used in the financial evaluation.

Rates for development and production activities were determined based on previous experience with other project as well as use of cycle-time based first-principle calculations. The LOM development rates have been prepared with consideration given to the potential bottlenecks such as equipment availability, capacity to move muck, congestion in the main ramp, and ventilation. Table 16-15 summarizes the development and production rates used in the mine schedule.

Table 16-15 Development and Production Rates

Development	Height (m)	Width (m)	m/day	m/month	Comments
Primary Ramp	5.0	5.0	4.00	120.0	includes allowance meters
Misc. Ramp Development	4.5	4.5	4.00	120.0	includes allowance meters
Haulage Level Galleries	4.5	4.5	3.33	100.0	includes allowance meters
Misc. Level Development	4.5	4.5	3.33	100.0	includes allowance meters
Stope Access X-Cuts	4.5	4.0	2.50	75.0	
Vent Raises	3.0	3.0	0.83	25.0	
Production	**Height (m)**	**Width (m)**	**t/day**	**t/month**	**Comments**
Transverse Stopes (10m wide)					
Bop/Bottom Sill Cuts	4.5	10.0	125	3,761	equiv to 2.3m per day
Stope Production	25.0	10.0	325	9,880	includes CRF placement
Transverse Stopes (15m wide)					
Stope Sill Cuts	4.5	15.0	152	4,571	equiv to 2.3m per day
Stope Production	25.0	10.0	364	10,921	includes CRF placement
Backfill	**Height (m)**	**Width (m)**	**t/day**	**t/month**	**Comments**
Transverse Stopes	30	10	1,581	48,228	
Longitudinal Stopes	30	8	1,581	48,228	
Jamming	4.5	4.5	450	13,725	assumes ~10 meters per day @ 4.5 x 4.5
Main Development Allowances					
Main Ramp	15%				incl. muckbays, passing bays, sumps, etc
Main Haulage Levels	12%				incl. muckbays, sumps, etc
Level Accesses	10%				incl. sumps

16.8.1 Development and Ore Production Profile

The schedule assumes development will commence in the first month in the GHFW and BW Ramps. Main levels will be developed in sequence as ramp development reaches the appropriate location. In areas where existing development is close to the planned stoping areas, stope access cross-cuts will be developed to provide ore production in the first 6 months of the schedule.

The maximum development target of 700 m per month was applied to main development; in addition, there was a percentage factor applied to the development rates in the first 3 months to account for operational ramp-up. The GHFW ramp is developed to the 1840mRL elevation which allows access to the entire upper mining block of the zone. Ramp development will be delayed for a period of approximately 6 months after which development continues to the bottom elevation (1740mRL). The development of the main haulage galleries within the GHFW zone was deemed to be critical to achieving the production targets as early as possible. The GHFW ramp was not scheduled to start until the third year of the mine plan with ore production not occurring until later that year.

The ore production target was set at 2,597 TPD with allowances for ramp up during the first six months. Initial ore production is targeted to come from the MXS and the AA zones as the existing underground development will allow for rapid access for ore development and stoping. GHFW zone will begin ore production from ore development in Q3 of Year1 with the first stopes being mined in the upper mining block in late Q4 of Year 1.

Table 16-16 below summarizes the development and ore production profile.

Table 16-16 Production and Development Schedule

	Name	Unit	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7
Development Breakdown	Dev_Ramp_m	m	4,268	1,732	839	1,696	-	-	-	-
	Dev_Level_m	m	2,067	432	655	558	422	-	-	-
	Dev_Haul_m	m	6,852	2,164	2,475	1,625	587	-	-	-
	Dev_StopeXC_m	m	8,196	1,515	2,591	1,932	1,349	766	43	-
	Dev_VentXC_m	m	605	126	200	214	55	10	-	-
	Dev_VR_m	m	782	99	253	292	137	2	-	-
	Dev_Total_m	m	22,771	6,068	7,014	6,318	2,550	778	43	-
Development by Zone	Dev_BW_m	m	891	891	-	-	-	-	-	-
	Dev_MXN_m	m	655	-	-	-	588	67	-	-
	Dev_MXS_m	m	2,426	518	683	539	584	103	-	-
	Dev_AA_m	m	798	684	87	27	-	-	-	-
	Dev_HW_m	m	4,140	-	350	2,651	1,129	10	-	-
	Dev_FW_m	m	13,861	3,975	5,894	3,101	249	598	43	-
Waste Breakdown	Dev Waste Tonnes	K-tonnes	1,285	357	390	363	133	40	2	-
	Sill Waste Tonnes	K-tonnes	159	27	28	50	39	15	-	-
	Total Waste Tonnes	K-tonnes	1,444	384	418	413	172	55	2	-
Backfill	Backfill_CRF	m3 x 1000	770	54	231	212	192	105	1	-
	Backfill_RF	m3 x 1000	789	21	88	130	142	223	193	-
	Backfill_Total	m3 x 1000	1,574	75	321	345	337	333	198	-
Ore By Zone	Ore_BW_t	K-tonnes	70	70	-	-	-	-	-	-
	Ore_MXN_t	K-tonnes	130	-	-	-	57	73	-	-
	Ore_MXS_t	K-tonnes	873	143	93	142	218	276	2	-
	Ore_AA_t	K-tonnes	120	84	9	27	-	-	-	-
	Ore_HW_t	K-tonnes	358	-	-	10	239	107	-	-
	Ore_FW_t	K-tonnes	3,093	71	807	765	428	487	535	-
Ore By Method	Sill_Cut_Ore_t	K-tonnes	1,210	173	355	244	227	184	26	-
	LOS_Ore	K-tonnes	3,431	195	555	698	715	759	510	-
Ore Breakdown	Ore_Total_t	K-tonnes	4,642	368	910	942	942	943	537	-
	Ore Cu	%	1.72	1.73	1.66	1.70	1.83	1.78	1.60	-
	Ore_Au	g/t	1.17	0.88	1.42	1.33	0.98	0.97	1.35	-
	Ore_Ag	g/t	19.2	21.1	19.1	18.3	19.0	19.3	20.0	-
	Ore As	ppm	1,609	1,343	1,576	1,556	1,546	1,560	2,142	-
	Ore_NSR	$/t	$ 99	$ 94	$ 102	$ 101	$ 99	$ 97	$ 98	$ -

Tonnes and grade reported in the mine plan represent the fully diluted and recovered ore material. All stopes and sill cuts are evaluated against the $60 NSR to determine if they are still economic. Stopes that fall below the economic cut-off do not report to the final ore production and are removed from the plan. Sill cuts that fall below the economic cut-off but are required to be developed to access stopes above and/or below are reported as sill waste but remain in the schedule. In the actual production setting, these sills would be reported as ore as long as they are above a marginal cut-off grade that considers costs for haulage to the mill, processing costs and any incremental administration costs.

16.9 MINING EQUIPMENT

The mining method employed at Aranzazu will require the use of standard mobile mining equipment to execute the LOM plan. All mobile mining equipment will be supplied by the mining contractor. The contractor is assumed to cover all equipment operating costs (including fuel, maintenance, and consumables) and these costs are included in the unit mining rates for development and production.
Development (ore and waste): assumes the use of 2-boom jumbos (Sandvik DD321 or similar) and bolting jumbos (Sandvik DS311 or similar) equipped for installation of resin-grouted rebar or split set friction bolts

Stoping: assumes the use of longhole drills (Atlas Copco E7C or similar) for stope drilling and a powder truck for the transport and loading of explosives.

Mucking: assumes the use of both 6.0 and 9.0 cubic yard load-haul-dump (LHD) units for mucking and loading of ore and waste.

Haulage: assumes the use of 14.0 cubic yard conventional haul trucks for ore/waste

Shotcrete: assumes the use of a shotcrete spray machine and two (2) transmixers.

Table 16-17 below summarized the proposed equipment fleet to be supplied by the contractor.

Table 16-17 Mobile Mining Equipment Fleet

Equipment Type	Model	Qty	Comment
Jumbo	Sandvik DD321	7	twin-boom jumbo
Rock Bolter	Sandvik DS311	4	install resin-rebar
LHD	Sandvik LH410	3	4.5 m^3 capacity
LHD	Sandvik LH514	5	7.0 m^3 capacity, remote equiped
Haul Trucks		15-17	14.0 m^3 capacity
Longhole Drill	Sandvik DH321	3	
Grader		1	Low Profile
Explosives Loader		3	
Telehandler		2	
Scissor Lift		2	
Light Vehicles	Toyota, Kubota	5	Man transport/maintenance

In addition to the equipment provided by the contractor, Aura Minerals will operate a small fleet of service and light vehicles to support the mining operations and perform routine maintenance of the mine infrastructure. This will include:

- 4-wheel drive utility tractors (Kubota) for engineering, geology, and geotechnical personnel
- 4-wheel drive pickup trucks (w/ flatbed configuration) for maintenance and ore control
- Service vehicle w/manlift to support survey and other elevated work functions

16.10 MANPOWER

The contractor will supply the effective manpower compliment to complete the work according to the LOM plan. The compliment will include miners, equipment operators, labourers, maintenance personnel, and supervision. It is estimate that the total manpower compliment will be between 80 – 90 people.

Aranzazu personnel will be responsible for providing technical support and supervision to the mining contractors in the areas of mine planning, ore control, and surveying. Aranzazu personnel are also responsible for the maintenance of the mine infrastructure including, but not limited to main pumps and fans, compressors and main electrical distribution network.

The Technical Services team will be comprised of planning engineers (4), geotechnical engineers (3) geologists (8) and surveyors (5). Total manpower compliment (including manager) of 20. The Mine Operation team will be comprised of a Mine Captain and three (3) supervisors to oversee the mining contractors. Total manpower compliment of 4. The Maintenance team will be comprised of a Superintendent, a planner and seven (7) mechanics and electricians. Total manpower compliment of 9.

The manpower compliment also included the position of the Mine Manager and a mine clerk. Total manpower compliment (Aranzazu staff) is 35.

16.11 BACKFILL

The LHOS mining method requires the placement of backfill material to fill the voids following extraction of the ore. In the case of a primary and secondary configuration of transverse open stoping, the primary stopes will require filling with a cemented rockfill (CRF) of sufficient strength to remain stable during the extraction of the adjacent secondary stopes. Secondary stopes or longitudinal stopes will be filled with uncemented waste rock that is either hauled from surface or from underground development location.

In addition to the material required for backfilling of the open stopes, it will also be necessary to place jammed backfill in the both the primary stope crosscuts (where the mine design incorporates the combined primary/secondary stope cross cut) and in the top-most sill cut of the all primary stopes. This will require the use of an LHD unit with a “Rammer Jammer” attachment to jam the backfill material tight to the back. Secondary stope sill cuts do not require backfill and will be left as open voids.

Based on the LOM schedule, a total of 1.5M m^3 of void space will require filling.

- Stope Fill (CRF) = 0.73 M cubic meters for 1.53 M tonnes
- Stope Fill (URF) = 0.76 M cubic meters for 1.45 M tonnes
- Stope Cross-Cut Jam (CRF) = 0.02 M cubic meters for 0.04 M tonnes
- Stope Top Sill Jam (CRF) = 0.06 M cubic meters for 0.12 M tonnes

The LOM plan will require approximately 2.8 M tonnes of aggregate to be supplied for backfill with 1.5 M tonnes required for CRF. This aggregate will require screening to produce the proper size distribution necessary to ensure the strength requirements of the CRF can be met. The geotechnical recommendations for CRF state that the material should be unaltered material, sulphide free and have a UCS of greater than 40 MPa. Site inspection of the Aranzazu waste dumps have shown there are sufficient quantities of Marble waste that would meet these criteria. In addition, underground waste from the intrusive rock mass will also meet these criteria and will be stockpiled during the initial mine development for use as backfill aggregate

CRF will be produced on surface and transported underground. Aggregate will be prepared onsite to the recommended specifications. Haul trucks will be loaded with prepared aggregate and weighed. Based on the recommended mix design of the CRF product, a pre-mixed dose of cement slurry will be pumped into the truck to meet the cement content requirement. Trucks will deliver the backfill to the active backfilling location where it will be dumped into a mixing pit on the level. An LHD will manually mix the CRF mixture to establish consistency of the product and then haul and dump the material into the stope.

At steady state production, the backfill system will be required to produce and place approximately 1,200 – 1,300 TPD. The mining contractor will supply the equipment necessary to produce the CRF. Aranzazu staff will provide design criteria and specification as well as conduct routine sampling and testing to confirm quality standards are being met.

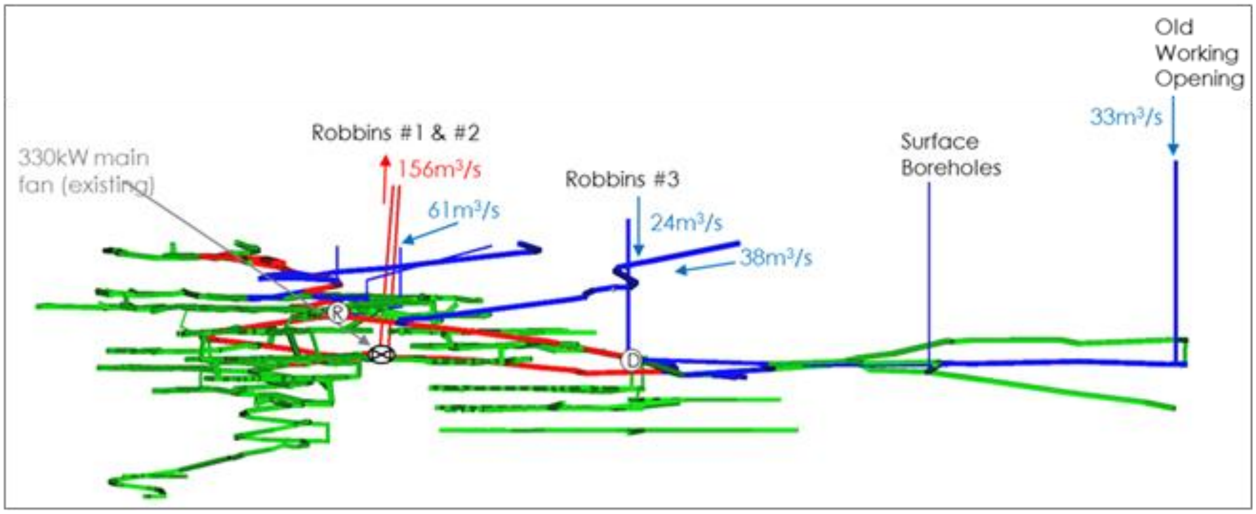
The cement content for the CRF has been estimated at between 5% - 8% depending of the CRF requirement however testing will be required to determine the exact mix based on the aggregate quality and size distribution. As of the writing of this report, the testing program has been approved but not started. Further refinements of the dosing and mixing strategy along with aggregate size specifications will be required following completion of the test program.

16.12 VENTILATION

The ventilation system at Aranzazu is designed to ensure that all active areas of the underground mine are adequately ventilated such that the health and safety of workers is maintained throughout development, and production at the planned production rates and to prevent accumulation of gases, dust and other contaminants within the mine. The system is designed as a “pull” system such that the negative pressure required to pull air through mine is developed by large exhaust fans. Fresh air will primarily enter the mine through the two (2) portal entrances along with small surface connection raises and exit through 2 main ventilation raises. The system is designed to supply a total airflow volume of 282 m^3/s.
In November 2017, Stantec Mining conducted a ventilation review and modelling to determine the ventilation requirements and the sequence of the ventilation stages. The results of this work have been used to define the ventilation strategy for the mine and are described below.

The current ventilation system is comprised of a pair of 2.4 m diameter circular raises (165 m in length). located within the Santa Barbara ramp. A single 2.35 m diameter, 300 kW Axial Fan is installed at the bottom of the Robbins #1/#2 twin raises. In addition, another 4.0 m diameter (131 m in length) circular raise was driven from the 3988 Ramp to surface. A second duplicate fan, which was previously purchased will to be installed at the top of the Robbins #3 raise. The raise is currently blocked due to poor ground conditions however the plan will require the raise to be rehabilitated before installation of the new Both fans are required to be installed in order to achieve the required airflow volumes and distribution. The majority of the fresh air is delivered to the mine from the two portals located within the AA and Security pits. In addition, there are two smaller diameter escapeway raises with connection the surface from the underground which provide additional fresh air to the mine. Figure 16.11 shows a schematic of the current ventilation configuration.

Figure 16.11 Schematic of Current Ventilation System System



Airflow requirements are based on equipment and manpower requirements. For equipment requirements, the airflow is determined from the equipment engine power, quantity operating underground, operating utilization and a recommend factor of 0.048 m3/s per kW engine power (75 cfm per HP). For manpower, the factor is 0.0235 m3/s (50 cfm) per person.

The calculation for the airflow requirements calculation for the estimated planned underground mobile fleet is outlined in Table 16-18. For full LOM production requirements, the total fresh air required underground is approximately 283 m3/s.

Table 16-18 Ventilation Requirements

		QTY	Engine (kw)	hr/day	Oper. Util. (%)	Eff. Util. (%)	Airflow (m3/s)
Equipment	Truck	15	337	22	75%	69%	165.29
	LHD	3	235	22	75%	69%	23.05
	LHD	5	260	22	75%	69%	42.51
	Jumbo	6	110	22	40%	37%	11.51
	Bolter	3	90	22	40%	37%	4.71
	Long Hole Drills	3	90	22	30%	28%	3.53
	Service Vehicle	5	75	22	50%	46%	8.17
	Light Vehicle	8	75	22	50%	46%	13.08
	Grader	1	110	22	75%	69%	3.60
	Explosive Loader	3	60	22	75%	69%	5.89
Personnel	Haulage	10					-
	Development/Production	20					-
	Maintenance	5					-
	Miscellaneous	10					-
	Production Crews	45					1.06
	Miscellaneous						-
	TOTAL						**282.40**

The ventilation system will be developed in stages starting with the initial development of the GHFW ramp to the full LOM production state. The ventilation for the initial ramp development and mining will be supported by the operation of existing 330 kW fan at the Robbins #1/#2 raise. If Robbins #3 is not available during this stage, the other existing openings can support delivery of the fresh air requirements. This will allow the fresh air to be pulled through the ramps and surface openings and directed to the active working areas. For the GHFW zone, fresh air will be provided from the Santa Barbara ramp. The ramp can be developed up to 1,200 m using forced ventilation – it is recommended that 2- 120 kW fans be installed in parallel using 1.2 m diameter ventilation ducting. Ventilation to the BW, AA, and MXS zones will be provided from fresh air coming down the main Conception ramp. During this phase, the system is expected to provide 130 -140 m3/s to the underground working.

During the second phase of the operation, the GHFW zone has been developed to the 1850 elevation and the main sub-level haulage drives have been established. On each main level, once the development has reached the designed location, ventilation raises will be established to connect to the level above. Raises are designed to be 3.0 m x 3.0 m square, developed using conventional techniques. The internal raise system is designed with 2 raises on each level – one on the west side and another on the east side of the GHFW zone. Ventilation control for the level will be achieved using bulkheads with regulators installed at the entrance of the ventilation raise cross cut; wood planks will be placed across the regulator opening to control the ventilation being pulled onto the level. For ventilation raises that will have secondary egress ladderways installed, the bulkheads will be equipped with a man door.

The ventilation raises will be connected from the top mining level (1970 level) to the existing working at the approximate 2005 level where it will be directed to the exhaust raise. At this point, the amount of production activity will require the re-establishment of the Robbins #3 ventilation raise and the installation of the second 300 kW fan. A total airflow requirement of 226 m^3/s will be required at this phase of the operation. Ventilation to the other mining areas remains the same as phase 1.

During this phase, the main GHFW ramp will continue to be developed using the forced ventilation system. Establishment of the internal ventilation raises on each of the main levels will allowing for flow through ventilation across that level. The fan and the ducting will be re-installed above the level which will provide sufficient ventilation to extend the ramp.

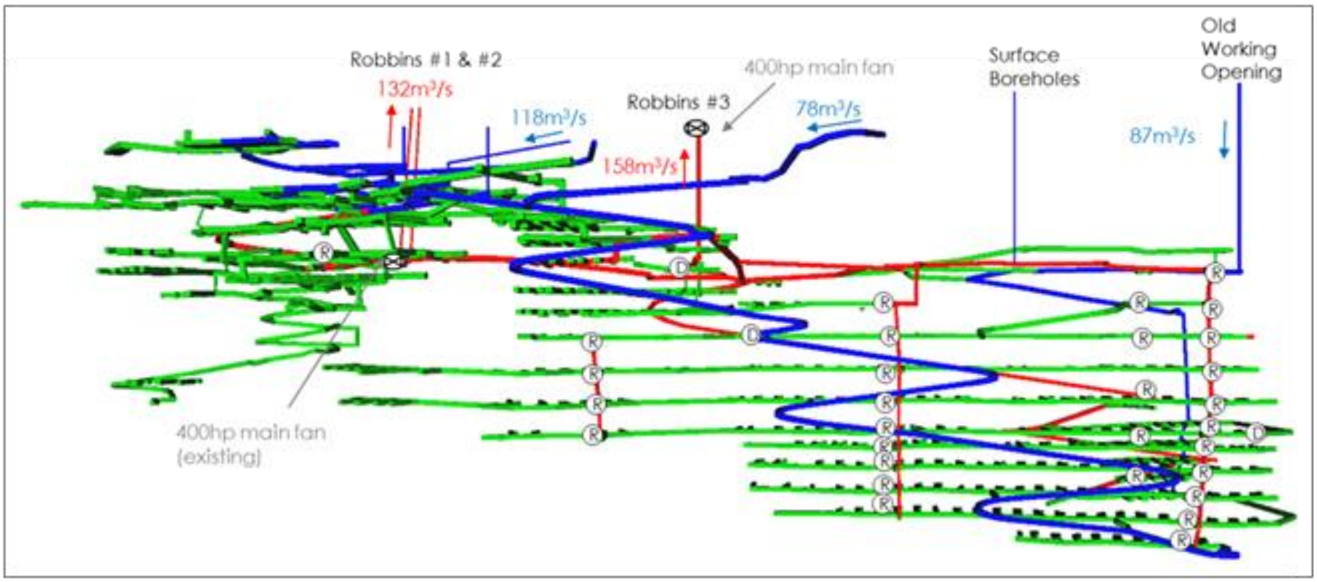
Stage 3 will be established to support the development of the GHHW ramp which is located at the bottom of the 3988 ramps. As with the GHFW ramp, ventilation required for development will be achieved using a similar forced ventilation system to supply fresh air to the face. A 3.0 m diameter raise will be developed from the bottom of the 4066-GHH1980 ramp to a cross cut located at the approximate 1865 elevation of the GHHW ramp. This will establish flow-through ventilation to the GHHW ramp and allow for extension of the forced ventilation system to continue the ramp development to the bottom elevation. At each level, a ventilation raise will be established to the level above and connected to the fresh air raise. Fresh air will flow down the raise system, across the levels and exhaust up the ramp. In order to mine the GHHW zone, it will be necessary to provide a dedicated intake for the North side of the mine. Previously there were a series of 5 – 30 cm boreholes drilled from surface. However, the effective area of the boreholes is not large enough to provide effective airflow. Fresh air to the GHHW zone (north side network) will be supplied via an existing opening on the bottom of the Security Pit. This connection is less than 20 m however it filled with surface waste. It will be necessary to re-establish the opening and install a 2.4 – 3.0 m diameter culvert to maintain the opening; if needed this can be equipped with a ladderway to act as secondary egress route.

At the full LOM stage, all the mining areas have been developed and are in production. Both the GHHW and GHFW ramp have reached the bottom mining elevation and the internal ventilation raise networks have been established. At full operation, the system will supply approximately 295 m^3/s to the underground working areas. Figure 16.12 shows the final ventilation configuration.

Ventilation control doors will be required in specific locations of the mine to ensure the ventilation network operates effectively. One vent door will be located just west of the Robbins #1/2, to avoid short circuiting, with the second located on the 3988 Ramp below the GHFW ramp entrance to prevent mixing of the fresh air and exhaust air streams. In addition, the design requires the construction of several permanent bulkheads to close off old working areas and prevent leakage.

In the case that the existing Robbins #3 ventilation raise cannot be successfully rehabilitated, it will be necessary to develop another raise in a suitable location to provide an exhaust route for the East side of the mine (GHFW and GHHW zones). The raise should be a minimum of 4.0 m in diameter and based on the experience of the existing Robbins #3 raise it should be lined and/or supported to avoid any ground instability. The location should consider the most suitable placement with respect to existing underground working and the planned ventilation network for the GHHW and GHFW zones along with the surface footprint for the fan and associated infrastructure.

Figure 16.12 Schematic of Final Ventilation Configuration



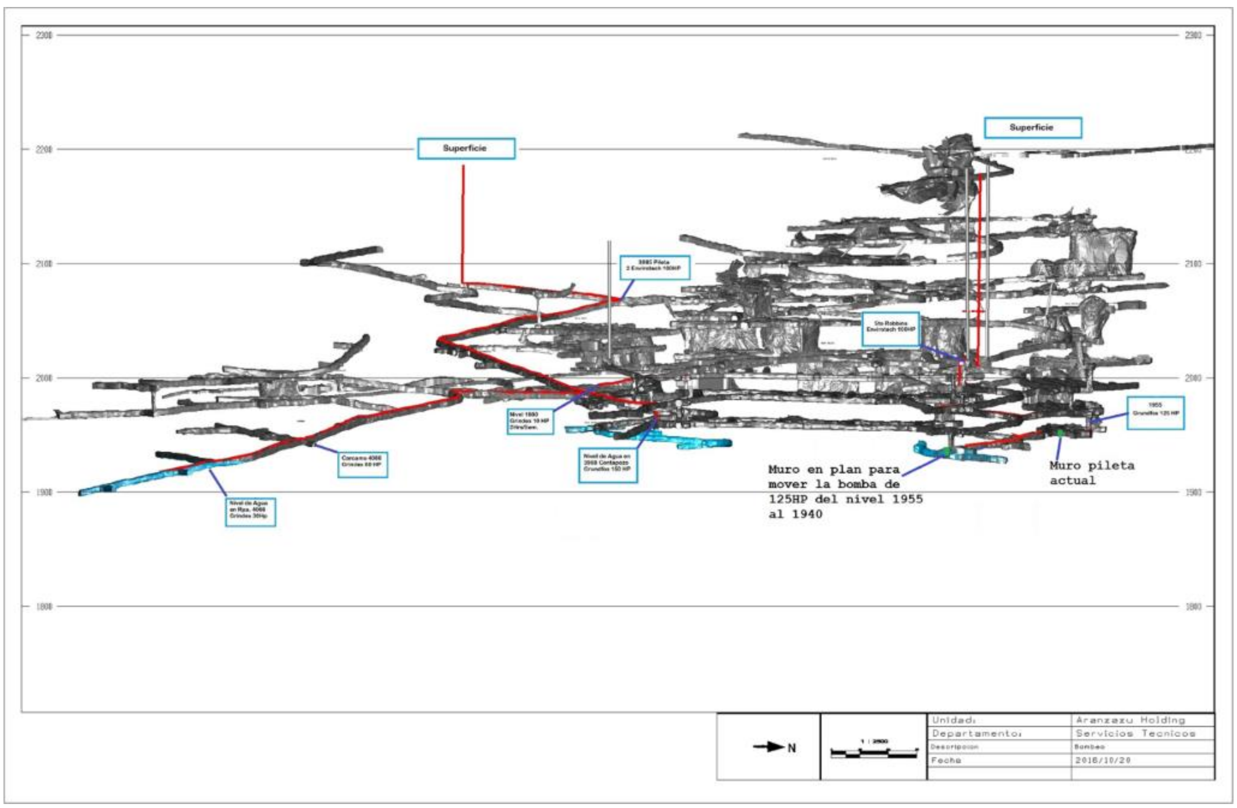
16.13 PUMPING AND DEWATERING

The current mine dewatering system is comprised of multiple sump pumps located at various points within the mine feeding a central sump located at the 3988 elevations of the main ramp. The main sump is composed of 2 – 75 kW Envirotech pumps which pump to water to the surface water ponds via a service borehole. The sumps are located throughout the mine. In addition, as a result of the mine being on care and maintenance for the last 3 years and increasing water levels in the lower areas of the mine, a temporary pumping system was installed on the West side of the mine to decrease the water level in this area. A 93 kW submersible pump feeds water to a 75 kW Envirotech pump which then pumps the water to surface through a 6" water line located at the Robbins #1/#2 ventilation raises. The total capacity, including the temporary system is estimated at 25 L/s. Figure 16.13 shows the current underground pumping configuration.

In November 2017, Stantec Mining conducted a review of the dewatering system and potential pumping rates and provided design recommendations for the new dewatering system. The results of this work are described below.

The system will require pumping all of the water from underground to the surface ponds. Potential sources of water include groundwater inflows and mine service water inflows. The hydrogeological study performed by CNI identified two potential sources of groundwater inflows into the mine; that from the groundwater infiltration and the other from intersection of perched water-bearing structures. The study provided values for both the average and peak inflows that would expected. Predicted groundwater inflows range from 20 – 30 L/s depend on the depth of the mine. Mine service water inflows are based on the quantity and type of units in the drilling fleet, the equipment utilization as well as other sources of water usage in the mine. Mine water usage is calculated in the range of 6-10 L/s. As a result, the water balance for the underground results in a mine dewatering requirement of approximately 20 – 30 L/s. A design criteria of 40 L/s was chosen as the system capacity.

Figure 16.13 Schematic of Current Dewatering System



The design concept for the new dewatering system incorporates a single-lift-clear water pumping system to surface from a central pumping location. The central pumping location will have all source of water feeding into it and is designed to allow for solids settling from the various sumps located in the mine. Dirty water submersible pumps will be installed in the sumps to handle both the groundwater and mine service water inflows.

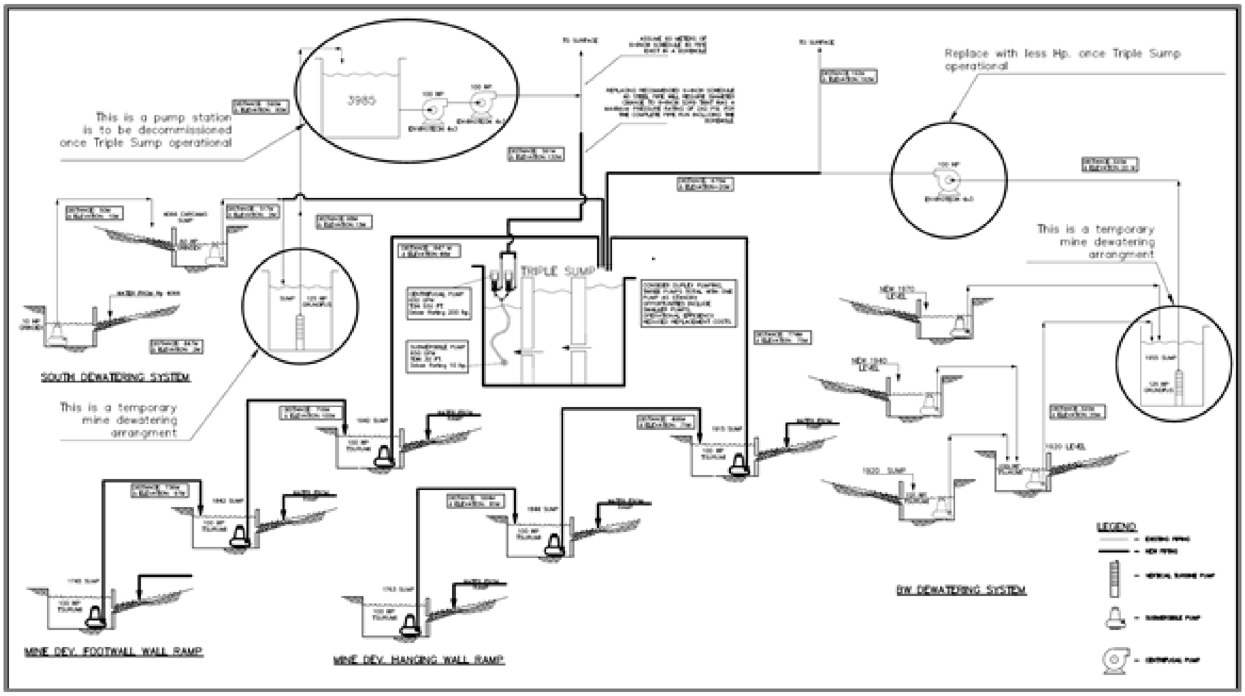
The main sump will be located at the bottom of the 3988 Ramp at the approximately 1990 elevation. An existing excavation comprised of three (3) parallel sumps had been previously developed for use at a central solid settling sump. The sumps are approximately 30 m in length and have small burn-cut openings developed between each of the sumps at mid-wall height to allow for water to flow from one sump to the next. Dirty water from the various level/ramp sumps will be fed into sump bay #1 where initial solids settling will occur. The sump will have a designed capacity of 250 m^3 which will provide for a retention time of between 2 – 4 hours depending on total flow rate. Partially clarified water will pass through the small opening into sump bay #2 for additional clarification. Sump Bay #2 will have a half-height decant wall installed along the length of the right wall (in front of the Sump #2 / #3 opening) to allow for additional solids settling with final clear water from Sump #2 being fed to sump bay #3. Two centrifugal pumps will to be installed to pump the clean water to surface. Each pump will be 150 kW capable of pumping 40 L/s allowing for one pump to remain on standby. The pumps will be controlled by simple level control switches to prevent running dry. Both sump bay #1 and #2 are design to allow for routine cleaning and removal of the slimes.

Secondary sumps will be located at specified elevation in the main ramps to collect water from the levels and the ramps. The systems will operate in a cascade arrangement with the lowest sump pumping up to the next sump with highest sump pumping to the main settling sump. Sumps will have single 75 kW submersible pumps capable of pumping dirty water at 40 L/s for up to 100 m vertically. Secondary sumps are designed to facilitate effective removal of the pump and ease of cleaning. Water from the mining levels will be delivered to the secondary sumps via small capacity submersible pumps or where possible, boreholes drilled from the level.

The majority of the piping for the dewatering system will be 150 mm diameter HDPE however due to the high pressures generated by the main pumps, it is recommended to install 150 mm diameter schedule 40 steel line from the main pumps to the borehole connection.

Figure 16.14 below shows a schematic of the dewatering system.

Figure 16.14 Schematic of Planned Dewatering System



16.14 MINE SERVICES

16.14.1 Electrical Distribution

The current electrical distribution system to the mine is fed from the main 35 kV powerline coming to the Property. There are two existing surface substations, Subestación Trio Ventilación and Subestación BW/Mexicana which together supply 4 MVa at 4,160 V. These substations provide power to both underground and surface facilities. These substations provide power to both underground and surface facilities. The Subestación Trio Ventilación feeds power to the Mexicana Norte /Mexicana Sur areas while the Subestación BW/Mexicana supplies pwer to the BW zone and the 300kW Alphair ventilation fan. There is a third 1,500 MVa substation that was purchased but never installed.

In November 2017, Stantec Mining conducted a review of the current electrical distribution system and the future demand load associated with resuming mining operations. They provided design recommendations for the new electrical system and the results of this work are described below.

Based on the expected electrical load increases, the current power underground power distribution system does not have the capacity to support the operations. Additional electrical power will be required for the second ventilation fans, secondary ventilation, dewatering system and mining equipment. It is estimated that the total connected underground loads will be approximately 7.5 MVa with a peak demand load estimated at 6.3 MVa. In order to accommodate the additional load requirements, it will necessary to install additional load distribution capacity. A new 5 MVa substation/transformer will be installed to support the additional loads. It has been recommended that this transformer provide 13,800V power to the mine. The new 13,800V substation will provide power to the following areas:

- 300 kW surface fan via a 500 kVa 13.8kV to 4160V transformer
- Glory Hole Footwall – Tap box at the 1970 Level and 9 other consecutive tap boxes at lower levels 1940, 1910, 1880, 1850, 1840, 1815, 1790, 1765, and 1740. In addition, this leg will feed the Mexicana South Lower
- Glory Hole Hanging Wall – Tap box at the 1860 Level, and 5 other consecutive tap boxes at lower levels 1840, 1820, 1800, 1780, and 1760. These tap boxes will be used to split the feed for local mine power centers

The supply voltage has been increased because the 13,800V will be more efficient for longer transmission distance and will reduce the cost of the electrical cables. Furthermore, the new substation will double the current capacity to support future mine development below the current depth.

The existing 4,160 substations will remain active and continue to support the mining activity in the BW and Mexicana zones.

Mine Load Centers (MLC) will be located at each activity mining level which will drop the voltage from the 13,800V feed to 440V for supply to ventilation fans, pumps, and mining equipment. Mine Load Centers should be skid mounted to enable effective relocation and configured to provide power to a number of pieces of equipment. Standard controls and safety switches should be included in the specifications to prevent damage to equipment. It is anticipated that these load centers should be rated for 750 – 1,000 KVa depending of the required load demand.

For the ramp development, a dedicated MLC is required to supply the necessary electrical power to the face. This load center will advance with the advancing ramp and can be at a lower supply voltage due to lower requirements. It will be necessary to have 2 similar units for this duty; one unit in operation and the other on standby. When the distance between the active face and the MLC exceeds the effective limit (300-400 m), the standby transformer will be installed closer to the face and activated and the other placed on standby. This strategy will limit the amount of downtime in ramp development due to disruption in power supply. Feeder breakers and tap boxes will be permanently positioned during the development sequence and will provide junction points for planned mining areas or future mine development.

Electrical cable requirements specify low-voltage cables with an insulation rating of 0.6/1kV are acceptable for use with power systems from 120/240V through 1000V; this would cover power distribution to lighting and service receptacle’s, 440V power feeds. Medium-voltage cables are acceptable and commonly available, with insulation ratings of 2.4kV (133% and 100% insulation), 5kV (133% insulation), 8kV (100% insulation), and 15kV (133% insulation). It is recommended that, in harsh environments such as those found in mining, a corrugated aluminum-armored and jacketed cable such as a Teck-90 style be used for all underground power distribution.

16.14.2 Compressed Air and Water

The existing compressed air system is sufficient to support the mining operations. There are four (4) compressors of varying equipment manufacturers and capacity installed. The total estimated capacity of the system is approximately 1.75 m3/s at 7.0 kPa (3,700 cfm at 95 psi) at the elevation of the mine (2,070 m). Assuming a loss of 20% to leakage, the delivered compressed air has been estimated at 1.4 m3/s (2,790 cfm). At full operation, the expected demand has been estimated to be approximately 1.02 m3/s (2,160 cfm).

The analysis indicates there is sufficient capacity with the four (4) compressors to meet the demand of the mining operations. It will permit three (3) units to be in operation and one (1) on standby. The largest unit (Kaeser FDS 400) has a capacity of 0.67 m3/s which represents approximately 40% of the system capacity. When this unit is down (breakdown or maintenance), there will not be sufficient capacity to supply the full requirements; air usage will need to be managed during this time.

Mine water delivery system has been established and is in working order. The system will need to supply between 5 – 6 L/s during normal operations however the peak demand will be in the range of 9-10 L/s. As the water system piping increases with depth, it will be important to install pressure reducing valves to control pressure in the lines. There is no flow meter on the mine water inflow and it is recommended that one be installed to track the amount of water usage in the mine.

16.14.3 Communications

The primary underground communications system will be a via leaky feeder mine radio system, which allows for two-way communication within the underground and between surface and underground. Underground mine equipment will be equipped with mine radios. In addition, hand-held radio units are to be provided to contractor supervision and Aranzazu management / technical staff. Base station units will be situated in the mine offices, main office, safety / mine rescue office and rescue chambers.

As a secondary communications system, a self-contained battery-operated emergency mine phone system will be installed as a backup to the mine radio system. Units will be located at major underground infrastructure along with specific surface facilities.

17. RECOVERY METHODS

17.1 GENERAL

The recovery of the metal values from the Aranzazu ore will employ the existing equipment, which utilizes conventional crushing and grinding operations to liberate the minerals with flotation used to concentrate the metals into a saleable concentrate. The primary, secondary and tertiary crushing circuits have the demonstrated capacity to crush the expected throughput of 2,597 TPD of run of mine ore to a size of 80% passing 8 mm. This crush size is fine enough to allow the single stage ball mills and the regrind mill, converted to primary grinding, to grind the ore to a projected 80% passing 135 microns, operating with 92% availability for 365 operating days per year. This yields a throughput of 940,240 tonnes/year, and a daily rate of 2,597 TPD. During 2014 the plant operating utilization was 97% so it has been shown that this operation is capable of achieving a very high operating time.

The flotation section of the plant will use the existing conventional flotation cells, in almost the same circuit arrangement as in the earlier operation, to produce a concentrate with a grade of 23% copper. The only change in the flowsheet will be that the operator will have the ability to direct some or all of the concentrate from the second bank of flotation cells, to final concentrate in addition to the concentrate from the first bank of cells. During the previous operation, the flotation equipment was shown to have the capacity to treat 3,000 TPD of ore and is therefore expected to easily handle the current design throughput.

In contrast with previous operations, where the arsenic in ore was generally in the 0.08 to 0.15 % range, and recoveries to concentrate averaged 33%, the deeper, higher grade ores that will be processed in the next five years have arsenic in the 0.2-0.3% range but, more importantly, the recovery of arsenic will be in the range of 80-90% so that arsenic in concentrate will exceed 3% at times which would incur significant penalties if shipped. The most critical factor in determining when such high arsenic concentrates will be produced is the copper to arsenic ratio in the ore. It has been determined that a Cu/As ratio of below 7.7 will almost certainly produce a concentrate with a grade of 3% arsenic or higher. The new mine plan which uses net smelter returns (NSRs) to determine mining limits will also consider this critical ratio.

Since the ore will, inevitably have a copper/arsenic ratio of less than 7.7 occasionally, the concentrate produced at such times will be stockpiled and blended with concentrate produced when the ore has a higher copper to arsenic ratio. Alternatively, the high arsenic concentrate may be blended with purchased concentrate.

17.2 PROCESS FLOWSHEET

The process flowsheets are shown below in Figure 17.1 for the crushing circuit and Figures 17.2 and 17.3 for the grinding, flotation and dewatering circuits. Changes in the process flowsheet and process control are indicated in red.

17.2.1 Crushing

Run of mine ore is scalped across a static slotted grizzly screen with 15-inch openings and then fed to a vibrating screen with 4 inch wide slotted openings before the 36 inch X 42 inch (250 hp) primary jaw crusher. Both screen undersize and crushed ore are conveyed to the secondary vibrating screen which has a slotted 9.5 mm lower deck and produces final crushed product while the oversize is crushed in the secondary 300 hp cone crusher. Secondary crusher product is conveyed to the tertiary vibrating screen which has a 9.5 mm slotted lower deck and produces final crushed product from the undersize with the oversize reporting to the 400 hp tertiary cone crusher. The tertiary crusher product is recycled to the tertiary screen forming a closed loop operation. Crushed ore is conveyed to the fine ore distributor above the stockpile. The stockpile provides live capacity for approximately 12 hours of operation of the plant.

17.2.2 Primary Grinding

Design feed tonnage to the grinding circuit is 116.7 tonnes per hour. The three 373kw (500 hp) primary single stage ball mills are fed by conveyors which have single idler weightometers. The existing 186 kw (250 hp) regrind mill, now known as the #4 primary mill, has been converted to primary grinding with the feed being ball mill discharge from two of the 298 kw (425 hp) primary mills. The tonnage is controlled by adjustment of the speed of the conveyors under the feed pipes from the stockpile. The ball mills discharge to primary cyclone feed pump boxes and the primary cyclones operate in closed circuit with the mills to produce a flotation feed with a p80 of 135 microns. Lime and Cytec 5100 are added to each ball mill feed to maintain a pH of 11.0 in the cyclone overflow and to provide the primary collector for both copper and gold flotation. Lime addition rate is controlled using pH measured in each cyclone feed pump box. There will be an addition of Cytec 5100 to the #4 primary mill to ensure that the fresh surface created there has sufficient collector.

17.2.3 Conditioning and Flotation

Ground ore feed flows by gravity from the cyclone overflows through a static sampler into the two conditioning tanks where it is conditioned with Cytec 3477 and frother (MIBC). The conditioning tanks provide approximately 7 minutes of residence time. The feed overflows the second conditioning tank into the first of four banks of four 300 cubic foot Denver DR flotation cells. The first bank and, at operator discretion, the second bank of flotation cells are used to produce the final concentrate at 22-23% copper grade. Additional Cytec 3477 is added into the feed box of each of the flotation banks. Banks 3 and 4 produce a scavenger concentrate which reports back to the feed box of the first flotation bank. Baffles will be added between the second and thirds cells of the first and second banks of cells to prevent back mixing and therefore improve the residence time distribution in the cells that are producing final concentrate. The baffle will have a level control valve with attendant level sensor to ensure that there is always a difference in the pulp level between the first two and the second two cells. The final concentrate is sampled using a cross cut sampler in the discharge line from the concentrate pump box. The tailings from bank 4 constitute final tailings which are pumped directly to the tailings area. There will be a static sampler on the final tailings stream.

17.2.4 Metallurgical and Mass Balance

The simplified metallurgical and mass balance shown in Table 17-1 is derived from the locked cycle test described in Item 13 with appropriate adjustments for the final concentrate grade produced and the grade of the ore mined compared to the grade of the composite sample.

Table 17-1 Metallurgical and Mass Balance

	METALLURGICAL BALANCE								MASS BALANCE			
	Grade					Recovery(%)			M3/hr	Solids	% Solids	Slurry
	Cu(%)	Au(g/t)	As(g/t)	Ag (g/t)	Wt %	Cu	Au	As	Slurry	SG	by Wt	SG
GRINDING CIRCUIT												
BALL MILL FRESH FEED	1.72	1.17	1609	19	100	100	100	100	36	3.62	97.0	3.358
BALL MILL CYCLONE OVERFLOW	1.72	1.17	1609	19	100	100	100	100	249	3.62	35.0	1.339
FLOTATION CIRCUIT												
FLOTATION CIRCUIT FRESH FEED	1.72	1.17	1609	19	100	100	100	100	249	3.62	35.0	1.339
FLOTATION CIRCUIT TOTAL FEED	1.76	1.22	1663	20	102	104.2	106.5	105.2	258	3.62	34.5	1.333
ROUGHER (FINAL) CONCENTRATE	23.1	12.4	20283	202	6.6	88.0	69.9	83.0	16.2	3.89	35.0	1.351
SCAVENGER CONCENTRATE	4.00	4.23	4639	89	1.8	4.2	6.5	5.2	5.5	3.38	30.0	1.268
SCAVENGER (FINAL) TAILS	0.22	0.38	294	6	93	12.0	30.1	17.0	248	3.60	33.3	1.317

17.2.5 Grinding Steel and Reagent Requirements

Table 17-2 Grinding and Reagent Requirements

Grinding Steel & Reagents	Function	Usage (grams/tonne)	Annual Requirements (tonnes)*	Cost per tonne (US$)	Total Cost/ year (US$)
			* based on annual throughput of 941,000 tonnes		
Grinding Steel, Bolas de	Media in Primary grind	350	329	1118	368000
Lime, Cal	pH Control	2000	1882	95	179000
Cytec 5100	collector	6.00	5.65	8810	50000
Cytec 3477	collector	6.00	5.65	3200	18000
MIBC	frother	50.00	47.1	2200	104000

Annual total (US$)	$719,000

Price of Cytec 3477 estimated by inflating a 2011 price of $3000/tonne

17.2.6 Concentrate Dewatering

Concentrate thickening is carried out in a 40ft (12.2m) diameter conventional thickener without flocculant addition although flocculant addition may become necessary with the much-increased concentrate production rate in years 3 to 5 of the production plan. Thickener overflow flows by gravity to a concrete lined decant pond adjacent to the thickener and the clarified water is pumped back to the process. Thickened concentrate at between 60 and 65% solids (w/w) is pumped directly to a TEFSA plate and frame pressure filter for final dewatering. The filter has 31 plates installed at present and has the capacity to take up to 42 and is expected to easily handle the increased concentrate production resulting from the higher ore grade.

17.2.7 Process Control

Process control will be improved dramatically compared to the previous operation by

- Providing enough instrumentation so that the feed tonnage can be automatically controlled based on the pulp levels in the primary cyclone feed pump boxes and the circulating load through the mills
- Automatically controlling the pulp density in the flotation feed by automatically adjusting the water addition to the circuit based on the feed tonnage

- Addition of an Online Stream Analyser (OSA) to provide copper, iron and arsenic assays of the four process streams approximately every ten minutes. The feed and tailings samples and the entire final and scavenger concentrate flows will be pumped to a multiplexer on top of the OSA building with the accounting sample collection and sample transmission to the OSA being achieved in the same unit. The feed, tailings sample and the scavenger concentrate streams will flow by gravity from the demultiplexer to the flotation feed while the final concentrate stream will be pumped to the thickener
- Providing a pulp density measurement on the thickener underflow so that the feed rate of concentrate to the pressure filter can be adjusted to maintain a constant high pulp density in the feed which will maximize filter efficiency
- Provision of a variable speed pump and level sensor to ensure that there is a smooth flow of scavenger concentrate to the head of the circuit
- Upgrading of the Programmable Logic Controller (PLC) so that there is support for the operating system and a back-up processor to ensure that the control would remain active in the event that the existing processor fails

17.3 INPUT REQUIREMENTS

17.3.1 Process Water

The current fresh water supply for the plant relies on a 14 km line from a bore field on the plains east of the operation (Pump Station 1) and three booster pumps: Pump Station 2, on the plains, Pump Station 3 on the edge of the town (Concepcion del Oro) and Pump Station 4 in the centre of town at the old smelter site. Tailings return water from the decant system and underdrains at the disused tailings impoundment facilities flow by gravity to Pump Station 3 to join the fresh water supply. The recovered water from the new tailings area (Area 5) will be pumped to Pump Station 3.

Water from underground workings is pumped to two concrete lined decant ponds above the process plant and flows by gravity to the plant. An allowance has been made in the water balance for water from the underground which will provide a small fraction of plant water requirements. Use of this water will also depend on the quality of the water which has in the past been good but may deteriorate when mining of the higher grade and higher pyrite ores is in progress.

Table 17-3 Water Balance

	Addition Point	Water		
		tonnes/hour	USGPM	litre/sec
Grinding	Water in ore	3.6	16	1.00
	Water to Ball mill feed	**26**	**113**	**7.10**
	Water to Cyclone feed	**171**	**753**	**47.5**
	gland water (4 cyclone feed pumps)	**9**	**40**	**2.50**
	gland water (3 transfer pumps)	**5**	**22**	**1.39**
	Water in Lime slurry	**2.3**	**10**	**0.64**
	Water with Reagents	Negligible		
Flotation	Circuit Fresh Flotation Feed	217	953	60.2
	Water to Copper Conc. launder	**5**	**22**	**1.38**
	Concentrate Pump Gland	**0.5**	**2**	**0.14**
	Copper Conc.	14	63	3.96
	Scavenger Feed	219	962	60.7
	Water to Scav. Conc. Launders	**1**	**2**	**0.14**
	Water to Scav. Conc. pump gland	**2**	**9**	**0.56**
	Scavenger Conc	3	13	0.80
	Water to Scav. tails pump gland	**2**	**9**	**0.56**
	Scavenger tails to impoundment	218	960	60.6
Dewatering	Conc. thickener feed	18	80	5.06
	Thickener sprays and pump glands	**5**	**22**	**1.39**
	Conc. thickener underflow	4	17	1.05
	Conc. thickener overflow (to pond)	19	82	5.16
	Filtered Concentrate	1	3	0.19
	Filtrate (to thickener feed)	3	14	0.86
Plant Water Requirement	Total to Process	234	1029	64.9
	Less water in ore	4	16	1.00
	Total water requirement	230	1013	63.9
	contingency at 10%	23	101	6.4
	Process Water Requirement	257	1131	71.3
Fresh Water Requirement	Total to Process	257	1131	71.3
	Total from Underground Mine	**15**	**66**	**4.2**
	Total water requirement	242	**1065**	67.2
	Recovered from tailings (40% Loss) *	131	576	36.4
	Fresh Water Requirement	111	488	30.8
	Water additions are in bold type * by evaporation and incorporation in tails			

Figure 17.1 Crushing Circuit Flowsheet

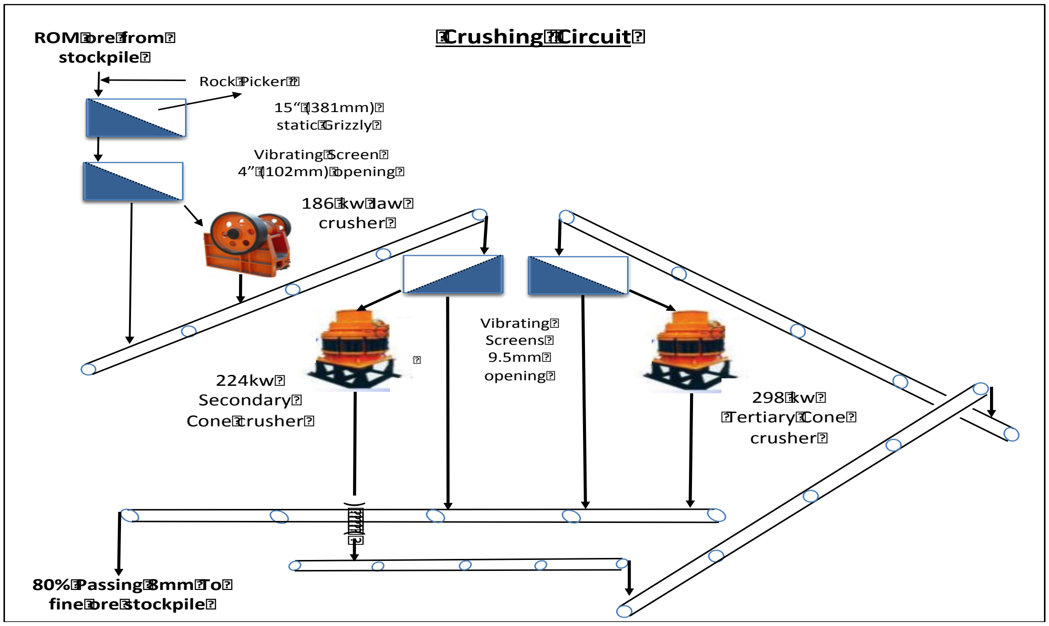
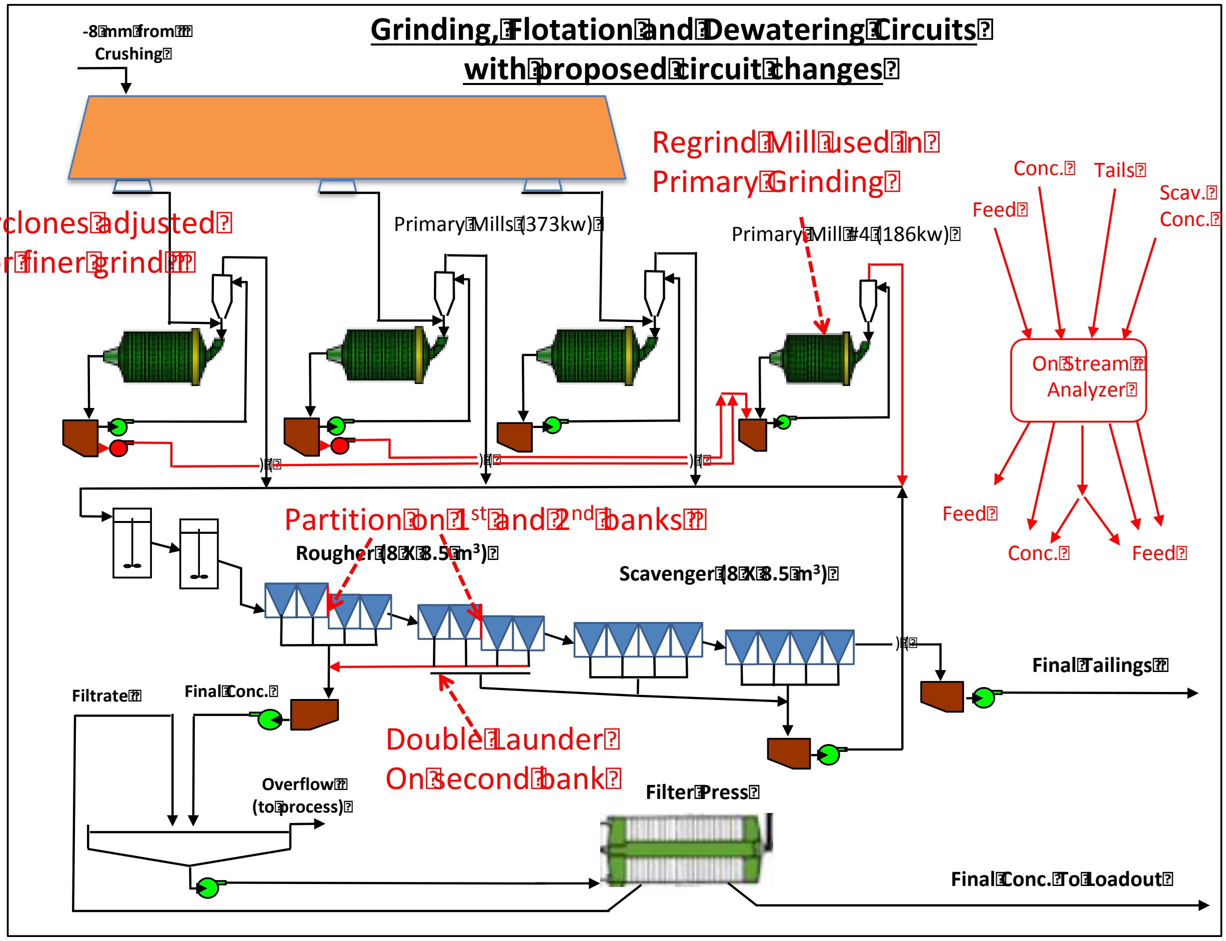
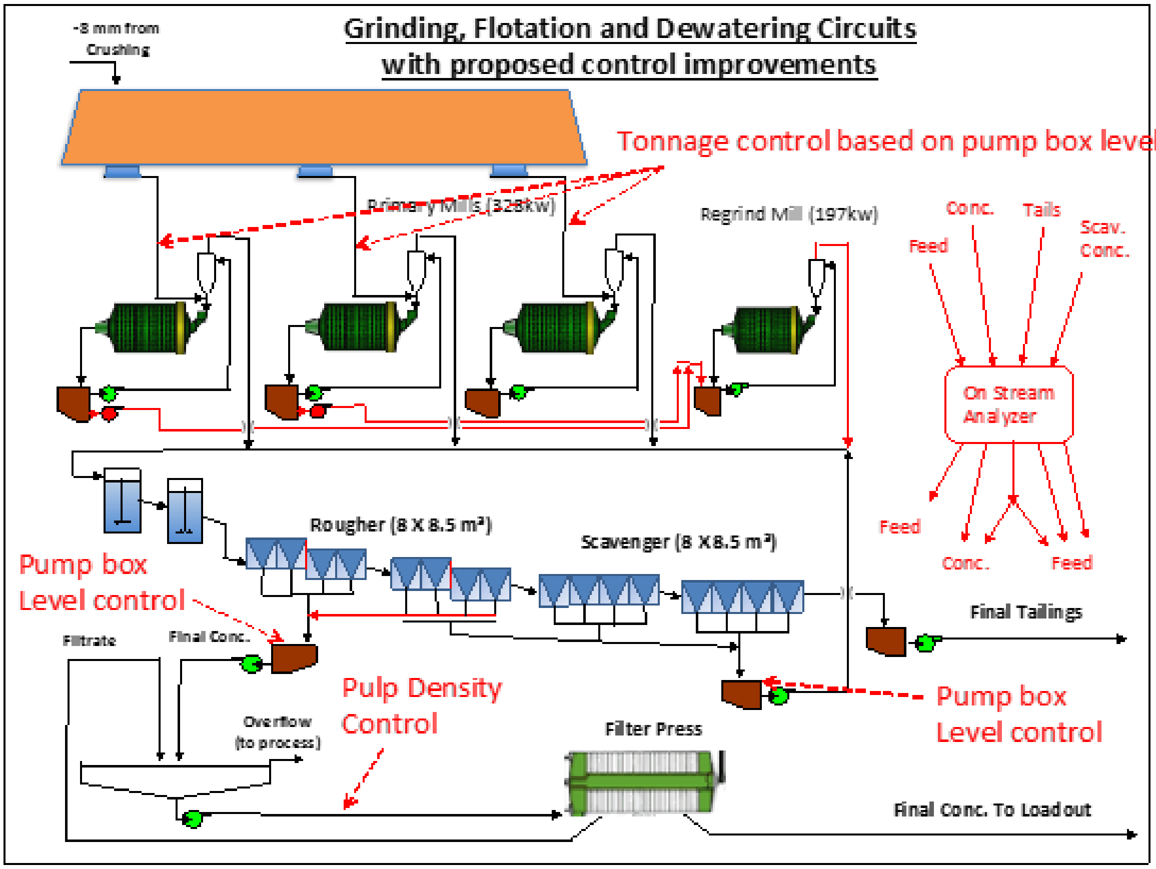


Figure 17.2 Grinding, Flotation and Dewatering Circuits with Proposed Control Improvements



Red colouration indicates where new equipment is required

Figure 17.3 Grinding, Flotation and Dewatering Circuits with Proposed Circuit Changes



Red colouration indicates where new equipment is to be installed

18. INFRASTRUCTURE

Item 18 describes the infrastructure requirements for the Project's start up. Aranzazu is an existing mine that was placed in care and maintenance in January 2015 and the infrastructure used at that time is still in good operating condition. The mine has all accesses, power and water requirements for the recommencement of operations.

The engineering activities carried out as part of the FS have revealed that the existing TD4 facility could be suitable for further tailings storage, but additional work is required to address identified risks in its foundation. Aura Minerals plans to address the recommendations provided by the engineer in record and will define whether TD4 will work as a temporary facility in the future. In the meantime, a new tailings storage facility TD5 has been designed and considered to be built to replace the existing TD4.

In addition to TD4, Aura Minerals identified the existing TD1/TD2 as another facility that could be suitable for further short-term tailings storage. However, additional work is required to condition the facility for additional tailings storage. Aura Minerals is currently performing upgrades at TD1/TD2, per recommendations described in Item 18.3.

The mine currently has all offices in good working condition for the different areas of the production chain including administrative offices, mine and processing offices, maintenance shops, laboratory, concentrate storage area and warehouse. The explosive storage area is also in good order condition.

18.1 NEW TAILINGS STORAGE FACILITY (TD5)

Wood Environment and Infrastructure Solutions, Inc. (Wood) prepared a detailed design for Stage 1 of the new TD5 tailings storage facility (TSF) to provide tailings storage of approximately 2.8 Mt for the first three years of operations. A conceptual level design was completed by Wood for expansions to the TD5 TSF by downstream raises of the tailings dams to provide approximately 10.1 Mt of total tailings storage.

18.1.1 Introduction

The tailings management design was prepared by Brett Byler, P.E. of Wood Environment and Infrastructure Solutions, Inc. (Wood). Mr. Byler visited the Aranzazu property on June 27 and June 28, 2018, and previously on May 19 and 20, 2017, and oversaw the TD5 tailings storage facility location. Additionally, Mr. Byler relied on site visits and geotechnical investigations at the TD5 location by other Wood geotechnical engineers.

18.1.2 Tailings Storage Facility TD5

Tailings will be disposed of at a new tailings storage facility designated "Tailings Dam 5" ("TD5") that is scheduled for construction in 2018. TD5 Stage 1 is designed to store conventional slurry flotation tailings deposited at a rate of 2,597 tpd that will be pumped from the process plant to TD5 (distance of approximately four km). TD5 will be formed by construction of two zoned earth fill tailings dams ("primary" and "south" dams) at the eastern side of TD5. The TD5 tailings dams will be constructed

by annual construction stages for the first three years using downstream construction methodology. Each sub-stage (1A, 1B and 1C) will provide approximately one year of tailings storage.
A conceptual level design was completed for expansions to the Stage 1 TD5 tailings dam to provide a total of 10.1 Mt of tailings storage. The expansion will be completed by sequential downstream raises to the tailings dam.

The basic engineering for TD5 was completed by SRK (2013). As part of SRK's scope of work, a geotechnical investigation campaign was carried out at TD5. A Manifesto de Impacto Ambiental (MIA) (Environmental Impact Assessment) was completed for TD5 in 2013 and approved by SEMARNAT in 2014. Tierra Group International, Inc (TGI) advanced the basic engineering design to final design in 2014 and 2015 (TGI, 2015), just prior to Aranzazu going to care and maintenance. TGI conducted a geotechnical investigation campaign at TD5 as part of their design package. In 2017, Amec Foster Wheeler Environment and Infrastructure Inc. (now Wood) prepared the detailed design package of Stage 1TD5 and conceptual design of the tailings dam expansion to 10.1 Mt tailings storage. The main components of the TD5 TSF are as follows:

- Primary and south tailings dams raised using downstream raise construction methods;
- Underdrains under each tailings dam;
- Unlined tailings storage impoundment;
- South and west diversion channels; and
- Lined primary and south seepage collection ponds located downstream of each dam.

TD5 is formed by two embankment dams herein designated as the "primary" dam and the "south" dam that abut the natural topography at the eastern limits of TD5 to form an enclosed basin for tailings storage.

The combined primary and south dam measures approximately 850 m (m) along the Stage 1C crest and have maximum heights of approximately 24 and 26 m, respectively. Tailings will be discharged from the crest of the embankment, forming a tailings pond at the southwestern area of the impoundment, away from the tailings dam. Water from the tailings pond will be recycled to the process plant by barge pumps (designed by others). The general layout of the tailings dams is presented on Figures 18-1 and 18-2.

Figure 18.1 Layout of the Stage 1 TD5 Tailings Dams
(Source: Amec Foster Wheeler, 2018)

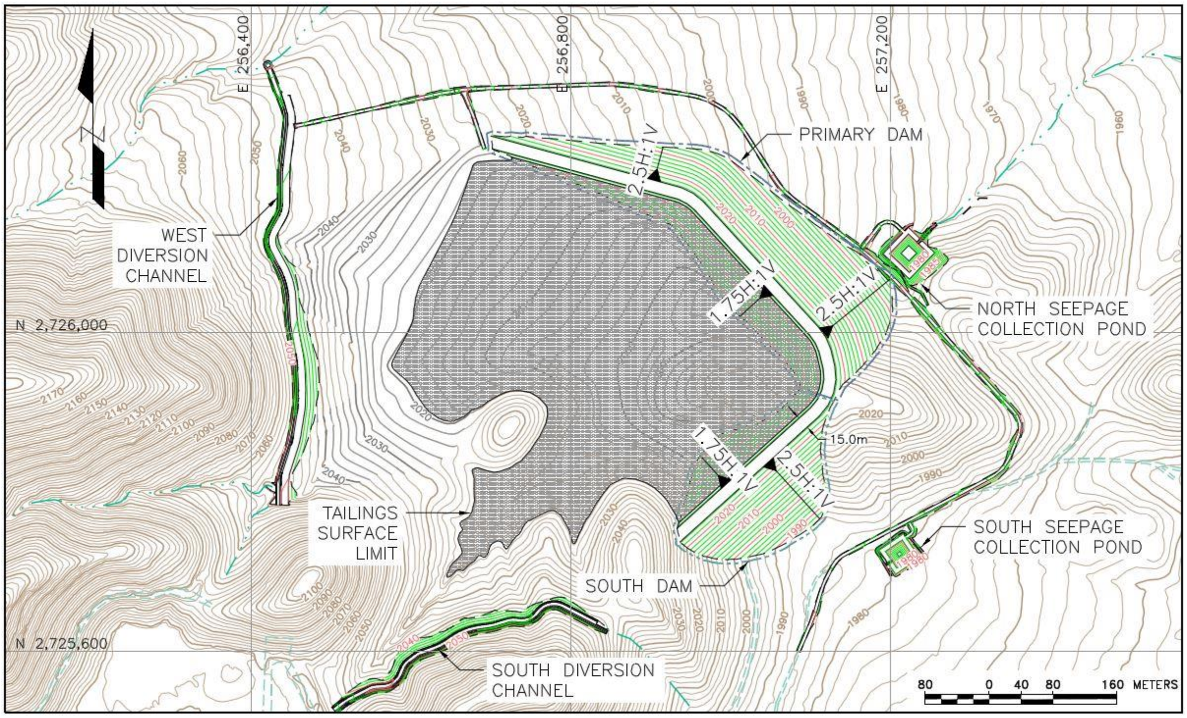
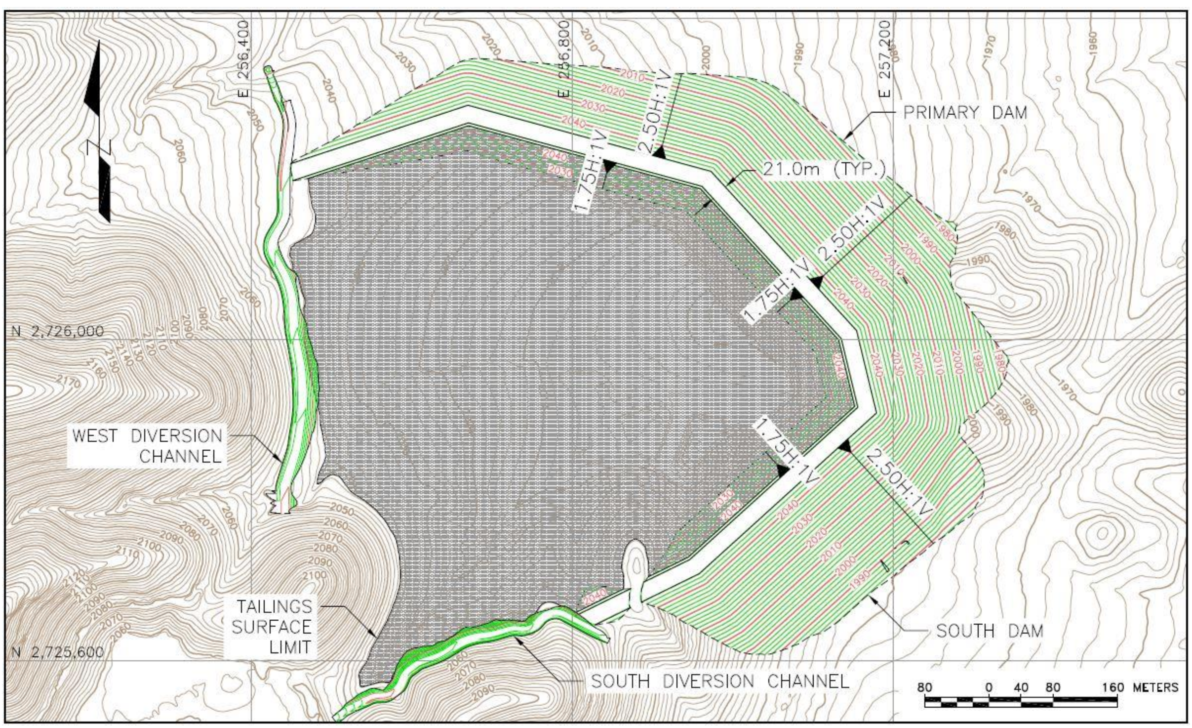


Figure 18.2 Layout of the TD5 Tailings Dams expansion to 10.1 Mt Storage Capacity
(Source: Amec Foster Wheeler, 2018)



18.1.3 Site Description

The proposed TD5 site is located approximately 3.5 km northeast of the Aranzazu process plant and approximately 1.0 km northeast of the existing TD4 facility. Elevations across the TD5 site range from approximately 1,980 to 2,050 m above sea level (masl). The area is moderately vegetated with shrubs, scrub trees and bushes and various cacti.

The existing ground surface ranges from gentle (5 to 10% slopes) within the tailings impoundment basin to steep (30 to 40%) on hill slopes on the southern and western portions of the facility. Minor ephemeral drainages cross the TD5 location at the primary and south tailings dam.

The municipality of Concepcion del Oro has operated a solid waste landfill at the location of the proposed TD5 facility. The municipal solid waste has been dumped behind an earthen embankment, ostensibly constructed to contain the waste. Few details are available as to the construction or operations of the landfill; however, it appears that the landfill is outside the limits of the proposed TD5 tailings dam footprint.

Several dirt roads currently access the landfill and cross the TD5 area. Aranzazu plans to relocate the landfill to a newly constructed facility outside of the proposed TD5 facility prior to construction of TD5. The landfill relocation will include complete removal of the waste material.

18.1.4 Geotechnical Investigations

Several geotechnical investigation campaigns, including boreholes and test pits, have been conducted at the TD5 site as follows:

- Condemnation Drilling by Aura Minerals, February 2013;
- Basic Engineering of TD5 by SRK Consulting (SRK, 2013); and
- Final Design of TD5 by Tierra Group International S.A.C. (TGI, 2015)

Wood reviewed the information from the previous geotechnical investigations and considered the investigations and methods to be appropriate. Wood also conducted additional geotechnical investigations at TD5 in 2017, including test pits to characterize the materials for dam construction and additional boreholes to characterize the hydrogeological conditions at the TD5 site (Amec Foster Wheeler, 2018).

18.1.5 TD5 Dam Design Overview

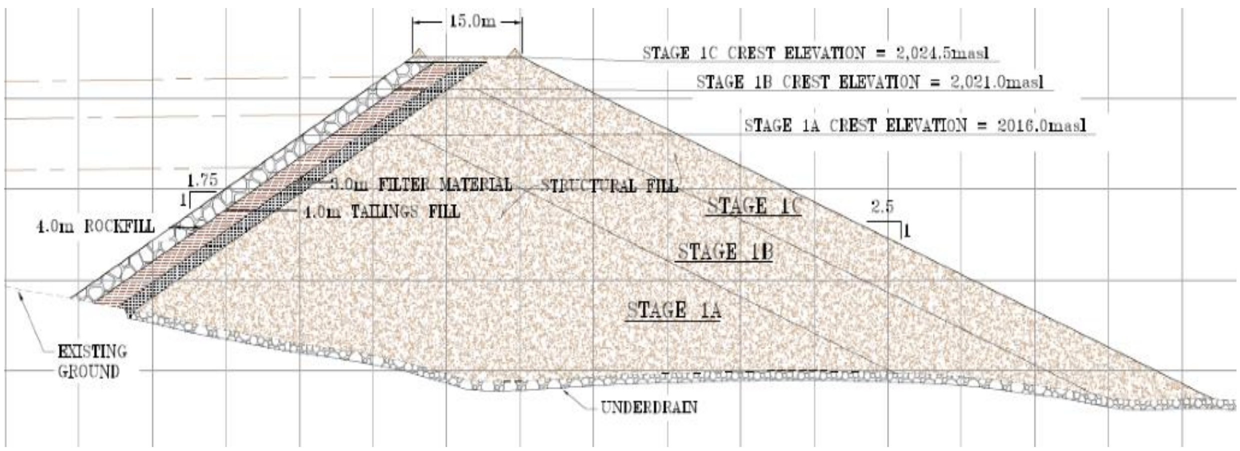
The TD5 tailings dam is a zoned earthfill embankment with 2.5H:1V (horizontal:vertical) downstream slopes and 1.75H:1V upstream slopes and a crest width of 15 m. The Stage 1 embankment construction will be completed in three construction stages (Stage 1A, 1B and 1C) using downstream raise methodology. Future expansions are anticipated to consist of sequential downstream raises of the tailings dam.

The dam will be constructed predominantly with Structural Fill material sourced from native alluvial soils from the TD5 impoundment basin. The native soils are typically composed of silty to clayey sands and gravels classifying as GM and GC according to the Unified Soil Classification System (USCS).

Three zones of fill will be constructed along the entire upstream face of the dam as follows (from upstream to downstream):

- 4 m-wide Rockfill zone (measured horizontally);
- 4 m-wide zone of compacted Tailings Fill (measured horizontally) sourced from historic tailings facilities at Aranzazu; and
- 3 m-wide zone of Filter Material (measured horizontally).
- A typical cross-section of the Stage 1 tailings dam is presented in Figure 18.3

Figure 18.3 Typical Cross-Section of Stage 1 TD5 Tailings Dam
(Source: Amec Foster Wheeler, 2018)



To control and limit the development of a phreatic surface within the TD5 dam, an inclined filter zone has been designed to provide both a filtering (i.e., particle retention) and drainage function. At the base of the inclined filter zone, a toe drain consisting of a dual-wall perforated corrugated pipe embedded in filter material will be constructed. The filter toe drains will feed underdrains at the primary and south tailings dams.

Seepage collected by the filter toe drains and underdrains will be routed by gravity via HDPE pipes to geomembrane-lined seepage collection ponds located downstream of each dam. Seepage collected in the ponds will be pumped to Pump Station No. 3 and recycled back to the process plant. Seepage water will not be discharged to the natural environment unless it meets water quality and permit requirements.

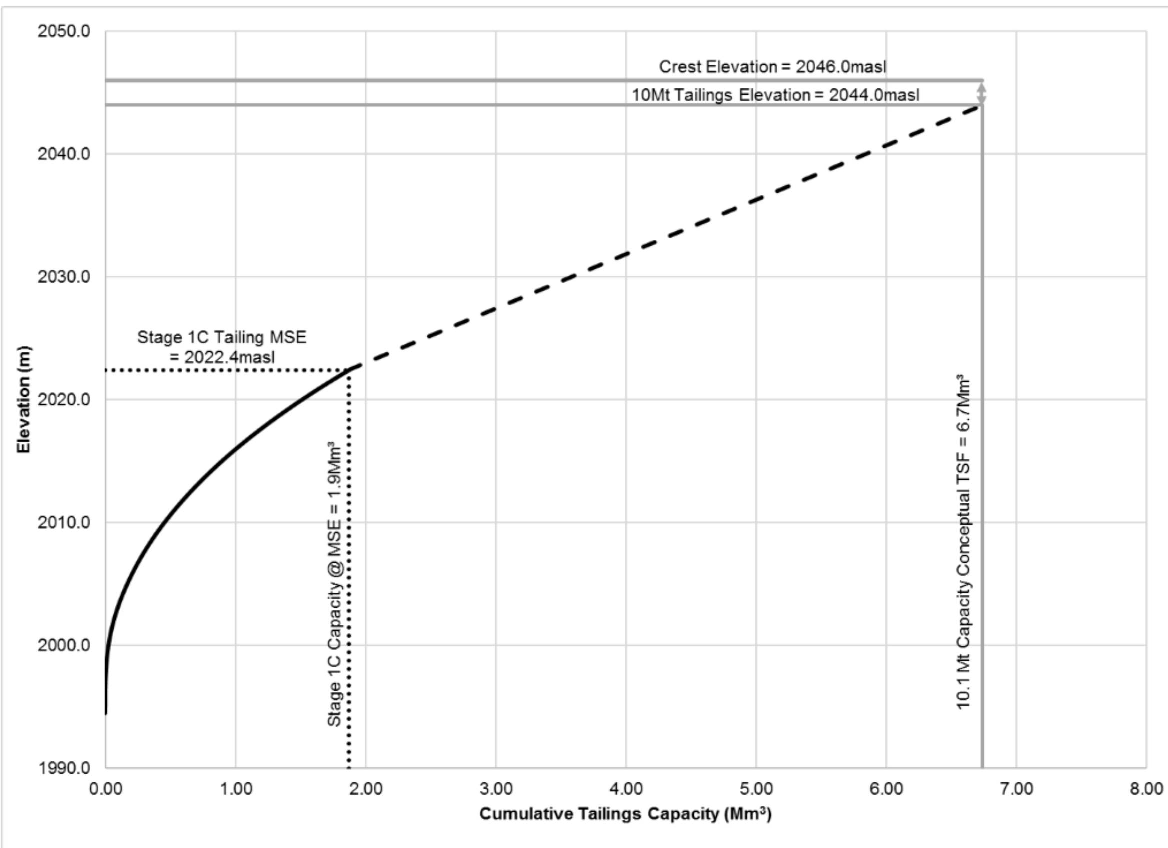
18.1.6 TD5 Tailings Storage Capacity

Table 18-1 summarizes the required tailings storage volumes for the first three years of the TD5 TSF. Figure 18.4 presents the elevation-storage curve for the Stage 1 TD5 tailings impoundment. The elevation-storage curve was developed based on an assumed 1% tailings slope.

Table 18-1 Required Tailings Storage for Stage 1 TD5

Year	Tailings Production Rate (tpd)	Annual Tailings Production (tonnes)	Cumulative Tailings Production (tonnes)	Tailings Dry Density (t/m^3)	Cumulative Tailings Volume (m^3)
1	2,600	949,000	949,000	1.5	632,667
2	2,600	949,000	1,898,000	1.5	1,265,333
3	2,600	949,000	2,847,000	1.5	1,898,000
10	2,600	949,000	9,490,000	1.5	6,326,667

Figure 18.4 TD5 Storage-Elevation Curve



18.1.7 Slope Stability

The Stage 1 TD5 tailings dam has been designed according to the following design criteria for static and seismic stability:

- Static FOS during construction > 1.3 (CDA, 2013)
- Static FOS during operations > 1.5 (CDA, 2013)

- Pseudo-static FOS >1.0 (CDA, 2013)
- Design earthquake event = 7,500-year return period = 0.121g (SRK, 2014)

The stability analyses for the Stage 1 TD5 tailings dam indicate that the static factors of safety exceed 1.3 for end of construction and 1.5 for operations. Furthermore, the Stage 1 tailings dam is stable under seismic loading conditions as evidenced by pseudo-static factors of safety greater than 1.0, although some deformations may occur during a seismic event.

18.1.8 TD5 Water Management

Two surface water diversion channels ("west" and "south" channels) will be constructed to intercept surface water runoff from watersheds upstream of the TD5 tailings impoundment. The channels have been designed to convey runoff from the 50-year precipitation event with 300 mm freeboard.

The TD5 design does not include an emergency spillway for operations, however, the impoundment has been sized to contain the normal operating pool plus the Inflow Design Flood (IDF) assuming failure of the diversion channels while maintaining 2.0 m of freeboard. The IDF was selected as 2/3 between the 1:1000-year flood and the probable maximum flood (PMF) based on criteria set forth by CDA (2013) for a "Very High" dam classification. A deterministic water balance was developed for TD5 to predict fluctuations in water volumes in the facility and ensure proper sizing of the embankments.

18.1.9 Water Reclaim System

Water will be reclaimed from the TD5 tailings pond via barge-mounted pumps at an average rate of 40 L/s (3,450 m^3/d) and recycled to pump station #3 where it will be pumped to the process plant. Wood is currently providing conceptual level design of the TD5 water reclaim system.

18.2 EXISTING TAILINGS STORAGE FACILITY TD4

TD4 is located approximately 2 km northeast of the mining operation. A historic tailings impoundment, referred to as Presa 4, forms the foundation of TD4. Presa 4 is a side hill impoundment constructed primarily through the 1990's to approximately 60 m in height using tailings and the upstream construction method.

Due to the construction of the TD4 tailings facility on the historic Presa 4, TD4 is exposed to a higher risk profile than is typically the case for more conventional tailings storage facilities. The design engineer provides Aura Minerals with the engineering design and guidance documents necessary to manage the risks associated with the construction and operation of TD4. This includes, for example, an operations, maintenance, and surveillance (OMS) manual and a schedule for regular monitoring of instrumentation, i.e. piezometers and survey monuments.

Further details regarding the history of TD4, its current condition and its availability for temporary disposal of tailings are provided below.

18.2.1 History of TD4

The conceptual design, detailed engineering design and stability analysis of the TD4 tailings storage facility “Raise” was completed in early 2010 by Promimet S.A.de C.V., a Mexican consulting firm. The initial design was based on the development of a 25 m high side hill impoundment to elevation 2,079 m over an existing 60 m high side hill tailings facility (Presa 4). The design accounted for an upstream raised facility with the outer face constructed at approximately 3.1H:1V, and constructed with compacted tailings. It also included details related to the decant line and filtration towers. SRK was retained to first undertake a senior desktop peer review of Promimet’s conceptual raise design in 2010. The SRK identified a series of design considerations which SRK recommended be addressed by Promimet as part of their final raise design (SRK, 2010).

In February 2012, Aranzazu commissioned SRK to complete a site selection study for tailings deposition beyond the operating life of TD4. This work commenced with a site visit during which SRK was asked to provide consulting advice in relation to the operation of TD4, specifically in regard to pond water clarification and dam construction/stability. Therefore, during the site visit, SRK completed a general site reconnaissance and test pits primarily focused on the eastern side of the historic Presa 4 where visible seepage was noted (SRK, 2012) along the toe of TD4. It should be noted through many of the historical documents focused towards TD4, the facility was referred to as the Pond 4 tailings facility. For consistency, this document refers to the facility as TD4; however, this is not consistent with the attachments and figures, which refer to the facility as Pond 4.

SRK subsequently undertook an additional site investigation program in order to improve the geotechnical characterization of the Presa 4 (SRK, 2013a). The program provided data from drill holes and CPT profiles which subsequently provided the basis for the detailed design of a raise of the TD4 to a crest elevation of 2,079 m. The 2,079 m design report recommended the construction of a buttress across the entire downstream face of the tailings dam, and a toe buttress to be constructed on the eastern side of the dam, to provide additional buttressing of the dam in the area where weak foundation conditions had been encountered. SRK also developed an operations, maintenance and surveillance (OMS) manual for TD4 (SRK, 2013b).

SRK subsequently evaluated the feasibility of safely raising TD4 above the current design elevation of 2,079 m to a maximum elevation of 2,088 m (SRK, 2014b). As a part of this evaluation, SRK completed a detailed assessment of the site seismic hazard, liquefaction potential of the tailings within Presa 4, in combination with the proposed raise of TD4 to 2,088 m, was assessed in greater detail.

After the project had initiated, a preliminary liquefaction assessment was completed which determined there were materials susceptible to dynamic liquefaction within the historical Presa 4. The analysis indicated that, depending on the phreatic surface within the historical Presa 4 impoundment, which had not been well defined previously, the material would liquefy based on the peak ground acceleration (PGA), which was determined to be 0.12 g.

It was hypothesized that, by dewatering the historical Presa 4 with pumping wells, it may be feasible to lower the phreatic surface to a level which would significantly limit the extent of potential liquefaction and thereby increase the stability of the facility during this transient event to a suitable level.

In July 2014, SRK was authorized to proceed with a drilling program with the intent of installing a pumping well and observation wells. However, during the course of the drilling program, insufficient water was encountered in the borehole to complete a pump test, and the focus of the investigation shifted towards installing piezometers that would provide a better understanding of the phreatic surface within TD4. The results of the drilling program were incorporated into an updated liquefaction assessment, and the results of the assessment were incorporated into the 2008 design report (SRK, 2014b).

After completing the design for an additional raise of the TD4 tailings dam to a maximum elevation of 2,088 m (SRK, 2014b), additional geotechnical characterization and hydrogeological data were collected during the course of the latest design phase, and the stability model was updated to reflect the new information. Of particular relevance was the SPT program, which focused on the eastern side of the new tailings facility foundation (i.e. within the historic tailings) and was used to increase the definition of a previously identified silty clay layer.

Previous reports indicated this area consists of weak foundation materials, specifically soft, silty clay layers consisting of hydraulically placed slimes tailings. As part of the 2,088 m raise design, SRK reviewed the stability of the dam based on the buttress construction progress to EL. 2,076, which Aura Minerals provided to SRK in July 2014. The stability analysis results for the 2,076 m dam and buttress configuration indicated the static factor of safety (FoS) met the design criterion (FS = 1.3), and the analysis was not explored further.

It was noted the material volume for the buttress and toe buttress between the 2,079 raise and the 2,082 m incremental crest elevation was relatively large given the 3 m raise, and SRK was asked to revisit and optimize the buttress volumes in an attempt to better distribute the volume (and capital expenditure) over the anticipated life of the 2,088 m dam raise. SRK subsequently reviewed the stability of the dam at 2,079 m based on the previously designed buttress configuration and, using the updated stability model (SRK, 2014b), determined that the downstream extent of the 2,079 m toe buttress would need to extend further to the south in order to provide an appropriate FoS (SRK, 2014a).

In conjunction with a separate but related scope of work, SRK was working towards updating the TD4 Operations, Maintenance, and Surveillance (OMS) Manual and recognized the 2,088 m toe buttress extended westward beyond the limits of the weak silty clay layer. The memo (SRK, 2014a) also presents a revised set of design drawings and staged material volumes for the buttress and toe buttress as the dam is raised to the 2,088 m crest elevation.

18.2.2 Failure Mode and Effects Analysis

In early 2015, TD4 entered a “care and maintenance” period. Shortly thereafter, the stability analysis was updated based on as-built topography provided by Aura and the stability analysis indicated, based on the as-built condition of the tailings buttress, the FoS of the tailings facility against undrained failure was below 1.3, the minimum acceptable safety factor (SRK, 2015). Following the 2016 Annual Inspection (SRK, 2016), it was noted groundwater elevations recorded to date may have been misreported (i.e. were reported lower than actual). Aura Minerals corrected the problem and, as of May 2017, updated groundwater elevations were provided to SRK. This resulted in some of the piezometer groundwater elevations increasing by as much as 2.5 m, and the need to update the stability analysis to confirm the increased phreatic level would not have a significant detrimental impact to the stability of the TP4 structure (SRK, 2017b).

In an effort to better assess the risk associated with TD4, Aura Minerals arranged for a Failure Modes and Effects Analysis (FMEA), which was part of the FS Study, to review TD4 with a panel of experts and included participation by representatives from Aura Minerals, SRK, AMEC Foster Wheeler and Geoconsultoria (SRK, 2017a).

The identified failure modes, effects and mitigative measures associated with TD4 concluded four failure surfaces were of primary concern. Most significantly, this included the potential for materials at the bedrock interface and toe of historical Presa 4 to fail in an undrained manner. However, insufficient data exists to determine the failure potential.

A stability analysis was completed using a range of reasonable undrained strengths for a hypothetical slimes layer to determine if an FoS could exist that did not meet the design criterion. The potential presence of a slimes layer was shown to have a considerable effect on the stability of the historical foundation (SRK, 2017b). Although the model used undrained shear strength values for relatively competent materials, in most cases an FoS of 1.3 was not achieved. This is a concern in terms of potential risk and therefore identifies a need for further investigation in the vicinity of the downstream toe of the historical tailings facility. A program to complete the investigation was developed and submitted to Aura Minerals (SRK, 2017c). As of July 2018, this program was in progress.

18.2.3 TD4 (Current Condition) and Temporary Disposal

In May 2018, Aura Minerals requested SRK evaluate the potential for TD4 to store between two and seven months of tailings production. This was based on the understanding the permitting delay will result in at least a two-month delay in the completion of TD5 construction. As Aura Minerals is proposing to recommission TD4, Aura Minerals would prefer to defer the capital cost of constructing TD5 until the mine is in operation and generating revenue. Based on the construction timeframe for TD5 (approximately five months), the permit delay would require the operation of TD4 for seven months. Aura Minerals has requested SRK consider raising the tailings facility be raised to support this full production period, if possible.

In response to this request, SRK representatives Erik Ketilson and Erick Lino visited the site on June 7 and 8, 2018. SRK prepared a memorandum outlines the path forward to recommissioning TD4.
It is essential that the buttress and toe-buttress be brought up to the appropriate design levels before either further tailings deposition occurs or the next lift is placed on the dam crest. It is SRK’s opinion that, if previous recommendations are carried out, TD4 can be recommissioned. However, prior to recommissioning, SRK recommends the following:

1. Prepare the site for buttress construction. This involves:
    - Cleaning tailings from within the drainage channels
    - Cleaning all diversion ditches
    - Raising piezometers within the buttress footprint
    - Testing the decant line to confirm that tailings have not plugged it

2. Commence buttress construction with waste rock. Construction should start immediately within the limits of Stage 1, with the rate of construction controlled by monitoring the survey monuments at the dam, and the piezometers. The construction should be focused towards extending the existing buttress downstream toe to the 2,079 m design toe limits.

2. Commence buttress construction with waste rock. Construction should start immediately within the limits of Stage 1, with the rate of construction controlled by monitoring the survey monuments at the dam, and the piezometers. The construction should be focused towards extending the existing buttress downstream toe to the 2,079 m design toe limits.

3. Procure and install inclinometers prior to construction of Stage 2 and 3 of the buttress (SRK, 2015) to monitor deformations associated with buttress construction and tailings deposition. SRK will monitor the installation of the inclinometers. The frequency of instrumentation monitoring should be as follows:
   a) Survey monuments should be monitored daily. However, it is noted that, historically, Aranzazu mine surveyors have not been able to survey these monuments to the necessary level of accuracy (millimeter precision). As such, buttress construction may be slowed or stopped if the required level of accuracy cannot be obtained.
   b) Piezometers should be monitored every two days, with the frequency increased depending on the results of the monitoring readings.
   c) Monitoring data should be reported to SRK on a daily basis.

4. Complete the site investigation recommended by SRK to obtain CPT and SPT results within TD4 and Presa 4.

These recommendations are being implemented in conjunction with ongoing buttress construction. Assuming the buttress is successfully constructed to the level currently required and allowing for the use of TD4 tailings in the construction of the TD5 dams and the TD4 buttress, TD4 has an available storage capacity of up to 259,500 dry metric tonnes. This additional storage capacity is equivalent to around 0.3 years of full production.

18.2.4 TD4 (Closure)

A detailed closure plan has not been developed for the TD4 TSF. However, current plans call for the deconstruction of the facility, and the tailings from TD4 would be used in combination with tailings from other historic facilities for on-going construction of the TD5 TSF.

In principle, this approach will help mitigate the long-term risk associated with TD4 during closure, but a deconstruction plan will need to be developed. The plan should include schematics for the progression of the deconstruction such that the deconstruction does not compromise the stability of the facility or otherwise cause it to lose containment.

18.3 EXISTING TAILINGS STORAGE FACILITY TD1/TD2

In May 2018, Aura Minerals retained Wood evaluate the potential to store up to four months of tailings production at the existing TD1/TD2 facility, due to the TD5 construction permit postponement that will result in at least a two-month delay in the completion of the TD5 construction.

In order to evaluate the potential for recommissioning tailings operations at TD1/TD2, Wood performed a geotechnical exploration campaign that included Standard Penetration Testing (SPT), Cone Penetration Testing (CPT), and laboratory testing on select recovered samples to estimate the existing material strength properties and to evaluate the stability condition at the facility under current and proposed future (after four months of tailings discharge) conditions. The results of the stability analyses at TD1/TD2 under existing conditions show that the estimated factors of safety at its current geometry did not meet the minimum required stability criteria. To achieve the recommended minimum factors of safety (FoS), for operational conditions, the construction activities listed below are required and are currently underway:

1. Flatten the overall downstream slope of the facility to an average slope of 2.1H:1V at selected segments of the facility (approximately 70% of the facility)
2. Regrade the upstream and downstream slopes of the upper embankment to 3H:1V and 2H:1V, respectively. Regrading shall be conducted by placement of compacted tailings fill as specified in the Technical Specifications included in the TD5 Detailed Design Package
3. Set the crest elevation at 2,059 m above sea level (masl) along the entire facility. This will not only provide additional storage, but also will provide a flat crest at a uniform elevation. This will be achieved by placing historic tailings excavated from the TD1/TD2 basin
4. Excavate and grade the TD1/TD2 basin to drain towards the decant system. This will facilitate the collection of reclaim water and the development of a tailings beach above water
5. After excavation, compact the subgrade throughout the entire basin per the TD5 Technical Specifications to prevent excessive seepage through the tailings / native soils foundation
6. Inspect and repair, if necessary, the existing decant pipe system prior to beginning the tailings discharge. Other water reclaim options are available in case the decant system is to be abandoned
7. Install the following instrumentation prior to recommissioning tailings deposition at TD1/TD2:
    a) Installation of six Casagrande piezometers to monitor phreatic levels within the embankment;
    b) Ten survey monuments installed on the embankment crest and slopes to monitor movement and deformation of the TD1/TD2 facility
8. Develop a tailings beach above water via operations as soon as discharge starts. A tailings deposition line consisting of spigots evenly spaced (i.e., at 20-meter spacing maximum) shall be installed prior to discharge
9. Perform standard monitoring procedures, such as daily patrols, especially along the downstream face of the facility to observe signs of deformation, saturation, seepage, or settlement

18.3.1 TD1/TD2 - Closure

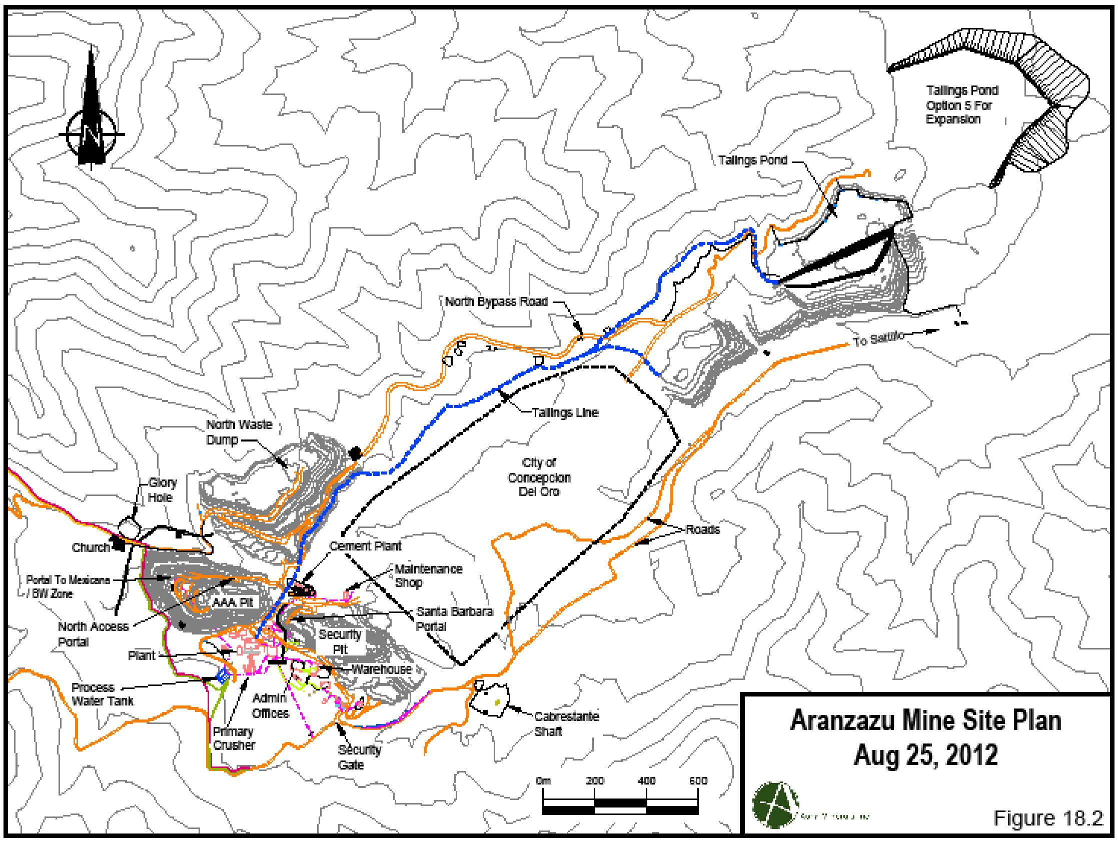
A detailed closure plan has not been developed for the TD1 / TD2 facility. However, Aranzazu plans to develop a detailed TD1 / TD2 closure plan after tailings discharge stops at the mentioned facility.

18.4 ROADS

The town of Concepcion de Oro lies 2 km west of the paved Mexican state Highway 54 between Saltillo and Zacatecas. Saltillo is approximately 120 km to the north and Zacatecas is 260 km to the south. The 2 km access road is paved to the town.

The Aranzazu Project is located approximately 1 km from the town by road, of which 500 m is paved before turning into a gravel road maintained by the mine. At one time, the only access to the mine was through the town. Given the high frequency of deliveries of goods, supplies and personnel transport through the town, the Mine has built a 1.7 km long road called the "North Bypass Road", as shown on Figure 18.5.

Figure 18.5 Surface Site Plan



18.5 CONCENTRATE TRANSPORT

The concentrates from Aranzazu will be trucked 900 km to the Mexican port of Manzanillo via Highway 54, where a concentrate trader blends as required to then send by ship to smelters located in China, India and Europe. The ocean freighters are typically the Handy and Handymax vessels, carrying concentrate weights up to the 28K and 58K dead weight tonnage (DWT) respectively.

18.6 BUILDINGS

Following is a list of existing buildings at Aranzazu. Some of these buildings, such as the engineering and administration buildings, will be enlarged to accommodate the expansion:

- Administration office, which houses purchasing, accounting, environment and safety departments as well as human resources and the General Manager
- Mine dry, engineering office and mine operations complex
- Warehouse and outside laydown area
- Assay lab
- Mobile equipment repair shop
- Various Contractor offices – portable trailer

18.7 FUEL STORAGE

The fuel distribution supply is by a local contractor. The diesel fuel storage tanks are as follows:

- 30,000 L
- 15,500 L
- 6,000 L

18.8 WORKSHOPS

There are currently five surface workshops:

- Mobile equipment shop
- Services – electrical / plumbing
- Plant maintenance
- 2 Contractor maintenance shops

There are currently two underground workshops:

- Mobile equipment
- Mine services – pumps / electrical / welding

A modern fully-equipped workshop has been planned for the 1880 level of the GHP zone.

18.9 EXPLOSIVES MAGAZINE

On surface there are two explosives magazines for packaged explosives totalling 43,000 kg of capacity and a detonator magazine of 200,000-unit capacity.

In the underground mine, there are plans for two explosive magazines with a total permitted capacity of 22,500 kg. The appropriate location of these magazines is yet to be determined.

For the expansion project, additional explosive space is not required since the open pit mining rate will be diminishing considerably as the underground mining rate ramps up.

18.10 POWER

Power to underground is supplied from the main 37 kVA surface substation located next to the concentrator. Underground substations step down 4160 V feed voltage to 440 V for all mining equipment, pumps and fans. At the time of writing, one active substation serves the AA mining area and a second substation is under construction for the BW and Mexicana areas. Substation feeders descend from surface via boreholes.

Permanent production loads being added during the mine expansion include two primary ventilation fans located in the BW and AA zones, the 1940 level pumping station, the 1880 level workshop, and the development and mining equipment in these areas. The primary ventilation fans will be powered from AA (existing) and BW (under construction) substations, whereas two new local substations will be required for the 1940 level pumping station and 1880 level workshops respectively.

The connected load for all underground electrical equipment (including ventilation, pumping mining equipment and infrastructure) has been calculated at approximately 5.7 KW with a peak demand load estimated at 4.5 kW. Based on the mine plan and anticipated usage of the electrical equipment, the projected power consumption for the underground (at full production) has been estimated at 3.1 MW-hrs per month.

Aura Minerals' plans to build a dedicated 6 km medium tension power line to Aranzazu; this new line will improve the voltage quality and also, support future increases in UG development and production. The capital allowance included for this specific project is 1.5M USD, estimate provided by Mr. Raul Botello, who is Aura Minerals' energy consultant in Mexico.

18.11 WATER

Fresh water is sourced from water wells about 14 km away from site. The fresh water system is comprised of four Pump Stations (PS). Fresh water is transported from PS1 to an intermediate booster PS2 and then to PS3 located at the toe of the old TD4 tailings dam. Fresh water is mixed with tailings reclaimed water in PS3 and then pumped to an intermediate PS4 before water arrives to the plant's storage ponds.

A new tailings thickener will also improve the recovery of water from tailings to the extent that the existing fresh water supply system does not require any capacity upgrade. Water from underground mining is another source of supply. A capital allowance of US$3.25M was included for the procurement and construction of a new tailings thickener to provide additional water buffer capacity.

18.12 VENTILATION

Ventilation to the mine is provided to the mine by a single 2.35 m diameter, 300 kW axial fan installed at the bottom of the Robbins #1/2 ventilation raises. Fresh air is brought into the mine from the two main surface portals along with several smaller surface raises. The fresh air is fed to the various working areas of the mine by smaller secondary fans and then directed to the exhaust raise. A second 2.35 m diameter, 300 kW axial fan was purchased which was to be installed on top of the Robbins #3 ventilation raise. This fan will be required to be installed to provide the necessary airflow to support the full mining operations.

18.13 COMPRESSED AIR

Compressed air is used for the jacklegs in underground drilling. Currently the compressor is located on surface. A 10 cm steel line is used to reticulate the air into the mine. Provisions for mobile compressors, as the mine expands, have been made in the proposed budget.

18.14 COMMUNICATIONS

Underground communication is by radios supported on a leaky feeder trunk system. Surface communication is by phone and radios. There are also provisions in the budget for purchase of a control system for the main equipment and mill circuits.

19. MARKET STUDIES AND CONTRACTS

Support for the sales and marketing of the concentrates to be produced from Aranzazu was provided by Exen Consulting Services ("Exen"). Exen's conclusions are summarized below.

19.1 OVERVIEW

Aranzazu will produce a complex copper-gold concentrate. The longer-term demand outlook for copper concentrates is favourable, with growing demand for copper matched by only limited mine supply growth.

A multi-year sales contract for 100% of the mine's output has been agreed with a major international trading house. Based on transportation logistics, the concentrates will be delivered to a warehouse in Manzanillo, Mexico after which they will be loaded onto vessels for shipment to buyers in Asia and elsewhere.

Metal prices used in derivation of the NSRs were as follows (all in US$):

Cu: US$3.00/lb ($6,613.87/t)
Au: US$1,280.00/oz
Ag: US$18.00/oz

19.2 CONCENTRATE TERMS

The mine is expected to produce approximately 55,000 to 60,000 DMT per annum of concentrates, on average, over the LOM. Although the concentrates contain deleterious elements, such as arsenic and bismuth, for which penalties will apply, strong buyer interest in the Aranzazu concentrates was seen during the sales and marketing process.

Using the metal prices as above and projected average concentrate grades, the following values for the payable metals and deductions were derived from the NSRs based on the contractual terms agreed with the Aranzazu concentrate off-taker (all figures in US Funds/DMT concentrate):

Payable Metals - Copper/Gold/Silver: US$1,991/dmt
Treatment/Refining Charges: US$167/dmt, DAP Manzanillo
Penalties: US$164/dmt

19.3 CONCENTRATE TRANSPORTATION LOGISTICS

The concentrates produced at Aranzazu will be loaded into trucks at the mine site for shipment to the buyer's warehouse in Manzanillo, Colima, MX, a distance of approximately 880 kms. Transportation costs for shipment/delivery of the concentrates to the buyer's warehouse in Manzanillo are projected at US $62.70/DMT.

All costs from Manzanillo onward, including material handling, vessel loading and ocean freight to the receiving smelter(s), will be borne by the buyer.

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 INTRODUCTION

The objective of the environmental review is to broadly assess the landscape associated with the mine re-opening, and to look for key issues and opportunities. It does not consist of a detailed description of potential environmental impacts. The descriptions found herein are based on the review of available project information, a review of project-collected data, a 2017 site visit and discussions with the Project team.

The review of environmental, permitting and socioeconomic factors principally involved the following considerations:

- Additions to Project footprint: construction of a new tailings storage facility (TD5);
- Management of mining waste products: tailings and waste rock
- Water Management: fresh water requirements, storm water management and diversions, water discharge
- Other Emissions and Effects (i.e. subsidence, air emissions)
- Permitting and regulatory requirements of Project
- Local socioeconomic effects of Project
- Closure: allowances for closure of existing site and for modifications resulting from the Project

20.2 CURRENT SETTING

20.2.1 Regulatory Landscape and Existing Authorizations

Environmental regulation for mining in Mexico is primarily governed by federal authorities, including SEMARNAT (Secretaria de Medio Ambiente y Recursos Naturales), CNA (Comisión Nacional de Agua), PROFEPA (Procuraduria Federal de Proteccion al Ambiente), and the Health Secretariat. Depending on local regulatory framework, there may be municipal permits with respect to non-hazardous solid waste management.

Much of the control over environmental activities is governed by the LGEEPA (Ley General de Equilibrio Ecológico y la Protección al Ambiente), passed in 1998 and last updated in 2015, and the Ley General para Prevención y Gestión Integral de Residuos, passed in 2003 and last updated in 2015. Since 2000, numerous regulations and standards to apply these laws have been passed, and generally speaking, the requirements are similar to those found in the U.S. or Canada. A Norma Oficial Mexicana (NOM) is a technically-based enforceable standard, and a key NOM relevant to the Project is NOM-141-SEMARNAT-2003, “Procedures and specifications for characterizing tailings, and specifications and criteria for the characterization and preparation of the site, design, construction, operation and post-operation of tailings impoundments”.

A new operation is required to have key approvals from SEMARNAT prior to construction: 1) approval of their Manifiesto de Impacto Ambiental or MIA (Environmental Impact Assessment), and 2) a Risk

Study. As soon as it is in operation, a mine submits certain operational and environmental performance data, and providing the information is consistent with that submitted in the MIA, SEMARNAT issues a Licencia Ambiental Única (LAU) within six to seven weeks.

A modification to a mining operation also requires approval, principally by way of submission of a Manifiesto de Impacto Ambiental Particular or MIA-P.

There is currently no requirement under Mexican legislation for posting of performance or reclamation bonds.

20.2.2 Natural Setting

The Project area is located in the Sierras Transversal geographical sub province, part of the central altiplano, or high plain of Mexico. Soil is of mainly residual origin with alluvial deposits found to a lesser extent. In most cases, the soils contain some salt concentration so that their fertility is low and their potential use is limited. Item 5.2 of this Technical Report provides additional information on the local setting.

As noted in the 2012 Technical Report, the Project is not located within any protected natural areas, priority terrestrial regions or areas of importance for wildlife conservation according to the publications by the National Commission for Knowledge and Use of Biodiversity (CONABIO).

The vegetation is best represented as a transition ecotone between two vegetation types: Rosifolius Scrublands and Microphyllous Scrublands with areas of natural grassland.

The Project's approved environmental impact assessment, compiled in 2010, concluded that there are no biologically significant areas affected by the Project.

20.2.3 Socioeconomics

Of a total of 8,748 inhabitants older than the age of 12 in the Municipality, 4,113 form the Economically Active Population (EAP), of which 4,078 are employed and 35 are idle (Corporacion Ambiental de Mexico, 2010).

EAP distribution by sector of activity in the municipality is as follows: primary sector (agriculture, livestock, forestry, fishing and hunting) accounts for 17.04%, the secondary sector (industrial) accounts for 37.12% and the tertiary sector (services) accounts for 42.17%.

The main industry in the State of Zacatecas is mining, largely concerned with the extraction of silver, gold, mercury, iron, zinc, lead, bismuth, antimony, salt, copper, quartz, kaolin, onyx, limestone, cadmium and wollastonite. The State comprises 13 mining districts, the most important of which are Zacatecas, Concepcion del Oro, Sombrerete, Chalchihuites, Mazapil and Noria de Angeles.

In the municipality of Mazapil adjacent to Concepcion del Oro, approximately 20 km west of Aranzazu, lies Peñasquito, a mining complex owned by Goldcorp, a Canadian company.

20.2.4 Existing Site

Aranzazu is a brownfields site and mining of the existing deposits has been carried out in several campaigns since 1962, with mining activity in the district documented as early as 1548.

The Aranzazu mine is immediately adjacent to the town of Concepcion del Oro, and most of the families have had a historic connection to mining. In the late 1800's the Mazapil Copper Company Ltd. constructed a large mill facility and smelter to process ore from the district. The remnants of the smelter complex, closed in the 1980's, still remains in the centre of the town. Several of the town's soccer and baseball fields are now situated on top of the smelter's large slag deposits.

As noted in Item 6, Aura Minerals acquired 100% of the Aranzazu mine in June 2008 and with this transaction acquired ownership and responsibility for older workings including abandoned shafts, the north waste rock pile, an abandoned oxide leach site, water pumping and conveyance systems, and a series of smaller tailings impoundments (TD1 through TD4, and historic TD5).

20.2.5 Background Metal Levels

Soils and stream sediments in the region tend to contain anomalous levels of metals including As, Cd, Pb and Sb. As noted in Item 20 of the 2012 Technical Report prepared by ERM Canada Corp., there is a background contamination of arsenic in the soil deposits of the region (AMC Mining Consultants, 2012). This is well documented within publicly available information. Concentrations of arsenic are also particularly high in the historical tailings that have been deposited in various areas on the Property throughout the course of the mine's history.

ERM notes that assays of the historical tailings have yielded heavy metals concentrations of up to 4,736 mg/kg of Cu, 1,264 mg/kg of Zn, 610 mg/kg of As, 230 mg/kg of Pb, 8.23 mg/kg of Cd. Concentrations of up to 4500 mg/kg of Cu, 500 mg/kg of As, 600 mg/kg of Pb and 1350 mg/kg of Zn in the surrounding soils and the sediments of the "El Principal" creek were reported by Castro-Larragoitia (2002).

Static assays performed on the tailings from the various deposits at Aranzazu in April 2010 yielded results ranging from 42 to 600 mg/kg of arsenic; 2017 tailings samples indicate a tighter range of 351 to 492 mg/kg arsenic. Leachate extractions on seven samples, however, show concentrations in the 0.0005 to 0.0813 mg/L range. This is well below maximum permissible levels of 5.0 mg/L for waste materials per Mexican regulations (NORMA Oficial Mexicana NOM-052-SEMARNAT-2005).

ERM also reported that sediment sampling was also conducted in the "Los Coyotes" stream which captures natural run-off and is also used by the town to discharge untreated sewage. Arsenic concentrations in the sediments of that stream were found as high as 545 mg/kg, consistent with the background concentration levels in soils discussed above. It is to be noted that some of the soils, sediments, and tailings in the region exceed threshold arsenic concentrations for residential and industrial use as specified in the Mexican legislation, NOM-147-SEMARNAT/SSA1-2004, 22 mg/kg for residential use and at 200 mg/kg for industrial use.

20.2.6 Water

Water for the Project's operation will be supplied from three sources:

1. Water from underground workings will be pumped to the process water pond for utilization in ore processing.
2. Well water from the "El Salero" wells, located 15 km away. This water will be stage-pumped to the processing plant through four pumping stations.
3. Reclaim water from the tailings impoundments will be conveyed to Pump Station 3 and will be mixed with fresh water from the Salero wells and stage-pumped through Pump Station 4 to the process water pond.

Water from the historic "Socavon" mine (underground gallery) will also be pumped by Aranzazu but only used to supply water to the town of Concepcion del Oro. The water is extracted with a submersible pump and transported by gravity to the process water pond and through a different pipe to the water storage tank for the town.

During operation Aranzazu conducted water quality monitoring since 2009 at seven surface water locations (including upstream and downstream of the town in the El Principal creek) and five groundwater locations (including in the El Salado well and within the tailings management facility).
The results obtained from the water sampling demonstrate that the water collected in underground workings can have elevated concentrations of arsenic (up to 15 mg/L). For reference, this value considerably exceeds the Mexican norm for discharge of water to water bodies NOM-001-SEMARNAT-1996 of between 0.2 mg/L (monthly average) and 0.4 mg/L (daily average). The anomalous arsenic concentration is linked to exposure to arsenic rich rock formations.

20.2.7 Site Monitoring

Prior to suspension of operations in early 2015, the site maintained an environmental monitoring matrix for operations including quarterly water quality sampling, and quarterly dust monitoring (PM10). Environmental inspections included pit diversions, tailings impoundments, tailings pipelines, solid waste, and septic system. Since that time environmental monitoring has been suspended and the frequency of inspections and regulatory reporting was not ascertained.

20.3 CONSIDERATIONS FOR THE PROJECT

20.3.1 Mine Workings and Resource to be Mined

The resource to be mined as part of the Project is contiguous with the areas mined previously. The Project will also involve construction of a new access ramp. All underground mining has been previously permitted.

Minimal risk of surface subsidence is anticipated from continued underground mining at Aranzazu. The mined stopes will be backfilled with either cemented rock fill or consolidated waste rock, leaving minimal voids underground after mining. Geotechnical studies indicate that the deposit and surrounding rock mass becomes more competent with depth. The application of good blasting techniques and effective ground support methodologies will assist in controlling the stability of the underground openings which will help in minimizing the risk of surface subsidence.

20.3.2 Waste Rock Management

No new waste rock storage facilities will be required for the Project. The estimated 1.5 Mt of waste rock produced during development and mining will be utilized for stope backfill. In addition, a further 1.5 Mt of waste rock is projected to be required for stope backfill and would be sourced from elsewhere, presumably from existing waste rock piles. The preparation of waste rock for backfill material (e.g. excavation and physical preparation) as well as waste rock scheduling still requires study, along with estimations of respective airborne emissions.

Geochemical testing of Aranzazu waste rock includes 25 drill core samples analysed in 2010, and seven test pit samples from the North Waste Rock pile analysed in late 2017 (Amec Foster Wheeler, 2018). Acid base accounting results indicate non-acid generating potential (NP:AP ≥ 3.0) for 16 of the 25 samples from 2010 and 6 of the 7 samples from 2017. Potential for net acidity (NP:AP < 1.2) was indicated for four samples from 2010 and one sample from 2017. The four samples from 2010 were comprised of intrusive and endoskarn rock types.

Mineralogical testing six of the 2017 samples indicated calcite content ranging from 3.0% to 91.0% and pyrite ranging from <1.0% to 4.0% by weight. Extracted solution from six waste rock samples tested in 2017 showed contaminant concentrations well below the stipulated limits for waste products (NORMA Oficial Mexicana NOM-052-SEMARNAT-2005). Nonetheless, AMEC Foster Wheeler note that some concentrations of certain elements that exceed the U.S. Environmental Protection Agencies (EPA) Maximum Contaminant Level (MCL), which is the legal threshold limit on the amount of a substance that is allowed in public water systems under the Safe Drinking Water Act.

Waste rock is therefore considered to have minimal potential for generation of net acidity, though having potential for solubilizing of some metals on contact with water.

20.3.3 Ore Processing

As discussed in Item 17.1, average process plant throughput will remain unchanged at 2,597 tpd, with no major changes to equipment, overall flowsheet, reagents or facility footprint.

20.3.4 Water Management

Water Use: Fresh water for processing and other Project needs are expected to be within the 1,081,495 m^3/year allowed under the existing water concession for the operation's wells near El Salero. Fresh water makeup requirements are expected to be highest during the first start up months of the Project, and during operation will vary depending on the contribution from mine dewatering. Aranzazu will continue to supply water from the Sovacon workings uphill of the Project to the town of Concepcion del Oro, using a stand-alone water pumping and conveyance system.

Mine Dewatering: mine dewatering outputs feed into the process circuit. As noted in Item 16.3, although no Project-specific dewatering studies have been carried out, dewatering volumes are expected to increase with depth. As noted in Item 16.3, mine dewatering capability is sized up to a maximum of 25 L/s.

Water Diversions: the existing diversion of El Principal creek above AAA pit will remain operational for the duration of the Project. A stormwater diversion system will be constructed above the ultimate elevation of TD5.

Discharges: there are no process discharges from the Project. Aura Minerals advises that under current authorizations, excess water from underground mine dewatering may be released to Los Coyotes Creek. The authorizations stipulate that mine water is to be monitored and characterized before and after initiating mining activities, and during the operating life of the mine. Results are to be presented in the bi-annual reports submitted to PROFEPA. Mine Domestic sewage undergoes basic treatment prior to discharge. Based on water quality analyses in El Principal and Los Coyotes creeks, the surface water quality is considered poor. The town of Concepcion del Oro discharges untreated municipal sewage to the Los Coyotes creek.

20.3.5 Tailings Management

As described in Item 18.1, the Project involves construction of a new tailings storage facility, “TD5”. The TD5 tailings impoundment site is favorably situated on a hillside well above the regional groundwater table (12 m to 33 m below the existing surface based on 2017 boreholes in the area of TD5). There are no perennial surface water sources in the area of the impoundment, however intense rain events can result in surface flows. Allowances to accommodate and/or pass the requisite 50-year recurrence interval, 24-hour duration storm events have been included in design of TD5.

The TD5 facility has undergone design improvements since it was first permitted in 2014. The updated design incorporates downstream dam construction methodology as well as zoned earth fill embankments with internal drainage to control the phreatic surface in the embankment and enhance stability. Collected seepage will be routed by gravity via HDPE pipe to geomembrane-lined seepage collection ponds located downstream of each of the facility’s two dams, then pumped to Pump Station No. 3 and recycled back to the process plant.

The TD5 Stage 1 design basis includes a deterministic water balance to ensure that a minimum 2.0 m freeboard will be maintained under extreme precipitation events, to predict fluctuations in water volumes in the facility; to ensure proper sizing of the embankments; and to estimate the magnitude of a deficit or excess water in the system.

Construction of TD5 will require clearing of vegetation. An ad-hoc municipal deposition/ recycling/burning operation has been carried out near the TD5 footprint for the past five to ten years; the site was visited in 2015 and test pit photographs from October 2014 reviewed; indications are that the solid waste is of solely domestic origin with no evidence of hazardous or industrial waste seen. Surveys at that time estimated 35,000 m^3 of waste. The landfill site appears to be outside the limits of the proposed TD5 tailings dam footprint, and Aura Minerals indicates that an allowance for removal and relocation of this material is included in the tailings construction costs.

Dam construction also entails utilizing tailings from some or all of legacy impoundments TD1, TD2,TD3, TD4 and/or TD5 (old); this is expected to have an overall positive effect by reducing the residual volume of tailings stored in these older impoundments.

As noted in Items 18.2 and 18.3, three existing tailings storage facilities (TD4, TD1, and TD2) offer additional storage capacity of up to 565,500 Dmt. Aura Minerals advises that as of July 2018 engineering review of these three facilities and buttress construction for TD4 are currently underway.

Geochemistry of tailings was studied in 2010 in accordance with requirements of Mexican tailings standard NOM-141-SEMARNAT-2003, and during 2017 as part of an assessment of materials for construction of TD5 dams (Amec Foster Wheeler, 2018). For the 2010 study, 18 metallurgical test samples were analysed for acid-base accounting and total metals. From the 18-sample suite, three composites were prepared to represent distinct ore zones and analysed for extractability of metals and metalloids. In 2017, samples from six test pits excavated in TD2, TD3, TD4, and old Tailings Dam 5 were analysed for total metals, net acid generation (NAG), and acid-base accounting. Four of these samples were then tested for leachate extraction via the meteoric water mobility procedure (MWMP), mineralogical composition via x-ray diffraction.

Results from 2010 testing indicated a relatively wide range of reactivity and potential for net acidity, with nine of the 18 samples showing low potential for net acidity (NP:AP $\geq$ 1.2) and nine samples showing some net acidity potential. Extracted solution from all three of the composite samples showed contaminant concentrations below the stipulated limits for waste products (NORMA Oficial Mexicana NOM-052-SEMARNAT-2005)

Testing from 2017 yielded similar results to the 2010 program. Five of the six samples showed low potential for net acidity (NP:AP $\geq$ 1.2), while one sample yielded a NP:AP value of 0.52 and pH of value of 3.69, indicating net acidity production. Leachate extraction concentrations were in similar range to those from the 2010 testing. Mineralogical testing indicates that calcite content in tailings typically comprises at least 10% by weight, with pyrite content varying from 3.0% to 10.0% by weight.

The tailings are therefore considered to have some potential for net acidity, and some potential for metal-leaching due to sulphide mineral content. Most tailings contain sufficient calcite and low sulphide mineral content such that production of net acidity is improbable, although tailings containing lower amounts of calcite or elevated pyrite may generate localized acidity if deposited in isolation. In their 2018 assessment, Amec Foster Wheeler conclude that leachate testing results show that *"several of the COPCs [constituents of potential concern] have the potential to be solubilized and leached from the material on contact with water. We recommend that runoff and leachate from the tailings impoundment be monitored closely to determine COPC leachability under site conditions."* Nonetheless, it is noted that transport mechanisms for potential contaminants likely are relatively limited given the local semi-arid climate and siting of the impoundments, in conjunction with design elements (such as seepage collection during operation and reducing potential for storm water ingress and wind transport of tailings following closure).

Three new groundwater wells were installed at the TD5 site in 2017 to acquire hydrogeological data and to develop an environmental baseline of groundwater conditions to support the TD5 design.

The TD5 Operations, Maintenance and Surveillance (OMS) Manual and Emergency Action Plan (EAP) will be developed and reviewed by the Engineer of Record.

Closure objectives for TD5 are to make the facility physically and chemically stable, safe, and non-erodible for the long term, and major components likely to include: i) construction of a durable and chemically stable cover system, ii) construction of a closure spillway, iii) grading to promote positive drainage towards a sediment control pond and closure spillway, iv) monitoring water flowrates and quality from the subdrainage system at the seepage collection ponds and determining management strategy, and v) post closure inspections and monitoring. Permit requirements for the previous TD5 approval also call for revegetation with locally native species. Soil development in the area is relatively poor, but to the extent possible, during construction the upper horizon of the soil surface will be stockpiled for future reclamation use.

20.3.6 Other Emissions

Management of air emissions in the form of fugitive dust will be an ongoing consideration particularly during operational stages. Potential sources include: 1) non-paved major thoroughfares such as haul roads and the bypass road to the site, 2) backfill preparation activities such as rock screening, and 3) wind entrainment of tailings. Mitigation incorporates standard dust abatement procedures on roadways, appropriate site selection for backfill preparation operations, and stabilization of tailings surfaces upon closure. Construction activities for TD5, may require additional dust abatement measures.

With the transition to a fully underground operation, noise and vibration emissions are expected to be lower than the mixed underground and open pit activities in recent years.

20.3.7 Permits and Authorizations

Review of the existing Aranzazu operation permits focussed on the major authorizations required for construction and operation of the facility and is not considered a comprehensive review.

A principal operating permit is approval of the operation's environmental impact statement (MIA) by SEMARNAT. The operation's overall MIA was approved in October 6, 2010 and expires October 6, 2020. This permit provides general authorization for the mining, mineral processing and associated activities described in the MIA document. According to the terms of the permit, SEMARNAT may extend the validity of the authorization beyond 2020 providing that Aranzazu submits an extension request at least 30 days prior to permit expiry along with validation from the enforcement agency (PROFEPA) that the company is up to date in compliance with the permit conditions. Since 2010 five MIA-P documents have been submitted for modifications of the operation and duly approved. Of relevance to the Project is a 2014 MIA-P resolution approving construction of TD5, based on SRK Consulting (Canada) report, 2013. Due to new design modifications and changes in the facility footprint, updated authorizations for TD5 are required prior to its construction and operation, and receipt of these approvals is expected before the end of August 2018. Aura Minerals advises that application for change of land use (Cambio de Uso de Suelo) for an additional area to accommodate seepage collection ponds at the dam toes has been submitted and that approval is anticipated before the end of July 2018. The updated facility-specific environmental impact assessment for TD5 Stage 1 (2.81 Mt capacity for an estimated three years of operation) is to be submitted to SEMARNAT in mid July 2018, with approval anticipated in mid to late August.

In order to provide sufficient tailings capacity for the Project beyond year three, Aura Minerals will be required to compile and submit design and environmental assessment documentation for the later stages of TD5 and obtain associated approvals and change of land use authorizations. Some additional property acquisitions will also be necessary.

AH currently holds industrial water use rights for up to 1,081,495 m^3/year for its El Salero well field, described in section 5.3, valid until 2028. AH currently holds or is re-activating other operational permits including waste management, and explosives storage and handling.

20.3.8 Employment

Aura Minerals maintains a positive relationship with the Municipality and mining has been an integral part of the community fabric and landscape for well over a century. The Project expansion is not expected to significantly alter the local socioeconomic conditions that existed at pre-expansion operating levels. Direct employment is not expected to increase over the previous operational phase as a result of the Project.

20.3.9 Environmental Management

The Project plan considers two environmental supervisors and one community relations liaison reporting to a Security, Health and Environmental Superintendent. Costs for required monitoring, inspections and reporting are considered as part of the operation’s general and administrative expenses.

20.3.10 Mine Closure

A conceptual closure assessment for the purposes of costing was developed for Aranzazu in 2009 and has since formed the basis of the site’s closure cost estimate. The assessment assumes a 5-year implementation period following permanent closure. The three main aspects included in the assessment are: 1) Studies (geotechnical and geochemical), 2) Structure Decommissioning (process plant, underground mine, open pits, administrative and maintenance areas, tailings impoundments, waste rock piles, and miscellaneous sites such as legacy workings), and 3) Monitoring.

As noted in Item 21.2.3, the Project cost model assumes US$6.5M for overall site closure costs (including both existing workings and the Project to be constructed), with US$0.5M allocated for each of Years 4 and 6, and the remaining US$5.5M expended at Year 7. Closure of the various tailings facilities accounts for a significant portion of the site closure cost. It is recommended that a comprehensive Closure Plan document and design be developed during the operating life of the Project.

21. CAPITAL AND OPERATING COSTS

All Capital and Operating costs are presented in nominal 2018 U.S. dollars. No escalation has been applied to revenues, capital or operating costs. An exchange rate of 18.0:1.00 (MXN:USD) has been used across all calculations in the cost model unless otherwise noted.

Costs have been derived based upon current market conditions, vendor quotes, design criteria developed by collaboration with other external consultants and specialists in different areas such as geology, mining, geotechnical, infrastructure, environment and mineral processing and benchmarks against similar existing projects.

21.1 CAPITAL COSTS

Total capital cost for the Project has been estimated at US$92.5M including mine development and equipment, plant and tailings, plant, drilling and infrastructure. A contingency of 10% has been applied to all costs except mine development.

The initial capital in Year 1 is estimated at US$32.1M which includes mine development, construction of tailings dam, plant refurbishment and upgrades, mine equipment along with construction of a new dedicated powerline. The Project has also included a capital allowance for pre-production costs in the first six months of operation. Sustaining capital over the following six years is estimated at US$60.4M. Closure costs have been estimated at US$6.5M (no contingency applied) which will occur primarily in Year seven of the Project. Table 21-1 summarizes the initial and sustaining capital breakdown and the capital spending profile and Table 21-2 shows the overall Capital expenditure over the Mine's LOM.

Table 21-1: Initial and Sustaining Capital Summary

Item	Initial (US$M)	Sustaining (US$M)
Pre-Production	$5.8	-
Underground Development	$12.2	$33.1
Tailings Dam	$6.9	$7.1
Mine Equipment	$2.2	$3.3
Plant	$2.0	$5.1
Powerline	$1.2	-
Exploration / Delineation Drilling	$0.5	$3.5
Sub-Total	**$30.8**	**$52.0**
Contingency (5%)	$1.3	$1.9
Closure Cost	$0.0	$6.5
Total	**$32.1**	**$60.4**

Table 21-2 Capital Development Schedule

Category	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total (US$M)
Pre-Production	$5.8	-	-	-	-	-	-	**$5.8**
UG Development	$12.2	$14.0	$12.6	$5.0	$1.5	$0.1	-	**$45.3**
Tailings	$6.9	$3.1	$0.3	$2.0	$1.5	$0.3	-	**$14.0**
Mine	$2.2	$1.6	$1.5	$0.2	-	-	-	**$5.5**
Plant	$2.0	$0.5	$3.7	$0.5	$0.2	$0.2	-	**$7.1**
Powerline	$1.2	-	-	-	-	-	-	**$1.2**
Drilling	$0.5	$0.5	$1.5	$1.5	-	-	-	**$4.0**
Sub Total	**$30.8**	**$19.7**	**$19.5**	**$9.1**	**$3.2**	**$0.5**	**-**	**$82.8**
Contingency (10%)	$1.3	$0.6	$0.7	$0.4	$0.2	$0.1	-	**$3.2**
Closure	-	-	-	$0.5	-	$0.5	$5.5	**$6.5**
Total	**$32.1**	**$20.3**	**$20.1**	**$10.0**	**$3.4**	**$1.1**	**$5.5**	**$92.5**

The largest capital costs in the Project are related to underground access development, construction of new tailings dam as well as subsequent raises and pre-production costs; followed by smaller capital expenses in operating areas such as mine, processing plant and the construction of a powerline.

21.1.1 Pre-Production

Table 21-3 shows the Pre-Production costs, which are estimated at US$5.8M in Year 1. These costs are associated to all expenses incurred before production of the first tonne of concentrate; and includes items such as initial hiring of people, consultants, land purchasing requirements, corporate support, fees associated to permits and licences and first fills for the processing plant.

21.1.2 Mining

Mine capital include cost for the development of the main ramps and levels along with capital equipment for continuation of mining (including ventilation, pumping, electrical and other infrastructure). The total cost for Mine development capital is US$45.3M. The total cost for Mine equipment and infrastructure capital is US$5.5M.

Underground development will be performed by an external contractor and costs for the development were based on quotes provided for the planned scope of work. Costs were obtained from three (3) contractors quoted in Mexican peso and converted to US dollars at an exchange rate of 18:1. Costs were provided for different types of development (profile, gradient) and different levels of ground support.

Table 21-3 Pre-Production Costs

Item	Capital (US$M)
Start-up Labor	$3.1
First fills	$1.5
Corporate and Consultants	$0.2
Land Purchasing	$0.5
Fees and other costs	$0.1
Admin Equipment	$0.4
HSE Supplies	$0.1
TOTAL	**$5.8**

The costs provided by the contractor included labor, materials and consumables (excluding explosives), and equipment operating costs; costs for explosives were calculated separately and added to the contractor costs. The costs also included the supply and installation of air/water/power services.

Ground support costs were based on the ground support standard for each geomechanically rock type; the overall development cost for each profile was determined using the weighted average of the geomechanical rock type contained within (as determined from the geotechnical block model).

The following table summarizes the capital development costs for the mine.

Table 21-4 Capital Development Unit Rates

Development Type	Profile (W x H)	Cost (US$/m)	Quantity (m)	Total Cost (US$M)
Main Ramps	5.0m x 5.0m @15%	$2,036	4,268	$8.7
Main Levels/Haulage Galleries	4.5m x 4.5m @ 2%	$2,077	8,919	$18.5
Stope X-Cuts	4.5m x 4.0m @ 2%	$1,970	8,196	$15.7
Misc. Development	4.5m x 4.0m @ 2%	$1,970	605	$1.2
Vertical Raise	3.0m x 3.0m	$1,590	782	$1.2
Total			**22.771**	**$45.3**

Capital equipment and infrastructure costs for the mine included costs for ventilation, dewatering, electrical distribution and communications, underground refuge, equipment and general construction.

The following provides a brief summary of the cost items included in the above categories.

- Dewatering: purchase and installation of sump pumps on main ramp, purchase and installation of pumps for main settling sump, miscellaneous piping allowance
- Ventilation: purchase of additional components and installation of 2nd main fan at surface, underground main ventilation fans, ducting and ventilation controls, rehabilitation of ventilation raise
- Electrical Distribution: new 13.8 KV surface substation, underground mine load centers, medium voltage electrical cable and accessories
- Communications: installation of mine radio and mine phone systems
- Underground Infrastructure: allowances for compressed air and mine water systems
- Underground Refuge and Escape: purchase of refuge chambers, escapeway ladders, mine rescue equipment
- Mobile Equipment: purchase of support and light vehicles to support underground activities.
- Underground Construction: general costs for installation of ventilation, dewatering, electrical infrastructure

Cost estimates for major equipment were based on quotes from supplier or from costing databases. Prices were obtained in US$ values however there is potential to source equipment within Mexico at local currency.

Total mine infrastructure and equipment cost is US$6.0M (including 10% contingency) and is summarized in the following Table 21-5.

Table 21-5 Mine Capital Summary

Category	**Cost Estimate ($USM)**	**Contingency ($USM)**	**Total ($USM)**
Underground Construction	$0.6	$0.06	$0.6
Mobile Equipment	$0.2	$0.02	$0.3
Dewatering	$0.3	$0.03	$0.4
Ventilation Systems	$1.1	$0.11	$1.2
Electrical Distribution	$2.1	$0.21	$2.3
Communications	$0.2	$0.02	$0.2
Mine Services	$0.1	$0.01	$0.2
Refuge and Escape	$0.8	$0.08	$0.9
Total	**$5.5**	**$0.52**	**$6.0**

21.1.3 Plant and Tailings

The process plant capital cost requirements are estimated at US$ 7.05M for refurbishment of the existing plant. in addition to US$14M for the tailings facility.

The Capital Expenditure schedule for the plant is shown in Table 21-6. Year 1 is considered the initial capital for plant repairs, while Year 3 includes a US$3.7M for the installation of a new tailings thickener as well as improvements in the water system.

Table 21-6 Plant Capital Schedule

Plant Capital	Year 1 (Initial Capital)	Year 2	Year 3	Year 4	Year 5	Year 6	Total
US$M	2.0	0.5	3.7	0.5	0.2	0.2	7.1

The process plant considers refurbishment activities and upgrades prior to resuming the operation which accounts for the initial Capital of US$2.0M. These activities will address current issues with the water system (i.e. piping and pumping), modifications in the processing plant for the grinding and flotation circuits, improvement of the crushing circuit as well as mechanical and structural repairs and enhanced instrumentation and control for the plant. Table 21-7 presents the breakdown as well as the source for the cost items.

Table 21-7 Initial Plant Capital Estimate

Line Item	**US$**	**Source**
Water Line Improvements	$134,000	Quote
Pump Station No. 3 Improvements	$125,000	Quote
Electrical Improvements	$27,000	Estimate
Pumps and piping	$47,000	Quote
Crusher improvements	$125,000	Quote
Grinding improvements	$162,000	Quote
Flotation improvements	$64,600	Quote
Thickener and filtration Improvements	$13,000	Quote
Others	$28,000	Estimate
IandC Components		
Crushing Circuit Logic Rewrites	$25,000	Estimate
Flotation Cell Logic Rewrites	$18,000	Estimate
Online Stream Analyzer (OSA)	$320,000	Quote
Pump Stations	$190,000	Quote
Crushing	$52,000	Quote
Grinding	$48,500	Quote
Flotation	$58,250	Quote
Filtration	$48,000	Quote
First fills	$300,000	Estimate
Commissioning Services	$120,000	Quote
Contingency	$95,268	Quote
TOTAL	**$2,000,618**	

The following table 21-8 provides the capital schedule for the new tailings dam development (TD5):

Table 21-8 Tailings Dam Capital Schedule

Plant Capital	**Year 1 (Initial Capital)**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Total**
US$M	6.9	3.1	0.3	2.0	1.5	0.3	14.0

The Starter Dam will provide one year of storage capacity and its cost estimate includes the construction of a new Municipal landfill required to dispose community waste as it is currently being disposed within the footprint of the new dam. The project considers the total removal of the existing garbage volume out of the dam footprint and redispose in the new landfill. The initial capital also includes the purchase of new water reclaim pumps, tailings line tie-ins and improvements in the existing tailings line and disposal system.

The construction costs presented in the below Table 21-9 are based on construction quotes provided by the Mexican construction companies in January 2018.

Table 21-9 Starter Dam Capital

Item	Description	Cost US$
1.0	Mobilization	121,800
2.0	Site Preparation	95,000
3.0	Removal of organic material, including loading and hauling to a final storage area (59,957 m3)	162,400
4.0	Structural fill excavation, including loading and hauling (643,400 m3)	1,937,200
5.0	Waste rock excavation, loading and hauling (147,600 m3)	127,600
6.0	Tailings excavation, loading and hauling (29,533 m3)	78,880
7.0	Filter material loading and hauling to construction area (27456 m3)	24,360
8.0	Dam Construction, including placing of waste rock, structural fill and compacted tailings. All material placed, leveled and compacted to 95% Proctor in layers of 30 cm.	1,690,000
9.0	Construction of the north water deviation channel including cut and fills, construction of safety berms, placing material to specs, etc.	105,560
10.0	Construction of the south water deviation channel including cut and fills, construction of safety berms, placing material to specs, etc.	54,520
11.0	Collection ponds North and South - Includes excavation, fill and compacted material	58,000
12.0	Double wall corrugated 8" diameter HDPE RD17 CPE pipeline (205 m)	6,960
13.0	Double wall corrugated 6" diameter HDPE RD17 CPE pipeline (190 m)	20,880
14.0	4” HDPE RD17 solid pipeline (635 m)	13,340
15.0	Others: couplings, valves, reducers, etc.	7,380
16.0	8Oz Geotextile (North and South deviation channels) - materials and installation of 5,100 m2	16,820
17.0	Geomembrane HDPE 1.5mm - materials and installation of 2,778 m2	30,160
18.0	Dam water reclaim pumps	266,800
19.0	Tailings line extension and system improvements	310,000
20.0	Construction of new Municipal Landfill	638,000
21.0	Waste removal from TD5 area	63,800
22.0	Instrumentation and control	115,000
23.0	Others (permits, complementary studies, etc.)	116,000
24.0	Contingency (15%)	840,000
	TOTAL	**6,900,500**

21.1.4 Other Capital

Other capital expenditures that have been incorporated into the study include.

Powerline
A new 6.0 km powerline will be constructed to provide dedicated feed of 6.0 MW from the Commission Federal de Energia (CFE) substation located at the entry of the Concepcion del Oro Township. This cost covers the electrical fee and guarantee to be provided to CFE, the purchase and installation of the high-voltage cable and poles, earthworks and electrical components. This cost is estimated US$ 1.2M and is based on a quote provided by a local Mexican contractor and is expected to be spent in Year 1.

Drilling
The resource at Aranzazu has been drilled extensively, sufficient for declaration of a Mineral Resource and Reserve. However, given the nature of the orebody there is additional delineation drilling required in order to better define the contacts of the economic ore zone in order to optimize the operational stope designs. The concept was that each stope should have at least 2 drill holes through it – one at the upper and one and the lower horizon to fully define the best location for the placement of the top and bottom sill cuts and the stope boundaries.

Drilling will be performed with a small diamond drilling unit from a location in either the main ramp or the haulage gallery to reduce the length of the holes. The drilling is to be scheduled in the production plan to ensure sufficient time for assay, analysis and design to be completed before commencing with stope development. An allowance of US$ 4.0M has been allocated for underground drilling to be spent over the first 4 years of the operation.

Closure
Site closure costs are associated with decommissioning and reclamation of the tailings storage facilities, waste rock piles and surface mining areas, removal of underground infrastructure and sealing of accesses (including non-working legacy facilities), processing plant and the general site.

Work will include surface capping, recontouring and re-vegetation of waste rock and tailings areas, sealing and/or securing underground openings, removal and disposal of equipment, dismantling of buildings and surface facilities, disposal of hazardous waste and/or chemicals, installation of water controls around open pits and general site cleanup.

Closure costs were reviewed by an independent consultant in late 2017; suggested adjustments and recommendations for additional information were provided to Aranzazu in January 2018. This current estimated closure cost for the Project considers approximate closure costs for new works, and where considered appropriate, adjusts existing estimates prepared by Aranzazu during previous operating years. The costs incorporate several years of post-closure monitoring and require regular review and refinement throughout the operating life of the Project.

Based on current estimates, the cost model includes US$6.5M in closure costs for the site. Initial costs of approximately US$1.0M will be incurred 1 to 3 years pre-closure to carry out engineering studies and develop closure plans as well as begin closure work on the decommissioned tailings storage facilities. Following cessation of mining operations, the remaining US$5.5M cost will be incurred.

21.2 OPERATING COSTS

Total operating cost for the Project has been estimated at US$267.6M including direct and indirect mining costs, plant and general administration costs. Operational costs for Aranzazu estimated at US$56.66/t as shown in Table 21-10. The mine operation has an estimated operational cost of US$38.86/t, US$10.91/t for processing and US$6.78/t for General and Administrative (G&A). Royalties paid to landowners is US$1.11/t.

The operating unit costs, with the exception of labor and contractor costs were based on the historic operating costs from the period of 2013-2014 adjusted for inflation and the fluctuations in the currency exchange rate. Labour costs were based on the August 2017 CAMIMEX benchmarking study which was used to determine the salary ranges and benefits for the identified positions in the manpower compliment. Details of the manpower compliment are shown later in this chapter. A summary of operating costs used in the financial analysis are shown in Table 21-10 below.

Table 21-10 LOM Operating Cost Summary

Category	**Unit Cost (US$/t)**	**Total Cost (US$M)**	**Comments**
Mining			
Contractor Mining	$34.74	$161.3	Direct mining costs
Owner’s costs	$4.11	$19.1	Operations/technical support, power, explosives
Total Mining	**$38.86**	**$180.4**	
Total Processing	$10.91	$50.6	
General and Admin.	$6.78	$31.5	Site management, fees, administration,
Total Operating Cost	**$56.54**	**$262.5**	
Royalties	$1.11	$5.2	Landowner royalties
Total	**$57.66**	**$267.6**	**Total Operating Cost**

21.2.1 Mining

Mining costs are estimated at US$38.86/t ore mined. The following Table 21-11 provide a summary of the direct (contractor) and indirect (owner) mining costs. The operating costs for the mine are based the direct and indirect costs. All mining operations are to be performed by an external contractor. Estimates for the direct mining costs have been based on quotes supplied by reputable Mexican contracting companies.

Cost estimates include the costs for ore production (both development), haulage to the mill, and backfill. Costs include labor, consumables and equipment costs (operating and maintenance). Major consumables such as fuel, explosives, cement ground support and drilling supplies are included in these costs however the study assumes that these items with will be sourced by Aranzazu directly.

Indirect costs are the owner's costs to provide operations and technical support to the contractor along with supplying and maintaining the primary underground infrastructure. These costs include wages and salaries for Aranzazu management, technical and maintenance staff, power and mine services, along with allowances for operating consumables (parts, fuel).

Table 21-11 Mine Operating Cost Summary

Category	US$/t	Comments
Direct Mining Cost		Contractor costs
Ore Development	$27.06	Include drill, blast, muck and support to 6m wide
Slot Raises	$0.75	3m x 3m slot raise
Sill Cut Slash	$7.56	Enlarge to full sill width (10 – 15m)
Long Hole Stopes	$17.26	Drill, blast and muck ore (remote LHD)
Total Ore	$16.50	Average ore mining cost
Haulage	$4.54	Average haulage cost
Backfill		
Cemented Rockfill	$19.43	Includes aggregate and 8% cement, haul and dump
Waste Rockfill	$8.00	
Backfill Total	$13.71	Average backfill cost
Total Direct Mining Costs	**$34.74**	
Indirect Mining Cost		Owner's cost
Labor	$1.53	Salary and Benefits
Consumables	$2.10	Maintenance parts and supplies
Services	$0.50	Power
Total Indirect Mining Cost	**$4.11**	
Total	**$38.86**	**Total Mine Operating Cost**

The annual labour costs are estimated to be US$1.45M or US$1.53/t of ore mined as shown in Table 21-12. Note that details of salaries and benefits per role are not provided due to safety concerns.

Table 21-12 Mine Labor Costs

Role	Quantity	TOTAL USD/Year
Management	3	348,370
Operations	3	124,760
Technical Services	21	728,110
Maintenance	10	161,700
Security	9	90,770
Total	**46**	**$1,453,710**

21.2.2 Processing

Processing costs includes a total workforce of 66 employees working on 12-hour shifts on a 14x7 rotation. The project considers that most, if not all, of the workforce will be Mexican and from the Zacatecas region.

During the project, salaries and benefits were fully reviewed and compared against the latest Mining Salary Survey provided by the Camara Minera de Mexico (CAMIMEX). From this review, benefits for specific positions were adjusted as the historic salary numbers coming from 2014 appeared to be lower than the average in the Mining industry in 2017. The annual labour costs are estimated to be US$1.9M or US$2.04/t of ore processed as shown in Table 21-13. Note that details of salaries and benefits per role are not provided due to safety concerns.

Table 21-13 Plant Labor Costs

Role	Quantity	TOTAL USD/Year
Management	3	175,000
Operations	38	1,030,200
Laboratory	6	168,000
Maintenance	19	567,500
Total	**66**	**1,940,700**

The annual cost of consumables for the processing plant is estimated at US$3.1M or US$3.27/t of ore processed, as presented in Table 21-14.

Table 21-14 Plant Consumable Costs

Category	Cost ($US/kg)	Total Cost (US$/year)
Water		$533,300
Grinding media and Steel	$1.34	$1,181,700
Lime	$0.16	$468,500
Collector I 5100	$9.18	$90,500
Collector II 3477	$3.94	$38,850
MIBC	$3.94	$194,200
Flocculant	$8.0	$157,700
Oil, fuel, lubricants, etc.		$410,000
Total		**3,074,750**

The total processing costs are estimated at US$10.91/t ore processed as shown in Table 21-15. Unit costs are based on a target production rate of 2,597 tpd at 92% availability. Table 21-15 summarized the plant operating costs.

Table 21-15 Plant Operating Cost Summary

Category	Cost ($US/t)	Comments
Labour	$2.04	See table 21-11
Consumables	$3.27	See table 21-12
Maintenance	$2.10	Parts and supplies
Services	$3.15	Power
Outside Contractors	$0.35	Equipment rentals, contract labour
Total	**$10.91**	**Total Plant Operating Cost**

21.2.3 General and Administration (G&A)

General and Administration Labour is comprised of 53 employees as shown in Table 21-16, for a total annual cost of US$1.9M or equivalent to US$2.02/t of ore mined. Note that details of salaries and benefits per role are not provided due to safety concerns.

Table 21-16 General and Adminstration Labor Costs

Role	Quantity	TOTAL USD/Year
Management	5	497,500
HSEC	7	235,500
Finance & Legal	7	311,700
Logistics & Purchasing	10	330,500
Human Resources	4	158,000
General Others	20	369,700
Total	**53**	**1,902,700**

The total General and Administration costs are estimated at $6.78/t ore mined as shown in Table 21-17. Unit costs are based on a target production rate of 2,597 tpd.

Table 21-17 General and Administration Cost Summary

Category	Cost ($US/t)	Comments
Labour	$2.02	Wages and Benefits
Consumables	$1.30	Site costs and supplies
Taxes, Fees and Insurance	$0.68	
Cost of Sales	$2.56	Concentrate transport and services
Services	$0.21	Power, communications, others
Total	**$6.78**	**Total G&A Cost**

The cost associated to sales of concentrate are based on previous concentrate agreements in 2014, which are no longer valid. New concentrate sale terms will be applied during the start-up and it is expected that those terms and conditions will be more favorable and will represent a lower cost compared to 2014.

21.2.4 Operational Cost Summary

Table 21-18 outlines the total cash operating cost before precious metal credits for the project at US$389.4M or US$2.51/lb Cu (including treatment charges, transportation costs and royalties). The reportable cash cost after credits is US$220.3M or US$1.42/lb Cu. The All-in-Sustaining Cost is US$1.81/lb Cu.

The reportable cash cost (after credits) of US$1.42/lb Cu is less than 50% of the average LOM price of copper that has been used in the study and the All-in-Sustaining Cost of US$1.81/lb Cu is below $2.00 per pound. The overall cost structure combined with the gold/silver credits provide a strong basis for a profitable operation.

Table 21-18 Site Operating Cost Summary

LOM Total Cost Breakdown	US$M	US$/lb Cu
Smelting, Refining, Treatment and Freight*	$121.7	$0.78
Cash Operating Costs	$262.5	$1.69
Royalties	$5.2	$0.03
Reportable Cash Costs	$389.4	$2.51
Credit: Gold Revenue	-$139.3	-$0.90
Credit: Silver Revenue	-$29.8	-$0.19
Reportable Cash Costs after precious metals credits	$220.3	$1.42
Copper Produced (M lbs.)	155.2	
Total Cash Costs (payable Cu)	-	**$2.51**
Total Cash Costs (payable Cu) After Credits	-	**$1.42**
Add: Sustaining Capital**	$60.4	$0.39
Total Costs incl. Sustaining Capital	$280.7	$1.81
***Includes Royalties, contingency, all sustaining capital after Year 1 and closure costs*		
All-in Sustaining Total Cash Costs**	**-**	**$1.81**

22. FINANCIAL EVALUATION

22.1 INTRODUCTION

Project economics were prepared on a monthly basis for the first two years and annually thereafter. Based upon design criteria presented in this Technical Report, the level of accuracy of the estimate is considered ±15%.

All economic results are presented in Q1 2018 US$ unless otherwise specified. No escalation has been applied to either capital or operating costs and no debt is assumed in the analysis other than the US$6.5M debt owed to certain creditors as discussed below. Tables and figures presented in this chapter are the summarized results of the full evaluation and as a result, the calculations inherently involve a degree of rounding. Where these occur, they are not considered to be material.

22.2 PRINCIPAL ASSUMPTIONS AND INPUT PARAMETERS

The metal prices used in the study which were based on long-term forecasted prices for copper, gold and silver from a leading Canadian schedule I Bank. Table 22-1 below summarizes the metal prices used in this study.

Table 22-1 Summary of Metal Prices

Commodity Price	Year 1	Year 2	Year 3 Onwards	LOM Average
Copper (US$/lb)	$2.90	$2.95	$3.10	$3.06
Gold (US$/oz)	$1,250	$1,299	$1,301	$1,297
Silver (US$/oz)	$18.23	$19.47	$19.83	$19.62

The Foreign Exchange rate considered was 18.0:1.00 MXN:USD according to projections provided by two leading Canadian banks.

The mine production considers 4.64M t ore over a 5,5-year mine life, average grades of 1.72 % Cu, 1.17 g/t Au, 19.2 g/t Ag for a total metal production of 155.1M lbs Cu, 122,149 oz Au, and 2.0M oz Ag.

All Capital and Operating costs are on quotations and estimates based on 2018 dollars. Total Project capital of US$92.5M is comprised of Initial Capital of US$32.1M and Sustaining Capital of US$60.4M.

22.3 CASHFLOW FORECAST

Due to the re-start nature of Aranzazu, availability of ore and modest capital expenditures required, the Project is able to generate revenue much earlier than a newly constructed mine (i.e., mill start up in Month 6 and thereafter commercial production in Q1 of Year 2). As such, after Aranzazu's shorter development and stockpiling period, its resulting cash flows upon mill start up provide it the ability to reach payback in less than two years. Table 22-.2 shows the full cash flow for the Aranzazu project.

The financial evaluation considers an outstanding debt of US$6.5M with suppliers and contractors who worked with Aranzazu before the 2015 shutdown. Aranzazu Holding is to commence payment to creditors two months after receipt of payment for the first concentrate shipment that may be any time between April 2018 and no later than April 2019. The debt is to be paid to each creditor in 36 equal monthly payments, with full payment by no later than April 2023.

22.4 TAXES, ROYALTIES AND OTHER INTERESTS

22.4.1 Taxes

Federal Tax: Corporate federal income tax is determined by subtracting all allowable operating expenses, overhead, depreciation, amortization and depletion from current year revenues to arrive at taxable income. The tax rate is then determined from the published progressive tax schedule. An operating loss may be used to offset taxable income, thereby reducing taxes owed.

As per Mexican Law, Aranzazu is also subject to an Extraordinary Mining Duty of 0.5% on precious metal revenues, a Special Mining Duty of 7.5% on EBITDA and a Corporate Income tax of 30%. The Extraordinary Mining Duty and the Special Mining Duty are both deductible from the taxable base in the Corporate Income tax calculation. There are currently 1.4B MXN (US$79M) in tax loss carry forwards that can be used to offset corporate taxes. It is estimated that US$10.1M will be paid in corporate taxes in addition to US$18.6M in Extraordinary Mining Duty and Special Mining Duty on behalf of Aranzazu.

State Tax: Aranzazu will also be subject to a state tax of 2.5%. It is estimated that US$3.1M will be paid to the state of Zacatecas, Mexico on behalf of Aranzazu. Taxes and royalties above have been included in the cash-flow model.

22.4.2 Royalties

Royalties are assessed on an NSR basis at a rate of 1% as per private landowner royalties (Macocozac – the previous owner of Aranzazu) and are included in the reported total cash costs per pound of copper produced. It is estimated to equate to US$5.2M over LOM.

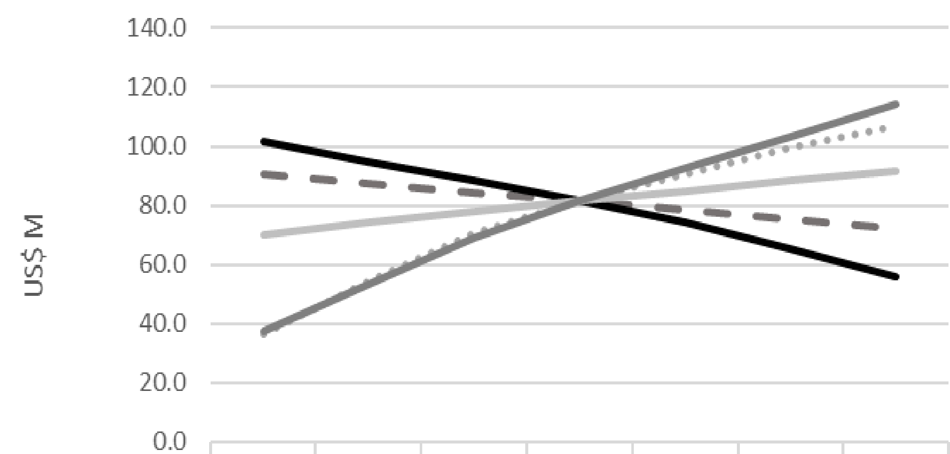
Table 22-2 Detailed Cash Flow Analysis

	LOM	Month 1	Month 2	Month 3	Month 4	Month 5	Month 6	Month 7	Month 8	Month 9	Month 10	Month 11	Month 12	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Tonnes Mined	**4,641,775**	14,606	13,245	10,431	18,812	23,316	25,828	41,501	47,614	38,388	44,294	57,222	32,448	367,704	910,488	941,592	942,341	942,858	536,792
Grade																			
Cu (%)	**1.72%**	2.06%	1.85%	1.92%	2.02%	1.91%	1.76%	1.55%	1.88%	1.53%	1.62%	1.57%	1.78%	1.67%	1.66%	1.70%	1.83%	1.78%	1.60%
Au (g/t)	**1.17**	0.95	0.83	0.63	0.67	0.59	0.67	0.95	1.01	0.80	0.89	1.00	1.10	0.95	1.42	1.33	0.98	0.97	1.35
Ag (g/t)	**19.2**	19.9	17.5	16.9	18.9	25.1	22.9	21.0	25.0	21.2	20.6	18.6	20.7	21.3	19.1	18.3	19.0	19.3	20.0
Tonnes Milled	**4,641,775**	0	0	0	0	0	0	0	79,856	77,280	79,856	77,280	53,432	367,704	910,488	940,240	940,240	940,240	542,863
Grade																			
Cu (%)	**1.72%**	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	1.83%	1.75%	1.70%	1.62%	1.72%	1.73%	1.66%	1.70%	1.83%	1.78%	1.60%
Au (g/t)	**1.17**	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.83	0.82	0.85	0.93	1.04	0.88	1.42	1.33	0.98	0.97	1.34
Ag (g/t)	**19.2**	0.0	0.0	0.0	0.0	0.0	0.0	0.0	22.0	21.8	21.3	19.8	20.3	21.1	19.1	18.3	19.0	19.3	20.0
Metal Contained																			
Cu (lbs M)	**176.4**	-	-	-	-	-	-	-	3.2	3.0	3.0	2.8	2.0	14.0	33.3	35.1	37.9	36.9	19.1
Au (oz)	**174,696**	-	-	-	-	-	-	-	2,137	2,048	2,173	2,320	1,779	10,458	41,510	40,217	29,753	29,343	23,416
Ag (oz M)	**2,871,579**	-	-	-	-	-	-	-	56,409	54,095	54,780	49,125	34,911	249,320	558,443	554,456	575,329	584,434	349,597
Recoveries																			
Cu (%)	**88.0%**	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%
Au (g/t)	**69.9%**	66.9%	66.9%	66.9%	66.9%	66.9%	66.9%	66.9%	66.9%	66.9%	66.9%	66.9%	66.9%	66.9%	71.9%	71.1%	67.8%	67.7%	71.3%
Ag (g/t)	**70.0%**	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%
Metal Produced																			
Cu (lbs M)	**155.2**	-	-	-	-	-	-	-	2.8	2.6	2.6	2.4	1.8	12.3	29.3	30.9	33.4	32.4	16.8
Au (oz)	**122,149**	-	-	-	-	-	-	-	1,429	1,369	1,453	1,552	1,190	6,993	29,850	28,582	20,173	19,868	16,684
Ag (oz M)	**2,010,105**	-	-	-	-	-	-	-	39,486	37,866	38,346	34,388	24,438	174,524	390,910	388,119	402,730	409,104	244,718
Revenue (US$M)	**620.8**	0.0	0.0	0.0	0.0	0.0	0.0	0.0	9.8	9.2	9.3	8.9	6.5	43.8	123.3	130.4	126.5	123.7	73.1
Total Operating Cost (US$M)	**262.5**	0.7	0.6	0.5	0.9	1.1	1.2	1.9	3.0	2.6	2.9	3.5	2.1	20.8	51.5	53.2	53.3	53.3	30.4
Royalties (US$M)	**5.2**	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.1	0.1	0.1	0.1	0.4	1.0	1.1	1.1	1.0	0.6
Total Cash Costs (US$M)	**267.6**	0.7	0.6	0.5	0.9	1.1	1.2	1.9	3.1	2.7	3.0	3.5	2.1	21.2	52.5	54.3	54.3	54.3	31.0
EBITDA (US$M)	**$231.40**	-$0.67	-$0.60	-$0.48	-$0.86	-$1.06	-$1.18	-$1.89	$4.78	$4.81	$4.53	$3.54	$3.13	$14.05	$47.66	$52.42	$47.74	$45.18	$24.36
After tax cash flow (US$M)	**$199.57**	-$0.67	-$0.60	-$0.48	-$0.86	-$1.06	-$1.18	-$1.89	$4.21	$4.24	$3.99	$3.12	$2.76	$11.58	$41.95	$46.38	$42.43	$37.82	$19.41
Initial Capital (US$M)	**-$32.1**	-$3.5	-$3.8	-$3.4	-$2.6	-$2.9	-$3.9	-$2.3	-$2.1	-$1.9	-$2.0	-$1.8	-$1.9	-$32.1	$0.0	$0.0	$0.0	$0.0	$0.0
Sustaining Capital (US$M)	**-$60.4**	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	-$20.3	-$20.1	-$10.0	-$3.4	-$1.1
Debt Repayments (US$M)	**-$6.5**	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	-$0.2	-$0.2	-$0.2	-$0.5	-$2.2	-$2.2	-$1.6	$0.0	$0.0
Mine Net Cash Flow (US$M)*	**$100.6**	-$2.5	-$2.8	-$4.4	-$3.9	-$3.6	-$4.5	-$3.0	$2.4	$1.3	$1.7	$2.0	$0.7	-$16.5	$20.7	$24.2	$29.6	$33.9	$16.5
**After working capital changes*																			
Cash Costs (US$M)	**220.3**	0.7	0.6	0.5	0.9	1.1	1.2	1.9	3.0	2.4	2.8	3.2	1.8	20.0	34.7	38.9	50.7	50.6	25.4
Total Cash Costs (US$/lb)	**1.42**	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.07	0.93	1.05	1.31	1.01	1.62	1.18	1.26	1.52	1.56	1.51
AISC (US$/lb)	**1.81**	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.07	0.93	1.05	1.31	1.01	1.62	1.88	1.91	1.82	1.66	1.57

22.5 PROJECT SENSITIVITY

Project Sensitivities (as shown in Figure 22.1) were evaluated at +/-15% range for copper and gold prices, capital and operating costs, and currency exchange. The Project is shown to be most sensitive to changes in the copper price, exchange rate and operating cost. The Project is less sensitive to gold price and capital cost.

Figure 22.1 Project Sensitivity



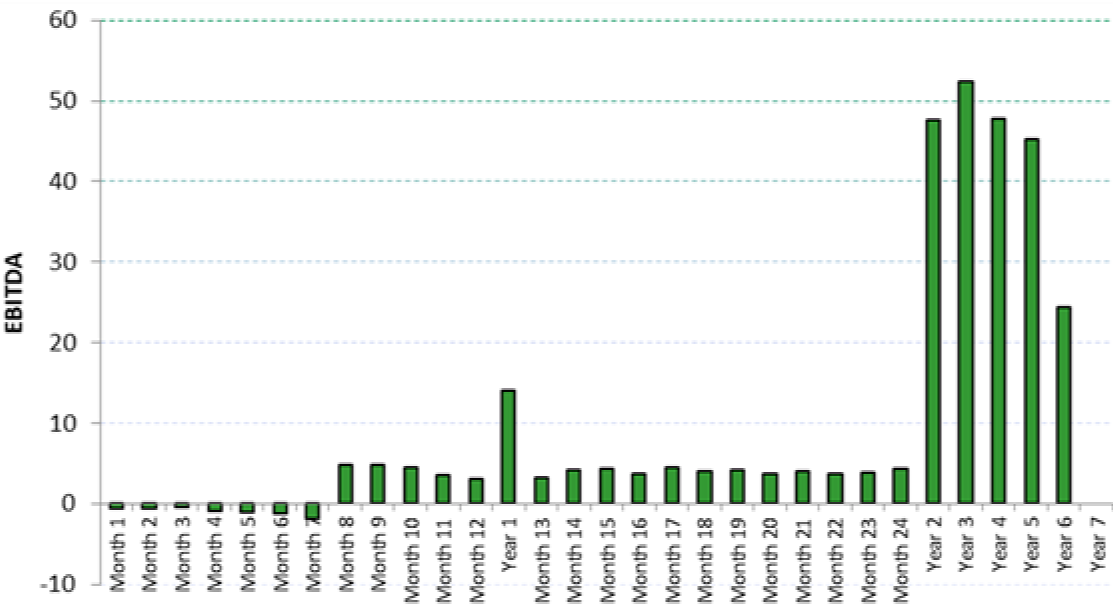
	85%	90%	95%	100%	105%	110%	115%
Cap Ex	90.6	87.5	84.4	81.4	78.3	75.2	72.1
Op Costs	101.8	95.0	88.2	81.4	74.5	65.4	55.7
Exch Rate	36.2	54.2	70.3	81.4	90.8	99.3	107.1
Gold Price	70.1	74.4	77.9	81.4	84.9	88.3	91.8
Copper Price	37.7	53.4	69.1	81.4	92.4	103.4	114.3

As with many operations in Mexico, fluctuations in the exchange rate will have a significant impact on the economics of the Project over its life. The trend over the past several years has seen a weakening in the MXN: USD exchange and projections for the next two to three years indicate either the same or slightly weaker exchange rates which would benefit the Project.

With respect to operating cost sensitivity, the Project economics will rely heavily on controlling the mining and processing costs. Regarding the mining costs, Aranzazu will take over the purchase and supply of major mining consumables which will provide a cost reduction. Other measures of cost control and reduction should be implemented as the operation matures.

In examining the effects of metal prices and exchange rate under a base case, downside case and upside case, results in the following financial scenarios:

Figure 22.2 Base Case FCF and EBITDA: (Cu :3.06US$/lb, Au : 1,297 US$/Oz, Ag : 19.6 US$/Oz, FOREX:18MX:1 USD)



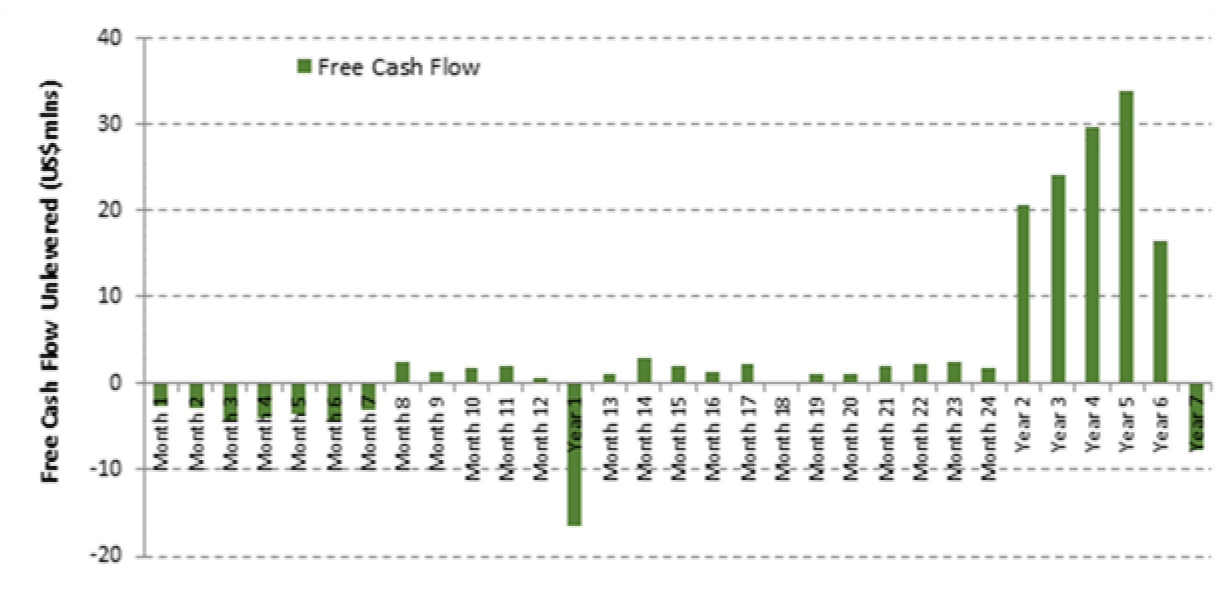
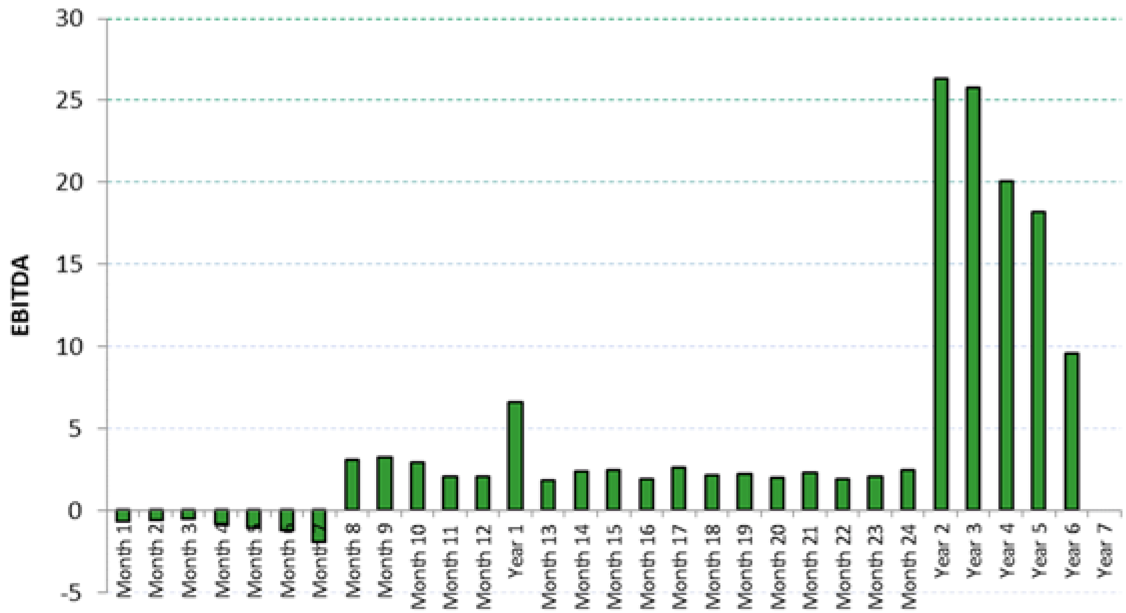


Figure 22.3 Conservative Case FCF and EBITDA: ($2.30/lb Cu, $1,200/oz Au, $17.00/oz Ag, FOREX:18MX:1 USD)



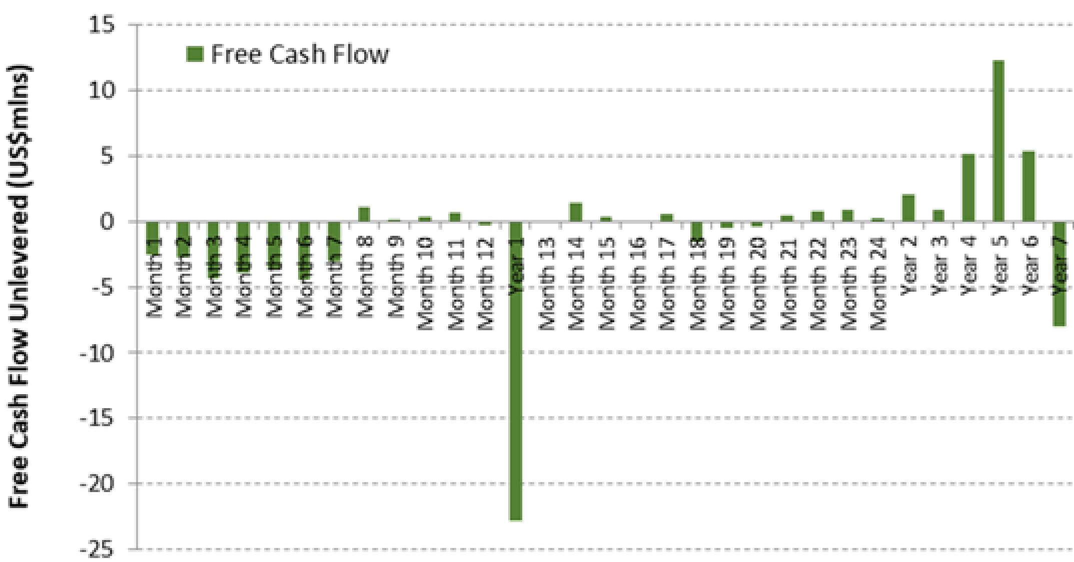
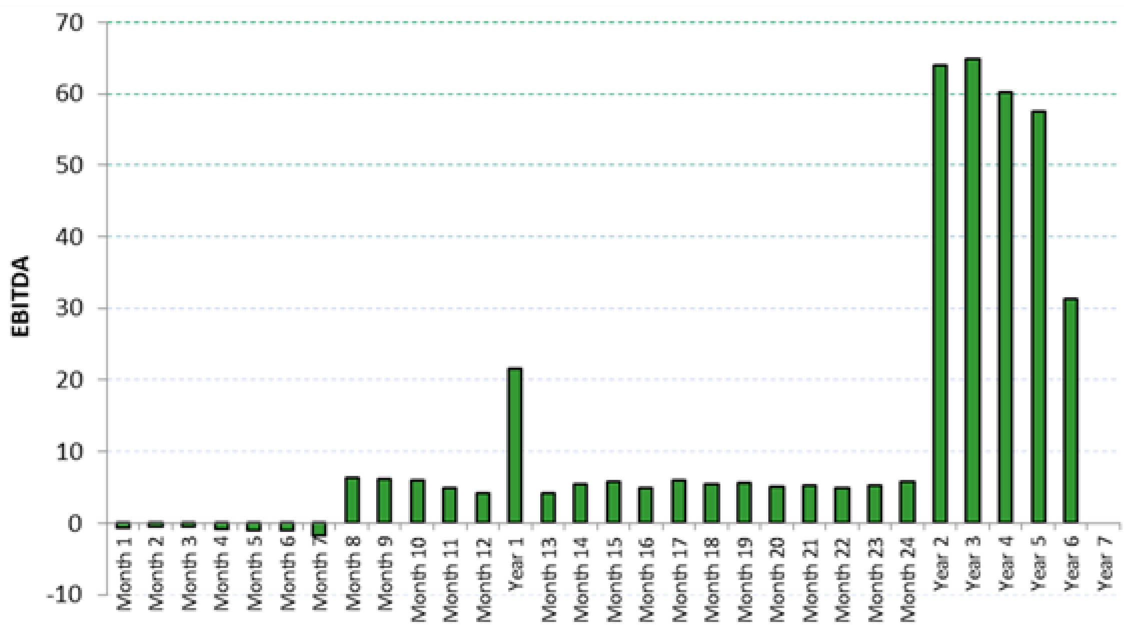
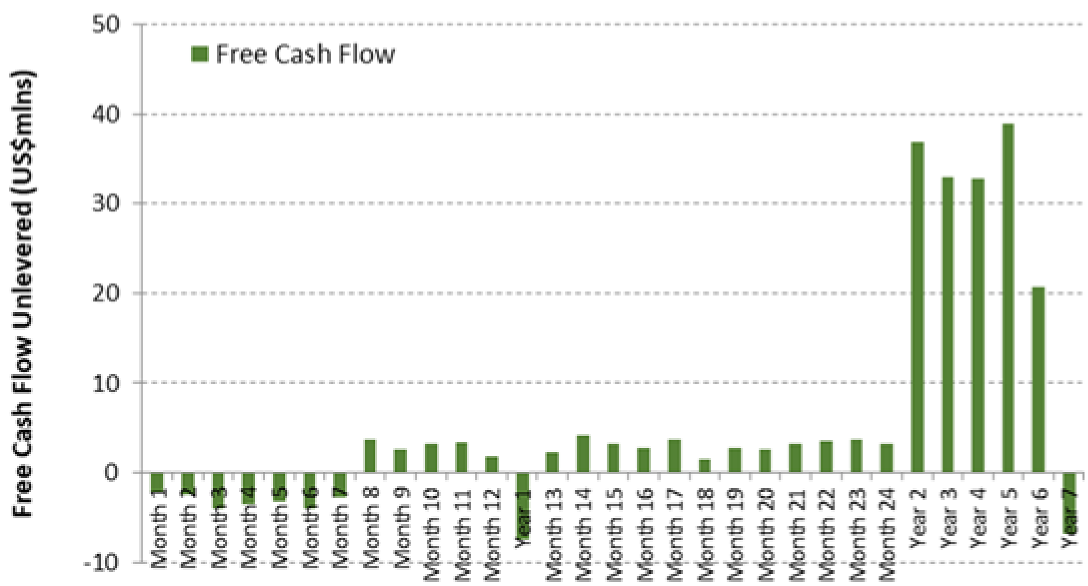


Figure 22.4 Upside Case FCF and EBITDA: ($3.30/lb Cu, $1,350/oz Au, $20.00/oz Ag, FOREX:18MX:1 USD)





22.6 PROJECT ECONOMIC RESULTS

The FS shows an after-tax NPV at 5.0% discount of US$81.4M and an IRR of 136.7%. The Project will produce a cash flow of US$100.6M with a payback of the capital in 22 months from start of production. The FS demonstrates the robust economic performance of the Project and results support the declaration of the Mineral Reserves. Table 22-3 summarizes the overall economics of the Project.

Table 22-3 Summary of Project Economics

Project Summary	**Units**	
Total Throughput	t	4,641,775
Mine Life	years	5.5
Recovered Metal in Conc.		
Cu production	t	70,416
Cu production	M lbs	155.2
Au production	oz	122,149
Ag production	oz	2,010,105
Concentrate Produced	t	306,156
Cu Concentrate Grade	%	23%
OPEX		
Total Cash Operating Costs*	US$M	$220.3
Total Cash Operating Costs*	US$/lb	$1.42
**After gold, silver credits*		
AISC**	US$M	$280.7
AISC**	US$/lb	$1.81
***Includes sustaining capital and closure costs*		
CAPEX		
Initial Capital	US$M	$32.1
Sustaining Capital	US$M	$60.4
Total Capital (incl. Closure)	US$M	$92.5
Financial Summary		
Total Revenue	US$M	$620.8
Net Smelter Return (NSR)	US$M	$499.0
LOM cash flow	US$M	$100.6
NPV (5.0%)	US$M	$81.4
IRR	%	137%
Payback	Months	22.0

23. ADJACENT PROPERTIES

No adjacent properties.

24. OTHER RELEVANT DATA AND INFORMATION

There is no other data or information that is relevant to the Project

25. INTERPRETATION AND CONCLUSIONS

25.1 CONCLUSIONS

The updated resource model and improved underground mining design has a positive effect on the financial performance of the asset.

The Mineral Resource Estimate was updated and validated by Farshid Ghazanfari, P.Geo., Qualified Person (QP) under NI 43-101 standards. The data verification process executed by the QP has led to confidence in Aura Minerals' database, which was previously compiled in-house by Aura Minerals. Therefore, the QP confirms that the database is suitable for use in the resource estimation reported herein and there is no bias in the nature of data and complied database.

The improved mining design has been developed from a geotechnical block model. The model was used to determine stope design widths, ground support recommendations, dilution estimates, pillar sizes and backfill strength.

The processing plant will be capable of treating 2,597 tpd, with the modification of the existing regrinding mill, as an auxiliary, to two of the primary grinding lines. The design and construction of the necessary modifications to achieve this should be carried out. The plant has treated 3,090 tpd in the past and the flotation circuit will be adequate with the minor modifications planned, largely the addition of proper level control and the ability to send concentrate from the second bank of cells either to final or scavenger concentrate streams. There will be an Online Stream Analyser (OSA) installed along with better sampling equipment. Instrumentation will be improved in both the grinding and flotation circuits. Extra plates will be added to the concentrate filter and density control added to the thickener underflow, to accommodate the extra tonnage of concentrate expected. These plans should be followed through to ensure that planned metal recoveries can be achieved.

In regards to metallurgy, it was found that finer primary grinding of the ore increases the overall recovery of the copper sulfides and gold significantly but does not increase the arsenic recovery to the same extent. It has been determined in open circuit tests that, for the general composite, grinding of the ore to 80% passing 150 microns compared to the previous 207 microns will increase the recovery of copper and gold in the concentrate from 82% to 88% and 62% to 70% respectively while only increasing the arsenic recovery from 80% to 82%. It was further observed that an even finer primary grind to a p80 of ~100 microns increased the copper and gold recoveries to 90% and 78% while arsenic recovery increased only to 83%. The composite sample used in these tests was made from ore from the Glory Hole Porfido zones only which represent 85% of the ore body scheduled to be mined in the next three years. Testing of variability samples showed that for the very high arsenic ores (0.3% As or higher) in this deposit, the finer grind will increase recovery of pay metals but also results in very high arsenic recovery and the differential (Cu recovery – As recovery) becomes smaller as the head grade increases. Grinding calculations indicate that the present primary circuit, with the addition of the regrind mill, can grind the ore to a p80 of 135 microns at the maximum required throughout so the expected recoveries will be intermediate between those observed with the 150 and 100-micron grinds.

A locked cycle test using a grind size (P80) of approximately 150 microns was performed and resulted in a concentrate grading 21% Cu, 14 g/t Au and 216 g/t Ag with recoveries in excess of 90% for copper and

69% for gold and silver. This recovery is downgraded to 88% for copper to allow for the slightly higher concentrate grade for ease of sale.

In regards to the new Tailings Dam No. 5, Aura Minerals will be required to compile and submit design and environmental assessment documentation for the later stages of TD5 beyond year and obtain associated approvals and change of land use authorizations. Some additional property acquisitions will also be necessary.

In regards to existing TD4, a number of actions must be completed before further tailings deposition can occur at TD4, including:

- Additional buttress construction at its toe,
- Installation of additional instrumentation to support buttress construction and short-term tailings deposition and
- Further site investigation to better characterize the tailings properties at TD4 and old Presa 4.

These actions are currently underway. Assuming the buttress is successfully constructed to the level currently required and allowing for the use of TD4 tailings in the construction of the TD5 dams and the TD4 buttress, TD4 has an available storage capacity of up to 259,500 dmt. This additional storage capacity is equivalent to around a one-third years of full production.

There is currently no comprehensive Closure Plan document for the Project, hence cost allowances have been made based on available information and review and updating of existing estimates. No other environmental, regulatory, social or community factors were identified as having potential to materially affect the construction, operation and decommissioning of the Project as described in this Technical Report.

Surface land tenure is noted as an area requiring special and ongoing attention. Squatting activities near the edges of the town of Concepcion del Oro on mineral concession areas is recognized as a potential issue, but thus far does not appear to have negatively affected the operation. As well, it is noted that a small parcel of land in the immediate area of TD5 is held by a third party and it will be necessary for Aura Minerals to construct and operate their facility without effects to this Property in the event that a land transfer does not proceed.

As demonstrated in Item 22, the Project exhibits attractive economics using base case price assumptions and at an 18.0:1.00 MXN:USD exchange rate. In addition, the Project economics are insulated somewhat from any modest downward pressures in metal prices, in particular, the copper price, due to the modest capital expenditures (US$32.1M in Year 1) required to restart the Project in attaining commercial production. Similarly, this allows a payback earlier than most development projects, not only from the reduced capital outlay, but also from the reduced time to reach commercial production (during Q1 Year 2). In addition, any further weakening of the MXN:USD exchange rate would also be beneficial on the Project's various metrics, in particular, on a NPV basis, as the expected increase in revenue from selling in US$ over LOM would outweigh the capital sensitivity in the first year.

Based on the entirety of the Project's analysis, resuming mining operations is recommended. However, the Project is sensitive to these prices and Aura Minerals should continue to evaluate future metal prices and consider various options to lock-in the metal prices to ensure continued viability of the Project.

25.2 RISK AND UNCERTAINTIES

25.2.1 Resource Estimate

The items below are considered to have a moderate risk level for current resource estimate:

- The Aranzazu deposit appears to be controlled by lithology and structures which resulted in some of domains having complex curvilinear shapes. The current model has nine mineralization domains; the number of domains could be increased to 30 with additional sub-domaining strategy.
- The assay density is high in or near to stopes and underground workings; while assay density decreases with depth.
- Unsampled intervals within the ore domains were assigned a median grade based on the logged LITHO type. This improved block model variability which is an inherent characteristic of the Aranzazu deposit.
- The default Specific Gravity (SG) values were applied to all missing intervals based on the mean and/or median combined lithological results.
- Pre-existing variograms were used to obtain the directions and ranges of continuity for anisotropic IDW estimation for all the nine domains.
- Classification was based on search radius (see above) but then adjusted according to inferred and indicated 'shells' that presided over measured and indicated. The 'shells' were based on confidence in areas and elevation. All blocks below 1750 masl were placed into the Inferred resource category. It is expected that there will be risks to the Project and several topics require further study. The risks identified herein are not the complete list of risks. They include only unusual risks related to technical issues.

25.2.2 Mining

The geotechnical data provided by Aura Minerals covers approximately two-thirds of the US$55/t NSR cut-off mineralized resource blocks, the data provided for geotechnical analysis should approach +90%. The geotechnical block model requires further refinement and validation from both underground maping and geotechnical logging of the additional drilling. Also, the final detailed stope-by-stope mining sequence and schedule should be geotechnically modelled to check the geotechnical integrity of the mine plan.

Other mining related items that should be reviewed in detail are:

- Comprehensive backfill study including mix design, strength testing and mixing/delivery procedures;
- More detailed haulage analysis from all areas;
- Further evaluation of the ventilation system based on the final mine plan and schedule;
- Analysis of ground support requirements and development of ground support management plan;
- Ongoing Hydrogeological investigations to validate the estimated inflow rates.

25.2.3 Metallurgy

- Arsenic levels in the ore during the final years of the LOM will require further evaluation for efficient control in the processing plant. It is envisioned that this risk can also be mitigated by effective blending with other low-arsenic bearing ores.

25.2.4 Infrastructure

25.2.4.1 Tailings Storage Facility TD1/TD2

Risks associated with recommissioning TD1/TD2 for tailings storage are:

- Performance of the TD1/TD2 facility is dependent on changes of the pore pressure regime and specifically how the phreatic surface will develop within the existing tailings embankment. If the phreatic surface exceeds those in the stability model, tailings deposition may need to be stopped prematurely. Instrumentation shall be installed to monitor the phreatic surface. A minimum 20m wide beach above water shall be developed and maintained for control of the phreatic surface.

25.2.4.2 Tailings Dam No. 5

Risks associated with Tailings Dam No. 5:

- Construction of a new municipal solid waste landfill is currently underway to accommodate waste from within the TD5 footprint and provide a waste disposal area for the municipality of Concepcion del Oro. Construction of the new landfill will need to be completed and waste from the TD5 impoundment area be relocated prior to commissioning TD5. Delays in construction/relocation of the new landfill could impact the construction or commissioning of TD5.
- Construction of Stage 1A of TD5 is planned for during the rainy season. Significant rain events could delay completion of the Stage 1A facility.
- The basis of the TD5 tailings storage facility was based on testwork conducted on historic tailings from other facilities on site. The grind of the TD5 tailings has since changed and the finer tailings may result in lower dry densities (and therefore lower tailings tonnage capacity), especially in the short-term until tailings can consolidate.
- Aranzazu will need to acquire a parcel of land downstream of the TD5 Stage 1 embankment to accommodate future downstream raises of the embankment. If this land cannot be acquired, alternatives for tailings storage beyond Stage 1 will need to be identified and evaluated.
- TD5 tailings storage capacity for Years 4 to the end of Project will require an additional licensing process and acquisition of additional property. Aranzazu is well-experienced in permitting, and management of this risk will entail a pro-active approach to design and planning for these later phases of TD5.

25.2.4.3 Tailings Dam No. 4

Risks associated with Tailings Dam No. 4:

- Due to the construction of the TD4 tailings facility on the historic Presa 4, the facility is exposed to a higher risk profile than is typically the case for more conventional tailings storage facilities. The design engineer provides Aura Minerals with the engineering design and guidance documents necessary to manage the risks associated with the construction and operation of TD4.
- As noted previously, a number of actions are currently underway to mitigate the geotechnical risks associated with short term deposition of tailings at TD4.

25.2.5 Metal prices

The NPV of the Project is sensitive to metal prices. As illustrated in previous sections, a 10% change in copper price would have the effect of changing the Project’s NPV by approximately US$28M with all other assumptions being equal.

26. RECOMMENDATIONS

26.1 GEOLOGY

The following recommendations have been made by Farshid Ghazanfari (Geological QP) regarding geology, QA/QC and the resource estimate. Prior to halting mining and exploration activities, Aura Minerals observed a few areas in the QA/QA process that must be addressed when operations resume; these suggestions have been included into the list below:

- The geological mapping of UG stopes and lateral development openings is the key to developing a short-term model. When mining operations recommence, detailed UG geological mapping needs to be done systematically to help better identify ore and waste boundaries while using the current NSR-based model.
- An additional high-grade field standard needs to be added to the suite of standards used previously as part of the QAQC program, especially for analyzing high grade gold and silver samples.
- Blank samples selected on site are not suitable as blank material as they contain weak mineralization. Rock that occurs well outside the deposit from a neighbouring limestone quarry needs to be collected and crushed. Ideally 10% of the blanks inserted into the exploration sample stream should be tested at the production lab to verify that they are in fact non-mineralized.
- Five percent of duplicates (from field, crush and pulp) should be sent to an independent laboratory and analyzed using the same methods as the primary laboratory. These results will be used as a duplicate check against bias. This laboratory should be in Canada, if possible.
- Standards need to be sent directly to ALS from SGS in pulp form.
- Previous ore control samples were not used in the resource estimation due to inherited error associated with survey location of samples. Proper ore control protocol needs to be established before start of operation
- For the current model, a US$45/t NSR cut-off was used to define limits of the mineralization. This limit was applied as a hard boundary to separate the higher-grade mineralization from the lower grade zone. The NSR model created several orphan zones, especially in the hanging wall of Mexicana and Glory hole zones, often with a different orientation compared to the main ore bodies. Implementation of sub-domains and different search orientations are needed to properly interpolate grade within these ore bodies in these areas.
- A rigorous geostatistical analysis has not been carried out for the current resource model. Aura Minerals needs to undertake a detailed geostatistical analysis derived from current 3D models using different orientations of the major domains and sub-domains.
- The NSR model is used as a hard boundary to avoid excessive dilution from adjacent lower grade zones in the Aranzazu mine; however, it would be beneficial for Aura Minerals to construct a lower grade shell that encompasses the new NSR-based wireframes for the next level of study. For future resource models Aura Minerals has been advised to use these grade shells for each zone and interpolate grade within lower grade shells using only a dataset outside of the current wireframes. This approach will be very beneficial for mining purposes including mine planning and calculation of dilution.
- A lower NSR cut-off (>30 US$/t) can be introduced to adjust the wireframe and produce an alternative model to investigate degree of bias (if any) compared to the current model.
- As geometry of the skarn is tending to be elusive, it is recommended that during definition drilling, some additional holes be designed to investigate the possibility of additional ore in the vicinity current designed stopes.

- Cabrestante zone, is classified as 'Inferred Mineral Resources' in the current model, while density of drilling maybe adequate to convert some of the resources to the 'Indicated Mineral Resources' category. Aura Minerals should investigate the possibility of opening the UG access to the Cabrestante zone and after this, the new classification scheme can be applied to this zone and some of the inferred resource may be converted to Indicated category.
- A significant portion of Inferred Mineral Resources in the current model are in Mexicana and Glory hole zones. UG development levels will provide an opportunity in the future to upgrade these resources and increase the life of mine.

26.2 MINING

- Cemented Rockfill testing was not completed prior to the writing of this Technical Report and it will be necessary to carry out a testing program to determine the appropriate cement content, aggregate size distribution, and mixing systems to achieve the recommended strength. The FS has identified enough quantities of suitable material within the existing waste dumps, but it will be necessary to select and prepare a bulk sample of the aggregate for testing. However, CRF is a commonly used backfill so all that remains is to determine the specifications prior to filling
- The delineation drilling should be planned and carried out such that it will occur at least one month ahead of the development of the stope top and bottom cross-cuts. The drill holes should be designed to ensure that each primary stope has at least one but preferably two drill hole intersection through it (including pre-existing resource diamond drill holes). Ideally the drill hole should be designed to intersect the ore zone at the elevation of the top and bottom cross-cuts
- Longitudinal stope designs. The majority of the Mineral Reserves have been designed as transverse stoping however there are opportunities to apply more longitudinal stoping. Mining longitudinal stopes has the advantage of potentially reducing development in waste as well as using main uncemented rockfill for backfill, both of which reduce costs and improve the operating margin. It is unlikely that longitudinal stopes can be more than 10 to 12 m wide, however geotechnical input will be required to determine what is likely to the stable
- The geotechnical database should be audited to determine if any additional instances of erroneous data exist and those errors fixed. The databased should be continually updated with new information and the results calibrated against underground mapping data
- A Ground Control Management Plan (GCMP) should be developed based on the designed recommendations in this Technical Report and submitted to the contactor to ensure understanding and compliance with the plan.
- The NSR value model was based on the proposed off-take terms and metallurgical recoveries from the 2015 study. New data will change the NSR valuation coefficients and the value of the ore blocks. The model should be updated according to the new NSR calculation
- Replacement ventilation raise for Robbins #3. As Robbins #3 is unlikely to be successfully rehabilitated, the development of new ventilation raise will be required to support the full LOM ventilation design. Potential locations will need to be identified and investigated both in terms of suitability for the ventilation design and favourable geotechnical characteristics. Once a final location has been selected, it is recommended that a geotechnical investigation drill hole be drilled prior to the drilling of the raise bore pilot hole
- The Mineral Reserves for this study focused primarily on the main zones of the mine and excluded other resource areas of the deposit (including satellite zones and areas considered remnants and/or too close to the old workings). These areas represent upside potential to both medium and long-term Mineral Reserves. Mine planning efforts should look to identify and examine whether these resources can be incorporated into the mine plan

- Significant inferred material exists in the GHFW zone below the 1740m level (lowest level of the LOM plan in this study) and represents the opportunity to extend the mine life. The mine planning work should begin to assess the mine design and development requirements as well as the impacts to the mine operations and infrastructure (haulage, ventilation, pumping, etc.)

26.3 METALLURGY

- Define a future metallurgical testwork to pursue other arsenic treatment routes for higher arsenic ores in Year 5 and Year 6.

26.4 ENVIRONMENT

- It is recommended that a comprehensive Closure Plan document and design be developed during the operating life of the Project.
- It is recommended that Aranzazu take a pro-active approach to design, planning and permitting of later phases of TD5 in order to ensure sufficient tailings capacity beyond Year 3.

26.5 INFRASTRUCTURE

26.5.1 Tailings Storage Facility TD1/TD2

Earthworks grading modifications of the upstream and downstream slopes and instrumentation installation are required at the TD1/TD2 facility prior to starting tailings deposition. Estimated costs of the facility modifications are US$0.3M.

26.5.2 Tailings Storage Facility TD4

Before the reactivation of the TD4 tailings storage facility, the buttress and toe-buttress need to be raised to the design level in accordance with the detailed recommendations provided previously to mitigate the risk of slope instability arising from pore pressure increases within the low strength historic tailings slimes immediately beneath TD4 and its buttress and toe-buttress.

26.5.3 Tailings Storage Facility TD5

Detailed design will be required for expansions to the TD5 tailings storage facility beyond Stage 1. Additionally, Aranzazu will need to acquire the necessary rights to the land and permits for dam raises subsequent to the Stage 1 tailings dam. Estimated cost for the detailed design of the tailings dam raises is US $0.2M. Estimated costs of consulting time for permit applications is US$30,000.

An Operations, Maintenance and Surveillance (OMS) manual should be developed prior to commissioning the TD5 tailings facility to outline the roles and responsibilities of personnel assigned to the TD5 facility, operational requirements, inspection and surveillance procedures and frequencies, and facility maintenance requirements for the safe and efficient operation of the tailings facility. The estimated cost to complete this task is US$20,000.

The mechanical design of the tailings pumping/piping and deposition and water reclaim system needs to be completed in a manner that is compatible with the design objectives of TD5 (e.g., maintain a wide tailings beach above water along the tailings dam). The estimated cost to complete this task is US$150,000.

26.6 COST OF RECOMMENDATIONS

Table 26-1 Summary of Recommendations

Recommendation	Estimated Cost (US$ 000')
Geology	120
Mining	180
Metallurgy	200
Infrastructure	500
Environment (Closure Plan)	500
TOTAL (US$)	**1,500**

27. REFERENCES

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28. QUALIFIED PERSONS CERTIFICATES

Certificates for the following Qualified Persons can be found under this Section 28.

- Farshid Ghazanfari P.Geo.
- Adam Wheeler C.Eng.
- Robert Dowdell C.Eng.
- Colin Connors R.M.
- Paul Cicchini P.E.
- Brett Byler P.E.
- Graham Holmes P.Eng.
- Cameron Scott P.Eng.
- Diane Lister P.Eng.
- Fernando Cornejo P.Eng.

FARSHID GHAZANFARI, P.GEO.

I, Farshid Ghazanfari, P. Geo., residing at 2135 Heidi Ave., Burlington, Ontario, L7M 3P4, do hereby certify that:

1. I am a Geological and Resource Estimation Consultant
2. I am a graduate of the Tehran University (Iran) having been awarded a M.Sc. (Hons.) Degree in Geology in 1992. My summarized career experience is as follows:

1990-1991: Mine Geologist and postgraduate student, Anguran Zinc Mine, Iran
1991-1993: Mineralogist, Geological Survey of Iran
1994-1997: Mineralogist, XRD/XRF Technologist and Field Geologist, Geo. Survey of Iran
1997-1998: Exploration Geologist, Minorco (Anglo-Gold), Tehran office, Iran
1999-2001: Exploration Geologist and Geological Consultant, Noranda Inc. Toronto, Canada
2001-2003: Geological Consultant, various companies
2003-2005: Exploration Geologist, Red Lake Mine, Goldcorp Inc.
2005-2006: Resource Modeling Geologist, Desert Sun Mining, Toronto, Canada
2006 -2009: Senior Resource Geologist , Largo Resources, Toronto ,Canada
2009-2013: Resource Modeling Consultant, Forbes and Manhattan, Toronto, Canada
2013-2018: Resource Modeling Consultant providing consulting services to the mining industry

1. I am a Professional Geologist in good standing with the Association of Professional Geologists of Ontario, License #1702.
2. I am responsible for Sections 7,8,9,10,11, 12, 14 and parts of 1, 25 and 26 of the technical report entitled “Feasibility Study (FS) of the Re-opening of the Aranzazu Mine, Zacatecas, Mexico” (The “Technical Report”) dated 31 January 2018.
3. I visited the Aranzazu mine in January 2018.
4. I have had prior involvement with the properties that are subject to the Technical Report.
5. I am an independent of the issuer prior to and as of the date of this certificate applying the test in Section 1.5 of NI-43101.
6. I have read NI 43-101 and Form 43-101F1 and the Report and the portion of the report for which I am responsible has been prepared in compliance therewith.
7. I am a “qualified person” for the purposes of NI 43-101 due to my experience and current affiliation with a professional organization (Professional Geologists of Ontario) as defined in NI 43-101.
8. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Effective date: January 31, 2018.
[SIGNED AND SEALED BY]

Farshid Ghazanfari, P.Geo.

ADAM WHEELER, C. ENG.

I, Adam Wheeler, C. Eng, do hereby certify that:

1. I am an independent mining consultant, based at Cambrose Farm, Redruth, Cornwall, TR16 4HT, England.
2. I hold the following academic qualifications:

B.Sc. (Mining) Camborne School of Mines 1981
M.Sc. (Mining Engineering) Queen's University (Canada) 1982

1. I am a registered Chartered Engineer (C. Eng and Eur. Ing) with the Engineering Council (UK). Reg. no. 371572.
2. I am a member in good standing of the Institute of Mining, Metallurgy and Materials (Member).
3. I have worked as a mining engineer in the minerals industry for over 35 years. I have experience with a wide variety of mineral deposits and reserve estimation techniques.
4. I have read NI 43-101 and the technical report, which is the subject of this certificate, has been prepared in compliance with NI 43-101. By reason of my education, experience and professional registration, I fulfil the requirements of a "qualified person" as defined by NI 43-101. My work experience includes 5 years as a mining engineer in an underground gold mine, 7 years as a mining engineer in the development and application of mining and geological software, and 23 years as an independent mining consultant, involved with evaluation and planning projects for both open pit and underground mines.
5. I am responsible for section 15, and co-authored section 16, of the Technical Report entitled "Feasibility Study (FS) of the Re-opening of the Aranzazu Mine, Zacatecas, Mexico" (The "Technical Report") and dated 31 January 2018 relating to the Aranzazu Property.
6. I visited the Aranzazu Mine in June and September of 2017.
7. As of the date hereof, to the best of my knowledge, information and belief, the technical report, which is the subject of this certificate, contains all scientific and technical information that is required to be disclosed to make such technical report not misleading.
8. I am independent of Aura Minerals Inc and its subsidiaries other than providing consulting services.
9. I consent to the filing of the report with any Canadian stock exchange or securities regulatory authority, and any publication by them of the report.

Effective date: January 31, 2018.
[SIGNED AND SEALED BY]

Adam Wheeler C.Eng.

ROBERT DOWDELL C. ENG.

I, Robert Stewart Dowdell, C Eng, reside at 14, The Fellside, Newcastle Upon Tyne, NE3 4LJ, United Kingdom, do hereby certify that:

1. I am an independent mining engineering consultant.
2. I graduated from the University of Newcastle upon Tyne with a B.Sc. (2.1 Hons) in Mining Engineering in 1965, and a Ph.D. in Rock Mechanics in 1968.
3. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfil the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 19 years as a mining engineer at various mines in Canada and overseas with Cominco Ltd, 1 year with an international consulting firm, 2 years with Geevor Plc, and 27 years as an independent mining consultant, involved in the evaluation and planning for both open pit and underground mines. See www.dowdell.co.uk for further details.
4. I am a registered Chartered Engineer (C.Eng.) with the Engineering Council (UK). Reg. Nº: 159780. I am also a member in good standing of the Institute of Mining, Metallurgy and Materials (Member).
5. I share responsibly for Sections 15 and 16 and parts of 1, 21 and 22 of this feasibility study "Feasibility Study (FS) of the Re-opening of the Aranzazu Mine, Zacatecas, Mexico" (The "Technical Report") and dated 31 January 2018 relating to the Aranzazu Property.
6. I have visited the Aranzazu mine on several occasions and spent in total around 90 days on site. Visits were made in July, Aug and Sept 2013; Jan and Dec 2014; April, May, July and Sept 2017; Jan and May 2018.
7. I have had no prior involvement with the properties that are subject to this feasibility study.
8. I am an independent of the issuer prior to and as of the date of this certificate applying the test in Section 1.5 of NI-43101.
9. I have read NI 43-101 and Form 43-101F1 and the Report and the portion of the report for which I am responsible has been prepared in compliance therewith.
10. As of the date of this certificate, to the best of my knowledge, information and belief, the feasibility study contains all scientific and technical information that is required to be disclosed to make the feasibility study not misleading.
11. I consent to the filing of the feasibility study with any Canadian stock exchange or securities regulatory authority, and any publication by them of the feasibility study.

Effective date: January 31, 2018.
[SIGNED AND SEALED BY]

Robert Dowdell C.Eng.

PAUL CICCHINI P.E.

1. I, Paul F. Cicchini, do hereby certify that I am a co-founder and senior principal of Call & Nicholas, Inc. (Geotechnical Engineering) with an office at 2475 N. Coyote Drive, Tucson, Arizona, USA.
2. I graduated with a Bachelor of Science in Geological Engineering from the University of Arizona in 1979.
3. I am a Registered Professional Engineer in the States of Arizona (Geological No.19629), Utah (Mining No.9590636-2202), Washington (Mining No.53170) and Alaska (Civil No.108950).
4. I am a member of the American Rock Mechanics Association, the Association of Engineering Geologists, the International Society of Rock Mechanics and the Society for Mining, Metallurgy, and Exploration.
5. I have practiced my profession continuously since 1979 and have worked as an international geotechnical engineering consultant for a total of 39 years. Most of my professional practice has focused on the geotechnical aspects of underground and surface mining ranging from scoping level studies thru feasibility, operation and closure in North America, Mexico, Central America, South America, Indonesia, Philippines, South Africa, Middle East, Europe and the Arctic.
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 16 (geotechnical aspects) in the Technical Report titled "Feasibility Study (FS) of the Re-opening of the Aranzazu Mine, Zacatecas, Mexico" dated 31 January 2018.
8. I last visited the Aranzazu Mine on July 3, 2017 for 3 days.
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.
11. I am independent of the Issuer applying the test in Section 1.5 of NI-43101.

Effective date: January 31, 2018.
[SIGNED AND SEALED BY]

Paul F. Cicchini, P.E.

COLIN CONNORS, RM-SME

1. I, Colin F. Connors., do hereby certify that I have been employed by the Issuer since July 2017 and am currently Director of Mining with Aura Minerals Inc.; located at 161 Bay St. Suite 2700, Toronto, Ontario, M5J 2S1
2. I graduated with a Bachelor of Applied Science in Mining and Mineral Process Engineering from the University of British Columbia (Vancouver, B.C.) in 1988 and with a Master of Science in Mining Engineering from Queen's University (Kingston, Ont.) in 1993.
3. I am a Register Professional Member of the Society of Mining Engineers (SME) License #4133660
4. I have practiced my profession continuously since 1993 in a range of technical, operational and consulting roles in Canada, United States, Russia, Brazil, Ghana and Mexico.
5. 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.
6. I am co-responsible for Sections 15, 16, and parts of Sections 14, 18, 21, 22, 25, and 26 in the Technical Report titled "Feasibility Study (FS) of the Re-opening of the Aranzazu Mine, Zacatecas, Mexico" dated January 31, 2018.
7. I am non-independent of the Issuer applying the test in Section 1.5 of NI 43-101.
8. I visited the Aranzazu Mine several times in 2017 and 2018, with a most recent visit dated May 2018.
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.
11. I consent to the filing of the report with any Canadian stock exchange or securities regulatory authority, and any publication by them of the report.

Effective date: January 31, 2018.
[SIGNED AND SEALED BY]

Colin F. Connors, RM-SME

GRAHAM P. HOLMES, P.ENG.

1. I, Graham P. Holmes of Mississauga, ON, Canada, do hereby certify that I am a Specialist Mineral Processing Engineer with Jacobs at 12 Wenonah Dr. Mississauga ON.
2. I graduated with a Bachelor of Mining Engineering with Mineral Technology Option, from the Royal School of Mines, Imperial College, London University in London UK in 1966.
3. I am a member of the Association of Professional Engineers of Ontario.
4. I have practiced my profession continuously since 1966 in a range of operational, technical and consulting roles.
5. 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.
6. I am responsible for Sections 13 and 17 in the Technical Report titled "Feasibility Study (FS) of the Re-opening of the Aranzazu Mine, Zacatecas, Mexico" dated 31 January 2018, (the "Report").
7. I last visited the Aranzazu Mine site in January 2018 for 3 days.
8. 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.
9. 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.
10. I am independent of the Issuer applying the test in Section 1.5 of NI-43101.

Effective date: January 31, 2018.
[SIGNED AND SEALED BY]

Graham P. Holmes, P.Eng.

CAMERON C. SCOTT, P.ENG.

1. I, Cameron C. Scott, do hereby certify that I am a Principal Consultant (Geotechnical Engineering) with the firm SRK Consulting (Canada) Inc. (SRK) with an office at Suite 2200, 1066 West Hastings St. Vancouver, British Columbia, Canada.
2. In 1974, I obtained a B.A.Sc. Degree in Geotechnical Engineering from the University of British Columbia. In 1984, I obtained a M.Eng. Degree in Civil Engineering (Geotechnical Option) from the University of Alberta.
3. I have been a Professional Engineer registered with the Association of Professional Engineers and Geoscientists of British Columbia (#11523) since 1978.
4. have practiced my profession continuously since 1974 and have worked as a Geotechnical Engineer for a total of 44 years. Most of my professional practice has focused on the Geotechnical and Hydrogeological aspects of mining, including the site selection, design, permitting, operation and closure of mine waste facilities in Canada, US, Mexico, Central and South America, Europe and various countries within the Former Soviet Union.
5. 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.
6. I am responsible for Sections 18.2 and parts of Sections 25 and 26 in the Technical Report titled “Feasibility Study (FS) of the Re-opening of the Aranzazu Mine, Zacatecas, Mexico (the “Technical Report”) dated 31 January 2018.
7. I last visited the Aranzazu Mine on June 28, 2018 for 2 days.
8. 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.
9. 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.
10. I am independent of the Issuer applying the test in Section 1.5 of NI-43101.

Effective date: January 31, 2018
[SIGNED AND SEALED BY]

Cameron C. Scott, P.Eng.

BRETT BYLER P.E.

1. I, Brett Byler, do hereby certify that I am an Associate Geotechnical Engineer with Wood Environment and Infrastructure Solutions, Inc. (Wood) with an office at 2000 South Colorado Blvd, Ste 2-1000, Denver, Colorado, USA.
2. This certificate applies to the technical report titled “Feasibility Study (FS) of the Re-opening of the Aranzazu Mine, Zacatecas, Mexico (the “Technical Report”) dated 31 January 2018.
3. I have been a Registered Professional Engineer in the State of Colorado (#39291) since 2005.
4. I graduated with a Bachelor of Science in Geological Engineering from the Colorado School of Mines in 1995 and with a Master of Science in Civil Engineering from the University of Colorado at Boulder in 2003.
5. I have practiced my profession for 21 years. I have been directly involved in studies, design, permitting, construction, operations and expansion of tailings storage facilities, heap leach pads and other mine infrastructure. My experience includes site selection studies, geotechnical investigation and characterization, instrumentation and performance monitoring, risk analyses, dam safety inspections and due diligence audits.
6. As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101) for the sections of the Technical Report that I am responsible for preparing.
7. I most recently visited the Aranzazu Project site on June 27 and 28, 2018. I also visited the site on May 19 and 20, 2017.
8. I have had prior involvement with the Aranzazu Property. I was retained by the issuer to participate in a risk assessment workshop for tailings dam TD4 in 2017.
9. I am responsible for Items 18.1, 18.3, 25.2.4.1, 25.2.4.2, 26.5.1 and 26.5.3, as pertaining to the tailings storage facilities TD5 and TD1/TD2 of the Technical Report.
10. I am independent of Aura Minerals, Inc. as independence is described by Section 1.5 of NI 43-101.
11. At the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
12. 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.

Signature date: January 31, 2018
[SIGNED AND SEALED BY]

Brett R. Byler, P.E.

DIANE LISTER, P.ENG.

1. I, Diane Lister of Marsh Lake, Yukon, Canada, do hereby certify that I am a consulting environmental engineer and principal of Altura Environmental Consulting with business address at 18 Michie Place, Marsh Lake, Yukon.
2. I graduated with a Bachelor of Applied Science in Geological Engineering in 1989 and with a Master of Applied Science in Mining Engineering in 1994 from the University of British Columbia, Vancouver B.C.
3. I am a Professional Environmental 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).
4. I have practiced my profession continuously since 1994 in a range of operational, technical and consulting roles.
5. 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.
6. I am responsible for Items 4, 5, 6, 20 and part of Items 25 and 26 in the Technical Report titled "Feasibility Study (FS) of the Re-opening of the Aranzazu Mine, Zacatecas, Mexico (the "Technical Report") dated 31 January 2018.
7. I, as the qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101 Standards of Disclosure for Mineral Projects.
8. I most recently visited the Aranzazu mine from August 21-23 2017. Previous site visits took place on July 30, 2015, January 28-29, 2010, June 23-24, 2009, December 9-10, 2008, September 25-27, 2008, April 17-18, 2008, and March 12, 2008.
9. I have had prior involvement with the Aranzazu Property. I was first retained by the issuer to assist with due diligence during Property acquisition in 2008, and from 2008 to 2010 I was periodically retained to provide input on various environmental management matters. In 2015 I was retained as a Qualified Person for the 2015 Technical Report of the Aranzazu Property.
10. 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.
11. 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: January 31, 2018.
[SIGNED AND SEALED BY]

Diane Lister, P.Eng.

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 161 Bay St. Suite 2700, Toronto, Ontario, M5J 2S1.
2. I graduated with a Bachelor Degree in Chemical Engineering, from Universidad Nacional de San Agustin, Arequipa, Peru in 2001 and a Masters Degree in Chemical Engineering from Ecole Polytechnique de Montreal, Canada in 2005.
3. I am a Professional Engineer registered with the Professional Engineers of Ontario.
4. I have practiced my profession since 2001 in a range of operational, technical and mineral processing consulting roles in Canada, Mexico and Peru.
5. 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.
6. I am responsible for Sections 1, 2, 3, and parts of 18, 21, 22, 25 and 26 in the Technical Report titled "Feasibility Study (FS) of the Re-opening of the Aranzazu Mine, Zacatecas, Mexico (the "Technical Report") dated 31 January 2018.
7. I have had prior involvement with the Aranzazu Property. I was first retained by the Issuer to develop the Basic Engineering Study of the Aranzazu Expansion from October 2012 to August 2013 while employed as a Project Manager by Jacobs Engineering, Toronto, Canada.
8. I am non-independent of the Issuer applying the test in Section 1.5 of NI 43-101.
9. I visited the Aranzazu Mine several times throughout the years and last visit was on January 2018.
10. 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.
11. 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: January 31, 2018
[SIGNED AND SEALED BY]

Fernando A. Cornejo, P.Eng.