MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES ON THE SAN ANDRÉS MINE IN THE MUNICIPALITY OF LA UNIÓN, IN THE DEPARTMENT OF COPÁN, HONDURAS

NI 43-101 Technical Report

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1.0 SUMMARY

1.1 Introduction

Aura Minerals Inc. ("Aura" or the "Company") has prepared a technical report (the "Report") compliant with National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") on the updated Mineral Resources and Mineral Reserves pertaining to its San Andrés mine (the "Mine" or the "Project").

1.2 Project Description and Location

The Mine is an open pit, heap-leach operation located in the highlands of western Honduras, in the municipality of La Unión, Department of Copán approximately 210 km southwest of the city of San Pedro Sula. The Mine's surface and mineral rights are owned by Minerales de Occidente, S.A. de C.V. ("Minosa"), a wholly-owned indirect subsidiary of Aura existing under the laws of Honduras.

1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

Access to the Mine is via paved highways and gravel roads approximately 210 km from San Pedro Sula or 360 km from the capital city of Tegucigalpa. Both cities are serviced by international airports with daily flights to the United States of America and cities in Latin America.

The Mine is located approximately 18 km due west of the town of Santa Rosa de Copán, the capital of the Department of Copán. The town site and property of San Andrés is reached via a 28 km paved highway from Santa Rosa de Copán, and then by a 22 km gravel road from the turn-off at the town of Cucuyagua. The gravel road is public, but Minosa assists local authorities with the maintenance of this road.

The climate of San Andrés is temperate, with a distinct rainy season locally called winter from May to November. Although parts of Honduras lie within the hurricane belt, the western Interior Highlands are generally unaffected by these storms.

Temperature decreases with increased elevation and as the Mine site is situated at an elevation of 1,200 m, the climate is quite temperate. Typically, December and January are the coolest months, with average daily temperatures of 17.9°C and 17.8°C, respectively. April and May are typically the warmest months, with average temperatures of about 22°C.

There are a number of mines operating in Honduras and throughout Central America. These mining operations are supplied and serviced by branch offices and facilities of international contractors and suppliers and by domestic contractors and suppliers. Cement and fuel are provided locally by Honduran companies. Spare parts and supplies from major centers in North or South America can be readily delivered to the site within a reasonable time.

Labour is sourced locally from the many communities located near the Mine. Educational, medical, recreational, and shopping facilities are established. Management and technical staff are available within Central America and from North or South America as is required. Aura also maintains a corporate office in Canada of experienced geologists and engineers to provide technical support and oversight for all of its projects, including the Mine.

The Mine has been in operation since 1983, and has a well-developed infrastructure which includes power and water supply, warehouses, maintenance facilities, assay lab and on-site camp facilities for management, staff and contractors. On-site communication includes radio, telephone, internet and satellite television services. Process water is supplied by rainwater run-off collected in a surge pond and by direct pumping from a water well pump station in the perennial Río Lara adjacent to the carbon-in-column adsorption, desorption and recovery plant ("CIC-ADR"). Chlorinated potable water for the town of San Andrés and camp facilities is supplied from a source originating upstream from San Andrés along the Río Lara, near the village of La Arena. Purified water for drinking and cooking is purchased from local suppliers.

1.4 History

The San Andrés property was explored in the 1930s and 1940s by numerous companies including Gold Mines of America and the New York and Honduras based Rosario Mining Company ("Rosario"). In 1945, the property was acquired by the San Andrés Mining Company and then purchased by the New Idria Company ("New Idria") (Malouf, 1985). A 200 short tons-per-day cyanide circuit was installed in 1948. Approximately 300,000 short tons of surface and 100,000 short tons of underground ore averaging 5.8 g/t Au were mined and milled by New Idria. In 1949, San Andrés became the first operation to use a carbon-in-pulp plant to recover gold and silver by adsorption using granular carbon, however, numerous problems including poor air travel support logistics and high underground mining costs caused the operation to close in 1954 (Marsden and House 2006). The area remained inactive until it was reopened in 1974 (Malouf, 1985).

In 1974, an exploration permit was granted to Minerales, S.A. de C.V. ("MINSA"), a Noranda Inc. subsidiary. MINSA then joint-ventured the property with Rosario and exploration efforts consisted of soil sampling, mapping and trenching with the purpose of identifying a large, disseminated, open pit gold deposit. Changes in the Honduran tax law forced MINSA to drop the concession in 1976. Compañía Minerales de Copán, S.A. de C.V. ("Minerales de Copán") acquired the property in January 1983 following changes in the Honduran tax laws. A 60 short tons-per-day heap leach operation was installed and 170 local residents were employed on a basic, shovel-and-wheel-barrow operation.

In 1993, Fischer-Watt Gold Company Inc. ("Fischer-Watt") acquired an option from Minerales de Copán to further explore the property. Fischer-Watt conducted additional mapping and sampling programs with encouraging results.

In 1994, Greenstone Resources Ltd. ("Greenstone") acquired the option from Fischer-Watt. The option was exercised in 1996 and Greenstone subsequently acquired in excess of 99% of Minerales de Copán. Feasibility studies began in 1996, and in 1997 Greenstone completed a feasibility study that evaluated mining the Water Tank Hill deposit. Proposed production was 2.1 million tonnes per annum ("Mtpa"), with the mine life estimated at seven years. The facilities were constructed to handle in excess of 3.5 Mtpa of ore and waste.

Following review and approval of the Environmental Impact Assessment ("EIA") for the mine, Greenstone Minera de Honduras, S.A. de C.V., Greenstone's wholly-owned Honduran subsidiary company, received the mining permit on December 9, 1998 and began mining in early 1999. Their first shipment of gold was on March 30, 1999. Due to cash flow problems within Greenstone, mining and crushing operations ceased at the Mine in mid-December 1999.

Greenstone subsequently defaulted of its obligations to its secured creditor, the Honduran Bank, Banco Atlántida, and the property rights and obligations associated with the mine were transferred to Banco Atlántida. Banco Atlántida formed Minosa to own and operate the Mine and on June 26, 2000 Banco Atlántida's real estate branch provided a bridge loan to Minosa for operations to resume. RNC Gold Inc. ("RNC") was retained to provide management services to Minosa, and mining operations resumed in early August 2000 at the Water Tank Hill deposit. The Water Tank Hill pit was depleted in early 2003 and production commenced in the East Ledge pit in March 2003.

On September 7, 2005, RNC purchased 100% of the Mine through the acquisition of 100% of Minosa. On February 28, 2006, Yamana Gold Inc. ("Yamana") acquired RNC and a 100% beneficial interest in Minosa, which was then acquired by Aura on August 25, 2009.

A summary of the historical and recent production at the Project by year is set out in Table 1-1 below.

Table 1-1: Historical and Recent Production

Year	Ore Leached Tonnes	Grade Au g/t	Gold Recovered (Oz)	Silver Recovered (Oz)
1983	21,480	-	-	-
1984	22,459	2.12	1,388	575
1985	22,332	2.46	1,433	636
1986	29,120	3.08	2,510	750
1987	40,178	2.46	2,710	806
1988	56,154	2.21	2,957	803
1989	76,209	1.87	3,406	1,247
1990	105,598	1.37	3,495	1,120
1991	133,084	1.93	4,813	1,385
1992	129,647	1.09	3,737	944
1993	138,766	1.15	4,607	1,100
1994	138,083	1.06	4,291	739
1995	130,956	0.93	3,482	708
1996	127,801	1.21	4,504	1,242
1997	42,885	0.87	1,048	262
1998	-	-	-	-
1999	1,357,544	2.04	42,455	44,392
2000	-	-	6,006	7,477
2000	719,631	1.85	17,508	22,841
2001	2,289,276	1.75	105,998	131,201
2002	3,378,116	1.09	99,064	108,694
2003	2,891,890	0.63	50,795	35,421
2004	3,793,870	0.69	65,032	18,502
2005	3,392,092	0.72	61,236	16,488
2006	3,732,049	0.70	70,779	-
2007	2,910,904	0.52	51,240	34,992
2008	3,567,279	0.58	47,761	17,636
2009	4,530,009	0.68	68,372	34,406
2010	4,913,900	0.70	70,641	52,394
2011	4,312,947	0.68	60,871	38,208
2012	4,372,598	0.61	59,751	41,487
2013	5,370,142	0.58	63,811	34,765

1.5 Geology and Mineralization

The gold deposits at the Mine are hosted within Tertiary-aged felsic volcanic flows, tuffs and agglomerates, thick inter-bedded silica breccias, primarily containing volcanic fragments and tuffaceous sandstones. These volcanic units occur on the south (hanging wall side) of the San Andrés Fault. The fault strikes west-east and dips at 60° to 70° south and it marks the northern boundary of the Water Tank Hill and East Ledge pits. The fault forms the contact between the Permian phyllites (metasediments) to the north and the volcanic units on the south. Mineralisation within the phyllites is limited to the Buffa Zone where quartz carbonate veining proximal to the San Andrés Fault. South of the

Mine area, where there is no alteration, the volcanic and sedimentary rocks have a distinctive hematite brick red color but, in the Mine area, they have been bleached to light buff yellow and grey colors due to alteration. The younger volcanic and sedimentary units typically have a shallow to moderate southerly dip and thicken to the south of the Mine area.

Structurally, the Mine area is transected by a series of sub-parallel, west to northeast-striking faults that are typically steeply dipping to the south and by numerous north and northwest-striking normal faults and extension fractures. The most prominent fault of the first set is the San Andrés Fault. The San Andrés Fault is parallel to, and coeval with, a major set of west to north-northeast trending strike-slip faults that form the Motagua Suture Zone, which is continuous with the Cayman Trough. The Motagua Suture Zone and the Cayman Trough result from the movement between the North American plate and the Caribbean plate. The direction of movement along these strike-slip faults, including the San Andrés Fault, is left lateral.

The normal faults and extension fractures occur within the volcanic and sedimentary units on the south side of the San Andrés Fault. Average strike of these structures is N25°W; dip is 50° to 80° to the southwest and northeast, forming grabens where the strata are locally offset. These faults and fractures are generally filled with banded quartz and blade calcite and have formed focal points the alteration and mineralisation fluids within the Mine area. These extensional structures are distributed over a wide area, from the East Ledge open pit to Quebrada Del Agua Caliente, approximately 1,500 m to the east, and from the San Andrés Fault, for at least 1,200 m south and are coeval with the strike-slip faults.

There are abundant occurrences of hot springs throughout Honduras and hot springs occur within the immediate vicinity of the Mine. These geothermal systems are most likely caused by thin crust and high regional heat flow resulting from the rifting associated with the Suture Zone. The hot springs are neutral to alkaline in pH and range in temperature from 120°C to 225°C. The high-temperature springs are currently depositing silica sinter with cooling. Structurally, the hot springs are associated with the northwest-trending extensional faults and fractures.

The San Andrés deposit is classified as an epithermal gold deposit associated with extension structures within tectonic rift settings. These deposits commonly contain gold and silver mineralization, which is associated with banded quartz veins. At the Mine, however, silver does not occur in significant economic quantities. Gold occurs in quartz veins predominantly comprised of colloform banded quartz (generally chalcedony with lesser amounts of fine comb quartz, adularia, dark carbonate, and sulphide material). The gold mineralization is deposited as a result of the cooling and interaction of hydrothermal fluids with groundwater and the host rocks. The hydrothermal fluids may have migrated some distance from the source; however, there is no clear evidence at the Mine that the fluids or portions of the fluids have been derived from magmatic intrusions.

The rocks hosting the San Andrés deposit have been oxidized near surface as a result of weathering. The zone of oxidation varies in depth from 10 m to more than 100 m. The zone of oxidation is generally thicker in the East Ledge deposit compared to the Twin Hills deposit.

In the oxide zone, the pyrite has been altered to an iron oxide such as hematite, goethite, or jarosite. The oxide zone generally overlies a zone of partial oxidation, called the mixed zone, which consists of both oxidized and sulphide material. The mixed zone may not occur continuously, but where it is present, it reaches thicknesses of over 50 m. below the zone of oxidation; the gold is commonly associated with sulphide minerals such as pyrite. The sulphide, or “fresh”, zone lies below the mixed zone.

The gold contained in the oxide zone is amenable to extraction by heap leaching using a weak cyanide solution. The gold recovery is reduced in the mixed zone as a result of the presence of sulphide minerals and the gold cannot currently be recovered economically from the sulphide zone by heap leaching. The estimated metal recovery by leaching from each zone is discussed in Section 17.

High clay content in the ore, resulting from alteration, is detrimental to the heap leaching process because of reduced through-put rates in the crushing plant and reduced permeability in the heap leach operation. This poor leaching situation is resolved by agglomerating the crushed ore by adding cement to increase the permeability of the heap prior to leaching.

Based on metallurgical studies, the gold is primarily contained in electrum as fine-grained particles. The particle size of the electrum grains varied from 1 micron ("μ") x 1 μ up to 10 μ x 133 μ. One native gold grain was noted. The silver generally occurs at about the same grade as gold and the correlation between silver and gold is low at 0.24. Silver is not considered important because of the lower price for silver compared to gold and the lower metal recovery of silver.

1.6 Exploration, Drilling, Sampling, Analysis, and Data Verification

Since the acquisition of Minosa by Aura on August 25, 2009, exploration activities conducted at the Project by Minosa personnel consists of property scale mapping, road cut channel sampling and a limited reverse circulation ("RC") drilling program in the Twin Hills Pit. During 2012, a new RC drilling programme was commenced in the Cerro Cortez and Cemetery areas for improving Mineral Resource and Mineral Reserve definition, this programme continued throughout 2013.

The following is a summary of exploration activities carried out at the Project by previous owners.

The drill hole database for the Mine, including condemnation drilling and drilling conducted prior to 1994 on the Water Tank Hill, consists of 740 drill holes for a total of 100,365 m.

Aerial photography was flown over the Project on March 31, 1996 by Hansa Luftild German Air Surveys of Munster, Germany. The aerial photographs were ortho corrected using seven ground control points and digital topographic maps with two-metre contour intervals created by Eagle Mapping Services Ltd. of Vancouver, British Columbia, Canada. The digital topography was used by Minosa in the design of the East Ledge and Twin Hills block models and resulting pit designs.

During 1997 and 1998 Greenstone carried out geological mapping and sampling that collected 1,700 bedrock channel samples from road cuts and outcrop exposures on the property. The results of this work helped to develop the geological model, define mineralized zones and define drill targets. As well, Quantec IP Inc. of Toronto, Ontario, Canada conducted induced polarization and magnetometer geophysical surveys consisting of 27.7 km, with readings at 12.5 m stations along lines 50 m apart, covering the Project from Water Tank Hill to south of Twin Hills and to the east over Cortez Ridge inside the San Andrés concession. The surveys identified four targets, three in a north to south corridor between Cerro Cortez and Twin Hills and a fourth located south of Water Tank Hill. Two of the targets have been mined and the third was drilled by Greenstone (SC-034) and intersected mineralization from surface to a depth of 50 m with individual sample grades up to 3.26 g/t Au with the remainder of the hole relatively barren. The fourth target on the east side of Cerro Cortez has not been drilled.

Geological mapping at 1:1,000 scales was conducted on the 1,150 m bench level of the Water Tank Hill pit in 2001. Mapping of the East Ledge pit high wall was conducted between the 1,120 m and 1,060 m elevations (11 benches) as the East Ledge pit was advanced from July through December 2004. The results of the mapping were used to assess the mineralization controls and the structural complexities of the deposit as well as for use in the geotechnical monitoring of the East Ledge Pit high wall. Geotechnical monitoring and geological mapping are continuing.

Drilling was initially carried out on the Water Tank Hill area because of the historical production from the area. The Twin Hills deposit was discovered in 1994 and the East Ledge deposit was discovered in 2001. Most of the drilling at the Project has been RC drilling.

Geological mapping and channel samples were completed in adjacent areas in 2010 and 2011 along with a RC drilling programme. Drilling targeted the Twin Hill South, Banana Ridge, Fault A, Cerro Cortez, Zona Buffa and Agua Caliente areas, totaling 6,209 m. The exploration program helped to develop the geological model and define future targets for infill drilling.

In 2012 and 2013, the RC drilling campaign conducted by Minosa was largely focused in Cerro Cortez and Cemetery areas.

A summary of the historical drilling at the Project by year and by drilling method is set out in Table 1-2 below.

Table 1-2: Summary of the Historical Drilling at the San Andrés Project

Company	Year	RC Holes		Core Holes		Total	
		No. of Holes	Metres	No. of Holes	Metres	No. of Holes	Metres
Fischer-Watt	EX-1992	22	2,717.40			22	2,717.40
Greenstone	EX-1994	63	5,008.30			63	5,008.30
	EX-1996	41	5,920.50			41	5,920.50
	EX-1997	101	11,601.40	9	1,323.5	110	12,924.90
	EX-1998	150	18,437.90	37	4536	187	22,973.90
Minosa	EX-2001	15	1,674.00			15	1,674.00
	EX-2002	49	6,306.50			49	6,306.50
	EX-2005	25	2,280.00			25	2,280.00
Minosa-Yamana	EX-2006	113	17,639.20	12	2,566.1	125	20,205.30
	EX-2007	59	8,316.10	28	6,253.4	87	14,569.50
	EX-2008	12	1,900.10	22	4,838.8	34	6,738.90
Minosa-Aura	EX-2010/2011	64	3,508.20			64	3,508.20
	SA-2010	9	426.8			9	426.8
	EX-2012	64	8,014.70			64	8,014.70
	SA-2012	21	853			21	853
	EX-2013	75	8,805.70			75	8,805.70
	SA-2013	22	1,400.6			22	1,400.60
	Total	**905**	**104,810.30**	**108**	**19,517.8**	**1013**	**124,328.1**

The RC and core drilling programs were designed to sample the entire oxide and mixed zones. Holes were generally drilled from 150 m to 200 m in depth and stopped in the sulphide zone. Some holes were drilled to sample the sulphide mineralization.

The RC sample collection procedures have been documented by Chlumsky, Armbrust, & Meyer L.L.C. ("CAM") (Armbrust et al., 2005) and by Scott Wilson RPA (2007). Samples were collected continuously from the collar to the end of the hole at 1.5 m intervals. The weight of the drill cuttings was measured and then the sample was split using a Gilson splitter and reduced to two samples of approximately 5 kg each and retained in poly bags marked with the sequence number, hole number and depth. One sample was then transported to the Mine assay lab for sample preparation and the other sample was sent to a secure storage facility for future reference. Every 20th sample was split for a duplicate assay check. All sampling was carried out by Company employees. A QA/QC program consisted of the use of duplicate samples, standards, and blanks. These QA/QC samples were inserted to assess the sample accuracy, the assay accuracy and to determine if there was cross contamination between samples.

At the San Andrés lab, the RC samples were recorded in a sample book, oven dried at 60°C, then crushed using a jaw crusher to approximately minus ¼-inch and a 50 g to 60 g subsample split was taken using a riffle splitter. The subsample was pulverized in a ring-mill pulveriser to 90% passing a 150 mesh screen. The pulverized sample or pulp was rolled and a sample was split off for fire assay. The pulps were packaged in plastic bags and then transported from the Mine site to Minosa offices in Santa Rosa de Copán and then shipped using an independent courier service to CAS de Honduras, S. de R.L. laboratory in Tegucigalpa ("CAS").

The samples collected for the 2012 and 2013 drilling campaign were prepped and assayed on site using the site lab with regular check samples sent to an independent lab operated by Inspectorate America Corporate ("Inspectorate"). Samples were shipped to the Inspectorate prep-laboratory in Guatemala for sample preparation and then to Reno, USA for analysis.

Core sample intervals were determined by the geologist, and were based on changes in rock type or structure, and ranged in length from 0.5 m up to 3.0 m. The sample intervals were clearly marked on the core prior to splitting. The core was sawn in half with a diamond saw, with one half being retained for reference and the other being submitted for sample preparation and assay. All sampling was conducted by Company employees. The sawn core samples were then transported from the Mine site to the Company offices in Santa Rosa de Copán and then shipped using an independent courier service to CAS.

Several different North American laboratories were used to assay the San Andrés samples, with the exception of the East Ledge drilling program by Minosa in 2001, 2002 and 2012 and Twin Hills and Cerro Cortez programs between 2010 and 2012, where the samples were analyzed in the Mine on-site lab. Fischer-Watt used American Assay Lab in Sparks, Nevada, USA during their 1992 drilling program. Greenstone started out by using Chemex Labs ("Chemex") located in Mississauga, Ontario, Canada, but switched to Barringer Assay Lab in Reno, Nevada, USA ("Barringer") in January 1998 (starting with RC hole SA-232 and core hole SC-5). In April 1997, a new procedure was initiated to reduce air freight costs where all samples were submitted first to McClelland labs in Tegucigalpa, Honduras, for partial sample preparation. At McClelland, the five kilogram samples were dried, crushed to -10 mesh and an 800 g to 1,000 g subsample produced. The subsample was then forwarded to a North American assay lab for final sample preparation and assay analysis.

All samples were analyzed for gold and most samples were analyzed for silver by fire assay methods with an atomic absorption spectroscopy ("AA") finish using a 29.162 g (1 assay-ton) sample. Except for the very early work (i.e., Fischer-Watt program), metal values were reported in g/t Au. All original assay certificates are on file on site.

The sample preparation and analytical procedures at both McClelland and the North American assay labs follow industry standards. The sample was dried in an oven at 60 °C, and then crushed to approximately -10 micron mesh. The crusher yielded a product where greater than 80% of the sample passed through a -10 micron mesh screen. A 200-400 g sub-sample was split off using a Jones Riffle Splitter, and the remaining portion of the -10 micron mesh reject was bagged and saved. The 200-400 g split was pulverized in a ring and puck pulveriser. The specification for this procedure was at least 90%

passing a -150 micron mesh screen. The pulverized sample (pulp) was rolled on a rolling cloth until fully homogenized and a 29.166 g (1 assay-ton) sample was split off for fire-assay.

Gold analysis was done by fire-assay with an AA finish. The sample was fused with a natural flux inquarted with 4 mg of gold-free silver and then cupelled. Silver beads were digested for 90 minutes in nitric acid to remove the silver, and then 3 ml of hydrochloric acid was added to digest the gold into solution.

The samples were cooled, made to a volume of 10 ml, homogenized and analyzed by AA for gold. Silver analysis was performed on a prepared sample that was digested in a hot nitric-hydrochloric acid mixture, taken to dryness, cooled and then transferred into a 250 ml volumetric flask. The final matrix was 25% hydrochloric acid. The solutions were then analyzed by AA.

1.7 Metallurgical Testing

The East Ledge deposit was assessed using bottle roll tests. Although bottle roll tests provide an indication that the ore is amenable to heap leaching, the tests do not provide quantitative estimates of the percent recovery. In the case of the East Ledge deposit, the recovery factors are based on production results. Historical production results between January 2003 and September 2007 indicate an overall recovery from the East Ledge deposit of 84%.

The Twin Hills deposit was assessed using a combination of bottle roll and column tests. Overall, column leach test data indicates that the Twin Hills bulk oxidized ore is readily amenable to heap leaching. Recoveries of 86.5%, 87.5%, and 87.2% in 68 days of cyanide solution contact were achieved from samples with a P_{80} of 3 inch, 1 inch, and ½ inch, respectively. Gold recovery rates were fairly rapid for all feed sizes, and extraction was substantially complete in 10 to 15 days of leaching. Additional gold was extracted after 15 days, but at a much lower rate.

Although the column test on the mixed zone from the East Ledge pit indicated a gold recovery of 43%, the test was conducted on coarse material (P_{80} of 2.5 inch) which predominantly consisted of fresh (sulphide) material. Additional column testing of material from the Twin Hills Pit of both clay type and rocky type mixed ores indicated recoveries ranging between 49% and 75% for ore crushed to a P_{80} of 3 inch.

Both the oxide and mixed ore recoveries are confirmed by historical production records, which show that between 2009 and 2013 approximately 6 Mt of mixed ore from the Twin Hills deposit was treated with a resultant recovery ranging from 73% to 82% for the oxide ore, and from 40% to 62% for the mixed ore.

Based on the bottle roll and column tests on the mixed zone at Twin Hills, and historical production records, a gold recovery of 57% and 76% for mixed ore and oxide ore respectively has been used for Mineral Reserve and Mineral Resource estimation and mine economics.

Although the test results indicated gold recoveries higher than 76%, at this stage, for the purposes of the Mineral Reserve estimate, Aura considers the 76% factor appropriate for the oxide zone.

The gold recovery based on production estimates for 2001 through 2013, is shown in Table 1-3.

Table 1-3: Gold Recovery Production[1]

Period	Ounces to Pad	Ounces Recovered	% Recovery
2001	128,645	105,998	82.4

2002	117,015	99,064	84.6
2003	58,800	50,795	86.4
2004	83,877	65,032	77.5
2005	78,231	61,236	78.7
2006	83,625	70,779	84.6
2007	49,068	51,240	104.4
2008	66,988	47,761	71.3
2009(2)	98,843	68,372	68.5
2010	110,518	70,641	63.9
2011	94,140	60,871	64.7
2012	86,292	59,751	69.2
2013	103,085	63,811(3)	61.9(3)

Note: Prior to February 2006, production was by RNC Gold Inc.
1. – From internal production data sheets
2. – Between 2009 and 2013, 6 Mt Ore from Mixed Zone Stacked and Leached.
3. – Due to labour strikes, most of the gold leached in December was not refined (effectively recovered in 2013).

A portion of the Mineral Reserves, located between, and adjacent to, the East Ledge and Twin Hills deposits, has not yet been tested. However, the geological setting and the style of mineralization are similar and the authors believe the recovery factor is consistent with what has been found to date.

As part of on-going leaching tests on the mixed zone, Aura has started the hot soluble cyanide gold assay procedure for both production blast hole assays and plant metallurgical control. This assay technique provides an excellent guide as to the degree of oxidation of the gold mineralization and its potential recovery.

1.8 Mineral Resources and Mineral Reserves

The Mineral Resources for the San Andrés deposit are estimated using ordinary kriging within 11 mineralisation domains defined by detailed geological modelling and reported by oxide, mixed, and sulphide boundaries. The Mineral Resources are also constrained by a 200 m exclusion zone along the Agua Caliente River. The block model used blocks measuring 10 m x 10 m x 6 m. The drillhole data was composited to 1.5 m and 6 m intervals depending on domain. The estimation search strategy was oriented to align with the variograms and 2 estimation runs applied within an octant search. Variable minimum and maximum values were set depending on composite lengths. The block model was then updated using the December 31, 2012 topography to account for previously mined material.

The estimation and classification of the Mineral Resources have been prepared in accordance with both Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") Best Practice Guidelines and NI 43-101 Technical Reporting standards. The classification of the Mineral Resources is based on two considerations, the search radius influence and a resource limit based on an optimized pit using a US$2,000/oz gold price.

The December 31, 2013 Mineral Resources estimated by Aura total 104.8 Mt of Measured and Indicated Mineral Resources at an average grade of approximately 0.49 g/t gold grade and Inferred Mineral Resource of 4.3 Mt at an average grade of 0.49 g/t gold grade, using a long term US$1,600 gold price and a 0.23 g/t Au cut-off for oxide and a 0.30 g/t cut-off for mixed material. The Mineral Resources pit shell optimization did not consider any sulphide material. Note that the Mineral Resources are inclusive of Mineral Reserves. Also note that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-4 sets out the estimated Mineral Resources for the Mine as of December 31, 2013.

Table 1-4: December 31, 2013 Mineral Resource Estimate*

Resources Category	Oxide			Mixed			Total		
	Tonne (t)'000	Au (g/t)	Oz' 000	Tonne (t)'000	Au (g/t)	Oz' 000	Tonne (t)'000	Au (g/t)	Oz '000
Measured	13,424	0.46	199	2,814	0.59	54	16,238	0.48	252
Indicated	63,201	0.47	945	25,402	0.57	462	88,603	0.49	1,407
Measured + Indicated	76,625	0.47	1,144	28,216	0.57	516	104,841	0.49	1,660
Inferred	3,319	0.42	45	1,029	0.74	24	4,348	0.49	69

Note*:

1. The Mineral Resources estimate is based on optimized shell using $1,600/oz gold.
2. The cut-off grade used was 0.23 g/t for oxide material and 0.30 g/t for mixed material.
3. Contained metal figures may not add due to rounding.
4. Surface topography as of December 31, 2013, and a 200m river offset restrictions have been imposed.
5. Mineral Resources are inclusive of Mineral Reserves.
6. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues.

The estimate of Mineral Reserves is based on a long range mine plan and plant production schedule developed by Aura. The economic criteria using the Lerches-Grossman algorithm for pit limit evaluations, including process recoveries and operating costs are provided in Table 1-5.

Table 1-5: Economic and Geometric Criteria

Pit and Cost Parameters	
Bench height (m)	6
Road width (m)	18
Overall Pit Slope (°)	Varies: 41 - 49
Bench face angle (°)	Varies : 65 – 70
Minimum pit bottom (m)	20
Berm width (m)	3.8
Ramp slope (%)	10
Gold Price (US$/oz)	1,300
Oxide recovery (%)	76
Mixed recovery (%)	57
Mining cost (US$/t)	2.41
Processing cost (US$/t)*	6.49
G & A cost (US$/t)	1.75

Note*– Includes maintenance costs

The December 31, 2013 Mineral Reserves estimated by Aura total 68.1 Mt of Proven and Probable Mineral Reserves at an average grade of 0.52 g/t Au. Table 1-6 summarizes the Proven and Probable Mineral Reserves for the Mine estimated using a long term cut-off grade using a $1,300/oz gold price of 0.28 g/t Au for oxide material and a cut-off grade of 0.37 g/t Au used for the mixed material as of December 31, 2013.

Table 1-6: December 31, 2013 Mineral Reserves Estimate*

Mineral Reserve Category	Oxide			Mixed			Total Material		
	Tonne (t)'000	Au (g/t)	Oz' 000	Tonne (t)'000	Au (g/t)	Oz' 000	Tonne (t)'000	Au (g/t)	Oz '000

Proven	12,369	0.48	190	2,346	0.63	47	14,714	0.50	237
Probable	43,838	0.50	702	9,549	0.62	190	53,388	0.52	892
Proven + Probable	56,207	0.49	892	11,895	0.62	238	68,102	0.52	1,129

Note*:

1. The Mineral Reserves estimate is based on an optimized pit, which has been made operational, using $1,300/oz gold.
2. The cut-off grade used was 0.28 g/t for oxide material and 0.37 g/t for mixed material.
3. Contained metal figures may not add due to rounding.
4. Surface topography as of December 31, 2013.

The Authors note that the Mineral Reserves are estimated in accordance with the CIM definitions and are considered to be NI 43-101 compliant. The reported Mineral Reserve estimate is reasonable for the remaining LOM Plan.

The Proven and Probable Mineral Reserves at the Mine contain approximately 1,129,400 oz of gold in 68 Mt of ore, sufficient for ten years of mine life at a calculated average production rate of approximately 7 Mt of ore per year. The Mine hosts a large Mineral Resource, and has had a good history of conversion of Mineral Resources into Mineral Reserves; as such there is a reasonable expectation that conversion of existing Mineral Resources into Mineral Reserves will extend the mine life beyond the current 10 years.

1.9 Mining and Processing

Mining at San Andrés is by conventional open pit methods. Historical production rates for the years 2009 to 2013 averaged approximately 13,000 t of ore and 10,000 t of waste produced daily with generally continuous mining 24 hours a day for 360 days per year. Operating phases (push-backs) have been designed to support the Mine production from initial topography of December 31, 2012.

The San Andrés Mine is anticipating a material expansion in ore throughput from approximately 5Mtpa to 7Mtpa. This expansion was justified by the improved incremental economics with modest capital investment.

Mine production utilizes conventional drill and blasting methods with excavation on 6m high benches. Blasted material is then loaded via shovels and excavators onto haul trucks and is hauled to one of two jaw crushers utilizing a contract haul fleet. All of the ore is processed through a two stage crushing circuit and transported on conveyors before being stacked as the final product sized at 80% passing 2.5 inches. The crushing and conveying circuit is designed for a nominal capacity of 1,100 t/h, which is adequate for the expanded production rate if operating at approximately 74% overall utilisation rate. For the expansion, most of the capital investment is applied to improve the secondary screening and crushing plant in order to consistently achieve or exceed 74% utilisation factor.

After the ore has been crushed it is treated with 2.5 to 4.0 kg/t of cement and 1.5 to 3.5 kg/t of lime before reaching the agglomerators where the ore is retained and mixed while adding an intermediate process solution to achieve the optimum moisture of 18%. The process solution contains up to 400 ppm cyanide solution.

The Mine production schedule was generated based on the December 31, 2013 Mineral Reserves within the designed pit phases and has considered restrictions of the planned waste dumps, previously mined areas and the cemetery. The detailed 2014 mine schedule is summarized by year in Table 1-7.

Table 1-7: Life of Mine Schedule

Year	Oxide Ore	Mixed Ore	Total Ore	Waste

	Tonne (t)'000	Oxide Gold Grade (g/t)	Gold Oz' 000	Tonne (t)'000	Mixed Gold Grade (g/t)	Oz' 000	Tonne (t)'000	Total Gold Grade (g/t)	Gold Oz '000	Tonne (t) '000	W/O
2014	6,333	0.47	97	0	0.00	0	6,333	0.47	97	3,986	0.63
2015	7,046	0.47	106	7	0.47	0	7,053	0.47	106	3,313	0.47
2016	6,582	0.49	103	449	0.59	9	7,030	0.49	112	5,363	0.76
2017	6,107	0.52	102	914	0.61	18	7,021	0.53	119	5,783	0.82
2018	6,839	0.47	102	263	0.62	5	7,102	0.47	108	6,741	0.95
2019	5,415	0.51	89	1,498	0.74	35	6,913	0.56	124	5,115	0.74
2020	4,207	0.51	69	2,894	0.56	52	7,101	0.53	121	3,075	0.43
2021	2,650	0.48	41	4,398	0.63	89	7,048	0.57	130	4,912	0.70
2022	6,080	0.49	95	707	0.61	14	6,786	0.50	109	5,739	0.85
2023	4,949	0.55	88	765	0.62	15	5,714	0.56	103	4,678	0.82
Total	**56,207**	**0.49**	**892**	**11,895**	**0.62**	**238**	**68,102**	**0.52**	**1,129**	**48,705**	**0.72**

The ore is stacked on the leach pad in 8 m lifts on previously leached ore that has been ripped and prepared. The ore is leached for an average of 120 days before the area is allowed to dry and prepared for the next lift. The solution used for leaching comes from the ADR plant after the cyanide concentration has been replenished.

The Mine leach pad facility is a monolithic leach pad that has been constructed in multiple phases. The first four phases of the leach pad facility were designed by the consulting firm SRK Inc., Denver, USA ("SRK").

Production rates from the current mining operation show that Phases III & IV of the existing heap leach pad would reach full capacity by the first quarter of 2015 without additional pad space. A new leach pad facility (Phase V), designed to be hydraulically independent from the existing Phase I-IV facility, was designed by the consulting firm AMEC, Denver, USA. The Phase V facility is being constructed in stages, with the first stage completed in 2013 and the final stage to be completed during the second quarter of 2015.

The Phase V heap leach pad expansion consists of a pad with a 32 hectare footprint, which partially overlaps with existing Phases II, III, and IV located immediately south of Phase V. Phase V heap leach pad provides for approximately 12 million m^3 of ore storage, or 19 million tonnes of ore capacity. The Phase V heap leach pad is considered a first stage of the potential further heap leach facility expansion. Further heap leach expansion may be constructed above or adjacent to the existing heap leach pads in the future.

Gold is recovered through the ADR plant, which has 12 carbon columns that can be configured in a two or three train configuration with a nominal capacity of 500 m^3/h per train. The assay lab which processes both Mine grade control samples and process plant samples is located in the same complex as the ADR plant. The gold produced at the ADR plant is analyzed prior to shipment for refining and sale. The ADR plant is being upgraded to couple with expanded capacity. Upgrades include improvements to the carbon handling and elution circuits and the addition of a number of cathodes and anodes to the existing electrowinning cells in the refining portion of the plant.

1.10 Environmental Considerations

An environmental management plan was formulated at the request of the government of Honduras and addresses the commitments made within the five EIA's; Water Tank Hill, Expansion Water Tank Hill (East Ledge), Twin Hills Phase II and IV, and Expansion Twin Hills; the Mitigation Contracts and recommendations issued by government agencies.

The plan defines and describes all references to the term "Best Management Practices" used in the EIA's. Overall, the plan allows for the orderly definition of commitments made to the Honduran government and to the Company's stakeholders for the protection of the environment and for mitigation of the potential environmental impacts caused by the construction and operation of the Project.

The management plan includes:

- Compliance with the International Cyanide Management Code, San Andrés is a certified operator;
- Environmental Monitoring Plan updated each year to adapt to new sampling requirements;
- Contingency Plan was updated and reviewed in 2012. This Plan has been discussed with key personnel in the operation to ensure procedures described are appropriate according to any given situation;
- Materials Management Plan, consisting of management of hazardous and nonhazardous materials, construction and management of facilities (i.e., land fill and ancillary facilities), education regarding good housekeeping, and organization of waste recollection and disposal;
- Spilled Soil Management and Remediation Plan, updated in 2004, that includes the development of treatment sites and technologies to decontaminate polluted soils (i.e., bioremediation of oil polluted soils in concrete tanks). Minosa possesses a THC analysis kit to verify THC concentration.
- Erosion Control Plan is updated every year to address yearly priorities;
- Explosives Management Plan, designed to comply with the Honduran and U.S. explosives management regulations.;
- Surface and Underground Water Management Plan, updated in 2004;
- Mine Waste Management Plan, updated yearly; main focus to use greater proportion of waste rock as material for contouring former mining areas;
- Wastewater Treatment and Management Plan, updated yearly depending on the quality of the water to be treated and/or managed.
- Health and Safety Plan, updated yearly under the commission of the Safety and Occupational Health Department. This plan consists of six main components; Occupational Clinic, program to assess the working environment, definition of required personal protection equipment, safety training program, mix health and safety Commission, health and safety surveillance.
- Reforestation Plan, updated in 2009 (the original plan was approved by COHDEFOR), the 2009 plan is pending approval by Forestry Conservation Institute ("ICF") and its implementation is the responsibility of a forestry engineer.
- A Conceptual Reclamation and Closure Plan is in place together with the International Financial Reporting Standards calculations.
- Plan of Sewage and Potable Water Management implemented in 2002.
- Plan to encapsulate AMD (Acid Mine Drainage) potential with inert waste implemented in 2004 and reviewed periodically.

The communities within the direct area of Mine influence have had a number of minor protests against Minosa and the Mine during late 2013 and early 2014. The protests have been settled through active engagement but have resulted in production stoppages, and or have prevented the delivery of goods and equipment, but have not negatively impacted the Mine's forecasted production.

1.11 Economic Considerations

The principal commodities mined at the Mine are freely traded, at prices that are widely published, so the sale of any production is not a material concern to Aura.

A post-tax cash flow model has been developed by Aura from the LOM production schedule, capital and operating cost estimates, and NSR's using $1,300/oz gold price. A review by Aura of the cash flow projections has found the after tax cash flow is positive, supporting the Mineral Reserve designation.

The sensitivity analysis has been completed that examined gold price, capital and operating costs ranging from +10 to -10%. The sensitivity analysis has been reviewed by Aura and it is concluded that when the gold price is reduced by 10%, or operating costs increase by 10%, or the capital costs increase by 10% the net present value remains positive.

1.12 Conclusions and Recommendations

Aura has prepared a Report compliant with NI 43-101 on the updated Mineral Resources and Mineral Reserves pertaining to its San Andrés Mine, located in the municipality of La Unión, in the Department of Copán, Honduras. The Project's mineral rights are owned by Minosa, a wholly-owned indirect subsidiary of Aura. The update became necessary due to the additional Mineral Resources and Mineral Reserves in connection with the Mine expansion plan, prepared by Aura.

The reported Mineral Reserve estimate is reasonable for the remaining LOM Plan.

The Authors recommend the following:

- A metallurgical study on the Zona Buffa Mineral Resources to determine leach recovery for inclusion of these resources into reserves. The approximate cost of this study is $5,000;
- As mining progresses, continued reconciliation needs to be reviewed and if parameters change, an update of the Mine plan should be developed;
- Operating costs should be reviewed on a regular basis to ensure operating cut-offs remain valid;
- The recovery rate for oxide, mixed and blends containing these types of ore should continue to be monitored and compared to equivalent column tests. It is also recommended that the on-going program of column tests (performed at site) is expanded for investigations of future production in accordance to the new Mine plan;
- Additional specific gravity measurements should be conducted on mixed zone material to determine an appropriate specific gravity that can be incorporated into the block model. This is estimated to cost $25,000; and
- That the operation continues with the QA/QC programme on the exploration and the production blast hole sampling to ensure that a comprehensive data set is obtained for future estimates, which yearly is estimated to be $15,000.
- Exploration of the Aguas Calientes and Banana Ridge areas, where there are a number of high grade intercepts is likely to see significant expansion to the resources and reserves.

2.0 INTRODUCTION

2.1 The Issuer

Aura Minerals Inc. ("Aura" or the "Company") is a Canadian-based company, located in Toronto, Ontario, that is focused on the development and operation of gold and base metal projects in the Americas. Aura's portfolio includes four producing mines: the San Andrés gold mine located in La Unión (the "Mine" or the "Project"), Honduras, the São Francisco and São Vicente gold mines, located in northwestern Brazil and the copper-gold-silver Aranzazú mine, located in Mexico. The Company's common shares trade on the TSX under the symbol "ORA".

Aura's portfolio also includes the copper-gold Serrote de Laje project located in Alagoas State, Brazil.

2.2 Terms of Reference

This technical report (the "Report") is compliant with National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") and includes updated Mineral Resources and Mineral Reserves for the Mine as announced by the Company on May 27, 2014. The Report was prepared by internal Company experts in close collaboration with San Andrés Senior Management and technical personnel to update the Mineral Reserves and Mineral Resources with current information.

The analysis and procedures for this Report was performed over the period February 2013 to September 2013 and incorporated Qualified Person expertise from Aura. Mine site personnel and management were closely consulted during the course of the study.

The names and details of the Qualified Persons and those who assisted the Qualified Persons, who have prepared or contributed this Report, are listed in Table 2-1.

Table 2-1: Qualified Persons

Qualified Person	Position	Employer	Date of Last Site Visit	Professional Designation	Sections of Report
Mr. B L Butcher	VP Technical Services	Aura Minerals Inc.	Jan 20-23,2014	P. Eng	1-5,15,16,19-22,25,26,27,28
Mr. B Bartlett	Manager, Mineral Resources	Aura Minerals Inc.	July 7-18, 2013	FAusimm	6-11,12,14,23,24
Mr. P Rosario	Former Principal Project Metallurgist	Aura Minerals Inc. (formerly)	Dec 9-13, 2013	P. Eng	13,17,18
Other Experts who Assisted the Qualified Persons					
Expert	**Position**	**Employer**		**Date of Last Site Visit**	**Sections of Report**
Mr. G Rios	VP Corporate Responsibility	Aura Minerals Inc.		June-18-21, 2014	4,16, 20
Mr. M Reed	General Manager	Minosa		Stationed at site	15,16,19,21,22
Mr. M Vergara	Superintendant of Technical Services	Minosa		Stationed at site	4,15,16
Mr. R Taylor	CFO	Aura Minerals Inc.		Oct 3-4, 2013	19,21,22
Mr. R Goodman	General Counsel & Corporate Secretary	Aura Minerals Inc.		Nov 15-16, 2012	4,23

All currency amounts are stated in US dollars, as specified, with costs and commodity prices. Quantities are generally stated in Système International d'Unités (SI) metric units, the standard Canadian and international practice, including metric tons (tonnes, t) and kilograms (kg) for weight, kilometres (km) or metres (m) for distance, hectares (ha) for area, grams (g) and grams per metric tonne (g/t) for gold (g/t Au). Wherever applicable, imperial units have been converted to metric units for reporting consistency. Precious metal grades may be expressed in parts per million (ppm) or parts per billion (ppb) and their quantities may also be reported in troy ounces (ounces, oz), a common practice in the mining industry. Attached hereto, as Appendix 1, is a glossary of common terms and abbreviations that may be used in this Report.

This Report follows the format and guidelines of Form 43-101F1, Technical Report of NI 43-101.

This Report includes technical information which requires subsequent calculations or estimates to derive sub-totals, totals and weighted averages. Such calculations or estimations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, Aura does not consider them to be material. Further, the Report summarizes the professional opinion of the Authors and includes conclusions and estimates that have been based on professional judgment and reasonable care.

The conclusions and estimates in this Report are consistent with the level of detail of this Report and based on the information available at the time this Report was completed. All conclusions and estimates presented are based on the assumptions and conditions outlined in this Report.

This Report is to be read in its entirety.

2.3 Sources of Information

The key sources of information used in this report include:

- The Company's NI 43-101 Technical Report entitled *Resource and Reserve Estimates on the San Andrés Mine in the Municipality of La Unión, Department of Copán, Honduras* dated March 28, 2012 (effective date of December 31, 2011) (the "2012 Technical Report").
- Monthly operating statistics and internal data and the 2014 Resource model.
- A full list of references is included in Section 27.

3.0 RELIANCE ON OTHER EXPERTS

In respect of acquisition agreements, status of mineral titles, concessions, taxes, NSR, ownership matters, etc as set out in Section 4 and 23 of this Report, the Authors have relied on information provided internally by Aura’s General Counsel with assistance from local counsel in Honduras.

In respect of the environmental section (Section 20) of this Report and specifically matters regarding environmental studies, permitting and social or community impact, the Authors have relied on Mr. G Rios, VP Corporate Social Responsibility for Aura.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Mine is located in the Department of Copán in the interior highlands of western Honduras approximately 210 km southwest of the city of San Pedro Sula within the Municipality of La Unión (Figure 4-1). The property is centered on latitude 14.76° North (UTM 1,632,640 m North) and longitude 88.94° West (UTM 291,085 m East).

Figure 4-1: Location Map

Source: United Nations- Map No. 3856 Rev. 3, May 2004

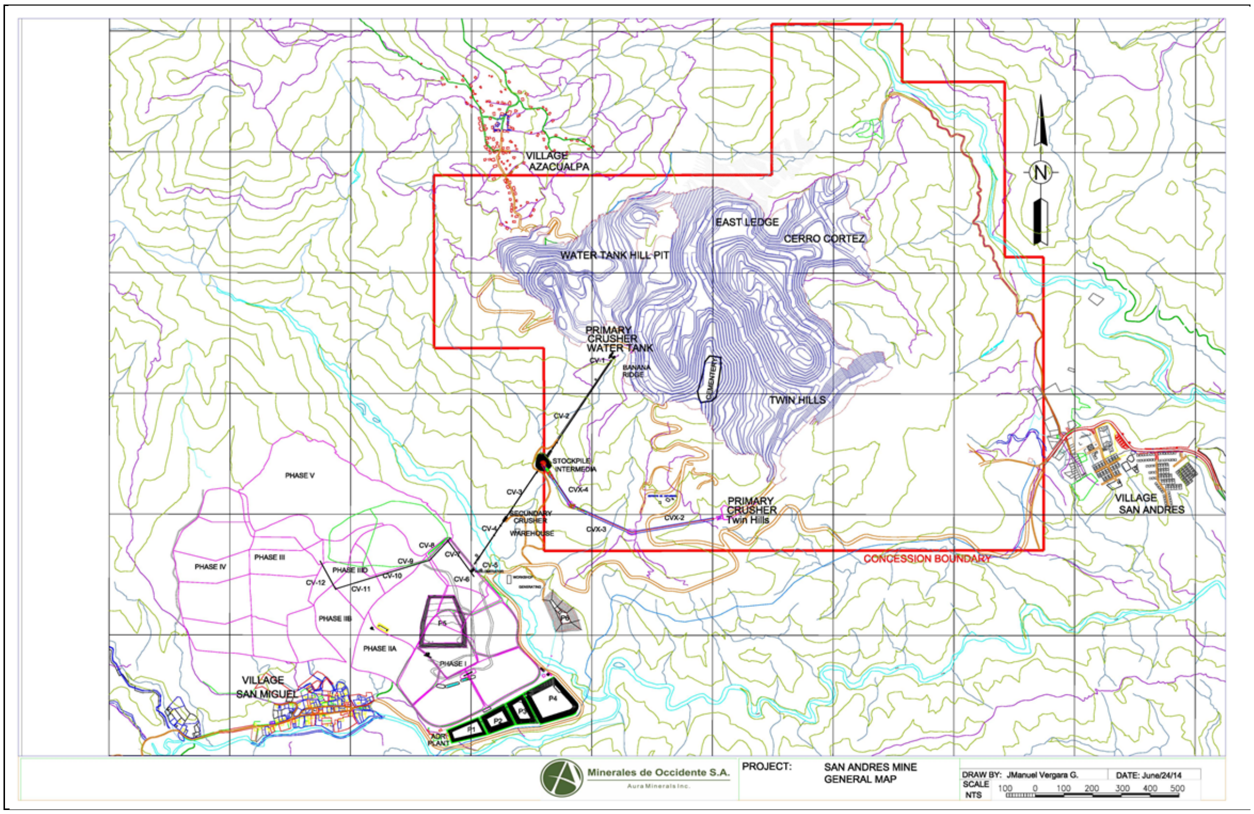
4.1 Location of Mine Workings, Mine Facilities and Cultural Features Relative to Property Boundaries, including Expansion Plan

Figure 4-2 shows the location of the Mine workings, Mine facilities, local communities, and cultural features relative to the property boundaries.

The first open pit mining at the Mine was carried out in the Water Tank Hill deposit. This pit was depleted by early 2003 and has since been reclaimed. Mining commenced in the East Ledge pit in early 2003 however it is currently shut down and is anticipated to re-start during the expansion plan implementation. The present production comes from the Twin Hills open pit and Cerro Cortez located just east of the East Ledge pit. Both the Twin Hills and Cerro Cortez open pit waste is currently transported and used to fill and reclaim the lower Twin Hills pit area.

The communities of San Andrés, San Miguel, Platanares and Azacualpa, located within and proximal to the mining concession, constitute individual "aldeas", or communities, and form part of a larger tract of land called "ejido", or public land. There is also a cemetery used by the communities located above a portion of the Mineral Resources and adjacent to the proposed pit expansion. An agreement with the communities in late 2012 allowed for relocation of this cemetery to accommodate expansion of the pit to recover the Mineral Resources located in this area. Agreements have been signed and are in the process of implementation to relocate the cemetery by the end of Q2 2015.

Figure 4-2: Site Location



4.2 Mining and Exploration Concessions

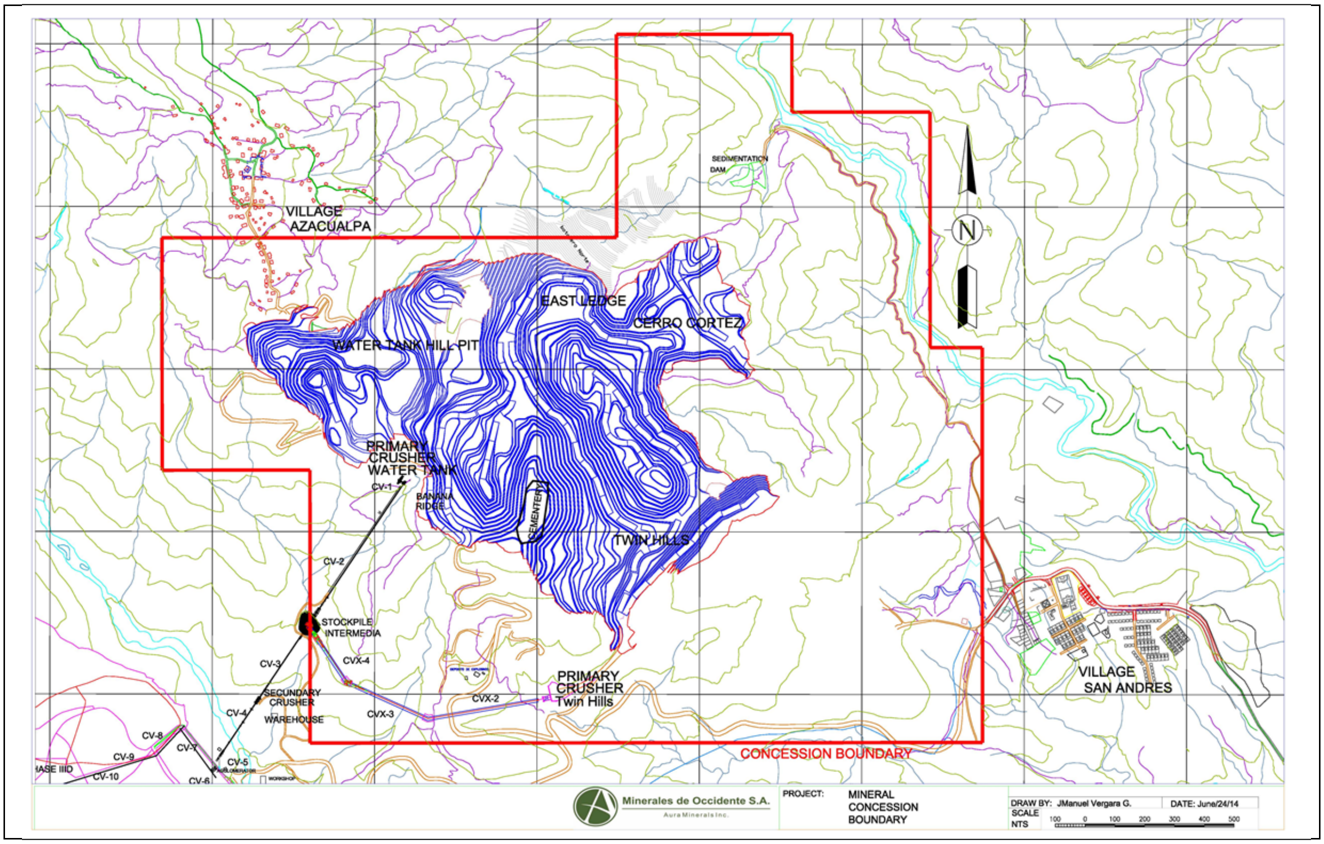
The exploitation and exploration concessions and all facilities of the mine were acquired by Aura in August 2009 from Yamana Gold Inc. The properties and assets are held by Aura's beneficially-owned subsidiary Minerales de Occidente, S.A. de C.V. ("Minosa"), which is a company existing under the laws of Honduras.

Minosa holds mineral tenure to the Mine pursuant to gold and silver exploitation concession called "San Andrés I" covering a total of 399.09 ha, or about 4.0 km^2. The San Andrés concession was first granted in an initial area of 100 ha to Compañía Minerales de Copán, S.A. de C.V. ("Minerales de Copán") on January 27, 1983. After an expansion of what was first reported as an additional 200 ha on November 18, 1992, a final addendum to the original claim with Greenstone Resources Ltd. ("Greenstone"), the original owner of the property at the time, was made on July 18, 1997, confirming the true area of the concession to be 399.09 ha. All mining claims are granted for a period of 40 years with a right to extend for an additional 20 years. This concession contains the Mineral resource and Mineral reserve estimates discussed in this Report.

In May 2002, Minosa filed applications for four additional mining concessions totaling 3,768 ha: San Andrés II (900 ha), San Andrés III (869 ha), San Andrés IV (999 ha), and San Andrés V (1,000 ha), however, the concessions have not yet been granted pending the final resolution of the Honduran mining law, but Minosa will have the first rights to the concessions.

The above concessions are contiguous and surround the existing San Andrés mining concession. The San Andrés mining concession that fully encompasses the Mineral Reserves are shown in Figure 4-3.

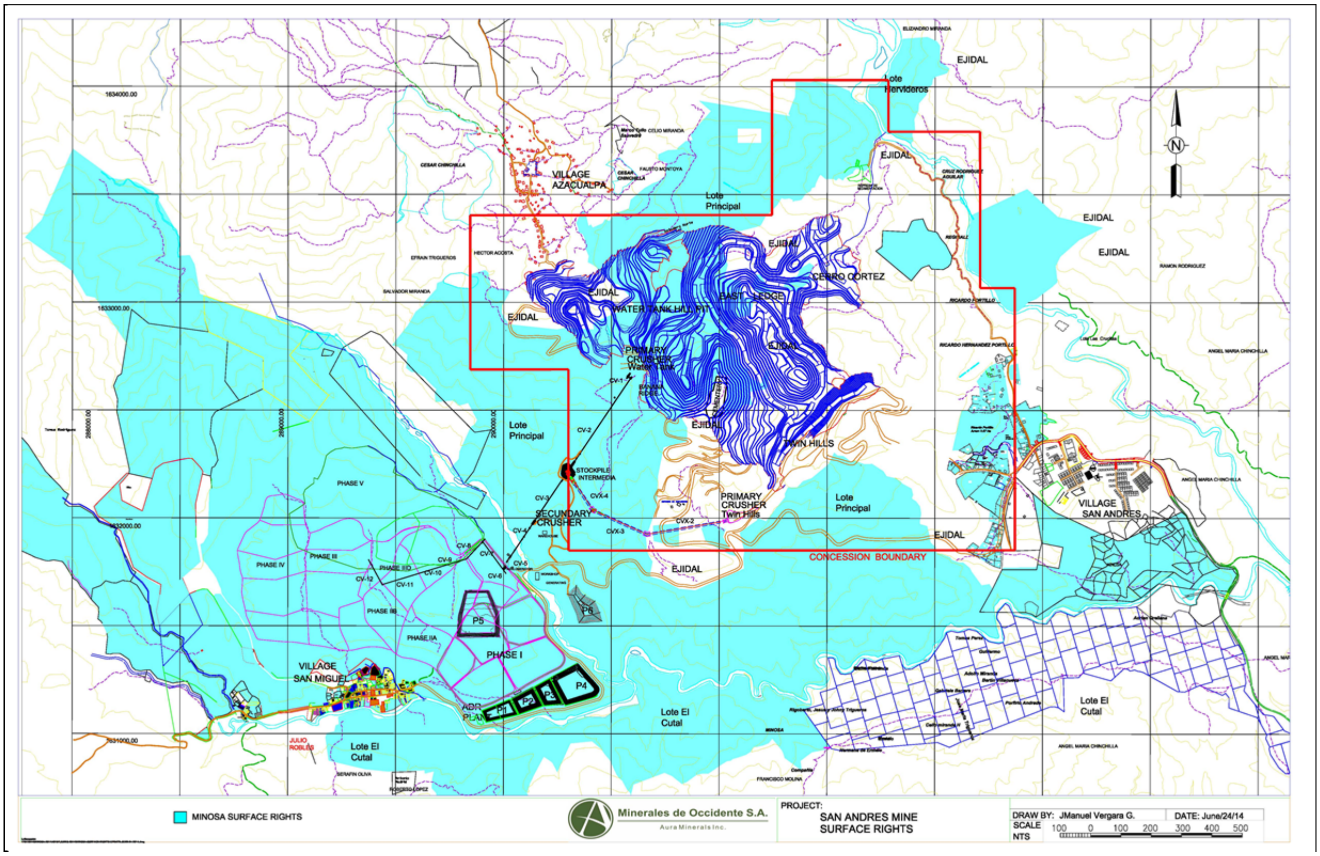
Figure 4-3: Concession Map



4.3 Surface Rights

The mining areas and primary crusher facilities are within the 399.09 ha mining concession. Other process facilities are located outside the concession, including the secondary crusher, agglomerators, leach pad, and CIC-ADR plant. Forty per cent of the surface rights on the mining concession are held by Minosa. A portion of the concession is government land whereby Minosa is provided access rights under the provisions of the mining concessions. Minosa privately owns the surface rights for the portion of the land occupied by the process facilities that are located outside the mining concession. The surface rights for the area required for the expansion of the leach pads to process the new Mineral Reserves have been purchased. Negotiations with the local communities are in progress to purchase the additional surface rights required for expansion of the open pit. Minosa's surface rights are shown in Figure 4-4.

Figure 4-4: Surface Rights



4.4 Permits

Mining concessions have been received for mining the East Ledge pit, the Twin Hills pit and the Cerro Cortez pit.

- East Ledge, Twin Hills, and Cerro Cortez sites/exploitation regions are all included within the San Andrés Mining Concessions, comprised of one single file, which is in force up to January 27, 2023.
- The Environmental License for East Ledge has expired and is currently inactive.
- The Environmental License for Twin Hills already includes the Cerro Cortez site, and is currently under renewal process.

Environmental policies are governed by the General Law of the Environment (‘Ley General del Ambiente’). The law came into effect 20 days after its publication on June 30, 1993 with the promulgation of Decree 104-93 by the National Congress in the capital city of Tegucigalpa.

Regulations for the General Law of the Environment are set forth in the ‘Reglamento de la Ley General del Ambiente’ (Executive Agreement 109-93, Tegucigalpa, December 20, 1993) and the ‘Reglamento del Sistema Nacional de Evaluación de Impacto Ambiental (SINEIA) General” (Executive Agreement 189 - 2009, Tegucigalpa, September 7, 2009).

Secretary of State for Energy, Natural Resources and the Environment, or Secretaría de Estado en los Despachos de Energía, Recursos Naturales y Ambiente ("SERNA") is the ministry responsible for Honduras' energy, environmental and natural resources policies. SERNA has several departments: Dirección General de Control Ambiental ("DECA"), Centro de Estudios y Control de Contaminantes ("CESCCO"), Dirección General de Biodiversidad ("DIBIO"), Dirección General de Recursos Hídricos ("DGRH"), Unidad de Servicios Legales ("USL") and others.

Pursuant to the Mining General Law (Decree 238-2012) and its Regulations ("Reglamento de la Ley General de Minería" approved through Executive Agreement 042-2013, Tegucigalpa, August 02, 2013) the mining authority is the Instituto Hondureño de Geología y Minas ("INHGEOMIN") depending on the Secretaría de Estado en el Despacho Presidencial (Ministry of the President).

INHGEOMIN regulates matters such as the technical aspects of mining, geology, environmental management and metallurgy. It also assigns metallic and non-metallic mineral concessions and administers the Mining Law in Honduras, under the direction of SERNA (depending on the Ministry of the President), but as a decentralized dependency.

DECA regulates technical matters of the environment. Honduras Law requires an environmental license for new projects or expansions and modifications of existing projects. Depending on the environmental impact that will be generated by the proposed project and its categorization according to the "SINEIA", the license is issued based on the approval of an Environmental Impact Assessment ("EIA"). The requirements for an EIA are governed by the Reglamento del Sistema Nacional de Evaluación de Impacto Ambiental ("SINEIA"), and the "Manual Técnico del Sistema Nacional de Evaluación de Impacto Ambiental" (1994). DECA reviews and approves EIAs submitted by mining companies

Municipal operating permits are also required for all activities in Honduras. Municipal laws and guidelines are described in the Ley de Municipalidades (Decree 134-90, Tegucigalpa, October 29, 1990) and its Reglamento General de la Ley de Municipalidades (Executive Agreement 018-93, Tegucigalpa, February 18, 1993).

The rules and regulations governing the granting of exploration and exploitation concessions and mineral rights are described in the Mining General Law approved by the National Congress on February 28, 2013 and its Regulations approved on August 2, 2013.

Applications for exploration concessions are submitted to, and approved by, the INHGEOMIN. Applications for Environmental Licenses for mining companies require the completion and submittal of an EIA.

As of the date of this Report, the permitting process is currently delayed as a result of proposed changes to the Mining General Law currently being discussed by the Honduran government. The primary changes being considered are:

- A proposed Increase in taxes from 1% to 2% to a social fund for affected municipalities, which is currently being paid voluntarily already by Minosa at 1%; and
- An additional increase of 1% of gross income to fund the mining authority (INHGEOMIN).

The status of the mining permits is outlined in Table 4-1.

Table 4-1: Status of Mining Permit and Environmental Licenses

Permitted Area	Date Approved	Comments
Water Tank Hill	1998	Initial Pit Completed
East Ledge	2002	Initial Pit Completed
Twin Hills – Cerro Cortez	2005	Currently mined pits
Pit Expansion	2010	Approved on November 26, 2010

4.5 Forestry Regulations

Instituto de Conservación Forestal, or Honduran Institute for Forest Conservation ("ICF") manages the countries forest resources. ICF is a decentralized organization with its own Executive Director, which is assigned by the President of the Republic of Honduras. The Executive Director has the ranking of a secretary of state and is a member of the cabinet council.

Minosa is allowed to cut all the trees in the areas that are permitted. Minosa has forest management plans over areas defined by their environmental department where surveys and inventories of trees are collected to calculate the volume of wood in each area. Permits from ICF are needed in order to cut any of the trees in these areas. The second step is to upgrade the management plans to operational plans where a specific area of tree cutting is expected to take place, for example, an area covering an open pit. Minosa has plans for the Twin Hills pit area and currently holds a valid operational plan permit over this area.

Anywhere outside this operational plan (currently restricted to the pit area) where trees have to be cut for exploration work, for example, specific permitting is required.

4.6 Environmental Liabilities

There are no un-mitigated historic environmental liabilities. There are a number of remediation measures that will be carried out upon completion of mining and which are included under the approved EIA. An EIA was completed in April 1998, following the completion of baseline studies and a review of Greenstone's 1996 Project Feasibility Study. The EIA was the source document used to create the Environmental Management Plan used for all permitting and compliance monitoring at San Andrés.

The EIA concluded that there are no serious long-term environmental hazards that would exist or be created by the operation as designed, considering that all reclamation and control programs were completed and are currently applied.

Since then, five more EIA reports have been completed. The first was prepared in 2002 (ECADEH), with the objective to license the extension of the Water Tank Hill pit, known as East Ledge. The Environmental License was issued by DECA/SERNA in 2003. A second, additional EIA to the first prepared by the consulting engineering firm SRK Inc. ("SRK"), was issued by Ecología y Servicios S.A. (Ecoservisa, 2004, Environmental Impact Assessment for the Development of Twin Hills and Cerro Cortez pits) resulting in an Environmental License issued by DECA/SERNA in 2004. The third, also issued by Ecología y Servicios S.A., for the expansion of the leach pads Phase III and IV, the environmental license was issued in 2009. The fifth EIA was issued by JATELCOM in 2012 with final approval by the environmental agency October 15, 2012. Table 4-2 summarizes the EIA.

Table 4-2: EIA Summary

Permitted Area	Date presented	Comments
Water Tank Hill, leach pads and ADR plant	1997	Initial Pit Completed
East Ledge	2002	WTH Expansion
Twin Hills – Cerro Cortez	2004	Pits to be Incorporated into new pit
Pit Expansion	2009	Approved on November 26, 2010
Phase III and IV	2006	Renewed until 2019
Phase V	2012	Expires in 2017

In all cases, Minosa and DECA signed a new Mitigation Measures Contract including items not covered in the first permit for very specific actions for the areas to be developed. One new feature of these was the introduction of compensatory measures which consist of the reforestation of municipal or private lands outside the concession area that are similar in size to those of the development projects.

4.7 Royalties Obligations and Encumbrances

The following NSR and taxes have been applied to the economic analysis:

- 1.5% NSR payable to an arm's length third party (part of the original purchase price)
- 2.0% Municipality tax (2013)
- 2.0% Security tax (2011)
- 1.0 % INHGEOMIN tax (2013 - Honduras' mining regulatory agency).

There are no back-in rights, payments, or other agreements and encumbrances to which the property is subject.

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

5.1 Accessibility

Access to the Project is via paved highways and a gravel road with a total driving distance of approximately 210 km from the city of San Pedro Sula or 360 km from the capital city of Tegucigalpa. Both cities are serviced by international airports with daily flights to the continental U.S.A. and to other cities in Latin America. San Pedro Sula is 40 km from Puerto Cortés, the major Honduran seaport on the Caribbean Sea as illustrated in Figure 4-1.

The Project is located approximately 18 km due west of the town of Santa Rosa de Copán, the capital of the Department of Copán. The town site and property of San Andrés is reached via a 28 km paved highway from Santa Rosa de Copán, and then by a 22 km gravel road from the turn-off at the town of Cucuyagua. The gravel road is public, but Minosa volunteers to maintain the road on behalf of the authorities.

5.2 Climate

The climate of San Andrés is temperate, with a distinct rainy season locally called winter from May to November. Monthly rainfall data was collected at San Andrés from 2002 to 2012. Total annual rainfall during this period averaged 1,500 millimetres ("mm"). The driest months are January and February (11.8 and 15.4 mm, respectively). The wettest months are June and September (301 and 284 mm, respectively). Although parts of Honduras lie within the hurricane belt, the western Interior Highlands are generally unaffected by these storms.

Accessibility during some of the wettest months is limited and as a result the available hours for production at the Mine have been reduced by about 6% due to delays.

Temperature decreases with increased elevation and as the Mine site is situated at an elevation of 1,200 m, the climate is quite temperate. Temperatures at San Andrés normally average 18°C in January and 22°C in May. Annual evaporation for the region fluctuates between 825 mm and 1,296 mm. The mean annual relative humidity at Santa Rosa de Copán is 82%. Winds are fairly constant throughout the year. Blowing predominantly from the north, wind speeds average consistent 10 km/h. Average monthly temperatures have been recorded at Santa Rosa de Copán for the period 1953 to the present. Typically, December and January are the coolest months, with average daily temperatures of 17.9°C and 17.8°C, respectively. April and May are typically the warmest months, with average temperatures of about 22°C.

5.3 Local Resources

There are a number of mines operating in Honduras and in adjacent countries in Central America. These mining operations are supplied and serviced by branch offices and facilities of international contractors and suppliers and by domestic contractors and suppliers. Cement and fuel are provided locally by Honduran companies. Spare parts and supplies from major centers in North or South America can be readily delivered to the site within a reasonable time.

Labor is sourced locally from the many communities in the immediate area of the Mine. Educational, medical, recreational, and shopping facilities are established. Management and technical staff are available within Central America or are brought in from North or South America. Aura also maintains a corporate office in Canada of experienced geologists and engineers to provide technical support and oversight for all of its projects, including the Mine and projects.

5.4 Infrastructure

The Mine has been in operation since 1983, and has a well-developed infrastructure which includes power and water supply, warehouses, maintenance facilities, assay lab, and on-site camp facilities for management, staff and contractors. On-site communication includes radio, telephone, internet and satellite television services.

The Mine operates a diesel power generation plant for operational needs. In late 2013, a solicitation for connection to the national power grid was made to the Honduras national power authority, Empresa Nacional de Energía Eléctrica (ENEE). ENEE has since responded that the connection is technically feasible and will be approved, reducing power costs by approximately 40%. The operating mode would be to buy power from the grid and maintain sufficient diesel generation capacity to back up the power grid.

Process water is supplied by rainwater run-off collected in the surge pond and by direct pumping from a pump station in the perennial Río Lara adjacent to the CIC-ADR plant. Flow measurements of feeder streams indicate that the Río Lara flow rate is in excess of 100 m^3/h at the driest time of the year.

Chlorinated potable water for the town of San Andrés and camp facilities is supplied from a nearby 72,000 gal storage tank which is fed via a 4-inch, 17-km metal water line from a source originating upstream from San Andrés along the Río Lara near the village of La Arena. Purified water for drinking and cooking is purchased from local suppliers.

5.5 Physiography

The Mine is located within the western Interior Highlands topographical province, a mountainous area that is covered mainly by short grasses and open pine and oak forests. Moderate relief with relatively steep slopes characterizes the area. The elevation in the area of the Mine ranges between 750 m and 1,300 m in a region of very steep slopes, but moderate topographic relief.

The San Andrés concession is located within the Río Lara catchment basin. The area affected by the operations forms only a small part of this catchment. The streams and creeks that drain the Project area eventually flow into Río Higuito located about 4 km southeast of the San Andrés area.

The principal drainages in the Project area include: Quebrada de San Miguel, which lies to the north of the Project and flows southeast and east into the Quebrada del Agua Caliente; Quebrada del Agua Caliente, situated along the eastern boundary of the concession area and draining southeast into the Río Lara; Río Lara, situated immediately south of the new processing facilities and flowing east into the Río Higuito; and Quebrada de Casas Viejas, which drains the southwest portion of the Project area and also flows southeast into the Río Lara. All streams flow year round.

6.0 HISTORY

6.1 Previous Owners

The San Andrés area is reported to be the first Spanish gold discovery in Honduras with initial production in the early 1500's. Until the 19th century, numerous attempts were made to develop small gold deposits hosted by the quartz veins at San Andrés. Numerous adits, drifts, shafts, declines, and prospect pits can still be seen. The streams in the San Andrés area were also extensively mined for placer gold by the local population (Malouf, 1985).

The San Andrés property was explored in the 1930s and 1940s by numerous companies including Gold Mines of America and the New York and Honduras based Rosario Mining Company ("Rosario"). Gold Mines of America and Rosario developed more than 3,140 m of underground drifts and cross-cuts in a zone of epithermal quartz veins and quartz stockwork. Gold Mines of America operated a small amalgamation plant in 1936 and 1937 to process the production.

In 1945, the property was acquired by the San Andrés Mining Company and then purchased by the New Idria Company ("New Idria") (Malouf, 1985). A 200-short tons per day cyanide circuit was installed in 1948, with all equipment air lifted to an airstrip located at Platanares. Approximately 300,000 short tons of surface and 100,000 short tons of underground ore averaging 5.8 g/t Au were mined and milled by New Idria.

In 1949, San Andrés became the first operation to use a carbon-in-pulp (CIP) plant to recover gold and silver by adsorption using granular carbon (Marsden and House 2006); however, numerous problems including poor air travel support logistics and high underground mining costs caused the operation to close in 1954.

In 1969, the Honduran Government, in association with the United Nations, restricted exploration in 10,800 km^2 of western Honduras for study purposes. This area included the San Andrés property. The area remained inactive until it was reopened in 1974 (Malouf, 1985).

In 1974, an exploration permit was granted to Minerales, S.A. de C.V. (MINOSA), a Noranda Inc. subsidiary. MINOSA then joint-ventured the property with Rosario and exploration efforts consisted of soil sampling, mapping and trenching with the purpose of identifying a large, disseminated, open pit gold deposit. Results of this activity indicated a potential for about a 20 million short ton resource grading 2.83 g/t Au (Malouf, 1985).

Changes in the Honduran tax law forced MINSA to drop the concession in 1976. Compañía Minerales de Copán, S.A. de C.V. (Minerales de Copán) acquired the property in January 1983 following changes in the Honduran tax laws. A 60 -short tons per day heap leach operation was installed and 170 local residents were employed on a basic, shovel-and-wheel-barrow operation.

As part of a Honduran grassroots exploration program, Fischer-Watt Gold Company Inc. (Fischer-Watt) became interested in the property following a reconnaissance trip to the nearby Platanares geothermal area. In 1993, Fischer-Watt acquired an option from Minerales de Copán to further explore the property. Fischer-Watt conducted additional mapping and sampling programs with encouraging results.

In 1994, Greenstone acquired the option from Fischer-Watt. The option was exercised in 1996 and Greenstone subsequently acquired in excess of 99% of Minerales de Copán. Feasibility studies began in 1996, and in 1997 Greenstone completed a Feasibility Study that evaluated mining the Water Tank Hill deposit. The project involved an expansion of the existing open pit mine within the concession area and the development of new heap leach processing facilities on adjacent land to the south. Other infrastructure components included a conveyor system to transport the ore to the processing facilities, a waste rock disposal area, a spoils pile, access/haul roads, and an on-site landfill site. The project required the relocation of the village of San Andrés. Proposed production was 2.1 Mtpa and mine life estimated at seven years. The facilities were constructed to handle in excess of 3.5 Mtpa of ore and waste.

Operations associated with the Mine were suspended in May 1997, pending the activities associated with the proposed project. To meet the mining construction schedule, access road improvements and construction of the leach pad, spoils pile and a new village began in July 1997 under the approval of Secretaría de Estado en el Despacho del Ambiente (SEDA).

Following review and approval of the EIA for the Project, Greenstone Minera de Honduras, S.A. de C.V., Greenstone's wholly owned Honduran subsidiary company, received the mining permit on December 9, 1998 and began mining in early 1999. Their first shipment of gold was on March 30, 1999. The mine employed about 344 people including 10 expatriates. Residents of the communities of San Andrés, Azacualpa, San Miguel, and other communities in the area provided most of the workforce.

In 1999, Greenstone projected resources (Measured, Indicated, and Inferred) of about 1.97 million oz (61.2 t) of gold at its Water Tank Hill deposit and the nearby Twin Hills deposit. Production was projected at 180,000 oz of gold per year. Due to cash flow problems within Greenstone, mining and crushing operations ceased at San Andrés in mid-December 1999.

Greenstone management and most expatriates left the country in March 2000. Many of the local employees also chose to leave. A small crew remained to manage leaching operations, maintain the site, and perform the actions required under the site permits.

The property rights and obligations associated with the Project were transferred to the Honduran bank, Banco Atlántida, and on June 26, 2000, its real estate branch provided a bridge loan for operations to resume. Minosa was formed to own and operate the mine.

RNC was retained to provide management services to Minosa. Most of the expatriates and senior management at the San Andrés Project worked for RNC. Minosa resumed mining operations in early August 2000 at the Water Tank Hill Deposit.

On September 7, 2005, RNC announced an agreement to purchase 100% of the Mine for approximately US$22.5 million. A private Belize company, San Andrés (Belize) Limited, which owned 75% of the mine equity, signed a letter of intent to sell its shares for US$12.0 million plus the NSR. The NSR was calculated as 1% on the first US$20.0 million of annual revenues reducing to 0.5% on the remaining annual revenues. The cumulative maximum NSR was US$1.5 million and has been fully paid. It was also agreed that the existing mine debts to a local Honduran bank of approximately US$5.5 million would be extinguished. A company controlled by senior executives of RNC agreed to sell the remaining 25% equity interest in the mine on the same basis.

On December 4, 2005, Yamana announced an agreement to acquire RNC and 100% of the Mine. As part of this transaction, Yamana advanced a US$18.9 million senior secured loan to RNC to fund the acquisition of the 75% interest in the mine, the repayment of debt by San Andrés (Belize) Limited and certain other transactions completed by RNC in connection with the acquisition. The remaining 25% interest was purchased for shares of Yamana. The acquisition of RNC by Yamana was approved at a shareholder meeting on February 17, 2006 and the transaction was completed on February 28, 2006.

Aura acquired a 100% beneficial interest in the Mine in August 2009.

6.2 Previous Mineral Resource and Mineral Reserve Estimates

Aura reported Mineral Resources and Mineral Reserves for the Mine in its Annual Information Form for the Fiscal Year ending December 31, 2011 (Table 6-1).

The Mineral Resources; classified as Measured, Indicated and Inferred; were based on cut-off grades of 0.28 g/t Au for oxide and 0.37 g/t for mixed material at a gold price of US$1,500/oz. Note Mineral Resources are inclusive of Mineral Reserves.

Table 6-1: December 31, 2011 Mineral Resource Estimate

Resources Category	Oxide			Mixed			Total		
	Tonne (t)'000	Au (g/t)	oz' 000	Tonne (t)'000	Au (g/t)	oz' 000	Tonne (t)'000	Au (g/t)	Oz '000
Measured	54,272	0.47	818	13,604	0.61	265	67,877	0.50	1,082
Indicated	20,101	0.47	306	10,962	0.69	243	31,062	0.55	549
Measured + Indicated	74,373	0.47	1,124	24,566	0.64	508	98,939	0.51	1,631
Inferred	1,557	0.44	22	361	0.82	10	1,918	0.51	32

Note:
1. a cut-off grade of 0.28 g/t Au for oxide material and a cut-off grade of 0.37 g/t for mixed material
2. numbers may not add up due to rounding.

The Mineral Reserves, classified as Proven and Probable, were based on a cutoff grade of 0.34 g/t Au for oxide material and a cutoff grade of 0.45 g/t Au for mixed material. Mineral Reserves have been estimated based on a US$1,250/oz gold price.

Table 6-2: December 31, 2011 Mineral Reserve Estimate

Ore (1)	Oxide			Mixed			Total ore		
	Tonne (t)'000	Au (g/t)	oz' 000	Tonne (t)'000	Au (g/t)	oz' 000	Tonne (t)'000	Au (g/t)	Oz '000
Proven	27,832	0.52	465	5,703	0.63	116	33,536	0.54	581
Probable	9,629	0.49	152	2,598	0.61	51	12,227	0.52	203
Proven + Probable	37,461	0.51	617	8,301	0.62	167	45,762	0.53	784

Note:
1. Reserves at cutoff of 0.34 g/t for oxide ore and 0.45 for mixed ore
2. Numbers may not add due to rounding.

Note that the 2011 Mineral Reserve estimate did not include the Cemetery zone because the Company was then negotiating the relocation of the cemetery to gain access to the Mineral Resources below it at that time. For the 2013 Mineral Reserve estimate the cemetery zone has been included since agreements have been entered into with the local communities to relocate the cemetery.

Minosa continues exploration drilling at the property, and has a 12,500 m program planned for 2014. Historical drilling has been primarily conducted in the areas between the East Ledge deposit and the Twin Hills with the planned drill holes focusing on the south Cemetery area and Cerro Cortez.

6.3 Historical Production

Minerales de Copán conducted mining operations at San Andrés from 1983 to 1997. Between 350 and 500 short tons per day of gold-silver ore were mined from exposed mineralized conglomerates. All mining operations at San Andrés ceased in May 1997 in preparation for the construction of the expanded facilities. Greenstone resumed mining activities in early 1999 and operated the mine until December 1999 when Greenstone went bankrupt. Minosa resumed mining activities in August 2000. The Water Tank Hill pit was depleted in early 2003 and production commenced in the East Ledge pit in March 2003 and in the Twin Hills pit in 2008. Historical and recent production figures are shown in Table 6-3.

Table 6-3: Historical and Recent Production[1]

Year	Ore Leached Tonnes	Grade g/t Au	Gold Recovered (Oz)	Silver Recovered (Oz)
1983	21,480	-	-	-
1984	22,459	2.12	1,388	575
1985	22,332	2.46	1,433	636
1986	29,120	3.08	2,510	750
1987	40,178	2.46	2,710	806
1988	56,154	2.21	2,957	803
1989	76,209	1.87	3,406	1,247
1990	105,598	1.37	3,495	1,120
1991	133,084	1.93	4,813	1,385
1992	129,647	1.09	3,737	944
1993	138,766	1.15	4,607	1100
1994	138,083	1.06	4,291	739
1995	130,956	0.93	3,482	708
1996	127,801	1.21	4,504	1,242
1997	42,885	0.87	1,048	262
Sub Total	1,214,753	1.41	44,381	12,317
1998	-	-	-	-
1999	1,357,544	2.04	42,455	44,392
2000	-	-	6006	7,477
2000	719,631	1.85	17,508	22,841
2001	2,289,276	1.75	105,775	131,201
2002	3,378,116	1.09	99,064	108,694
2003	2,891,890	0.63	50,795	35,421
2004	3,793,870	0.69	65,032	18,502
2005	3,392,092	0.72	61,236	16,488
2006	3,732,049	0.70	70,779	-
2007	2,910,904	0.52	51,240	34,992
2008	3,567,279	0.58	47,761	17,636
2009	4,530,009	0.68	68,372	34,406
2010	4,913,900	0.70	70,641	52,394
2011	4,312,947	0.68	60,871	38,208
2012	4,263,953	0.64	59,751	41,487
2013	5,370,142	0.58	63,811	34,765

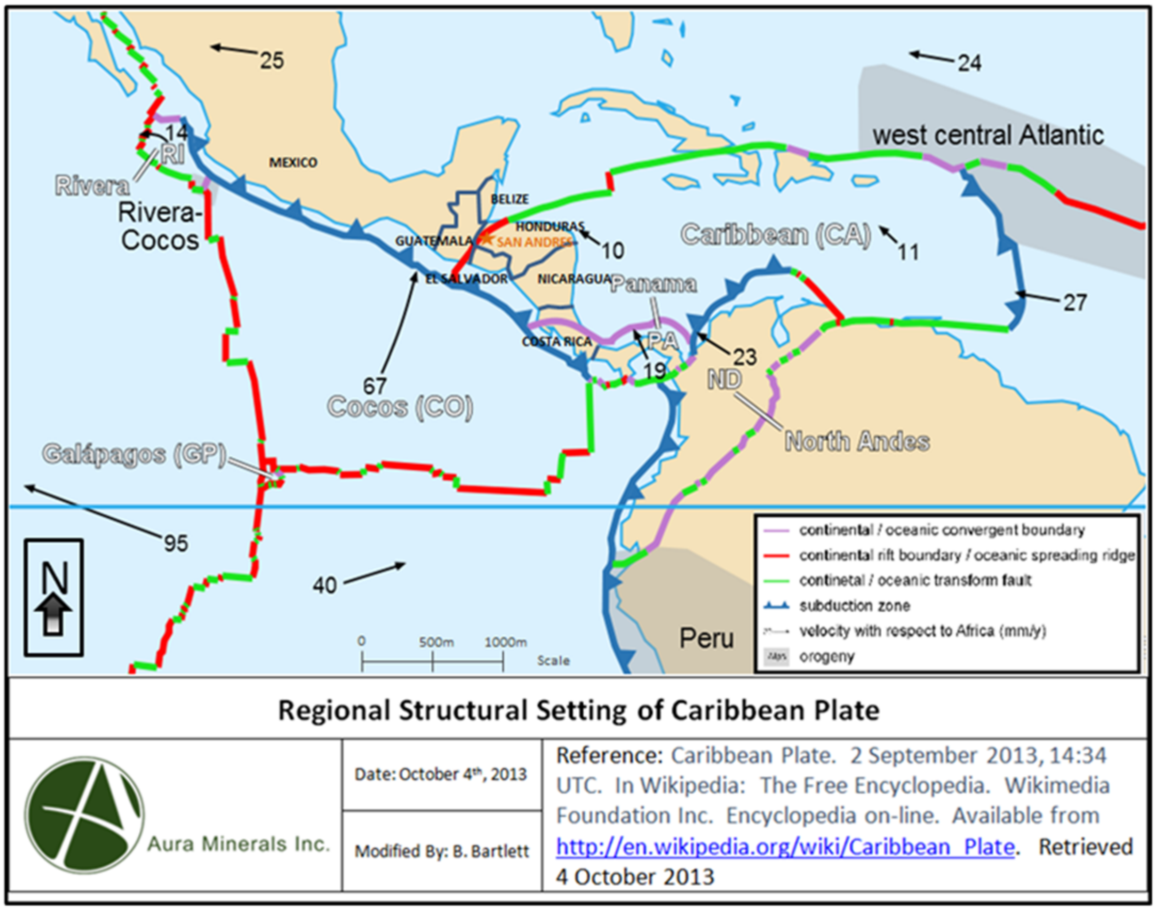
Note: Sourced from internal production data

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

Five lithospheric plates form boundaries in the region. Most of Central America between Guatemala and Costa Rica lies on the Caribbean Plate. The Mine is located on the northern edge of the Caribbean Plate at the boundary with the North American Plate. A chain of active volcanoes situated along the Pacific coast from Guatemala to Costa Rica marks the subduction zone which lies between the Cocos and Caribbean plates along the Middle American Trench. Figure 7-1 shows the regional structural setting of the Caribbean plate.

Figure 7-1: Regional Structural Setting of the Caribbean Plate



The boundary between the Caribbean and North American plates in Central America is marked by the Motagua Suture Zone. The suture zone is approximately 80 km wide and extends through Honduras and Guatemala. Three major faults are recognized within the zone: the Polochic Fault, the Motagua Fault, and the Jocotan Fault. These faults are predominantly strike-slip with left-lateral movement and are seismically active. The Motagua Suture Zone is terminated to the west by the Middle America Trench, marking the boundary with the Pacific plate.

The strike-slip faults forming the Motagua Suture Zone extend offshore to the east and form the Cayman Trough. The Cayman Trough formed as a result of strike-slip faulting on the Swan Islands Fault and the Oriente Fault. The Swan Islands Fault and the Oriente Fault are transform faults that form respectively, the southern boundary and the northern boundary of the Cayman Trough. The Cayman Trough is approximately 100 km wide and extends from the coast of Honduras to Hispaniola on the east, cutting through the Oriente Province in southeast Cuba (Gordon, 1997). The trough terminates at the Puerto Rico Trench. The Cayman Trough and the Motagua Suture Zone extend over a distance of about 2,500 km from the Puerto Rico Trench to the Middle America Trench on the west.

The Cayman Trough contains a slow-spreading center, referred to as the Mid Cayman Rise (Ten Brink et al., 2002). This feature is oriented north-south and connects the transform faults forming the boundaries of the Cayman Trough. It is a major extensional fracture formed at the same time as the strike-slip faults. Both of these structural elements result from the relative movements of the North American and Caribbean plates. Rock samples collected from the rise are largely basaltic in composition. The Mid Cayman Rise is a zone of high heat flow, which is typical of spreading centers.

Further evidence of the formation of north-south extensional fractures is also demonstrated from mapping in the vicinity of the Oriente Fault in Southeast Cuba (Rojas-Agramonte et al., 2003) and is exemplified by karst-filled extensional veins and normal faults.

7.2 District Geology

Honduras can be divided geologically into three zones. The northern third of the country, the Cordillera del Norte, generally consists of Permian metamorphic rocks ranging in age from 280 to 225 Ma. The central third of the country, the Cordillera Central, consists primarily of Cretaceous sedimentary rocks ranging in age from 136 to 65 Ma. The southern third of the country, the Cordillera del Sur, is dominated by Tertiary volcanic rocks that range in age from 65 to 2 Ma. A generalized geological map is shown in Figure 7-2 and a stratigraphic section is shown in Figure 7-3.

Figure 7-2: District Geology (After Bikerman 2003, CAM 2004 and Eppler et al. 1986)

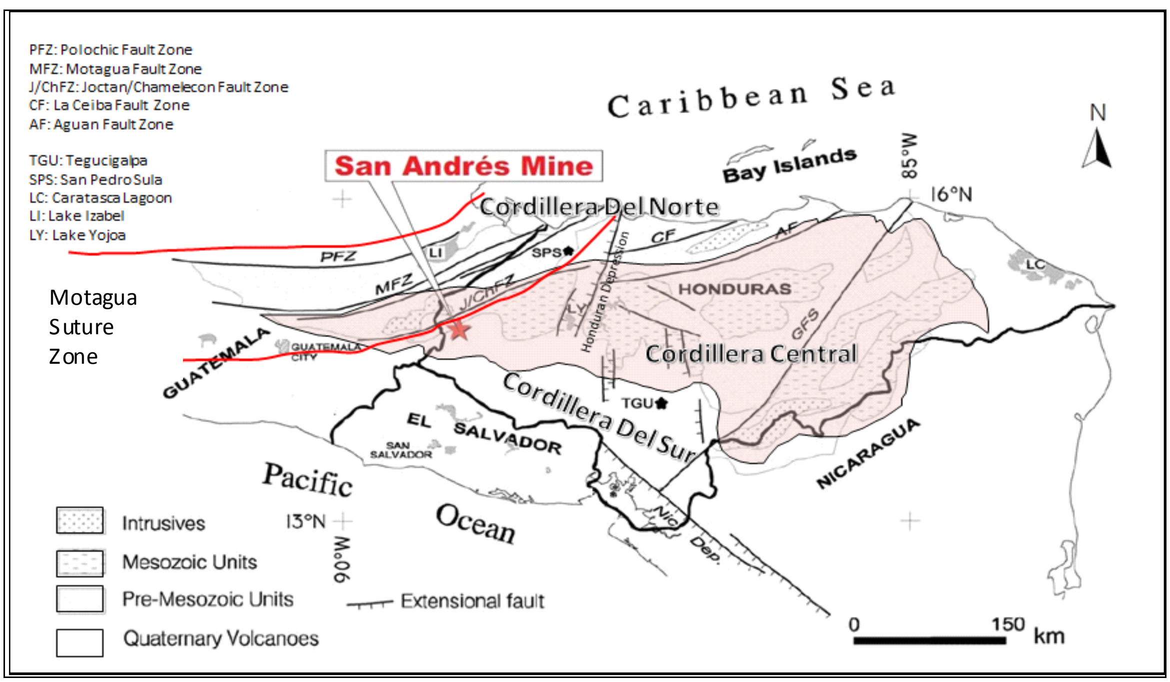
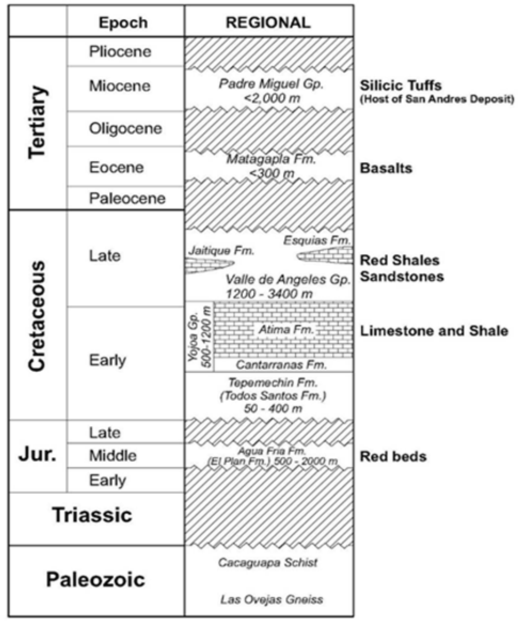


Figure 7-3: Statigraphic Column



Source: Eppler et al., 1986.

North-trending grabens associated with the "Honduras Depression" cut across Honduras from San Pedro Sula to the Golfo de Fonseca. The faults bounding these grabens, as well as a host of other faults throughout Honduras, average a northerly strike. However, they are stepped along northeast and northwest-trending faults that are dominantly normal-slip, but also have strike-slip components (Eppler et al., 1986). Eppler et al. also suggested that this extension was causing crustal thinning in the region.

Pflaker (1976) suggested that the northwest portion of the Caribbean Plate was fragmenting along these north-trending grabens as a result of the eastward movement of the Caribbean Plate. He suggested that the greatest movement on the graben faults was associated with the eruption of the Padre Miguel tuffs in the Miocene and Pliocene and that minor adjustments along these faults may be occurring today. He suggested that the east-west extension implied a north-trending horizontal principal stress.

There are abundant occurrences of hot springs throughout Honduras and hot springs occur within the immediate vicinity of the Mine. A hot spring was encountered during mining of the Water Tank Hill pit. Eppler et al. (1986) carried out an extensive study of these hot springs to assess the potential for geothermal resource sites. In their opinion, the absence of young silicic volcanism suggests that cooling plutons are not the heat source for the hot spring activity. Rather, the geothermal systems are caused by thin crust and high regional heat flow. In their inspection of several sites, Eppler et al. noted the association of the springs with the north-trending extensional faults and fractures.

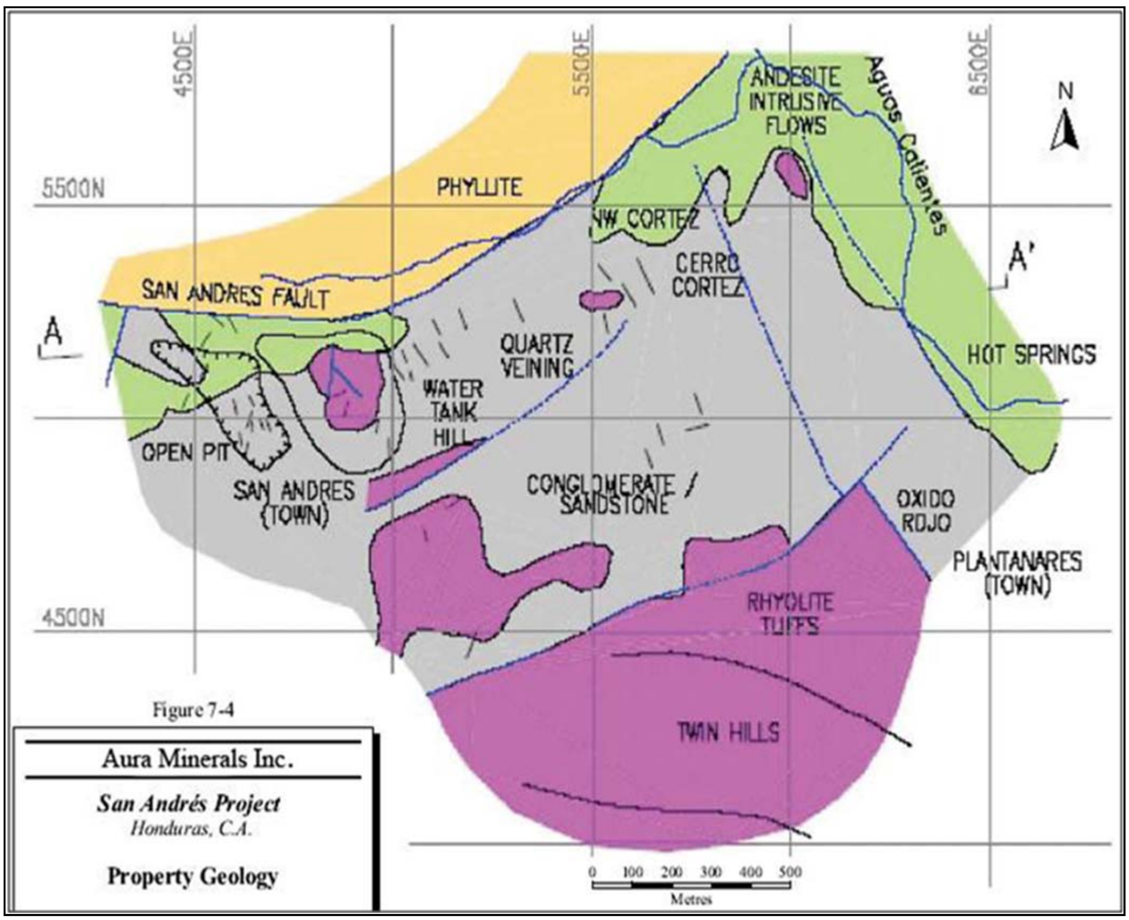
Eppler et al. indicated that the geothermal fluids were neutral to alkaline in pH and were best classified as Na-HCO_3-SO_4-Cl waters. They also observed that the systems with the highest temperature generally deposited silica sinter. Temperatures of the fluids varied from 120°C to 225°C.

7.3 Property Geology

The Mine is situated along the southern margin of the Motagua Suture Zone. The deposits occur on the south, or hanging wall side of the San Andrés Fault.

The oldest rocks recognized at San Andrés are Permian metasediments which are grey green to locally black in colour and appear to be a thick sequence of metamorphosed shales, sandstones, and arkosic sands (Figure 7-4). The metasediments (phyllites) are exposed on the north, or footwall side of the San Andrés Fault, located immediately to the north and northwest of the Water Tank Hill pit and mine area. Drilling in the phyllites demonstrate that they are carbonaceous, with 1% to 2% sulphide in the form of pyrite, and contain narrow veins of massive white (milky) quartz. The quartz is largely barren and interpreted to be pre-mineralisation, however to the North East corner of the mine lease local high grade (>2g/t Au) mineralised quartz and carbonate veining has been identified. This mineralisation in the immediate footwall of the San Andrés Fault has been identified and defined as the Buffa Zone.

Figure 7-4: Property Geology (Scott Wilson RPA)



The phyllites are overlain by porphyritic andesites of the Tertiary Matagalpa Formation. This unit underlies much of the Mine, the Cerro Cortez Hill area, and extends eastward to the Quebrada del Agua Caliente. The explosive phase of the andesites consists of agglomerates, flows and tuff breccias. Locally, the andesite appears to have intruded into the overlying sandstone and conglomerate rock units. The andesite-conglomerate contact is often very irregular in form and typically exhibits shearing. There are zones of mixed rock along the contact where angular fragments of andesite porphyry are found in a matrix of conglomerates and sandstones. In places, the andesite is grayish green in colour and consists of moderately abundant plagioclase and hornblende phenocrysts in a felsic to glassy matrix.

Overlying the andesites is a thick red bed sequence of quartz conglomerates, medium to fine sands and silts. These rocks are believed to be the Tertiary Subinal Formation. South of the Mine area, where unaltered, these rocks have the distinctive hematite brick red colour, but in the Mine area they have been bleached to light buff yellow and grey colours. These units typically have a shallow to moderate southerly dip and they thicken to the south of the Mine area.

The Subinal Formation sediments are overlain by poorly to moderately welded rhyodacite and rhyolite tuffs. Fine sands and silts with a tuffaceous matrix and quartz fragments of volcanic origin are also part of these tuff units and appear to be the basal portion in contact with the conglomerates and sands of the Subinal Formation which represents a change in the geologic environment. These units occur as intercalated, thin, discontinuous beds. The rhyodacite and rhyolite tuffs consist of crystal-rich, poorly to locally moderately welded tuffs with abundant biotite in a felsic groundmass. Quartz pyroclasts are not always present, but may be locally abundant. This unit is considered to be the Tertiary-aged Padre Miguel Group. The rhyolite tuffs crop out on the hill immediately to the east of the past producing Water Tank Hill pit and form thick ridge cappings in the southern and eastern portion of the district. A thin remnant of biotite crystal-rich rhyolite tuff was mapped on Water Tank Hill before it was mined.

The youngest sequences present in the area are Quaternary and recent alluvial deposits which fill canyon bottoms and stream valleys and occur locally as slope cover.

7.4 Structure

The Mine area is dominated by a series of sub parallel and widely spaced east to northeast-striking faults that are typically steeply dipping to the south and by numerous north to northwest-striking faults that also dip moderately to steeply to the east and to the west. The most prominent fault on the property is the San Andrés Fault which strikes east to east-northeast and dips steeply south. The San Andrés Fault forms a distinct boundary between the phyllites to the north (footwall side of the fault) and the host rocks for the Mine to the south (hanging wall side), which include the andesite, conglomerates, and rhyolite tuff units. The San Andrés Fault is parallel to, and coeval with, the major set of east to north-northeast trending strike-slip faults that form the Motagua Suture Zone and the direction of movement along the San Andrés Fault is also left lateral. Drilling has shown that the fault strikes east-northeast and dips at 70° to 80°south at the Mine.

Within the volcanic and sedimentary units on the south side of the San Andrés Fault, numerous and more closely spaced extensional faults and fractures have been mapped. The average strike of these faults is N25°W and the dip is 50° to 80° to the southwest and northeast, forming graben-like blocks where the strata are locally offset. Many of the extensional faults have numerous shallow north to northeast dipping linking structures with localised movement. Faults and fractures are generally filled with banded quartz and blade calcite with elevated mineralisation associated with intersecting structure sets. These extensional structures are distributed over a wide area, from the East Ledge open pit to Quebrada del Agua Caliente, approximately 1,500 m to the east, and from the San Andrés Fault, for at least 1,200 m south. At Platanares, the active geothermal springs and seeps are also associated with a northwest trending fault and fracture system. These extensional faults and fractures are interpreted as being coeval with the regional extensional structures resulting from the current rifting process and primary conduits for the mineralising fluids.

These extensional structures most likely exhibit low levels of seismicity. Micro-seismic monitoring systems could be used to identify zones containing major extension fracture systems for exploration purposes.

Juliano de Carvalho (2006) completed a structural analysis of the Mine area, focusing particularly on the East Ledge pit area. He divided the East Ledge pit into three main domains: North, East, and Central West based on similar structural features. The northern portion of the East Ledge pit is dominated by two main sets of structures: E-W strike-slip faults, probably related to the San Andrés Fault, and north-striking faults and extensional structures. He noted that strong hydrothermal alteration and mineralization is always associated with the extensional faults.

De Carvalho considered the Central West Domain to be the central part of a graben-like structure. This type of structure is cited in a number of papers as the principal control on the mineralization in the mined-out Water Tank Hill pit. These graben-type structures are thought to have formed in conjunction with the displacement on the San Andrés Fault.

The East Domain is in the east-southeast portion of the pit (Twin Hills) and is different from the other domains being characterized by a major concentration of northwest striking fractures. Alteration is dominated by strong silicification and quartz veining. This area is considered by de Carvalho to be a potential main pathway for the mineralizing fluids.

A prominent north-striking structure in the southwest wall of the East Ledge pit is reported to display a steep lineation of the slickenside on the footwall of one of the faults, indicating vertical movement. The fault was filled with banded chalcedony and bladed calcite and varied in width from one meter to 0.2 m. The strike of the fault was NNE and it dipped at 55° east.

7.5 Alteration

Rock alteration associated with the deposit includes an outer halo of bleaching and propylitization with mixed argilization and silicification central to the gold mineralization.

Minor to locally moderate amounts of sulphides, such as pyrite and possibly marcasite, were also introduced into the host rocks, but are now nearly all oxidized and occur as hematite, goethite, and jarosite. Oxidation in the ore zones extends to at least 100 m vertically in the East Ledge pit.

Propylitization is seen in the andesite flows and intrusive as a grayish green colouration with various amounts of very fine disseminated pyrite, chlorite, and calcite. The andesite shows weak to moderate argillic alteration throughout, with the plagioclase and groundmass altered to soft light coloured clays, but with phenocrysts still visible.

Within the sediments, the silty and fine-grained matrix has been strongly clay altered near the underlying andesite intrusive contact and along faults and vein structures. Weak to moderate argillic alteration is present over broad areas within the sedimentary rocks and is associated with hematite, goethite, and jarosite development.

Silicification varies from weak through intense to total replacement in both sediments and andesite. Although silicification is generally associated with veins and faults, local areas of flooding are noted. The silica flooded areas are locally blanket-like zones associated with controlling feeder structures. The silicic alteration is strongest in the tuffs.

7.6 Mineralization

At San Andrés, gold and silver mineralization is associated with a high level epithermal, quartz-carbonate-adularia system consisting of veins, stockworks, and disseminations. In the andesite, overlying conglomerate and rhyodacite, the quartz veins are typically composed of banded chalcedony and fine grained white quartz which has replaced calcite. The bladed calcite texture seen in veining is ubiquitous and the quartz replacement is almost always complete.

Metallurgical studies show that the gold is primarily contained in electrum as fine-grained particles. The particle size of the electrum grains varied from 1 micron ("μ") x 1 μ up to 10 μ x 133 μ. One native gold grain was noted.

The silver mineralization in the San Andrés deposit generally occurs at about the same grade as the gold mineralization. Scott Wilson RPA assessed the correlation between the silver assays that were available with the gold assays. The correlation coefficient was low at 0.24. However, because of the much lower price for silver and the lower metal recoveries, the value of the silver recovered is less than 1% of the value of the gold produced. Therefore silver samples are sporadic and localised to certain domains and largely have an inadequate data density to reliably estimate the grade; therefore Silver grades are not reported in the Mineral Resource statement.

7.6.1 East Ledge and Water Tank Hill Deposits

The rock formations and structural setting of the East Ledge zone are similar to that observed in the Water Tank Hill pit 200 m to the west. The highest gold grades and the most intense hydrothermal alteration are associated with the north to north-northwest striking normal faults and extension fracture system. Gold mineralization has not been observed in the east and east-northeast trending faults that parallel the main San Andrés Fault. The San Andrés Fault and associated faults are shear-type faults and extensional-type structures have not been observed paralleling these faults.

The gold grade in the Water Tank Hill pit is higher than the gold grade at East Ledge. The higher gold grade in the Water Tank Hill deposit is attributed to a flexure along the San Andrés Fault zone and massive Andesite, which has subsequently resulted in a greater concentration of north-striking normal faults and extension fractures and the greater width of the quartz veins. The quartz veins at Water Tank Hill averaged one to two metres in width. At East Ledge, the veins are thinner and form more of a stockwork. The north-striking extensional structures are still present, but they are not as prominent. The quartz veins average only about 0.5 m in width and at the Twin Hills deposit, the veins are even thinner.

The preponderance of extensional fractures in the Water Tank Hill deposit is most likely a result of the presence of the andesite unit and the flexure along the San Andrés Fault Zone. The andesite is more massive and brittle than the conglomerate unit and possibly may have been more amenable to fracturing as a result of tectonic stress. The fractures provided permeable channels and traps to allow the flow of the hydrothermal fluids and concentrate the gold deposition. The conglomerate unit is much weaker and the fractures may not have formed as readily. The gold mineralization in the conglomerate unit may have been primarily deposited in the pore space, forming a more disseminated style of mineralization.

Within the East Ledge pit, the conglomerate and tuff are argilized to a buff-colour and the intensity of alteration appears to fade to the east towards Cerro Cortez and the gold grade also decreases. This relationship provides a qualitative indication of the limit of the economic gold mineralization within the pit for both planning and production purposes.

A new resource area, Zona Buffa, was defined by the 2008 drill-hole program. The resources are situated northeast of the East Ledge pit within the footwall of the San Andrés Fault and hosted within the phyllite, previously thought to be barren of mineralisation. The Buffa zone has been considered in the estimation of Mineral Resources but not in the estimation of Mineral Reserves, pending further detailed environmental and technical studies.

7.6.2 Twin Hills Deposit

The gold mineralization at Twin Hills occurs in a more widespread, disseminated pattern with less quartz veining and structural control within the Padre Miguel Group rhyolite tuffs with a thickness of more than 75 m. The low-grade disseminated mineralization is strata bound within the sequence. The mineralized zone dips to the south at about 10° to 20°. The deposit is oxidized from 10 m to 50 m below surface. There are few high-grade areas along defined structures although in places higher-grade zones appear to largely inter-finger with the lower grade mineralization.

De Carvalho (2006) describes three types of mineral association within the Twin Hills area:

- Gold mineralization associated with quartz veining. The veins are normally contained within the north-northwest trending fractures and vary in size from a few millimeters up to one meter. The veins may have associated pyrite in the fresh zone at depth and in the oxide zone; the associated minerals are hematite, goethite, and jarosite. The gold grades are generally highest in the quartz veins, with grades up to 17 g/t Au reported.
- Gold mineralization associated with strongly altered rocks. This style of mineralization is primarily hosted in conglomerates and arenites of the Subinal Formation. Hydrothermal alteration is characterized by strong propylitization, argillization, and silicification. Gold grades are variable, but rarely exceed 1.5 g/t Au. De Carvalho reports the presence of intense fracturing, which normally occurs in the hanging wall of the north to northeast striking extension fractures, and normal faults although some of the mineralization may be associated with permeability within the conglomerate and arenite units.
- Gold mineralization within permeable rock units. De Carvalho describes this style of mineralization in the Twin Hill area within unfractured conglomerate. This style of mineralization forms a more wide-spread disseminated pattern and quartz veining is rare. De Carvalho also suggests that this style of mineralization may be stratigraphically controlled rather than structurally controlled. The conglomerate layers are interlayered with tuffs and siltstones.

As part of a resource and reserve study in April 1999, Mine Development Associates of Reno, Nevada, USA (“MDA”) completed a correlation study of geological features versus precious metal grade. Their conclusions are summarized below:

- precious metal grades are positively related to quartz veins in all rock types, with minor variations;
- precious metal grades are related to quartz after calcite, with minor variations;
- calcite veining does not form a definitive relationship with grade;
- in general, there is an increase in grade with an increase in hematite/limonite;
- silicification and grade is positively correlated;
- oxidation does not form a relationship with grade; and
- Propylitic alteration is inversely related to grade.

MDA coded the gold assays by geological zone and performed a similar study. It was found that the geological zones honoured the grade-geology relationships described above. For example, as the geologic zone increased from a low grade to a high-grade zone, the logged limonite, hematite, fractures, quartz veins, silicification and quartz after calcite increased. Carbonate decreased along with propylitic alteration while the present oxidation and calcite veining did not show consistent trends with the gold grades.

7.7 Near-Surface Oxidation

The rocks hosting the San Andrés deposit have been oxidized near surface as a result of weathering. The zone of oxidation varies in depth from 10 m to more than 100 m. The zone of oxidation is generally thicker in the East Ledge deposit compared to the Twin Hills deposit.

In the oxide zone, the pyrite has been altered to an iron oxide such as hematite, goethite, or jarosite. The oxide zone generally overlies a narrow zone of partial oxidation, called the mixed zone, which consists of both oxidized and fresh material. The mixed zone may not occur, but where it is present, it reaches thicknesses of over 50 m.

At depth, below the zone of oxidation, the gold is commonly associated with sulphide minerals such as pyrite. This fresh or “sulphide” zone lies below the mixed zone.

The contact between the oxide zone and the sulphide zone is very clear in the pit mapping, in the core, and in the chip samples from the RC drilling.

The gold contained in the oxide zone is amenable to extraction by heap leaching using a weak cyanide solution. The gold recovery is reduced in the mixed zone as a result of the presence of sulphide minerals and the gold cannot currently be recovered economically from the sulphide zone by heap leaching. The estimated metal recovery by leaching from each zone is discussed in Section 13.

A high content of clay minerals in the ore resulting from the alteration is detrimental to the heap leaching process because of reduced permeability in the pile. This situation is resolved by agglomerating the crushed ore by adding cement to increase the permeability of the pile prior to heap leaching.

8.0 DEPOSIT TYPES

The San Andrés deposit is an epithermal gold deposit associated with extensional structures within tectonic rift settings. These deposits commonly contain gold and silver mineralization, which is associated with banded quartz veins. At San Andrés, silver is not economically material. Gold occurs in quartz veins that are predominantly comprised of colloform banded quartz, generally chalcedony, with lesser amounts of fine comb quartz, adularia, dark carbonate, and sulphide material. The gold mineralization is deposited as a result of cooling and the interaction of hydrothermal fluids with groundwater and the host rocks. The hydrothermal fluids may have migrated some distance from the source; however, there is no clear evidence at San Andrés that the fluids, or portions of the fluids, have been derived from magmatic intrusions.

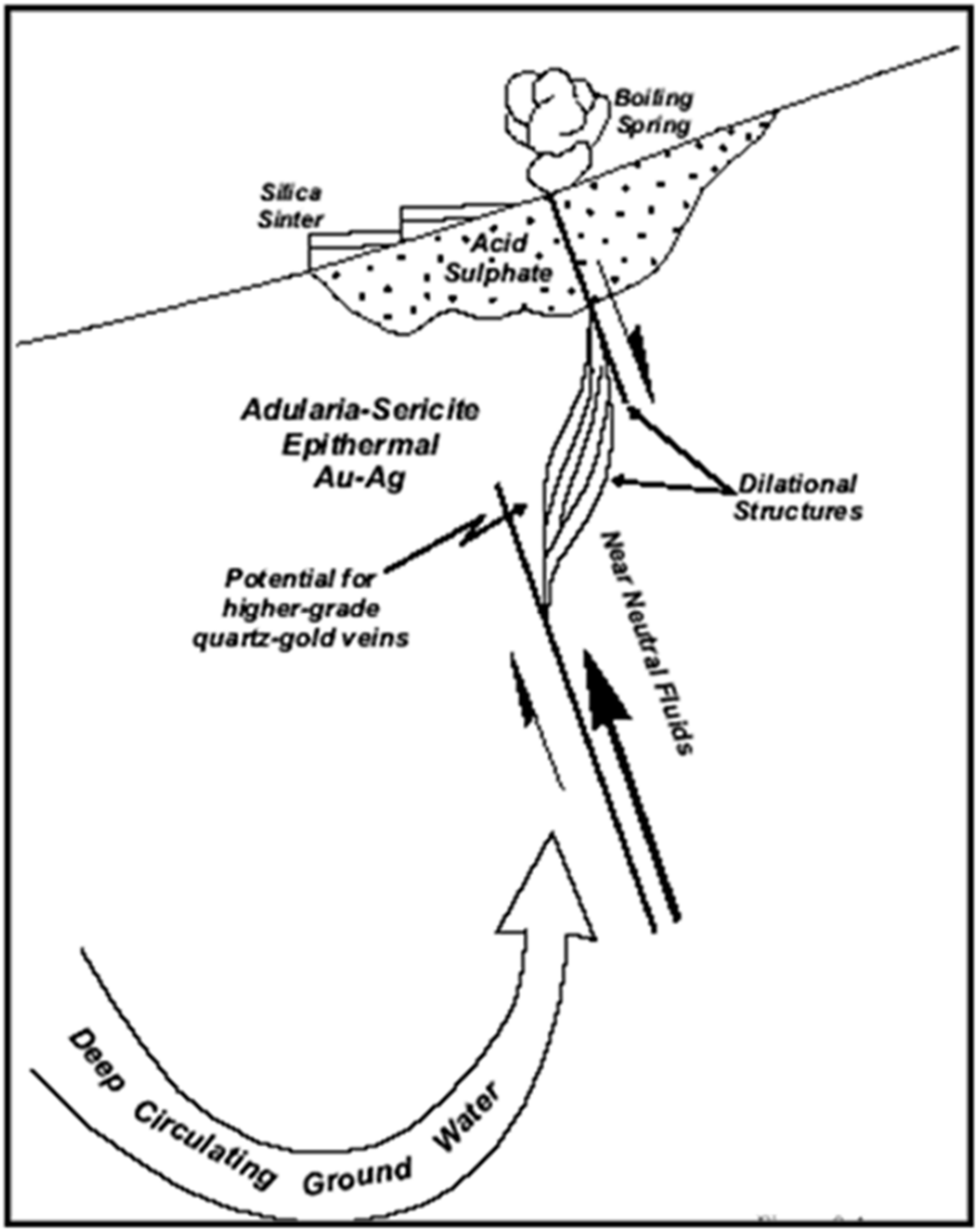
Many of these low sulphidation epithermal deposits occur in felsic volcanic sequences where geothermal fluids are circulating. Near surface, many deposits are capped by eruption breccias which are formed by the rapid expansion of depressurized geothermal fluids. These breccias are characterized by intensely silicified matrix and angular fragments of the host rock. Wall rock alteration forms as halos to veins and includes sericite grading to peripheral smectite and marginal chlorite alteration.

Corbett (2002) suggests that structure and the competency of the host rocks may be important ore controls for the vein systems. The extension fractures form in the stronger, more competent rocks. Higher grade ore shoots generally develop in areas with a greater frequency of extensional structures, or at dilational jogs or flexures in the veins.

The mineralization at San Andrés appears to be in an upper level epithermal system as indicated by the hydrothermal alteration patterns, the disseminated style of mineralization, the presence of both gold and silver associated with quartz veining, the presence of active hydrothermal fluid flow at the property and the actively forming extensional fracture system, which creates the permeability.

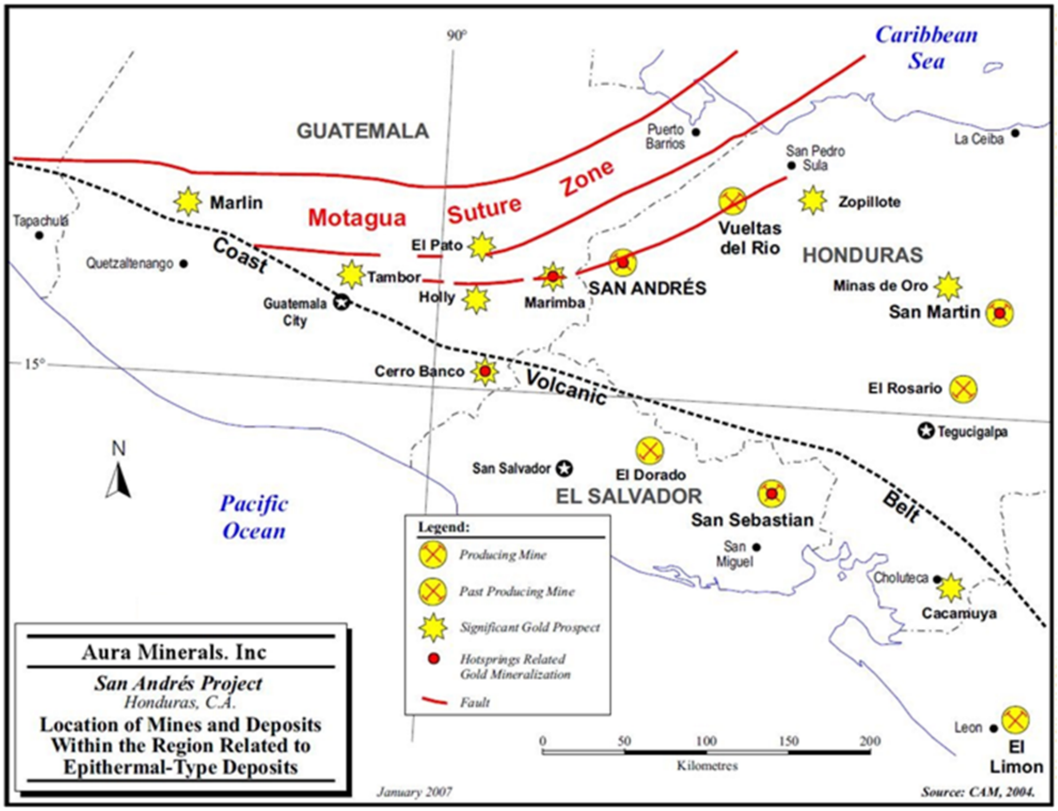
Figure 8.1 shows a schematic drawing of epithermal deposits showing the likely position of the San Andrés deposit, adapted from Corbett (2002).

Figure 8-1: Schematic Diagram of a Low Sulphidization Epithermal Deposit (Scott Wilson RPA)



The region hosts a number of mines and mineral deposits with a similar style of mineralization as the Mine and epithermal-style mineralization formed deeper in the system. The location of these mines and deposits is shown in Figure 8-2.

Figure 8-2: Location of Mines and Deposits (CAM 2004)



9.0 EXPLORATION

CAM (Armbrust et al., 2005) provides a summary of previous exploration work on the property from which much of this section has been drawn. Much of the information in the CAM report was drawn from Bikerman (2003a and 2003b).

9.1 Previous Exploration

Following Yamana's acquisition of RNC in 2005, they carried out exploration programs from 2006 to 2008 consisting of RC and diamond drilling, and geological mapping and sampling of rock exposures in road cuts. This work was carried out to better define the pit boundaries, test areas between the pits, and assess potential extensions of the mineralization in other areas of the mining lease. Results from the Yamana drilling and the previous drilling programs are discussed further in Section 10.

The core drilling, in particular, has generated new insight into the deposit geology and the controls and extent of the mineralization. Previous exploration was focused on the three principal pit areas (Water Tank Hill, which is now depleted; East Ledge and Twin Hills which are currently being mined); however, a recent review of the drilling results by Aura indicates that the gold mineralization system is more extensive than previously recognized. Economically extractable gold mineralization is continuous between the East Ledge pit and the Twin Hills deposit and extends to the east, west, and south of these deposits. The current plan is to combine the two pits into one large pit extending into surrounding areas of Cerro Cortez and the Banana Ridge areas. Further expansions may be warranted depending on the results of any continuing exploration program.

The structural geology study completed in April 2006 (de Carvalho, 2006) with particular emphasis on the exposures in the East Ledge pit was discussed in the previous sections. This study provided a better understanding of the structural controls on mineralization and several recommendations were made for targets to be drilled. A study of the alteration and mineralization by Hodder (2007) has better defined the controls and distribution on gold mineralization.

Since Aura acquired a 100% interest in the Mine in August 25, 2009, Minosa has carried out further exploration activities at the mine consisting of property scale mapping, road cut channel sampling and trenches in Cerro Cortez, Agua Caliente, Zona Buffa and Fault A and reverse circulation drilling targeting the Twin Hill South, Banana Ridge, Fault A, Cerro Cortez, Zona Buffa and Agua Caliente areas, totaling 6,209 m. The exploration program helped to develop the geological model and define future targets for infill drilling. These works were carried out to better define the pit boundaries and test areas to assess potential extension.

9.1.1 Topographic Surveys

The San Andrés property was flown for aerial photography on March 31, 1996 by Hansa Luftbild German Air Surveys of Munster, Germany. A total of 24, 1:15,000 scale colour photographs were taken over a total of three flight lines. These aerial photos were ortho corrected and topographic maps with two-meter contour intervals were created by Eagle Mapping Services Ltd. of Vancouver, British Columbia, Canada. The topographic map is centered on the old San Andrés town site and covers approximately 25 km^2. A total of seven ground survey control points were established, four of which were resurveyed for confirmation. Using this control point data the photos were ortho corrected and the two-meters contour topographic maps were completed in November 1997. The data were provided in digital format and are all in local mine grid coordinates.

This digital topography was used by Minosa in the design of the East Ledge and Twin Hills block models and resulting pit designs in 2005 (CAM, 2005). Additional topographic data are routinely collected and incorporated into the mine database by qualified survey engineers using modern electronic total stations and AutoCAD software.

9.1.2 Geological Mapping and Sampling

Geological mapping at a scale of 1:1,000 was conducted on the 1,150 m bench level of the Water Tank Hill pit in 2001 in order to determine possible geologic controls to mineralization that could be used in the block model for the deposit. Control points with surveyed mine grid coordinates were established at approximately 30 m intervals along the bench and plotted onto the bench map. A 100 m tape was then used to accurately locate geologic features such as faults, fractures, lithological contacts, and alteration. Geologic features seen in the bench face are projected to the toe elevation of the bench.

Geologic mapping of the East Ledge pit high wall was conducted between the 1,120 m and 1,060 m elevations (11 benches) as the East Ledge pit was advanced (July through December 2004). Survey control points were established along the high wall and a string gauge (hip chain) was used for the plotting of geologic information at a scale of 1:1,000. This information was used to assess the mineralization, ore controls, and structural complexities of the East Ledge deposit as well as for use in the geotechnical monitoring of the high wall area of the pit. Geotechnical monitoring and geological mapping are on-going.

Production mapping by mine geologists in the Twin Hills pit and the final excavated East Ledge pit were completed during 2011 and 2012 and used to generate a 3 dimensional structural model for mine planning and determining structural controls on the mineralisation within the Mine.

9.1.3 Geochemical Surveys

During 1997 and 1998, geological mapping and sampling programs undertaken by Greenstone over Twin Hills and surrounding areas involved the collection of nearly 1,700 bedrock channel samples from all significant road cut and outcrop exposures on the property. The mapping and channel sampling programs helped in the development of the geological model, in the definition of mineralized zones, and definition of targets for drilling. These samples were not used in the estimation of the Mineral resources. A geological outcrop-style map covering most of the San Andrés concession area was prepared by Barranca Resources, Inc. on behalf of Greenstone in November 1997.

9.1.4 Geophysical Surveys

Between December 1997 and February 1998, Quantec IP Inc. (Toronto) conducted an Induced Polarization ("IP") and magnetometer geophysical survey covering an area from Water Tank Hill, south to Twin Hills and east over Cerro Cortez within the San Andrés concession. Approximately 27.7 km were covered with readings taken every 12.5 m along lines spaced 50 m apart, oriented at N65°E. Previously mapped structures were identified by the IP survey.

Interpretation of the data from the survey resulted in the identification of four structurally favorable zones, three in the north to south corridor between Cerro Cortez and Twin Hills and a fourth located south of the Water Tank Hill deposit. The IP survey also identified several Priority 2 targets. The Priority 1 targets are located in the centre of the old Water Tank Hill pit, the centre of the pre-2006 Twin Hills resource area (it should be noted that these two areas had already been defined by exploration drilling at the time of the survey), the northern limit of the Twin Hills resource, and the east side of Cerro Cortez. The first two targets are in areas that have or will be mined out. The target identified along the northern limit of the pre-2006 Twin Hills resource area was tested with one Greenstone-era core hole (SC-034), which intersected mineralized material down to a depth of 50 m with individual samples up to 3.26 g/t Au. From 50 m to the total depth of 151.50 m, the hole was relatively barren.

9.2 Recent Exploration

Minosa commenced a Mineral Resource and Mineral Reserve delineation campaign in September 2012 focussing on the Cerro Cortez and Cemetery areas. This campaign consists of 30,000 m of RC drilling of which 16,820.4 m were completed for the Mineral Resource and Mineral Reserve update. An additional 802.7 m of in-pit RC drilling were also completed for the Twin Hills open pit. All samples have been analysed at the onsite lab with a detailed QA/QC and independent lab checking program in place through Inspectorate.

Two RC drill rigs were used in the 2012 campaign, the first is owned and operated by Minosa and the second is owned and operated by Rodio Swissboring Honduras S.A ("Swissboring"), contractor drilling services. Swissboring has completed earlier drilling programs on site and is familiar with the San Andrés drilling environments. For the 2013 drilling campaign the Mine utilized only the Minosa drill rig.

Inspectorate was selected as the independent sample check lab due to practicality of transporting samples from Honduras and the Inspectorate prep lab in Guatemala is the closest independent lab services to the Mine. All analysis of samples from the Guatemala lab is completed in Reno, USA.

10.0 DRILLING

Drilling data used by Minosa in the December, 31 2013 Mineral Resource included RC, Diamond and Production blast hole information. Production data was limited to immediate production areas where mine to mill production reconciliation supports the use of the close spaced production data.

10.1 Exploration Drilling

Drilling by Greenstone in 1994 was initially carried out at Water Tank Hill because of the historical production from the area. The Twin Hills deposit was discovered by Greenstone in 1994 and the East Ledge deposit was discovered by Minosa in 2001.

Most of the drilling at San Andrés has been RC drilling. As different drill campaigns have been largely focused on defining particular open pit resources, the drilling is discussed initially by principal target areas, followed by a summary.

The drill section line spacing ranges between 25 m intervals to 100 m intervals depending on the relative maturity of the ore zones. The drill spacing ranges from:

- ~25 m x 20 m at Water Tank Hill
- ~25 m x 25 m at East Ledge and Cemetery area
- ~50 m x 50 m at Twin Hills
- >50 m x 50 m for Cerro Cortez, Twin Hills North, Buffa Zone, Agua Caliente and Central Zone Areas.

Reconnaissance drilling by Greenstone, Minosa and Yamana has also been conducted at wider spacing in other areas within the San Andrés concession.

Table 10-1 presents a summary of the drilling by method and by mineralized area. Figure 10-1 displays the location of the drilling. Table 10-2 present the summary of the drilling by method and campaign.

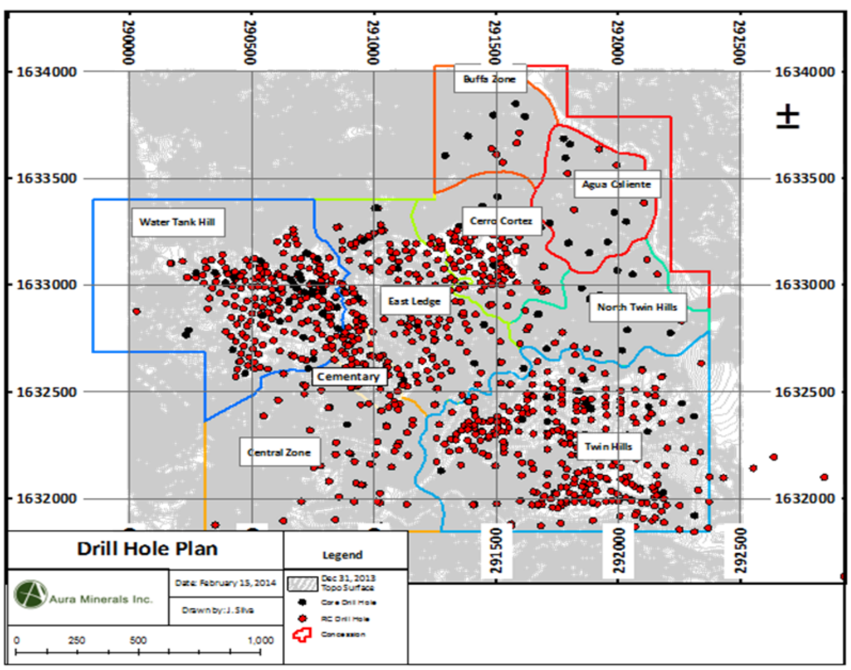
Table 10-1: Diamond & Reverse Circulation Drilling Summary by Area

Area	RC		Core		Total	
	No. of Holes	Metres	No. of Hole	Metres	No of Holes	Metres
Cemetery	61	8,249.0			61	8,249.0
Water Tank Hill	213	22,590.9	34	4,205.0	247	26,795.9
East Ledge	142	18,903.2	14	3,071.6	156	21,974.8
Twin Hills	297	29,385.2	18	4,093.8	315	33,479.0
Central Zone	50	7,432.5	4	985.7	54	8,418.2
North Twin Hills	11	1,681.1	16	3,781.0	27	5,462.1
Cerro Cortez	103	13,929.0	7	1,382.1	110	15,311.1
Agua Caliente	7	704.1	10	1,413.1	17	2,117.2
Buffa Zone	6	688.5	5	585.5	11	1,274.0
Others	15	1,246.9			15	1,246.9
Total	**905**	**104,810.3**	**108**	**19,517.8**	**1,013**	**124,328.1**

Table 10-2: Diamond and Reverse Circulation Drilling Summary by Campaign

Company	Campaign	RC		Core		Total	
		No. of Holes	Metres	No. of Hole	Metres	No. of Holes	No. of Holes
Fischer-Watt	EX-1992	22	2,717.40			22	2,717.4
Greenstone	EX-1994	63	5,008.30			63	5,008.3
	EX-1996	41	5,920.50			41	5,920.5
	EX-1997	101	11,601.40	9	1,323.5	110	12,924.9
	EX-1998	150	18,437.90	37	4536	187	22,973.9
MINOSA	EX-2001	15	1,674.00			15	1,674.0
	EX-2002	49	6,306.50			49	6,306.5
	EX-2005	25	2,280.00			25	2,280.0
MINOSA-Yamana	EX-2006	113	17,639.20	12	2,566.1	125	20,205.3
	EX-2007	59	8,316.10	28	6,253.4	87	14,569.5
	EX-2008	12	1,900.10	22	4,838.8	34	6,738.9
MINOSA-Aura	EX-2010/2011	64	3,508.20			64	3,508.2
	SA-2010	9	426.8			9	426.8
	EX-2012	64	8,014.70			64	8,014.7
	SA-2012	21	853			21	853.0
	EX-2013	75	8,805.70			75	8,805.7
	SA-2013	22	1400.6			22	1,400.6
	Total	**905**	**104,810**	**108**	**19,518**	**1,013**	**124,328.2**

Figure 10-1: Drill Hole Location Map



10.2 East Ledge

The earliest exploration work in the East Ledge zone consisted of surface geologic mapping and outcrop sampling, followed with three drill holes by Fischer-Watt during their 1991-1992 exploration programs. Greenstone carried out a limited amount of drilling on the peripheries of the zone during their development drilling program on the Water Tank Hill deposit in 1998. Despite the known existence of mineralization in the area, the geologic interpretations of the area at that time were thought to preclude the possibility of a mineable gold deposit.

In March 2001, Minosa drill tested the East Ledge while evaluating several locations around the Water Tank Hill pit for additional waste dump sites. The 2001 drilling was originally planned for three holes to condemn the area for a waste dump site. The drill program was expanded to 12 holes as the first holes encountered significant mineralization, totaling 1,368 m drilled at the East Ledge deposit. The program was conducted using RC drilling with down-hole face sampling hammers and tri-cone bits.

A development drilling program to evaluate mineralization in the East Ledge zone began on February 1, 2002 using RC methods. The general plan for testing the East Ledge zone consisted of drilling on 50 m grid spacing with in-fill holes at 25 m when deemed necessary for better sample and geology resolution. A total of 6,000.5 m were drilled in 46 drill holes, with an average hole depth of 131 m in the 2002 drilling campaign and the drill holes were stopped when the sulphide zone was intersected.

In 2005, Minosa completed seven RC holes totaling 792.2 m at East Ledge. No diamond drilling was carried out in 2005. The program began in the area immediately south of the existing East Ledge pit limit with the drilling of MO-05-01. An additional five holes (MO-05-09 to MO-05-11, MO-05-23, and MO-05-25) were drilled in the South East ledge zone with one additional hole (MO-05-24) drilled from within the northern end of the East Ledge pit following up a high grade intercept in MO-02-33 drilled during the 2002 East Ledge definition drilling program. Minosa continued with exploration drilling in early January 2006 with drill holes MO-06-01 through MO-06-03 located between the depleted Water Tank Hill pit and the East Ledge pit. Mining over portions of this area during the first quarter of 2003 showed the presence of low to moderate-grade material in this zone and additional drilling was needed to better quantify the presence of economic mineralization.

Two holes (MO-05-12 and MO-05-13) were drilled immediately southwest of the East Ledge pit limit in the Central Zone exploring the Cemetery and Fault A trends. Three holes (MO-06-20 to MO-06-22) were also drilled in the Cerro Cortez zone in October towards the end of the 2005 exploration program.

In March 2006 drilling by Yamana focused on better defining the limits of the mineralization around the pit and testing the area between the East Ledge and Twin Hills zones. Drilling continued in the Central Zone with an additional eight holes (MO-06-04 to MO-06-12 and MO-06-14). A second RC drill, along with a CS-1500 core drill, was contracted through Swissboring (Guatemala) beginning at the end of March, which brought the total number of drills to three. Total drilling by Yamana in 2006 at East Ledge consisted of 5,992.5 metres of drilling in 36 holes and in 2007, a total of 1,397.7 m of drilling was completed at East Ledge.

The drill holes testing the deposit at East Ledge and south to the Twin Hills deposit are oriented at 065° and 245° directions to intersect the NNW striking extension fractures at approximately a right angle. A small percentage of the holes have been drilled in a NNE direction parallel to the strike of the extension fractures.

A summary of the drilling at East Ledge is presented in Table 10-3. A map showing the spacing and the location of the drill holes is shown in Figure 10-1.

Table 10-3: Diamond & Reverse Circulation Drilling Summary, East Ledge

Company	Year	Name of holes	No. of holes	Metres
Fisher-Watt	1992	SA-007, SA-008, SA-009, SA-011, SA-012, SA-013	6	839.7
Greenstone	1996	SA-108	1	159.0
Greenstone	1997	SA-226	1	147.0
Greenstone	1998	SA-231, SA-234, SA-236, SA-269, SA-272, SA-297, SA-298, SA-299, SA-300, SA-301, SA-314, SA-315,SA-316, SA-317, SA-318	15	2,067.0
Minosa	2001	MO-01-001, MO-01-002, MO-01-003, MO-01-004, MO-01-005, MO-01-006, MO-01-007, MO-01-008, MO-01-009, MO-01-010, MO-01-011, MO-01-013	12	1,368.0
Minosa	2002	MO-02-001, MO-02-002, MO-02-003, MO-02-004, MO-02-005, MO-02-006, MO-02-007, MO-02-008, MO-02-009, MO-02-010, MO-02-011, MO-02-012, MO-02-013, MO-02-014, MO-02-015, MO-02-016, MO-02-017, MO-02-018, MO-02-019, MO-02-020, MO-02-021, MO-02-022, MO-02-023, MO-02-024, MO-02-025, MO-02-026, MO-02-027, MO-02-028, MO-02-029, MO-02-030, MO-02-031, MO-02-032, MO-02-033, MO-02-034, MO-02-036, MO-02-037, MO-02-038, MO-02-039, MO-02-040, MO-02-041, MO-02-042, MO-02-043, MO-02-046, MO-02-047, MO-02-048, MO-02-049	46	6,000.5
Minosa	2005	MO-05-01, MO-05-09, MO-05-10, MO-05-11, MO-05-23, MO-05-24, MO-05-25	7	792.2
Minosa - Yamana	2006	MO-06-02, MO-06-03, MO-06-05, MO-06-09, MO-06-10, MO-06-12, MO-06-13, MO-06-14, MO-06-15, MO-06-16, MO-06-18, MO-06-30, MO-06-31, MO-06-32, MO-06-33, MO-06-34, MO-06-35, MO-06-36, MO-06-54, MO-06-55, MO-06-57, MO-06-59, MO-06-60, MO-06-61, MO-06-63, MO-06-65, MO-06-75, MO-06-77, MO-06-79, MO-06-81, MO-06-82, MO-06-91, MO-06-95, MO-06-97, MO-06-102	36	5,992.5
Minosa - Yamana	2007	MO-07-21, MO-07-26, MO-07-28, MO-07-32, MO-07-33, MO-07-34, MO-07-38, MO-07-39, MO-07-40	9	1,397.7
Minosa -Aura	2013	MO-13-05, MO-13-06, MO-13-07, MO-13-09, MO-13-11, MO-13-12, MO-13-13, MO-13-14, MO-13-17, MO-13-18, MO-13-21, MO-13-22, MO-13-23, MO-13-25, MO-13-26	15	1,234.4
	RC Total		142	18,903.2
Greenstone	1998	SC-027, SC-031, SC-033, SC-035	4	528.0
Minosa - Yamana	2006	MC-06-01, MC-06-02, MC-06-04, MC-06-09	4	745.7
Minosa - Yamana	2007	MC-07-03, MC-07-04, MC-07-05	3	987.0
Minosa - Yamana	2008	MC-08-13, MC-08-18, MC-08-19	3	811.0
	Core Total		14	3,071.6
	Total		**156**	**21,974.8**

10.3 Twin Hills

A total of 33,479 m of drilling in 315 drill holes has been drilled in the Twin Hills area. Of this total, 297 (29,385.2 m) were RC holes and 18 (4,093.8 m) were core holes for a total of 22,012 assays. The deposit has also been extensively channel sampled; however, only the drill hole (exploration and production) assays have been used in the estimation of the Mineral Resources and Mineral Reserves.

The objectives of the 1998 diamond core drilling program carried out by Greenstone at Twin Hills were to provide samples of ore and waste for specific gravity determinations; to provide samples of ore from which certain geological data not otherwise obtainable through RC drilling could be recorded; and to provide assay data that could be used as a check against the assay data being generated by the RC drilling. Eight core holes (SC-19, 28 to 30, 33, 34, 36 and 38) for a total of 1,127 m were drilled in the Twin Hills deposit.

In 2005, drilling was carried out by Minosa in an area south of the proposed Twin Hills pit to follow up on mineralization encountered in MO-02-44 drilled as a condemnation hole in the proposed Twin Hills waste dump location during 2002. A total of 13 short, vertical holes were drilled in the area of the proposed Twin Hills waste dump (MO-05-02 to MO¬05-08 and MO-05-14 to MO-05-19) on approximately 50 m centres during two phases of drilling.

In 2006 Yamana drilled a total of 6,784.1 m of drilling in 43 holes in the Twin Hills area and in 2007 Yamana drilled a total of 5,746.8 m of drilling in 33 holes drilled in the Twin Hills area.

During 2010 to 2013, Aura drilled a total of 6,188.6 m of drilling in 116 drill holes. The purpose of these holes was to increase drill hole density within the production areas.

At Twin Hills, the drilling is oriented in a north-south direction based on the general orientation of the host conglomerate and rhyolite formations. The holes are generally drilled at dips ranging from -45° to 60°. Some holes have been drilled on an ENE orientation and a number of holes have been drilled vertically.

A summary of the core and RC drill holes for the Twin Hills area is shown in Table 10-4. A map showing the spacing and the location of the drill holes is shown in Figure 10-1.

Table 10-4: Diamond & Reverse Circulation Drilling Summary, Twin Hills

Company	Year	Name of holes	No. of holes	Metres
Greenstone	1994	SA-056, SA-057, SA-086, SA-087, SA-088	5	527.4
	1996	SA-089, SA-090, SA-091, SA-092, SA-093, SA-094, SA-095, SA-096, SA-103, SA-111, SA-112, SA-113, SA-115, SA-117, SA-118, SA-119, SA-120, SA-121, SA-122, SA-123, SA-124, SA-125, SA-126, SA-127	24	4,036.5
	1997	SA-133, SA-134, SA-135	3	417.0
	1998	SA-274, SA-277, SA-278, SA-285, SA-302, SA-303, SA-304, SA-305, SA-306, SA-307, SA-308, SA-309, SA-310, SA-311, SA-312, SA-313, SA-325, SA-326, SA-327, SA-328, SA-329, SA-330, SA-331, SA-332, SA-334, SA-336, SA-338, SA-340, SA-341, SA-342, SA-343, SA-344, SA-345, SA-346, SA-347, SA-348, SA-349, SA-350, SA-351, SA-352, SA-353, SA-354, SA-355, SA-357, SA-358, SA-359, SA-360, SA-361, SA-362, SA-363, SA-364, SA-365, SA-366, SA-367, SA-368, SA-369, SA-370, SA-371, SA-372, SA-373, SA-374, SA-375, SA-376, SA-377, SA-378	65	7,345.4
Minosa	2002	MO-02-044, MO-02-045	2	163.0
	2005	MO-05-02, MO-05-03, MO-05-04, MO-05-05, MO-05-06, MO-05-07, MO-05-08, MO-05-14, MO-05-15, MO-05-16, MO-05-17, MO-05-18, MO-05-19	13	763.4
Minosa - Yamana	2006	MO-06-17, MO-06-19, MO-06-20, MO-06-21, MO-06-22, MO-06-23, MO-06-25, MO-06-47, MO-06-56, MO-06-58, MO-06-66, MO-06-67, MO-06-69, MO-06-70, MO-06-71, MO-06-73, MO-06-74, MO-06-76, MO-06-78, MO-06-80, MO-06-87, MO-06-89, MO-06-90, MO-06-92, MO-06-93, MO-06-94, MO-06-96, MO-06-101, MO-06-103, MO-06-104, MO-06-105, MO-06-106, MO-06-107, MO-06-108, MO-06-109, MO-06-110, MO-06-111, MO-06-112, MO-06-113	39	5,844.4
	2007	MO-07-01, MO-07-02, MO-07-03,MO-07-05, MO-07-07, MO-07-09, MO-07-11, MO-07-14, MO-07-16, MO-07-17, MO-07-18, MO-07-19, MO-07-41, MO-07-42, MO-07-43, MO-07-44, MO-07-45, MO-07-46, MO-07-47, MO-07-48, MO-07-49, MO-07-50, MO-07-51, MO-07-52, MO-07-53, MO-07-54, MO-07-55, MO-07-56, MO-07-57, MO-07-58	30	4,099.6
Minosa - Aura	2010	MO-10-01, MO-10-02, MO-10-03, MO-10-04, MO-10-05, MO-10-06, MO-10-07, MO-10-08, MO-10-09, MO-10-10, MO-10-11, MO-10-12, MO-10-13, MO-10-14, MO-10-15, MO-10-16, MO-10-17, MO-10-18, MO-10-19, MO-10-20, MO-10-21, MO-10-22, MO-10-23, MO-10-24, MO-10-25, MO-10-26, MO-10-27, MO-10-28, MO-10-29, MO-10-30, MO-10-31, MO-10-32, MO-10-33, MO-10-34, MO-10-35, MO-10-36, MO-10-37, MO-10-38, MO-10-39, MO-10-40, MO-10-41, MO-10-42, MO-10-43, MO-10-44, MO-10-45, MO-10-46, MO-10-47, MO-10-48, MO-10-49, MO-10-50, MO-10-51, MO-10-52, MO-10-53, MO-10-54, MO-10-55, MO-10-56, MO-10-57, MO-10-58, MO-10-59, MO-10-60, MO-10-61, MO-10-62, MO-10-63, MO-10-64, TH-10-01, TH-10-02, TH-10-03, TH-10-04, TH-10-05, TH-10-06, TH-10-07, TH-10-08, TH-10-09	73	3,935.0
	2012	MO-12-01, MO-12-02, MO-12-03, MO-12-04, MO-12-05, TH-12-01, TH-12-02, TH-12-03, TH-12-04, TH-12-05, TH-12-06, TH-12-07, TH-12-08, TH-12-09, TH-12-10, TH-12-11, TH-12-12, TH-12-13, TH-12-14, TH-12-15, TH-12-16	21	853.0
	2013	TH-13-01, TH-13-02, TH-13-03, TH-13-04, TH-13-05, TH-13-06, TH-13-07, TH-13-08, TH-13-09, TH-13-10, TH-13-11, TH-13-12, TH-13-13, TH-13-14, TH-13-15, TH-13-16, TH-13-17, TH-13-18, TH-13-19, TH-13-20, TH-13-22, TH-13-23	22	1,400.6
	Total RC		297	29,385.2
Greenstone	1998	SC-019, SC-028, SC-029, SC-030, SC-034, SC-036, SC-037, SC-038	8	1,126.5
Minosa - Yamana	2006	MC-06-08, MC-06-10, MC-06-11, MC-06-12	4	939.3
	2007	MC-07-10, MC-07-14, MC-07-16, MC-07-18, MC-07-21, MC-07-27	6	2,028.0
	Total Core		18	4,093.8
	Total		**315**	**33,479.0**

10.4 Cerro Cortez

A total of 15,311 m of drilling in 110 drill holes has been drilled in the Cerro Cortez area and these are detailed in Table 10-5. Of this total, 103 were RC holes (13,929 m) and seven were core holes (1,382.1 m) for a total of 9,940 assays. The deposit has also been channel sampled; however, only the drill hole assays have been used in the estimation of the Mineral Resources and Mineral Reserves.

The objective of the 1998 drilling program carried out by Greenstone at Cerro Cortez was to provide exploration information about this target. Three RC holes for a total of 585.2 m were drilled in the Cerro Cortez deposit.

Between 2002 and 2005, a new RC drilling program was carried out by Minosa in the Cerro Cortez area. This program was done to confirm the mineralization and potential of this target. A total of four holes were drilled in the area totaling 597.5 m.

Between 2006 and 2008, Yamana drilled a total of 3,451.1 m of drilling in 19 holes in the Cerro Cortez area. This data provided information to preliminary Mineral Resources estimation.

During 2012 and 2013 Minosa drilled a total of 10,680.2 m in 84 RC drill holes. The purpose of this campaign was decrease spacing between the holes and increases the level of reliability of the Mineral Resource and Mineral Reserve estimation.

Table 10-5: Reverse Circulation Drilling Summary, Cerro Cortez

Company	Year	Name of holes	No. of holes	Metres
Greenstone	1998	SA-256, SA-271, SA-273	3	582.2
Minosa	2002	MO-02-35	1	143.0
	2005	MO-05-20,MO-05-21, MO-05-22	3	454.7
Minosa - Yamana	2006	MO-06-11,MO-06-44, MO-06-46, MO-06-48, MO-06-51, MO-06-62, MO-06-64, MO-06-68, MO-06-72	9	1,527.3
	2007	MO-07-04, MO-07-06, MO-07-08	3	541.7
Minosa - Aura	2012	MO-12-06, MO-12-07, MO-12-08, MO-12-09, MO-12-10, MO-12-11, MO-12-12, MO-12-13, MO-12-14, MO-12-15, MO-12-16, MO-12-17, MO-12-18, MO-12-19, MO-12-20, MO-12-21, MO-12-22, MO-12-23, MO-12-24, MO-12-25, MO-12-26, MO-12-27, MO-12-28, MO-12-29, MO-12-30, MO-12-31, MO-12-32, MO-12-33, MO-12-34, MO-12-35, MO-12-36, MO-12-37, MO-12-38, MO-12-39, MO-12-40, MO-12-41, MO-12-42, MO-12-43, MO-12-44, MO-12-45, MO-12-46, MO-12-47, MO-12-48, MO-12-49, MO-12-50, MO-12-51, MO-12-52, MO-12-52A, MO-12-59, MO-12-61, MO-12-62	51	6,277.4
		MO-13-02, MO-13-27, MO-13-28, MO-13-29, MO-13-45, MO-13-46, MO-13-47, MO-13-48, MO-13-49, MO-13-50, MO-13-51, MO-13-52, MO-13-56, MO-13-57, MO-13-58, MO-13-59, MO-13-60, MO-13-61, MO-13-62, MO-13-63, MO-13-64, MO-13-65, MO-13-66, MO-13-67, MO-13-68, MO-13-69, MO-13-70, MO-13-71, MO-13-72, MO-13-73, MO-13-74, MO-13-75, MO-13-76	33	4,402.8
	Total RC		103	13,929.0
Minosa - Yamana	2006	MC-06-03, MC-06-07	2	445.5
	2007	MC-07-06	1	139.5
	2008	MC-08-06, MC-08-07, MC-08-08, MC-08-10	4	797.1
	Total Core		7	1,382.1
	Total		**60**	**8,872.2**

10.5 Cerro Cemetery

A total of 8,249 m of drilling in 61 drill holes has been drilled in the Cerro Cemetery area; all of them are RC holes, totaling 5,292 assays. Table 10-6 details the drilling completed in Cerro Cemetery.

The objective of the 1996-1998 drilling program carried out by Greenstone at Cerro Cemetery was to provide exploration information about this target. Eight RC holes for a total of 1,153.5 m were drilled in the Cerro Cemetery deposit.

In 2005, a new RC drilling program was carried out by Minosa in the Cerro Cemetery area. This program was done to confirm the mineralization and potential of this target. A total of 2 holes were drilled in the area totaling 269.8 m.

In 2006, Yamana drilled a total of 1,656.3 m of drilling in 9 holes in the Cerro Cemetery area. These data provide information to preliminary resources estimation.

In 2012 and 2013, Minosa, drilled a total of 4,905.8 m in 40 RC drill holes. The purpose of this campaign was decrease spacing between the holes and increases the level of reliability of the Mineral Resource and Mineral Reserve estimation.

Table 10-6: Reverse Circulation Drilling Summary, Cerro Cemetery

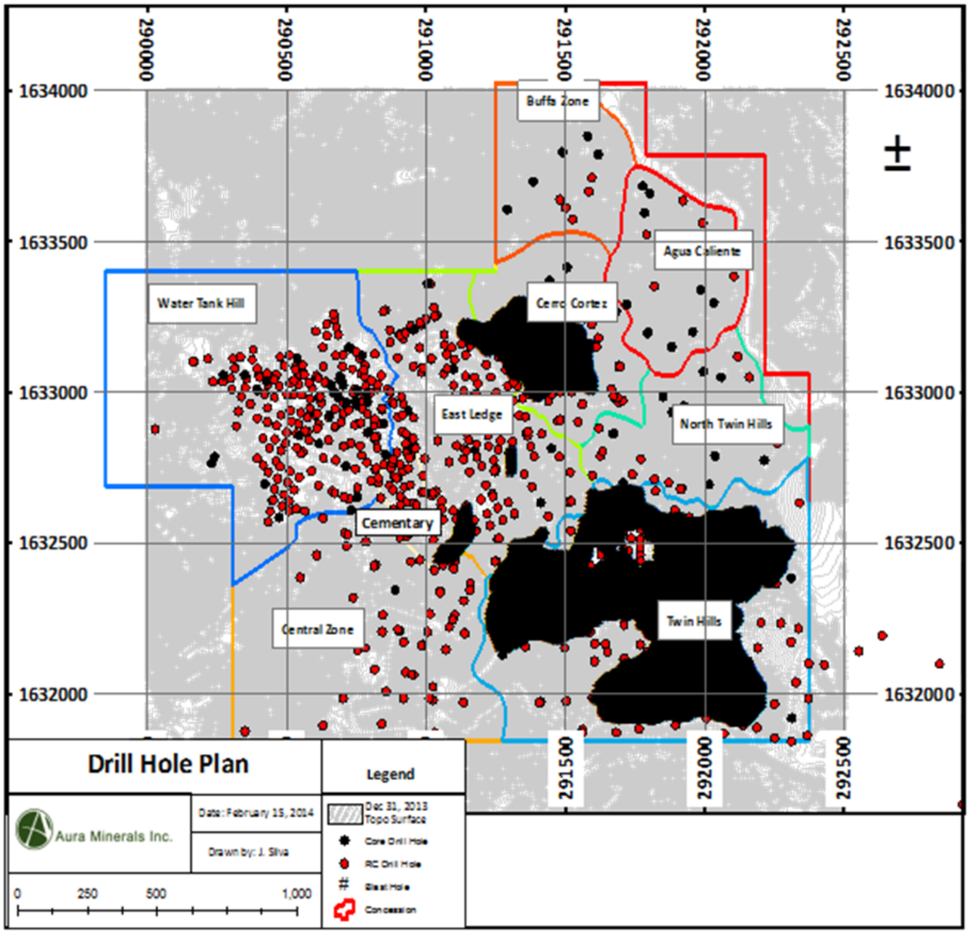
Company	Year	Name of holes	No. of holes	Metres
Fischer-Watt	1992	SA-010, SA-019	2	263.6
Greenstone	1996	SA-107	1	168.0
	1997	SA-222	1	156.0
	1998	SA-267, SA-268, SA-283, SA-284, SA-286, SA-287	6	829.5
MINOSA	2005	MO-05-12, MO-05-13	2	269.8
Yamana	2006	MO-06-01, MO-06-02, MO-06-24, MO-06-27, MO-06-30, MO-06-31, MO-06-59, MO-06-81, MO-06-86	9	1,656.3
Aura	2012	MO-12-53, MO-12-54, MO-12-55, MO-12-56, MO-12-57, MO-12-58, MO-12-60, MO-12-63, MO-12-64, MO-12-65, MO-12-66, MO-12-67, MO-12-68	13	1,737.4
	2013	MO-13-01, MO-13-03, MO-13-04, MO-13-08, MO-13-10, MO-13-15, MO-13-16, MO-13-19, MO-13-20, MO-13-24, MO-13-30, MO-13-31, MO-13-32, MO-13-33, MO-13-34, MO-13-35, MO-13-36, MO-13-37, MO-13-38, MO-13-39, MO-13-40, MO-13-41, MO-13-42, MO-13-43, MO-13-44, MO-13-54, MO-13-55	27	3,168.4
	Total RC		61	8,249.0
	Total		**61**	8,249.0

10.6 Production Drilling

A total of 165,996 production blast holes with 166,350 samples are included in the drill hole database in the Cerro Cortez, Twin Hills and East Ledge domains. Figure 10-2 outlines the location of the production drilling included into the database in relation to the deposit areas and exploration drilling.

Minosa blast hole drilling was completed using an air-track drill rig and an Atlas copco ROC L8 mobile production rig. The majority of blast holes are designed to 6.5 m depth with 6.0 m being sampled with and the 0.5 m subdrill often unsampled.

Figure 10-2: Full Database Drill Hole Location Map



10.7 Condemnation Drilling

Before Greenstone started construction on the leach pads, condemnation drilling was performed in the area around the proposed crusher facilities, waste dump, and new town site locations. Condemnation drilling totaled 4,905 m in 42 holes. With the exception of certain holes drilled immediately north of the Water Tank Hill open pit, all condemnation drilling confirmed that no significant gold existed in these areas.

In 1996, six RC holes (SA-97 to SA-102) for a total of 321 m were collared by Greenstone in the San Miguel area. This program effectively condemned the ground under the leach pads, pond and CIC-ADR plant. In the same year, to begin the process of condemning a proposed secondary crusher site south of Water Tank Hill, four RC holes (SA-109, 110, 114, 116) were drilled for a total of 515 m.

In 1997, three additional RC condemnation holes (SA-130 to 132) totalling 294 m were drilled by Greenstone at San Miguel. This was followed by an additional 13 RC holes (SA-136 to 148) totalling 1,835 m in and around the initially proposed secondary crusher site south of Water Tank Hill. One hole, SA-159 (155 m), condemned the eventual secondary crusher site. Three RC holes (SA-208 to SA-210) totalling 269 m were drilled in the Platanares area to condemn the area selected for the new town site location.

In May 2002, two condemnation holes (MO-02-41 and MO-02-42) were drilled in the area of the proposed Twin Hills waste dump.

In 2013, three condemnation holes (MO-13-45, MO-13-46 and MO-13-47) were drilled in the south of Cerro Cortez in preparation for a new waste dump area.

10.8 Drilling Types and Procedures

From 2006 to 2008, there were two RC drills and one core drill at the Mine site. The core drill and one RC drill were operated by the contractor, Swissboring, and one RC drill was operated by Minosa. A geologist and a sampling crew from Minosa were present at the RC drills at all times to log the chip samples and collect the samples for analysis and storage. Minosa employees collected the drill core and transported it to the Minosa's core logging and secure sample storage facility.

From 2010 to 2013, there were intermittent periods where up to two RC drill rigs were in use, one operated by Swissboring, and the other one operated by Minosa. A technician and sample crew were present all times to collect the samples. The geologists supervised all drilling process and the geological logging was completed at the site office.

10.8.1 Logging Procedures for RC Drilling

Logging procedures for pre-2005 RC drilling at the East Ledge and Twin Hills deposits were reported by CAM (Armbrust et al., 2005) and the logging codes are listed below. The main points are summarized briefly:

- Logging was undertaken in a well-lighted office, with large tables suitable for laying out several holes at once (full cross sections). RC drill chips were mostly logged using binocular microscopes. RC chips were logged in conjunction with nearby holes for comparative purposes.
- All previous drill holes were relogged according to the new format and the geological parameters entered into the database. Sectional interpretation was on-going throughout the logging process.
- The geological logs were designed to capture the main geologic criteria used to delineate the model zones. In order to facilitate this process, new logging sheets were created for the RC samples. A coding system was developed to expedite the entry of the various geological parameters into the computer database.
- At Twin Hills, all of the holes drilled prior to 1997 were relogged in 1997 and 1998. The objective of the relogging effort was to improve the overall consistency of the logging done during the various drilling programs.
- The geological data entered into the model was checked against the original drill logs. All lithological breaks were checked and most other geological parameters were spot checked at least every 30 m to 50 m down-the-hole.

The 2005-2008 logging procedures as reported by Scott Wilson RPA are summarized below:

- All coarse chip logging was done at the drill site at a table adjacent to the drill. The logging site was covered with a tarpaulin.
- A sieve was used to collect a sample of the coarse chips from the RC sample at the cyclone discharge. One sample of coarse material was taken for every 5 ft (1.5 m) sample.

- The coarse chips were examined by the geologist using a hand lens at the drill site. The rock type, alteration types and intensity, percentage quartz veins, oxide zone, mixed zone or sulphide zone, whether the sample was wet or dry, and structures were logged and entered into designated columns and a general description is entered into a comments column.
- The sample chips are placed in a sample tray and stored at the secure sample storage facility at the Mine site.

The 2005-2008 RC drilling procedures were the same as reported by CAM (Armbrust et al., 2005) for the pre-2005 drilling procedures, with the exception of a change in the drill hole logging glossary and code definitions shown in Table 10-4. Wet drilling conditions were not encountered during the 2005 and 2006 drilling campaigns; so only the dry sample collecting procedures were utilized.

The logging of the 2005 to 2011 RC drilling programs was completed in the field as each hole was drilled. A geologist was present to oversee that the sample procedures were carried out correctly as well as to perform geologic logging of the chips using standard hand lens and field equipment. Lithology, alteration type and intensity, percent quartz veins, percent pyrite, along with any visible structures, were logged. The oxidation state of the sample was also classified. Once the field log was completed, the chips and log were revisited in the office where chips from one hole could be compared with those of nearby holes for comparative purposes. A binocular microscope was also available for more detailed examinations.

During the 2012-2013 drilling program a geological technician was present full-time at the rig and was responsible for the supervision of the sampling and QA/QC procedures. A supervising geologist would complete regular checks of the rig, assay lab and coordinate the selection of QA/QC duplicates and repeats along with the analysing of the results. Logging of the sample chips was completed in the office.

The codes used in the logs for the rock types, alteration type and minerals, oxidation state, presence of pyrite and structure for the logging of the RC samples are shown in Table 10-5 to Table 10-13.

Table 10-7: Lithology Codes

Code	Lithology Code	Description
1	Qc	Colluvium and Overburden
2	Trv	Rhyolite tuff, ignimbrite and related clastic and pyroclastic units
3	Tcgla	Pebble conglomerate, sand and siltstones
4	Trv1	Quartz-eye ryolite tuffs to dacite, sills or lenses in Tcgla
5	Ta	Andesite porphyry, agglomerate, or tuff
6	Td	Dacite porphyry, agglomerate, or tuff
7	PM	Phylite
8	K	Valle de Angeles (Red-Beds)

Table 10-8: Colour Intensity

Code	Colour Intensity
1	Light
2	Medium
3	Dark

Table 10-9: Colour Codes

Code	Colour Intensity
1	Red
2	Orange
3	Yellow
4	Green
5	Brown
6	Purple
7	Gray
8	White
9	Black

Table 10-10: Oxide – Sulphide Oxides

Code	Oxide / Sulphide Code	Description
1	Oxidized	All Oxidized (brick red hematite and brown yellow goethite jarosite
2	Mix 1	Trace sulphides (pyrite) in ground mass, oxides dominate
3	Mix 2	Sulphides (pyrite) in ground mass and fracture coating, minor oxides
4	Sulphides	No significant amount of hematite or goethite jarosite present

Table 10-11: Alteration Intensity Codes (For Silicification, Argilization, Propylitization, Hematite and Jarosite)

Code	Alteration Intensity
1	Very Weak
2	Weak
3	Moderate
4	Strong
5	Intense

Table 10-12: Percentage Codes (For Quartz-Calcite Veining and Sulphides)

Code	Percentage
1	Traces to <1.0
2	1.0 to 5.0
3	5.0 – 10.0
4	10.0 – 25.0
5	25.0 – 50.0
6	>50.0

Table 10-13: Dry Wet Sample Codes

Code	Description
1	Dry
2	Damp, above water table
3	Damp, below water table
4	Wet, above water table
5	Wet, below water table
6	Wet, water table unknown

Table 10-14: Structure Codes

Code	Structure Code	Alteration
1	QV	Quartz vein
2	CV	Calcite vein
3	QC	Mixed Quartz / Calcite vein
4	BZ	Breccia zone
5	FZ	Fault zone
6	GZ	Gouge zone
7	WT	Water table

Table 10-15: Primary Alteration Codes

Code	Alteration
1	Propylitic
2	Agrilic
3	Silicification
4	Argilic-Silicic (Argilic dominant)
5	Silicic-Argilic (Silicic dominant)
6	Argilic-Proprolitic (Argilic dominant)
7	Proprolitic – Argilic (Proprolitic dominant)

10.8.2 Logging Procedures for Core Drilling

The drill core was emptied from the core tube and placed in a core box by the driller. The core tube is 10 ft (3.1 m) in length, but was often not filled due to blockages. Each core run was marked using wooden blocks with the metreage marked on the block. Note that the core was not oriented. The core box was covered once it was filled and the hole number and metreage were marked on the box. The core boxes were then transported to the core logging facility by Minosa employees, where it was logged in a well-lighted room. The core was logged by a geologist using the codes shown in Tables 10-4 to 10-10. Once the core was logged, sample intervals were marked on the core by a geologist based on changes in the rock type or structure and can range from 0.5 m up to 3 m in length.

10.8.3 Logging Procedures for Production Drilling

The geological logs were designed to capture the main geologic criteria used to delineate the model zones. In order to facilitate this process, new logging sheets were created for the RC chips. A coding system was developed to expedite the entry of the various geological parameters into the computer database.

The coarse chips were examined by the geologist using a hand lens at the drill site. The rock type, alteration, alteration intensity, weathering zone (oxide/mixed/sulphide), are logged and entered into designated columns. A general description is entered into a comments column.

The codes used in the logs for the rock types, alteration type and minerals, oxidation state, characterization for the logging of the BH samples are shown in Table 10-5, Table 10-8, Table 10-9 and Table 10-14.

Table 10-16: Production Alteration Codes

Code	Characterization
1	Clay
2	Clay – Rock
3	Rock
4	Rock – Clay

10.8.4 Authors' Opinion on Logging Procedures

For Mineral Resource estimation purposes, the most important parameters that require accurate identification are the contacts between, and the extent of, the oxide, mixed, and sulphide zones to ensure the gold recovery factor applied in the block model is accurate. Based on the description of the logging procedures, a review of the logs and discussions with Aura and Minosa geologists, the Authors are satisfied that these rock types can be reasonably identified by the logging procedures using the presence, relative amount, or absence of sulphide minerals and the amount of hematite (brick red) and goethite-jarosite (brown yellow).

Further to the identification of weathering profiles in the logging process, the 2012/2013 drilling program introduced the practice of analysing exploration samples using hot cyanide (HCN) analysis (partial gold analysis). The ratio between fire assay and HCN assay is used reliably within the operations grade control procedures to classify material as oxide, mixed or sulphide and the FA: AUHCN ratio was used to validate the weathering surfaces where sampling was available.

11.0 SAMPLE PREPARATION, ANALYSIS AND SECURITY

11.1 Sampling Method and Approach

The sample collection method is primarily based on RC drilling with a minor amount of core drilling for exploration holes and blast hole sampling for production holes and for each method the entire hole is logged and sampled. Although an extensive amount of channel sampling was conducted at the surface of the San Andrés deposit, this information was not used for the Mineral Resource and Mineral Reserve estimate. The pit mapping in the East Ledge and Twin Hills pits were used in the generation of the geological models and the blasthole sample assays from the production drilling at the East Ledge pit were included into the Dom 1 and 3 resource estimation.

A total of 1,013 exploration drill holes with 82,348 samples are included in the drill hole database (877 within the project limits) that was used to estimate the Mineral Resource. A total of 165,996 production blast holes with 166,350 samples are included in the drill hole database in the Cerro Cortez, Twin Hills and East Ledge domains. The samples taken from surface exposures for mapping purposes were not used in the Mineral Resource estimation.

11.1.1 Sampling Method for RC Drilling

CAM (Armbrust et al., 2005) reported the procedures for RC drilling in the technical report prepared for RNC for the drilling programs conducted prior to October 2005. Scott Wilson RPA reviewed and reported the sampling procedures used by Yamana in Yamana's 2007 technical report.

Minosa RC drilling programs have been designed to sample the entire oxide and mixed zones with a small number of holes extended into the sulphide zone. The holes were drilled from 50 m to 200 m in depth and planned to intersect the oxide–sulphide contact with several holes planned to penetrate deeper into the sulphide zone. Samples are taken continuously throughout the entire length of the hole from the collar to the end at 1.5 m (5 ft) intervals. The sampling procedures used by Yamana and Minosa for RC and core drilling are essentially the same as those described by CAM for the previous drilling. The sampling procedures are described below.

RC drill cuttings are collected from the cyclone discharge into 5-gallon plastic buckets. Each sample represents 5 ft (1.5 m), of drilling. The drill rods are in 10 ft (3 m) lengths and the rod holder is marked at the point when the rod is halfway through the run. At that point, a new sample is collected.

- The weight of the chips collected in the buckets is measured and recorded in the drill log. Sample recoveries are estimated from weight of the sample compared to the calculated weight from the volume of a 1.5 m (5 ft), sample interval.
- The sample splitter and the sample buckets are cleaned using compressed air after every sample.
- When drilling dry, the recovered sample is passed from the bucket through a Gilson splitter and reduced to two samples of about five kilograms each. Splits are retained in poly bags with a sequence number. A correlation between sample number / hole number and depth is written in the drill log.
- One sample was transported to the assay lab at the Mine site for sample preparation. The other sample remains on site in a storage facility for future reference. Every 20th sample is split for a duplicate assay check.

- When drilling wet, a rotary wet splitter was used to produce the two samples. The wet samples are passed through the Gilson splitter if further size reduction is necessary.
- All sampling was conducted by Minosa employees.
- Samples are taken through the entire drill interval.
- In addition to the duplicate samples, standards and blanks are inserted to assess for sample accuracy, contamination, and assay accuracy.

Based on the ratio of the measured weight of the sample, compared to the calculated weight, the sample recovery was estimated to be between 80% and 85%. Scott Wilson RPA reviewed 20 RC drill holes and sample recovery and noted eight samples for which no sample cuttings were recovered. On a drilled interval basis for which no sample was collected (approximately 3,000 m of drilling), the sample recovery is estimated to be greater than 99%.

The RC samples were then transported from the drill site to the San Andrés sample preparation facility by Minosa employees. The samples were then recorded in a sample book at the lab and signed by the person delivering the samples. The sample pulps have been prepared as follows:

- The sample was dried in an oven at 140°F.
- The sample was crushed to approximately minus ¼-inch by passing the sample through a small jaw crusher.
- A 300 g subsample was split off using a splitter.
- The remaining portion of the minus 10 mesh reject was bagged and saved.
- The 300 g split was pulverized on a ring-mill pulveriser. The specification for this procedure was at least 90% passing a 150 mesh screen.
- The pulverized sample (pulp) was rolled on a rolling cloth until fully homogenized and a sample was split off for fire assay.
- The splitter and the pulveriser were cleaned after each sample using compressed air.
- The pulps were packaged in plastic bags, labeled and assayed by the Minosa lab on site. The remaining sample is placed in storage on site and labelled in the event that the sample is to be used in the QA/QC process.

Prior to 2009

Historically the samples were transported from the Mine site to Minosa's Santa Rosa de Copán office via a company vehicle. From Santa Rosa de Copán the samples were shipped directly to CAS de Honduras, S. de R.L. (CAS), an assay lab in Tegucigalpa, via a courier service. This lab has since closed operations and the 2008 drilling campaign was the last to use this service.

A work request form accompanied the samples and, upon receipt of the samples by CAS, the contents of the shipment were verified with the sample numbers on the work request form. CAS then advises Minosa via email if there are any discrepancies with the shipment versus sample numbers on the work request form. Once available, the assays were sent electronically and original signed assay certificates were sent via courier to the Santa Rosa de Copán office where they were later transported for filing in the exploration files at the mine. Bulk rejects were also sent back via courier from CAS to Santa Rosa de Copán and then transported by the company for storage at the mine sample storage facility.

Minosa 2012-2013 Inspectorate Check Assay Process

The check samples requiring sample preparation were sent to Inspectorate laboratories via ADL, a transportation company, who picked up the samples on site and took them to the Inspectorate's sample preparation lab in Guatemala City. The prepared pulp samples were then shipped to Inspectorate's analysis lab in Reno. Check samples already pulverized were sent directly to Inspectorate's lab in Reno.

A work request form accompanied the samples and, upon receipt of the samples by Inspectorate, the contents of the shipment were verified with the sample numbers on the work request form. Inspectorate then advises Minosa via email if there were any discrepancies with the shipment versus sample numbers on the work request form. Once available, the assays were sent electronically and the original signed assay certificates were sent via courier to the site. Bulk rejects and pulp were also sent back via courier from Inspectorate to the site and then stored at the mine sample storage facility.

11.1.2 Sampling Method for Core Drilling

- The core tube was emptied directly into the core box by the driller. The core tube is 10 ft (3.1 m) in length but was often not filled due to blockages.
- The end of the core run was marked by wooden plates with the metreage marked on the plates.
- The core was not oriented.
- The core box was covered immediately after the box was filled and the hole number and metreage were marked on the box.
- The core boxes were transported to a core logging facility at the sample storage site by Minosa employees.
- The core was logged at the sample storage facility in a well-lighted room.
- The sample intervals were determined by a geologist, and are based on changes in rock type or structure, and range in length from 0.5 m up to 3.0 m.
- The sample intervals were then clearly marked on the core prior to splitting.
- The core has been sawn in half with a diamond saw.
- One-half of the core was placed in a plastic bag and the hole number, sample number, and depth were recorded.
- The other half of the core was kept in the core box and stored at site in a covered building for future reference.
- All sampling was conducted by company employees.
- Duplicate samples, sample blanks, and standard samples have been inserted at regular intervals as part of the QA/QC program.
- The samples of sawn core were transported to the company offices in Santa Rosa de Copán by Minosa employees and then sent via an independent courier service to CAS for sample preparation and assay.

11.1.3 Sampling Method for Production Drilling

C. Keech (Keech, 2010) produced a comprehensive technical memorandum detailing the procedures for blast hole grade control sampling at the Mine. The technical memorandum reported the sampling procedures used by Minosa in the 2012 Technical Report.

Minosa blast hole drilling was completed using an air-track drill rig and an Atlas copco ROC L8 mobile production rig. The majority of blast holes are designed to 6.5 m depth with 6.0 m being sampled with and the 0.5 m subdrill often unsampled.

The sampling procedures are described below.

- Twin samples ranging from 6 to 12 kg, were collected using pie-shaped sampling trays. The second sample provided a duplicate sample for QA/QC measures.
- This material was retained in poly bags with a sequence number. A correlation between sample number / hole number and depth is written in the drill log.
- The chips were collected with a sieve and the log is done in the field.
- The sample was transported to the assay lab at the Mine site for sample preparation.
- All sampling was conducted by Minosa employees.

In addition to the duplicate samples, standards and blanks are inserted to assess for sample accuracy, contamination, and assay accuracy.

11.2 Sample Preparation Methods for Analysis and Analytical Methods

CAM (Armbrust et al., 2005) reported that several different North American assay labs were utilized for the San Andrés samples except for the East Ledge drilling program by Minosa in 2001 and 2002 and Twin Hills and Cerro Cortez programs during 2010 and 2012, where the samples were analyzed in the Mine lab.

Fischer-Watt used American Assay Lab in Sparks, Nevada, USA during their 1992 drilling program. Greenstone started out by using Chemex Labs located in Mississauga, Ontario, Canada but switched to Barringer Assay Lab in Reno, Nevada, USA in January 1998 (starting with RC hole SA-232 and core hole SC-5).

In April 1997, a new procedure to reduce the air freight costs was initiated where all samples were submitted first to McClelland labs in Tegucigalpa, Honduras, for partial sample preparation. At McClelland, the five kilogram samples were dried, crushed to -10 mesh and an 800 g to 1,000 g subsample produced. The subsample was then forwarded to a North American assay lab for final sample preparation and assay analysis.

All samples were analyzed for gold and most samples for silver by fire assay methods with an atomic absorption spectroscopy (“AA”) finish using a 29.162 g (1 assay-ton) sample. Except for the very early work (i.e., Fischer-Watt program), metal values were reported in g/t Au. All original assay certificates are on file on site.

The sample preparation and analytical procedures at both McClelland and the North American assay labs follow industry standards. These procedures as summarized by CAM (Armbrust et al., 2005) are outlined below:

- The sample is dried in an oven at 140°F.
- The sample is crushed to approximately minus 10-mesh. The crusher yields a product where greater than 80 %of the sample passes through a 10-mesh screen.
- A 200 to 400 g sub-sample is split off using a Jones Riffle Splitter.
- The remaining portion of the minus 10-mesh reject is bagged and saved.
- The 200 to 400 g split is pulverized on a ring-mill pulveriser. The specification for this procedure is at least 90% passing a 150-mesh screen.
- The pulverized sample (pulp) is rolled on a rolling cloth until fully homogenized and a 29.166 g (1 assay-ton) sample is split off for fire-assay.

Gold analysis is done by standard fire-assay techniques. A one assay-ton sample is fused with a natural flux inquarted with four mg of gold-free silver and then cupelled. Silver beads are digested for one and one-half hours in nitric acid to remove the silver, and then three ml of hydrochloric acid is added to digest the gold into solution. The samples are cooled, made to a volume of 10 ml, homogenized and analyzed by AA for gold.

- Silver analysis is performed on a prepared sample which is digested in a hot nitric-hydrochloric acid mixture, taken to dryness, cooled and then transferred into a 250 ml volumetric flask. The final matrix is 25% hydrochloric acid. The solutions are then analyzed by AA.

CAM described the procedures for sample preparation and analyses at the Mine lab for the 2001 and 2002 drilling programs at East Ledge as follows:

- All samples from Minosa's 2001 and 2002 East Ledge drilling campaigns were assayed at the Mine lab. The Mine lab used the same procedure for sample prep and analysis, as was the practice for blasthole drill samples. All samples were analyzed for gold and most samples for silver by fire assay methods with an AA finish.

Sample preparation and analyses procedures generally follow industry standards. These procedures are outlined below:

- The sample is dried in an oven at 140°F.
- The sample is crushed to approximately minus ¼-inch by passing the sample through a small jaw crusher.
- A 50 to 60 g sub-sample is split off using a splitter.
- A remaining portion of the minus 10-mesh reject is bagged and saved.
- The 50 to 60 g split is pulverized in a ring-mill pulveriser. The specification for this procedure is at least 90 % passing a 150-mesh screen.
- The pulverized sample is rolled on a rolling cloth until fully homogenized and sample is split off fire-assay.
- Gold analysis is done by standard fire-assay techniques. The sample is fused with a natural flux inquarted with gold-free silver and then cupelled. Silver beads are digested for one and a half hours in nitric acid to remove the silver then hydrochloric acid is added to digest the gold into solution. The samples are cooled, made to a volume of 10 ml, homogenized and analyzed by AA for gold.

- Silver analysis is performed on a prepared sample which is digested in a hot nitric-hydrochloric acid mixture, taken to dryness, cooled and then transferred into a volumetric flask. The final matrix is 25 % hydrochloric acid. The solutions are then analyzed by atomic absorption spectroscopy.

11.2.1 2006 – 2008 Drilling

Beginning in February 2006, all exploration samples were sent for gold analysis to the CAS lab in Tegucigalpa. CAS is a division of Custom Analytical Services, Inc. based in Washington State. The RC samples were still prepared at the Mine lab, but the core samples are sent to CAS for sample preparation and analysis. CAS was not an accredited lab.

The sample preparation procedure at CAS was as follows:

- All samples were dried at a temperature of 60°C.
- The sample was crushed to -10 mesh and split in a Jones Riffle Splitter until a 250 g to 300 g representative sample was obtained.
- A blank sample material was regularly introduced into the preparation circuit to ensure no cross-contamination occurred.
- A representative split was pulverized with a ring and puck pulverized to 90% passing -150 mesh.
- Manual screen analysis tests were performed on sample pulps, one in every 15 samples were processed, to ensure that proper grinding was maintained.
- Pressurized air and a silica (glass) rinse were used between each sample to clean the milling rings and bowls to ensure no cross contamination between each sample occurs.
- All coarse rejects were stored indoors for a period of 30 days free of charge. All sample pulps were stored for up to sixty days at no charge.

The analytical procedures at CAS were as follows:

- All pulps were fire assayed using classical methods and an aliquot of 30 g. The resulting doré was finished through AA procedures.
- Each set of samples assayed (usually 28 in a set) includes a blank, a standard, and two random repeats. In addition, every sample finished by AA that reports greater than 1,000 ppb Au was repeated using standard fire assay methods and a gravity finish.
- Blank assays were run with each set to address contamination concerns, duplicate assays were run to address precision concerns, and a standard were run with each set to address accuracy concerns.
- All repeats, standards, and blanks are reported.

CAS emphasized that its QA/QC procedures were conducted for internal purposes. It was not intended to replace a good QA/QC program implemented by its clients. CAS advised all of its clients to include their own blanks, certified standards, and duplicate samples as an independent effort to establish acceptable confidence in the same three major points of the lab/field sampling and analysis.

In March 2006, prior to restarting the exploration drilling program following the purchase of the Mine by Minosa, reviews were carried out at the San Andrés lab and at the CAS lab in Tegucigalpa. Rod Hanson, a sampling consultant, David Turner, and Sergio Brandão Silva, Senior Geologists for Minosa reviewed procedural and equipment issues within the Mine lab. A due diligence visit was conducted at the CAS lab in Tegucigalpa on March 30, 2006. Overall, Minosa and its consultant considered the equipment and procedures at the CAS lab to be satisfactory.

11.2.2 Exploration and Production Drilling

Samples from the 2012-2013 drilling campaign were on the whole sent to the mine lab apart from 15 holes (MO-12-41 to MO-12-55), which were sent to inspectorate. The inspectorate lab utilizes similar procedures to the mine lab and is Minosa's QA/QC check assay lab.

The mine lab uses the same sample prep and analysis procedures for blasthole and exploration samples and all samples are analyzed for Gold only. Since late 2012 all samples, production and exploration utilized fire assay and hot cyanide methods with an AA finish. Exploration samples up to late 2012 were only analyzed using the fire assay method. Processing and analysis of all exploration samples were completed in batches to ensure maximum efforts were made to separate these from production samples to avoid contamination.

Sample preparation and analyses procedures follow industry standards. These procedures are outlined below:

- The sample is dried in an oven at 140°F.
- The sample is crushed to approximately minus ¼-inch by passing the sample through a small jaw crusher.
- A 300 g sub-sample is split off using a splitter.
- A remaining portion of the minus 10-mesh reject is bagged and saved by the exploration team for QA/QC purposes and external checks.
- The 300 g split is pulverized in a ring-mill pulveriser. The specification for this procedure is at least 90% passing a 150-mesh screen.
- The pulverized sample is rolled on a rolling cloth until fully homogenized and sample is split off for fire-assay.
- Gold analysis is done by standard fire-assay techniques. The sample is fused with a natural flux inquarted with gold-free silver and then cupelled. Silver beads are digested for one and a half hours in nitric acid to remove the silver then hydrochloric acid is added to digest the gold into solution. The samples are cooled, made to a volume of 10 ml, homogenized and analyzed by AA for gold.
- Hot cyanide analysis, 10 g are collected and put into vials. In this vile, 20 g of HCN (10,000 ppm) is added and put into an agitator (brand: Thermo Scientific, model: Precision), for approximately one hour. The vials are then removed from the agitator and put into a centrifuge (brand: Thermo Scientific, model: Heraeus Megafuge 16) for approximately three minutes. After this procedure, the material is analyzed by AA.

In August 2012, prior to restarting the exploration drilling program, Minosa engaged Inspectorate to complete a series of check sample analysis. Guilherme Canedo, Exploration programme manager reviewed the inspectorate procedures and certification along with the process and security of sample movement between site and Guatemala City in Guatemala, this is described in Section 11.1.1. Overall, Minosa considered the procedures and the certification at the Inspectorate laboratory to be satisfactory.

11.3 Sample Security

As noted above, the sampling has been carried out by Minosa employees and both the RC and core samples have had a secure chain of custody from the drill site to the CAS and Inspectorate assay laboratories. The drill core and RC samples continue to be stored at a secure storage facility at the Mine site. Based on a review of the security protocols and discussions with Company and MINOSA personnel, the Authors are of the opinion that the sample security procedures at the Mine are within industry standards.

11.4 Authors' Opinion on Sample Preparation, Analysis and Security

In the opinion of the Authors, the selected sample intervals are appropriate for both the RC and the core sampling and the work has been conducted using standard industry methods. The sampling has been conducted at a secure site and the samples were transported by Minosa employees to their office prior to being sent to the external lab.

Production samples are selected, sampled and transferred to the site lab in a systematic and well-coordinated process.

There is no evidence that the samples have been altered or contaminated. In the event of an independent courier service transporting the samples to external laboratories the provider was unaware of the sequence of blanks, standards and duplicates, and the sequence of their insertion is varied. San Andrés is a producing mine and overall, the gold production reconciles reasonably well with the estimates determined from the sampling data.

Overall, the Authors believe that the sample preparation, analysis and security procedures are within industry standards and that the samples are sufficiently reliable so that they can be used in the estimation of Mineral Resources and Mineral Reserves.

11.5 Comparison of RC Sampling and Core Sampling

MDA conducted a review of the RC sampling method at the Twin Hills deposit in 1999 for Greenstone. MDA compared the gold assays between a pair of twin holes, two sets of nearby core, and RC holes (20 m to 50 m apart), and core and RC drill samples from similar areas. MDA reported a poor correlation between the twinned core and the RC holes. However, the core data was not available for a 30 m interval within the mineralized zone casting doubt on the reliability of the comparison.

The comparison between the core and the RC holes for the nearby holes and holes drilled in the same area showed similar gold grades and MDA concluded that the sample methods provided similar results.

MDA did indicate that the samples from approximately 37 RC holes drilled prior to 1997 taken below the water table might have been contaminated. In an effort to retain more of the sample fines from being lost to the overflow when drilling wet, the sample collection methodology was changed in April 1997. The new method involved the collection of the wet sample into two 25-gallon drums placed in series before the 5-gallon plastic bucket. The sample slurry would cascade and collect into all three containers. Even when the volume of water was very high and overflow occurred at the last container, overall sample recovery was improved as the new method promoted the decanting and settling of the sample fines.

The assays were also reviewed for the potential of down-the-hole contamination. MDA performed an audit and each hole was checked statistically for contamination at rod changes. They also performed grade decay analysis to check for overall contamination. Individual drill chip records were also reviewed visually for potential contamination. The statistical analysis of the assays and the visual review of the RC drill chips resulted in no clear indication of any problems with contamination.

As the oxide zone at the Twin Hills deposit is shallow and lies above the water table and given the small number of holes and the restriction of the sampling problem to only a small number of wet samples, and that no clear indication of contamination was noted by MDA. In the Authors' opinion, the potential for sample contamination is not considered to materially affect the estimation of Mineral Resources and Mineral Reserves.

11.6 Pre-2006 Drilling QA/QC Programs

Fisher-Watt submitted samples for assay to the American Assay Lab of Sparks, Nevada. Details of any QA/QC programs are unknown. Greenstone submitted samples for assay to Chemex Lab of Mississauga, Ontario during 1994 to 1997, but later changed to Barringer Assay Lab of Reno, Nevada for the samples collected in 1998. From 2001 to 2005 Minosa commonly used the Mine assay lab, with some check samples sent to CAS of Tegucigalpa, Honduras.

11.6.1 East Ledge

Two separate check assay programs were run on the RC drill samples in 2002. The first program was performed by sending a sample split to the lab at the same time as the primary sample. This was performed for all 47 drill holes of the 2002 drill campaign.

The second program was initiated after the drilling of the first 14 holes had been completed and the sample assay results from these holes were known. It was noticed that the check assay results for samples submitted at the same time as the primary sample showed good correlation whereas check samples submitted at a later date had poorer correlation. As such, starting on the 15th hole of the 2002 drill campaign, a second duplicate sample was submitted at least two days after the primary sample and the first duplicate were submitted.

CAM (Armbrust et al., 2005) reviewed the QA/QC data on the drilling at East Ledge and concluded, after completing a variety of statistical checks and checks of data entry, that the exploration database had been prepared to industry standards and was suitable for the development of geological and grade models.

11.6.2 Twin Hills

Check assay programs on the drill data were reported by CAM (Armbrust et al., 2005) to have been conducted in four phases. A check-assay program was run on the RC drill samples in 1995. The program occurred while drilling was in progress and consisted of one random 1.5 m duplicate sample taken approximately every 100 m and sent to either American or Chemex Labs for assay. Duplicate assays correlate very well with the original assays.

A second check assay program for Twin Hills was instituted in early 1998 on 1,544 duplicate samples from 136 drill holes (SA-149 through SA-285). Samples were taken from Greenstone's 1997-1998 RC drill programs and included holes collared at Twin Hills and other nearby prospects. While drilling, a random, duplicate sample was taken every six to ten metres (15% of all samples) and sent for gold and silver analysis. Statistical analysis of the check assay results was performed by MDA. The results show that the correlation between the original assays and the duplicates taken at the drill are excellent. The correlation between the original Au and the duplicate Au was 0.96 and the correlation between original Ag and duplicate Ag was 0.94. A summary of the results is shown in Table 11-1.

Table 11-1: Check Assay Program #2 – Summary of Results (MDA, April 1999 from CAM, 2005)

Population / Element	Valid N	Mean (g/t)	Min. (g/t)	Max. (g/t)	Std. Dev.	Skew	Kurtosis	CV*
Original-Au	1,544	0.44	0.00	21.84	1.26	9.44	118.65	2.85
Duplicate-Au	1,544	0.44	0.00	23.76	1.28	9.75	128.86	2.92
Original-Ag	1,544	2.28	0.00	91.00	5.57	7.49	79.30	2.44
Duplicate-Ag	1,544	2.19	0.00	68.00	5.24	7.03	64.49	2.39

*CV=Coefficient of Variation=Standard Deviation/Mean.

As part of a fourth check assay program, 86 pulp samples from Chemex were sent to Barringer, and 92 pulps were sent from Barringer to Chemex. The procedure was implemented as a means of checking the variability between labs. Also, 118 coarse rejects were sent from McClelland labs in Tegucigalpa to CAS labs in Tegucigalpa, and assayed by both Barringer and CAS to check the sample preparation procedures at McClelland.

Results of these showed a good correlation coefficient for gold (r = 0.950 - 0.997) between labs and confirmed that the assay results were reproducible within industry standards. Metallic screen assays were done on 47 samples to check for coarse gold. Although an average 4% of the total gold was in the plus 150-mesh portion of the sample, MDA concluded that this did not present a problem in assay reproducibility.

11.7 2006-2008 QA/QC Program

The QA/QC program completed by Minosa - Yamana between 2006 and 2008 is extensively documented by Reed et al, 2012 Technical Report and should be used for a more detailed reference. Minosa - Yamana utilized the following processes during this period:

- Standard Reference Material, six certified standard reference materials (“SRMs”) ranging from 0.33 g/t Au to 6.83 g/t Au.
- Check Assay program, where every 10th sample was submitted to both Minosa (the Mine lab) and ACME Analytical Laboratories (Vancouver) Ltd. (“ACME”).
- Blank Samples, where blanks were inserted at regular intervals throughout the sample stream.
- Duplicate samples, where Minosa collected two samples from the RC cuttings. Sample “A” is sent for analysis and sample “B” is stored. A duplicate sample was collected from sample B every tenth sample and submitted for assay.

11.7.1 Authors’ Opinion on 2006-2008 QA/QC Program Results

A statistical review of the QA/QC data for the 2006-2008 period indicates that these assay results and drill hole database are sufficiently reliable and can be used for Mineral Resource and Mineral Reserve estimation.

11.8 2012 QA/QC Exploration Program

11.8.1 Standard Reference Material

For the 2012 drilling program completed by Minosa, six certified SRMs were inserted into the sample stream at a rate of 7%. The SRMs were purchased from Geostats Pty Ltd, Australia. The SRMs range in grade from 0.32 g/t Au to 1.41 g/t Au and a further three standards were used by the lab (STD 1 to 3). The SRM's used in the 2012 drilling campaign are detailed in the table 11-2 and table 11-3 below.

Table 11-2: Check Standard Reference Material used in 2012 drilling campaign

Standard	Name	Standard Value	Standard Deviation	Min. Acceptable	Max. Acceptable
G300-7	Au_FA_ppm	1.00	0.04	0.92	1.08
G305-2	Au_FA_ppm	0.32	0.02	0.28	0.36
G306-1	Au_FA_ppm	0.41	0.03	0.35	0.47
G902-7	Au_FA_ppm	1.41	0.10	1.21	1.61
G999-2	Au_FA_ppm	0.63	0.06	0.51	0.75
G910-10	Au_FA_ppm	0.97	0.05	0.87	1.07
STD-1	Au_FA_ppm	0.32	0.02	0.28	0.36
STD-2	Au_FA_ppm	0.52	0.05	0.42	0.62
STD-3	Au_FA_ppm	0.80	0.07	0.66	0.94

Table 11-3: Check Standard Reference Material results for 2012 drilling campaign

Standard	Name	Standard Value	Number Standards Submitted	Mean	Min	Max	% Variance to Std Mean
G305-2	Au_FA_ppm	0.32	90	0.30	0.28	0.35	-5.70
G306-1	Au_FA_ppm	0.41	2	0.43	0.39	0.47	4.88
G999-2	Au_FA_ppm	0.63	129	0.59	0.51	0.72	-6.37
G910-10	Au_FA_ppm	0.97	8	0.98	0.96	0.99	0.63
G300-7	Au_FA_ppm	1	101	0.99	0.94	1.07	-0.95
G902-7	Au_FA_ppm	1.41	2	1.28	1.24	1.32	-9.22
STD-1	Au_FA_ppm	0.32			0.28	0.36	
STD-2	Au_FA_ppm	0.52			0.42	0.62	
STD-3	Au_FA_ppm	0.8			0.66	0.94	

Figures 11-1 to 11-3 show graphs of the SRM results for Minosa lab assays compared to the expected value of the SRM. The standards G306-1, G910-10 and G902-7 have a limited number of samples therefore are not a representative sample however all samples submitted returned within 2 standard deviations.

An upper and lower detection limits are shown as red lines and the average of the results is shown as the dark black line. In general, the SRM's submitted through the Minosa lab show that the assay results are reliable to be used for Mineral Resource and Mineral Reserve estimation. There is however, a small bias towards underestimation of the SRM grade across all but one standard with mean assay values varying between +4.9% to -9.2% as detailed in the table 11-3.

Figure 11-1: QA/QC Program – Standard G300-7, 2012

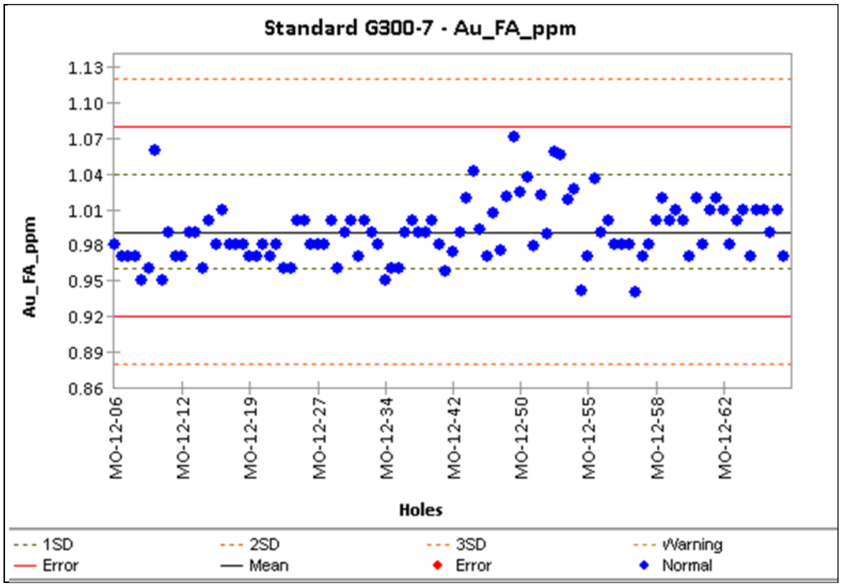


Figure 11-2: QA/QC Program – Standard G305-2, 2012

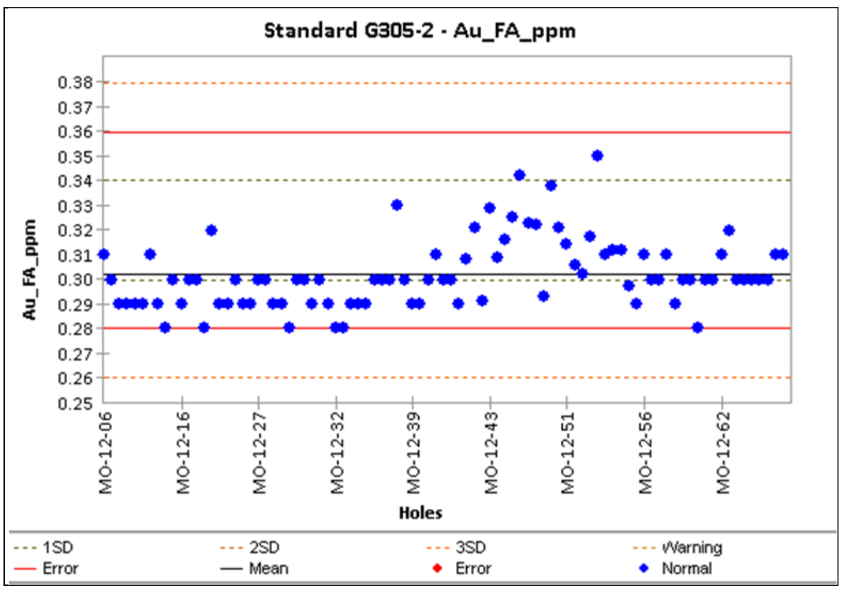
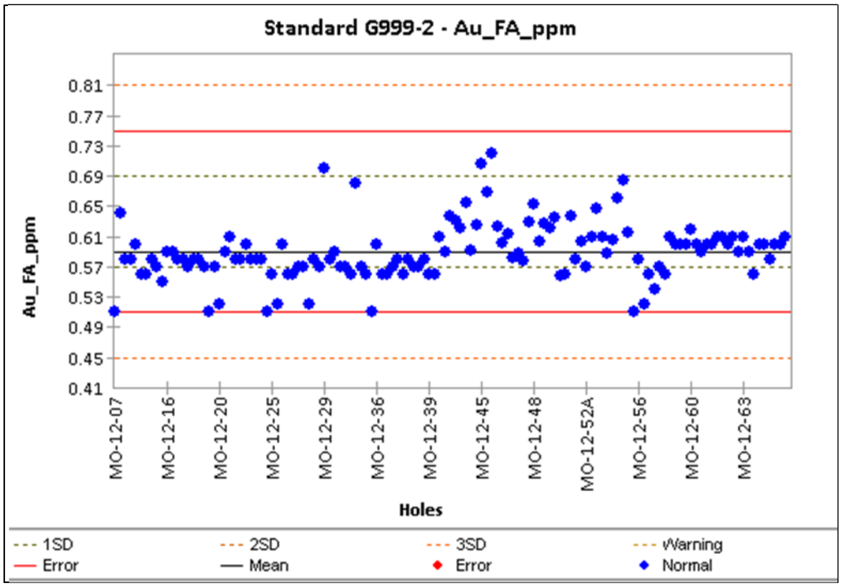


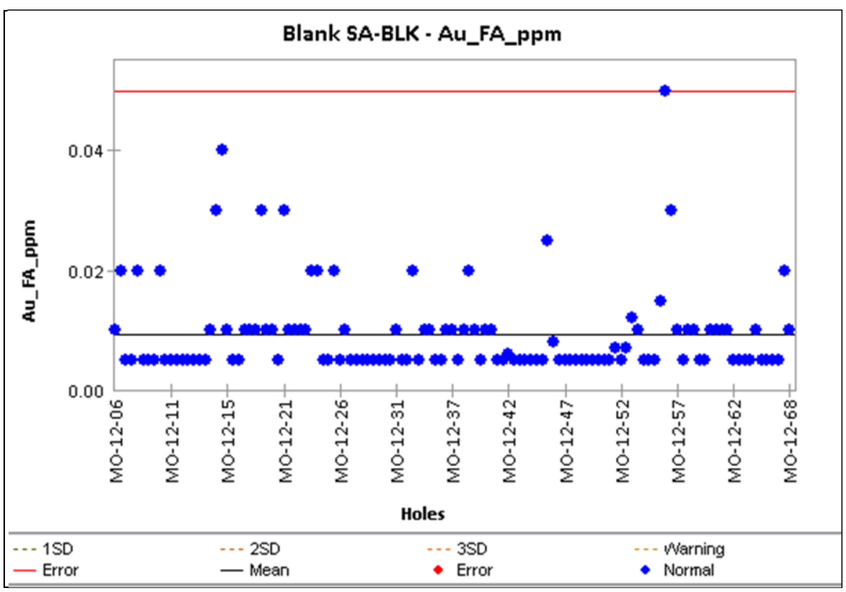
Figure 11-3: QA/QC Program – Standard G999-2, 2012



11.8.2 Blank Samples

For the 2012 drilling campaign, a total of 121 blanks (samples with a grade of 0.0 g/t Au) were sent to the lab to test for potential contamination issues. Minosa inserted two blank samples per hole, placed within the mineralized zones as defined by the interpreted mineralisation model. The results are shown in Figure 11-4. All results of the assays of blank material are acceptable and confirm that there was likely minimal cross contamination between samples. The line in red is the limit for blank values.

Figure 11-4: QA/QC Program –Blanks to 2012



11.8.3 Duplicate Samples

Minosa collected two samples during the RC drilling, sample A and B as illustrated in Figure 11-5 below. Sample A was sent for analysis at the Minosa site lab and sample B is stored in a secure storage area for the purpose of duplicate assay checks.

Three different types of duplicates were analyzed in 2012 drilling campaign: field duplicate (sample B), reject duplicate (coarse reject of sample A) and pulp duplicate (pulp reject of sample A). A total of 6% of all samples were tested by pulp duplicates, four holes were tested by field duplicate and three by reject duplicates and in the case for the field and reject duplicates only the mineralized zone was tested.

Results of the duplicate samples are illustrated in Figures 11-6 to 11-8 where the samples had a good correlation coefficient (>0.88 for field duplicates; 0.97 for course rejects; 0.93 for pulp duplicates). The duplicate data points are illustrated as blue, red and green the graphs with the following categories:

- green points indicate samples below the threshold
- red points indicate samples above the threshold
- blue points indicate samples within threshold.

The proportion of duplicates above or below 20% variation remains relatively high; between 20 and 30% however this is not unexpected in a gold deposit. The proportion of duplicates outside of 20% variation is:

- 16% of the pulp duplicates
- 29% of the field duplicates
- 13% of coarse reject duplicate.

Figure 11-5: Photograph – Field Duplicates, 2012

Figure 11-6: QA/QC Program – Field Duplicates, 2012

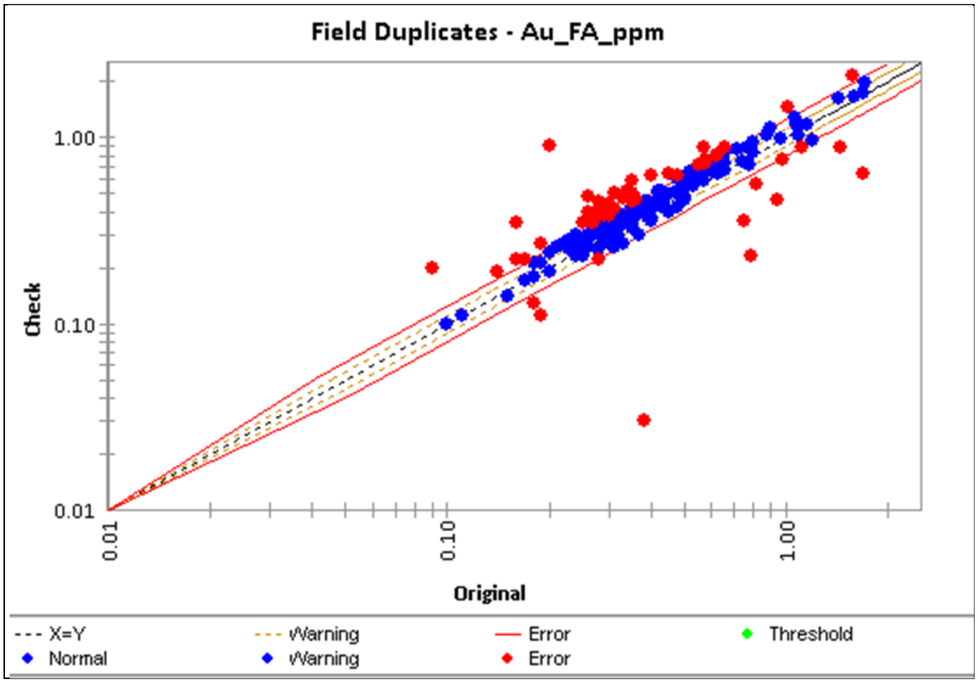


Figure 11-7: QA/QC Program – Pulp Duplicates, 2012

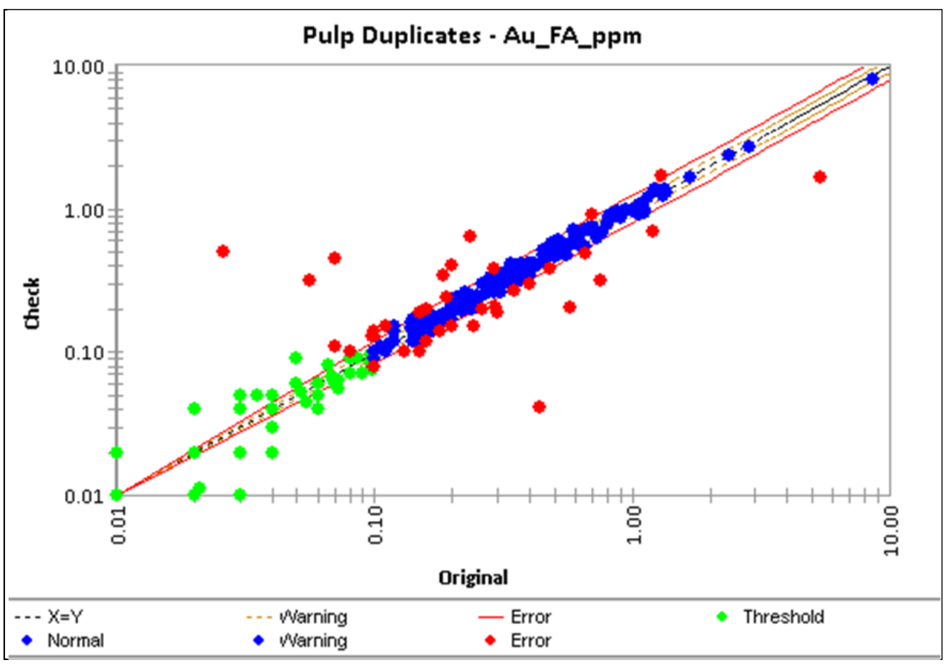
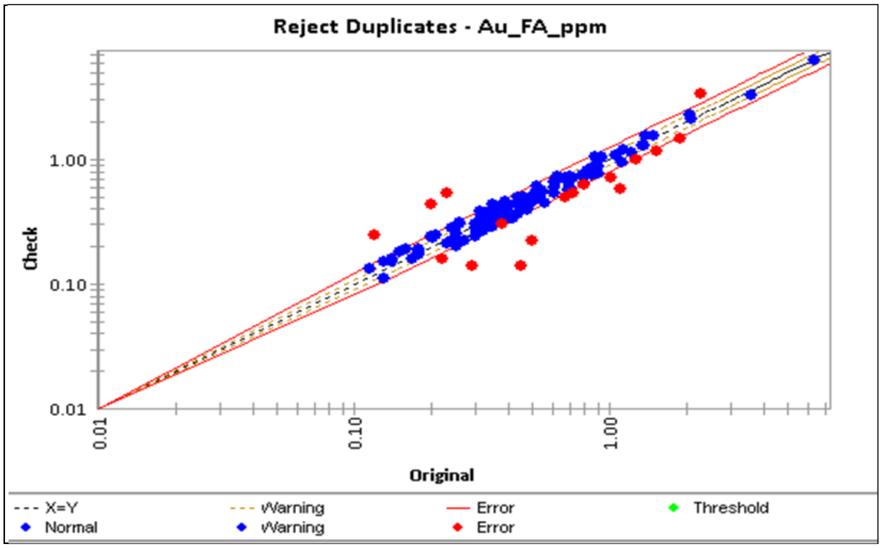


Figure 11-8: QA/QC Program – Reject Duplicates, 2012



11.8.4 Check Samples

The check samples were sent to Inspectorate and like the duplicates samples analysed at the Minosa lab, three different check duplicates were analyzed in 2012 drilling campaign. Duplicates analysed included; field duplicate (sample "B"), reject duplicate (coarse reject of sample A) and pulp duplicate (pulp reject of sample A). The results from Inspectorate are consistent with that of the Minosa lab, providing a reliable independent check on the sample preparation methods and analysis results from the site lab.

Pulp Duplicate

A total of 241 samples were tested by pulp duplicates. The result of this analyse indicate a very good correlation between Minosa and Inspectorate assays (corr. coeff. of 0.98). The major part of the samples is in the range of 10% of variance. Figure 11-9 bellow illustrated this correlation and Table 11-4 summarizes the drill holes that were sampled for pulp duplicate.

Figure 11-9: Inspectorate Check Samples – Pulp Duplicates, 2012

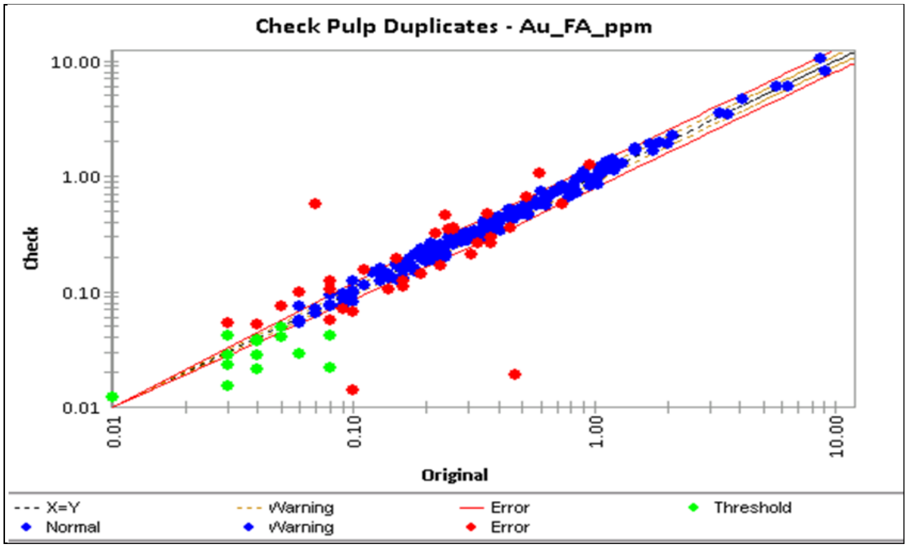


Table 11-4: Inspectorate Check Samples – Pulp Duplicate Drill holes

Drill hole	Sample	Drill hole	Sample	Drill hole	Sample	Drill hole	Sample
MO-12-06	13	MO-12-19	5	MO-12-32	2	MO-12-59	4
MO-12-07	11	MO-12-20	3	MO-12-33	3	MO-12-60	6
MO-12-08	6	MO-12-21	4	MO-12-34	4	MO-12-61	4
MO-12-09	8	MO-12-22	4	MO-12-35	6	MO-12-62	5
MO-12-10	9	MO-12-23	6	MO-12-36	5	MO-12-63	3
MO-12-11	11	MO-12-24	3	MO-12-37	3	MO-12-64	2
MO-12-12	11	MO-12-25	2	MO-12-38	2	MO-12-65	2
MO-12-13	5	MO-12-26	4	MO-12-39	5	MO-12-66	2
MO-12-14	6	MO-12-27	3	MO-12-40	3	MO-12-67	2
MO-12-15	4	MO-12-28	6	MO-12-41	3	MO-12-68	8
MO-12-16	5	MO-12-29	4	MO-12-56	8		
MO-12-17	4	MO-12-30	2	MO-12-57	5		
MO-12-18	5	MO-12-31	4	MO-12-58	6		

Coarse Reject Duplicate

A total of 111 coarse reject duplicate samples were tested by Inspectorate. The correlation between Minosa and Inspectorate assays is in line with the duplicate samples with 21% of the data outside of the 2 standard deviations. The correlation coeff of 0.67 is significantly lower due to a larger variation in the samples outside of the thresholds however, with nearly 80% of the Inspectorate pulp duplicate data within 2 standard deviations of the Minosa lab results the Minosa data is considered to be reliable. Figure 11-10 illustrates the correlation between Inspectorate and Minosa coarse rejects and Table 11-5 list the drill holes sampled for coarse reject duplicates.

Figure 11-10: Inspectorate Check Samples – Coarse Reject Duplicates, 2012

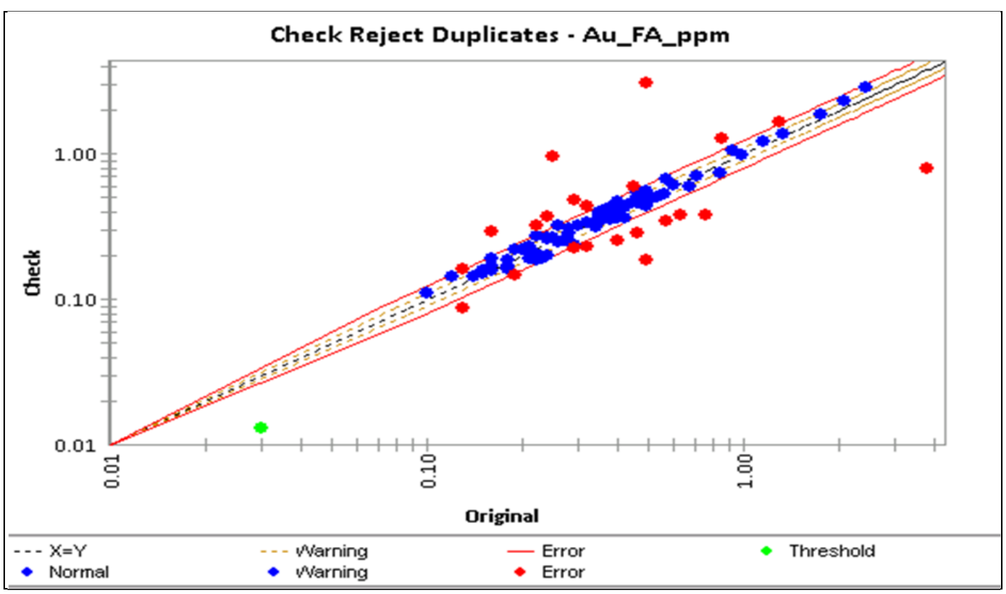


Table 11-5: Inspectorate Check Samples – Coarse Reject Drill holes

Drill hole	Sample
MO-12-20	65
MO-12-26	14
MO-12-28	11
MO-12-37	21

Sample B Duplicate

A total of 110 sample B duplicate samples were tested by Inspectorate. The correlation between Minosa and Inspectorate assays is very good, like in the pulp duplicates (corr. coeff. of 0.90). Just 10% of the samples fall outside of 2 standard deviations and are very close to the limits. Figure 11-11 illustrates the correlation between Inspectorate and Minosa sample B duplicates and Table 11-6 list the drill holes sampled for sample B duplicates.

Figure 11-11: Inspectorate Check Samples – Sample B Duplicates, 2012

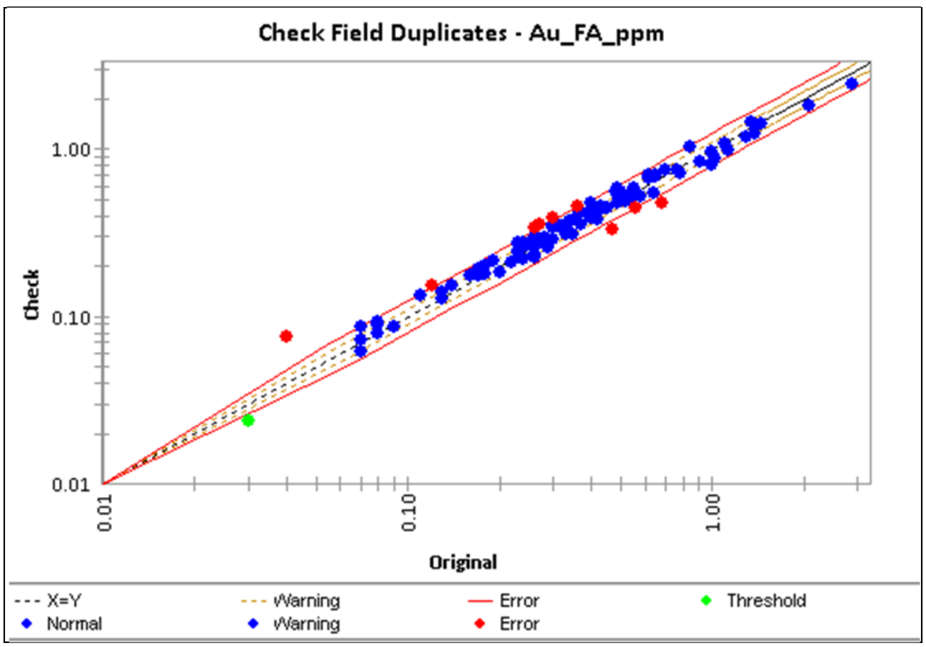


Table 11-6: Inspectorate Check Samples – Sample B Drill holes

Drill hole	Sample
MO-12-17	53
MO-12-29	46
MO-12-36	11

15 drill holes were analyzed at Inspectorate (Sample A) and a total of 33 Sample B duplicate samples were sent and tested at the Minosa mine lab. The correlation between Minosa and Inspectorate assays is the same of the check assays done in Inspectorate (corr. coeff. of 0.90). 24% of the samples fall

outside of 2 standard deviations and are very close to the limits. Figure 11-12 illustrates the correlation between Inspectorate and Minosa duplicates and Table 11-7 list the drill holes sampled for duplicates.

Figure 11-12: Minosa Check Samples – Sample B Duplicates, 2012

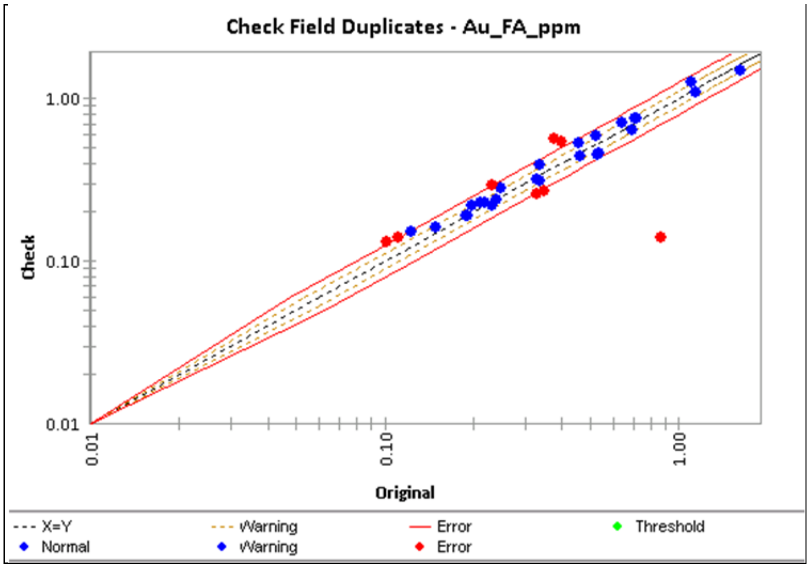


Table 11-7: Minosa Check Samples – Sample B Drill holes

Drill hole	Sample
MO-12-49	33

Standard Samples

A total of 31 standard samples were sent to Inspectorate and all of them returned within acceptable limits. The standards used are the same as those sent to the Minosa site lab and detailed in Table 11-8. Standard G300-7 had a mean of 1.03 g/t and a BIAS of 3.65%. Standard G999-2 had a mean of 0.62 with a BIAS of -1.56%. Figures 11-13, 11-14 and 11-15 illustrate the results and it is the Author's view that these assay results are reliable to be used for Mineral Resource and Mineral Reserve estimation.

Table 11-8: Check Standard Reference Material results for 2012 drilling campaign

Standard	Name	Standard Value	Number Standards Submitted	Mean	Min	Max	% Variance to Std Mean
G300-7	Au_FA_ppm	1.00	6	1.04	1.01	1.06	3.65
G999-2	Au_FA_ppm	0.63	8	0.62	0.56	0.66	-1.57
G305-2	Au_FA_ppm	0.32	4	0.31	0.28	0.33	-1.72
G311-1	Au_FA_ppm	0.52	5	0.50	0.49	0.51	-3.81
G910-10	Au_FA_ppm	0.97	6	0.95	0.90	1.04	-1.94
G998-6	Au_FA_ppm	0.80	2	0.83	0.83	0.84	4.13

Figure 11-13: Standard G300-7 sent to Inspectorate

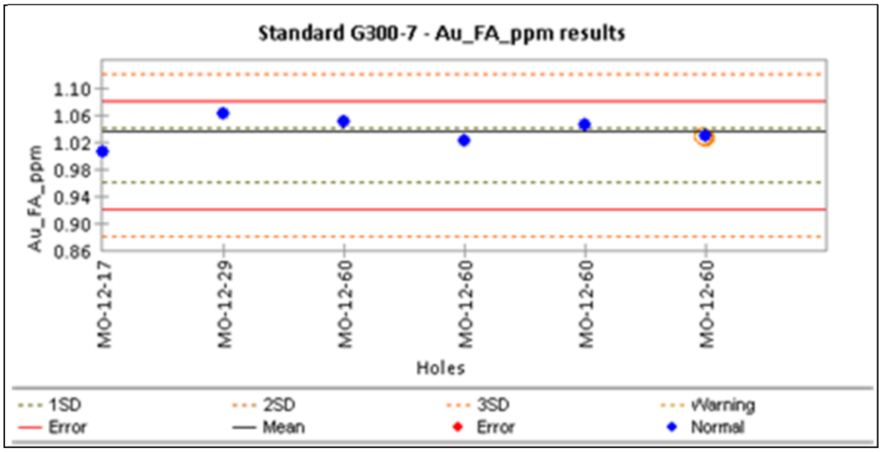


Figure 11-14: Standard G999-2 sent to Inspectorate

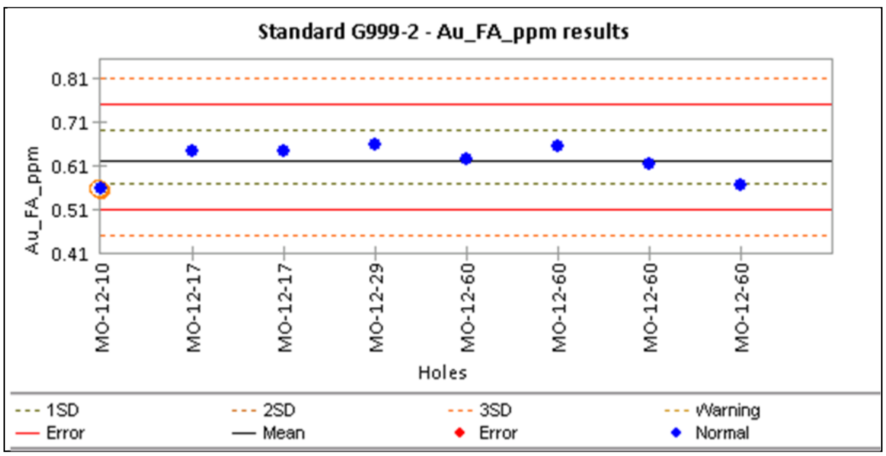
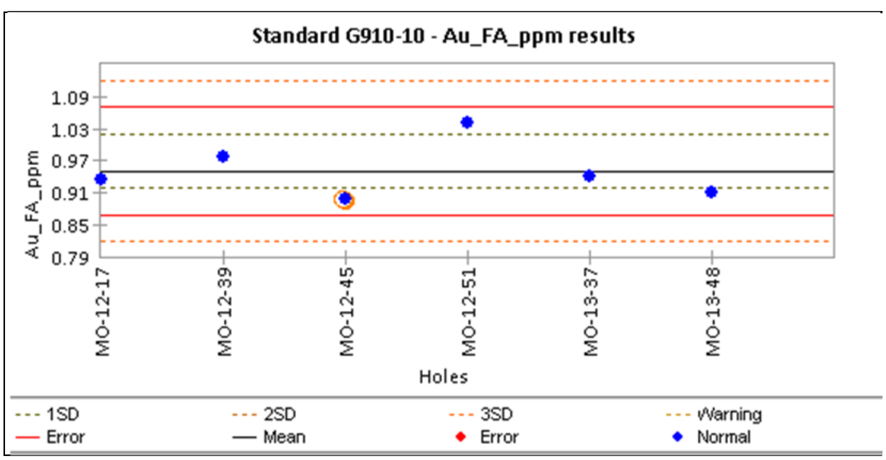


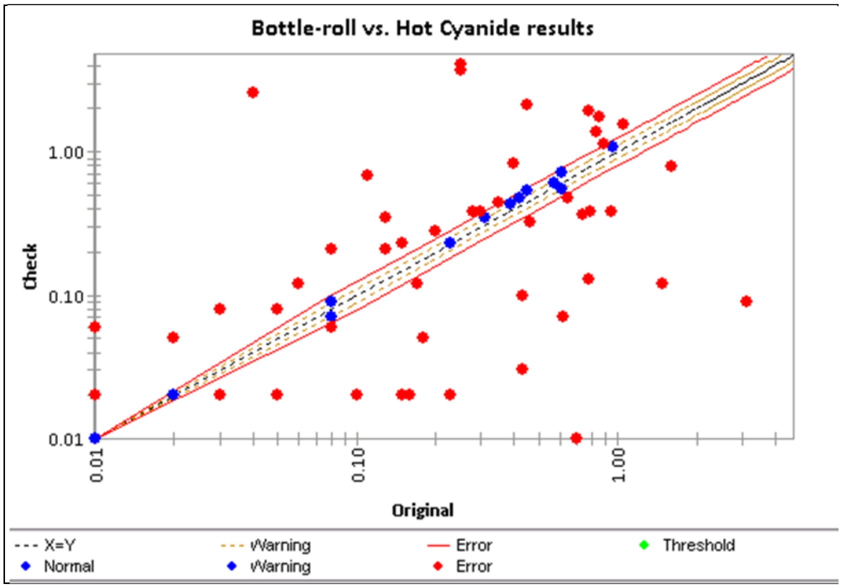
Figure 11-15: Standard G910-10 sent to Inspectorate



11.8.5 Hot Cyanide Assay comparisons

A bottle roll analyses was completed to the check hot cyanide (AUHCN) results used to assist in determining the oxide, mixed ore and sulphide boundaries. Seventy-one samples from 27 holes were analysed in Minosa's lab with 78% of the samples have more than 20% variance. Figure 11.16 below illustrates these results. However it is the Author's assessment that the results of the AUHCN analysis is sufficiently reliable to be used as a guide along with the geological description for the delineation of cyanide recoverable Au (Oxide / Mixed / Sulphide contacts).

Figure 11-16: Bottle-roll vs. Hot Cyanide analysis



11.9 2013 QA/QC Exploration Program

11.9.1 Standard Reference Material

For the current drilling programme commenced in 2013 by Minosa five certified standard reference materials (SRMs) were inserted into the sample stream at a rate of 7%. The SRMs were purchased from Geostats Pty Ltd, Australia. The SRMs range in grade from 0.32 g/t Au to 1.00 g/t Au a further two standards were used by the lab (STD 1 to 2). The SRM's used in the 2013 drilling campaign are detailed in the table 11-9 and table 11-10 below.

Table 11-9: Check Standard Reference Material used in 2013 drilling campaign

Standard	Name	Standard Value	Standard Deviation	Min Acceptable	Max Acceptable
G300-7	Au_FA_ppm	1.00	0.04	0.92	1.08
G305-2	Au_FA_ppm	0.32	0.02	0.28	0.36
G311-1	Au_FA_ppm	0.52	0.04	0.44	0.60
G910-10	Au_FA_ppm	0.97	0.05	0.87	1.07
G999-2	Au_FA_ppm	0.63	0.06	0.51	0.75
STD-1	Au_FA_ppm	0.32	0.02	0.28	0.36
STD-2	Au_FA_ppm	0.52	0.05	0.42	0.62

Table 11-10: Check Standard Reference Material results for 2013 drilling campaign

Standard	Name	Standard Value	Number Standards Submitted	Mean	Min	Max	% Variance to Std Mean
G300-7	Au_FA_ppm	1.00	81	0.99	0.96	1.03	-1.01
G305-2	Au_FA_ppm	0.32	118	0.30	0.28	0.32	-6.41
G311-1	Au_FA_ppm	0.52	30	0.50	0.47	0.52	-3.53
G910-10	Au_FA_ppm	0.97	39	0.99	0.96	1.02	1.64
G999-2	Au_FA_ppm	0.63	132	0.59	0.57	0.62	-6.35
STD-1	Au_FA_ppm	0.32					
STD-2	Au_FA_ppm	0.52					

Figures 11-17 to 11-19 show graphs for the SRM results for Minosa lab assays compared to the expected value for the SRM. All samples submitted returned within 2 standard deviations.

The upper and lower detection limits are shown as red lines and the average of the results is shown as the dark black line. In general, the SRM's submitted through the Minosa lab show that the assay results are reliable to be sued for Mineral Resource and Mineral Reserve estimation. There is however, a small bias towards underestimation of the SRM grade across all but one standard with mean assays values varying between +1.6% to -6.4% as detailed in Table 11-10.

Figure 11-17: QA/QC Program – Standard G300-7, 2013

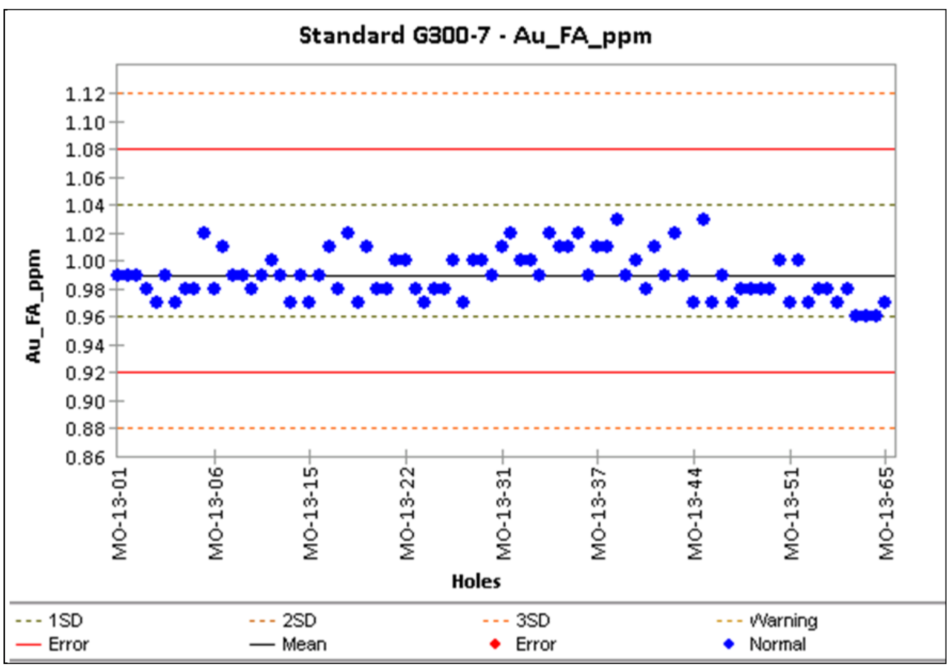


Figure 11-18: QA/QC Program – Standard G305-2, 2013

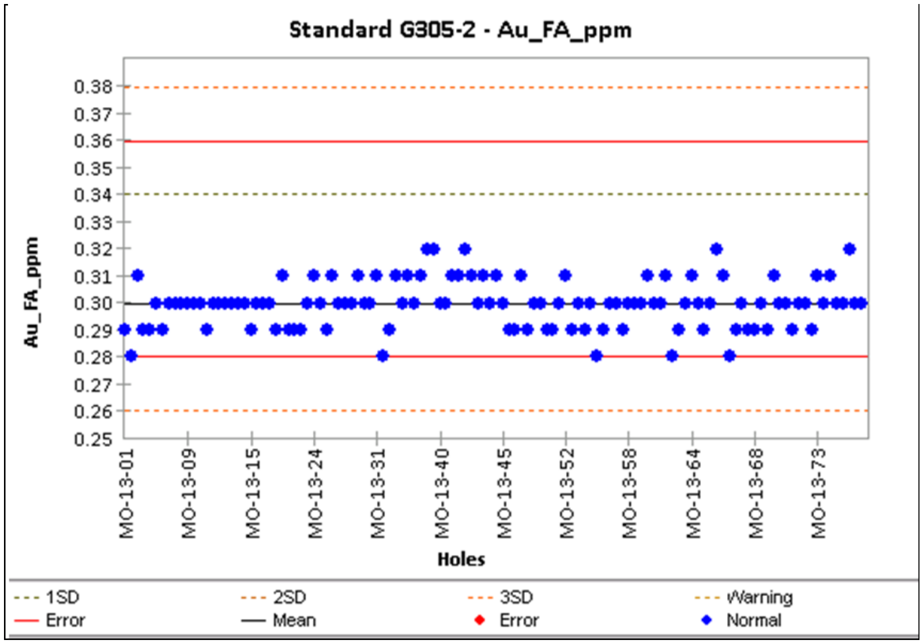
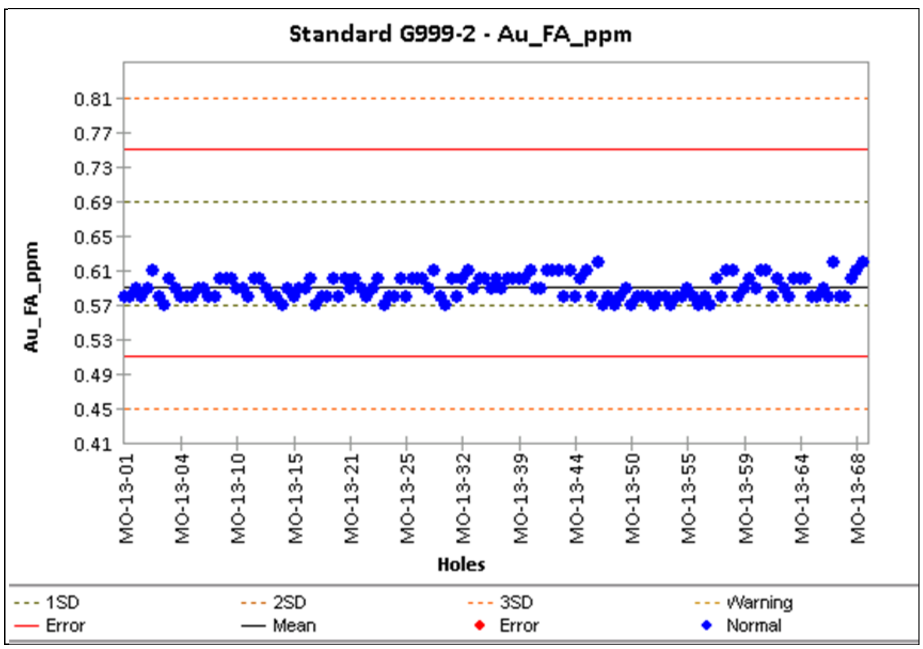


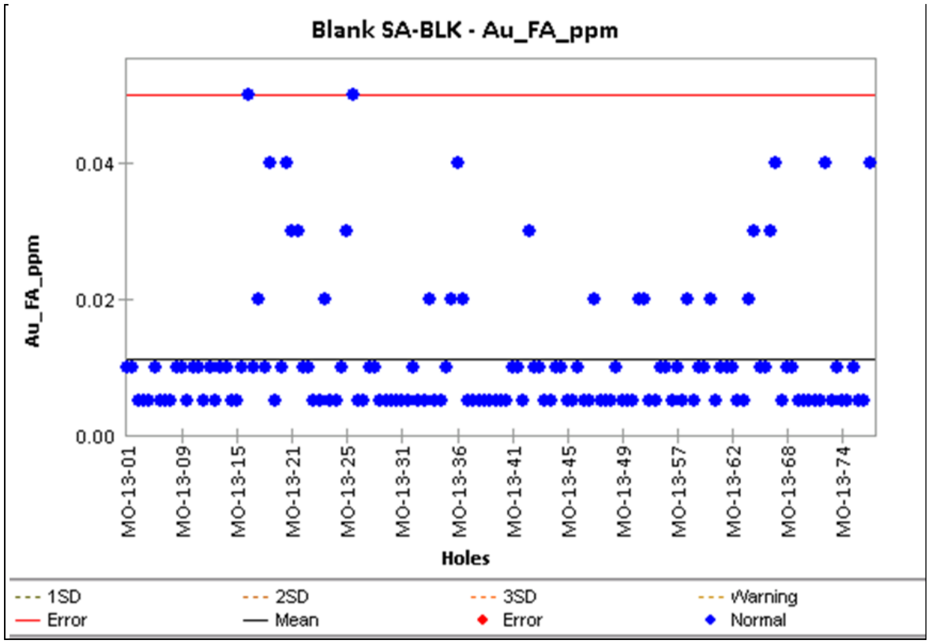
Figure 11-19: QA/QC Program – Standard G999-2 2013



11.9.2 Blank Samples

For the 2013 drilling campaign, a total of 136 blanks (samples with a grade of 0.0 g/t Au) were sent to the laboratory to test for potential contamination issues. Minosa inserted two blank samples per hole, placed within the mineralized zones as defined by the interpreted mineralisation model. The results are shown in Figure 11-20. All results of the assays of blank material are acceptable and confirm that there was likely minimal cross contamination between samples. The line in red is the limit for blank values.

Figure 11-20: QA/QC Program – Blank Samples 2013



11.9.3 Duplicate Samples

Minosa used the same procedure of 2012 campaign where collected two samples during the RC drilling, sample A and B. Sample "A" was sent for analysis at the Minosa site laboratory and sample "B" is stored in a secure storage area for the purpose of duplicate assay checks.

Three different types of duplicates were analyzed in 2013 drilling campaign: field duplicate (sample B), reject duplicate (coarse reject of sample A) and pulp duplicate (pulp reject of sample A). A total of 5% of all samples were tested by pulp duplicates, three holes were tested by field duplicate, other three by reject duplicates and in the case for the field and reject duplicates only the mineralized zone was tested.

Results of the duplicate samples are illustrated in Figures 11-21 to 11-23 where the samples had a good correlation coefficient (>0.95). The duplicate data points are illustrated as blue, red and green the graphs with the following categories:

- green points indicate samples below the threshold
- red points indicate samples above the threshold
- blue points indicate samples within threshold.

The proportion of duplicates above or below 2 standard deviations remains relatively high; between 20 and 40% however this is not unexpected in a gold deposit. The proportion of duplicates outside of 2 standard deviations is:

- 23% of the pulp duplicates
- 40% of the field duplicates
- 27% of coarse reject duplicate.

Figure 11-21: QA/QC Program – Field Duplicates 2013

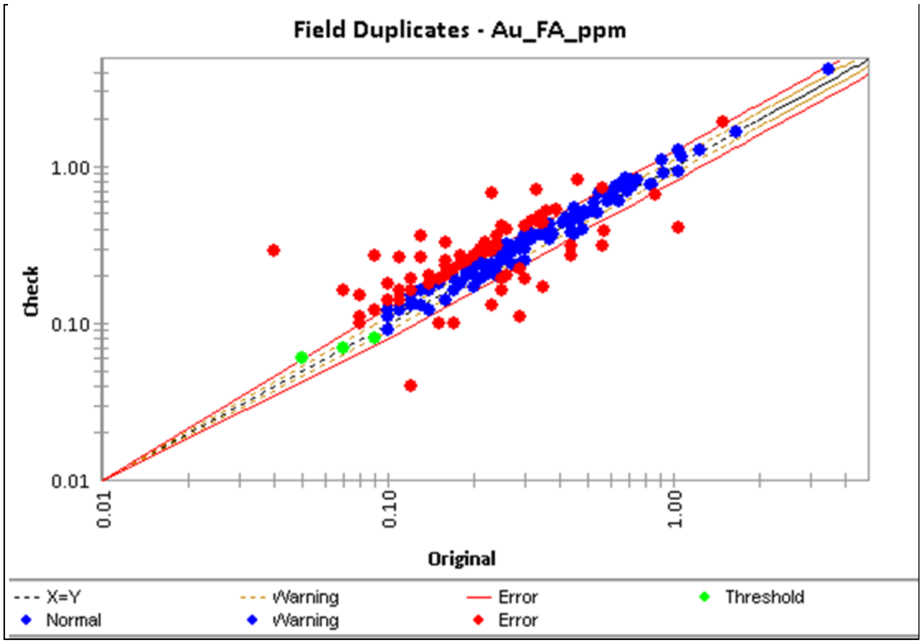


Figure 11-22: QA/QC Program – Pulp Duplicates 2013

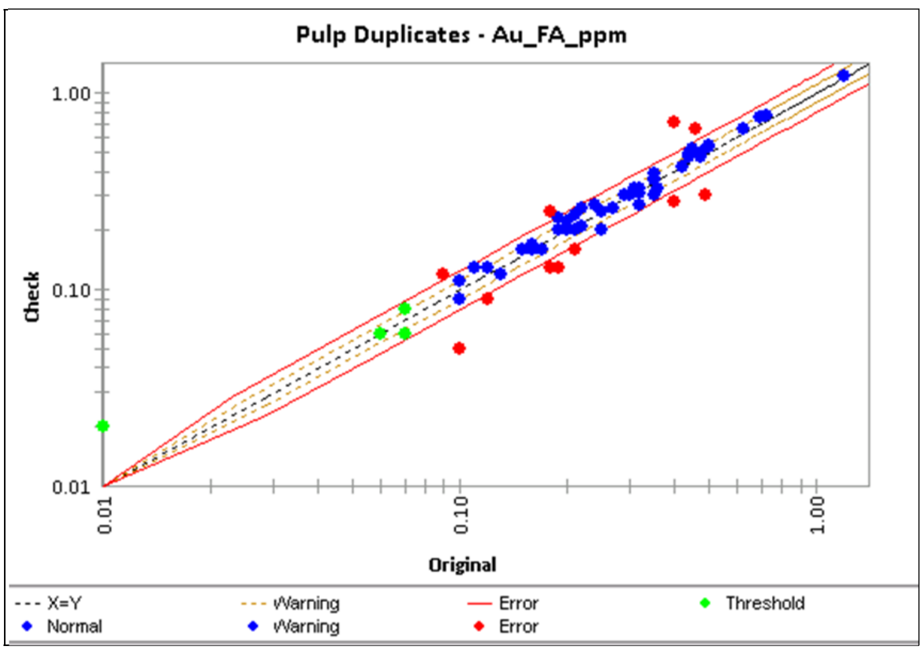
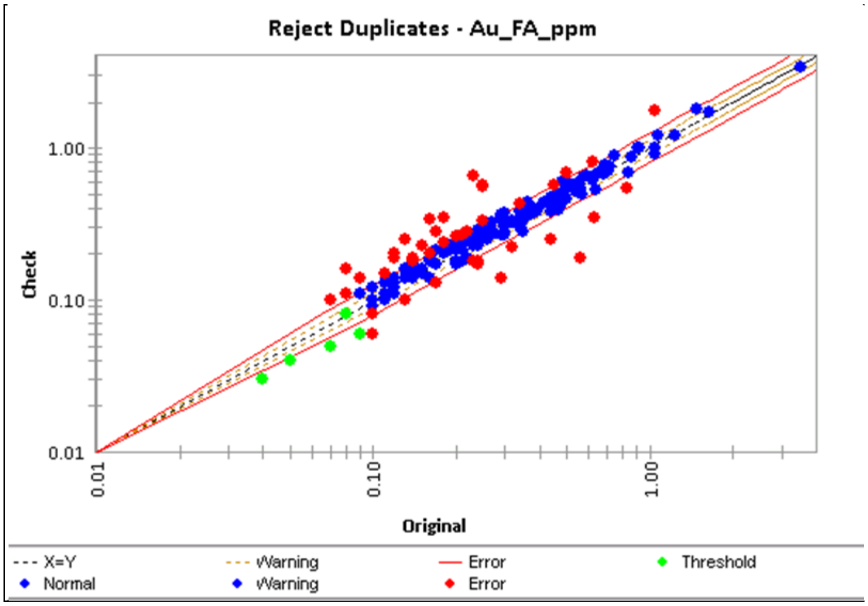


Figure 11-23: QA/QC Program – Reject Duplicates 2013



11.9.4 Check Samples

The check samples sent to Inspectorate for the 2013 campaign consisted of Minosa lab pulp duplicates. The results from Inspectorate are consistent with that of the Minosa lab, providing a reliable independent check on the sample preparation methods and analysis results from the Mine lab.

Pulp duplicate

A total of 293 samples were tested by pulp duplicates. The result of this analyse indicate a good correlation between Minosa and Inspectorate assays (corr. coeff. of 0.88) and on the whole samples are in the range of 20% of variance. Figure 11-24 bellow illustrated this correlation and Table 11-11 summarizes the drill holes that were sampled for pulp duplicate.

Figure 11-24: Inspectorate Check Samples – Pulp Duplicates, 2013

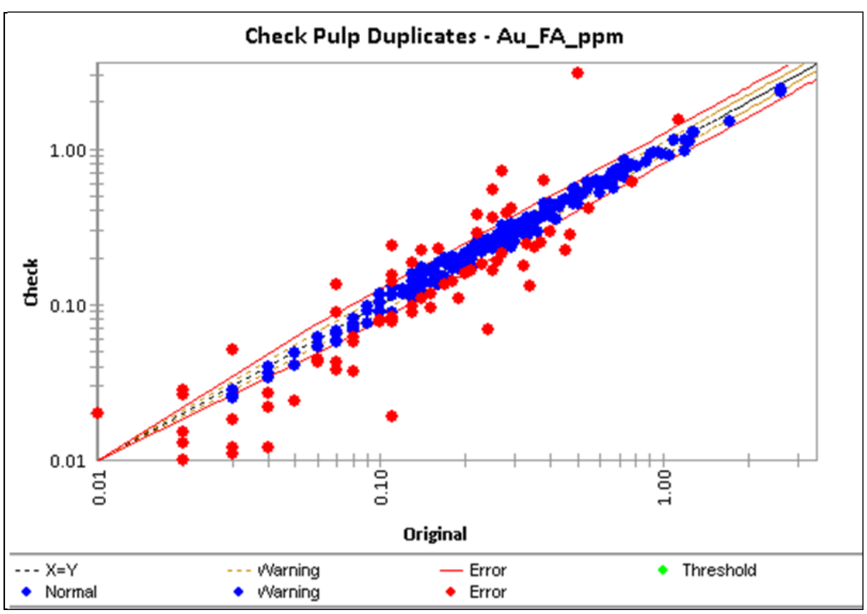


Table 11-11: Inspectorate Check Sample – Pulp Duplicate Drill holes, 2013

Drill hole	Samples	Drill hole	Samples	Drill hole	Samples	Drill hole	Samples	Drill hole	Samples
MO-13-01	5	MO-13-16	2	MO-13-31	7	MO-13-47	2	MO-13-63	4
MO-13-02	7	MO-13-17	3	MO-13-32	3	MO-13-48	5	MO-13-64	5
MO-13-03	3	MO-13-18	3	MO-13-33	2	MO-13-49	4	MO-13-65	4
MO-13-04	7	MO-13-19	3	MO-13-34	2	MO-13-50	4	MO-13-66	4
MO-13-05	2	MO-13-20	4	MO-13-35	3	MO-13-51	6	MO-13-67	8
MO-13-06	2	MO-13-21	3	MO-13-36	2	MO-13-52	7	MO-13-68	4
MO-13-07	2	MO-13-22	3	MO-13-37	6	MO-13-54	4	MO-13-69	5
MO-13-08	2	MO-13-23	3	MO-13-38	2	MO-13-55	7	MO-13-70	5
MO-13-09	2	MO-13-24	7	MO-13-39	3	MO-13-56	5	MO-13-71	4
MO-13-10	7	MO-13-25	2	MO-13-40	3	MO-13-57	3	MO-13-72	3
MO-13-11	3	MO-13-26	2	MO-13-41	8	MO-13-58	6	MO-13-73	4
MO-13-12	2	MO-13-27	3	MO-13-42	3	MO-13-59	4	MO-13-74	4
MO-13-13	4	MO-13-28	3	MO-13-43	4	MO-13-60	4	MO-13-75	4
MO-13-14	3	MO-13-29	4	MO-13-44	4	MO-13-61	4	MO-13-76	7
MO-13-15	6	MO-13-30	2	MO-13-45	4	MO-13-62	4		

Standard Samples

A total of 20 standard samples were sent to Inspectorate and all of them returned within acceptable limits. The standards used are the same as those sent to the Minosa lab and detailed in Table 11-12. Six standards were sent, G305-2, G311-1, G910-10, G998-6, G300-7 and G999-2. Standard G300-7 had a mean of 0.99 g/t and a BIAS of -0.20%. Standard G999-2 had a mean of 0.62 with a BIAS of -6.40%. Figures 11-25 and 11-26 illustrate the results and it is the Authors' view that these assay results are reliable to be used for Mineral Resource and Mineral Reserve estimation.

Table 11-12: Check Standard Reference material results for 2013 Drilling

Standard	Name	Standard Value	Number Standards Submitted	Mean	Min	Max	% Variance to Std Mean
G300-7	Au_FA_ppm	1.00	3	1.00	0.93	1.05	-0.20
G999-2	Au_FA_ppm	0.63	3	0.67	0.59	0.75	6.40
G305-2	Au_FA_ppm	0.32	3	0.32	0.31	0.33	1.46
G311-1	Au_FA_ppm	0.52	4	0.51	0.50	0.52	-2.45
G910-10	Au_FA_ppm	0.97	4	0.94	0.91	0.98	-2.96
G998-6	Au_FA_ppm	0.80	3	0.85	0.83	0.87	5.71

Figure 11-25: Standard G300-7, Inspectorate

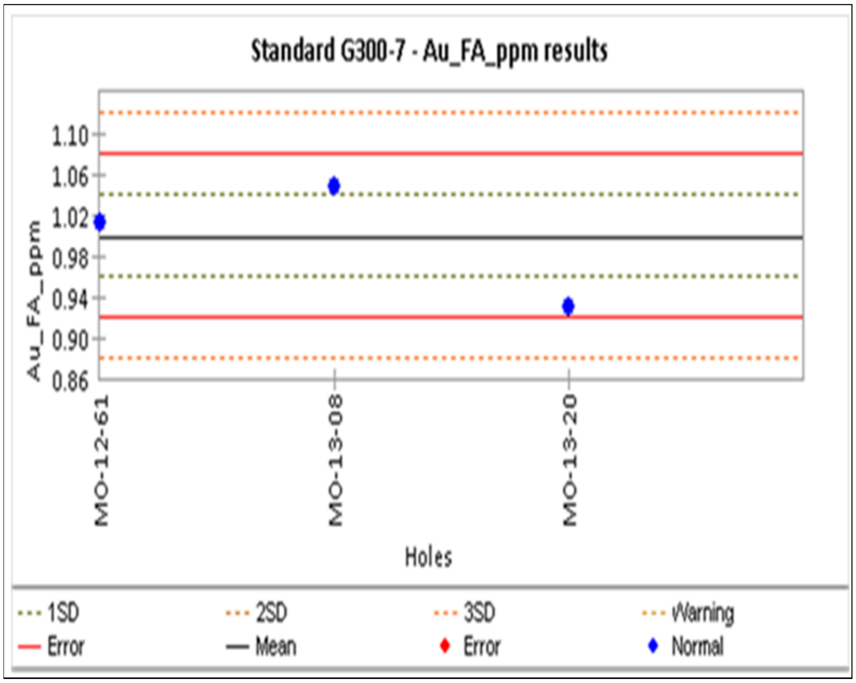
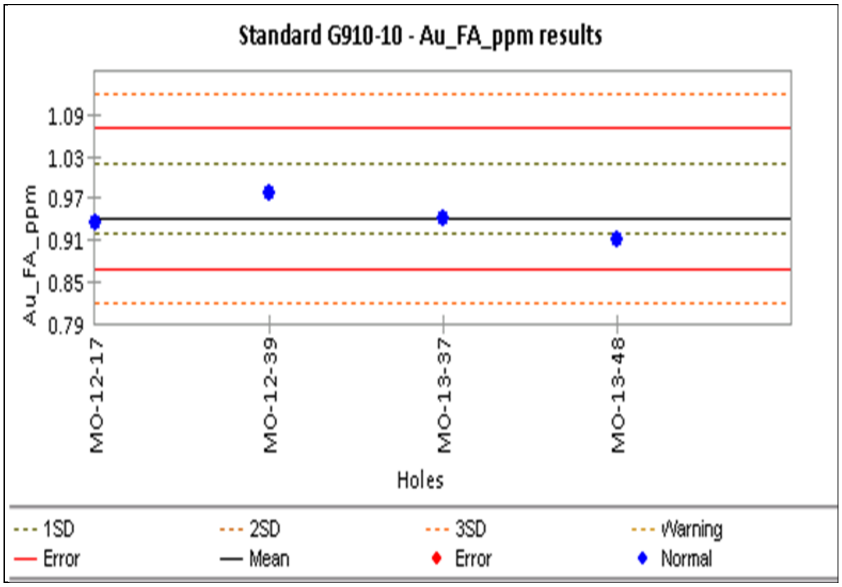


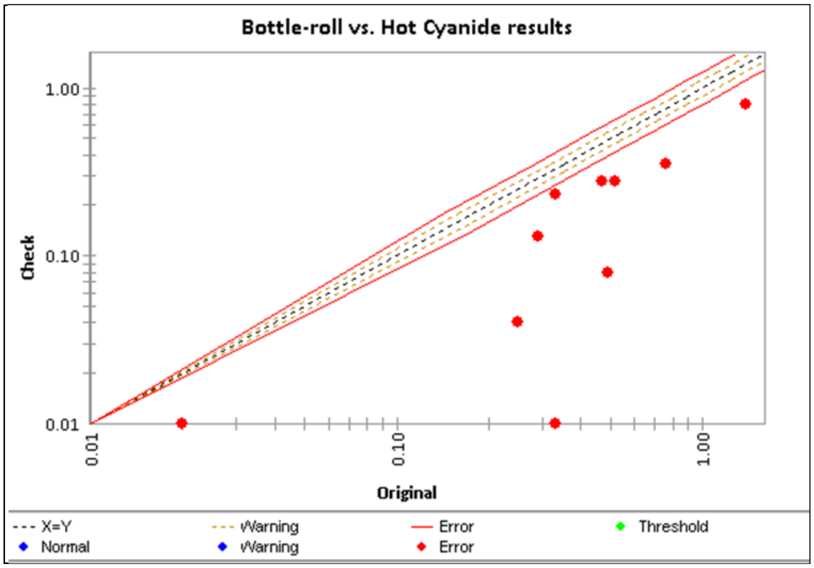
Figure 11-26: Standard G910-10, Inspectorate



11.9.5 Hot Cyanide Assay Comparisons

A Bottle roll analyses was completed to the check hot cyanide (AUHCN) results used to assist in determining the oxide, mixed ore and sulphide boundaries. A few samples, 10 samples from 6 holes, were analysed in internal lab, and all samples have more than 20% variance. The assays show that the hot cyanide results are higher than bottle roll, indicating that the hot cyanide results may be overestimating the recovered gold. However, 10 samples were not representative and not conclusive. Figure 11.27 below illustrates these results.

Figure 11-27: Bottle roll vs. Hot Cyanide Analysis



11.10 Production QA/QC Program, 2012/2013

11.10.1 Standard Reference Material

Minosa started to include standard samples to test blasthole just in November 2012. Prior to this the QA/QC program was limited to duplicate and blank samples. Minosa included 606 standard samples to test blasthole during 2012/2013. Six certified SRMs were inserted into the sample stream for each 20 samples. The SRMs were purchased from Geostats Pty Ltd, Australia. The SRMs range in grade from 0.32 g/t Au to 1.41 g/t Au a further three standards were used by the lab (STD 1 to 3). The SRM's used in the 2012/2013 blastholes are detailed in the Table 11-13.

Table 11-13: Check Standard Reference Material results for 2012/2013 production drilling

Standard	Name	Standard value	Number standards Submitted	Mean	Min	Max	% Variance to Std Mean
G305-2	Au_FA_ppm	0.32	125	0.32	0.25	0.97	0.92
G999-2	Au_FA_ppm	0.63	258	0.59	0.31	1.33	-5.73
G902-7	Au_FA_ppm	1.41	154	1.32	0.28	1.82	-6.09
G311-1	Au_FA_ppm	0.52	56	0.50	0.29	0.58	-3.19
G998-6	Au_FA_ppm	0.80	3	0.85	0.83	0.87	5.83
G300-7	Au_FA_ppm	1.00	10	1.16	0.97	1.32	15.50
STD-1	Au_FA_ppm	0.32			0.28	0.36	
STD-2	Au_FA_ppm	0.52			0.42	0.62	
STD-3	Au_FA_ppm	0.80			0.66	0.94	

Figures 11-28 to 11-30 show graphs of the SRM results for Minosa lab assays compared to the expected value of the SRM. An upper and lower detection limits are shown as red lines and the average of the results is shown as the dark black line. In general, the SRM's submitted through the Minosa laboratory show that the assay results are reliable to be used for Mineral Resource and Mineral Reserve estimation however there are 31 assays out of the accepted limits. These are most likely inaccurate selection of

SRM in the field as there is a consistency in the results of these samples. The bias has a range of +15% until -6% as detailed in the Table 11-13.

Figure 11-28: Production QA/QC Program – G305-2, 2012/2013

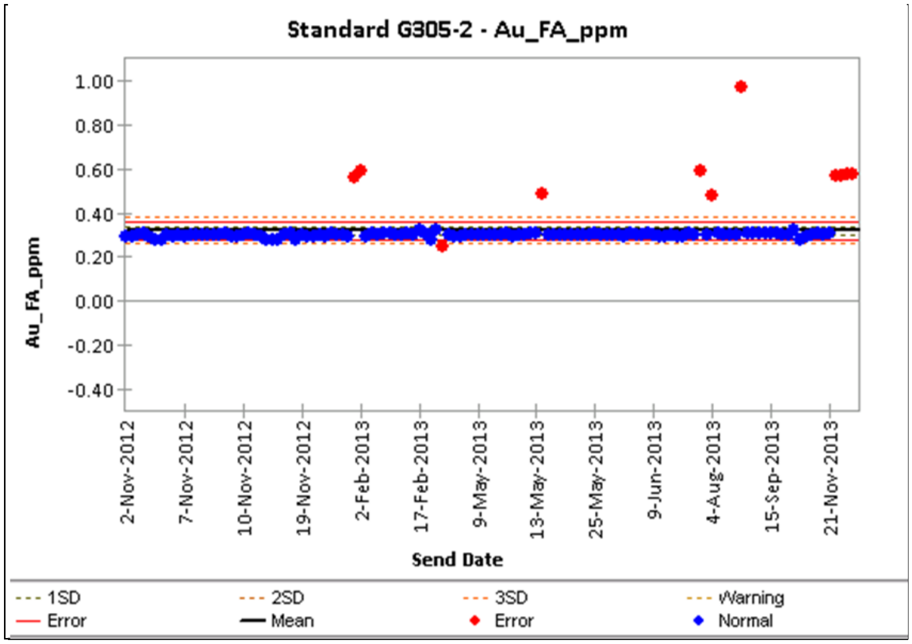


Figure 11-29: Production QA/QC Program – G902-7, 2012/2013

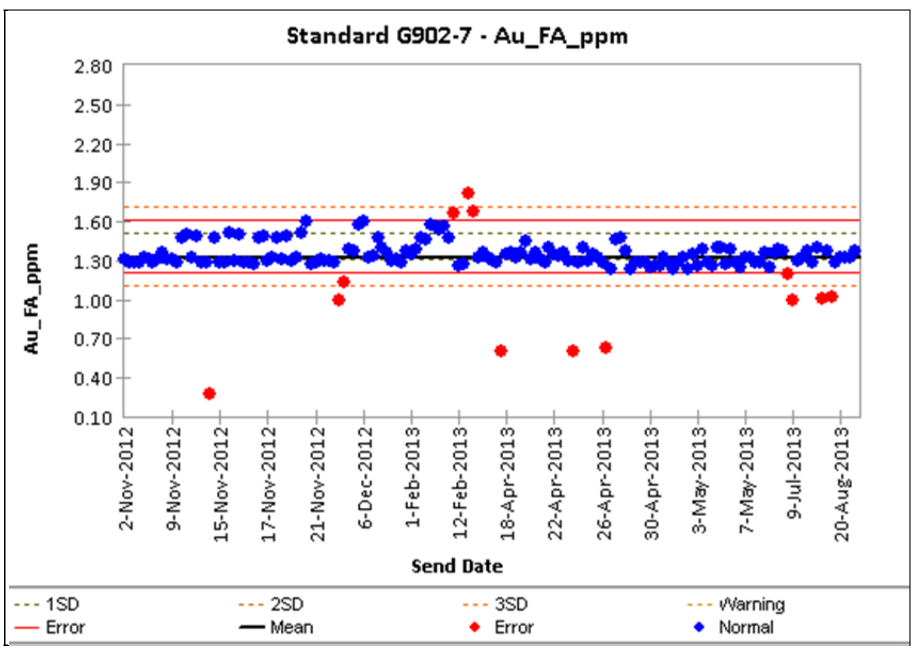
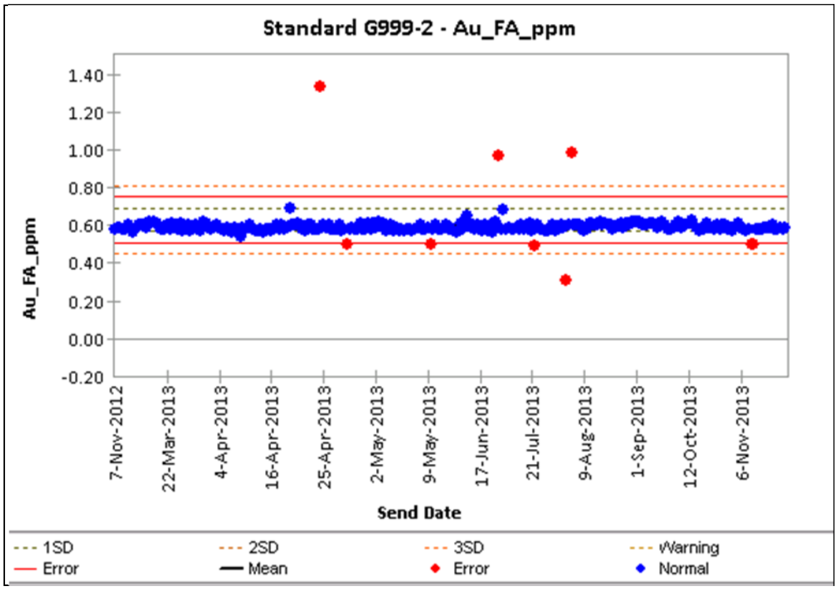


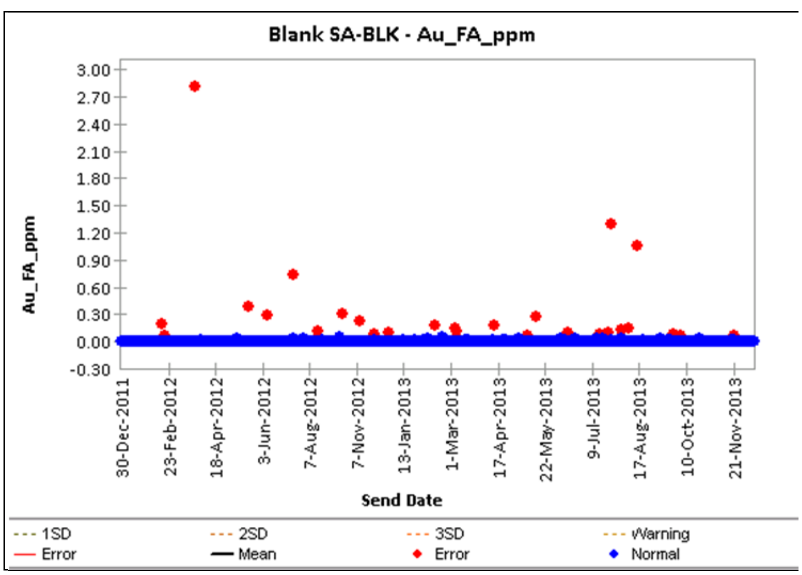
Figure 11-30: Production QA/QC Program – G999-2, 2012/2013



11.10.2 Blank Samples

For the 2012/2013 blasthole, a total of 2,727 blanks (samples with a grade of 0.0 g/t Au) were sent to the lab to test for potential contamination issues. Minosa inserted blank samples each 40 blast samples (2.5%). The first blank was included after the 13th blast sample of each batch. The results are shown in Figure 11-31. 27 samples returned with higher than expected assays, indicating some contamination of samples, but the number is very small and not consistent to indicate that the samples are routinely contaminated. It is the Author's view that the there are some concerns with likely contamination occurring in the sample preparation stage and for batches where blank samples have failed the duplicate sample be submitted for analysis. However in general the assays of blank material are acceptable and confirm that there was likely minimal cross contamination between samples. The line in red is the limit for blank values.

Figure 11-31: Production QA/QC Program – Blanks, 2012/2013



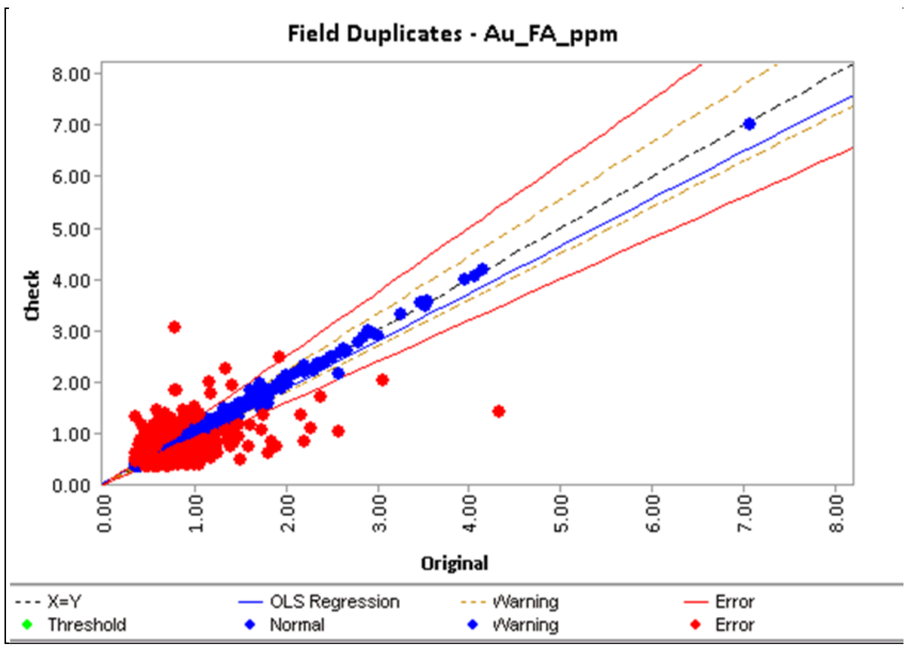
11.10.3 Duplicate Samples

Minosa collected field duplicates each 20 samples (5%), starting after the 15th sample of each batch. Results of the duplicate samples are illustrated in Figure 11-32 where the samples had a good correlation coefficient (0.94). The duplicate samples are illustrated as blue and red in the graphs with the following categories:

- red points indicate samples with more than 20% of variance.
- blue points indicate samples with less than 20% of variance.

The proportion of duplicates above or below 2 standard deviations remains relatively high; between 10 and 20% however this is not unexpected in a gold deposit.

Figure 11-32: Production QA/QC Program – Duplicate Samples, 2012/2013



11.11 Density

Density measurements for the East Ledge and Twin Hills deposit areas as reported by CAM (Armbrust et al., 2005) were made using the "weight in air – weight in water" method. The sample procedure for density measurements is started by air-drying the samples for two to four weeks. Using a balance, the sample is weighed (i.e., weight in air). The sample is then weighed in water using a cradle suspended from the base of the balance and submerged in a barrel of water. The actual weight of the sample in water is the difference between the weights of the cradle/sample in water less the weight of the empty cradle in water.

Moderately to strongly argillaceous samples were wrapped in plastic to prevent water from collecting in the pore spaces and fractures and/or being absorbed by argillic alteration minerals. Tests were done on whole pieces of core (i.e., unsplit core).

No additional specific gravity determinations were completed during the 2012 drilling campaign therefore; there is no change to the assigned value in the 2013 Mineral Resources.

11.11.1 East Ledge Density

In 1998, Greenstone conducted density measurements on a total of 460 pieces of core from eight PQ-diameter metallurgical holes and twelve HQ-diameter exploration holes from the Water Tank Hill area. The samples were grouped together into the principal mineralized and barren rock types and an overall average density was calculated for each rock type. Table 11-14 summarizes the results of the density testing program.

Table 11-14: Specific Gravity Test Results – All Samples

Rock Type	Specific Gravity	Number of Samples
Rhyolite	2.23	197
Andesite	2.22	122
Conglomerate	2.37	94
Quartz veins	2.36	39
Calcite Veins	2.51	8
Total	-	460

After Armbrust et al., 2005

The data set was then further refined to reflect the effect that mineralization has on the density of the material. The mineralized (i.e., >0.50 g/t Au) samples, which are in most cases are also strongly silicified and or quartz veined, were separated from the general sample population and a separate average density was calculated for these samples (Table 11-15).

Table 11-15: Specific Gravity Test Results – Mineralized Samples > 0.50 g/t Au

Rock Type	Specific Gravity	Number of Samples
Rhyolite	2.29	153
Andesite	2.32	23
Conglomerate	2.45	17
Quartz veins	2.37	41
Total	-	234

After Armbrust et al., 2005

As the geology is very similar between the Water Tank Hill and East Ledge pits, additional density tests were not completed at East Ledge and the values obtained from the Water Tank Hill areas were applied to East Ledge.

11.11.2 Twin Hills Density

In 1998, Greenstone conducted density measurements on a total of 191 pieces of core from ten HQ-diameter exploration holes from the Twin Hills area. The results were calculated separately for the oxide and mixed zones.

In the oxide zone there were 151 samples taken, of which 140 samples had rock types coded to the samples (Table 11-16). The average density of these 140 samples was 2.25, with a standard deviation of 0.15.

Table 11-16: Specific Gravity Test – Oxide (After CAM, 2005)

Item	Specific Gravity	Number of Samples
Mean	2.25	151
Median	2.24	151
Standard Deviation	0.15	-

In the mixed zone, there were 40 samples taken, all of which had rock types coded to the samples (Table 11-17). The average density of these 40 samples was 2.37, with a standard deviation of 0.29.

Table 11-17: Specific Gravity Test – Mixed (After CAM, 2005)

Item	Density	Number of Samples
Mean	2.37	40
Median	2.50	40
Standard Deviation	0.29	-

Although the rock types are defined in the geological model and the density of the mixed zone is higher, at 2.37, there are a limited number of samples tested from this zone and there is a lack of production data for material from the zone. Therefore, a density of 2.25 is used for all rock types and ore types throughout the model.

It is recommended that additional density measurements be conducted on mixed zone material to determine an appropriate density that can be incorporated into the block model.

11.12 Grade Control Sampling

The drill cuttings from the blastholes are sampled. The blastholes are generally drilled on a 3.5 m x 4 m pattern, but may be drilled on a 3.0 m x 3.5 m pattern in harder ground. The blasthole location is initially surveyed using a tape measure from surveyed lines. The holes are 12 cm in diameter and drilled using an air track drill rig and / or an Atlas Copco ROC L8 top-hammer hydraulic drill rig. The chips are sampled by placing a two pie-shaped sample tray against the drill rod to collect the cuttings as the hole is drilled. The cuttings are placed in a plastic bag, which is placed at the collar of the hole. The collars are surveyed prior to blasting and the samples are marked by the surveyor and delivered to the sample preparation and assay lab for sample preparation and assay. The samples weigh approximately five kilograms.

All production samples were analysed for Au using both Fire Assay (FA) and Hot Cyanide Leach (HCN) methods. The FA and HCN assays were then returned by the lab to the operations department and validated prior to being uploaded into the database.

Upon receipt of the valid production data the Mines Technical Services Group and the assay and surveyed collar are plotted on bench plans. The assay of the blasthole cuttings is applied to the blast pattern block of 4 m x 4 m x 6 m. The pit geologist assesses the blasthole assay results for the bench to determine the portion of the bench being blasted that will be sent to the waste dump and the portion sent for processing using the short term production cut-off grade of 0.25 g/t in HCN and 0.32 g/t in FA. Lines are drawn on the bench plan designating the waste and the ore. When the bench is blasted, the broken material remains in place.

The surveyors then survey the lines separating the waste from the ore and flags are placed on the broken material for the shovel operator. The smallest mining unit that can effectively be separated is believed to be three blastholes, giving a selective mining unit ("SMU") of approximately 12m x 12m x 6m.

The production grade of the ore sent for processing is estimated on the basis of the average of the blasthole grades and the tonnage is estimated based on the truck counts. The trucks are counted when they leave the pit and when they dump the ore at the primary crusher.

The ore is sampled by a belt sampler after it has been crushed at the secondary cone crusher (-5 cm). The belt sampler consists of a bucket that is passed through the ore as it drops from the crusher onto the belt feeding the agglomerator. A sample is taken approximately every 400 tonnes and passed through a splitter. The sample is placed in a pail and taken to the lab for sample preparation and assay. The ore is weighed by a belt weightometer, which is installed prior to the addition of the cement.

Table 11-18 provides a comparison between the grade of the ore placed on the leach pads based on the tonnage and grade estimate from the blasthole cuttings and the grade based on the weightometer and the estimate from the belt sampler.

The gold grade, as measured by the belt sampler, is very similar to the blasthole assays, and the mine grade and the plant grade do track very well, indicating that the blasthole samples are providing a reasonable estimate of the production on a monthly basis. Any differences are probably due to the difference in the sampling methods.

Table 11-18: Comparison of Mine Production and Plant Monitoring

Period	Mine Tonnage	Mine Grade g/t Au	Plant Tonnage	Plant Grade g/t Au
2005				
Jan-05	341,598	0.744	343,595	0.744
Feb-05	305,784	0.673	343,006	0.673
Mar-05	297,192	0.627	342,200	0.627
Apr-05	280,402	0.657	316,418	0.657
May-05	290.578	0.639	326,178	0.640
Jun-05	199,676	0.628	207,708	0.628
Jul-05	212,980	0.737	215,649	0.737
Aug-05	272,360	0.689	269,346	0.689
Sep-05	223,380	0.754	236,320	0.754
Oct-05	211,812	0.845	223,226	0.845
Nov-05	277,496	0.876	278,612	0.876
Dec-05	272,504	0.835	289,843	0.849
Total 2005	**3,185,662**	**0.723**	**3,392,092**	**0.717**
2006				
Jan-06	311,632	0.824	306,624	0.824
Feb-06	261,226	0.821	266,131	0.821
Mar-06	326,964	0.736	323,244	0.737
Apr-06	334,720	0.644	335,220	0.660
May-06	301246	0.643	304,558	0.642
Jun-06	226,738	0.751	198,948	0.751
Jul-06	306,772	0.663	286,013	0.662
Aug-06	352,892	0.600	332,442	0.599
Sep-06	287,592	0.649	282,272	0.647
Oct-06	380,454	0.622	375,454	0.622
Nov-06	350,573	0.607	355,573	0.607

Period	Mine Tonnage	Mine Grade g/t Au	Plant Tonnage	Plant Grade g/t Au
Dec-06	368,534	0.671	365,534	0.671
Total 2006	**3,809,343**	**0.679**	**3,732,049**	**0.697**
		2007		
Jan-07	346,477	0.64	332,064	0.611
Feb-07	339,003	0.616	345,064	0.626
Mar-07	348,529	0.476	383,272	0.463
Apr-07	290,000	0.485	326,252	0.460
May-07	273527	0.58	296,102	0.462
Jun-07	233,170	0.693	268,132	0.644
Jul-07	171,435	0.682	185,410	0.672
Aug-07	98,778	0.765	110,782	0.658
Sep-07	75,662	0.61	68,336	0.617
Oct-07	167,363	0.469	221,078	0.347
Nov-07	201,002	0.523	224,510	0.443
Dec-07	112,605	0.643	119,222	0.302
Total 2007	**2,657,551**	**0.584**	**2,910,904**	**0.588**
		2008		
Jan-08	255,625	0.61	255,652	0.581
Feb-08	333,519	0.54	333,519	0.366
Mar-08	279,988	0.57	279,988	0.471
Apr-08	419,534	0.60	419,534	0.460
May-08	346,993	0.58	346,993	0.385
Jun-08	305,810	0.64	305,810	0.457
Jul-08	238,417	0.79	238,417	0.514
Aug-08	248,685	0.71	248,685	0.482
Sep-08	282,828	0.86	282,828	0.826
Oct-08	219,739	0.84	239,859	0.810
Nov-08	312,625	0.86	378,687	0.848
Dec-08	219,296	0.94	237,307	0.941
Total 2008	**3,463,086**	**0.70**	**3,567,279**	**0.584**
		2009		
Jan-09	342,447	0.890	342,447	0.790
Feb-09	309,971	0.861	309,971	0.847
Mar-09	442,065	0.833	442,065	0.754
Apr-09	387,246	0.757	387,246	0.535
May-09	319,024	1.237	319,024	0.712
Jun-09	322,894	1.004	322,894	0.797
Jul-09	455,160	0.822	455,160	0.667
Aug-09	441,373	0.774	328,125	0.660
Sep-09	400,974	0.705	369,149	0.649
Oct-09	444,872	0.666	400,159	0.602
Nov-09	256,729	0.654	230,548	0.587
Dec-09	407,254	0.751	328,488	0.584
Total 2009	**4,530,009**	**0.821**	**3,709,056**	**0.679**
		2010		
Jan-10	489,331	0.719	489,331	0.612
Feb-10	464,170	0.808	464,170	0.668
Mar-10	438,710	0.764	438,710	0.770
Apr-10	380,657	0.793	380,657	0.780
May-10	369,163	0.726	369,163	0.742

Period	Mine Tonnage	Mine Grade g/t Au	Plant Tonnage	Plant Grade g/t Au
Jun-10	305,948	0.662	305,948	0.661
Jul-10	369,517	0.645	369,517	0.647
Aug-10	389,634	0.676	389,634	0.664
Sep-10	413,544	0.673	413,544	0.649
Oct-10	421,025	0.640	421,025	0.661
Nov-10	443,756	0.768	443,756	0.759
Dec-10	428,445	0.770	428,445	0.774
Total 2010	**4,913,900**	**0.720**	**4,913,900**	**0.700**
		2011		
Jan-11	472,278	0.796	472,278	0.727
Feb-11	408,753	0.794	408,753	0.733
Mar-11	435,834	0.779	435,834	0.766
Apr-11	431,632	0.811	431,632	0.773
May-11	401,351	0.814	401,351	0.804
Jun-11	270,522	0.629	270,522	0.548
Jul-11	267,473	0.595	267,473	0.523
Aug-11	401,908	0.729	401,908	0.645
Sep-11	276,132	0.871	276,132	0.820
Oct-11	268,131	0.645	268,131	0.625
Nov-11	342,382	0.601	342,382	0.526
Dec-11	336,551	0.659	336,551	0.516
Total 2011	**4,312,947**	**0.737**	**4,312,947**	**0.679**
		2012		
Jan-12	349,356	0.609	362,995	0.664
Feb-12	348,555	0.705	340,480	0.671
Mar-12	390,714	0.734	362,601	0.678
Apr-12	404,477	0.699	379,666	0.704
May-12	448,896	0.751	477,120	0.682
Jun-12	347,876	0.724	379,150	0.750
Jul-12	383,074	0.716	347,890	0.734
Aug-12	243,807	0.598	228,572	0.656
Sep-12	255,763	0.593	242,428	0.580
Oct-12	294,701	0.508	311,632	0.483
Nov-12	468,249	0.440	396,236	0.433
Dec-12	437,131	0.532	435,181	0.514
Total 2012	**4,372,598**	**0.635**	**4,263,953**	**0.629**
		2013		
Jan-13	467,742	0.558	499,681	0.512
Feb-13	446,101	0.572	413,629	0.562
Mar-13	462,759	0.605	494,362	0.605
Apr-13	515,543	0.560	473,525	0.563
May-13	520,148	0.607	500,733	0.577
Jun-13	482,806	0.755	492,357	0.664
Jul-13	537,469	0.629	516,490	0.601
Aug-13	498,871	0.565	473,335	0.541
Sep-13	388,558	0.613	444,106	0.563
Oct-13	584,066	0.550	541,032	0.556
Nov-13	378,919	0.524	445,244	0.460
Dec-13	182,056	0.500	75,647	0.494
Total 2013	**5,465,039**	**0.589**	**5,370,142**	**0.564**

12.0 DATA VERIFICATION

The quality and reliability of the drill hole database used for the estimation of Mineral Resources and Mineral Reserves has been tested four separate times. CAM carried out an independent verification of the data in 2005, Minosa carried out a verification of 105 drill holes in 2006, RPA Scott Wilson carried out a verification of 19 RC holes in 2007 and in 2012 Joao Francisco Silva (Database & GIS Manager Exploration, Minosa) and Benjamin Bartlett (Manager, Mineral Resources) completed a full database review where 46 holes were modified for inconsistencies. Joao Francisco Silva oversaw the management and validation of the database throughout 2013.

Minosa's 2012 data verification program included the following tasks:

- Collar validation, database against paper logs.
- Survey validation, check against paper logs.
- Geological descriptions, check against paper logs.
- Assay validation with below detection limits modified from -1, check against lab reports.

Historical validation checks completed included, verification of the survey and topographic data.

12.1 Survey and Topographic Data

Beginning with the 1996 RC drill program, drill hole collars were surveyed using modern, electronic survey equipment. The 1997-1998 programs utilized a Topcon FC/48GX total station instrument from drill hole SA-149 onwards. About one in 15 holes were randomly selected and resurveyed to check the quality of the survey results.

It is not exactly known how the drill holes were surveyed prior to 1996. However, various surveyed reference points were known to exist around the Water Tank Hill area as far back as the 1992 Fischer-Watt program. The exploration personnel used these surveyed points to identify the drill hole collar coordinates in the early programs. It is possible that some of the final coordinates of these holes were determined by compass and tape methods.

CAM (Armbrust et al., 2005) reports that the survey and collar information was converted from a mine grid coordinate system to UTM coordinates by Minosa, prior to entering into the database. All drill holes drilled prior to 1996 were in mine grid coordinates and had to be converted to the UTM coordinate system. Those holes that were still locatable in the field were resurveyed into UTM directly. Those that were no longer locatable were converted arithmetically into UTM based on their original mine grid coordinates and a mine grid to UTM conversion factor.

Down-the-hole surveys were performed on 14 core holes (SC-12, 14, 16-18, 20, 21, 23-27, 31, and 32) using a Sperry Sun single shot instrument. No factor was applied to the orientation of any of the holes that were not surveyed down-the-hole.

12.2 Data Verification for the Pre-2005 Drill Programs

CAM (Armbrust et al., 2005) reported that, for the 1997-1998 Twin Hills drill programs and for the 2002 East Ledge drill hole program, all assay results were sent electronically from the assay lab and downloaded directly into the database to prevent data entry errors. All assay results from the prior drill programs were entered manually. These data were all verified against the original assay certificates before modeling was initiated.

CAM reported that 80% of the drill hole assay data had been transferred electronically from the lab and entered directly into the database, minimizing the chance for any data-entry errors. For the other 20% of the drill holes (8 out of 86 holes at East Ledge and 29 out of 97 holes at Twin Hills), assays were manually entered into the database. CAM checked the assay certificates for seven holes, which had been entered manually and checked the data from the certificates against the data in the database. No errors were found.

CAM was provided with the 2005 exploration data sets and they ran their standard check procedure on the exploration databases. This check procedure includes:

- Check for duplicate collars.
- Check for twin holes.
- Check for statistically anomalous down hole surveys.
- Review of assay statistics by grade class.
- Review of assay statistics by length class.
- Check for assay values successively the same.
- Check for assay spikes.
- Check for down hole contamination by decay analysis.
- Check of total grade thickness by hole.

Although a few anomalies were noted, and were forwarded to Minosa, the number and type of anomalies were within industry norms for databases of this size (even if the anomalies were errors they would not have had an effect on the overall Mineral Resource estimate). On the basis of these statistical checks, and the checks of the data entry, CAM concluded that the exploration database had been prepared according to industry standards and was suitable for the development of geological and grade models.

12.3 Data Entry Procedures for the 2005-06 Drill Programs

The Minosa database entry procedures for the 2005-2006 drill programs were as follows:

- The coded geologic information contained in the handwritten logs for each drill hole is manually entered into one master database. EXCEL is used to create a spreadsheet that can be read directly into the MineSight software modeling program. Along with the collar location (northing, easting, elevation), survey information (bearing and dip), and total depth of the hole, the spreadsheet contains the following columns: drill hole identification, the from and to interval for each sample, sample length, Au grade (g/t), Ag grade (g/t) (when available), lithology, alteration, % quartz veins, oxide/sulphide, % pyrite, dry/wet, hematite, jarosite/goethite, structure, and sample identification number.
- Upon the completion of a drill hole, all of the drill and geological information is entered into the EXCEL master database. The assay data are entered when the assays are received from the lab. Assay certificates are received electronically and the information is cut from the electronic certificates and pasted into the master database. The assays for each sample are also hand entered into the individual paper drill logs. This exercise provides verification that the assays match the sample numbers and the drill log entries are complete.
- Once the data entry procedure for each hole is finished, a quick visual review of the file is performed to check for completeness.

12.4 Data Verification for 2005-06 Drill Programs

In April 2006, Minosa completed a verification of the database for all drilling available as of that date, a total of 526 drill holes, up to and including drill hole MO-06-11. Randomly, 20%, or 105 drill logs, were selected and the following drilling information was compared between the original paper logs and assay certificates with the electronic database: hole identification, total depth, collar (northing, easting, and elevation), survey information (bearing and dip), oxidation state, and assay information were checked. No significant errors or discrepancies were found.

In the January 2007 technical report entitled "Technical Report on the San Andrés Gold Project, Honduras", prepared for Yamana, Scott Wilson RPA reported that they checked 19 RC logs and one core log from the 2005 and 2006 drilling programs against the master data file for data entry errors. No errors were found in the collar surveys, but errors were found in two drill holes between the dip recorded on the drill-hole log and the database (MO-06-46 and MO-06-48). In both instances, the borehole log showed a dip of -45 degrees and the dip in the data file was -50 degrees. These entries have been corrected.

No errors were found in the recording of the oxidation state between the drill hole logs and the database. All other geological information on the logs was transcribed accurately to the data file. In some entries, the structure-type was not transcribed from the log to the database.

Scott Wilson RPA also audited the gold assays shown on the handwritten logs, the assay certificates from the lab, and the assays entered in the data file. A total of 1,990 samples were checked from the 19 RC holes and the one core log. No errors were found in the data entries. For duplicate samples, in all cases, the assay from the "A" sample was consistently entered in the database.

For duplicate samples taken on the pulps at the CAS lab, the lab consistently entered the assay for the first sample pulp on the assay certificate.

Scott Wilson RPA concluded that the differences found by their check procedures were considered minor. The geological information in the database was consistent with the information contained in the drill hole logs and the geological and grade information in the database was considered to be of high quality and acceptable for use in the estimation of Mineral Resources and Mineral Reserves.

While MCB did not carry out an independent sample verification of the database, they concluded that because the Mine has been operating for more than 10 years, and because the grade of the drill holes compares well with the mined grade (Table 12-1), this vets the drilling and sampling programs that have been implemented. MCB concluded that the drill hole data was suitable for the estimation of Mineral Resources and Mineral Reserves.

12.5 Production data validation

Minosa completed a verification of the database for all production drilling available as of December 31, 2013, a total of 165,996 production blast holes were included. During the validation exercise a total of 104 blast holes were identified with incorrect collar elevations. These holes were not removed and do not impact the Mineral Resources. The Authors recommend that these holes be removed from future evaluations.

The production data is validated through GIM Acquire's import system, where the data generated by the geology, planning, topography and lab are cross referenced for:

- Topography x Mining plan.

- Collar x Topography.
- Samples x assays.

This process is then checked by the operations technical support team to visually check the production data against the mining block.

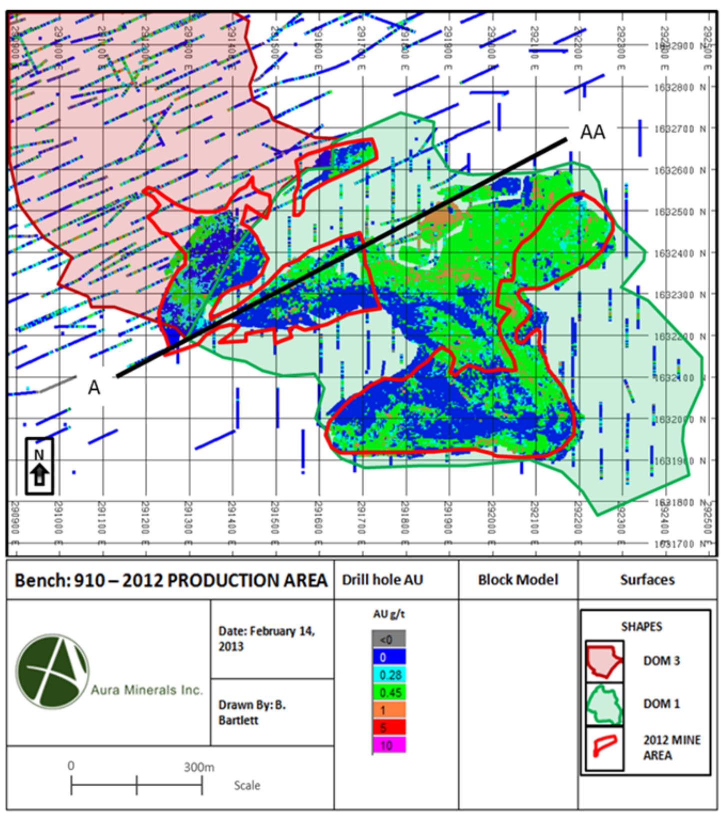
12.6 Block Model Reconciliation – Production drill hole data

In January 2013, Minosa carried out a resource validation study using actual Mine production and reconciling this with the interim January 2013 resource block model for the 12 months of 2012. The interim model was used for mine planning and resource estimation parameter validation. The reconciliation study, covered the Twin Hills (Dom 1) area along with a small section of the Twin Hills (Dom 3) reproduced each months production by material type (oxide / mixed) and applied the forecast recoveries to the predicted ounce production. A 12 month period was deemed sufficient to account for the lag time between production and leaching time which is approximately +3 months from production.

The study also provided a validation check on the use of production blast hole data in areas where wide spaced resource drilling (>50 m x 50 m) has proven to be unreliable in short range grade predictions required for production areas.

Figure 12-1 illustrates the production areas, production drilling and the domain areas.

Figure 12-1: Resource Model Validation - 2012 production areas



The reconciliation results are summarised in Table 12-1 below and show that the resource model predicted 59,354 recovered ounces at 0.43 g/t versus an actual recovered total of 59,751 oz at 0.44 g/t. This is a 0.7% variance in ounces. This study supports the use of production data in the resource model for both interpretation of the oxide, mixed and sulphide boundary and for grade estimation. The forecast recoveries used are based on the operational forecast figures for the Twin Hills area and are marginally higher than that used for the San Andrés total Mineral Resources. When using, 0.76% (oxide) and 0.57% (Sulphide) recoveries the resource model predicted variance to actual produced ounces is -3.1%.

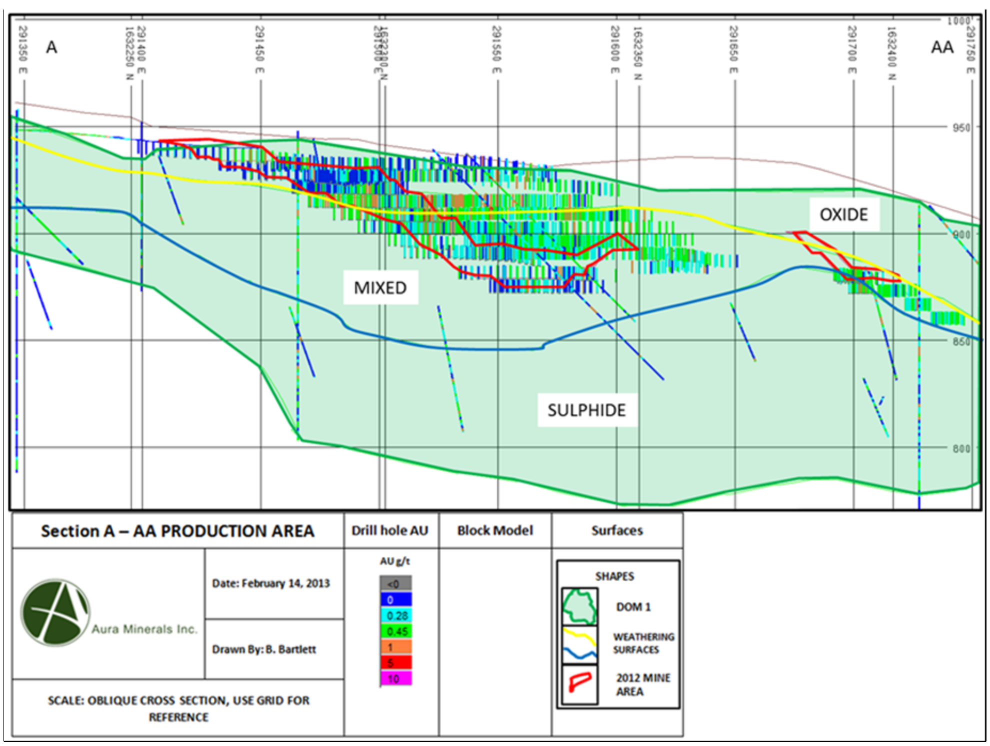
Table 12-1: Resource Model Reconciliation – 2012 Production

Jan 2013 Model Predicted	Recovered Oz	4,456	4,795	6,886	6,865	7,294	5,527	5,290	3,339	2,290	2,986	3,880	5,746	59,354
	Recovered grade	0.43	0.47	0.45	0.42	0.45	0.45	0.51	0.42	0.39	0.36	0.37	0.44	0.43
Actual Recovered Production	Produced Oz	5,163	4,174	4,048	5,994	6,496	5,641	5,528	6,432	4,338	3,036	3,628	5,272	59,751
	Grade (g/t)	0.44	0.38	0.35	0.49	0.42	0.46	0.49	0.88	0.56	0.30	0.28	0.38	0.44
Variance	Oz	708	-621	-2,838	-871	-798	113	239	3,093	2,048	50	-252	-475	397
	Grade (g/t)	0.01	-0.08	-0.10	0.08	-0.03	0.01	-0.01	0.46	0.16	-0.05	-0.08	-0.07	0.00

*Forecast recoveries of 78% (Oxide) and 62% (Mixed) applied to predicted raw ounces

Figure 12-2 shows a cross section A-AA of the long-term drilling and production drilling model at the Twin Hills. Note that the drill spacing of the exploration drill holes in this area are between 50 to 100 m and are insufficient to accurately define the location of the oxide, mixed, and sulphide horizons and precisely estimate the local grade variability required for production forecasts. The short-term blasthole assays found additional mineralized material and defined the local variability, so that there is are real positives in using production information in areas where the exploration data is inadequate.

Figure 12-2: 2012 Production Area- Section A-AA



*Clipping +/-12.5m.

12.7 Authors' Opinion on the Reliability of Drill Hole Database

The Authors believe that based on the description of the database management procedures and on the findings of the three verification programs carried out on the drill hole database; that the drill hole database has been maintained to industry standards and it is sufficiently reliable and can therefore be used for the estimation of Mineral Resources and Mineral Reserves.

The Authors also believe that based on detailed 12 months reconciliation of the production blast hole data against actual mill production their use in the Mineral Resource estimation for active producing domains is warranted. The reconciliation data also indicates that the use of production data in the estimate provides a more reliable short term estimate in areas where there is some wide spaced (>50 m x 50 m) drilling, this is further highlighted in Section 24.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 East Ledge Deposit

The East Ledge deposit was assessed using bottle roll tests. Although bottle roll tests provide an indication that the ore is amenable to heap leaching, the tests do not provide quantitative estimates of the percent recovery. In the case of the East Ledge deposit, the recovery factors are based on production results. Historical production results between January 2003 and September 2007 indicate an overall recovery from the East Ledge deposit of 84%.

13.2 Twin Hills Deposit

A total of eight composites, one surface sample, and one bulk ore sample from Twin Hills were column tested (Allen, C. and McClelland, G. 1997). Direct agitated cyanidation (bottle roll) tests were conducted on all 10 composites and samples at P_{80} 10 mesh and 200 mesh feed sizes to determine gold recovery, recovery rate, reagent requirements, sensitivity to feed size, and preliminary amenability to heap leaching treatment. Column percolation leach tests were conducted on the bulk ore samples at P_{80} 3 inch, 1 inch and ½ inch feed sizes to determine gold recovery, recovery rate, reagent requirements, and optimum heap leach crush size. Overall bottle test data for the Twin Hills composites/samples indicate that the Twin Hills ore would be amenable to heap leaching treatment. The Twin Hills bulk ore sample was readily amenable to heap leach cyanidation treatment at all three crush sizes. Gold recovery rates were fairly rapid for all feed sizes, and extraction was substantially complete in 10 to 15 days of leaching. Additional gold was extracted after 15 days, but at a much slower rate. Gold recoveries for P_{80} 10 mesh feeds ranged from 63.0% to 93.5% in 144 hours of cyanidation treatment. Grinding to P_{80} 200 mesh improved gold recovery for all composites/samples except for composite 4 (SA-091, 37.5m to 42.0 m). Gold recoveries from P_{80} 200 mesh feeds ranged from 73.4% to 96.6% and averaged 89.4% in 144 hours of leaching.

Overall, column leaching test data show that the Twin Hills bulk ore is readily amenable to heap leaching. Gold recoveries of 86.5%, 87.5%, and 87.2% were achieved from P_{80} 3 inch, 1 inch, and ½ inch, respectively, in 68 days of cyanide solution contact.

Gold recovery rates were fairly rapid for all feed sizes, and extraction was substantially complete in 10 to 15 days of leaching. Gold was extracted after 15 days, but at a much lower rate.

13.3 Gold Recoveries Based on Production Results

The gold recovery based on production estimates for 2001 through to 2013, is shown in Table 13 -1.

Table 13-1: Gold Recovery – Production[1]

Period	Ounces to Pad	Ounces Recovered	% Recovery
2001	128,645	105,998	82.4
2002	117,005	99,064	84.6
2003	58,800	50,794	86.4
2004	83,877	65,032	77.5
2005	78,231	61,236	78.7
2006	83,625	70,779	84.6
2007	49,068	51,240	104.4
2008	66,988	47,761	71.3
2009(2)	98,843	68,372	68.5
2010	110,518	70,641	63.9
2011	94,140	60,871	64.7
2012	86,292	59,751	69.2
2013	103,085	63,811(3)	61.9(3)

Note. Prior to February 2006, production was by RNC Gold Inc.
1. – Sourced from internal production data
2. – Between 2009 and 2013, 6 Mt Ore from Mixed Zone Stacked & Leached
3. – Due to labour strikes, most of the gold leached in Dec. was not refined (effectively recovered in 2013)

13.4 Projected Gold Recovery from the Mixed Zone

A sample of sulphide-rich conglomerate was collected from the north end of the East Ledge pit in February 2006 for metallurgical testing. The sulphide-rich conglomerate was in direct contact with a strongly oxidized conglomerate, with little or no mixed zone present between the sulphide and oxide zones. The estimated grade of the sulphide-rich sample was between 0.40 g/t Au and 0.65 g/t Au. A column test was conducted on the coarse material (2.5 inch) and the gold recovery was 36.1% after 5 days, 42.2% after 10 days and 43.3% after 20 days. There is an increase in cyanide consumption compared to the column tests on the oxide material. McLelland and Allen (1997) report that overall recovery for the three feed sizes indicate that the bulk ore samples are not ore size sensitive. The gold recovery rate increases slightly with decreased crush size, but not sufficiently to warrant additional crushing costs.

The pit evaluation parameters used to estimate the Mineral Reserves are 76% recovery for the oxide zone and 57% for the mixed zone for all of the mineralization.

For the oxide zone, Scott Wilson RPA considers a gold recovery estimate of 78% to be reasonable and conservative (January 2007 Technical Report). The gold recovery estimate for the East Ledge deposit is based on 2004 and 2005 production results rather than test samples. Scott Wilson RPA notes that the gold recovery for the production from the East Ledge pit from 2003 to September 2006 averaged 82.1%. For the Twin Hills deposit, Scott Wilson RPA reviewed the borehole logs and, in their opinion, the samples from the oxide zone are representative of the mineralization. Scott Wilson RPA noted in the McLelland and Allen Report that the one sample (Sample 6) from Twin Hills did not respond to leaching in bottle roll tests. The sample was from drill-hole SA-112 - a 43.5 m to 49.5 m interval. A review of the log determined that the sample was from the fresh (sulphide) zone.

The bottle roll tests on sample 4, taken from the same drill hole (37.3 m to 42.0 m), indicated gold recoveries between 70% and 75% compared to the recovery of 90% achieved for other samples (excluding Sample 6) at the same grind size (80% passing 200 mesh). A review of the drill log indicated that the sample was taken from the mixed zone (containing both oxidized and fresh material).

Although the test results indicated gold recoveries higher than 76% for oxide material, at this stage, for the purposes of the Mineral Reserve estimate, Aura considers the 76% factor appropriate for the oxide zone and 57% factor appropriate for the mixed zone. Although the column test on the mixed zone from the East Ledge pit indicated a gold recovery of 43%, the test was conducted on coarse material which predominantly consisted of fresh (sulfide) material. Although not definitive, the bottle roll tests on the mixed zone at Twin Hills indicated a gold recovery in the 75% range.

Historical production records show that between 2009 and 2013 approximately 6 Mt of mixed ore from the Twin Hills deposit was treated with a resultant mixed ore recovery ranging from 40% to 62%. A high variability in the mixed zone recovery can be expected. The recovery will be dependent on the ratio of fresh sulphide present.

As part of future leaching tests on the mixed zone, Aura has started the hot soluble cyanide gold assay procedure for both production blast hole assays and plant metallurgical control. This assay technique provides an excellent guide as to the degree of oxidation of the gold mineralization and its potential recovery.

The estimated recovery breakdown between oxide and mixed ore is shown in Table 13– 2 below.

Table 13-2: Breakdown of Recovery between Oxide and Mixed Ore - Production

Period	Oxide Ore Ounces to Pad	Mixed Ore Ounces to Pad	Total Ounces Recovered to Carbon	Oxide Ore Recovery (%)	Mixed Ore Recovery (%)
2009	86,489	11,678	98,167	74.0	46.0
2010	67,863	43,059	110,922	76.0	50.6
2011	54,187	46,887	101,074	73.0	39.6
2012	67,718	21,079	88,797	82.0	61.0
2013	61,696	41,389	103,085	77.0	62.0

Note: Between 2009 and 2013, 6 Mt Ore from Mixed Zone Stacked & Leached

Based on the bottle roll and column tests on the mixed zone at Twin Hills, and historical production, a gold recovery of 57% and 76% for mixed ore and oxide ore respectively has been used for mineral reserve and resource estimation and mine economics.

A portion of the Mineral Reserves, located between, and adjacent to, the East Ledge and Twin Hills deposits, has not yet been tested. However, the geological setting and the style of mineralization are similar and Aura believes the recovery factor is consistent with what has been found to-date.

14.0 MINERAL RESOURCE ESTIMATE

The Mineral Resource estimate was prepared by Minosa using a geological model, ordinary kriging (OK) interpolation and the commercial mine planning software MineSight. The geological interpretation considered the San Andrés deposit as a gold deposit associated with extensional structures within tectonic rift settings. Commonly, these deposits contain both gold and silver mineralization associated with banded quartz veins. At San Andrés, silver is not considered to be economically material, and only the gold grade has been considered for the estimation of the Mineral Resource.

Over the course of 2012 Minosa completed interpretations and developed geological models for the structural, lithological, alteration, weathering and mineralisation at the San Andrés deposit, these models were tested in the field and incorporated into interim production and short term mine planning models. Previous geological models defined the mineralisation domains to the rhyolite Tuff (Trv) and oxidation surfaces, on review this simplified the resource and did not take into account the local variation in data populations or the structural and lithological controls on the mineralisation. The changes to the geological model have produced 12 mineralisation domains determined by changes in the structural and lithological controls impacting mineralisation orientation differences. There was no further segregation on weathering following a review into the impact of the weathering profiles on mineralisation, which found that there is a minimal impact on the grade distribution over the weathering profile. The oxidation zones are not uniformly populated; therefore data distribution largely influenced historical segregation of the mineralisation on weathering.

The block model limits are based on the UTM system and use 10 m x 10 m x 6 m blocks. The drill hole and production gold assays were composited to two different lengths 1.5 m and 6.0 m. Composite lengths were determined by domain and whether production data was within the domain, Table 14-1 details the variable search parameters (section 14-6 Block Model Parameters).

Mineralisation domain boundaries were considered as hard boundaries. The estimation strategy used two passes, with the search ellipsoid size and orientation based on the anisotropy determined by the variography.

Mineral Resources were tabulated below the December 31, 2013 topography surface to account for previously mined material and also considered a US$1,600/oz, resource pit shell. Note that the resource limit pit shell did not consider any of the sulphide material.

14.1 Drill Hole Database

The San Andrés exploration drill hole database was maintained in MineSight format and contains 1,013 holes, 1,006 of which are inside the MineSight project limits. The full data set has drilling comprised of 104,810.3 m of reverse circulation drilling in 905 holes and 19,517.8 m of core drilling in 108 holes for a total of 124,328.1 m in 1013 drill holes. A total of 83,348 gold assays with sample lengths ranging from 0.01 m to 6.0 m with the most common sample length being approximately 1.5 m.

The San Andrés production drill hole database was maintained in MineSight format and contains 165,996 holes inside the MineSight project limits and impacting domains 1, 3 and 4 (Twin Hills and Cerro Cortez areas). The drilling is comprised of 1,026,852.92 m of blast hole drilling with most of the holes drilled to 6.5 m with the last 0.5 m the sub-drill, which were generally not sampled. A total of 166,350 gold assays with sample lengths ranging from 2.0 m to 8.5 m with the most common sample length being approximately 6.0 m.

The MineSight database contains both exploration and production drill holes containing a total of 248,698 gold assays gold assays with sample lengths ranging from 0.01 m to 8.5 m. A subset of 190,258 gold assays has been tagged with the domain code wireframes that define the mineralization and has been used for the estimation of Mineral Resources.

A significant change from the 2011 Minosa Mineral Resource is the use of production data for the Twin Hills and Cerro Cortez areas (Dom 1, 3 & 4) where much of the exploration drill hole data is in excess of 50 m x 50 m drill spacing. Poor mine reconciliation to the resource model in these areas requires that closer spaced data be used to improve accuracy and precision of the resource model in areas where production data is infilling wide spaced exploration data. The introduction of production data coincided with implementing a QA/QC programme for the production data (see Section 11.9)

14.2 Geological Modeling

The geological interpretation of the structures, lithology, alteration, were completed by Minosa in 2012 using the Datamine software package (CAE Mining Canada Inc) and imported into MineSight as AutoCad dxf files. These models were later validated through mapping and updated in using MineSight in 2013. The weathering and mineralized domains used for mineral resource estimation were completed by Minosa in MineSight. Validations of the geological interpretations were completed by in field mapping and observations by Ben Bartlett, Manager Mineral Resources, Aura (QP).

The geological interpretation was carried out on vertical cross sections, by connecting continuous geological units being modelled. Domain mineralized zones were defined using the following assumptions:

- Nominal resource cut-off grade of 0.20 g/t.
- Geological continuity defined by spatial location, structural orientations and lithological controls.
- Mineralisation continuity and grade distribution.

Interpretations were completed on 25 m oblique NE-SW sections and taking care to snap to the ends of the samples, where-ever possible to ensure geological continuity was not compromised. Figure 14-1 shows across section (291250 E) through the East Ledge and Cemetery of the interpreted 3D wireframe of the lithology, structures, topography and drill holes Figure 14-2 shows a plan view on the 920 m bench. Figures 14-3 and 14-4 illustrate the Mineralisation Domains on plan (950 m bench) and section 291400 E and their relationship with the structures and overlapping lithological contacts.

Figure 14-1: Cross Section – 291250 E – Lithology and Structure

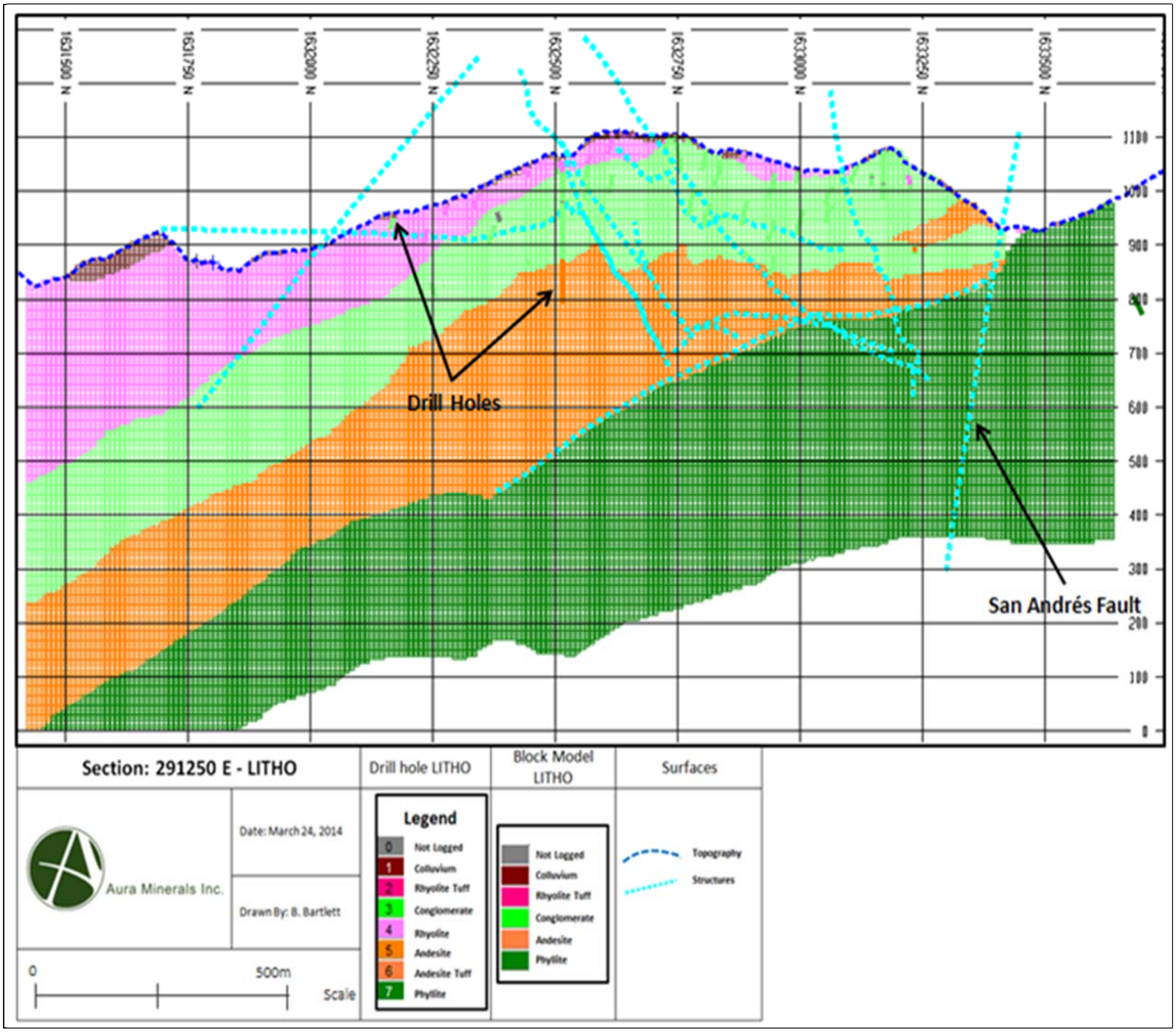


Figure 14-2: Bench Plan 920 mRL – Lithology and Structure

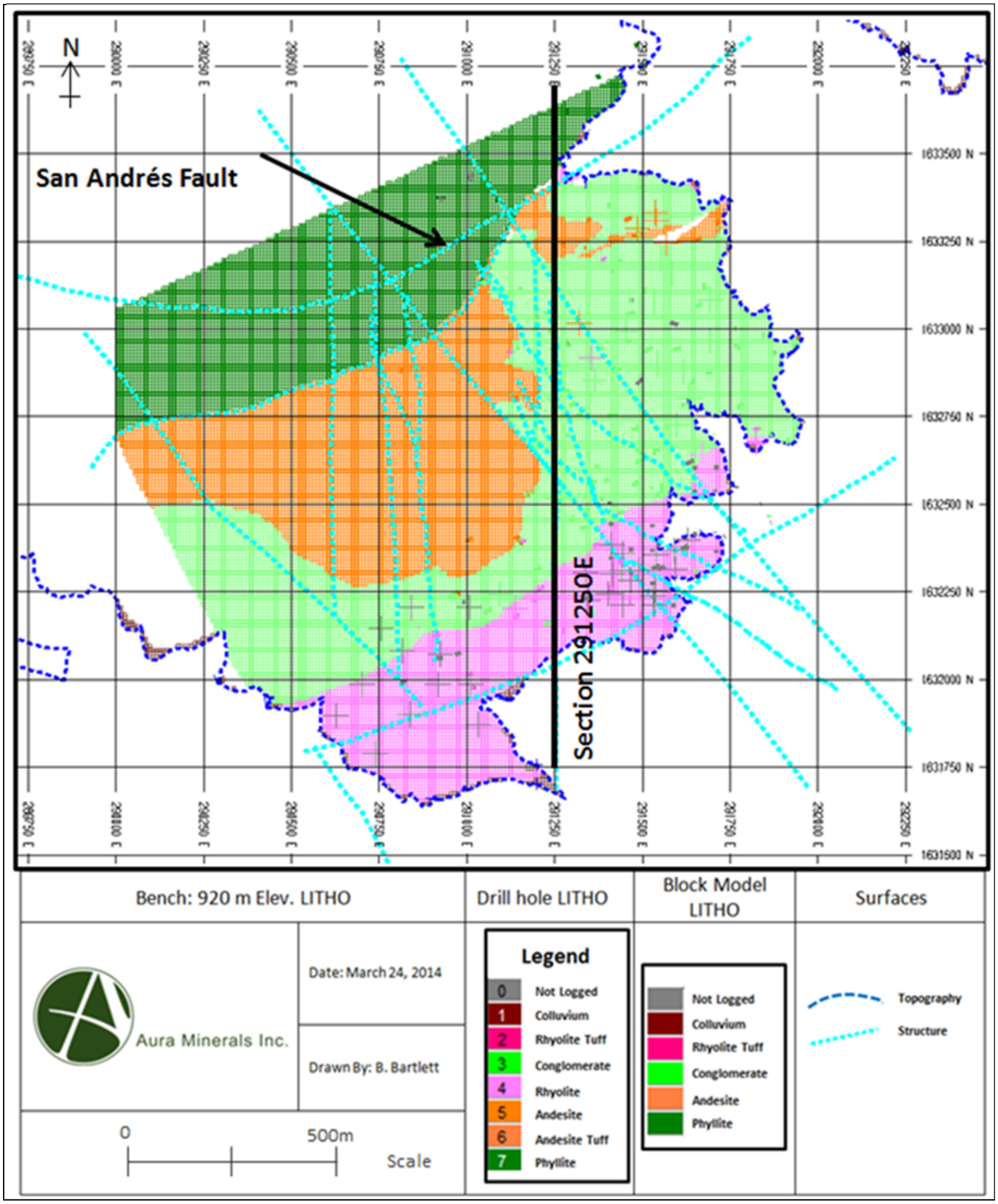


Figure 14-3: Bench Plan – 950 m Elev – Mineralization Domain Codes

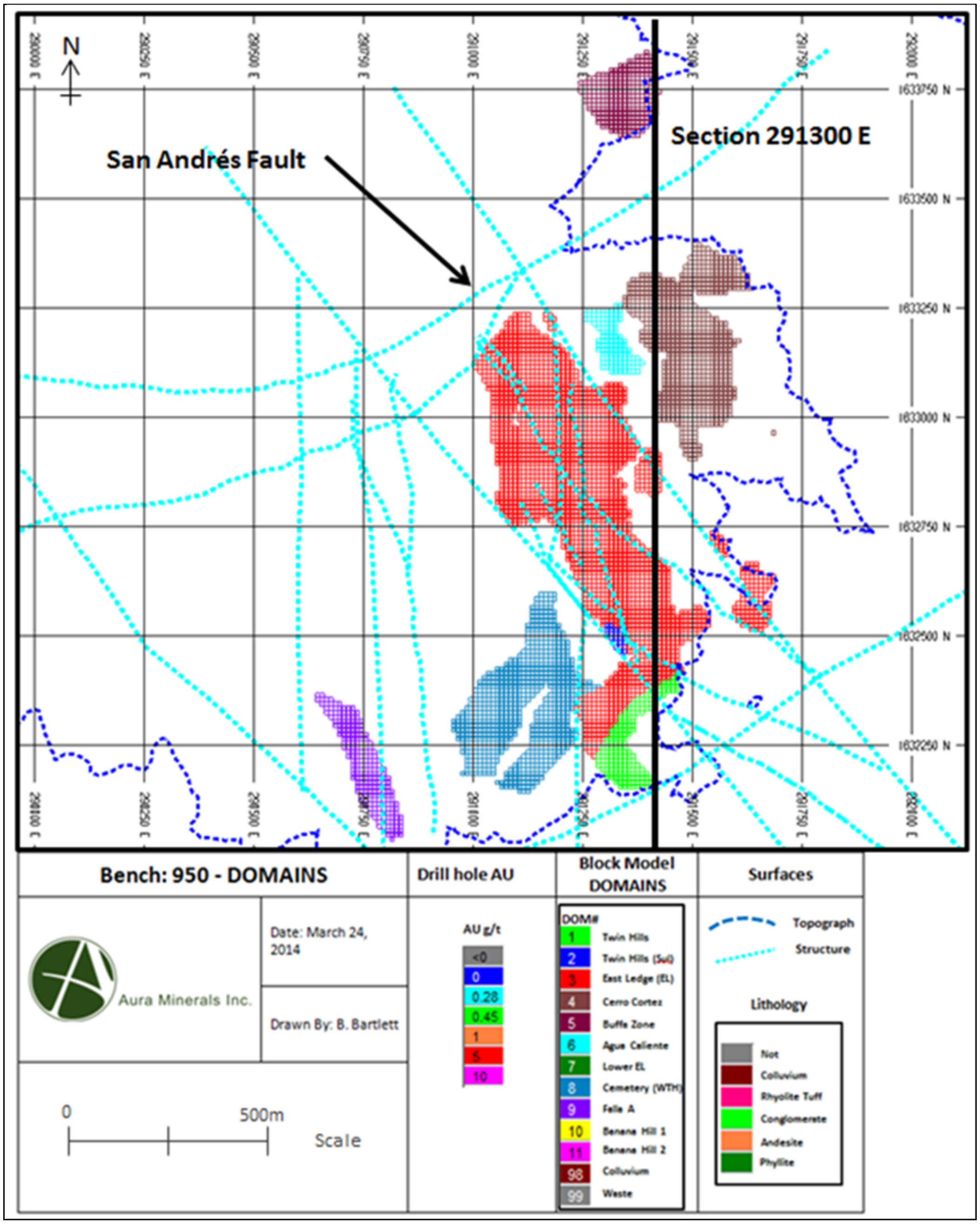
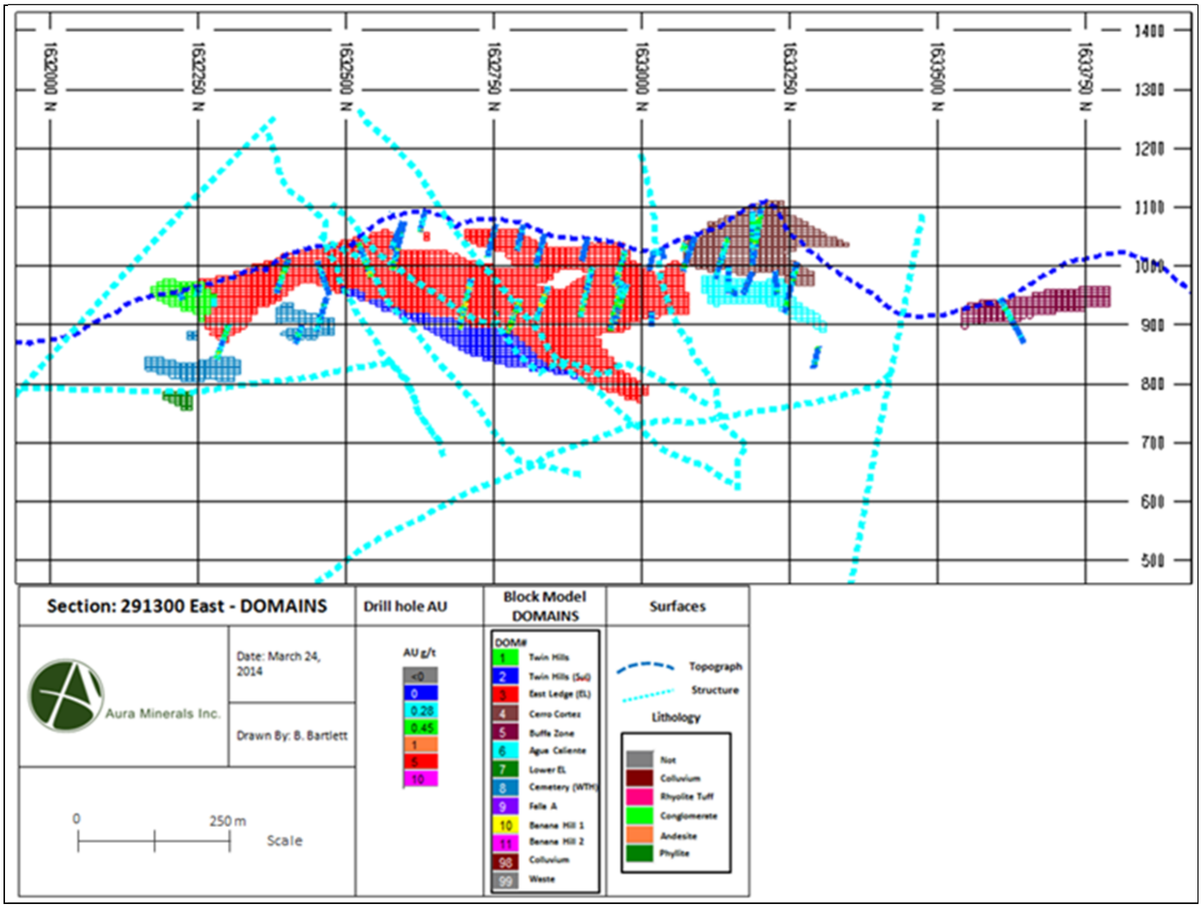


Figure 14-4: Section 291300 E Mineralization Domain Codes



The extent of the bottom of oxide, and the top of sulphide zones were also interpreted on vertical cross sections and linked as 3D surfaces, these used both logged oxide codes and where sufficient data was available, AuHCN (hot cyanide leach) assays to interpret the contact zones. These surfaces were then used to identify the oxide, mixed and sulphide categories for reporting the minerals resources. Figure 14-5 displays a bench plan at 950 m elevation showing the oxide, mixed and sulphide zones and Figure 14-6 displays a vertical cross section at 291400 E.

Figure 14-5: Section 291400 E Weathering Codes

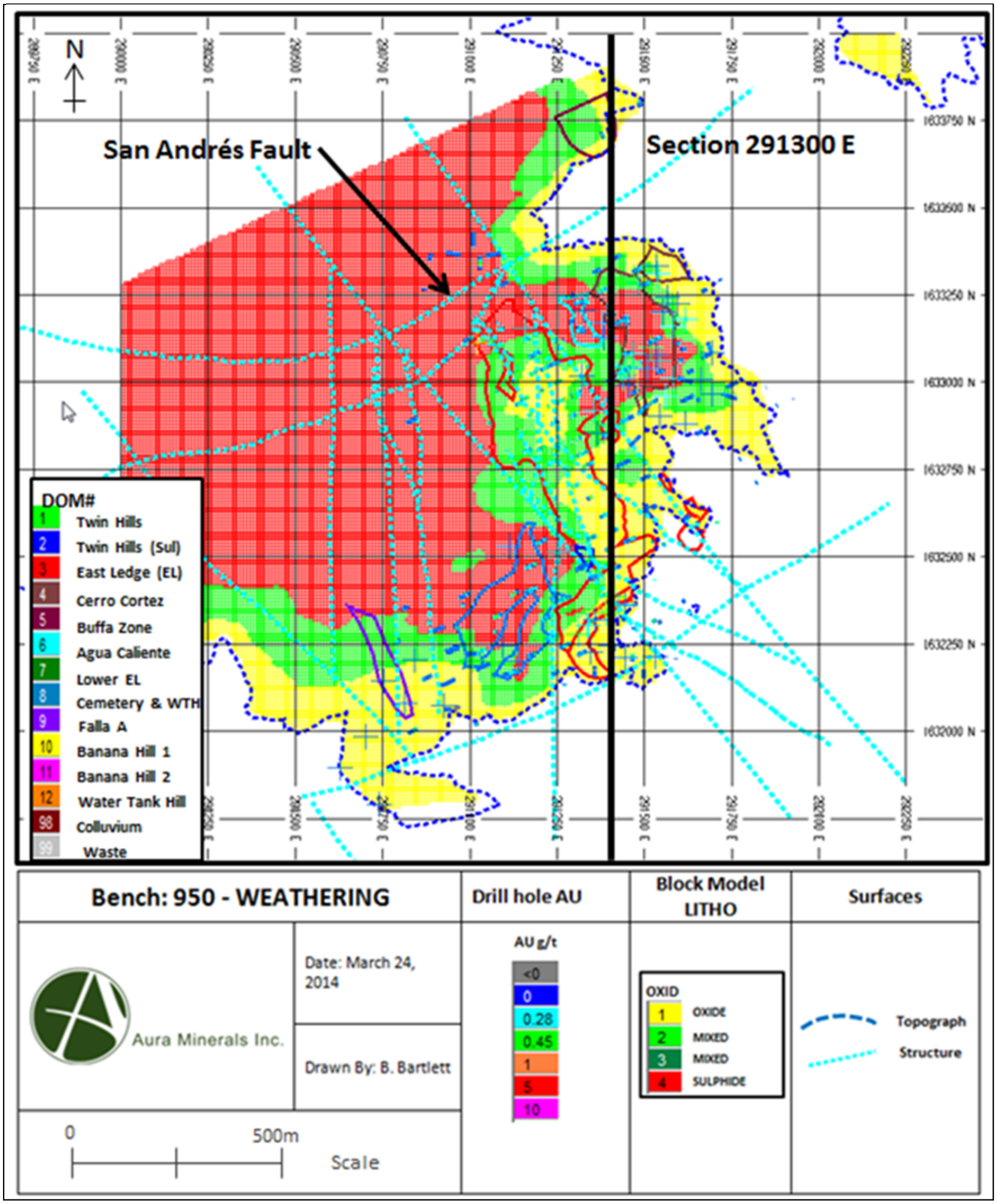
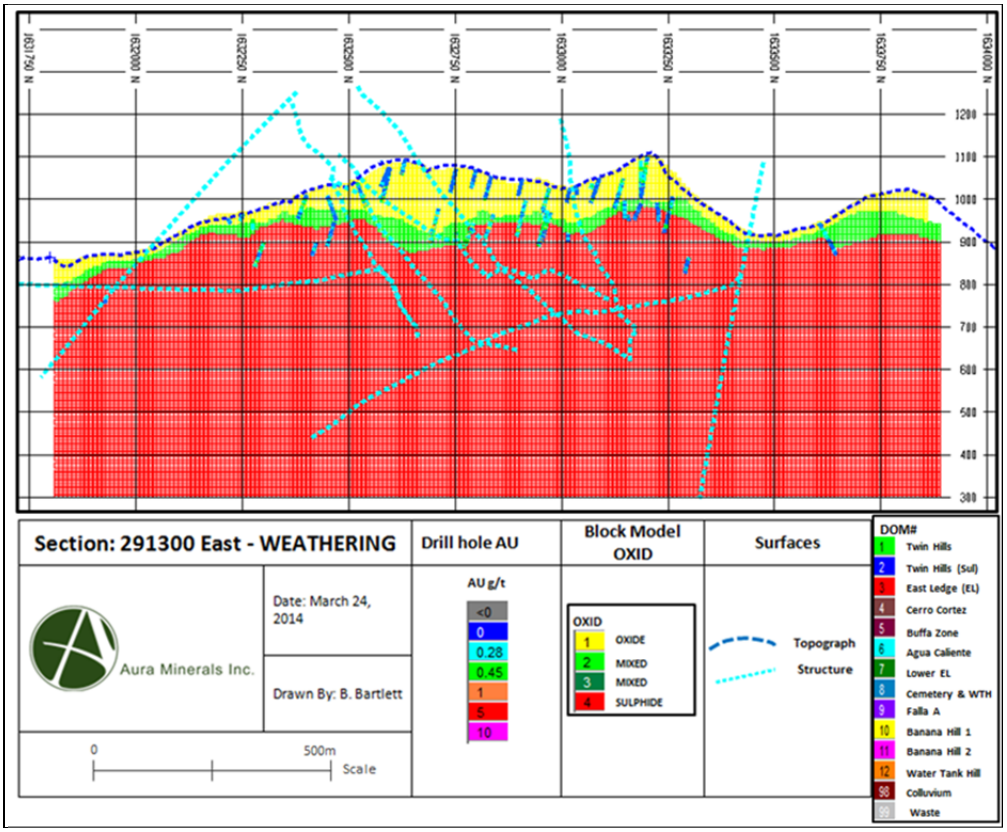


Figure 14-6: Section 291400 E Mineralization Domain Codes



14.3 Exploratory Data Analysis

Exploratory data analysis was completed using Snowden's Supervisor software package. Statistical and graphical summaries of the assays were produced to understand the gold distribution of the San Andrés gold deposit. Figure 14-7 displays a histogram showing the positively skewed histogram of gold assays, which is typical of this type of gold deposit. The mean gold grade is 0.58 g/t Au, with a Coefficient of Variation ("CV") of 1.41 and a maximum gold grade of 170.28g/t Au intersected in blast hole CC03301020, which is located in the Cerro Cortez area (Dom 4). For the exploration drill holes a maximum gold grade of 39.2 g/t Au intersected in blast hole SA-034, which is located in the Water Tank Hill area (Dom 8).

Table 14-1 displays a summary of the gold assay mean grade and CV for each mineralisation domain. Table 14-2 displays the breakdown by weathering (oxide, mixed and Sulphide). The gold assays in the oxide material have a lower mean grade than in the mixed material and the oxide zone has a higher CV. This is largely attributed to increased data distribution and closer spaced sampling in the production holes relative to the mixed populations. Figures 14-8 to 14-10 show histograms of the gold grades for the respective mineralisation domains. These histograms are also positively skewed with a long tail to the high-grade side of the histogram.

The changes to the weathering profiles during the production process and infill drilling indicate that there is likely to be a broad transition zone between layers and the similarity in the raw statistics between oxide and mixed indicate that there is little difference between the populations. This is further supported by grade distributions and variography evaluation indicating that mineralisation trends extend beyond sub horizontal weathering profiles. Therefore unlike previous resource estimations the mineralisation domains were not segregated into weathering profiles and were treated as one homogenous population within each of the domains.

Table 14-1: Sample Summary Statistics for Gold by Domain (g/t)

Domain	No. of Assays	Au (g/t)	
		Mean	CV
1	121,127	0.61	0.77
2	444	0.49	1.19
3	25,461	0.47	1.36
4	28,113	0.45	2.55
5	298	0.98	1.61
6	1,310	0.44	3.41
7	361	0.50	1.69
8	6,132	0.77	2.31
9	1,562	0.52	1.33
10	1,261	0.53	2.30
11	180	0.56	1.09
12	4,009	1.00	2.41
Total	**190,258**	**0.58**	**1.41**

1: Assays weighed by sample length.
2: Includes only assays inside the mineralized domains

Table 14-2: Weathering Statistics for Gold by Domain (g/t)

Weathering	No. Of Data points	Au (g/t)	
		Mean	CV
Oxide	104,405	0.56	1.43
Mixed	73,031	0.62	0.85
Sulphide	10,775	0.49	1.78
Ox-Mx	177,436	0.58	1.21
Colluvium	2,047	0.59	6.5

Figure 14-7: Histogram of RAW Gold Assays All Mineralisation Domains

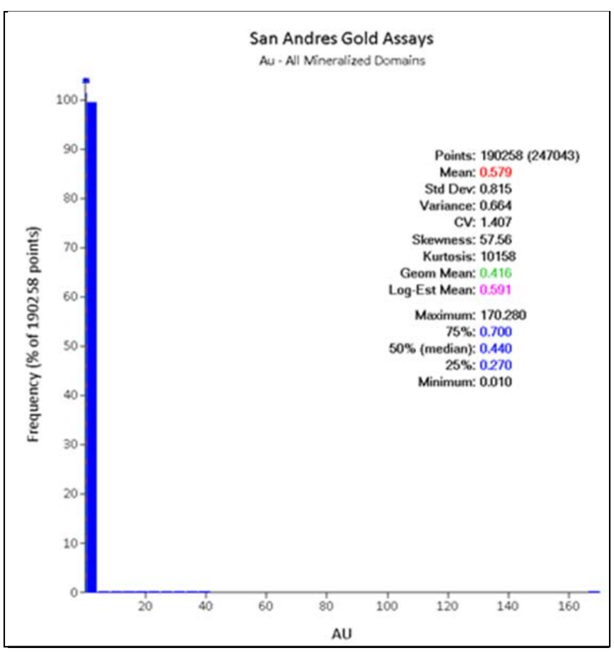


Figure 14-8: Histogram of Gold Assays – Domain 1 to 4

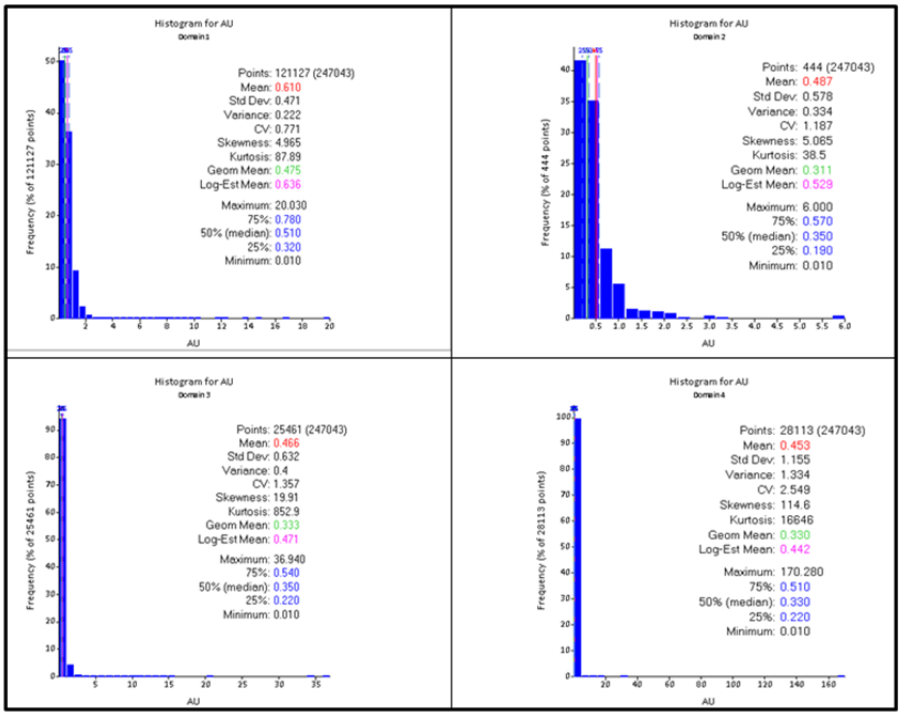


Figure 14-9: Histogram of Gold Assays – Domain 5 to 8

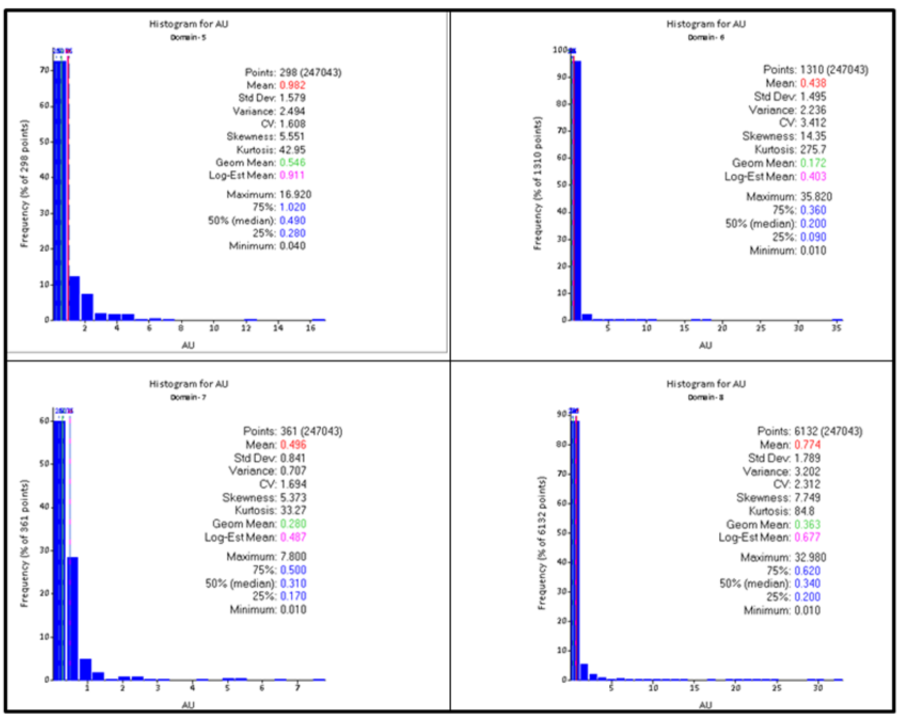
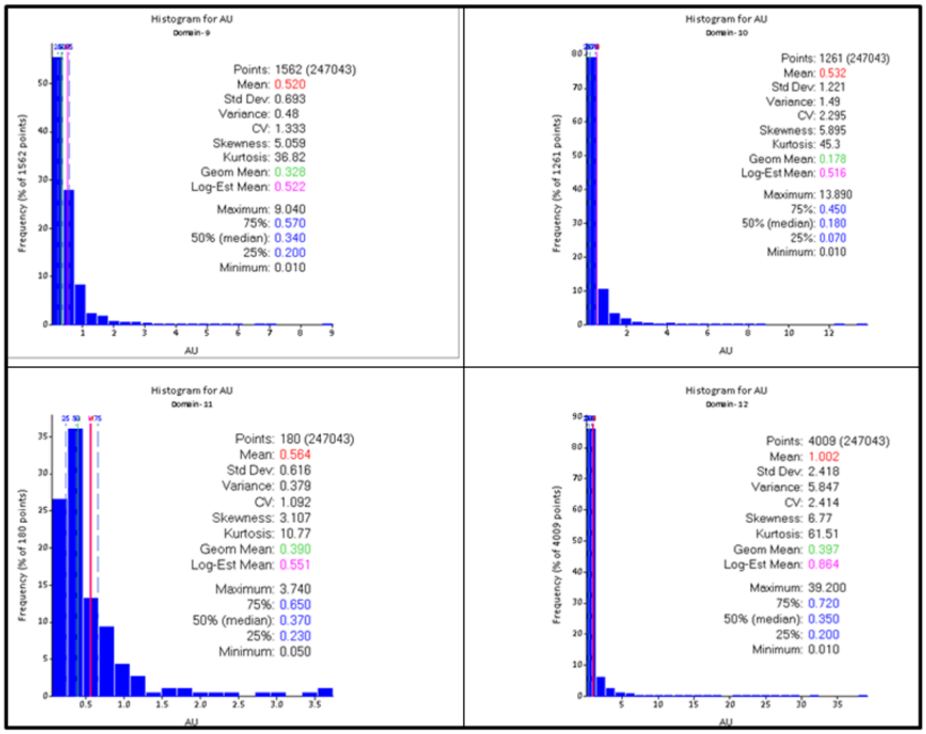


Figure 14-10: Histogram of Gold Assays – Domain 9 to 12



14.4 Composites

The main reason for compositing the original drill hole assay data is to normalize the assay grade lengths and the Mine data base consists of both exploration and production sample data, which have different mean sample lengths. The majority of the exploration data is sampled to 1.5 m, whereas the production data is sampled on 6 m lengths. Given the two distinct data sources and the variability in the mineralisation domains two composite lengths were chosen, 1.5 m and 6.0 m, the latter for Dom 1, 3 and 4, which have the largest proportion of production data. The data was then composited down-the-hole to either 1.5 m or 6.0 m with the composites broken at the domain code boundaries. Summary statistics of the 1.5 m and 6.0 m composites are shown in Table 14-3. In previous resource estimates the gold grades were composited to 6 m and a comparison table is shown in Table 14-4 which illustrates little variance between the mean and CV between 6 and 1.5 m composites.

There is minimal difference in the means of the composite data versus the raw assays, 0.63 g/t vs. 0.62 g/t. However the CV (the standard deviation divided by the mean) is greatly reduced over all the domains except Dom 8 and 10. Individual domains have highly variable CV's resulting from their data and grade distributions; this was largely masked in previous estimates by combining the data into three large mineralisation domains based on weathering horizons. Overall the moderate CV indicates that the grade distribution of the composites is not strongly affected by high-grade outliers and this was validated by log probability plots for each domain, however there were minimal capping values applied to a number of domains.

Figure 14-11 displays a histogram of the 1.5 m composite gold grades. The distribution of the composite gold grades is also still positively skewed, but not as much as the raw assay gold grades.

Table 14-3: Composite Summary Statistics for Gold (g/t)

Description	Domain	Composite Length (m)	No. of Comps	Au (g/t)	
				Mean	CV
Twin Hills	1	6	113,250	0.612	0.742
Twin Hills Sulphide	2	1.5	421	0.477	1.09
East Ledge	3	6	16,359	0.437	0.917
Cerro Cortez	4	6	24,583	0.466	2.611
Zona Buffa	5	1.5	310	0.994	1.365
Lower Cerro Cortez	6	1.5	1294	0.351	1.865
Lower Twin Hills	7	1.5	315	0.414	0.987
Cemetery – Water Tank Hill	8	1.5	6249	0.772	2.276
Fault A	9	1.5	1607	0.519	1.266
Banana Ridge1	10	1.5	1228	0.534	2.214
Banana Ridge2	11	1.5	181	0.58	1.128
Water Tank Hill	12	1.5	4107	0.98	2.172
All Domains		-	169,904	0.59	1.15

1: Composites weighted by length,
2: Excludes 3,038 non assayed intervals set to absent value.

Table 14-4: Raw vs. Composite Statistics for Gold (g/t)

Domain	Samples			Mean			CV		
	RAW	6m	1.5m	RAW	6m	1.5m	RAW	6 m	1.5 m
1	121,127	113,250		0.61	0.621		0.77	0.742	
2	444		421	0.49		0.477	1.19		1.09
3	25,461	16,359		0.47	0.437		1.36	0.917	
4	28,113	24583		0.45	0.466		2.55	2.611	
5	298		310	0.98		0.994	1.61		1.365
6	1,310		1294	0.44		0.351	3.41		1.865
7	361		315	0.50		0.414	1.69		0.987
8	6,132		6249	0.77		0.772	2.31		2.276
9	1,562		1607	0.52		0.519	1.33		1.266
10	1,261		1228	0.53		0.534	2.30		2.214
11	180		181	0.56		0.58	1.09		1.128
12	4,009		4107	1.00		0.98	2.41		2.172
Total	**190,258**			**0.58**			**1.41**		

Figure 14-11: Histogram of Gold 1.5 m Composites - Domain 2, 4 to 11

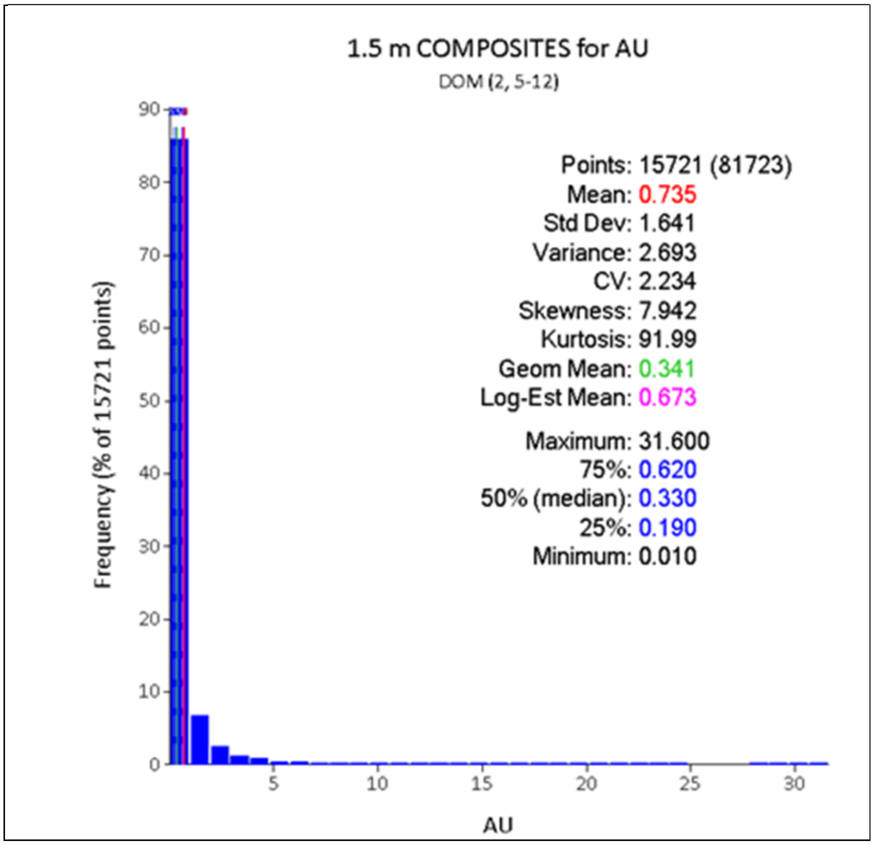
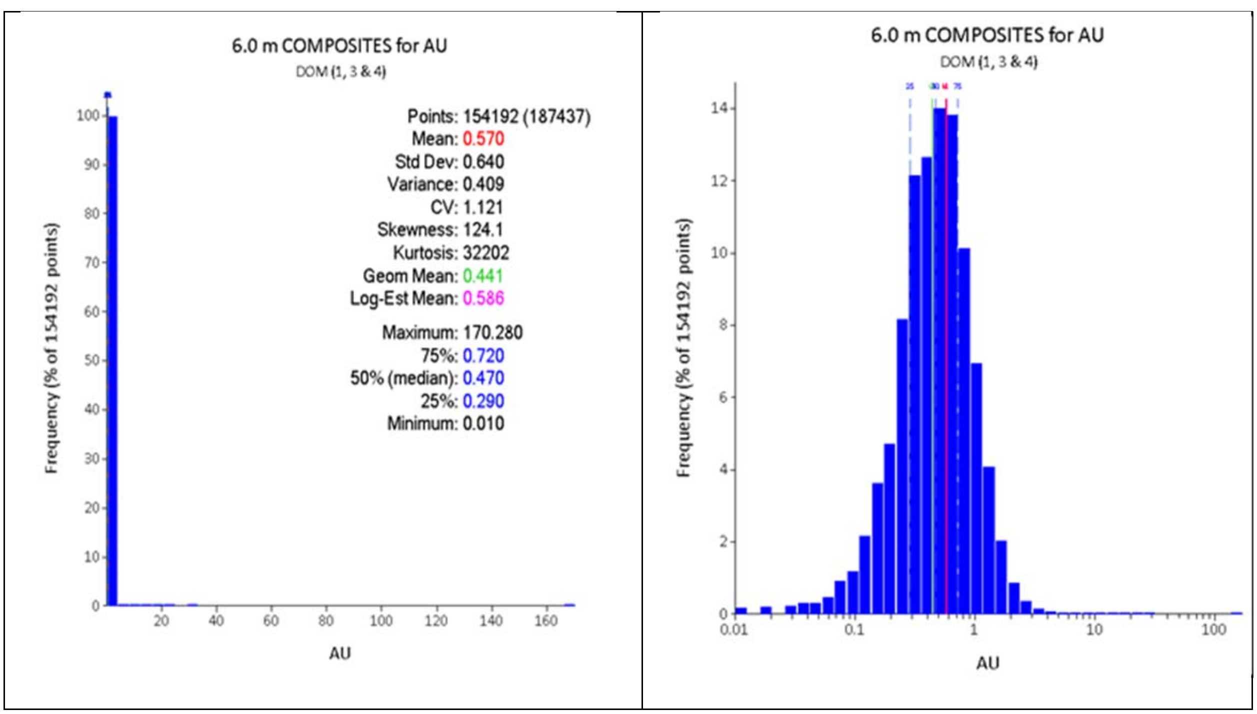


Figure 14-12: Histograms of Gold 6.0 m Composites – Domain 1 and 3



14.5 Variography Analyses

Variograms were calculated for 8 of the domains with four domains; 2, 5, 7 and 11 containing too few data points to accurately produce reliable variograms. The modelled variogram parameters are detailed in Table 14-5 used in the kriging estimate. Two examples of the variogram models and ellipses orientations are illustrated in Figure 14-13 and Figure 14-14.

The variograms were fitted with both two or three nested spherical models and a nugget effect. The nugget effect ranges between 26% and 59% of the total variability, indicating a good to average spatial correlation for closely spaced composites and illustrates the significant variances in grade distributions between mineralization domains. These are typical of many gold epithermal deposits.

Table 14-5: Variogram Model Parameters for Kriging Au Composites

Domain 1							
Direction	**C_0**	**C_1**	**A_1**	**C_2**	**A_2**	**C_3**	**A_3**
-008-->040	0.29	0.26	30	0.4	70	0.05	130
006-->310	0.29	0.26	30	0.4	70	0.05	130
-080-->260	0.29	0.26	16	0.4	20	0.05	40
C_0=nugget effect, C_1=first contribution, C_2=second contribution, C_3=third contribution							
A_1=range of first structure (m), A_2=range of second structure (m), A_3=range of third structure (m)							
Rotation							
	Type	**Angle 1**	**Angle 2**	**Angle 3**			
MineSight	ZXY	39.6	-7.6	-6.5			
Domain 3							
Direction	C_0	C_1	A_1	C_2	A_2	C_3	A_3
000-->010	0.53	0.28	25	0.13	65	0.06	200
020-->280	0.53	0.28	25	0.13	65	0.06	105
-070-->280	0.53	0.28	7	0.13	15	0.06	30
C_0=nugget effect, C_1=first contribution, C_2=second contribution, C_3=third contribution							
A_1=range of first structure (m), A_2=range of second structure (m), A_3=range of third structure (m)							
Rotation							
	Type	**Angle 1**	**Angle 2**	**Angle 3**			
MineSight	ZXY	10	0	-20			
Domain 4							
Direction	**C_0**	**C_1**	**A_1**	**C_2**	**A_2**	**C_3**	**A_3**
-16-->072	0.42	0.38	17	0.08	40	0.13	100
50-->002	0.42	0.38	17	0.08	40	0.13	100
35-->150	0.42	0.38	12	0.08	25	0.13	60
C_0=nugget effect, C_1=first contribution, C_2=second contribution, C_3=third contribution							
A_1=range of first structure (m), A_2=range of second structure (m), A_3=range of third structure (m)							
Rotation							
	Type	**Angle 1**	**Angle 2**	**Angle 3**			
MineSight	ZXY	71.8	-16.3	-53.3			
Domain 6							
Direction	**C_0**	**C_1**	**A_1**	**C_2**	**A_2**	**C_3**	**A_3**
02-->300	0.59	0.3	80	0.11	260		
20-->210	0.59	0.3	80	0.11	260		
70-->035	0.59	0.3	10	0.11	20		
C_0=nugget effect, C_1=first contribution, C_2=second contribution, C_3=third contribution							
A_1=range of first structure (m), A_2=range of second structure (m), A_3=range of third structure (m)							

Rotation							
	Type	**Angle 1**	**Angle 2**	**Angle 3**			
MineSight	ZXY	-59.7	1.7	-19.9			
Domain 8							
Direction	**C_0**	**C_1**	**A_1**	**C_2**	**A_2**	**C_3**	**A_3**
00-->330	0.35	0.37	15	0.15	75	0.14	300
20-->240	0.35	0.37	10	0.15	25	0.14	60
70-->060	0.35	0.37	10	0.15	25	0.14	40
C_0=nugget effect, C_1=first contribution, C_2=second contribution, C_3=third contribution							
A_1=range of first structure (m), A_2=range of second structure (m), A_3=range of third structure (m)							
Rotation							
	Type	**Angle 1**	**Angle 2**	**Angle 3**			
MineSight	ZXY	-30	0	-20			
Domain 9							
Direction	**C_0**	**C_1**	**A_1**	**C_2**	**A_2**	**C_3**	**A_3**
07-->348	0.48	0.4	90	0.12	231		
-29-->262	0.48	0.4	25	0.12	60		
60-->245	0.48	0.4	10	0.12	20		
C_0=nugget effect, C_1=first contribution, C_2=second contribution, C_3=third contribution							
A_1=range of first structure (m), A_2=range of second structure (m), A_3=range of third structure (m)							
Rotation							
	Type	**Angle 1**	**Angle 2**	**Angle 3**			
MineSight	ZXY	-11.9	7.4	29.1			
Domain 10							
Direction	**C_0**	**C_1**	**A_1**	**C_2**	**A_2**	**C_3**	**A_3**
-008-->076	0.33	0.49	50	0.18	150		
049-->355	0.33	0.49	25	0.18	75		
-040-->340	0.33	0.49	8	0.18	24		
C_0=nugget effect, C_1=first contribution, C_2=second contribution, C_3=third contribution							
A_1=range of first structure (m), A_2=range of second structure (m), A_3=range of third structure (m)							
Rotation							
	Type	**Angle 1**	**Angle 2**	**Angle 3**			
MineSight	ZXY	76.5	-7.6	-49.6			
Domain 12							
Direction	C_0	C_1	A_1	C_2	A_2	C_3	A_3
-04-->334	0.28	0.33	20	0.22	50	0.16	90
45-->247	0.28	0.33	15	0.22	45	0.16	60
45-->060	0.28	0.33	6	0.22	25	0.16	35
C_0=nugget effect, C_1=first contribution, C_2=second contribution, C_3=third contribution							
A_1=range of first structure (m), A_2=range of second structure (m), A_3=range of third structure (m)							
Rotation							
	Type	**Angle 1**	**Angle 2**	**Angle 3**			
MineSight	ZXY	-26.5	-3.5	-44.9			

Figure 14-13: Dom 4 Variograms AU

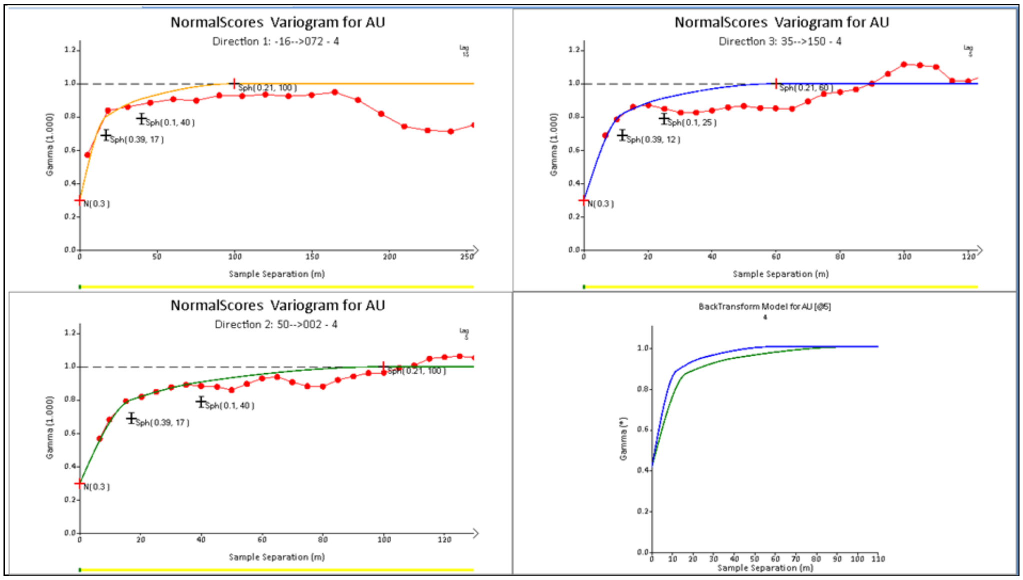
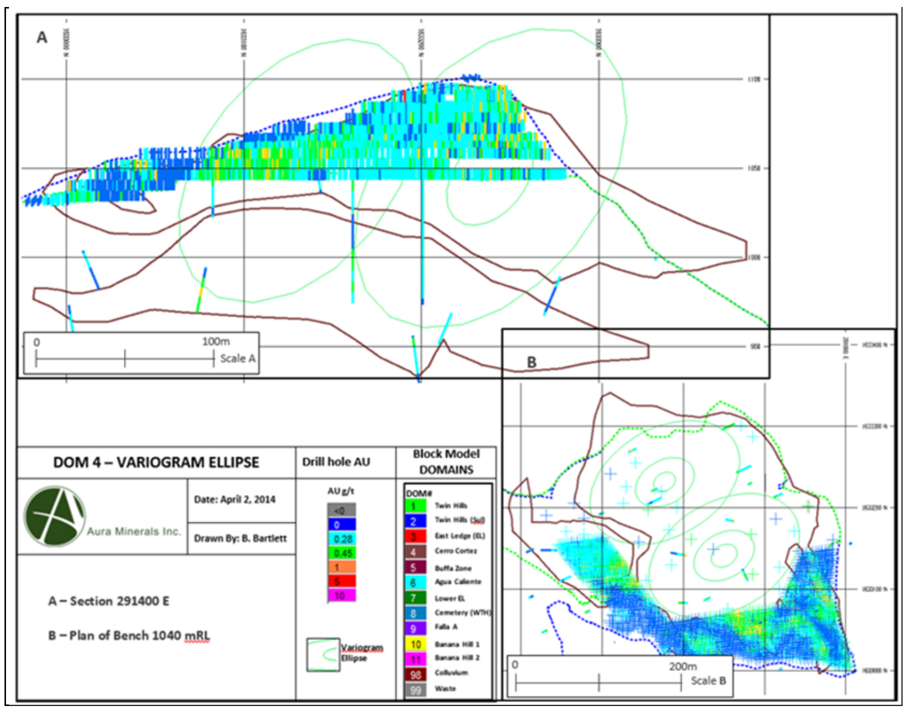


Figure 14-14: Dom 4 Variograms Ellipse Diagrams



The variography study resulted in 4 domains with very poor variogram maps to accurately define the dominant directions, these include; two sulphide domains (not included in the Mineral Resource), 2 (97% Sulphide) and 7 (100% Sulphide), the Buffa Zone, domain 5 and domain 11 in the Banana Hill area. All of these domains combined constitute 3.8% of the total mineral resource therefore are not a material proportion. For these domains, variograms and searches were defined on the mineralisation domain orientation, size and relative anisotropic relationships and where appropriate variograms used parameters from similar domains.

To validate this approach, an inverse distance squared estimate was completed along with swath and QQ plots and comparative estimation statistics to ensure that the variogram parameters used were not biasing the estimate and provided a reasonable representation of the grade distribution within each domain. Table 14-6 presents a summary of the variogram model used for kriging of the composite gold grades for domains 2, 5, 7 and 11. Variogram ellipses diagrams for domain 5 are detailed in figure 14-15, along with a Swath plot for the corresponding northings in Figure 14-16. The QQ plot for AUK and AUID in Figure 14-17 illustrates clearly that the AUK estimate is conservative to the AUID with the mean grade of 0.83 g/t for AUK compared to 0.92 g/t for the AUID and 0.92 g/t for the drill hole composite grade.

In the Authors' opinion the approach to applying reasonable variogram parameter has not biased the estimate for domains 2, 7, 5 and 11; however the Authors recommend that additional data is required in these domains to reliably determine the close spaced grade variability.

Table 14-6: Variogram Model Parameters for kriging

Domain 2							
Direction	**C_0**	**C_1**	**A_1**	**C_2**	**A_2**	**C_3**	**A_3**
-008-->040	0.43	0.39	20	0.1	54	0.08	180
006-->310	0.43	0.39	20	0.1	49	0.08	125
-080-->260	0.43	0.39	6	0.1	30	0.08	50
C_0=nugget effect, C_1=first contribution, C_2=second contribution, C_3=third contribution							
A_1=range of first structure (m), A_2=range of second structure (m), A_3=range of third structure (m)							
Rotation							
	Type	**Angle 1**	**Angle 2**	**Angle 3**			
MineSight	ZXY	-120	50	0			
Domain 5							
Direction	C_0	C_1	A_1	C_2	A_2	C_3	A_3
00-->035	0.26	0.46	100	0.28	150		
40-->305	0.26	0.46	100	0.28	150		
50-->125	0.26	0.46	20	0.28	30		
C_0=nugget effect, C_1=first contribution, C_2=second contribution, C_3=third contribution							
A_1=range of first structure (m), A_2=range of second structure (m), A_3=range of third structure (m)							
Rotation							
	Type	**Angle 1**	**Angle 2**	**Angle 3**			
MineSight	ZXY	35	0	-40			
Domain 7							
Direction	**C_0**	**C_1**	**A_1**	**C_2**	**A_2**	**C_3**	**A_3**
00-->025	0.29	0.49	20	0.22	50		
20-->295	0.29	0.49	20	0.22	50		
70-->115	0.29	0.49	5	0.22	15		
C_0=nugget effect, C_1=first contribution, C_2=second contribution, C_3=third contribution							
A_1=range of first structure (m), A_2=range of second structure (m), A_3=range of third structure (m)							
Rotation							
	Type	**Angle 1**	**Angle 2**	**Angle 3**			
MineSight	ZXY	25	0	-20			
Domain 11							
Direction	**C_0**	**C_1**	**A_1**	**C_2**	**A_2**	**C_3**	**A_3**
	0.33	0.44	11	0.24	80		
	0.33	0.44	11	0.24	80		
	0.33	0.44	5	0.24	15		
C_0=nugget effect, C_1=first contribution, C_2=second contribution, C_3=third contribution							
A_1=range of first structure (m), A_2=range of second structure (m), A_3=range of third structure (m)							
Rotation							
	Type	**Angle 1**	**Angle 2**	**Angle 3**			
MineSight	ZXY	-10	0	-15			

Figure 14-15: Dom 5 Variograms Ellipse Diagrams

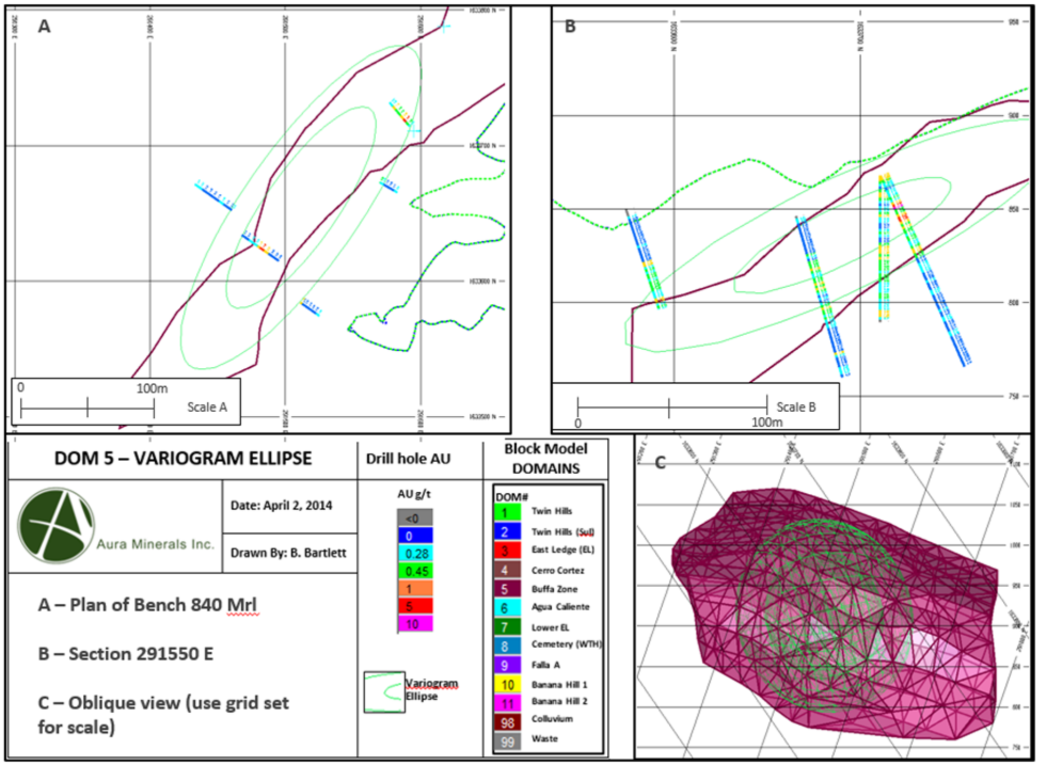


Figure 14-16: Dom 5 Swath Plot – Northing

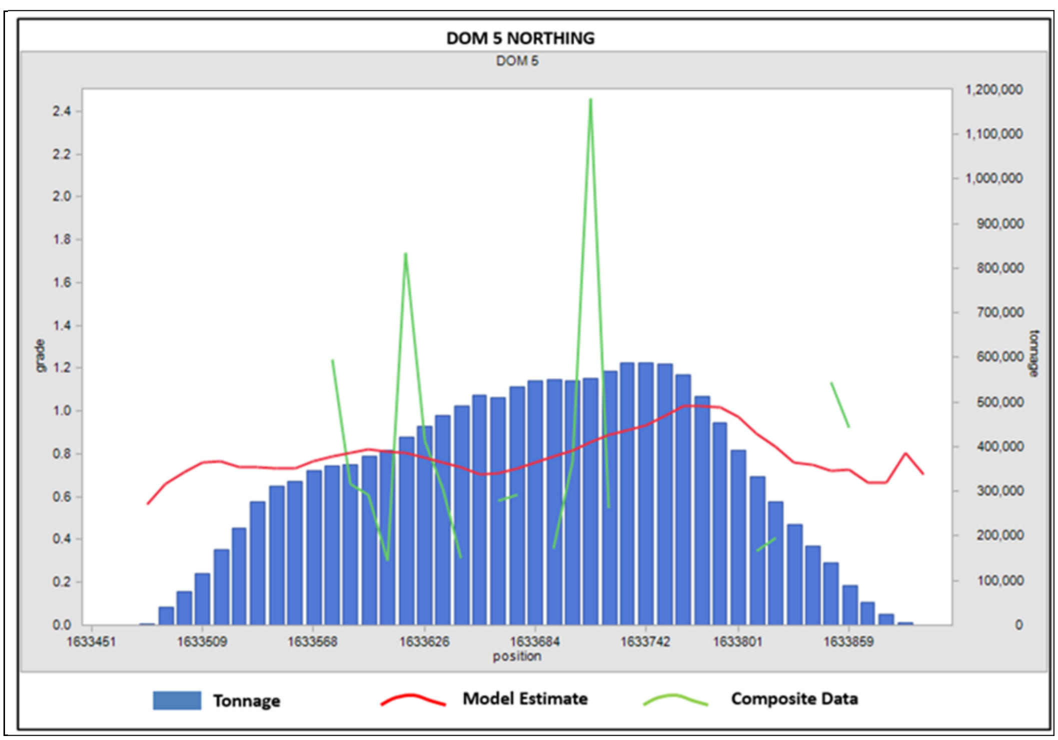
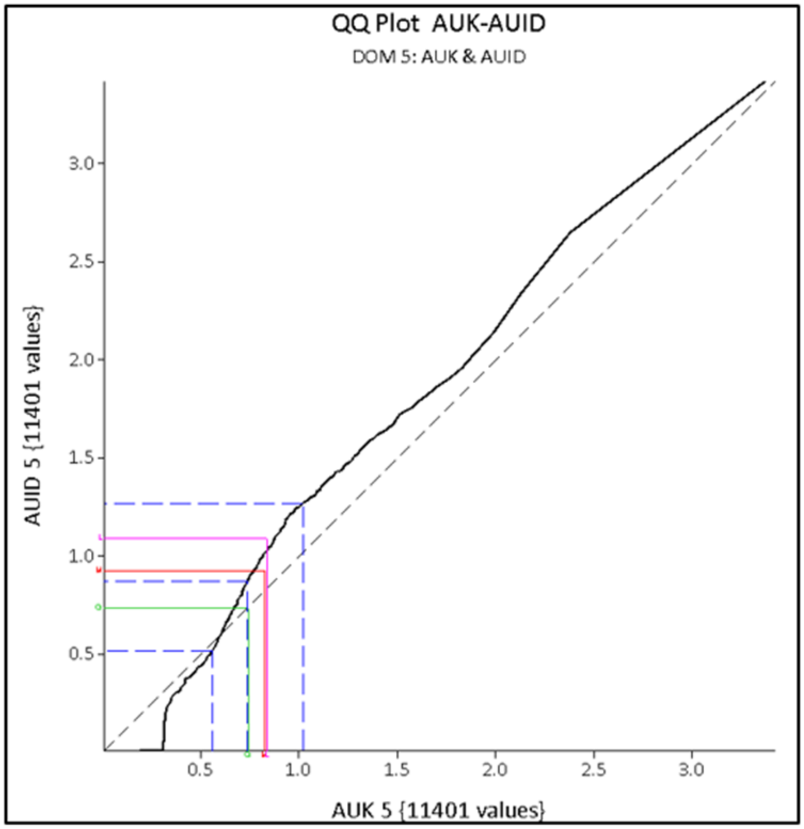


Figure 14-17: Dom 5 QQ Plot AUK vs. AUID



14.6 Search Parameters

On completion of the variograms the search ellipse parameters were chosen using the Variogram anisotropy distances as a guideline for defining two sets of search parameters. Two estimation runs were completed to utilise a close spaced search and a wide spaced search to populate the mineralisation domains. The first search ellipse was defined using 2x; the variogram ranges with the second search ellipse (close spaced) based on the maximum variogram ranges. The two composite lengths used; 1.5 m and 6.0 m impacted on the number of samples selected for the estimate and these are detailed in Table 14-7 with Table 14-8 and Table 14-9 detailing the search ellipse parameters for each estimation run.

For the waste domain (DOM 99) only a single run was completed and DOM 3 parameters were utilised for the search and variogram parameters.

The composite data sets used in the estimation contained a variety of drill spacing from close spaced production blast holes to wide spaced exploration drill holes. In order to account for the clustering of data, limits were placed on the number of samples available per hole and per octant. This approach will limit the impact of clustered data.

Table 14-7: Block Model AUK & AUID Sample Selection Parameters

Domain	Composite Length	Pass 1				Pass 2			
		Min #	Max #	Min Comps / Hole	Max Comps / Octant	Min #	Max #	Min Comps / Hole	Max Comps / Octant
1	6	4	20	2	4	6	20	3	4
2	1.5	4	35	4	6	6	35	4	6
3	6	4	20	2	4	6	20	3	4
4	6	4	20	2	4	6	20	3	4
5	1.5	4	35	4	6	6	35	4	6
6	1.5	4	35	4	6	6	35	4	6
7	1.5	4	35	4	6	6	35	4	6
8	1.5	4	35	4	6	8	35	4	6
9	1.5	4	35	4	6	6	35	4	6
10	1.5	4	35	4	6	8	35	4	6
11	1.5	4	35	4	6	6	35	4	6
12	1.5	4	35	4	6	8	35	4	6
99	6	4	20	2	4	6	20	3	4

Table 14-8: Search #1 Parameters

Domain	Angle 1 (Major)	Angle 2 (Semi-Mjr)	Angle 3 (Minor)	Distance 1 (Major)	Distance 2 (Semi-Mjr)	Distance 3 (Minor)
1	39.6	-7.6	-6.5	260	260	80
2	-120	50	0	360	250	100
3	10	0	-20	400	210	60
4	71.8	-16.3	-53.5	200	200	120
5	35	0	-40	300	300	60
6	-59.7	1.7	-19.9	520	520	40
7	25	0	-20	100	100	30
8	-30	0	-20	150	50	50
9	-11.9	7.4	29.1	460	120	40
10	76.5	-7.6	-49.6	300	150	48
11	-10	0	-15	160	160	30
12	-26.5	-3.5	-44.9	180	120	70
99 (Waste)	10	0	-20	400	210	60

Table 14-9: Search #2 Parameters

Domain	Angle 1 (Major)	Angle 2 (Semi-Mjr)	Angle 3 (Minor)	Distance 1 (Major)	Distance 2 (Semi-Mjr)	Distance 3 (Minor)
1	39.6	-7.6	-6.5	130	130	40
2	-120	50	0	180	125	50
3	10	0	-20	200	105	30
4	71.8	-16.3	-53.5	100	100	60
5	35	0	-40	150	150	30
6	-59.7	1.7	-19.9	260	260	20
7	25	0	-20	50	50	15
8	-30	0	-20	75	25	25
9	-11.9	7.4	29.1	230	60	20
10	76.5	-7.6	-49.6	150	75	24
11	-10	0	-15	80	80	15
12	-26.5	-3.5	-44.9	90	60	35
99 (Waste)	Not re-estimated					

The search parameters resulted in a small number of blocks (<0.005%) on the periphery of Domains 2, 3, 5, 8 and 10. For all missing AUK estimates an assigned value of 0.01 was applied, ensuring that no blocks outside of the assigned search parameters were included in the mineral resource. In the case for Domain 5 the missing values were also used to define the approximate limits of the classification between indicated and inferred, with much of the inferred portion of Domain 5 outside of the mining permit area (<200 m of Agua Caliente River), thus these blocks are not included within the resource.

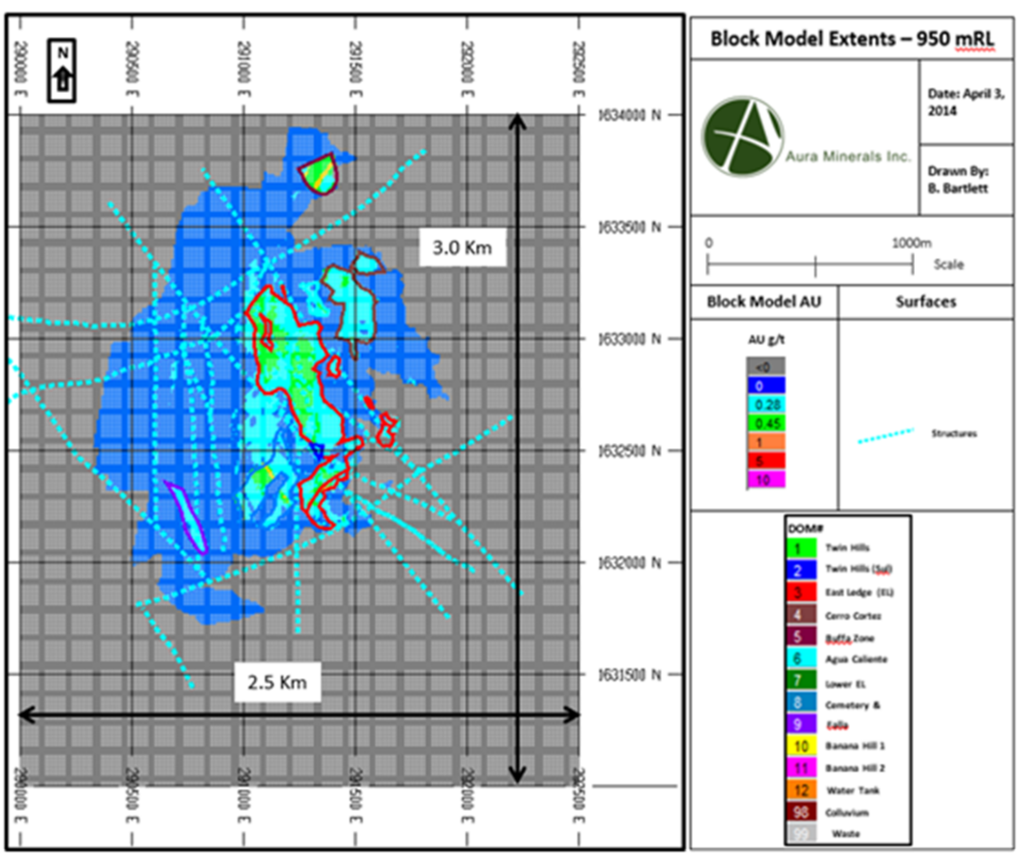
14.7 Block Model

The block model limits were defined using UTM coordinates and the block size was 10 m x 10 m x 6 m. The block size was selected to properly discretize the rock code 3D wireframes and approximately represents the level of selectivity considered appropriate for the mine production rate. This block size is also the same as that used in previous resource estimates. Table 14-10 shows the block model limits and Figure 18 displays the block model limits and AU grades with structures and domain solids.

Table 14-10: Block Model Parameters (MineSight)

	East	North	Elevation
Minimum (m)	290,000	1,631,000	0
Maximum (m)	292,500	1,634,000	1,296
Block Size (m)	10	10	6
Number of blocks	250	300	216
Rotation (degrees)	0	0	0

Figure 14-18: Block Model Area



The block model variable DOM was coded using the domain code 3D wireframe solids along with LITHO (Lithology), ALTN (alteration), OXID (weathering), MINE (Current mine surface). As well, the DOM code 3D wireframes were used to code the percentage of a block that was inside these wireframes. These partial blocks were coded in the block model item ORE%. The block model was also coded with the topography (item TOPO %), such that a block completely below the topography surface was given a value of 100%, while a block completely above the topography surface was given a value of zero percent.

A constant density of 2.25 t/m^3 was assigned to all blocks in the block model.

14.8 Grade Estimation

The methodology chosen for the mineral estimation is unchanged to previous resource estimation by Minosa and its consultant and the fields AUK and AUHK in the block model (sa15.014) were estimated using Ordinary kriging. The block model used a block discretization of 4 x 4 x 2 and the block grades were estimated utilizing the following constraints:

- two estimation passes
- 1.5 m and 6.0 m drill hole composite files (fdh209.dat & fdh609.dat)
- drill hole composites coded by DOM
- model blocks coded by DOM
- 12 domain defined variogram parameter files.

The variogram model parameters used are shown in Tables 14-5 and Table 14-6. The search parameters are shown in Table 14-7 to Table 14-9.

An Inverse distance squared (AUID) estimation was completed for all domains as part of the validation checks and comparison evaluation.

Upon completion of the estimate the block model was checked for missing values. All missing AUK and AUID estimates (<0.005% of DOM coded blocks) were re-assigned to 0.01 g/t as they fell outside of the wide spaced search criteria. For domains where mineralisation interpretations were projected above the TOPO surface, blocks where TOPO=100%, AUK and AUID were re-assigned an absent value.

14.9 Block Model Validation

Validation of the block model estimates was carried out using a series of checks that include a global bias check, a global variance check, a trend check, block estimation parameter checks, reconciliation against the 2012 production and a visual validation in plan and section.

14.9.1 Global Bias Check

The global bias check is carried out by comparing the mean grade of the declustered composites to the mean grade of the OK estimates. For this check, the inverse distance squared estimates (ID^2) will be used as the declustered mean. Table 14-11 shows the mean grade of the OK estimates for all rock codes is 0.50 g/t Au (see column "OK AU Mean"). The mean grade of the ID^2 estimates for all rock codes is 0.47 g/t Au. There is a good comparison between the two estimates, indicating that the OK estimates are globally unbiased. The CV ratio between "OK AU CV" and "ID^2 AU CV" for all the estimates is 0.83, which indicates an appropriate global change in variance.

Table 14-11: Summary Statistics for Block Model Estimates

Domain	No. of Blocks	OK AU Mean	OK AU CV	No. of Blocks	ID2 AU Mean	ID2 AU CV	Mean Ratio	CV Ratio
1	60,904	0.54	0.445	60,904	0.54	0.474	1.00	0.94
2	22,888	0.46	0.298	22,888	0.43	0.569	1.07	0.52
3	95,467	0.51	0.339	95,467	0.49	0.443	1.04	0.77
4	27,780	0.37	0.562	27,780	0.37	0.52	1.00	1.08
5	11,401	0.83	0.478	11,401	0.92	0.543	0.90	0.88
6	34,673	0.39	0.463	34,673	0.37	0.658	1.05	0.70
7	6,511	0.39	0.493	6,511	0.26	0.884	1.50	0.56
8	39,863	0.5	0.93	39,863	0.36	0.747	1.39	1.24
9	13,975	0.44	0.499	13,975	0.31	0.752	1.42	0.66
10	5,081	0.51	0.686	5,081	0.56	1.006	0.91	0.68
11	1057	0.52	0.372	1057	0.29	0.84	1.79	0.44
12	12741	0.68	1.259	12741	0.67	1.287	1.01	0.98
All Domains	332,341	0.5	0.637	332,341	0.47	0.764	1.06	0.83

14.9.2 Trend Check

Local trends in the grade estimates were assessed via swath plots for each domain code. The average estimate for the ID^2 and OK models for 20 m wide swaths in easting and northing were compared with the composite data used to derive the estimates. No elevation swath plots were generated due to the shallow nature of the mineralisation domains.

Figure 14-19 and Figure 14-20 display a representation of the swath plots for the easting and northing directions for Domain 4 (“Cerro Cortez”). The yellow line shows the mean gold grade obtained from the ID^2 block model estimates and the green line shows the mean gold grade obtained from the OK block model estimates. The red line represents the mean gold grade of the composite data used to calculate the model estimates. Note that at the limits of the block model data becomes sparse grade estimates are often biased by the limited data; therefore estimates on the fringes of the model are often less reliable.

Figure 14-19: Easting Swath Plot of Au Estimates for Domain 4

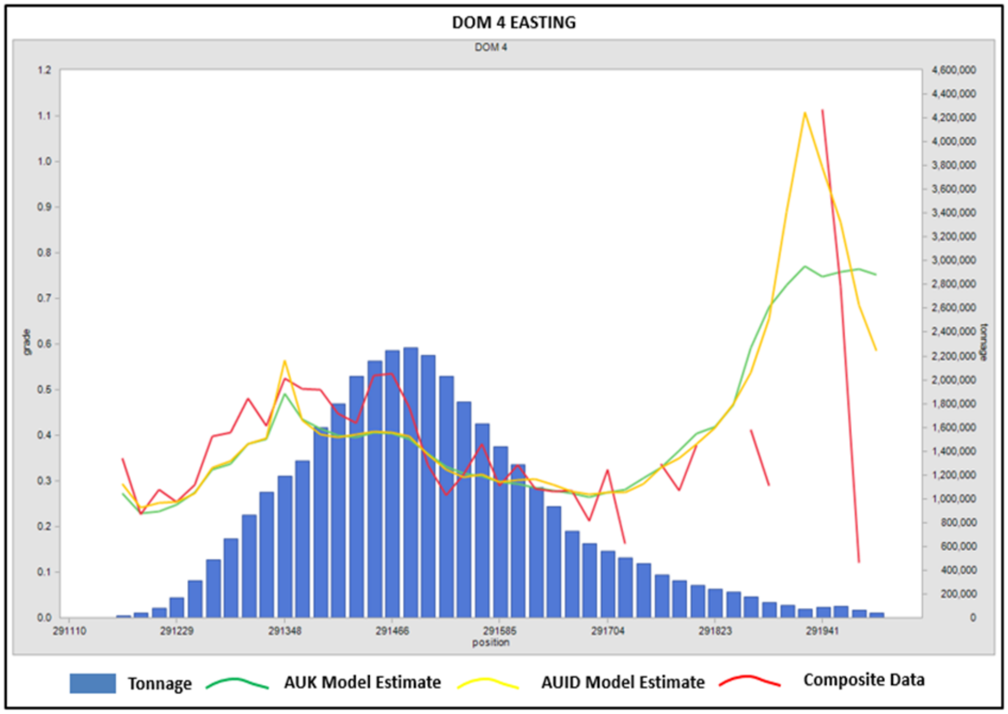
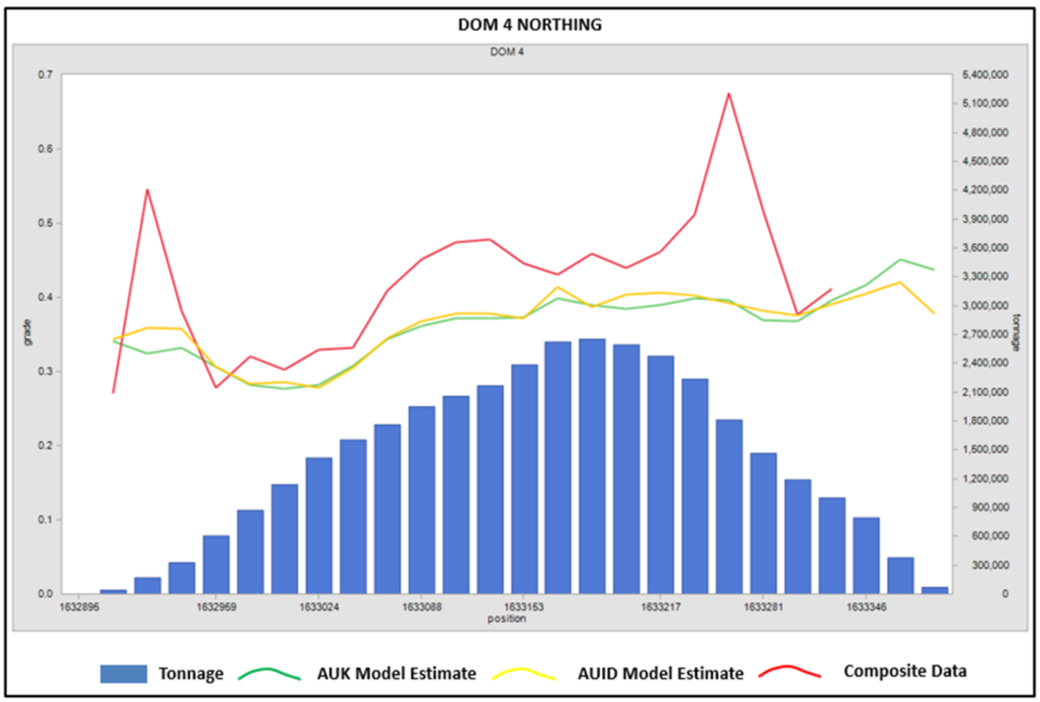


Figure 14-20: Northing Swath Plot of Au Estimates for Domain 4



In general, where there are sufficient block estimates to make reliable comparisons between the two block estimates and there is a good agreement between the OK and ID^2 models along with the composite data used to estimate the grades. The swath plots indicate that the estimates honor the data trends and smooth out the high and low extremes in the data as expected. In the example shown both the Northing and Easting swath plots are illustrated as in the Northing orientation the swath plot indicates a potential underestimation of model grade in relation to the composite data, this is countered by the Easting direction which shows a good correlation between composite and model grades.

In the Authors' opinion the swath plots support that the local trends in the gold grades are being appropriately modeled.

14.9.3 Reconciliation Validation

Minosa completed reconciliation on the January 2013 planning model to validate the use of production data in the resource estimation process, along with significant changes to the domain interpretations and weathering surfaces. It is the Authors' opinion the results from the reconciliation validate the use of production data with < 1.0% variance in predicted to actual produced ounces. Minimal changes have been made between the January 2013 and January 2014 Mineral Resource estimates therefore the reconciliation is still deemed relevant to the validation of the processes used.

The reconciliation study, covered the Twin Hills (Dom 1) area along with a small section of the Twin Hills (Dom 3) reproduced each months production by material type (oxide / mixed) and applied the forecast recoveries to the predicted ounce production. A 12 month period was deemed sufficient to account for the lag time between production and leaching time which is approximately +3 months from production.

The reconciliation study concluded the model estimate in the production areas predicted within +/- 4% of the actual recovered ounces. Month to month variation is predictable due to the lag time within the leaching process. Table 14-12 details that the resource model predicted 59,354 recovered ounces at 0.43 g/t versus an actual recovered total of 59,751 oz at 0.44 g/t; this is a 0.7% variance in ounces. The forecast recoveries used are based on the operational forecast figures for the Twin Hills area and are marginally higher than that used for the San Andrés Mineral Resources. When using, 0.76% (oxide) and 0.57% (sulphide) recoveries as per the resource model assumptions the predicted variance to actual produced ounces is -3.1%.

The reconciliation process and illustrations are discussed in Section 12.6.

Table 14-12: Resource Model Reconciliation – 2012 Production

Jan 2013 Model Predicted	Recovered Ounces	4,456	4,795	6,886	6,865	7,294	5,527	5,290	3,339	2,290	2,986	3,880	5,746	59,354
	Recovered grade	0.43	0.47	0.45	0.42	0.45	0.45	0.51	0.42	0.39	0.36	0.37	0.44	0.43
Actual Recovered Production	Produced Ounces	5,163	4,174	4,048	5,994	6,496	5,641	5,528	6,432	4,338	3,036	3,628	5,272	59,751
	Grade (g/t)	0.44	0.38	0.35	0.49	0.42	0.46	0.49	0.88	0.56	0.30	0.28	0.38	0.44
Variance	Ounces	708	-621	-2,838	-871	-798	113	239	3,093	2,048	50	-252	-475	397
	Grade (g/t)	0.01	-0.08	-0.10	0.08	-0.03	0.01	-0.01	0.46	0.16	-0.05	-0.08	-0.07	0.00

*Forecast recoveries of 78% (Oxide) and 62% (Mixed) applied to predicted raw ounces

14.9.4 Change of Support Validation

An assessment of the amount of smoothing in the block model estimates was not completed for the January 2014 Mineral Resource estimate as there have been no changes to the SMU / block size and estimation methodology as for previous estimates. The conclusions from the change of support validation process contained in the 2012 Technical Report stand and were as follows (page 101):

"When the NN block model gold grade estimates are adjusted to 30 x 30 x 6 m SMUs, there is good agreement between the OK block model gold grade estimates and the 30 x 30 x 6 m SMUs over the full range of the mining cut-offs from 0.30 to 0.5 g/t Au. This indicates that the OK estimates have a theoretical SMU of 30 x 30 x 6 m". (B. Reid et al, 2012)

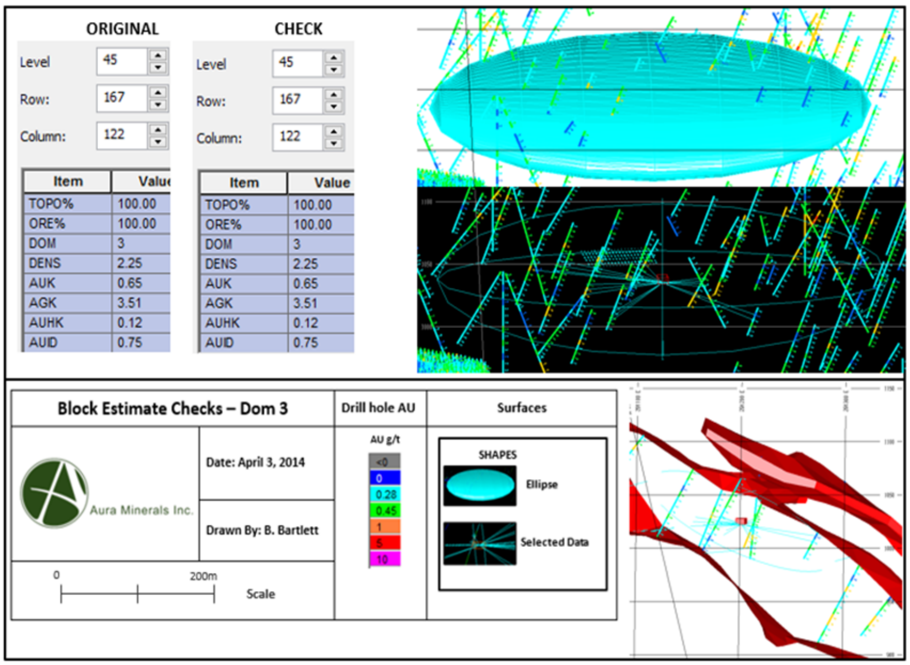
The Authors of the 2012 Technical Report recommended at page 101: "The SMU of 30 x 30 x 6 m requires further validation through reconciliation of the block model (long-term model) with the grade control model (short-term model)". The reconciliation exercise of completing a model with an SMU of 30 x 30 x 6 m has not been completed; however the reconciliation of the January 2013 production model against the 2012 Mine production resulted in a variance of +/-4%, indicating that the current model support assumptions are within acceptable limits for grade estimation.

14.9.5 Block Validation

Individual blocks were selected to test the estimation parameters and a search ellipse generated to illustrate the search used and the samples selected. Figure 14-21 illustrates a block chosen in domain 3 by which the block was re-estimated and the ellipse and composites used defined for visual validation.

It is the Authors' opinion that the estimation parameters were correctly applied and individual block checks provide repeatable estimates.

Figure 14-21: Block Validation of Estimation Parameters



14.9.6 Visual Validation

Detailed visual inspections of the gold grade estimates were conducted in both plan and section to ensure that interpolation results honoured the geological boundaries and the drill hole composite data. This validation included confirmation of the proper coding of blocks for each of the rock codes and the block grade estimates relative to drill hole composites to ensure that the drill hole data were properly represented in the model.

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 variogram model used, and the kriging plan applied (Figures 14-22 and 14-23).

Figure 14-22: Bench 930 m Elevation, OK Estimates for Domain Codes

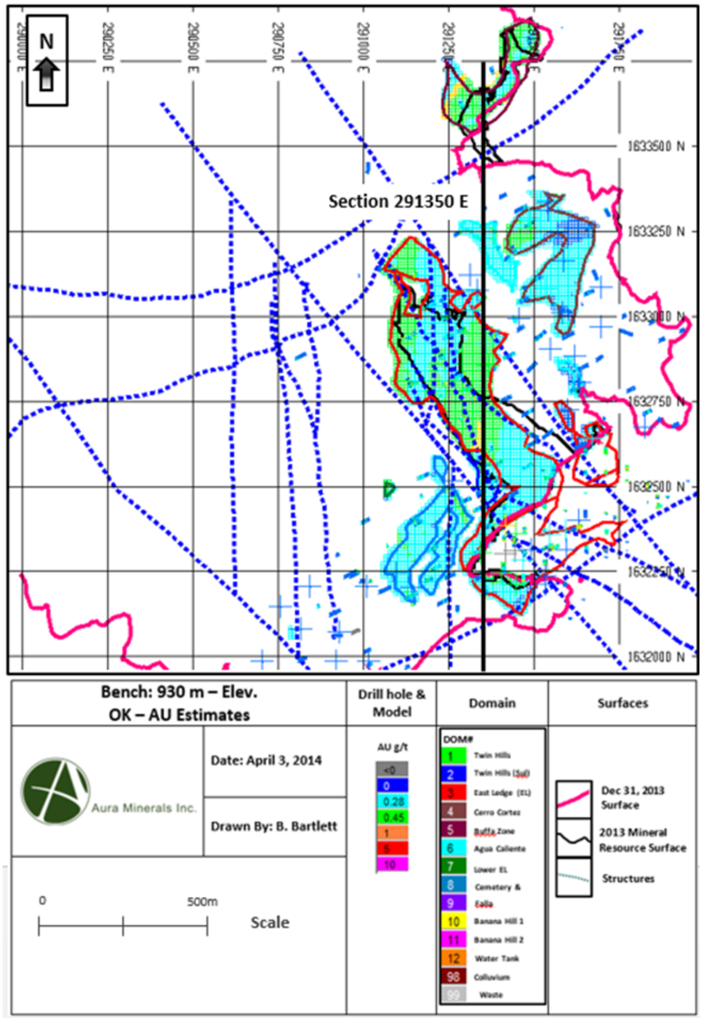
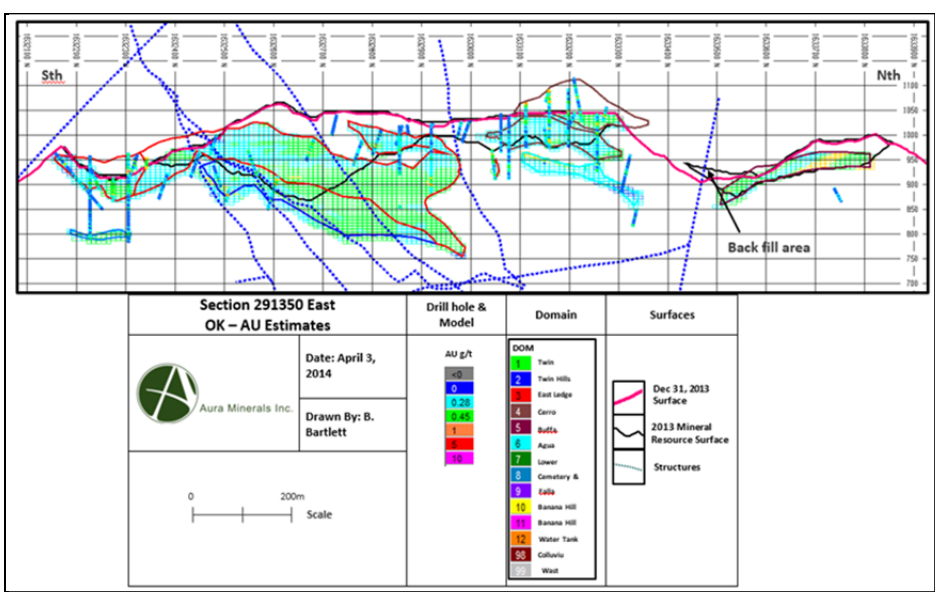


Figure 14-23: Section 291,350 E m OK Estimates for all Mineralised Domains



14.10 Mineral Resource Classification

The mineralization at the Mine has been classified in accordance with the definitions of NI 43-101 and the CIM Standards.

The relevant definitions for the CIM Standards are as follows:

A **Mineral Resource** is a concentration or occurrence of diamonds, natural solid inorganic or fossilized material, or natural sold fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth’s crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

The term “reasonable prospects for economic extraction” implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. In other words, a Mineral Resource is not an inventory of all mineralization drilled or sampled, regardless of cut-off grade, likely mining dimensions location or continuity. It is a realistic inventory of mineralization, which under assumed and justifiable technical and economic conditions might become economically extractable.

An **Inferred Mineral Resource** is that part of a Mineral Resource, for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.

An **Indicated Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

A **Measured Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.

The criteria applied by Minosa for the 2013 Mineral Resource classification differs from previous Mineral Resource estimates which used only a search distance criteria for classification. The change in approach was determined by the significant changes to the geological model, mineralisation domains and the applied estimation parameters between the models. The criteria chosen to classify the Mineral Resources are based on three fundamentals:

- Data density

- Geological Continuity
- Grade Continuity

Using this approach the Mineral Resource was classified by defining areas which were coded for CLASS as measured, indicated or inferred. The classification used the following criteria:

Measured Mineral Resource:

- Well established geological continuity (In pit mapping and logging evidence)
- Well defined drill hole intervals for grade continuity, generally <25 m x 25 m apart
- Well defined variography and grade continuity with blocks estimated within the close spaced search ranges (Section 14.6, Search Parameters)
- Plus / minus production blast hole data within the immediate area

Indicated Mineral Resource:

- Established geological continuity (Mapping and / or logging evidence)
- Established drill hole intervals providing reliable grade continuity, generally 25 m x 50 m but up to 100 x 50 m apart where well defined grade and geological continuity exist
- Reliable variography and grade continuity with blocks estimated within both the close and wide spaced search ranges (Section 14.6, Search Parameters)

Inferred Mineral Resource:

- Assumed geological continuity (Mapping and / or logging evidence)
- Limited drill hole intervals providing assumed grade continuity, generally >100 x 50 m apart
- Poor variography and assumed grade continuity with blocks estimated within the close and wide spaced search ranges (Section 14.6, Search Parameters)
- Periphery interpretation where few drill holes close the mineralization off

Figure 14-24 to Figure 14-26 illustrate the classification coding and Table 14-13 details the areas defined as Measured, Indicated or Inferred Mineral Resource. For areas covered by production blast holes the classification generally remained as indicated as much of the drilling in these areas is wide spaced 50 x 50 m to 100 x 100 m therefore the influence of the production data is limited to the immediate areas of influence.

Figure 14-24: Classification Coding

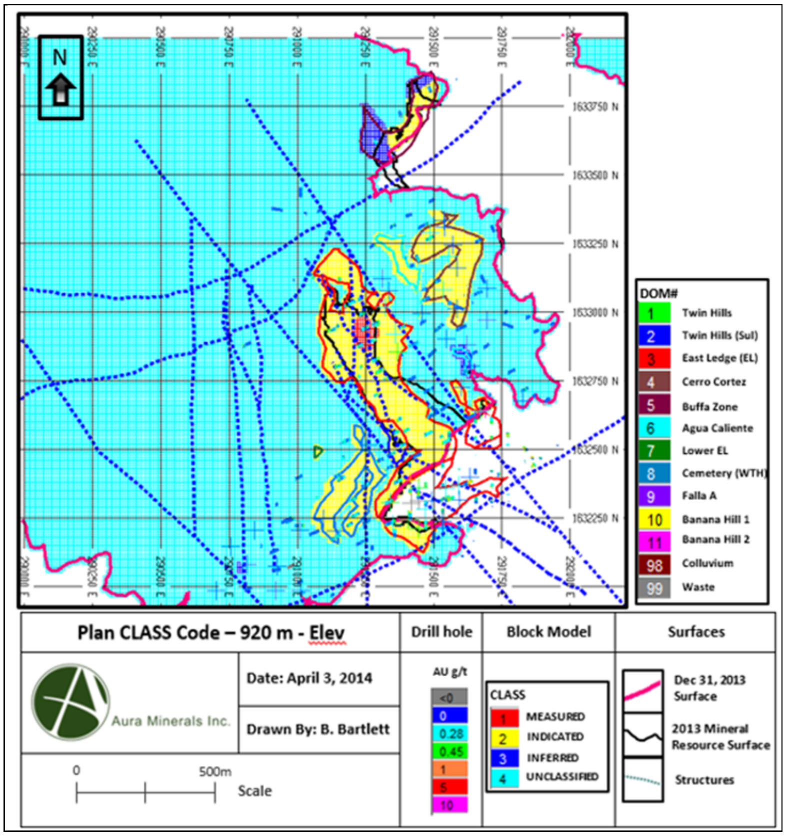


Figure 14-25: Classification Area Coding – Level Plans

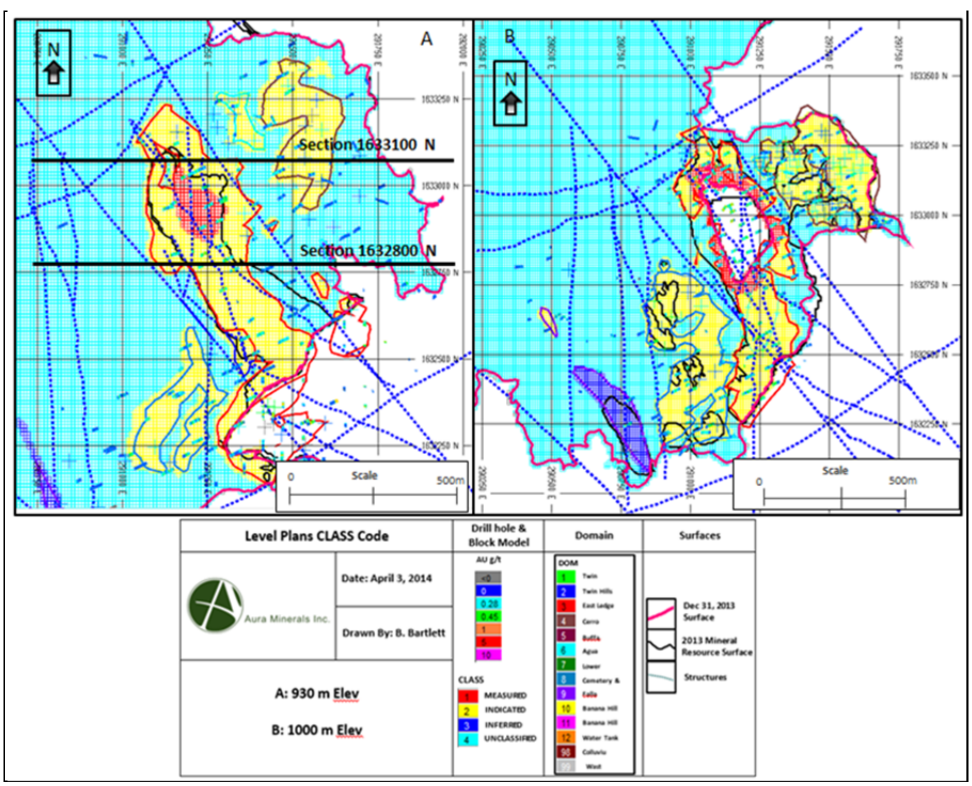


Figure 14-26: Classification Area Coding – Sections

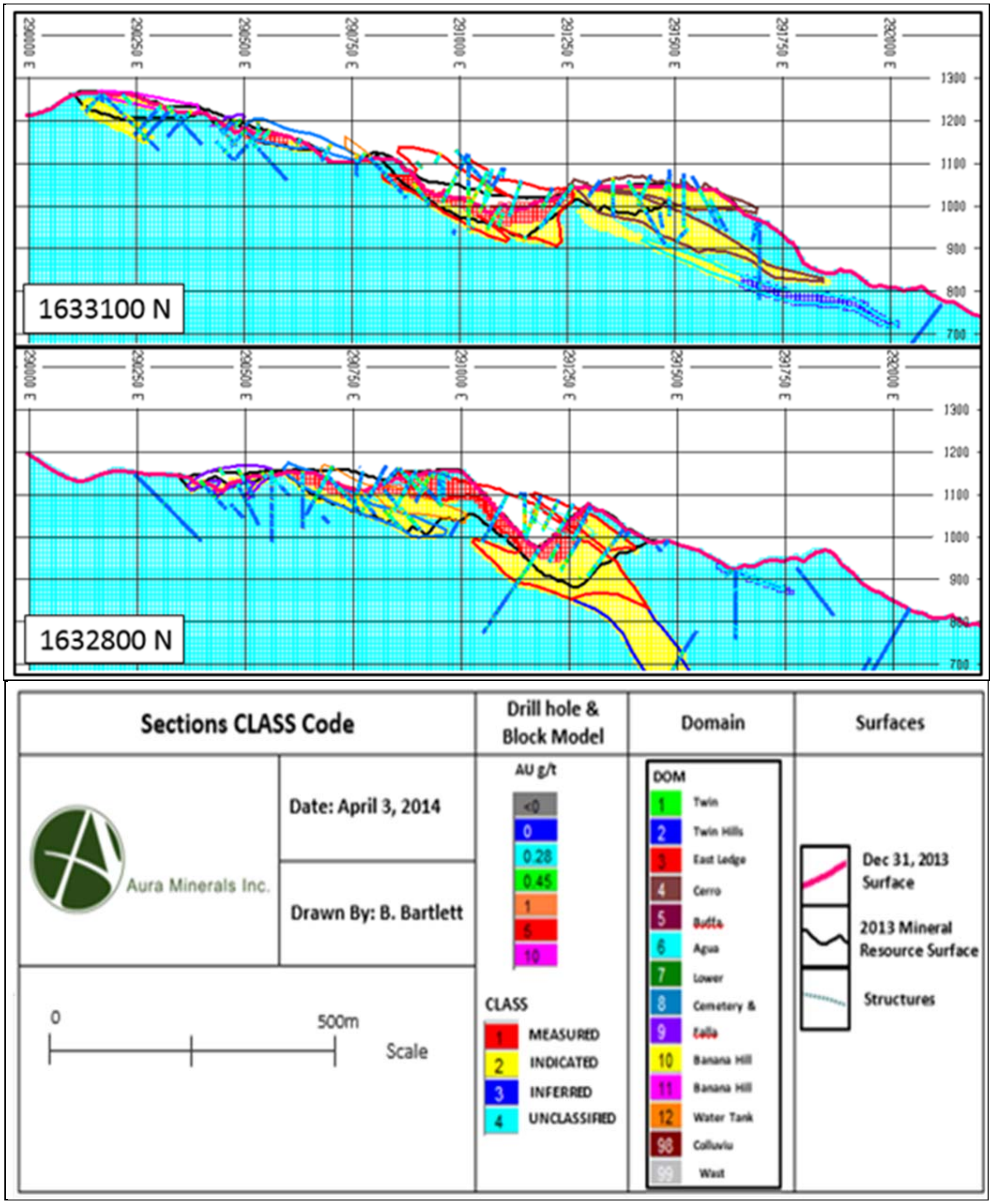


Table 14-13: Defined Classification areas

Resource Category	Class Code	Average	
		Area	**Comments**
Measured	1	East Ledge & Water Tank Hill	• Well established production and reconciliation • Close spaced drilling <= 25 x 25 m • Well defined grade distribution
Indicated	2	Majority of mineralisation domains outside of Measured and Inferred areas	• Established geological continuity • Drill spacing ranges from 25 x 25 m to 100 x 50m • Established grade continuity with reasonable variogram models
Inferred	3	Sections of Fault A, Cerro Cortez, Agua Caliente & Buffa Zone	• Wide spaced drilling with assumed geological continuity • Poor variograms and grade continuity

14.11 Mineral Resource Summary

The 2013 Mineral Resource estimate is constrained by the following parameters:

- An optimised pit shell using a gold price of US$1,600/oz
- December 31, 2013 topography surface
- Mineralisation cut-off grades of 0.23 g/t for OXIDE and 0.30 g/t for MIXED.

The December 31, 2013 Measured and Indicated Mineral Resource is estimated by Aura to be a total of 104.8 Mt at 0.49 g/t Au for 1.7 Moz and an Inferred Mineral Resource of 4.3 Mt at 0.49 g/t Au for 69 koz. This is an increase of 8.3 Mt and 66 koz on the December 31, 2011 Mineral Resources. The Mineral Resources for December 31, 2013 are detailed in Table 14-14.

The Resource Pit shell optimization did not consider any sulphide material as was done in previous years and the Mineral Resources are inclusive of Mineral Reserves.

Table 14-14: December 31, 2012 Mineral Resource Estimate*

Resources Category	Oxide			Mixed			Total		
	Tonne (t)'000	Au (g/t)	Ounces '000	Tonne (t)'000	Au (g/t)	Ounces '000	Tonne (t)'000	Au (g/t)	Ounces '000
Measured	13,424	0.46	199	2,814	0.59	54	16,238	0.48	252
Indicated	63,201	0.47	945	25,402	0.57	462	88,603	0.49	1,407
Measured + Indicated	76,625	0.47	1,144	28,216	0.57	516	104,841	0.49	1,660
Inferred	3,319	0.42	45	1,029	0.74	24	4,348	0.49	69

Note*:
1. The Mineral Resource estimate is based on an optimized shell using $1,600/oz gold.
2. The cut-off grade used was 0.23 g/t for oxide material and 0.30 g/t for mixed material.
3. Contained metal figures may not add due to rounding.
4. Surface topography as of December 31, 2013, and a 200m river offset restrictions have been imposed.
5. Mineral Resources are inclusive of Mineral Reserves.
6. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues.

The change in the Mineral Resources from 2011 to 2013 reflects the changes in the following areas:

- Mineralisation and oxidation domains
- Classification approach
- Decrease in the cut-off grade for oxide material from 0.28 g/t Au to 0.23 g/t Au and in the mixed material from 0.37 g/t Au to 0.30 g/t Au
- Depletion of Mineral Resources due to mining
- Exploration and production delineation drilling.

15.0 MINERAL RESERVE ESTIMATE

15.1 Pit-Optimization Method

The estimate of Mineral Reserve is based on a LOM plan and plant production schedule developed by Aura. The pit was optimized using MineSight Software. A raw pit shell was created using the parameters listed and then smoothed and detailed to include ramps and honour the pit slope criteria. Other than the internal dilution inherent through the resource estimation process, no additional dilution or mining loss was factored into the Mineral Reserve estimates. Reconciliation studies between the block model and actual grades and tonnes as described in Section 24.1 are within acceptable differences. Major factors that could materially affect the Mineral Reserve estimate are gold price, operating costs, metallurgical recoveries, available land, pit slope angle parameters and mineral resource estimates. Note that the Mineral Reserve estimate now includes those portions of Mineral Resources as of December 31, 2011 that was located beneath the cemetery region. Now that community agreements have been achieved, this area is now accessible for mining. As of December 31, 2011, this area only had reasonable prospects of achieving gaining access and therefore was not included as Mineral Reserves.

The economic criteria using the MineSight software Lerches-Grossman algorithm for pit limit evaluations, including process recoveries and operating costs, geometric parameters are provided in Table 15-1.

Table 15-1: Pit's Main Parameters

Parameters	
Bench height	6 m
Road width	18 m
Overall Pit Slope	Varies: 41 - 49
Bench face angle	Varies : 65 – 70
Minimum pit bottom	20 m
Berm width	3.8 m
Ramp Slope	10%

The cost parameters to develop the analyses for the definition of Mineral Reserves and mining planning are outlined in Table 15-2. These parameters are reasonable estimates based on the current operation.

Table 15-2: Cost Parameters

Gold Price (US$/oz)	1,300
Oxide Recovery	76%
Mixed Recovery	57%
Costs - US$/t	
Mine	2.41
Plant*	6.49
G&A	1.75

Note*– Includes maintenance costs

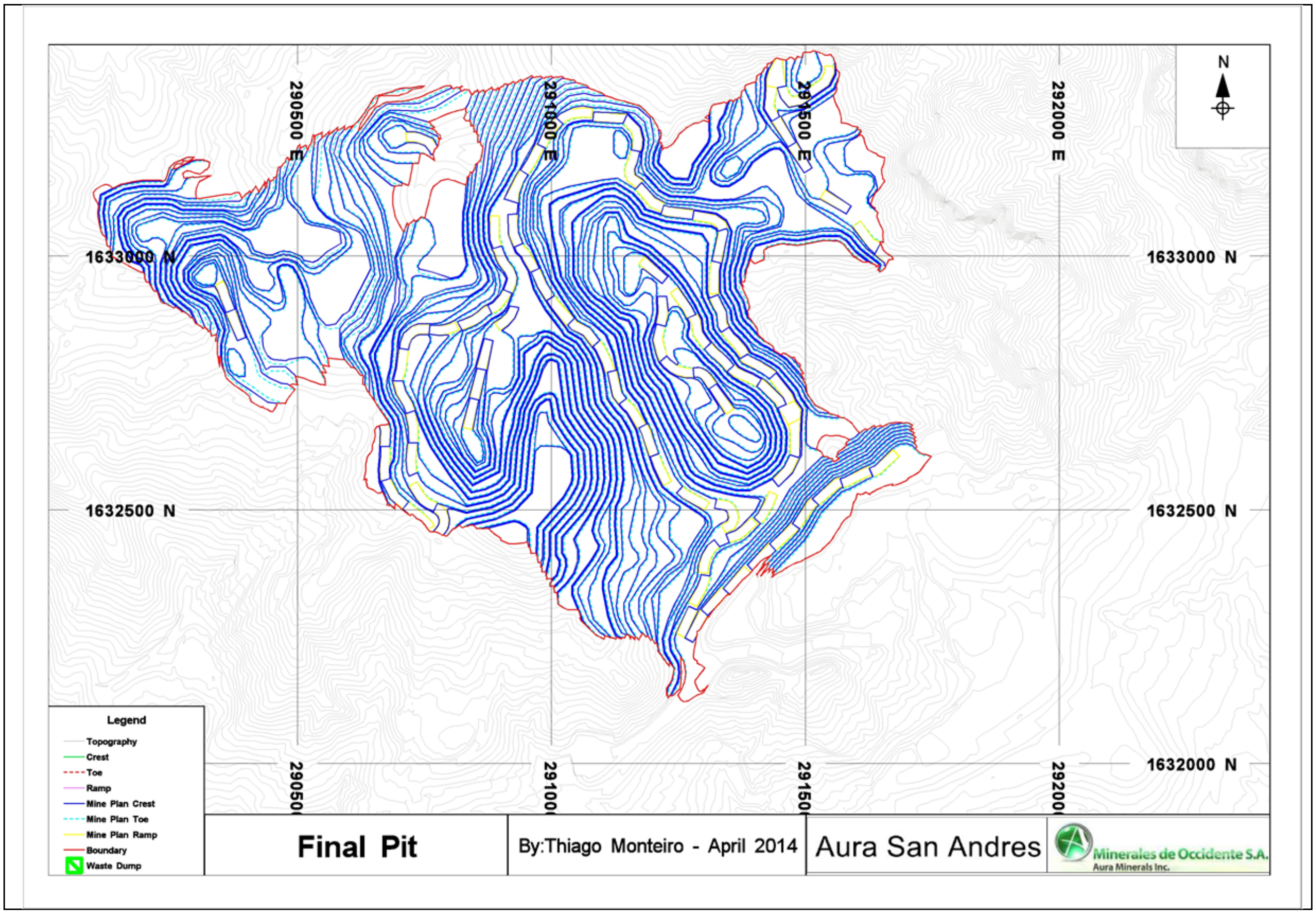
The 2013 Mineral Reserve estimate is shown in the Table 15-3 and the ultimate pit is illustrated in Figure 15-1 below.

Table 15-3: December 31, 2013 Mineral Reserves Estimate

Mineral Reserve Category	Oxide			Mixed			Total Material		
	Tonne (t)'000	Au (g/t)	Oz ' 000	Tonne (t)'000	Au (g/t)	Oz '000	Tonne (t)'000	Au (g/t)	Oz '000
Proven	12,369	0.48	190	2,346	0.63	47	14,714	0.50	237
Probable	43,838	0.50	702	9,549	0.62	190	53,388	0.52	892
Proven + Probable	56,207	0.49	892	11,895	0.62	238	68,102	0.52	1,129

Note:
1. The Mineral Reserve estimates are based on an optimized pit, which has been made operational, using $1,300/oz gold.
2. The cut-off grade used was 0.28 g/t for oxide material and 0.37 g/t for mixed material.
3. Contained metal figures may not add due to rounding.
4. Surface topography as of December 31, 2013.

Figure 15-1: Final Pit Design



15.2 Mine Sequence

The end of period maps from 2013 to 2019 are illustrated in Figures 15-3 to 15-9, while the summarized production schedule is shown in Table 15-4.

Figure 15-2: Year End Projection – 2014

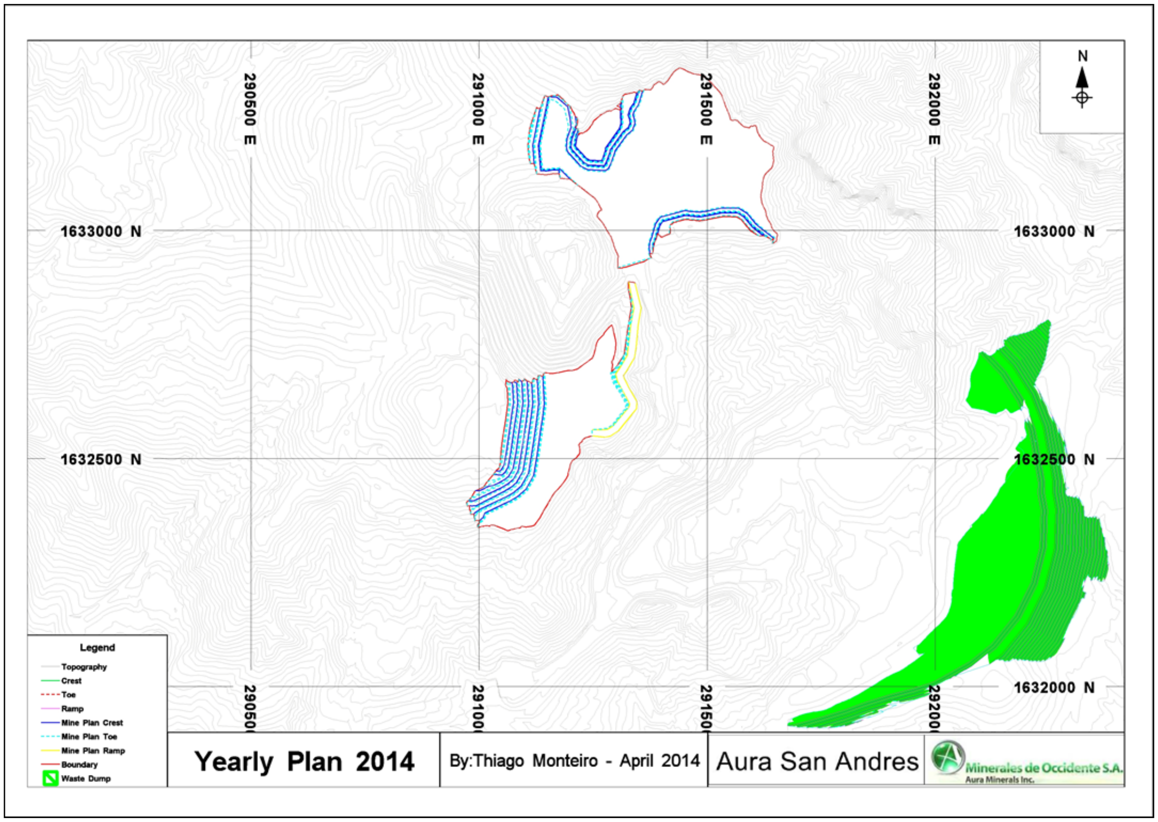


Figure 15-3 Year End Projection – 2015

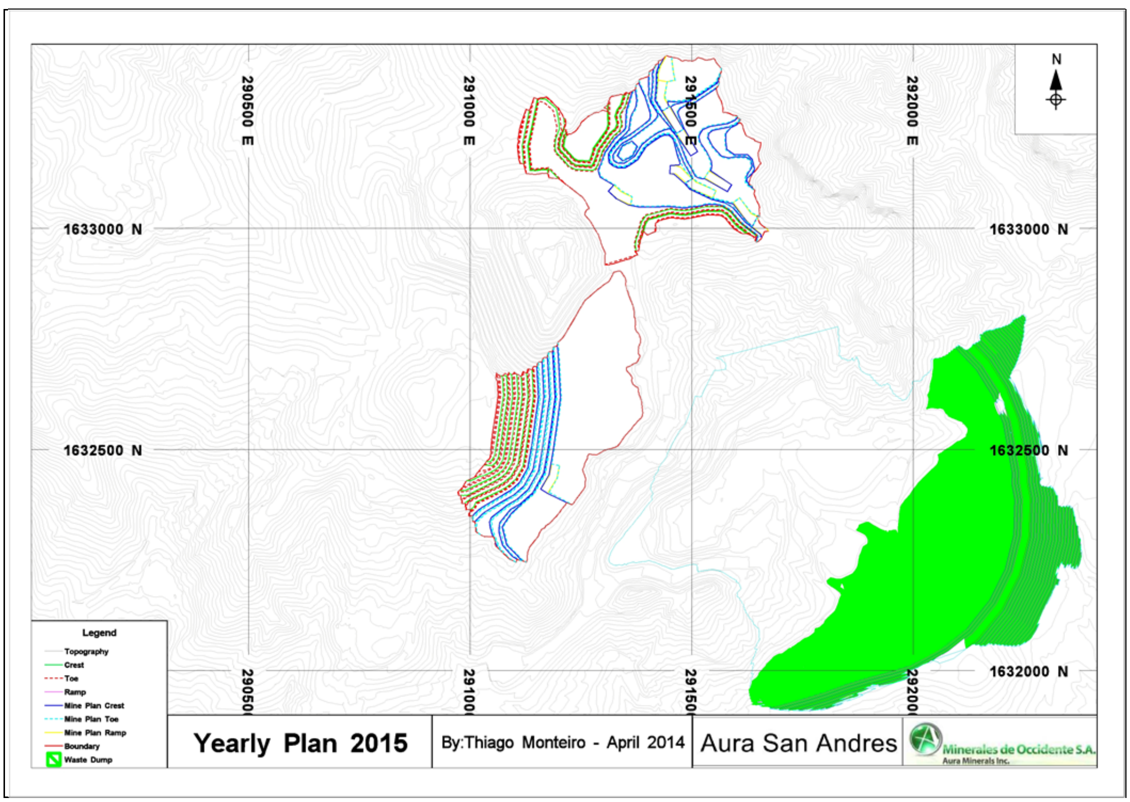


Figure 15-4: Year End Projection – 2016

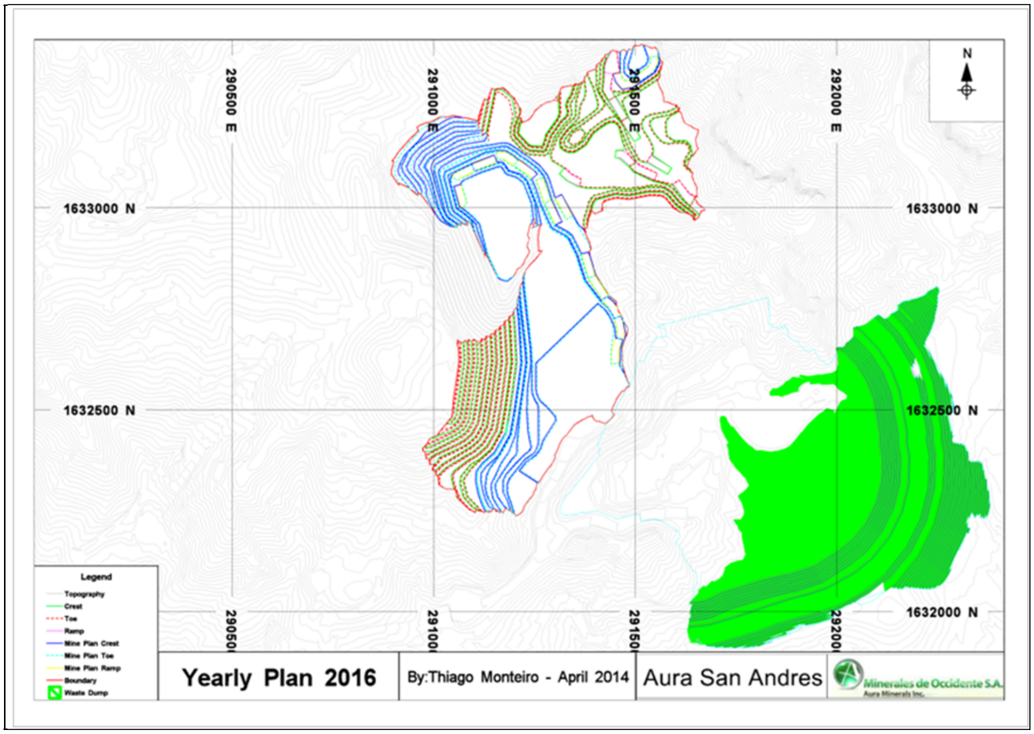


Figure 15-5: Year End Projection – 2017

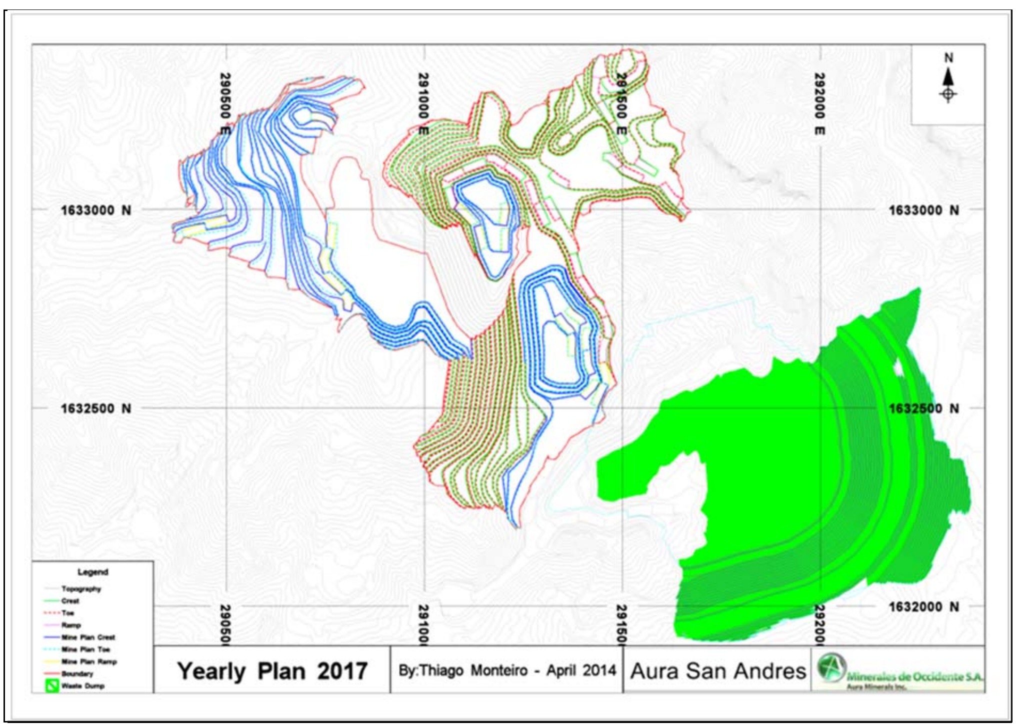


Figure 15-6: Year End Projection – 2018

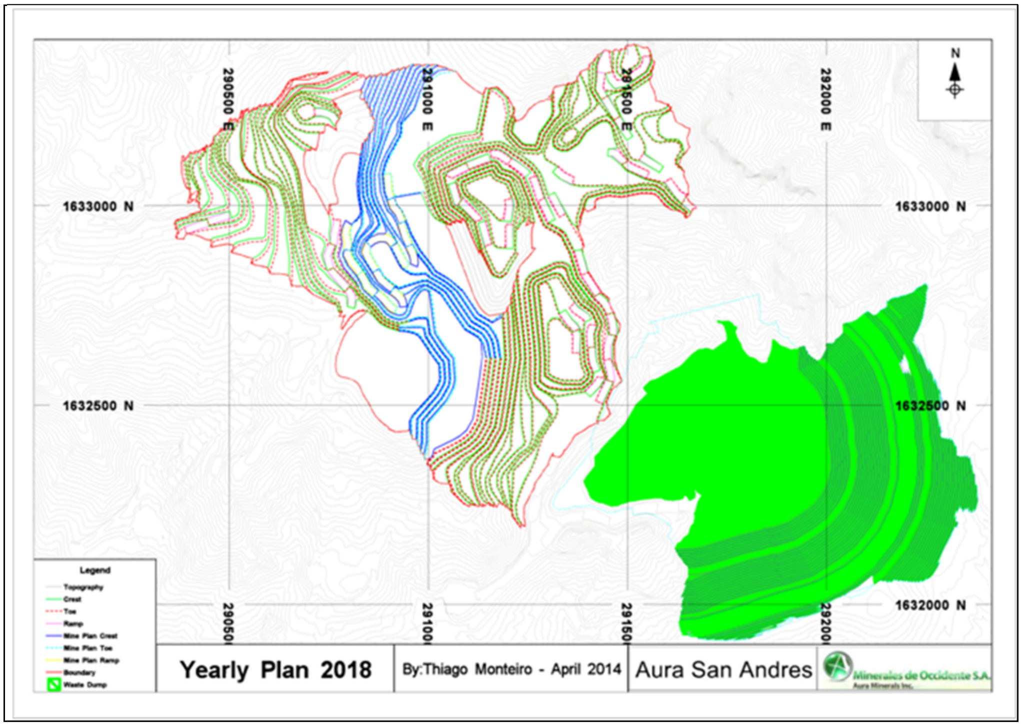


Figure 15-7: Year End Projection – 2019

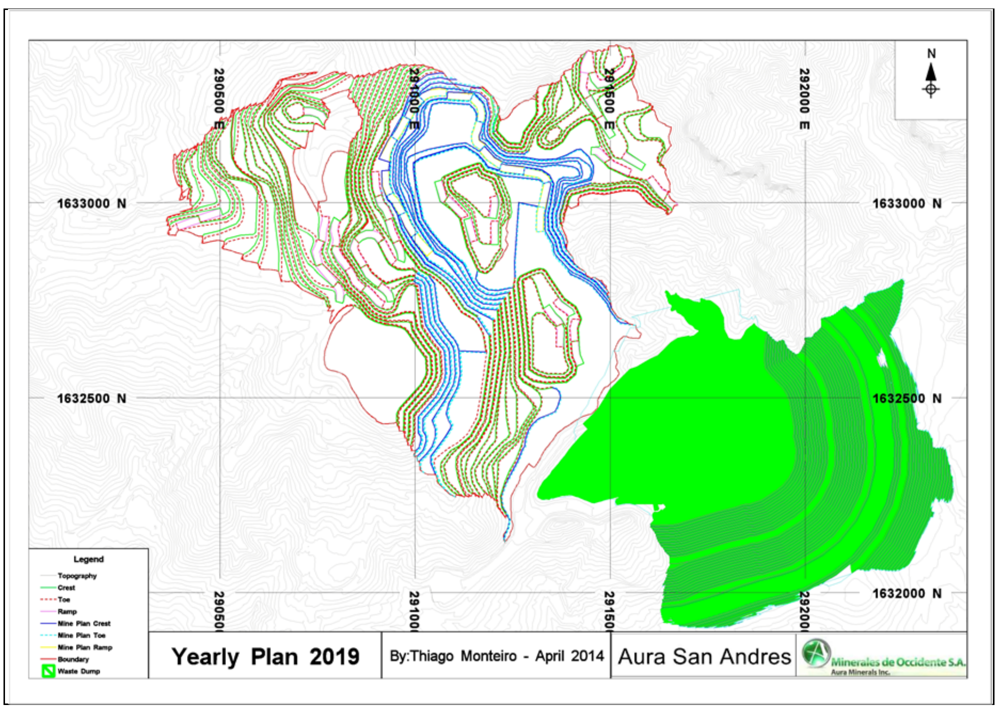


Figure 15-8: Year End Projection – 2020

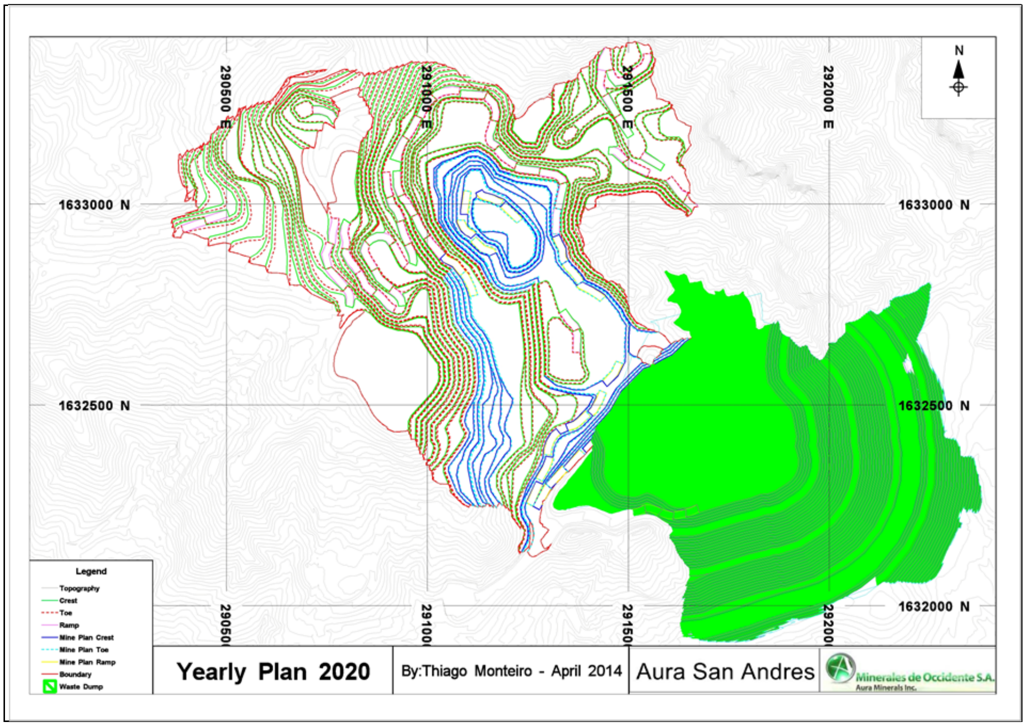


Figure 15-9: Year End Projection – 2021

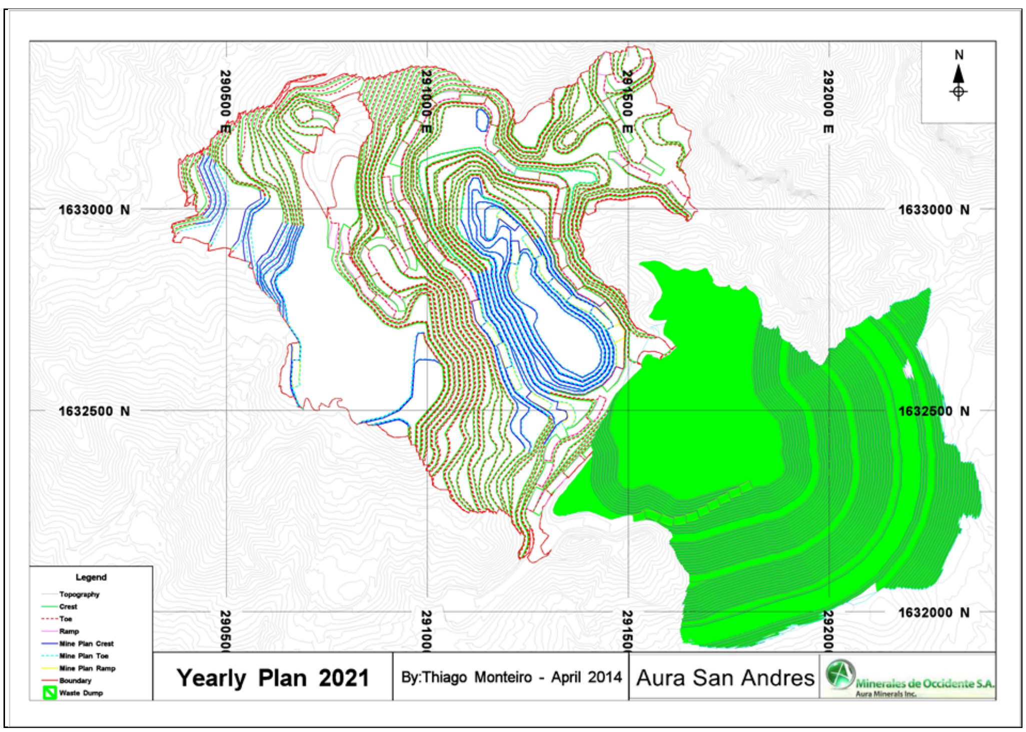


Figure 15-10 Year End Projection – 2022

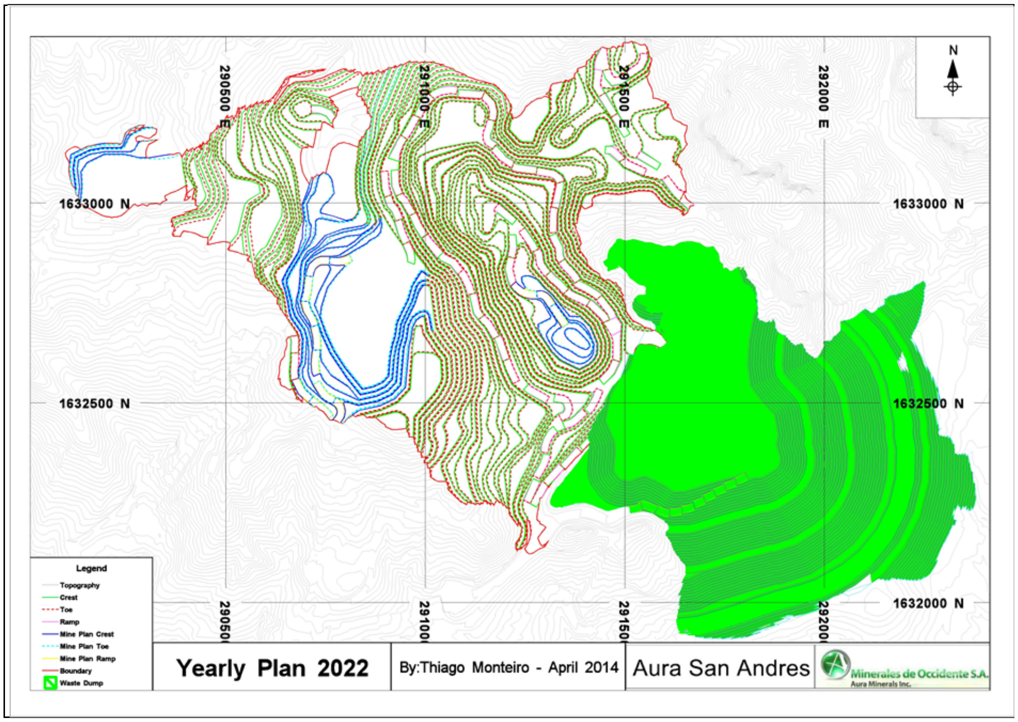


Figure 15-11: Year End Projection – 2023

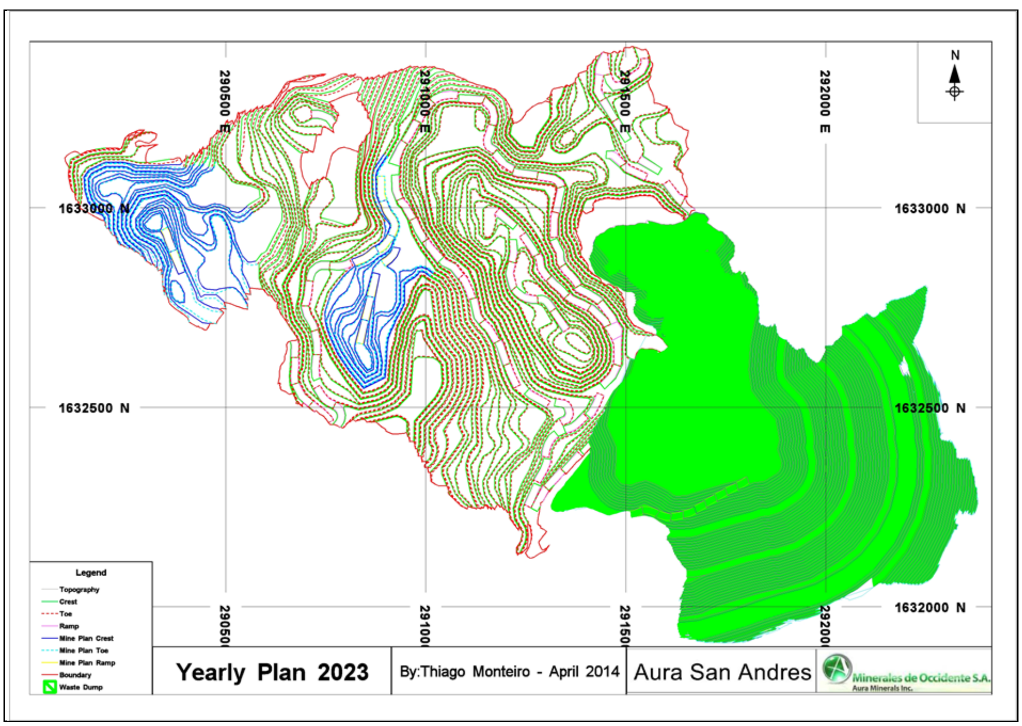


Table 15-4: Mine Production Schedule

Year	Oxide Ore			Mixed Ore			Total Ore			Waste	
	Tonne (t)'000	Oxide Gold Grade (g/t)	Gold Oz' 000	Tonne (t)'000	Mixed Gold Grade (g/t)	Oz' 000	Tonne (t)'000	Total Gold Grade (g/t)	Gold Oz '000	Tonne (t) '000	W/O
2014	6,333	0.47	97	0	0.00	0	6,333	0.47	97	3,986	0.63
2015	7,046	0.47	106	7	0.47	0	7,053	0.47	106	3,313	0.47
2016	6,582	0.49	103	449	0.59	9	7,030	0.49	112	5,363	0.76
2017	6,107	0.52	102	914	0.61	18	7,021	0.53	119	5,783	0.82
2018	6,839	0.47	102	263	0.62	5	7,102	0.47	108	6,741	0.95
2019	5,415	0.51	89	1,498	0.74	35	6,913	0.56	124	5,115	0.74
2020	4,207	0.51	69	2,894	0.56	52	7,101	0.53	121	3,075	0.43
2021	2,650	0.48	41	4,398	0.63	89	7,048	0.57	130	4,912	0.70
2022	6,080	0.49	95	707	0.61	14	6,786	0.50	109	5,739	0.85
2023	4,949	0.55	88	765	0.62	15	5,714	0.56	103	4,678	0.82
Total	**56,207**	**0.49**	**892**	**11,895**	**0.62**	**238**	**68,102**	**0.52**	**1,129**	**48,705**	**0.72**

Figure 15-12: Results of Mine Sequence – Total

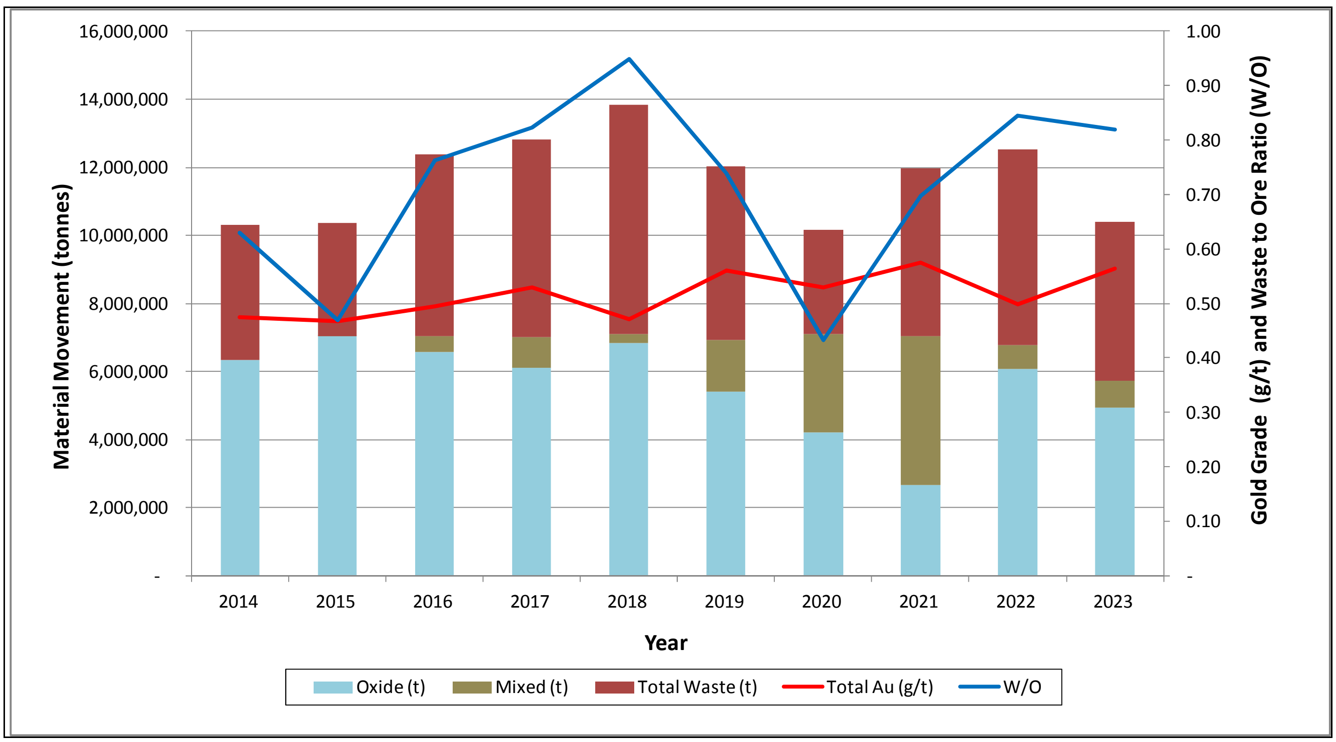


Figure 15-13 shows a plan view of the production areas.

Figure 15-13 Production Areas

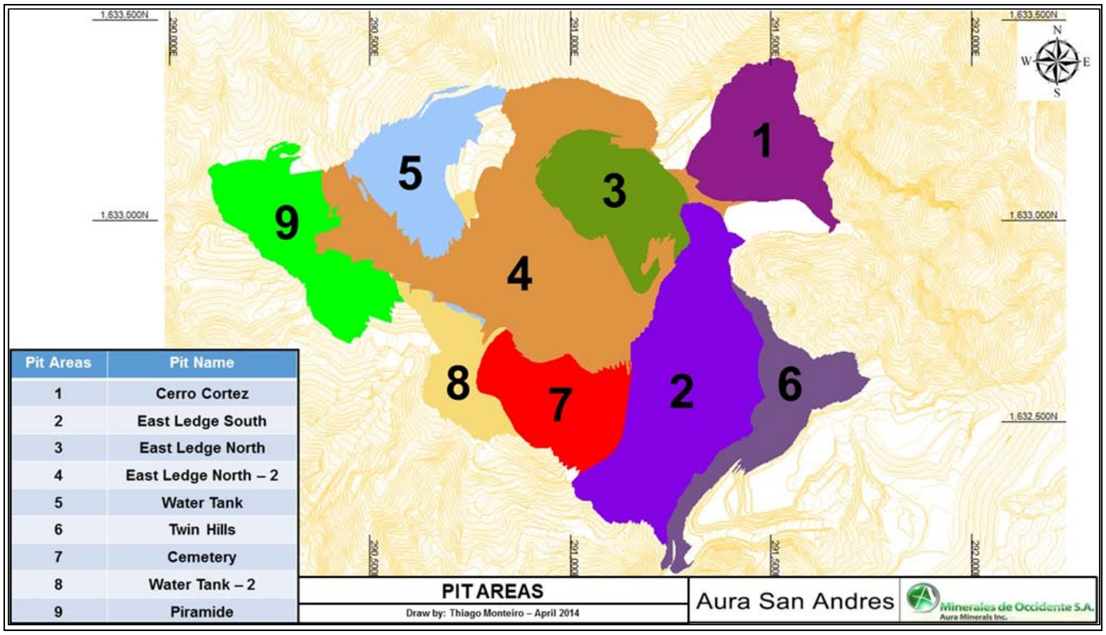


Table 15-5 shown below outlines the Life of Mine production plan by area.

Table 15-5: Mine Production Schedule by Area

Year		2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	Total
Cerro Cortez	Grade (g/t)	0.44	0.39	0.40	-	-	-	-	-	-	-	**0.42**
	Ore (tx 1000)	4,491.52	2,661.65	183	-	-	-	-	-	-	-	**7,336**
	Waste (tx1000)	1,775.32	529.87	34	-	-	-	-	-	-	-	**2,339**
East Ledge South	Grade (g/t)	0.56	0.52	0.48	0.52	-	-	-	-	-	-	**0.51**
	Ore(tx 1000)	1,841	4,391.45	5,756	3,495	-	-	-	-	-	-	**15,484**
	Waste (tx 1000)	2,210	2,668	3,363	1,169	-	-	-	-	-	-	**9,411**
East Ledge North	Grade (g/t)	-	-	0.58	0.59	-	-	-	-	-	-	**0.58**
	Ore (tx 1000)	-	-	1,089.70	487.73	-	-	-	-	-	-	**1,577**
	Waste (tx 1000)	-	115	1,709	295	-	-	-	-	-	-	**2,120**
East Ledge North-2	Grade (g/t)	-	-	0.49	0.41	0.47	0.57	0.57	-	-	-	**0.52**
	Ore (tx 1000)	-	-	2.52	2,206	4,598	4,466	3,735	-	-	-	**15,008**
	Waste (tx 1000)	-	-	229,38	2,954	3,468	3,076	467	-	-	-	**10,195**
Water Tank	Grade (g/t)	-	-	-	0.83	1.03	-	-	-	-	-	**0.84**
	Ore (tx 1000)	-	-	-	832	61	-	-	-	-	-	**894**
	Waste (tx 1000)	-	-	27	1,365	33	-	-	-	-	-	**1.426**
Twin Hills	Grade (g/t)	-	-	-	-	-	0.43	0.44	-	-	-	**0.44**
	Ore (tx 1000)	-	-	-	-	-	253.16	1,066.81	-	-	-	**1,320**
	Waste (tx 1000)	-	-	-	-	-	1,417	1,664	-	-	-	**3,081**
Cemetery	Grade (g/t)	-	-	-	-	0.47	0.55	0.52	0.59	0.68	-	**0.55**
	Ore (tx 1000)	-	-	-	-	2,442	2,194	2,247	5,891	434	-	**13,209**
	Waste (tx 1000)	-	-	-	-	2,894	587	447	1,515	71	-	**5,514**
Water Tank - 2	Grade (g/t)	-	-	-	-	-	-	0.57	0.49	0.48	0.48	**0.48**
	Ore (tx 1000)	-	-	-	-	-	-	52	1,157	5,778	2,458	**9,444**
	Waste (tx 1000)	-	-	-	-	346	12	496	3,397	4,562	430	**9,244**
Pyramid	Grade (g/t)	-	-	-	-	-	-	-	-	0.58	0.63	**0.62**
	Ore (tx 1000)	-	-	-	-	-	-	-	-	574	3,256	**3,830**
	Waste (tx 1000)	-	-	-	-	-	23	-	-	1,106	4,248	**5,377**
Total	**Grade (g/t)**	**0.47**	**0.47**	**0.49**	**0.53**	**0.47**	**0.56**	**0.53**	**0.57**	**0.50**	**0.56**	**0.52**
	Ore (tx 1000)	**6,333**	**7,053**	**7,030**	**7,021**	**7,102**	**6,913**	**7,101**	**7,048**	**6,786**	**5,714**	**68,102**
	Waste (tx 1000)	**3,986**	**3,313**	**5,363**	**5,783**	**6,741**	**5,115**	**3,075**	**4,912**	**5,739**	**4,678**	**48,705**

16.0 MINING METHODS

Mining is by conventional open pit methods. Benches are 6 m high. Operating phases (push-backs) have been designed to support the mine production.

The Mine is, as of the date of this Report, working 24 hours per day 7 days per week. Previously, the Mine worked on a 6 day per week schedule.

16.1 Drilling

The blasthole drilling is done by the Minosa employees and equipment. Minosa has an Atlas Copco L8, a Furukawa DCR 20 and an Ingersoll Rand ECM590 hydraulic percussion drills for blasthole drilling, as well a new Furukawa DCR 20 drill. These drills are operated and maintained by Minosa employees. The drilling is done normally on a 3.5 m by 4 m (spacing and burden) pattern with a depth of 6.6 m. The bench height is 6 m and 0.6 m for sub drilling. Minosa also has a MAXCAT RC drill used in exploration that can be converted to a DTH drill in a few hours in order to back up any further necessity in blasthole drilling.

16.2 Blasting

The blasting is carried out by Minosa employees. Blasting is done on the day shift using conventional blasting agents including ANFO, Emulsions, Dual Delay detonators, MS surface delays and is initiated using electric detonators. All of the materials are stored in the explosives magazines that are designed and maintained using North American standards. The Powder Factor is approximately 0.18kg/t. The pit walls are protected from impact by using either pre-shearing techniques or cushion blasting techniques.

16.3 Loading and Hauling

All of the material movement at the Mine is done by the Honduran contractor “Inversiones y Comercializadora Benitez” (Incobe). Incobe are responsible for all of the material movement within the pit and surrounding areas. Incobe has its own employees including maintenance staff, operating staff, and purchasing staff. They are responsible for all of their logistics such as shops, transportation, and accommodations. All of these services are provided at the current bank-cubic-meter (BCM) rate. Extra activities are covered on a cost plus rate that have to be authorized by the engineer in charge for Minosa.

As of the date of this Report, the Cerro Cortez open pit is being mined with an average haul distance to the primary crushers of 1.5 km. The old Water Tank Hill crusher is in operation as well, so that the Mine has flexibility as to which one is best suited to direct the ore. The distance from the pits to the primary crushers will vary with the pits evolution.

A listing of equipment by Incobe as of January 2014 is shown in Table 16-1 below:

Table 16-1: Equipment List

Equipment	Make	Model	Year	Units
30 tonne Excavator	Caterpillar	336 DL	2013	3
50 tonne Excavator	John Deere	450 DL	2014	3
Front Loader	Caterpillar	966H	2010	1
Track Dozer	Case	9050	2005	1
Track Dozer	John Deere	850J	2013	1
Track Dozer	Caterpillar	D6D	2000	1
Track Dozer	Caterpillar	D6R	2006	1
Track Dozer	Caterpillar	D8T	2010	1
30 t articulated haul truck	Caterpillar	730	2012	3
40 t articulated haul truck	Caterpillar	740D	2010	7
40t articulated haul truck	Caterpillar	740B	2014	8
40t articulated haul truck	John Deere	410E	2014	5
20t rigid body	Mack	Granite	2014	22
20t rigid body	International	Star	2013	3
Grader	John Deere	670G	2013	1
Compactor	Caterpillar	CS533	2013	1
Water Trucks	Various	Various	n/a	4
Fuel and Service trucks	Various	Various	n/a	2

This current fleet is expected to be able to fulfill production requirements over the LOM. If extra equipment should be required, Incobe management has indicated they can bring in additional equipment relatively easily from other sources.

As mining requirements increase over the LOM, Incobe has committed to supplying extra equipment.

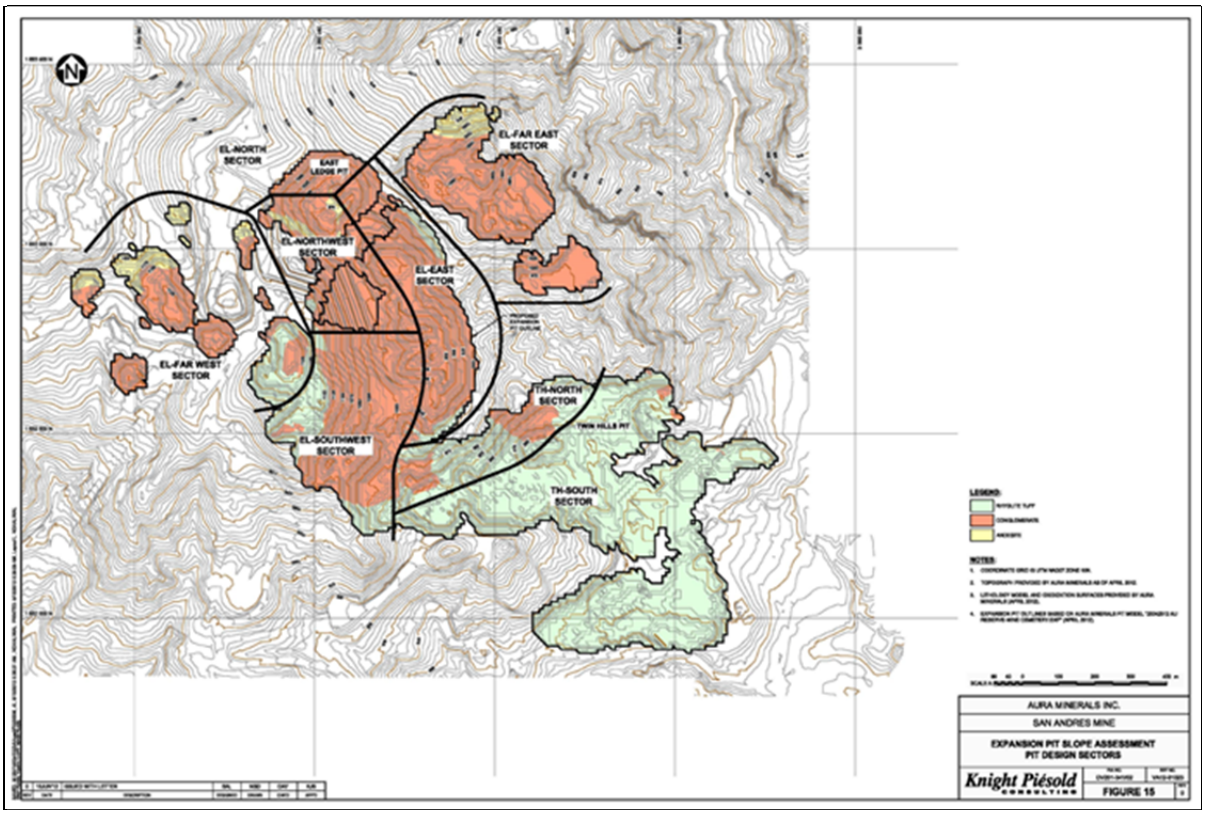
16.4 Pit Slope Stability

Knight Piesold geotechnical consultants completed a review of the pit slope conditions and recommended some minor changes to previous assumptions. Table 16-2 and Figure 16-1 shown below outline the current pit slope design parameters that were used for final design. Ongoing geotechnical support is continuing.

Table 16-2: Geotechnical Pit Slope Design Parameters

Expansion Zone	Pit Design Sector	Geotechnical Unit	Bench Face Angle (°)	Bench Height (m)	Bench Width (m)	Inter-Ramp Angle (°)
East Ledge	EL-North	Conglomerate	70	12	6	49
	EL- East	Conglomerate	70	12	6	49
	EL-Southwest	Conglomerate	70	12	8	44
	El-Northwest	Conglomerate	70	12	6	49
	EL-Far East	Conglomerate	70	12	6	49
	EL-Far West	Conglomerate-Rhyolite Tuff	70	12	6	49
Twin Hills	TH- North	Rhyolite Tuff	65	6	4	41
	TH-South	Rhyolite Tuff	70	6	4	44

Figure 16-1: Geotechnical Pit Slope Design Parameters Map



16.5 Mine Waste

Local soils are found to have moderate to poor drainage and a sandy-silty texture. The soils are relatively shallow in depth, being generally less than five centimeters. The pH of the soil is slightly acidic to near neutral (4 to 6.5), characterized mostly by pine or pine-oak associations. Deeper soils are found along the streams and allow the establishment of a more diverse forest. The low pH of the soil, poor drainage and high erosion are all cited as natural limitations to the development of an extensive vegetative cover. However, substrate resulting from the piling of waste rock has proven to have moderate to good drainage as a result of the breaking that occurred during the blasting and mixture of diverse grain materials.

Substrates resulting from the contouring and piling of waste rock have proven to require minimum fertilization, although in some cases the hard nature of rocky material has required covering with top soil to promote plant growth. Revegetation consists in covering the resulting exposed surface with Alicia grass or Brachiaria grass.

In 2011 a small localized acidic seepage was identified in the Twin Hills area. A water treatment plant was established to treat this water before discharge. An Acid Mine Drainage specialist was commissioned to report on the system and recommend any upgrades or modifications. A summary of completed recommendations are listed below:

- Provide continuous lime dosification into the treatment dam (night and day)
- Acquire and install a pump / pH controller with proportional feedback loop
- Increase the treatment dam size to 200 M3
- Muck out sediments from the treatment dam

In addition to these, it was also recommended to encapsulate the Twin Hills waste rock piles with soil to inhibit water, and oxygen. As of August 2013, the encapsulation was 75% complete.

Minosa continues to monitor and manage this issue so that environmental compliance targets are achieved. A plan to have a passive treatment installed is in place which is expected to be implemented in 2014.

16.6 Water Monitoring

As a part of the Environmental Management Plan, Minosa has developed a water monitoring program that is updated annually. The program includes the monitoring of water quality from surface, ground water, drains coming from recently developed waste rock dumps and subgrade drains under leaching facilities. The installation of monitoring piezometers around solution ponds and leach pads adds new monitoring stations. Ground water monitoring stations continue to monitor older facilities. The main criteria for water analysis continue to be the pH, conductivity, presence of heavy metals, and free cyanide in drains coming from the new facilities. Water quality data has changed since the initial baseline study, with special significance in the drains coming from the North waste rock dump (former tailings dump site of previous operations). The change has been substantially positive since most of "polluting" parameters are undetectable or found only in trace amounts.

16.7 Mine Drainage

Outside of the pits, drainage ditches will be opened to turn aside rainfall water. Drainage of the Mine will be natural, while the Mine is operating with the open-type benches, through properly positioned drainage ditches.

Mining closed-type benches will take place right in the beginning of mining operations. In this case drainage will be carried out through pumping from one of the extreme points of the pit to where all the water can be directed to the properly positioned collecting ditches. Small dikes will be constructed in the valley downstream of the pits, to hold solids carried by the Mine drainage flows.

16.8 Waste Disposal

The waste disposal is carried out in a controlled manner, with the formation of the dump in ascendant sequence in consecutive, in small lifts of 3 m. The movement of equipment (trucks and tractor) on top of these layers will make them compact, assuring the stability of the dumps and making it harder for rainwater to infiltrate and cause erosion.

The location of the waste disposal piles are beside the open pit area, where transportation costs would be minimized. Waste piles are also located in areas within the pit.

The design parameters for waste disposal inside the mined out pits are outlined in Table 16-2.

Table 16-2: Design Parameters for In Pit Waste Disposal

Dynamic dumping angle	32°
Lift Height	3.0 m
Berm width	4.0 m
Average overall dump angle	23°

The design parameters for waste disposal outside of the mined out pits are outlined in Table 16-3.

Table 16-3: Design Parameters for Ex Pit Waste Disposal

Dynamic dumping angle	38°
Lift Height	3.0 m
Berm width	3.0 m
Average overall dump angle	27°

Drainage ditches will be opened upstream of the heaps to turn aside rainwater. The berms will have a small inclination toward the toe of the dump face, to minimize the erosion from the effects of heavy rain. Drainage ditches will also be constructed on the toes of the waste face to collect and remove rainfall water from the area.

17.0 RECOVERY METHODS

17.1.1 Process Plant Description

Mining at San Andrés is by conventional open pit methods. Benches are 6 m high. Operating phases (push-backs) have been designed to support the Mine production from initial topography of December 31, 2010 up to the final pit geometry.

Ore is hauled to the jaw crushers utilizing a contract haul fleet. All of the ore is processed through a two-stage crushing circuit and transported on conveyors before being stacked. After the ore has been crushed, it is treated with cement and lime before reaching the agglomerators.

The Mine production schedule was generated based on the December 31, 2013 Mineral Reserves. The detailed 2014 mine schedule is summarized by year in Table 17-1. The project LOM will be 10 years. The total run-of-mine production of 68.1 Mt contains approximately 1,129,000 oz Au at an average grade of 0.52 g/t.

The estimated metallurgical recoveries are expected to be 76% for oxide ore and 57% for mixed ore. Annual gold production will be dependent on the average grade of ore mined and the blend between the oxide and mixed ores processed.

Table 17-1: Life of Mine Schedule – 31 December 2013

Year	Oxide Ore			Mixed Ore			Total Ore			Waste	
	Tonne (t)'000	Oxide Gold Grade (g/t)	Gold Oz' 000	Tonne (t)'000	Mixed Gold Grade (g/t)	Oz 000	Tonne (t)'000	Total Gold Grade (g/t)	Gold Oz '000	Tonne (t) '000	W/O
2014	6,333	0.47	97	0	0.00	0	6,333	0.47	97	3,986	0.63
2015	7,046	0.47	106	7	0.47	0	7,053	0.47	106	3,313	0.47
2016	6,582	0.49	103	449	0.59	9	7,030	0.49	112	5,363	0.76
2017	6,107	0.52	102	914	0.61	18	7,021	0.53	119	5,783	0.82
2018	6,839	0.47	102	263	0.62	5	7,102	0.47	108	6,741	0.95
2019	5,415	0.51	89	1,498	0.74	35	6,913	0.56	124	5,115	0.74
2020	4,207	0.51	69	2,894	0.56	52	7,101	0.53	121	3,075	0.43
2021	2,650	0.48	41	4,398	0.63	89	7,048	0.57	130	4,912	0.70
2022	6,080	0.49	95	707	0.61	14	6,786	0.50	109	5,739	0.85
2023	4,949	0.55	88	765	0.62	15	5,714	0.56	103	4,678	0.82
Total	**56,207**	**0.49**	**892**	**11,895**	**0.62**	**238**	**68,102**	**0.52**	**1,129**	**48,705**	**0.72**

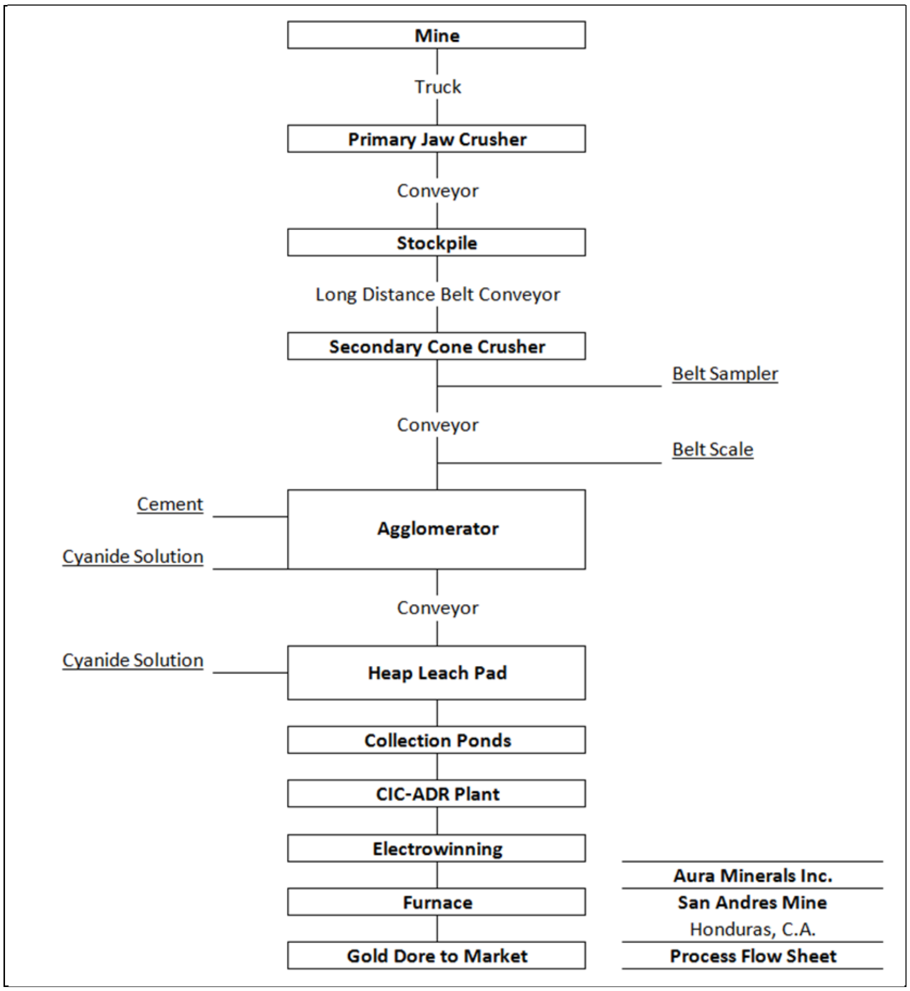
The ore is stacked on the leach pad in 8 m lifts on previously leached ore that has been ripped and prepared. The solution used for leaching comes from the Adsorption Desorption Refinery plant ("ADR") after the cyanide concentration has been replenished.

Gold is recovered through the ADR plant, which has 12 carbon columns that can be configured in a two or three train process with a nominal 500 m³/h per train. The assay lab which process both mine grade control samples and process plant samples is located in the same complex as the ADR. The gold produced at the ADR plant is analyzed prior to shipment for refining and sale.

The ADR plant is currently being updated to accommodate the required higher capacity for the process of 7 Mtpa of ore. The current number of carbon columns is sufficient for the expanded absorption capacity and all units have been inspected and are under a special maintenance program for improved reliability. For the absorption (stripping) part of the plant, modifications will be done to the strip vessel (addition of a pair of parallel basket strainers), the acid wash system will be upgraded (replacement of the dilute caustic tank), and a new neutralization tank will also be added in the short term. The refining system will have its capacity substantially increased with the additional of a number of cathodes, anodes and the replacement of the rectifiers.

The process flow sheet for San Andrés is presented in Figure 17-1.

Figure 17-1: Process Flow Sheet



17.2 Plant Design, Equipment Characteristics, and Specifications

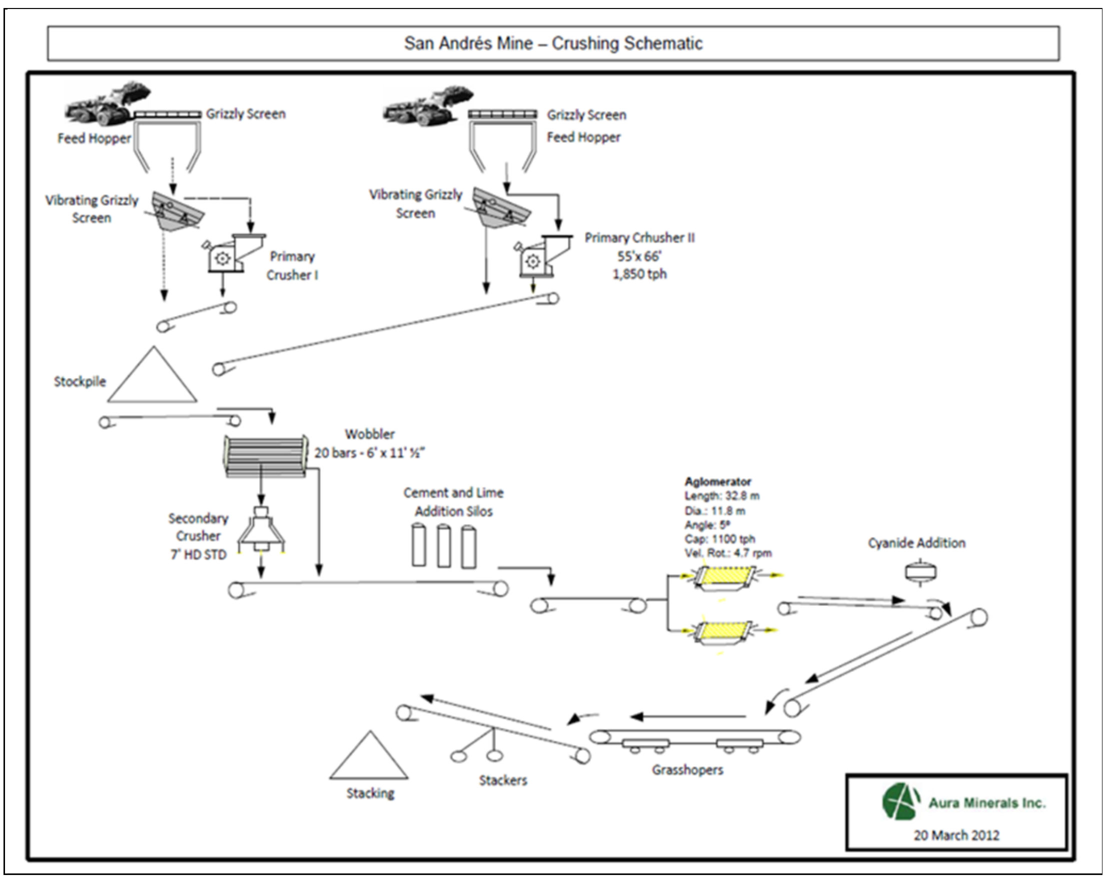
17.2.1 Crushing Circuit

The crushing circuit was designed with a daily operating time of 18 hours and a throughput nominal capacity of 1,100 t/h. For the expanded capacity, the system will operate 24 hours per day with an overall utilisation factor of 74%. In addition, a slight increase in nominal capacity is expected to occur with the replacement of the secondary crusher and the corresponding scalper feeder that was implemented in the fourth quarter of 2013. Later, a more significant increase in nominal capacity is anticipated when the parallel secondary crushing system is put in operation in the second half of 2014.

The ore is mined in the open pit and is hauled to one of two jaw crushers using contract haul trucks. All of the ore is processed through a two-stage crushing circuit and transported on conveyors before being stacked as the final product sized at 80% passing 2.5 inch. The “Water Tank” primary crusher is a 55 inch x 66 inch Telsmith Jaw crusher that produces a 6 inch product which is transported to the intermediate stockpile by a 490 m long conveyor. From the intermediate stockpile the material is fed to a second series of conveyors by the reclaim tunnel. The second series of conveyors totals 510 m. During 2009, a 44 inch x 48 inch Svedala primary crusher and conveyor system was installed in the Twin Hills area, with commissioning completed in the second quarter of 2010. Modifications to the stacking system were completed during the fourth quarter of 2010. These modifications allow the crushing and stacking of up to 1,100 t/h of rocky ore. From the reclaim tunnel the ore is passed through a Symons 7 foot Standard Cone crusher set at 2.5 inches. The secondary crusher system (the cone crusher and its scalping wobbler) is a bottleneck when processing wet ore. The feeder/scalper wobbler was replaced with a double-deck vibrating screen during the fourth quarter of 2013 and an additional secondary crusher system is planned to be installed in 2014 in order to achieve higher and more consistent throughput capacity. After the ore has been crushed, 2.5 kg/t to 4.0 kg/t of cement is added before reaching the agglomeration circuit. Between 1.5 kg/t to 4.0 kg/t of Lime is also added for pH control. In the two 10 m long x 3.5 m diameter agglomerators, the ore is retained and mixed for five minutes while adding intermediate solution to achieve the optimum moisture of 18%. Before reaching the stacker, the ore is sprinkled with 400 ppm cyanide solution at three different points on the conveyor belts.

Figure 17-2 below shows a Process Flow Diagram of the Crushing Circuit.

Figure 17-2: Process Flow Diagram of the Crushing Circuit



17.2.2 Heap Leach Pads

The ore is stacked on the leach pad in 8 m lifts on previously leached ore that has been ripped and prepared. The ore is left for 72 hours to cure before any solution is applied. The ore is leached for an average of 90 days before the area is allowed to dry and prepared for the next lift. The solution used for leaching comes from the ADR plant after the cyanide concentration has been replenished.

The Mine leach pad facility is a monolithic leach pad that has been constructed in multiple phases. Phase I was completed during initial construction and start-up, Phase II-A was an expansion that was constructed in 2002, and Phase II-B was another expansion that was completed in July 2005. The Phase III and IV heap leach pads are currently being loaded and are located immediately west of Phase IIB. The total capacity of the four phases of the leach pad facility is approximately 23.5 Mt. These leach pad facilities were designed by the consulting firm SRK, with all of the engineering for stability, water balances, liner specifications, and earth work specifications completed by them.

Production rates from the current mining operation show that Phases III & IV of the existing heap leach pad will reach full capacity in the 1st quarter of 2015. A new leach pad facility (Phase V), which will be hydraulically independent from the existing Phase I-IV facility has been designed by the consulting firm AMEC, Denver, USA. The Phase V heap leach pad is being constructed in stages and construction initiated in November 2012 with the first stage completed in Q2 2013. The Phase V heap leach pad is expected to be completed by May of 2015.

The Phase V heap leach pad expansion consists of a pad with a 32 hectare footprint, which partially overlaps with existing Phases II, III, and IV located immediately south of the proposed facility. This facility provides for approximately 12 million m^3 of ore storage, or 19 million tonnes of ore capacity. Further heap leach expansions may be constructed above or adjacent to the existing heap leach pads. A geomembrane liner will be installed at the base so that the existing and new heap leach pads will be hydraulically isolated.

17.2.3 Carbon ADR Plant

The operational gold extraction of the carbon ADR facility is approximately 96%, the current design capacity of the ADR facility is 1,000 m^3/h and with an estimated gold in solution feed grade to the circuit of 0.30 g/m3, the facility will produce approximately 8 oz Au/h.

The recovery process is achieved through the ADR plant. The plant has twelve carbon columns that can be configured in a two or three train arrangement with a capacity of up to 500m^3/h per train.

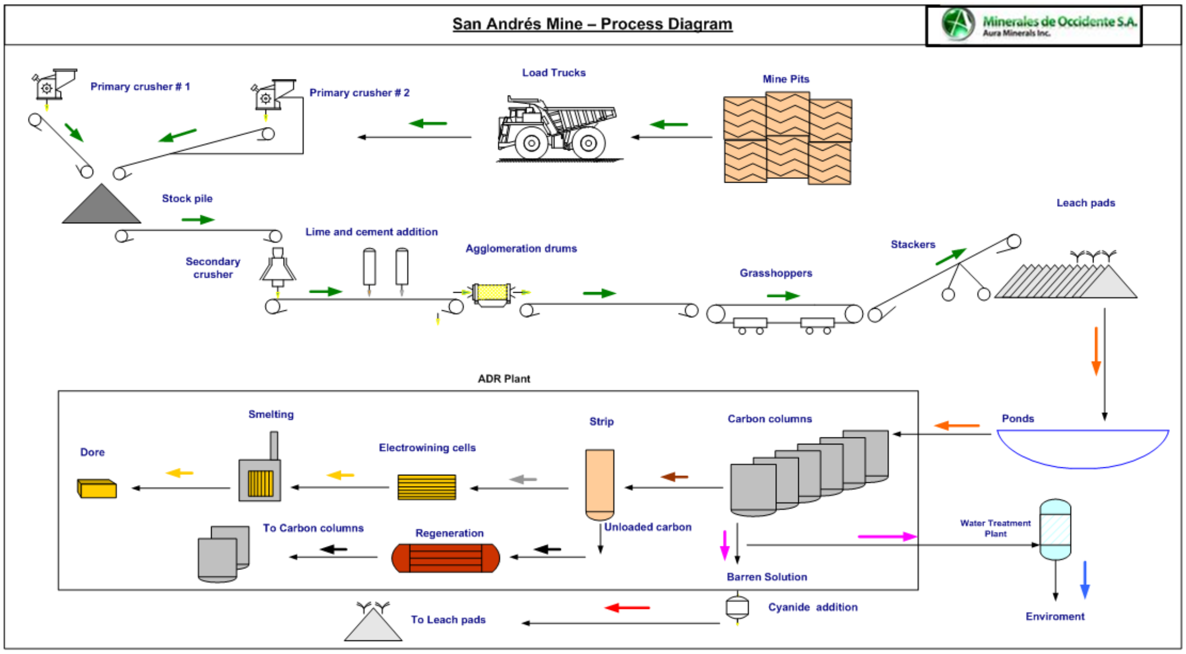
Prior to stripping, carbon is acid washed using a dilute hydrochloric acid solution. This acid solution dissolves the carbonates in the carbon micro pores. Following acid washing and stripping, barren carbon is sent for regeneration in the carbon reactivation circuit. The carbon reactivation system consists of one 5-tonne carbon reactivation kiln, operating at approximately 10 tonnes a day.

The gold recovery system consists of an electro-winning circuit with design capacities of 60 gallons per minute. The refinery has two parallel electro-winning cells currently equipped with 13 cathodes and 14 anodes each, running at a current of 1,000 and 2,500 A and a voltage of 15 V. The actual nominal capacity is significantly lower than the design value and for the expansion the electro-winning circuit will be retrofitted to operate with 74 cathodes and 76 anodes in each cell and the rectifiers will be replaced with new units of 3,000 A. In the refinery there is also a mercury-retort system for the removal of mercury prior to smelting. The assay lab is located in the same complex as the ADR plant. The gold produced at the ADR plant is analyzed prior to shipment to Johnson-Matthey Refining Corporation ("J-

M") for refining and sale. J-M analyzes the doré received prior to refining. The gold received by J-M is based on the metal purchased and refined and can vary significantly based on inventory and time of purchase; however, comparison of ore recovered shipment shows that the gold ounces as measured at the ADR plant are no more than 1% above the J-M measurements.

Figure 17-3 below shows a simplified Process Flow Diagram of the Gold Recovery Circuit.

Figure 17-3: Gold Recovery Circuit



18.0 PROJECT INFRASTRUCTURE

The Mine has been in operation since 1983 and has a well-developed infrastructure which includes power and water supply, warehouses, maintenance facilities, assay lab, and on-site camp facilities for management, staff and contractors. On-site communication includes radio, telephone, internet and satellite television services.

The Mine operates a diesel power generation plant for operational needs. Current power requirements are approximately 4Mw but future requirements rise to almost 7Mw due to increased hydraulic head and therefore increased pumping requirements. The cost of diesel generation is dependent almost entirely on the cost of diesel fuel. In late 2013, a solicitation for connection to the national power grid was made to the Honduras national power authority, Empresa Nacional de Energia Electrica (ENEE). ENEE has since responded that the connection is technically feasible and will be approved, reducing power costs by approximately 40%. The operating mode would be to buy power from the grid and maintain sufficient diesel generation capacity to back up the power grid. This is the assumed mode of operation in the LOM from 2015 forward. In addition, solar power options are being solicited to further reduce power costs during daylight hours.

Process water is supplied by rainwater run-off collected in the surge ponds and by direct pumping from a pump station in the perennial Río Lara adjacent to the CIC-ADR plant. Flow measurements of feeder streams indicate that the Río Lara flow rate is in excess of 100 m^3/h at the driest time of the year.

Chlorinated potable water for the town of San Andrés and camp facilities is supplied from a nearby 72,000 gallon storage tank which is fed via a 4-inch, 17-km metal water line from a source originating upstream from San Andrés along the Río Lara near the village of La Arena. Purified water for drinking and cooking is purchased from local suppliers.

If necessary, surplus water from the process plant is treated by several steps to perform the sequential removal of cyanide, arsenic, mercury and cobalt. Cyanide destruction takes place with the addition of calcium hypochlorite. Arsenic is then precipitated from the solution with ferrous sulfate. Following arsenic treatment, mercury is then encapsulated with a chelating agent ("Polymeric dithiocarbamate") which is highly selective for mercury. The precipitates of arsenic and mercury are removed from the solution using a coagulant and flocculant addition, followed by sedimentation in a pond. The treated water is then pumped to the final stage of the water treatment process for the removal of cobalt using an ion exchange resin. In order to maintain optimum working resin, water is filtered using filter presses in two stages prior to cobalt removal. Filtration removes all fine suspended sediments which were not removed by the coagulant and flocculant.

19.0 MARKET STUDIES AND CONTRACTS

The principal commodities at San Andrés are freely traded, at gold prices that are widely known, so the sale of any production is not a material concern to Aura. Aura has not engaged in any recent market studies for the purposes of selling gold.

The Company currently has several major contracts including the mining haulage contract, an explosives contract and several other commodity supply contracts such as cyanide, lime and cement. These contracts are within industry norms.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Management Plan

The management plan was formulated at the request of the government of Honduras and addresses the commitments made within the five Environmental Impact Assessments ("EIAs"), the Mitigation Contracts and recommendations issued by government agencies.

The plan defines and describes all references to the term "Best Management Practices" used in the EIAs. Overall, the plan allows for the orderly definition of commitments made to the Honduran government and to the Company's stakeholders for the protection of the environment and for mitigation of the environmental impacts caused by the construction and operation of the Project.

The management plan includes:

- Compliance with the International Cyanide Management Code, San Andrés is a certified operator.
- Environmental Monitoring Plan updated each year to adapt it to new sampling requirements; the last update for the water monitoring plan was done on 2013.
- Air quality monitoring plan, which includes; noise and particulate material of mobile and fixed emission sources.
- Contingency Plan updated in 2003 and reviewed in 2004. This Plan has been discussed with key personnel in the operation to ensure procedures described are appropriate according to any given situation.
- Materials Management Plan, consisting of management of hazardous and nonhazardous materials, construction and management of facilities (i.e., land fill and ancillary facilities), education on good housekeeping, and organization of waste recollection and disposal.
- Spilled Soil Management and Remediation Plan, updated in 2004, include the development of treatment sites and technologies to decontaminate polluted soils (i.e., bioremediation of oil polluted soils in concrete tanks); Minosa possesses a THC analysis kit to verify THC concentration.
- Erosion Control Plan is updated every year to address yearly priorities.
- Explosives Management Plan, designed to comply with the Honduran and U.S. explosives management regulations.
- Surface and Underground Water Management Plan, updated in 2004.
- Mine Waste Management Plan, updated yearly; main focus is to use a greater proportion of waste rock as material for contouring former mining areas.
- Wastewater Treatment and Management Plan, updated yearly depending on the quality of the water to be treated and/or managed Health and Safety Plan, updated yearly under the commission of the Safety and Occupational Health Department. This Plan consists of six main components; Occupational Clinic, program to assess the working environment, definition of required personal protection equipment, safety training program, mix health and safety Commission, health and safety surveillance.
- Reforestation Plan, updated in 2009 (the original plan was approved by COHDEFOR, the 2009 plan is pending approval by ICF, Forestry Conservation Institute) and its implementation is responsibility of a forest engineer.

- A Conceptual Reclamation and Closure Plan is in place together with the Canadian GAAP Asset Retirement Obligation calculations.
- Plan of Sewage and Potable Water Management, permanent since 2002.
- Plan to encapsulate waste rock with AMD (Acid Mine Drainage) potential, written in 2004, and reviewed periodically.

To the best of the Authors' knowledge, there are no material environmental issues that affect Minosa's ability to extract the mineral resources.

Section 4.4 of this Report discusses permitting in detail.

The communities within the direct area of Mine influence have had a number of minor protests against Minosa and the Mine during late 2013 and early 2014. The protests have have been settled through active engagement but have resulted in production stoppages, and or have prevented the delivery of goods and equipment, but have not negatively impacted the forecasted production for the Mine.

21.0 CAPITAL AND OPERATING COST ESTIMATES

Capital costs for the Mine operation are indicated in Table 21-1. The capital cost for the Mine averages approximately US$9 million per year until 2023 not including any reclamation and closure costs. This represents approximately US$1.32/t of ore. The majority of the capital cost estimate, obtained through supplier quotes, is associated with the new leach pad construction projects, as well as community and sustaining capital projects. The majority of the Community costs are for housing projects. The community and sustaining cost estimates were developed by obtaining vendor quotes and in-house estimates. Also included is the improvement and repairs to the power generating system in 2014.

The following is excluded from the capital cost estimate:

- Project financing and interest charges
- Owner's costs
- Land acquisition, leases rights of way and water rights
- Permits, fees and process royalties
- Environmental impact studies
- Taxes
- Import duties and custom fees
- Cost of geotechnical investigation
- Working capital
- Sunk costs
- Pilot Plant and other testwork
- Relocation of any facilities, if required
- Catering costs.

Table 21-1: Capital Expenditures

Area	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	Total
Mine	231,113	1,251,500	0	0	2,800,000	0	0	0	0	0			**4,282,613**
Exploration	1,341,301	1,728,000	1,440,000	1,440,000	1,152,000	1,152,000	1,152,000	1,152,000	1,152,000				**11,709,301**
Process	6,227,125	4,692,628	5,980,000	6,300,000	5,900,000	4,000,000	4,000,000	2,250,000	0	0			**39,349,753**
Maintenance	648,052	647,000	556,583	565,000	500,000	720,000	500,000	535,000	525,000	545,000			**5,741,635**
Community	3,245,080	4,580,965	6,335,713	4,219,735	478,074	0	0	0	0	0			**18,859,567**
Environmental	26,814	143,500	32,000	25,000	0	0	32,000	0	0	0			**259,314**
IT	36,468	96,319	54,334	25,000	0	0	0	0	0	0			**212,121**
Other	50,000	0	0	100,000	100,000	100,000	100,000	100,000	100,000	100,000			**750,000**
Reclamation and Closure								1,290,218	1,290,218	6,370,629	1,290,218	5,563,506	**15,804,789**
Total	**11,805,953**	**13,139,911**	**14,398,630**	**12,674,735**	**10,930,074**	**5,972,000**	**5,784,000**	**5,327,218**	**3,067,218**	**7,015,629**	**1,290,218**	**5,563,506**	**96.969,092**

The 2013 unit operating cost breakdown for the Mine is shown in Table 21-2.

Table 21-2: Operation Cost Breakdown – 2013

Area	Unit Costs
Mine (US$/t moved)	
Drilling	0.24
Blasting	0.41
Haulage / Loading / Auxiliary (Contractor)	1.91
Geology, QA, Topography	0.01
Mine Admin	0.07
Total Mine (US$ / t moved)	**2.63**
Process ($/t leached)	
Crushing	1.13
Stacking + Agglomeration	2.08
Leaching	1.68
ADR, Refinery, Metallurgy, Lab, Plant Admin, Maintenance	1.89
G&A	2.09
Total ($/t leached)	**8.88**

For the economic analysis in Section 22, and the expansion to 7Mt/yr, the following operating costs have been used:

Table 21-3: Operation Cost Breakdown – 2014 Forecast LOM

Operation Cost Breakdown – LOM 2014	**Total LOM**
Mine (US$/t Moved)	
Drilling	0.20
Blasting	0.36
Haulage/Loading/Auxiliary (Contractor)	1.66
Geology, QA, Topography	0.01
Technical Services	0.03
Mine Administration	0.03
Total Mine (US$/t Moved)	**2.29**
Process (US$/t Leached)	
Crushing	0.98
Stacking + Agglomeration	1.80
Leaching,	1.46
ADR, Refinery, Metallurgy, Lab, WTP, Plant Admin, Maintenance	1.63
G&A	1.81
Total (US$/t Leached)	**7.67**

The 2014 LOM cost forecast is justified through increase in tonnes processed, and recent reductions of major commodity supply such as cyanide, fuel and lime. Minosa has undergone a major cost reduction program where all areas of the business were analyzed. For loading and hauling, the mine has brought in another contractor resulting in better unit rates. One of major cost drivers as mentioned in Section 18 was the fuel required for operating the power generators. Available grid power is estimated to reduce process costs by 40%.

22.0 ECONOMIC ANALYSIS

A post-tax Cash Flow Model has been developed by Aura from the LOM production schedule and capital and operating cost estimates. Recoveries of 76% oxide, 57% mixed material as well as an overall ADR recovery of 96% has been used. A review by Aura of the Net Present Value ("NPV") projections indicated that using the operating and capital cost estimates noted in Section 21, and a gold price of $1,300, the after tax NPV is positive, supporting the Mineral Reserve designation.

For the purposes of the economic evaluation, the following taxes and NSRs have been applied:

- 1.5% NSR payable to an arm's length third party (part of the original purchase price)
- 2.0% Municipality tax (2013)
- 2.0% Security tax (2011)
- 1.0 % INHGEOMIN tax (2013 - Honduras' mining regulatory agency)

The sensitivity analysis has been completed that examined gold price, capital and operating costs ranging from +10 to -10%. The sensitivity analysis has been reviewed and it is concluded that when the gold price is reduced by 10%, or operating costs increase by 10%, or the capital costs increase by 10% the net present value remains positive.

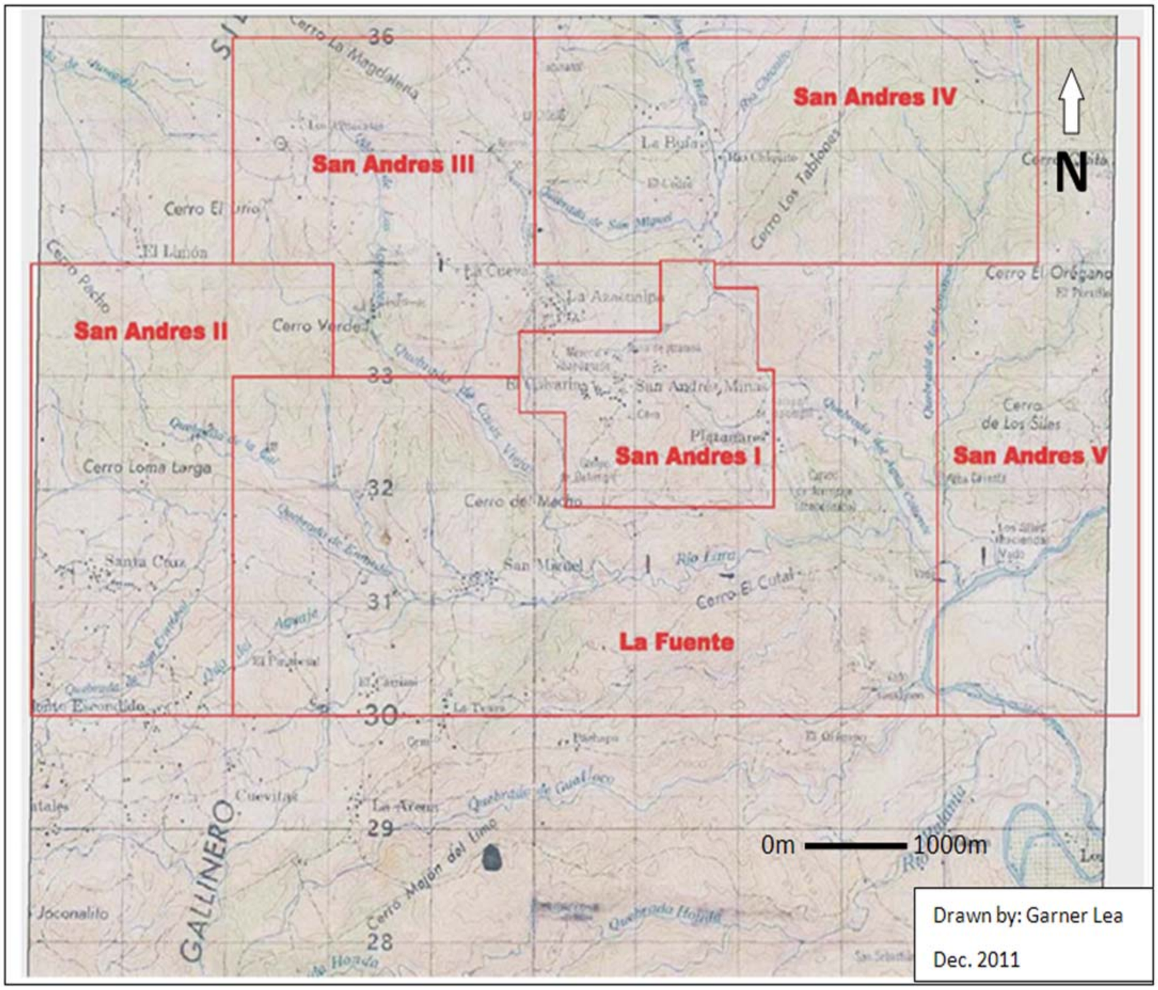
23.0 ADJACENT PROPERTIES

In May 2002, Minosa filed applications for four additional mining concessions totaling 3,768 ha: San Andrés II (900 ha), San Andrés III (869 ha), San Andrés IV (999 ha), and San Andrés V (1,000 ha), however, the concessions have not been granted yet pending the final resolution of a new Honduran mining law. Minosa is required to pay fees as if the concessions were active. On January 15, 2007, Minosa paid US$0.75/ha, which corresponds to the fifth year fee for these concessions. The payments were made upon the completion of each year, and this fee was last paid in 2006.

Neighbouring the San Andrés concession to the south and east is the La Fuente concession which has been held by Minera Energética Centro Americana (MECA) since 1995 (Figure 23-1). On June 3, 2002, Minosa entered into a contractual agreement with Empresa Minero Energética Centroamericana, S.A. de C.V. ("EMECA"), a local Honduran company that holds the La Fuente concession. Pursuant to this agreement Minosa has the explicit exploration rights over the area.

The current Honduran mining law states that after the 8th year concessions are required to have a minimum production level of US$500 of sales per hectare or they are subject to a penalty (the fee rises to $3.00/ha and continues to double every year thereafter until these production levels are met). EMECA has held the concession since December 18, 1995 and as such would be required to have minimum production levels by 2004. In order to avoid paying the penalty for not having production, EMECA applied to DEFOMIN (Honduras' mining regulatory authority) to subdivide the concession into three smaller concessions covering the exact same area. Since all new concession applications are frozen the La Fuente concession, with the application to subdivide also became frozen and as a result additional exploration work is prohibited. Once the concession applications are granted, it is assumed that Minosa will have to negotiate another agreement with EMECA similar to that signed in 2002.

Figure 23-1: Adjacent Property Map



24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 2012 and 2013 Reconciliation

The Mine carries out reconciliation between the long-term block model and the short-term production estimate produced from grade control blast holes, this is completed on a monthly basis throughout the year. Table 24-1 presents the results of the monthly reconciliation for 2012 and Table 24-1 presents 2013.

The short-term production estimate contained 28.4% more tonnes at a 5.8% higher grade than was predicted by the long-term block model for 2012 and 2.98% more tonnes at a 2.60% higher grade than was predicted by the 2013, long-term block model. A significant reduction in variance in both the tonnes and grade between the long-term and short-term models is a direct result of the changes made at the start of 2013, including the use of production blast hole data into the long-term model. In 2012 the 5.8% grade variance is slightly over the industry standard of 5% acceptable variance however this reduces in 2013 to 2.60% and is typical for this style of gold deposit. This is mainly due to additional localized high grade mineralization, which is poorly identified in the long-term global model. The difference in metric tonnes is due to a lack of information which causes the model to report more waste material than there actually is.

There are also certain months that show large differences between the long-term and short-term production estimates. To determine if these large differences would be considered significant, a three-month (quarterly) moving average was used to reduce the variability of the differences that can occur in a single month.

Table 24-1: Reconciliation between Long Term and Short-term Production Estimate - 2012

	Long Term Model			Short-term Production Estimate		
Period	**Tonnes**	**Grade (g/t)**	**Metal Au (oz)**	**Tonnes**	**Grade (g/t)**	**Meta Au (oz)**
31-Jan-12	362,841	0.625	7,297	349,356	0.609	6,840
29-Feb-12	287,702	0.635	5,876	348,555	0.705	7,896
31-Mar-12	359,184	0.686	7,920	390,714	0.734	9,220
30-Apr-12	362,003	0.643	7,481	404,477	0.699	9,090
31-May-12	318,787	0.664	6,800	448,896	0.751	10,838
30-Jun-12	252,098	0.653	5,290	347,876	0.724	8,099
31-Jul-12	208,899	0.580	3,892	383,074	0.716	8,821
31-Aug-12	182,553	0.582	3,417	243,807	0.598	4,687
30-Sep-12	19,605	0.488	2,503	255,763	0.593	4,874
31-Oct-12	223,731	0.491	3,532	294,701	0.508	4,812
30-Nov-12	277,492	0.525	4,684	468,249	0.440	6,629
31-Dec-12	409,035	0.529	6,959	437,131	0.531	7,465
Total	**3,403,930**	**0.600**	**65,651**	**4,372,599**	**0.635**	**89,270**

Table 24-2: Reconciliation between Long Term and Short-term Production Estimate - 2013

	Long Term Model			Short-term Production Estimate		
Period	**Tonnes**	**Grade (g/t)**	**Metal Au (oz)**	**Tonnes**	**Grade (g/t)**	**Metal Au (oz)**
31-Jan-13	462,269	0.635	9,436	467,742	0.558	8,391
29-Feb-13	344,497	0.603	6,675	446,101	0.572	8,204
31-Mar-13	495,760	0.612	9,760	462,759	0.605	9,001
30-Apr-13	524,721	0.648	10,931	515,534	0.560	9,282
31-May-13	597,872	0.612	11,771	520,148	0.607	10,151
30-Jun-13	610,384	0.664	13,024	482,806	0.755	11,720
31-Jul-13	549,712	0.561	9,912	537,469	0.629	10,869
31-Aug-13	481,757	0.512	7,929	498,871	0.565	9,062
30-Sep-13	358,089	0.482	5,548	388,558	0.613	7,658
31-Oct-13	400,569	0.461	5,931	584,066	0.550	10,328
30-Nov-13	320,430	0.475	4,896	378,919	0.524	6,384
31-Dec-13	160,733	0.502	2,593	182,056	0.500	2,927
Total 2013	5,306,793	0.577	98,404	5,465,030	0.592	103,976

Figure 24-1 displays the quarterly moving averages of the percentage differences in tonnes, grade and metal for the 2012 reconciliation results. The dashed black line shows the differences in tonnes percentage, the solid red line shows the differences in grade percentage and the dotted green line shows the differences in metal percentage.

Upper and lower threshold limits of 15% have been defined as the acceptance level for any quarterly moving average difference, confirming the reliability of the mineral resource estimate.

Figure 24-1 illustrates that there is significant deviation from the tolerance levels in both tonnage and grade compared to the Long-term model. This variance was largely due grade control delineation drilling expanding the resource into areas which were poorly defined by wider spaced drilling. These areas were subsequently mined. The inclusion of the production blast holes into the December 2012 Long-term model has significantly reduced this variance in the current production areas along with the infill drilling within Cerro Cortez; this can be seen in Figure 24-2.

Figure 24-1: Quarterly Moving Average Comparison of Long-term model and Short-term Production Estimate - 2012

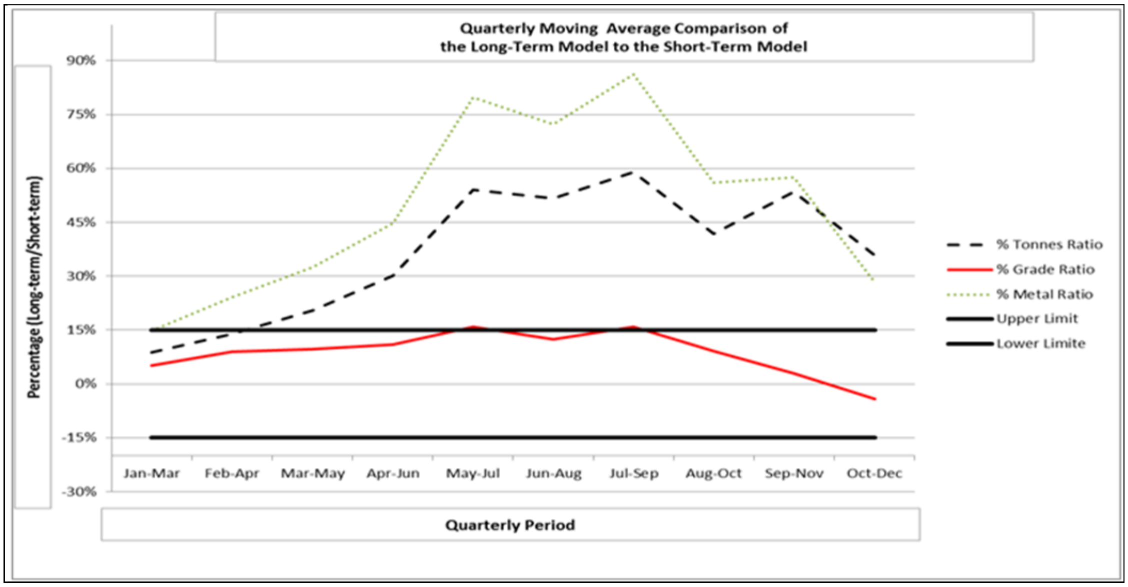
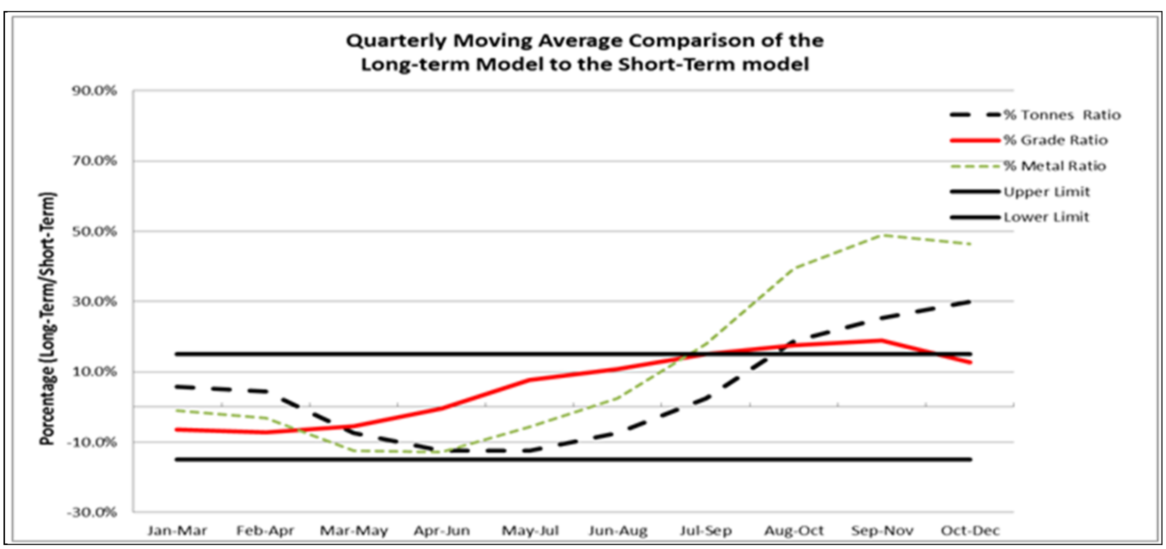


Figure 24-2 displays the quarterly moving averages of the percentage differences in tonnes, grade and metal for the 2013 reconciliation results. The dashed black line shows the differences in tonnes percentage, the solid red line shows the differences in grade percentage and the dotted green line shows the differences in metal percentage.

Figure 24-2 illustrates that there is a marked improvement on 2012 and the deviation from the tolerance levels in both tonnage and grade compared to the Long-term model is only significant in the last quarter of 2013. This variance was largely due mining progressing into wider spaced drilled areas of Cerro Cortez where additional tonnes and higher ore grades were seen in the production drilling compared to the long-term model.

Figure 24-2: Quarterly Moving Average Comparison of Long-term model and Short-term Production Estimate – 2013



25.0 INTERPRETATION AND CONCLUSIONS

Aura has prepared this Report compliant with NI 43-101 on the updated Mineral Resources and Mineral Reserves pertaining to its 100% owned San Andrés Mine, located in the municipality of La Unión, in the Department of Copán, Honduras.

The major assets and facilities associated with the Mine are:

- An open pit mine that has treated approximately 68 Mt of material at an average grade of 0.52 grams per tonne gold ("g/t Au") from 2014 to the end of 2023. An increase in throughput to 7 Mt/yr with modest capital expenditure was executed in 2013 and 2014. A plant, heap leach pads with carbon recovery system that produces gold and a cyanide destruction circuit as well as mine and plant infrastructure including office buildings, shops, and equipment. Major cost reductions are anticipated by switching from diesel generators to grid power.
- On-site accommodations, recreation facilities, and cafeteria for the workforce.
- A controlled solution discharge system including pregnant, intermediate, and barren ponds and a security dam where excess solutions, generally during the rainy season, are collected and neutralized prior to final discharge.

The San Andrés deposit is classified as an epithermal gold deposit associated with extension structures within tectonic rift settings. These deposits commonly contain gold and silver mineralization associated with banded quartz veins. At San Andrés, however, silver is not economically material. Gold occurs in quartz veins predominantly comprised of colloform banded quartz (generally chalcedony with lesser amounts of fine comb quartz, adularia, dark carbonate, and sulphide material). The gold mineralization is deposited as a result of cooling and the interaction of hydrothermal fluids with groundwater and the host rocks. The hydrothermal fluids may have migrated some distance from the source; however, there is no clear evidence at San Andrés that the fluids or portions of the fluids have been derived from magmatic intrusions.

The 2013 Mineral Resources estimated by Aura, using a long term US$1,600 gold price and a 0.23 g/t Au cutoff for oxide and a 0.30 g/t cutoff for mixed material, is a total of 104.8 Mt of Measured and Indicated Mineral Resources at an average grade of 0.49 g/t Au and an Inferred Mineral Resource of 4.3 Mt at an average grade of 0.49 g/t Au. Note that the Mineral Resources are inclusive of Mineral Reserves. Also note that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The 2013 Mineral Reserves estimated by Aura total 68.1 Mt of Proven and Probable Mineral Reserves at an average grade of 0.52 g/t Au. The Proven and Probable Mineral Reserves estimated using a cutoff grade 0.28 g/t Au for oxide material and a cutoff grade of 0.37 g/t Au used for the mixed material using long term gold price of $1,3000/oz.

The Authors note that the Mineral Reserves are estimated in accordance with the CIM definitions and are considered NI 43-101 compliant. The reported Mineral Reserve estimate is reasonable for the remaining Life of Mine plan.

The phrase "reasonable prospects for economic extraction" implies a judgement by the Authors of this Report. Reasonable prospects for economic extraction will be closely related to the prevailing price of gold, gold pricing assumptions, operating costs, metallurgical recoveries, available land and other key assumptions contained within this Report. Key factors influencing the price of gold include the supply of and demand for these commodities, the relative strength of currencies (particularly the U.S. dollar) and macroeconomic factors such as current and future expectations for inflation and interest rates. The Authors believe that the short-to-medium term economic environment is likely to remain relatively supportive for gold but with continued volatility. Readers are encouraged to read this Report in its entirety.

26.0 RECOMMENDATIONS

The Authors recommend the following:

- A metallurgical study on the Zona Buffa Mineral Resources to determine leach recovery for inclusion of these resources into reserves. The approximate cost of this study is $5,000;
- As mining progresses, continued reconciliation needs to be reviewed and if parameters change, an update of the Mine plan should be developed;
- Operating costs should be reviewed on a regular basis to ensure operating cut-offs remain valid;
- The recovery rate for oxide, mixed and blends containing these types of ore should continue to be monitored and compared to equivalent column tests. It is also recommended that the on-going program of column tests (performed at site) is expanded for investigations of future production in accordance to the new Mine plan;
- Additional specific gravity measurements should be conducted on mixed zone material to determine an appropriate specific gravity that can be incorporated into the block model. This is estimated to cost $25,000; and
- That the operation continues with the QA/QC programme on the exploration and the production blast hole sampling to ensure that a comprehensive data set is obtained for future estimates, which yearly is estimated to be $15,000;
- Exploration of the Aguas Calientes and Banana Ridge areas, where there are a number of high grade intercepts is likely to see significant expansion to the resources and reserves.

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28.0 DATE AND SIGNATURE PAGE

This Technical Report entitled *Mineral Resource and Mineral Reserve Estimate on the San Andrés Mine in the Municipality of la Unión, in the Department of Copán, Honduras* and dated July 2, 2014 with an effective date of December 31, 2013 was prepared by the following co-authors:

Dated July 2, 2014	(Signed & Sealed) "*Bruce Butcher"* Bruce Butcher P. Eng. Vice President, Technical Services, Aura Minerals Inc.
Dated July 2, 2014	(Signed & Sealed) "*Benjamin Bartlett*" Benjamin Bartlett FAusIMM. Manager Mineral Resources, Aura Minerals Inc.
Dated July 2, 2014	(Signed & Sealed) "*Persio Rosario*" Persio Rosario – P. Eng. Former Principal Metallurgist, Aura Minerals Inc.

CERTIFICATE OF QUALIFIED PERSON

As a co-author of the Technical Report entitled "Mineral Resource and Mineral Reserve Estimates on the San Andrés Mine in the Municipality of la Unión, in the Department of Copán, Honduras", dated 2nd July, 2014 with an effective date of December 31, 2013 and prepared for Aura Minerals Inc. (the "Company"), I, Bruce Butcher, do hereby certify that:

1) I am the Vice President, Technical Services of the Company with an office at 595 Howe Street, Suite 308, Vancouver, British Columbia, V6C 2TC.

2) I am a member of the Association of Professional Engineers and Geoscientists of British Columbia (APEGBC, member #29970). I graduated from Queens University with a Bachelor of Applied Science degree in Mining Engineering in 1988.

3) I have practiced my profession continuously since 1988 and have been involved in the design, construction and operation of mines in Canada, Chile, Mexico, Honduras, Brazil, the United States, and Peru.

4) I am a "qualified person" as that term is defined in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101").

5) My most recent personal inspection of the San Andrés Mine was from January 20 to 23, 2014. Prior involvement with the Mine includes Life of Mine planning, exploration planning, Geotechnical support and author of the previous NI 43-101 report on the San Andrés Mine published in March 2012.

6) I am not independent of the Company applying the test set out in section 1.5 of NI 43-101.

7) I am responsible for sections 1 - 5, 15, 16, 19, 20, 21, 22, 25, 26, 27 and 28 of the Technical Report.

8) As of the date of this certificate, to the best of my knowledge, information and belief, the portions of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

9) I have read NI 43-101 and confirm that the portions of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.

Dated this 2nd day of July, 2014

"Bruce Butcher" (Signed and sealed)
Bruce Butcher, B.Sc., P.Eng.
Vice President, Technical Services, Aura Minerals Inc.

CERTIFICATE OF QUALIFIED PERSON

As a co-author of the Technical Report entitled "Mineral Resource and Mineral Reserve Estimates on the San Andrés Mine in the Municipality of la Unión, in the Department of Copán, Honduras" dated 2nd July, 2014 with an effective date of December 31, 2013 and prepared for Aura Minerals Inc. (the "Company"), I, Benjamin Bartlett, do hereby certify that:

1) I am the Manager, Mineral Resources of the Company with an office at suite 595 Howe Street, Suite 308, Vancouver, British Columbia, V6C 2TC, Canada.

2) I am a Fellow of the Australian institute of Mining and Metallurgy (FAusIMM, 990986). I graduated from Otago University, New Zealand, with a Bachelor of Science with Honours in Geology in 1993.

3) I have practiced my profession continuously since 1994 and have been involved in the estimation of mineral resources of precious and base metal deposits in Australia, Papua New Guinea, Russia, Kyrgystan, Honduras, Mexico, and Brazil.

4) I am a "qualified person" as that term is defined in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101").

5) I have been responsible for the Geological database, models and Mineral Resource estimates of the San Andrés Mine since January, 2012 and my most recent personal inspection of the San Andrés Mine was from November 28 to December 2, 2013.

6) I am not independent of the Company applying the test set out in section 1.5 of NI 43-101.

7) I am responsible for sections 6, 7, 8, 9, 10, 11, 12, 14, 23 and 24 of the Technical Report, as well as those portions of sections 1, 25 and 26 that pertain to the foregoing sections.

8) As of the date of this certificate, to the best of my knowledge, information and belief, the portions of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

9) I have read NI 43-101 and confirm that the portions of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.

Dated this 2nd day of July, 2014

<u>*"Benjamin B Bartlett" (Signed and sealed)*</u>
Benjamin B Bartlett, B.Sc. (Hons), FAusIMM.
Manager Mineral Resources, Aura Minerals Inc.

CERTIFICATE OF QUALIFIED PERSON

As a co-author of the Technical Report entitled "Mineral Resource and Mineral Reserve Estimates on the San Andrés Mine in the Municipality of la Unión, in the Department of Copán, Honduras" dated 2nd July, 2014 with an effective date of December 31, 2013 and prepared for Aura Minerals Inc. (the "Company"), I, Persio P. Rosario, do hereby certify that:

1) I was employed as the Principal Project Metallurgist of the Company until April 1st, 2014 with an office at Suite 308 – 595 Howe Street, Vancouver, British Columbia, Canada V6C 2T5. My participation on the preparation of this report was concluded in March 2014.

2) I am a member of the Association of Professional Engineers and Geoscientists of British Columbia (APEGBC, member #32355). I graduated from Mackenzie University, Brazil, with a Bachelor of Applied Science in Mechanical Engineering in 1989, and from the University of British Columbia with a Masters in Applied Science in 2003 and with a Doctor of Philosophy in 2010, both in Mining & Mineral Process Engineering.

3) I have practiced my profession continuously since 2001 and have been involved in the design, construction and operation of mines in Canada, Peru, Panama, U.S.A., Mexico, Brazil and Honduras.

4) I am a "qualified person" as that term is defined in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101").

5) My most recent personal inspection of the San Andrés Mine was from December 9 to 13, 2013. Prior involvement with the Mine includes metallurgical testwork supervision, optimization study and design of process modifications for the comminution plant, review of operation's progress against budgets and assessment of operating practices.

6) I am not independent of the Company applying the test set out in section 1.5 of NI 43-101.

7) I am responsible for sections 13, 17, and 18 of the Technical Report as well as those portions of sections 1, 25 and 26 that pertain to the foregoing sections.

8) As of the date of this certificate, to the best of my knowledge, information and belief, the portions of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

9) I have read NI 43-101 and confirm that the portions of the Technical Report that I am responsible for have been prepared in compliance with the NI 43-101.

Dated this 2nd day of July, 2014

"Persio P. Rosario" (Signed)
Persio P. Rosario, M.A.Sc, PhD, P.Eng
Former Principal Project Metallurgist of Aura Minerals Inc.

APPENDIX 1

GLOSSARY OF MINING AND RELATED TERMS AND ABBREVIATIONS

GLOSSARY OF MINING AND RELATED TERMS AND ABBREVIATIONS

The following is a glossary of certain mining terms and abbreviations that may be used in this Technical Report.

A

Adit	A horizontal underground opening, open to the atmosphere at both ends.
Assay	A chemical test performed on a sample of ores or minerals to determine the amount of valuable metals contained.
Au	Gold.
Aura Minerals	Aura Minerals Inc., including, unless the context otherwise requires, the Company's subsidiaries.

B

Blasthole	A drill hole in a mine that is filled with explosives in order to blast loose a quantity of rock.
Bulk sample	A large sample of mineralized rock, frequently hundreds of tonnes, selected in such a manner as to be representative of the potential mineral deposit (orebody) being sampled and used to determine metallurgical characteristics.
Bullion	Precious metal formed into bars or ingots.
By-product	A secondary metal or mineral product recovered in the milling process.

C

Channel sample	A sample composed of pieces of vein or mineral deposit that have been cut out of a small trench or channel, usually about 10 cm wide and 2 cm deep.
Chip sample	A method of sampling a rock exposure whereby a regular series of small chips of rock is broken off along a line across the face, back or wall.
CIM	The Canadian Institute of Mining, Metallurgy and Petroleum.
CIM Standards	The CIM Definition Standards on Mineral Resources and Mineral Reserves adopted by CIM Council from time to time. Last revised on December 11, 2005 and currently undergoing a review process for a further update possibly in 2011.
Concentrate	A fine, powdery product of the milling process containing a high percentage of valuable metal.
Contact	A geological term used to describe the line or plane along which two different rock formations meet.
Core	The long cylindrical piece of rock, about an inch in diameter, brought to surface by diamond drilling.
Core sample	One or several pieces of whole or split parts of core selected as a sample for analysis or assay.
Custom smelter	A smelter which processes concentrates from independent mines. Concentrates may be purchased or the smelter may be contracted to do the processing for the independent company.
Cutoff grade	The lowest grade of mineralized rock that qualifies as ore grade in a given deposit, and is also used as the lowest grade below which the mineralized rock currently cannot be profitably exploited. Cutoff grades vary between deposits depending upon the amenability of ore to gold extraction and upon costs of production.
CV	The standard deviation divided by the mean.

Cyanidation	A method of extracting exposed gold or silver grains from crushed or ground ore by dissolving it in a weak cyanide solution. May be carried out in tanks inside a mill or in heaps of ore out of doors.
Cyanide	A chemical species containing carbon and nitrogen used to dissolve gold and silver from ore.
D	
Dacite	The extrusive (volcanic) equivalent of quartz diorite.
Dilution	Rock that is, by necessity, removed along with the ore in the mining process, subsequently lowering the grade of the ore.
Diorite Dip	The angle at which a vein, structure or rock bed is inclined from the horizontal as measured at right angles to the strike.
E	
Epithermal	Hydrothermal mineral deposit formed within one kilometre of the earth's surface, in the temperature range of 50–200°C.
Epithermal deposit	A mineral deposit consisting of veins and replacement bodies, usually in volcanic or sedimentary rocks, containing precious metals or, more rarely, base metals.
Exploration	Prospecting, sampling, mapping, diamond drilling and other work involved in searching for ore.
F	
Fault	A break in the Earth's crust caused by tectonic forces which have moved the rock on one side with respect to the other.
Fold.	Any bending or wrinkling of rock strata.
Footwall	The rock on the underside of a vein or mineral deposit.
Fracture	A break in the rock, the opening of which allows mineral-bearing solutions to enter. A "cross-fracture" is a minor break extending at more-or-less right angles to the direction of the principal fractures.
G	
g/t	Grams per metric tonne.
Galena	Lead sulphide, the most common ore mineral of lead.
gpt	Grams per tonne.
Grade	Term used to indicate the concentration of an economically desirable mineral or element in its host rock as a function of its relative mass. With gold, this term may be expressed as grams per tonne (g/t) or ounces per tonne (opt).
Gram	0.0321507 troy ounces.
H	
Hanging wall	The rock on the upper side of a vein or mineral deposit.
High grade	Rich ore. As a verb, it refers to selective mining of the best ore in a deposit.
Host rock	The rock surrounding a mineral deposit or orebody.
HQ	Drill core with a 63.5 millimetre diameter.
Hydrothermal	Processes associated with heated or superheated water, especially mineralization or alteration.

I

Indicated Mineral Resource	An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.
Inferred Mineral Resource	An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.
Intrusive	A body of igneous rock formed by the consolidation of magma intruded into other.

K

km	Kilometre(s). Equal to 0.62 miles.
kg/t	kilograms per tonne.
kg	kilograms (metric).

L

Leaching	The separation, selective removal or dissolving-out of soluble constituents from a rock or ore body by the natural actions of percolating solutions.
Limestone	A bedded, sedimentary deposit consisting chiefly of calcium carbonate.

M

m	Metre(s). Equal to 3.28 feet.
Marble	A metamorphic rock derived from the recrystallization of limestone under intense heat and pressure.
Measured Mineral Resource	A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.
Metallurgy	The science and art of separating metals and metallic minerals from their ores by mechanical and chemical processes.
Metamorphic	Affected by physical, chemical, and structural processes imposed by depth in the earth's crust.

M

Milldrum used for the grinding	A plant in which ore is treated and metals are recovered or prepared for smelting; also a revolving of ores in preparation for treatment.
Mine	An excavation beneath the surface of the ground from which mineral matter of value is extracted.
Mineral	A naturally occurring homogeneous substance having definite physical properties and chemical composition and, if formed under favourable conditions, a definite crystal form.
Mineral Claim	That portion of public mineral lands which a party has staked or marked out in accordance with federal or state mining laws to acquire the right to explore for and exploit the minerals under the surface.
Mineral Deposit	A mass of naturally occurring mineral material, e.g. metal ores or non metallic minerals, usually of economic value, without regard to mode of origin.
Mineralization	The process or processes by which mineral or minerals are introduced into a rock, resulting in a valuable or potentially valuable deposit.
Mineral Resource	A concentration or occurrence of diamonds, natural, solid, inorganic or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the earth's crust in such form and quantity and of such grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a mineral resource are known, estimated or interpreted from specific geological evidence and knowledge. The term mineral resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which mineral reserves may subsequently be defined by the consideration and application of technical, economic, legal, environmental, socio-economic and governmental factors. The phrase reasonable prospects for economic extraction imply a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. A mineral resource is an inventory of mineralization that under realistically assumed and justifiable technical and economic conditions might become economically extractable. The term mineral resource used in this Technical Report is a Canadian mining term as defined in accordance with NI 43-101 – Standards of Disclosure for Mineral Projects under the guidelines set out in the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM), Standards on Mineral Resource and Mineral Reserves Definitions and guidelines adopted by the CIM Council on December 11, 2005 (the CIM Standards).
Mt	Millions of tonnes, with a metric ton of 1,000 kilograms (2,205 pounds).

N

NaCN.	Sodium Cyanide
Net Smelter Return (NSR)	A payment made by a producer of metals based on the value of the gross metal production from the property, less deduction of certain limited costs including smelting, refining, transportation and insurance costs.
NI 43-101	National Instrument 43-101 Standards of Disclosure for Mineral Projects as adopted by Canadian Securities Administrators.

NQ	Drill core with a 47.6 millimetre diameter.
O	
Open Pit	A mine that is entirely on surface. Also, referred to as open-cut or open-cast mine.
Ore	Mineralized material that can be extracted and processed at a profit.
Orebody	A term used to denote the mineralization contained within an economic mineral deposit.
Ounce	A measure of weight in gold and other precious metals, correctly troy ounces, which weigh 31.1 grams as distinct from an imperial ounce which weigh 28.4 grams.
Outcrop	An exposure of rock or mineral deposit that can be seen on surface, which is, not covered by soil or water.
Oxidation	A chemical reaction caused by exposure to oxygen that results in a change in the chemical.
oz	Ounce.
P	
Plant	A building or group of buildings in which a process or function is carried out; at a mine site it will include warehouses, hoisting equipment, compressors, maintenance shops, offices and the mill or concentrator.
Ppm	parts per million.
Pyrite	A common, pale-bronze or brass-yellow, mineral. Pyrite has a brilliant metallic luster and has been mistaken for gold. Pyrite is the most wide-spread and abundant of the sulphide minerals and occurs in all kinds of rocks.
Q	
Qualified Person	Conforms to that definition under NI 43-101 for an individual: (a) to be an engineer or geoscientist with at least five years' experience in mineral exploration, mine development or operation or mineral project assessment, or any combination of these; (b) to have experience relevant to the subject matter of the mineral project and the technical report; and (c) to be a member in good standing of a professional association that, among other things, is self-regulatory, has been given authority by statute, admits members based on their qualifications and experience, requires compliance with professional standards of competence and ethics and has disciplinary powers to suspend or expel a member.
QA/QCQuality Assurance / Quality Control	Systems and methodologies to ensure samples and data have good confidence of accuracy.
R	
Reclamation	The restoration of a site after mining or exploration activity is completed.
Recovery Rate	A term used in process metallurgy to indicate the proportion of valuable material obtained in the processing of an ore. It is generally stated as a percentage of the material recovered compared to the total material present.
Refining	The final stage of metal production in which impurities are removed from the molten metal.

R

Refractory ore	Ore that resists the action of chemical reagents in the normal treatment processes and which may require pressure leaching or other means to effect the full recovery of the valuable minerals.
Rod mill	A steel cylinder filled with steel rods into which crushed ore is fed. The rod mill is rotated, causing the balls to cascade and grind the ore.
RQD - Rock Quality Designation	A system which ranks rock strength. From the RQD index the rock mass can be classified as follows:

RQD	Rock Mass Quality
<25%	very poor
25-50%	poor
50-75%	fair
75-90%	good
90-100%	excellent

S

Shoot	A concentration of mineral values; that part of a vein or zone carrying values of ore grade.
Skarn	Name for the metamorphic rocks surrounding an igneous intrusive where it comes in contact with a limestone or dolostone formation.
SMU	Is short for selective mining unit - the smallest tonnage of material that can be separated from waste and mined as ore.
Sphalerite	A zinc sulphide mineral; the most common ore mineral of zinc.
Stockpile	Broken ore heaped on surface, pending treatment or shipment.
Strike	The direction, or bearing from true north, of a vein or rock formation measure on a horizontal surface.
Stringer	A narrow vein or irregular filament of a mineral or minerals traversing a rock mass.
Sulphides	A group of minerals which contains sulphur and other metallic element such as copper and zinc. Gold and silver is usually associated with sulphide enrichment in mineral deposits.

T

Tailings	Material rejected from a mill after most of the recoverable valuable minerals have been extracted.
Tailings pond	A low-lying depression used to confine tailings, the prime function of which is to allow enough time for heavy metals to settle out or for cyanide to be destroyed before water is discharged into the local watershed.
Tonne	A metric ton of 1,000 kilograms (2,205 pounds).
t	A metric tonne.
t/h	Tonne per hour.
tph	tonnes per hour.
t/m^3	Tonnes per cubic metre.
t/d	Tonnes per day.
tpd	tonnes per day.
Tunnel	A horizontal underground opening, open to the atmosphere at both ends.

V

Vein	A fissure, fault or crack in a rock filled by minerals that have travelled upwards from some deep source.

W

Wall rocks	Rock units on either side of a mineral deposit (orebody). The hanging wall and footwall rocks of a mineral deposit (orebody).
Waste	Unmineralized, or sometimes mineralized, rock that is not minable at a profit.

Z

Zone	An area of distinct mineralization.