SECURITIES AND EXCHANGE COMMISSION
Washington, D.C. 20549

Form 6-K

Report of Foreign Private Issuer

Pursuant to Rule 13a-16 or 15d-16
of the Securities Exchange Act of 1934

For the month of March 2007

Commission File Number 000-51690

Baja Mining Corp.

(Translation of registrant's name into English)

2350 – 1177 West Hastings Street,
Vancouver, British Columbia T2N 1X7

(Address of principal executive offices)

Indicate by check mark whether the registrant files or will file annual reports under cover Form 20-F or Form 40-F.

Form 20-F ☐ Form 40-F ☑

Indicate by check mark if the registrant is submitting the Form 6-K in paper as permitted by Regulation S-T Rule 101(b)(1): ☐

Note: Regulation S-T Rule 101(b)(1) only permits the submission in paper of a Form 6-K if submitted solely to provide an attached annual report to security holders.

Indicate by check mark if the registrant is submitting the Form 6-K in paper as permitted by Regulation S-T Rule 101(b)(7): ☐

Note: Regulation S-T Rule 101(b)(7) only permits the submission in paper of a Form 6-K if submitted to furnish a report or other document that the registrant foreign private issuer must furnish and make public under the laws of the jurisdiction in which the registrant is incorporated, domiciled or legally organized (the registrant's "home country"), or under the rules of the home country exchange on which the registrant's securities are traded, as long as the report or other document is not a press release, is not required to be and has not been distributed to the registrant's security holders, and, if discussing a material event, has already been the subject of a Form 6-K submission or other Commission filing on EDGAR.

Indicate by check mark whether by furnishing the information contained in this Form, the registrant is also thereby furnishing the information to the Commission pursuant to Rule 12g3-2(b) under the Securities Exchange Act of 1934.

Yes ☐ No ☑

If "Yes" is marked, indicate below the file number assigned to the registrant in connection with Rule 12g3-2(b): 82 -

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SIGNATURES

Pursuant to the requirements of the Securities Exchange Act of 1934, the registrant has duly caused this report to be signed on its behalf by the undersigned, thereunto duly authorized.

Baja Mining Corp.
(Registrant)

Date: March 23, 2007

By: /s/ John Greenslade

John Greenslade
President

BAJA MINING CORP.

AN UPDATED PRELIMINARY ASSESSMENT OF EL BOLEO COPPER COBALT PROJECT

BAJA CALIFORNIA SOUTH, MEXICO

Prepared for

Baja Mining Corp.

by

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31 January 2007

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1 SUMMARY

1.1 GEOLOGY AND MINERAL RESOURCE MODELLING

El Boléo Cu-Co-Zn-Mn deposit is located near the town of Santa Rosalia, Baja California Sur, Mexico.

The deposit, which occurs in the Boléo Formation, comprises seven mineralized units called "Mantos" (manto is a Spanish term used in mining parlance for a generally mineralized layer or stratum). The mineralized mantos dip gently to the east but faulting, which is common throughout the project area, produces a step-like pattern in the present position of the mantos.

The deposit was actively mined from 1886 to 1972, during which time an estimated 18 Mt of ore were treated. Modern exploration activities took place during the years 1993 to 1998 and again in 2004 to 2006, at which time the drill hole assay database used in the current Resource Study was acquired.

Independent geologic consultants, Hellman & Schofield (H&S) of Sydney, Australia were engaged in 2004 to oversee the geologic modelling of the Boléo deposit and to produce a resource estimate in accordance with NI 43-101 standards.

Analytical processes and quality control methods have been studied at length. Matrix matched assay standards were developed for the project and round-robin assay trials used to establish the best analytical method. The assays were deemed accurate and precision values were obtained from original samples versus diamond core duplicates indicating they were representative of the drill hole intervals.

A gridded block model for each Manto was developed for open pit mine design in preference to a gridded seam model because it allows greater versatility to evaluate mining methods at various cutoff grades. Block dimensions used were 50 m EW x 100 m NS and 1 m vertical.

Gridded seam models have been developed for Mantos 1, 2, 3 and 4 for underground mine planning. These models incorporate minimum and maximum mining heights.

A 3D digital geological interpretation was developed from data and information supplied by the company. This incorporated both Manto surfaces and faults.

Statistical analysis of the assay data from each Manto showed that the histograms of Cu, Co, Zn and Mn are not highly skewed, indicating that ordinary kriging is an appropriate estimation method.

To account for the displacement of mantos by the numerous faults, block models and drill hole intersections for each Manto were re-aligned at the same height so the data was effectively re-positioned approximating pre-faulting locations. This removed the need to have a complex series of fault bounded data domains and significantly streamlined the resource estimation process. These block models are referred to as 'flat' models. Block centroid heights in real space, taken from gridded surfaces of the Manto footwalls, were used to translate flat model grade estimates into true 3D blocks.

Resource classification was determined by the number of composite data available for grade estimation, from increasingly localised data search regimes.

For 3D Block and Seam models:

- Measured – search 200 m x 250 m x 2 m
- Indicated – search 400 m x 500 m x 2 m
- Inferred – search 400 m x 500 m x 4 m.

For 3D Block models:

- Measured – minimum data 14 composites from 4 octants
- Indicated – minimum data 14 composites from 4 octants
- Inferred – minimum data 7 composites from 2 octants.

For Seam models:

- Measured – minimum data 4 composites from 4 octants
- Indicated – minimum data 4 composites from 4 octants
- Inferred – minimum data 2 composites from 2 octants.

The total reported Measured and Indicated Resource, based on copper equivalent cutoff grades of 0.5 and 1.0% Cu are as follows:

CuEq. Cutoff Grade		0.5%	1.0%
Measured	Tonnes (10^6)	59.4	53.2
	CuEq.%	2.15	2.30
	Cu%	0.86	0.94
	Co%	0.088	0.091
	Zn%	0.46	0.48
	Mn%	2.77	2.87
Indicated	Tonnes (10^6)	173.4	128.3
	CuEq. %	1.72	2.05
	Cu%	0.76	0.96
	Co%	0.055	0.064
	Zn%	0.54	0.60
	Mn%	2.74	3.06
Total	Tonnes (10^6)	232.8	181.5
	CuEq. %	1.83	2.13
	Cu%	0.79	0.96
	Co%	0.064	0.072
	Zn%	0.52	0.56
	Mn%	2.75	3.00

The additional Inferred Resource is:

CuEq.. Cutoff Grade		0.5%	1.0%
Inferred	Tonnes (10^6)	202.6	114.3
	CuEq. %	1.32	1.76
	Cu%	0.46	0.66
	Co%	0.043	0.055
	Zn%	0.65	0.88
	Mn%	2.67	3.38

A model was created to account for past work in the historic mining areas of Mantos 1 and 3. Tonnes were factored down by 12% to account for material extracted and processed.

Metallurgy and Process Design

A 'proof of concept' pilot plant campaign was conducted at the SGS-Lakefield Testing Laboratories, Lakefield, Ontario, Canada from November 16th to 28th, 2004 treating a bulk oxide sample of underground Boléo ore grading 1.6% Cu, 0.087 % Co, 0.58% Zn, 3.23% Mn and 8.71% Fe. At this time various other bench scale test work was conducted including settling rate determination, slurry viscosity work, tailings characterisation test work (for environmental permitting purposes) and bench scale leach testing.

The major focus of the proof of principle pilot plant was to confirm 1) the clay based Boléo ores could be thickened and washed in a conventional CCD train using high rate thickeners and 2) the CSIRO “DSX” solvent extraction system could be used to recover a cobalt and zinc product.

Since the initial proof of concept pilot plant tests, further work was undertaken to 1) optimize the DSX solvent extraction reagent composition, 2) incorporate and test a manganese recovery process, 3) test the suitability of soda ash for pH adjustment in the circuit and 4) utilize local Boléo carbonate for bulk neutralization.

The integrated demonstration pilot plant campaign to verify these changes was run from June 5th to 24th, 2006 on a bulk oxide sample of underground Boléo ore grading 2.18% Cu, 0.135 % Co, 0.491% Zn, 5.0% Mn and 8.26% Fe.

The zinc solvent extraction and cobalt solvent extraction and electrowinning circuits were run in a separate campaign on accumulated solutions during the period of July 4th to 15th, 2006 thereby allowing the SGS, Bateman and client teams to initially focus on the processes that were either new to the flowsheet or modified since the original proof of concept pilot campaign.

Once again extensive bench scale test work was conducted including:

- settling rate determination
- agitator power estimation in the leach and iron removal circuits
- tailings slurry viscosity and pumping test work

- bulk materials handling test work
- filtration testing of iron residues and manganese carbonate product
- tailings characterisation test work for environmental permitting purposes.

The results of the test work campaign have been employed to generate plant design data for the purposes of advancing the definitive feasibility study for a planned greenfield development consisting of open pit and underground mines, a processing plant and all associated facilities and infrastructure necessary to produce copper and cobalt metal and a zinc sulphate monohydrate salt on site. Manganese carbonate production as an additional product was also considered.

The process plant is being designed to produce and treat 2.6 Mdmt/a of plant feed at an average head grade 2.2% Cu, 0.1% Co, 0.6% Zn and 2.2% Mn through an integrated hydrometallurgical facility to produce the following products and tonnages:

- Up to 60,000 t/a of copper cathode
- Up to 3,100 t/a of cobalt cathode
- Up to 36,000 t/a of zinc sulphate monohydrate salt.

In addition, the potential to produce up to 100,000 t/a of a manganese carbonate product has been reviewed and is discussed in Section 17 of this report.

It is intended to "de-bottleneck" the plant operations at production Year 5 to ensure the product production levels remain at the target values as ROM head grade begins to decrease. Debottlenecking of the production facility will require a modest capital injection at the appropriate time. Capital costs have been allowed by Baja Mining Corp for this eventuality in the financial analysis of all of the base case scenarios.

MINING

The potential for surface and underground mining were assessed for the H&S resource model. A series of underground mining operations, supported by small open-cut surface mines, delivering targeted high-grade copper-cobalt-zinc manganese ore (0.5% to 2.5% Cu) to the process facility was selected as the best alternative for the scale of operation envisaged by Baja Mining Corp.

The seam formation and low material strength of the mantos suggested conventional "soft rock" mining methods such as used in coal, potash, or salt mining would be successful. Common methods were examined and considered for their application to the deposit. Longwall mining was discounted due to the faulted and dipping manto structure and high initial capital cost. Shortwall mining was also discounted when efficient mine layouts could not be readily designed due to the manto structure. Room-and-pillar mining using continuous miners was chosen because of the methods' flexibility for layout designs, its' efficient recovery of resources and lower initial capital cost. The resource seam model was used to define manto areas that could be economically mined. The basic criteria used were:

- Minimum mining height of 1.8 m to allow working room for the machines. If the economic thickness of the manto was less than this it was diluted by the lower grade blocks above up to 1.8m height.
- Maximum mining height of 4.2 m, matching the reach of the continuous miner. Economic blocks above this height were ignored.
- The composite copper equivalent grade of the manto over the mining height must exceed a cutoff grade of 0.5% Cu. For mining purposes, the copper equivalent cutoff grade was calculated based upon base case metal prices.
- An allowance for voids in old works and recovery of “retaque”, or previously mined material, was made in terms of both recovery and density.

Underground mining trials to test equipment, working methods and geotechnical ground responses to the chosen method were undertaken in two stages in the years 2005-2006 under the supervision of consultants, Australian Mine Design & Development (AMDAD), Sydney, Australia, and Agapito Associates, Inc. (AAI), Grand Junction, Colorado, USA. These tests were very successful and confirmed the suitability of continuous mining methods to the Boléo deposit.

Initial mining plans for both underground and surface mining targets in all mantos have been advanced by AAI and AMDAD to provide ore feed at targeted production levels and head grades based on process plant schedules. A limestone source, located on the Boléo property, has also been modelled as the source of calcium carbonate needed for process plant operations.

Preliminary Economic Assessment:

A financial model was created utilizing the current mine production schedule over an initial 20 years, the associated diluted metal grades based on the H&S geological resource and mining schedule, capital and operating costs as set out herein and base case metal prices of copper US$1.25/lb, cobalt US$12.00/lb and zinc sulphate US$950/mt.

In addition, sensitivity analysis was also conducted at various increased metal prices. The project is sensitive to four key variables; copper price; cobalt price; capital costs and operating costs. The sensitivity of the After-Tax IRR and NPV (at 8% discount rate) relative to the Base Case is shown in the table below to indicate the effect of plus or minus 10% changes in the key variables. Note that the changes to the copper price apply to all of the annual prices, starting in 2009, and not just the long term price.

	After Tax IRR				After Tax NPV at 8% ($ Millions)		
	-10%	Base Case	+10%		-10%	Base Case	+10%
Copper Price	16.0%	19.0%	21.7%		$236	$333	$420
Cobalt Price	18.1%	19.0%	19.9%		$298	$333	$367
Capital Cost	21.5%	19.0%	16.6%		$375	$333	$283
Operating Cost	20.7%	19.0%	17.1%		$396	$333	$266

The modelling, based on the current un-optimised preliminary mine schedule, indicates that the project is financially attractive at base-case metal prices. Financial modelling, using the base case prices and only 20 years for the project life, shows that the project could generate a net after- tax Internal Rate of Return (IRR) of 19.0% with a discounted present value, at an 8% discount rate, of US$333.4 million. Using a 6% discount rate generates an NPV, after tax, of $US445.9 million.

2 INTRODUCTION

The Boléo deposit was actively mined from about 1888 to 1972. During that time approximately 18 Mt of ore were mined and processed. More recently, an intensive phase of exploration drilling was carried out between 1993 and 1998, by International Curator Resources Ltd (ICR), focused on defining a large resource amenable to open-cut mining. The project was not developed at that time and in 2001 ownership of the project passed to Minera y Metalurgica del Boléo S.A de C.V. (MMB). Various resource studies and a second intensive phase of exploration and infill drilling between 2004 and 2006 has been completed by MMB with the program focus being to define higher grade resources that could be exploited by underground mining methods.

The current study was commissioned to:

- Produce a digital geological model, including a topographic surface that honours the faults, tops and bases of the seven mantos using a set of interpreted cross sections.
- Produce a digital grade model, for all mantos, for Cu, Co, Zn, and Mn.
- Produce an underground void model.
- Provide a Confidence Classification, taking into account drill spacing, structural and other controls on mineralization, suitable for quotation to NI43-101 standards.
- Review the QA/QC data as relevant to sampling, assaying precision and accuracy.
- Assess the methodology for determining density and the adequacy of the density database, leading to the production of a density model for conversion of volumes to tonnages.

MMB supplied digital drill hole assay and geology data, geological interpretation, drill hole recovery data and rock density data. Information and data regarding assay and sampling quality control was sourced from various reports by consultants G. Peatfield, B. Smee and D. Mehner (Peatfield and Smee 1997, Peatfield 1997, Peatfield 1998, Mehner 2003).

Drilling, sampling and assaying activities providing a significant part of the data used in the current resource study were completed between 1993 to 1998. H&S, whose involvement with the project commenced in August 2004, were, therefore, unable to observe any drilling, sampling or assaying activities related to the drilling carried out during the 1990s. Three infill drill programs were carried out by Minera y Metalurgica del Boléo, S.A. de C.V., the first from December 6, 2004 to January 29 2005 (DDHs 928 to 941), the second program from May 11, 2005 to July 3, 2005 (DDHs 942 to 959), and the third program from February 1, 2006 to September 15, 2006 (DDHs 960 to 1082). These programs were observed and monitored by H&S's representative Dr. B Yeo.

3 RELIANCE ON OTHER EXPERTS

The capital cost for development of the Boléo Project has been developed by a number of specialist organizations. These organizations are listed below in a table that summarises areas of significant capital cost and the organizations responsible for development of capital costs for these respective areas. The Capital Cost Estimate for the project development has been co-ordinated and integrated by Wardrop Engineering on behalf of Bateman Engineering Canada Corp.

Major Cost Area	Consultant	Location
Open Pit Mining	AMDAD Pty Ltd	Sydney, Australia
Underground Mining	Agapito Associates, Inc	Golden, Colorado
Mining Surface Infrastructure	Wardrop Engineering	Vancouver, Canada
Process Plant and General Infrastructure	Wardrop Engineering	Vancouver, Canada
Tailings Dam	Arcadis Geotechnica	Santiago, Chile
Co-Generation Plant	Fransen Engineering Ltd	Vancouver, Canada
Acid Plant	Fenco Pty Ltd	Toronto, Canada
SO_2 Gas Production Facility	Noram Engineering & Constructors Ltd	Vancouver, Canada
Barging Facility	ATI	Vancouver, Canada
Liquid Sulphur Infrastructure	ICEC Canada Ltd	Calgary, Canada
Mexican Construction Labour Rates	UHDE Jacobs	Mexico City

With the exception of AMDAD Pty Ltd, Agapito Associates and Wardrop Engineering the consultants listed above cannot be classified as qualified persons for the purposes of this report.

Bateman Engineering Canada Corp & Wardrop Engineering have relied on the consultants listed above for the generation of capital cost estimates in their particular areas of expertise. Neither Bateman Engineering Canada Corp nor Wardrop Engineering have attempted to formally verify the accuracy or sufficiency of the cost estimates provided by these consultants.

The portion of the report to which the above disclaimer applies is to Section 18.1.3, Capital Cost Estimate.

4 PROPERTY DESCRIPTION AND LOCATION[1]

4.1 LOCATION

The Boléo project is located along the east coast of the Baja peninsula centred on the port town of Santa Rosalia in Baja California Sur, Mexico (Figure 1). The town is approximately 850 km. south of San Diego, California, USA. Coordinates for the project are Latitude 27°14´ to 27°25´ N, Longitude 112°14´ to 112°22´ W.

Figure 1: El Boléo Location Map

[1] Information based on Mehner 2003, reviewed and updated by T. Albinson (MMB) Nov 2004. Maps supplied by MMB.

4.2 DESCRIPTION

The Boléo property consists of 18 total mineral concessions covering 19,519.1872 ha, of which 17 concessions are contiguous. The "San Bruno" concession is not contiguous and is located 30 KM south of Sta. Rosalia in the San Bruno basin area. The titled concessions are listed in Table 1 and shown in Figure 2. One concession is in the process of becoming titled ("San Luciano 5"claim). It should be noted as of January 1st, 2006, all claims in Mexico are "Concesiones Unicas" (Sole Concessions) and the older classification of Exploration claims evolving after 6 years to Exploitation claims is no longer applicable or in use.

Table 1: Boléo Property, Sole Concessions, January 2007

Claim	Title No.	Surface Area (ha)	Date Initiated	Expiry Date	Annual Taxes
El Boléo	218082	4,975.6132	Sept. 29-2000	Sept. 28-2050	285,004
El Boléo I	218092	72.4463	Aug. 31-2000	Aug. 30-2050	4,150
El Boléo19, 256.1872 II fracc. Uno	218179	1,296.6156	Sept. 29-2000	Sept. 28-2050	74,270
El Boléo II fracc. Uno A	218180	507.2841	Sept. 29-2000	Sept. 28-2050	29,058
Boléo III	212148	224.6410	Aug. 31-2000	Aug. 30-2050	12,868
Nuevo San Luciano	214189	150.0000	Aug. 10-2001	Aug. 9-2051	4,272
Boléo II fracc. IV	218975	267.1579	Jan. 28-2003	Jan. 27-2053	7,608
Boléo X fracc. 5	211055	1.3829	Mar. 24-2000	Mar. 23-2050	80
Boléo X fracc. 8	211058	3.9486	Mar. 24-2000	Mar. 23-2050	226
Boléo X fracc. 9	211059	9.9612	Mar. 24-2000	Mar. 23-2050	570
Boléo X fracc 12	211062	3.1241	Mar. 24-2000	Mar. 23-2050	178
Boléo X fracc 16	211066	0.0068	Mar. 24-2000	Mar. 23-2050	2
Biarritz B	219819	0.0055	April 16-2003	April 15-2053	2
San Luciano 2	220740	670.0000	Sept. 30-2003	Sept. 29-2053	9,220
San Luciano 3	221073	1,899.0000	Nov. 19-2003	Nov.18-2053	26,130
San Bruno	222772	8,783.0000	Aug. 27-2004	Aug. 26-2054	120,854
San Luciano 4	223358	392.0000	Dic. 3-2004	Dic.2-2054	5,394
San Luciano 5	E-429	263.0000			
Total (Pesos)		19,519.1872			579,886
Total (US$)					53,635

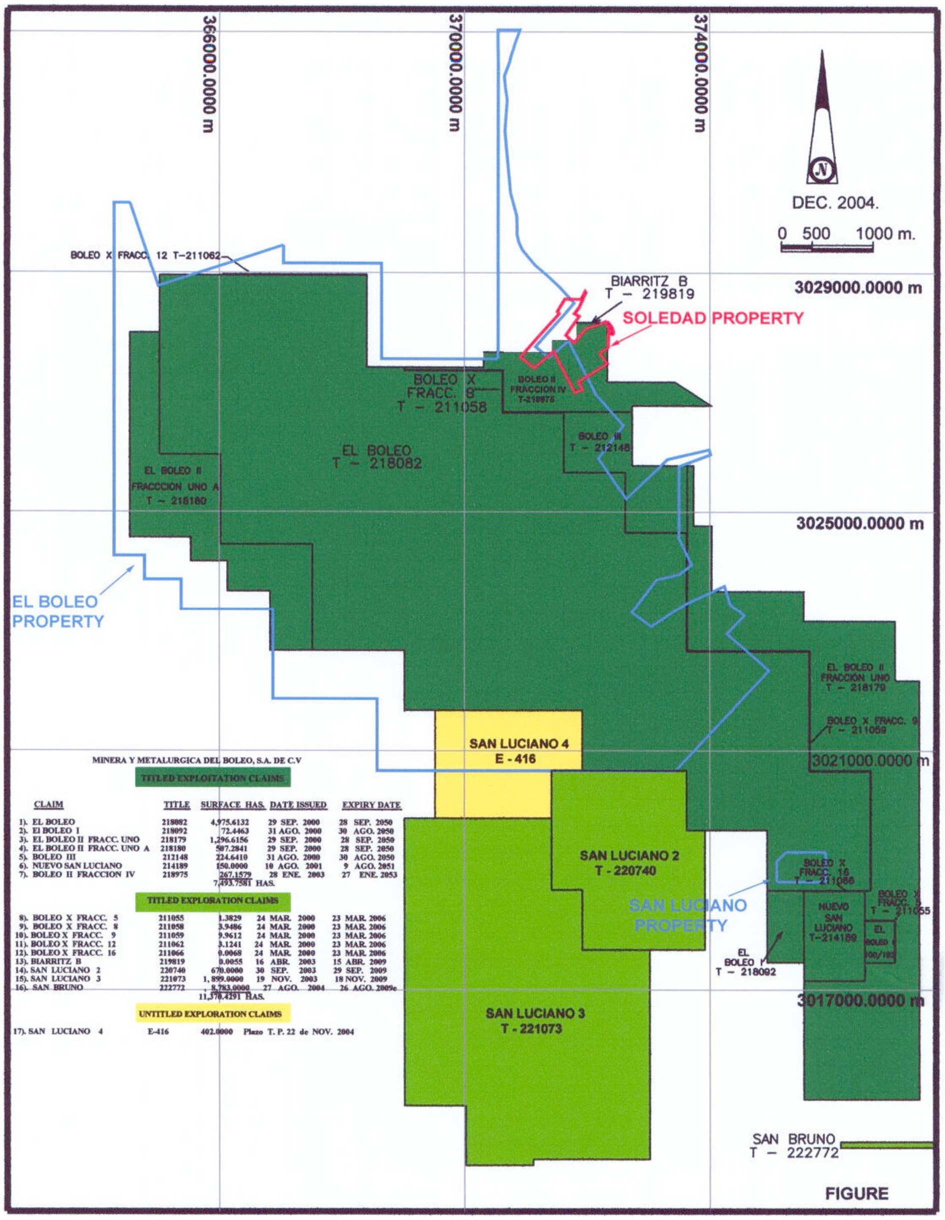
Note: the exchange rate used is Pesos 10.8116 =US $1

The project also includes three surface lots totalling 6,692.58 ha as shown in Table 2 and Figure 2.

Table 2: Boléo Project, Surface Property and Annual Taxes

Surface Lot	Size (ha)	Annual Taxes
El Boléo	6,553.55	21,604
Soledad Property	99.91	23,786
San Luciano Property	39.12	5,301
Total (Pesos)	6692.58	50,691
Total (US$)		4,688

Figure 2: El Boléo Property Map



4.3 OWNERSHIP

The mineral concessions covering El Boléo copper-cobalt-zinc manganese deposit are 100% owned by Minera y Metalurgica del Boléo S.A. de C.V. (MMB), a Mexican company involved in mineral exploration and development and a wholly owned subsidiary of Baja Mining Corp, who recently listed on the Toronto Stock Exchange (TSX).

4.4 TAXES AND ASSESSMENT WORK REQUIREMENTS

4.4.1 TAXES

Total annual fees payable in January 2007 as of this report are 579,886 pesos for mineral concessions and 50,679 pesos for surface leases, or using the exchange rate as of January 02, 2007 (10.8116 pesos/US$), US$58,323 (see Tables 1 and 2). The calculated annual fees are based on the latest published government tax guides.

4.4.2 WORK REQUIREMENTS

Work obligations on the property (known in Mexico as "Informes de Conprobaciones de Obras") are in good standing. Based on past work expenditures of approximately US$22 million, enough credits have been accrued to keep the property in good standing until 2013.

4.4.3 OPTION PAYMENTS

There are no royalties payables on the properties and there are no other agreements or encumbrances.

4.5 PERMITS AND LIABILITIES

4.5.1 PERMITS

During 2006, MMB successfully completed a full Environmental Impact Assessment that covers the construction, operation and closure phases of the Boléo project. Given the complexities of the project itself and the environmental sensitivity surrounding the project location, the Mexican Federal environmental agency, Secretaria de Medio Ambiente y Recursos Naturales, (SEMARNAT) requested the submittal of an Environmental Impact Manifest with a regional scope. The change of scope required additional field-work to fully characterize the regional area of influence of the project. This, in turn, caused a delay in the EIM submission date from February to May 2006. The evaluation process also included a request from SEMARNAT to submit additional information relating to the project to better clarify the identified environmental impacts. This request was given to MMB on July 5th, 2006. The information was formally filed on October 2nd, 2006.

Finally, after incorporating the observations and recommendations from the National Commission for Natural Protected Areas; the Secretariat for Urban Planning, Infrastructure and Ecology of the State Government of Baja California Sur and the Municipal Presidency of Mulegé at Baja California Sur, the environmental impact resolution was issued on November 27th, 2006 and delivered to MMB on December 7th, 2006. This resolution authorizes the construction, operation and closure of El Boléo Mining Project. The official document number containing this resolution is S.G.P.A.-DGIRA.-DDT.-2395.06 and is signed by the General Director for Environmental Impact and Risk (DGIRA).

This authorization allows MMB to initiate the procedures to obtain more specific permits. In 2007, MMB will concentrate its efforts in securing these additional permits and in managing the terms and conditions that were established in the environmental impact authorization.

4.5.2 Liabilities

There are no outstanding liabilities associated with the property. The most recent disturbances were caused by the drilling and metallurgical sampling programs of International Curator in 1997-98 and MMB in 2004-2006, including the development and operation of the underground test mine. All of the disturbed areas from 1997-98 were remediated without any assessed environmental liabilities from that period. The current MMB drilling program work is in the process of being remediated and no liabilities have been recorded at this time and no liabilities are expected to be incurred from this work going forward. The test mine work remains and has been incorporated into the EIM permit as an identified disturbed area allowed under the permit.

The project is located within the “buffer zone” of the Vizcaino Biosphere Reserve which is centred on the Desierto de Vizcaino on the west central coast of the Baja. The Biosphere extends south to encompass the historic Boléo Mining District and to protect certain environmental and cultural features in the town of Santa Rosalia. It is believed the Biosphere intended to protect the historic buildings dating from the late 1800s associated with the early mining of the Boléo district. It should be noted these buildings are outside the Boléo project and study area boundary and will not be directly affected by project development.

Since the Boléo district has been mined for copper and cobalt since 1865, and with two large gypsum quarries currently operating in the region, the authorities have designated the local land suitable for mining and have established land management directives within the Biosphere for development. The area within the reserve is therefore managed relative to a specific land usage description.

There are no tailings ponds on the lands owned by MMB or on the referenced concessions. There are 88 small mine waste dumps[2] located at the portals of historic mine workings.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY[2]

5.1 ACCESS

El Boléo property is located on tidewater on the east coast of the State of Baja California Sur, Mexico, adjacent to the town of Santa Rosalia. Current access for construction equipment would principally be via the Trans-peninsular highway, a trip roughly 850 km south of the US border. This highway passes directly in front of the plant site location and through Santa Rosalia. The highway carries heavy traffic volumes year round. Heavy construction equipment and project supplies would also be brought in by barge to the port of Santa Rosalia and the Pacific Coast marine facilities at Guerro Negro. There are regularly scheduled air services from both the United States and mainland Mexico to the towns of Loreto, which is a two hour drive away to the south, and La Paz which is a six hour drive to the south. The closest private airstrip is at Palo Verde a half hour drive south. Port facilities, which originally serviced the copper processing plant and mines until 1985, are still being used on a regular schedule by a ferry service to the mainland at Guaymas.

5.2 CLIMATE

The project area is immediately adjacent to the Gulf of California, with a climate typical of the Sonoran desert region with warm to hot temperatures and minimal seasonal precipitation. Rainfall is confined mainly to heavy cloud bursts at intervals of several years during tropical cyclones. Mining operations can be scheduled 365 d/a heavy rainfall events not withstanding.

5.3 LOCAL RESOURCES

According to the most recent Mexican census, nearly 40,000 people live in Mulegé County, which is over 10% of the Baja California Sur population. Population concentrations within the county include Santa Rosalia, Santa Agueda, Mulegé (town), Guerrero Negro, San Ignacio, Bahía Tortugas, San Marcos Island, Gustavo Diaz Ordaz and Bahía Asunción. Santa Rosalía is the largest town in the county. Santa Rosalia has a population of 10,000 people and services a fishing fleet, fish processing facilities and two open pit gypsum mines. Education levels are reasonably high but unemployment is also high. An informal socio-economic study prepared in 1997 suggested almost all of the non-staff positions might be filled from the local population.

Hotel accommodation, gasoline, groceries and various hardware goods can all be purchased in Santa Rosalia. Other items including machinery and trained personnel are readily available from

[2] Information derived from Mehner, 2003

mainland Mexico via the ferry or airplane. Services and supplies are also obtainable from California, USA.

Fresh water for domestic and drilling purposes is scarce and the town currently obtains most of its supply from wells in Palo Verde, 30 km away. The planned process plant and mines are expected to use seawater for 100% of its requirements. A desalination plant with a capacity of 200 m^3/h will be required to supply process and potable water for use at the plant and mines, including sufficient fresh water for the final stages of metal production.

5.4 INFRASTRUCTURE

Aside from the many kilometres of drill roads built along each arroyo to access the property, the only other infrastructure on the plant site property is a warehouse and fenced yard. This was constructed in 2002 by MMB as a site improvement and will be used as the site office and base of operations for ongoing development.

A test underground mine and portal site was constructed in 2005 near the Texcoco area of the Boléo arroyo and is currently idle. The site was used to conduct underground mining tests for geotechnical information and mine equipment evaluations. The site includes:

- an over-the-road container trailer modified to house a diesel electrical generator, caplamp light rack, mining supplies and some tools
- several lean-to sheds for roof bolt supply storage and temporary office
- a fuel tank (w/containment)
- a small mine fan
- steel tunnel cowlings
- several pieces of underground mine equipment, including a continuous miner, electrical power center, diesel LHD, two diesel mine trucks, and a portable hydraulic roof drilling machine.

At present, there are no plans to continue mining via these portals.

5.5 PHYSIOGRAPHY

Property topography is best described as mesa-arroyo with relatively flat plateaus cut by deeply incised arroyo valleys resulting in rugged, steep sided valleys with arroyos that drain into the Gulf of California. Project site elevations vary from 50 masl to 350 masl.

The project site is very arid with vegetation consisting of a wide variety of cactus. Over most of the project area vegetation is quite sparse and only locally along the mesa tops, a few kilometres in from the coast, does it occur in significant amounts.

6 HISTORY

The discovery of copper in the Boléo district is attributed to local rancher, Jose Rosa Villavicencio who found copper nodules, “boleos” in 1868 while traversing down an arroyo not far from present day Santa Rosalia (Wilson and Rocha, 1955). The property was sold to two individuals from Guaymas, Sonora who in 1872 began mining and hand sorting high-grade oxidized copper ores from trenches and open cuts and shipping them to smelters in Europe and Guaymas.

Lower grade material was left on dumps or used as backfill in the stopes. This continued until 1884 when declining copper made operations difficult and the firm failed. Until then production was estimated to have been 60,000 short tons grading 24% Cu (Wilson and Rocha, 1955). A further 120,000 tons averaging about 8% Cu is estimated to have been deposited on dumps or used as backfill in stopes.

In 1884, a number of French geologists and mining engineers including Messrs. Eduoard Cumenge and G. de la Bouglise visited Boléo and after recognizing the vast potential recommended a significant investment to develop the district. On May 16, 1885 the Compagnie du Boléo (later to be known in Mexico as the Compania del Boléo, S.A. - “the Boléo Company”) was formed in Paris, backed mainly by the banking interests of the French House of Rothschild.

On July 7, 1885, the Boléo Company acquired, from the Mexican government, all mining claims in the region and a concession of about 20,655 ha. Operations began in 1885 and early work involved a systematic organization of mining and construction of a smelter, port facility, town site and other infrastructure.

Production started in 1886 and by 1894 had reached over 10,000 annual tons of copper contained in copper matte and “black copper”, which were transported to Europe for treatment. In 1922, a new smelter was built to produce blister copper, which was shipped to Tacoma, Washington, for refining. The Compagnie du Boléo was active from 1885 to 1938, when it went into liquidation. It continued operations on a small scale until 1948, when it was reorganized as the Boléo Estudios e Inversiones Mineras, S.A. From 1938 on, much of the smelter feed was supplied by small groups of independent miners called “poquiteros”, who re-worked backfilled stopes, robbed pillars and worked smaller, lower grade mines. Their work is poorly documented. Smelting operations were initially suspended in 1954. Nearly all of the ore mined in this period was sourced from the numerous small underground mines throughout the district.

In 1954, operations were taken over by the Compañia Minera Santa Rosalia, S.A., jointly owned by Federal and State Governments and private Mexican interests and managed by the Comisión de Fomento Minero (Fomento Minero is the Mexican Bureau of Mines). Fomento Minera attempted to sustain copper production by re-opening the smelter and building a leach-precipitation-flotation (LPF) plant to treat dump material, including small amounts of underground ore still being produced by the poquiteros, all to produce a concentrate for the smelter. Recoveries in the LPF plant are reported to have been about 60% in the early years but diminished with time as the plant deteriorated. The smelter continued operation, treating material produced by poquiteros and concentrates from off-shore, until final closure in 1985.

During the latter years of operation at Boléo, there was some exploration in the form of diamond and churn drilling by both French and Mexican concerns. Shafts were also sunk to intercept the high-grade mineralization. This work was concentrated in a few relatively restricted areas of the district since the smelting operations needed a cutoff grade of > 4.5%. This exploration work showed that the required grades lay near the southeast corner of the present property and at depths deeper than 200 m. It should be noted that these early operators assayed only for copper and only portions of the mineralized units were sampled. The results were thus of little importance in the overall context of a modern exploration program.

During the 1960s and early 1970s, the Compañia Minera Santa Rosalia S.A, in an effort to find more reserves for the LPF plant, commenced an underground program in which it blocked out a measured resource of backfill material in the Apollo mine area reported to be in the order of 660,000 tonnes grading about 1.60% Cu, with an unknown cobalt and zinc content. This material was never mined due to lack of funding from the government.

Table 3: Historical Mining Activities at Boléo

Period of Activity	Metric Tonnes Mined	Ave. Copper Grade	Tonnes Copper Produced
To 1884	~54,400	24.0%	~10,400
1888 – 1947	13,622,327	4.81%	540,3342
1948 – 1952	817,300	3.95%	~27,000
1953 – 1972	1,118,200	3.95%	~36,500
1973 – 1985	720,900	3.02%	~18,000
1964 – 1972	2,500,000	1.40%	n/a

After cessation of operations at Boléo in the 1980s, the bulk of the district was held in the Mexican Strategic National Mining Reserve until 1991. Some months after the release of the ground from the reserve, much of the district was acquired by Minera Terra Gaia, S.A. de C.V., a wholly-owned subsidiary of Terratech Environmental Corporation (Terratech), Barbados, which subsequently optioned the concessions to International Curator Resources Ltd. and its' subsidiary, Mintec International Ltd., a Barbados company, now Mintec Processing Ltd. (Mintec) as a result of a continuation of Mintec International to British Columbia, Canada in 1993.

Over the period October 1993 to March 1997, Curator completed 68,685 m of HQ coring in 828 holes. In addition, there were 28 holes either re-drilled or twinned, and 58 large diameter holes drilled to recover metallurgical test samples. This was supplemented with 108 hand or excavator dug trenches put in to expose mineralized mantos for both assay data as well as to better define erosional limits of various mantos. Ten larger trenches were dug, using bulldozers to expose the base of Mantos 2 and 3 and to provide sites for bulk sampling. Only six of these trenches were successful in exposing the desired contact and, of these, 5 contained Manto 3.

By the end of 1997 a pre-feasibility study incorporating all work completed since 1993 was prepared by Fluor Daniel Wright and presented to the underlying owners.

In summary, Fluor Daniel Wright estimated proven reserves at Boléo to be 71.2 M tonnes grading 1.44% Cu, 0.092% Co and 0.55% Zn with further probable reserves of 13.1 M tonnes grading 1.57% Cu, 0.065% Co and 0.81% Zn. These were deemed sufficient to support an 11,500 t/d operation for about 17 years with an estimated capital cost of about US$440.5 million. The operation was envisioned to be an open-pit mine with on-site processing utilizing a hydrometallurgical plant producing copper, zinc and cobalt cathode with an option to produce a cobalt sulphide product instead of cathode. Metal recovery would involve acid leaching with copper, cobalt, and zinc recovered from the leach slurry using a "novel", in pulp method of recovery.

In 2001, following a significant down-turn in metal prices, International Curator withdrew from the project by handing back its interest in Mintec and the Boléo concessions to Terratech.

After re-gaining control of Boléo, Mintec engaged Bateman Engineering Pty. Ltd. of Australia to assess the pre-feasibility work conducted by Fluor Daniel back in 1997 and determine if significant improvements in mining, processing and capital costs could be achieved relative to the costs presented by Fluor Daniel.

Most of Bateman's work concentrated on the metal recovery part of the proposed flow sheet where they proposed using a conventional counter current decantation (CCD) solid-liquid separation circuit followed by base metal recovery from the CCD wash solution. The Bateman flow sheet also involved acid leaching of the copper, zinc, cobalt, and manganese followed by rejection of the leach residue and separate recovery of copper, zinc sulphate, cobalt either as a metal product or a high value cobalt intermediate precipitate and manganese carbonate.

As a result of the Bateman work and the belief that the new process would greatly improve the economics of the project, Mintec embarked on a corporate re-organization. In April, 2002, Mintec International Corp. acquired all of the rights to the copper/cobalt/zinc/manganese concessions, as described above and registered them with Mintec's wholly owned Mexican subsidiary Minera y Metalurgica del Boléo S.A. de C.V.

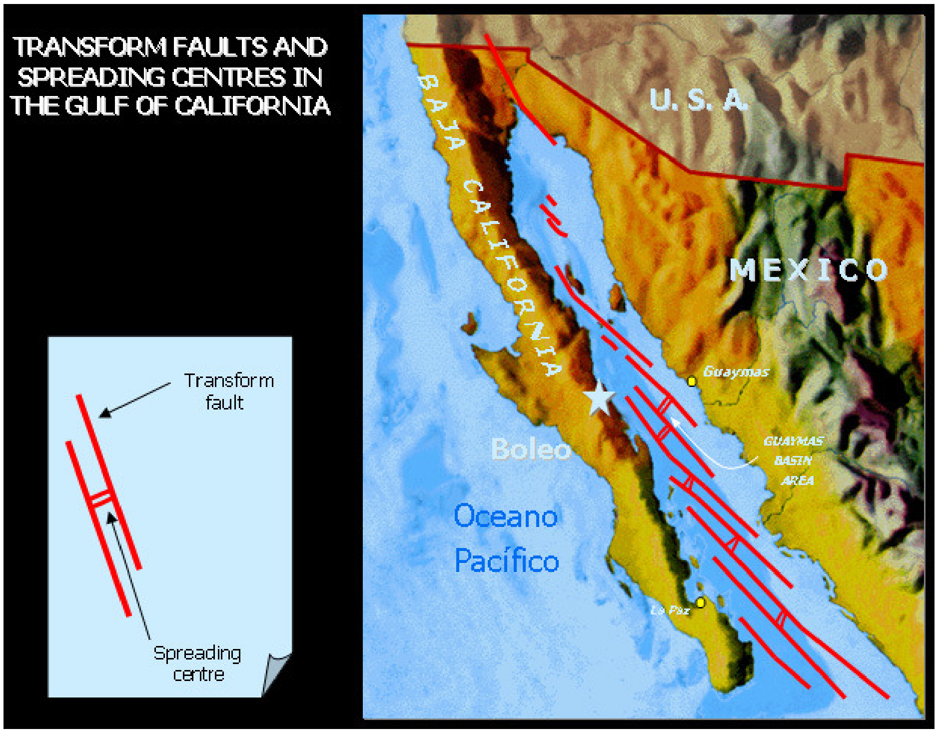
In November 2003, the shareholders of Mintec entered into an agreement with First Goldwater Resources Inc., a TSX-Venture Exchange listed public company, to exchange all the issued shares of Mintec for shares in First Goldwater (now Baja Mining Corp) and effectively purchased the Boléo copper-cobalt-zinc asset by way of a reverse take-over. The TSX-V Exchange approved the transaction and, after an initial Cdn$10 million capital raising in April 2004 the name First Goldwater Resources Inc. was changed to Baja Mining Corp (Baja), trading on the TSX-V under the symbol BAJ until February 2007 when the shares of Baja Mining Corp were listed for trading on the TSX main board under the symbol BAJ-T.

7 GEOLOGICAL SETTING[3]

7.1 REGIONAL GEOLOGY

The Boléo deposits occur within the Boléo sub-basin of the Santa Rosalia basin. This basin formed as a result of Miocene rifting in the Gulf of California extensional province (Figure 3). The northward extension of this province is the Basin and Range province of the southwest United States.

Figure 3: El Boléo Geological Setting



The timing of initial rifting varies from 13 Ma to 8 Ma. In the Boléo District, which is located near the western edge of the Gulf extensional province, rifting is believed to have started some time after 10 Ma (Sawlan and Smith, 1984).

The early rifting direction was east-northeast and produced north-northwest oriented basins and ranges in basement Miocene volcanic rocks flanking the rift axis. The latest movement, occurring after the Gulf transform and San Andreas wrench-fault systems were initiated, has been right-lateral oblique movement (Stock and Hodges, 1989). This has moved Baja California approximately 350 km. northwest relative to mainland Mexico and has created a number of deep pull-apart basins along the axis of the Gulf of California (Bailes, et al., 2001).

[3] Information from Mehner, 2003

Stratigraphically the Boléo copper-cobalt-zinc manganese deposits occur within the late Miocene age succession of fine to coarse clastic sedimentary rocks of the Boléo Formation, lying unconformably on andesitic rocks of early to middle Miocene age called the Comondú Volcanics. The Boléo Formation is characterized by a number of coarsening upward cycles of sediments that are believed to represent deltaic deposition in a shallow, near-shore marine basin. The upper part of the formation has been locally eroded and unconformably overlain by similar but barren and fossil-rich sedimentary successions of Pliocene and Pleistocene age delta and beach deposits, known as the Gloria, Infierno and Santa Rosalía Formations.

The Boléo and overlying formations collectively make up the so-called Boléo Basin. Locally, the entire succession is capped by Pleistocene to recent flows and pyroclastic rocks of the Tres Virgenes Volcanics. The geology of the district has been described in detail by Wilson and Veytia (1949) and by Wilson and Rocha (1955), in privately prepared reports for International Curator by Peatfield (1995) and Christoffersen (1997) and in numerous other published and unpublished papers and reports referred to in the above mentioned documentation.

7.2 Property Geology

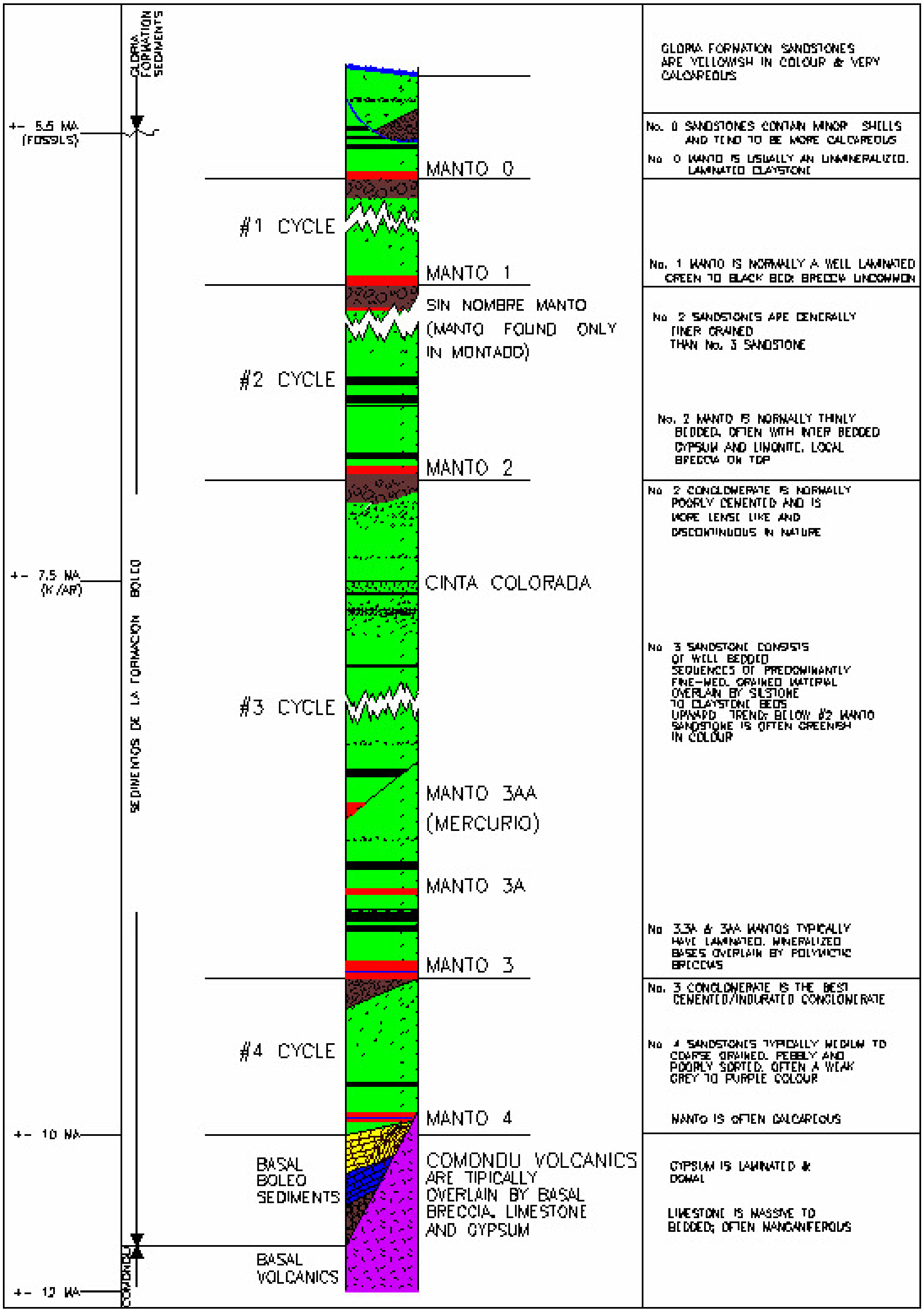
The oldest rocks outcropping on the property are andesitic volcanics of the Comondú Formation. They include sub-aerially erupted flows and coarse explosive breccias which grade into coeval epiclastic sediments to the west. The volcanics have been dated from 24 to 11 Ma. They are underlain by Cretaceous granodiorite (Schmidt, 1975).

The overlying Boléo Formation consists of five coarsening upward cycles of sedimentation numbered “4” at the base and “0” at the top (Figure 4). This interpretation is based on work by Curator from 1993 to 1997 and is different from that published by Wilson and Rocha (1955) who interpreted conglomerates, the coarsest units in the stratigraphy, to be the basal unit in each cycle.

The basal unit in the Boléo Formation is a 1 to 5 meter thick limestone unit. It contains cherty lenses and non-diagnostic fossil fragments. Its occurrence atop very steep paleo surfaces, combined with banding parallel to its base and the cherty horizons, suggests it is at least in part a chemical sediment.

Overlying the limestone or laying directly on Comondú over parts of the district, particularly over much of the coastal area, is an extensive gypsum deposit up to 80 m thick. Although a few dome or mound structures have been noted, the gypsum unit is characteristically flat to shallow dipping exhibiting laminated to massive and even brecciated textures. Intraformational carbonate beds are rare.

Figure 4: Boléo Formation Stratigraphic Column



On top of the gypsum/limestone beds is the cyclic succession of clastic beds that average 150 m and range to 270 m thick. Individual cycles range from 20 to 140 m thick and consist of a basal mud and fine volcanic ash horizon (now altered to montmorillonite clay) that hosts the copper-cobalt-zinc manganese mineralization (the manto).

These are overlain by progressively coarser material of maroon coloured, tuffaceous claystone, siltstone, feldspathic sandstone, pebbly sandstone and eventually cobble to boulder orthoconglomerates.

Typically the earliest cycles (manto 4, then 3) are thickest with each successive cycle being thinner. The last cycle is thin and believed to be incomplete. All cycles thin over basement highs and wedge out toward the basin margins. The copper-cobalt-zinc manganese stratiform deposits only occur within Boléo formation rocks.

Unconformably overlying the Boléo clastics are fossiliferous marine sandstones and conglomerates of the lower Pliocene (about 5.3 Ma.). Gloria formation (Bailes, et al., 2001). These in turn are overlain by slight unconformity with a sequence of fossiliferous marine sandstones and conglomerates of the Infierno formation. Unconformably overlying these are fossiliferous sandstone and conglomerate of the Pleistocene, Santa Rosalia formation (Wilson and Rocha, 1955).

7.3 STRUCTURAL GEOLOGY

The Boléo Formation rests on an irregular volcanic basement, with several distinct basement highs and intervening troughs. In places, these basement highs are so pronounced that they have influenced the deposition of the lower mantos, such that these pinch out against the volcanics and only the upper mantos are present. There is also a tendency for the sediments of each cycle to thin towards the high ground, giving a stratigraphic compression and thus less vertical separation of mantos.

Faulting is common throughout the district. The dominant faults are northwest to north-northwest striking and steeply dipping with normal movements. These faults have downthrows to both east and west, with more of the major faults down dropping to the west. This, coupled with the generally easterly dip of the mantos, yields a stepwise pattern of the present position of the mineralized beds (Figure 5). Many of the faults appear to be long-lived, probably with their first movements influencing the initial basin formation and with continuing movements throughout time to the present day. Vertical displacements can be as much as 50 m to 200 m maximum on the major faults, with much lesser movements toward the ends of these faults and on lesser structures throughout the district. Fault displacements will obviously be important in detailed mine planning; fortunately, in much of the district, the faults and their displacements are well documented in old mining records.

Major faults at Boléo are, in most cases, laterally separated by several hundred, to in some cases, over a thousand m. Lesser faults are common and more closely spaced. Faults displace mantos and as a consequence of their dip, may form “fault windows” in which the mantos are not present. However an order of magnitude calculation suggests that the windows may represent less than 2% of the total area.

Many of the major faults have zones a few metres to tens of metres wide in which the rocks, including the mineralized mantos, are highly disrupted.

There has also been some oblique strike-slip movement on many of the faults. The sense of this movement appears to be predominantly right lateral which would be expected given the spreading regime in the Gulf of California.

8 DEPOSIT TYPES[4]

The Boléo District hosts a number of mineral deposit types that have the potential to be of sufficient size and grade to be economically mined and processed. The most important of these and the subject of this report are the manto hosted copper-cobalt-zinc manganese deposits which occur in Boléo formation clastic sediments.

Another possible targets are the extensive gypsum beds which occur over portions of the property, particularly north and east of Arroyo Saturno. It is believed they occur along the same stratigraphic position as those currently being mined immediately north of Boléo and 20 km southeast on San Marcos Island.

[4] Data from Mehner, 2003

9 MINERALIZATION[5]

Deposits of copper-cobalt-zinc manganese mineralization in the Boléo District occur within widespread, stratiform clay-rich horizons or beds known as “mantos” (manto is a Spanish term used in mining parlance for a generally mineralized layer or stratum). Within Boléo formation stratigraphy there are up to seven mantos, including two of very limited extent, that occur as relatively flat to generally shallow dipping, stratabound and stratiform beds. These include, with increasing depth, manto 0, 1, 2, 3AA, 3A, 3 and 4. Historically the major producing manto has been number 3, which yielded approximately 83% of all production between 1886 and 1985 when the plant shut down. Most of the remaining production has come from manto 1 in the southeast portion of the Boléo area where manto 3 is absent. A small amount of production has come from the widespread but generally thin manto 2 while an even smaller level of production has come from the relatively restricted manto 3A. Based on previous studies and exploration work, mantos 1, 2 and 3 still offer the most potential for hosting significant economic reserves.

The mantos themselves tend to be clay rich (ash altered to montmorillonite) with laminated basal zones generally less than 1 m thick overlain by intrabasin slump breccias up to 20 m thick. Underlying lithologies vary from predominantly ortho-conglomerates in the heart of the Boléo basin to coarse sandstones typically containing pebbles of Comondú volcanics. The contact between the mantos and footwall rocks is sharp.

Overlying lithologies vary from fine to medium grained sandstones. The contact between them and the clay rich slump breccias is gradational.

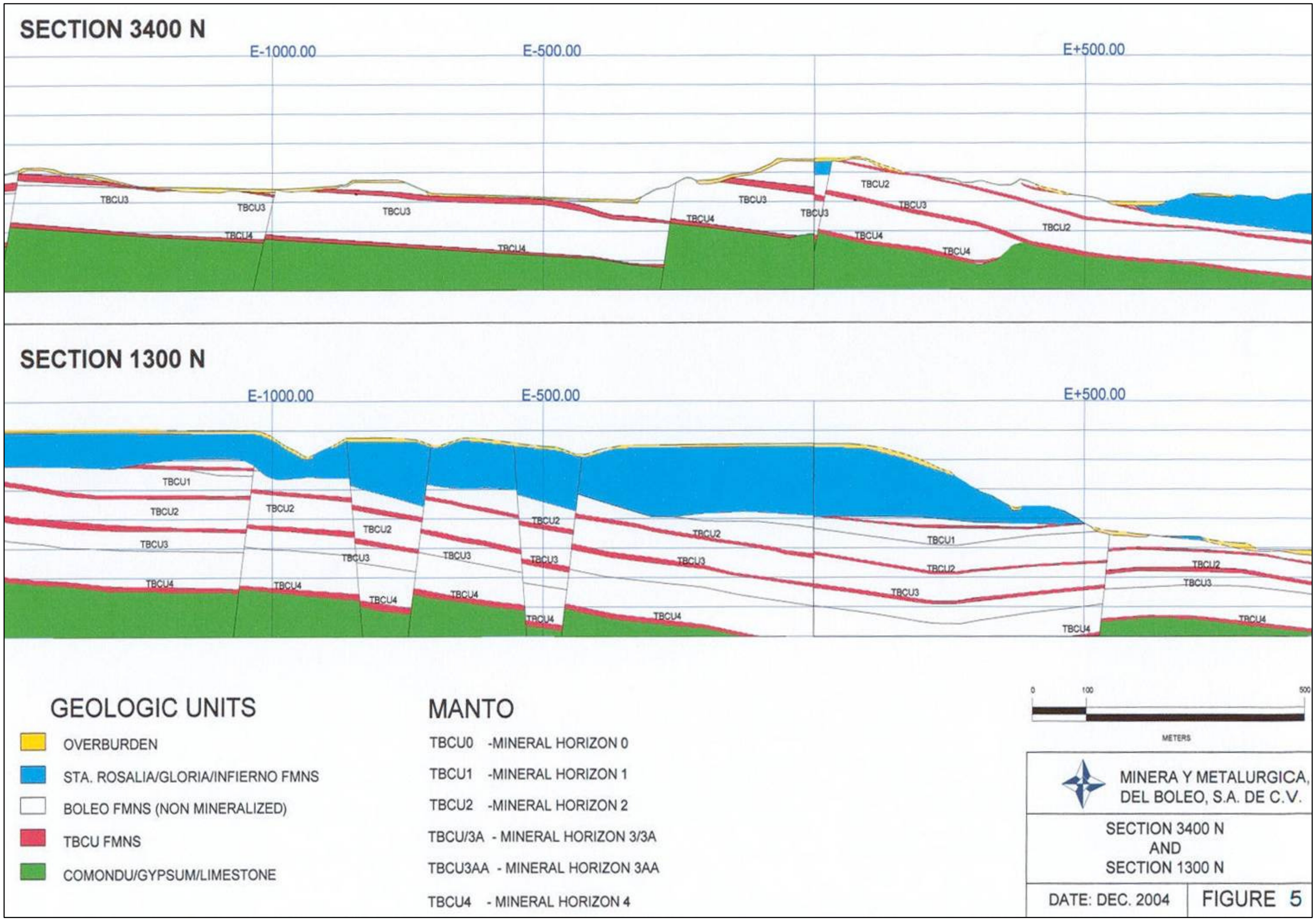
In a general sense each manto has distinctive characteristics, especially with regard to copper-cobalt ratios and relative concentrations of zinc, manganese and carbonates.

Metals of interest in the mantos include copper, cobalt, zinc and manganese. Ore minerals include a fine grained, complex assemblage of primary sulphides including pyrite, chalcocite, chalcopyrite, bornite, carrolite, sphalerite and secondary minerals including malachite, azurite and the rare minerals of boleite, pseudoboleite and cumengite. Mineralization is generally finely disseminated over intervals up to 20 m thick in the slump breccias. The richest material typically occurs in the laminated basal section of the Manto, which was historically mined from 1886 to 1972 to an average of about 80 cm and graded 4% to 5% Cu.

Recently, the presence of a thin layer of claystone within a manto creating a “false bottom” over large areas of mantos 1 and 3 has been recognized. Within these areas, the rich material is often in the higher portions of the mantos above the false bottom layer. For a detailed discussion on the concept of the "false bottom" please refer to Section 9.1 of this report.

[5] Data from Mehner, 2003

Figure 5: Geological Cross-Sections



Zoning of the principal economic metals occurs both vertically and laterally. Within individual mantos, copper is enriched at the base, zinc towards the top and cobalt is more or less evenly distributed. Stratigraphically, vertical zoning shows a trend of zinc enrichment from the lowest manto (4) to the uppermost mantos. Lateral variations indicate the central core of the Boléo sub-basin is copper rich flanked by a zinc rich marginal zone. Cobalt is variable and shows no clear correlation with copper or zinc.

Individual mantos and their enclosing strata are "time transgressive", in that they are progressively younger toward the present Gulf of California. One very distinctive unit, the "Cinta Colorada" or "red ribbon" is a layer of reddish andesitic-basaltic tuff up to two m thick. This is interpreted as the product of a single explosive volcanic event, which probably blanketed the entire region. The Cinta Colorada represents a true "time horizon", and can be seen to transgress stratigraphy, in some places lying within the unit 2 conglomerate (below Manto 2) and elsewhere in the underlying unit 3 clastic succession. Thus it demonstrates the time transgressive nature of the enclosing stratigraphic units.

Individual mantos have great lateral continuity and relatively consistent thicknesses. In the principal areas of interest, the lowest Manto (4) lies at the base of the Boléo Formation, directly on the Comondú Formation. Manto 3 is widespread and thick, and accounting for the largest proportion of the mineral resource. Mantos 3A and 3AA are less continuous and thinner, lying higher in the succession. In some places, especially in the Saturno-Jalisco area, 3A merges with 3. Manto 2 is stratigraphically very continuous but because of its higher stratigraphic position, it is more commonly eroded.

Manto 1 makes up the bulk of the mineral resource in the southeast portion of the Boléo property, where the lower mantos (3 and 4) were for the most part not deposited. The beds dip to the southeast and as a consequence, Manto 1 lies deeply buried in this area. To the northwest, Manto 1 overlies the well mineralized portion of Manto 3 (and in many places, 3A, 3AA and 2).

9.1 BOLÉO DISTRICT FALSE BOTTOMS

Although in the greater part of the district the mineralized mantos sit directly and in sharp contact on conglomerate footwall, there is also a large portion of the district where the best mineralized horizon is detached from the conglomerate footwalls, or exhibits no conglomerate in the footwall. The "old timers" named these footwalls "false bottoms" which were usually described as sandstones (Wilson, 1955). The extensive infill drill program carried out by Baja Mining during 2005 and 2006, in conjunction with the drill programs conducted by International Curator Resources during the 1990s have clarified that false bottoms consist not only of sandstone, but of limestones, claystones and polymictic breccias. For the purpose of resource estimation, the presence of a false bottom is only defined if two or more adjacent diamond drill holes exhibit similar and correlate-able footwall stratigraphy.

The first new significant false bottom was identified in Manto 1 on the eastern basin-side of the manto where the footwall rocks are comprised mostly of finely stratified light coloured limestones (see Figures 6 and 7).

Figure 6: Manto 1 Footwall Geology

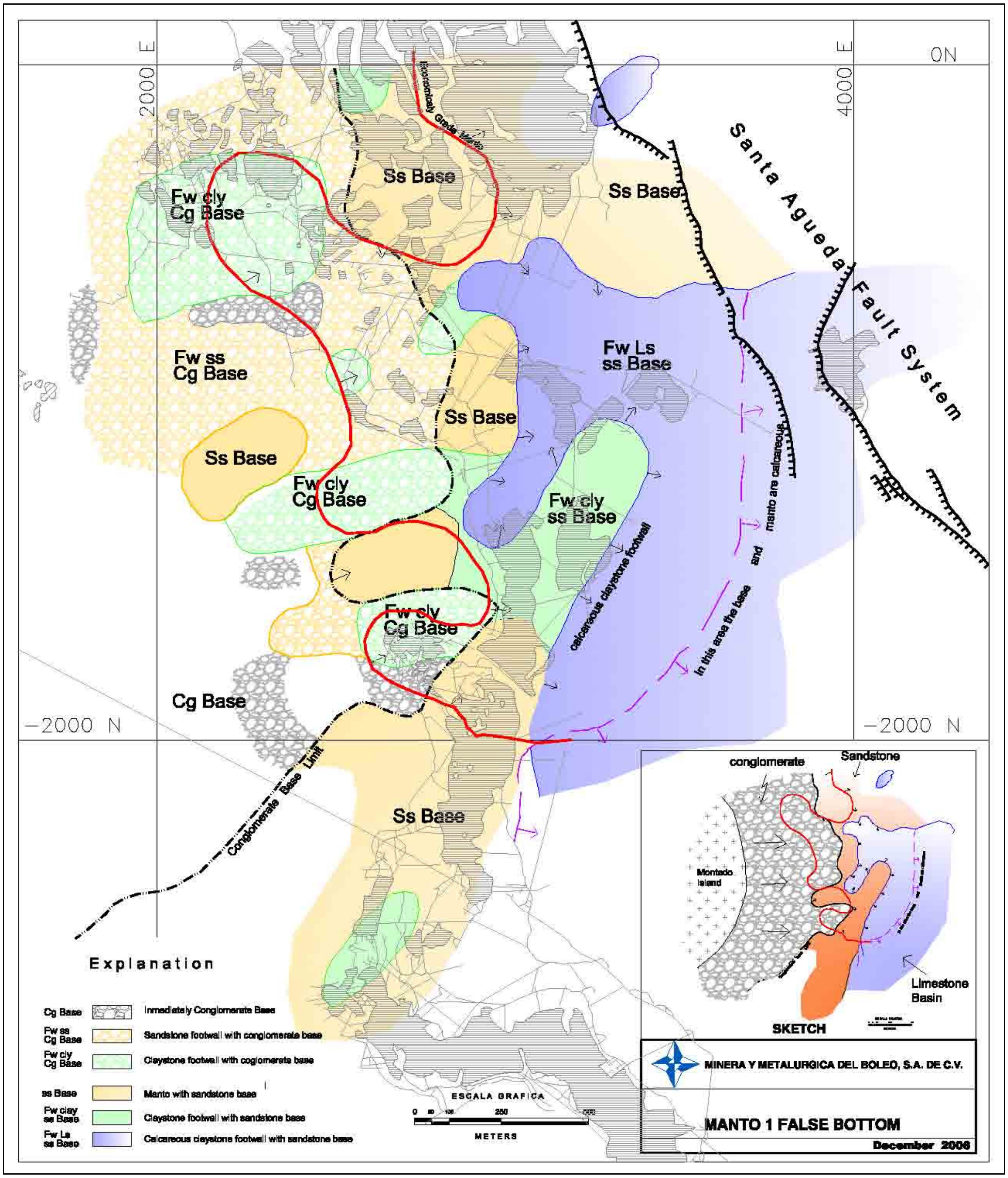


Figure 7: Manto 1 Hole DDH955 – False Bottom Geology

In reference to Figure 6, Manto 1 has formed above the complex footwall stratigraphy reflecting a deepening basin on the eastern side of Cerro Montado island. The buried Cerro Montado island exhibits typical Boléo formation conglomerate directly overlying the island's Comondú volcanics on the western side of the basin. The manto occurs directly above conglomerate footwall in a patchy pattern only in the western sector, although sandstones and claystone false bottoms are also present above the conglomerate in the western sector. The eastern limit with presence of conglomerate in the footwall is shown as an interrupted black line.

East of this line there is no conglomerate present in the footwall rocks of the manto. The western limit of economic mineralization is shown as a thick red line in Figure 6 and mineralization tends to occur where "fingers" of Manto 1 overlie conglomerate with claystone false bottoms. This finger-like geometry seems to reflect the details of paleo-topography with well mineralized manto occurring along valley bottoms coming off Cerro Montado Island, where deeper and quieter submarine conditions led to deposition of claystone first and the entrapment of later richer brines forming Manto 1.

Moving westwards, the conglomerate facies disappears altogether, and the footwall stratigraphy is dominated first by sandstones and claystones and further to the east by a base of well laminated hard and light coloured limestones which are clearly identifiable (Figure 7).

The thickness of this unit at the base of Manto 1 ranges from a few tens of centimetres to over 1 meter. Even further to the east, the manto itself exhibits an increase in calcareous content indicating contemporaneous deposition of limestone and mineralized manto. Some of the highest copper grades (3% ~4% Cu) are associated with the zone where Manto 1 directly overlies either claystone or limestone in what may have represented a feeder zone that was coincident with a ridge.

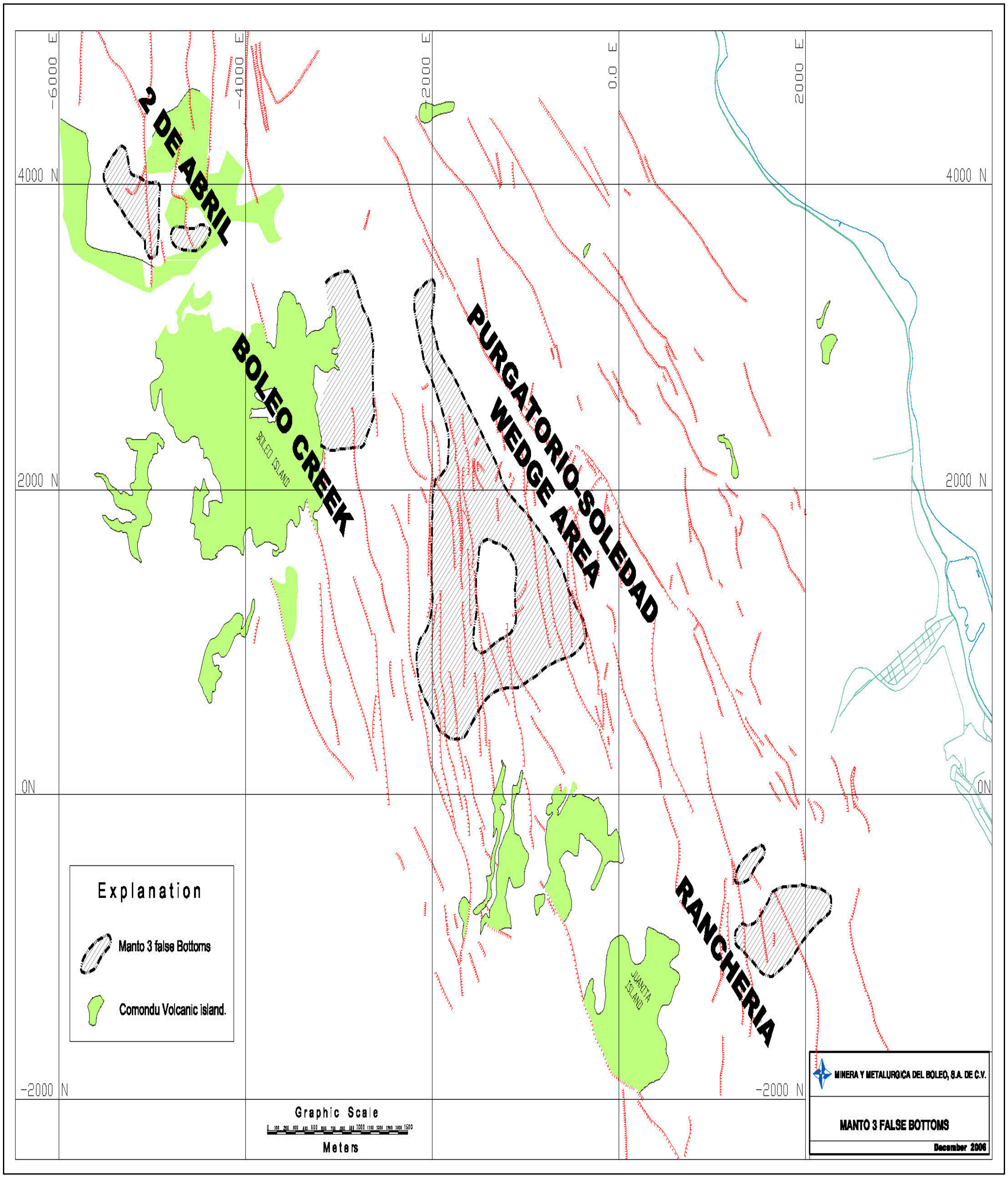
The location of old stopes also partly mimics the Manto 1 paleo-topography. Thus, the evidence provided by the stratigraphic changes in the footwall of the manto, as well as the pattern of the old stopes, and the paleo-topography of the base of Manto 1 all indicate that a hard finger-like geologic boundary for Manto 1 occurs to the west where the distribution of the brine was sharply limited by impoundment against the paleo-topographic barrier created by the buried Cerro Montado island. To the east, the manto shows a more gradational zoning pattern, where the best copper occurs in the central portion of the basin in the downthrown block of the Santa Agueda fault system, providing progressively better cobalt and lower copper grades further eastwards on the up thrown side of the Santa Agueda faults.

Several other areas with false bottoms have been identified in Manto 3 (see Figure 8). The Rancheria false bottom is located in the southern extreme of Manto 3 where it ends against the northern flank of the buried Cerro Montado Island. Where the manto under Providencia Creek sits directly above its footwall conglomerate, towards the south under the Rancheria ridge, the well mineralized horizon sits either in the middle (as in DDHs 06-463, 06-1072 and 06-1004) or towards the top (as in DDH06-970 and 06-971) of a wedge of low grade manto which occurs on the flanks of the buried island. This stratigraphic change is demonstrated by the position of old stopes that followed the well mineralized horizon downwards to a position only metres away from the high grade DDH06-1072. The surface area influenced by this facies change is approximately 600 m long and 250 m wide.

The second and largest area of Manto 3 over which a false bottom occurs is in the central portion of the district in between and eastwards of the Boléo and Juanita islands (Purgatorio-Soledad wedge). The funnel-shaped wedge is approximately 2.5 km long, 1.5 km wide at its southern base and less than 200 m wide at its northern top. The wedge is situated where the structural pattern of the district is dominated by north-south trending faults, reflecting the position of an extensional rip-apart sub-basin. This sub-basin is related to local normal and dextral fault displacements, which are responsible on a more regional scale for the formation of the Gulf of California.

What is common within the wedge is the presence of a low grade claystone or polymictic breccia false bottom above the footwall conglomerate, with the well mineralized horizon sitting at the top of Manto 3. The thickness of the false bottom ranges from less than a meter to several metres wide. In the central portion of the wedge the false bottom is absent and an “island” of footwall conglomerate directly underlies the mineralized manto.

Figure 8: Manto 3 Location of False Bottoms



Both vertical grade distribution in the metals, as well as the position of old stopes in this area provide additional evidence the well mineralized horizon occurs at the top of the manto. The example from three diamond holes (Table 4) demonstrates vertical change in grade between the claystone false bottom and the well mineralized top of Manto 3 in this area.

Table 4: Manto 3 - Vertical Grade Variation Associated With False Bottoms

DDH	Interval (m)	Width (m)	Cu%	Co%	Zn%	Mn%	Lithology
95-167	117.68-118.41	0.73	1.33	0.112	0.67	2.15	Retaque
	118.41-119.17	0.76	4.17	0.118	0.44	3.61	Retaque
	119.17-120.09	0.92	0.52	0.013	0.11	0.29	Claystone
	120.09-120.69	0.60	0.97	0.011	0.08	0.36	Claystone
	120.69-121.72	1.03	0.50	0.010	0.10	0.11	Claystone
95-193	139.54-140.15	0.61	1.27	0.126	0.12	4.25	Breccia
	140.15-141.14	0.99	2.75	0.161	0.16	4.20	Breccia
	141.14-142.13	0.99	0.08	0.007	0.06	0.15	Claystone
	142.13-142.50	0.37	0.02	0.010	0.13	0.25	Claystone
95-192	54.1-55.5	1.40	3.06	0.141	0.60	3.22	Polymictic Breccia
	55.5-56.9	1.40	0.63	0.024	0.27	0.22	Claystone

DDH 95-167 shows that the better mineralized part of the manto is in fact “retaque” (stope fill) indicating that the old stopes were located at the top of the manto and that mining took place above a claystone false bottom.

The Boléo Creek false bottom is located where Manto 3 approaches and ends up against the southeastern side of Boléo Island. The "2 de Abril" sub-basin, which is located in the northwestern sector of the district and at the northwestern side of Boléo Island, also exhibits the better mineralized horizon of Manto 3 at the top of the manto.

A false bottom has also been identified in parts of Manto 4 where the best mineralized area is also characterized by the higher grades being at the top of the manto.

10 EXPLORATION

Since acquiring control of the Boléo project in 2001, and prior to December 2004, MMB has concentrated on carrying out a complete geological, mining and processing review of the Boléo Property in place of a straight field exploration work. This includes an independent review of the open pit copper-cobalt-zinc manganese reserves, an in-depth review of alternate flow sheets for the processing of ore from the Boléo Property and an investigation into alternative mining methods, in particular the potential to use underground mining techniques.

This review resulted in a new pre-feasibility study being issued by Bateman Engineering Pty Limited, of Perth, Western Australia in February 2002, which focused principally on a new metallurgical flow sheet for the processing of ore from the Boléo Property.

Drilling activities commenced on the property in December 2004 (see Section 10.3 Drilling, below).

All historical work, including that carried out by Curator in conjunction with Mintec prior to its re-organization, is documented in Section 10.2 under History.

11 DRILLING

11.1 GENERAL

The current resource study uses all the drill holes completed by previous owners of the Boléo property, plus the additional holes of the three infill drill programs carried out by MMB between December 2004 and September 15, 2006 (DDHs 04-928 to 06-982).

A small number of these holes can be classified as exploration holes and did not serve the purpose of infill drilling. These holes are numbered 04-928, 04-929, 05-947 and 05-948.

11.2 HISTORICAL

The oldest recorded drilling program was carried out between 1927 and 1940 when 10,237 metres were drilled in 46 vertical churn holes (Wilson and Rocha, 1955). Most of these holes were drilled in the southeast portion of the property to explore for manto 1 in the Rancheria, San Luciano and Montado areas. Further diamond drilling was carried during the latter years of mining operations when Fomento Minero was looking for high-grade reserves to exploit. Records of this drilling were either never kept or have been lost or destroyed.

By far the most extensive drilling program ever conducted in the Boléo District was that of International Curator Resources Ltd. between 1993 and 1997. All exploration diamond drilling was completed using skid mounted Longyear 38 drill rigs moved with logging skidders. Core size was HQ (63.5 mm diameter), reduced to NQ (47.6 mm diameter) when necessary because of drilling problems. In areas where considerable thickness of overlying barren stratigraphy (Gloria, Infierno and Santa Rosalia formations) was expected, the upper portion of the hole was triconed before coring was begun near the target horizon.

The need to increase the Boléo resource to the measured and indicated classification led to the execution of three drill programs between December 2004 and January 2007.

11.3 2004 – 2007

Three infill drill programs were carried out by MMB on the property between December 2004 and January 2007. These drill programs were designed to infill the existing drill hole coverage with the aim of improving the confidence in the resource estimates. The location of all the drill holes is shown in Figure 9.

The first program comprised 15 holes for 2,566 m, between December 6, 2004 to January 29 2005 (DDHs 928 to 941), the second program comprised 18 holes for 3,174 m from May 11, 2005 to July 3, 2005 (DDHs 942 to 959), and the third program which comprised 123 holes for a total of 20,609 m from February 1, 2006 to September 15, 2006 (DDHs 960 to 1082). A total of 156 new drill holes were completed for a total of 26,349 m.

Figure 9: Drill Hole Locations –Historical & Infill

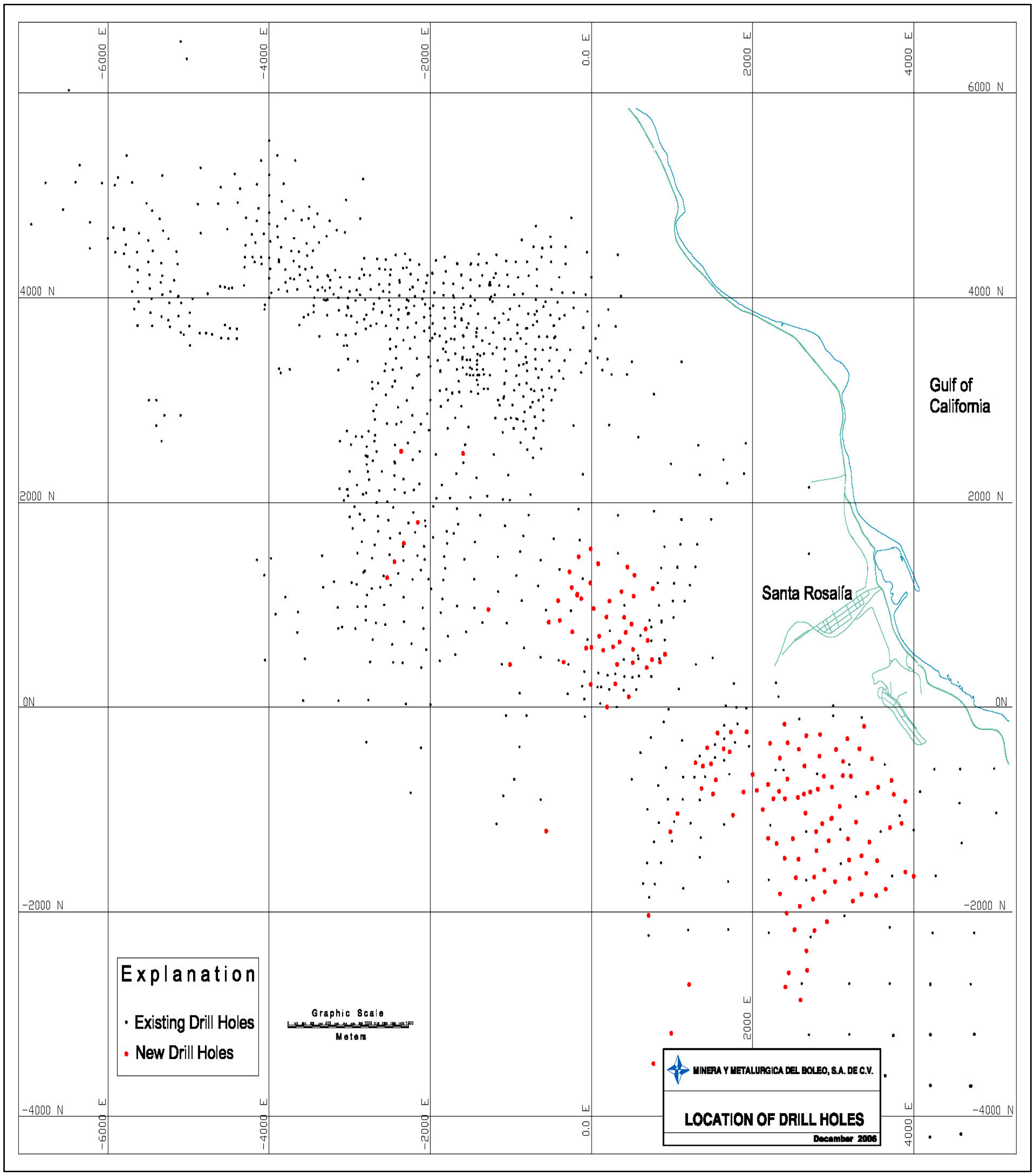


Table 5 contains the drill hole location information for the latest drill hole programs, covering diamond drill holes DDH 928 to 1082.

Table 6 contains the various results of the latest drilling programme, covering diamond drill holes DDH928 to 1082. Results are reported for the logged manto intersections only from each hole and are quoted as single composite intervals for each manto.

Table 5: Diamond Drill Hole Locations – Current Programs (DDH928-1082)

Hole ID	Easting	Northing	El.	Total Depth
04-0928	-561.96	-1209.94	264.22	190.75
04-0929	769.22	-3482.88	228.34	252.05
05-0930	-2156.14	1806.45	152.55	67.05
05-0931	-2328.86	1601.64	161.79	57.55
05-0932	-2444.79	1421.11	174.14	132.5
05-0933	-2535.9	1265.82	180.07	195.25
05-0934	-1005.47	420.98	107.45	159.85
05-0934a	-1010.42	417.43	107.62	192.15
05-0935	-344.8	440.01	80.44	167.75
05-0936	193.85	3.64	74.55	182.65
05-0937	-913.46	516.64	105.29	124.6
05-0938	1068.7	-1038.58	89.46	188.5
05-0939	978.24	-1216.07	93.01	142.85
05-0940	708.32	-2033.87	119.99	186.05
05-0941	-1279.98	954.24	267.34	326
05-0942	-177.44	1094.05	232.21	208.15
05-0943	-175.59	1103.88	231.82	344.65
05-0944	515.74	566.14	61.86	216.25
05-0945	-1593.29	2478.35	117.04	208.9
05-0946	-2362.16	2500.04	252.38	230.8
05-0947	1212.12	-2708.22	243.84	117.9
05-0948	989.89	-3186.24	238.08	238
05-0949	3119.52	-668.48	183.83	259.4
05-0950	3422.26	-839.11	77.5	143.4
05-0951	3079.72	-969.8	101.52	125.05
05-0952	2982.47	-1083.21	110	173
05-0953	3557.88	-782.85	67.35	112.5
05-0954	3751.26	-852.11	56.39	106.65
05-0955	3448.8	-1318.79	74.89	130.15
05-0956	317.8	418.34	69.97	56.9
05-0957	95.48	693.49	184.51	161.2
05-0958	405.25	877.91	158.56	165.6
05-0959	522.47	1083.51	150.71	175.8
06-0960	3350.7	-1452.221	85.664	135.11
06-0961	3196.833	-1492.658	96.4	169.88
06-0962	2809.012	-800.552	199.362	205.72
06-0963	2636.241	-849.439	204.97	202.82
06-0964	2327.664	-821.916	211.163	252.09
06-0965A	2984.115	-779.126	187.485	208.54
06-0966	2883.369	-674.971	190.566	202.82
06-0967	3218.229	-673.91	193.04	250.3
06-0968	2655.783	-1034.359	207.774	224.17
06-0969	2642.35	-573.727	204.795	227.22
06-0970	2053.019	-811.571	214.976	230.27
06-0971	2000.533	-657.837	218.508	245.52
06-0972	2336.122	-496.693	205.894	254.2
06-0973	-530.029	829.648	241.89	190.25
06-0974	-395.135	848.398	240.541	190.2
06-0975	-238	735.73	234.91	195
06-0976	-414.44	1041.63	235.267	195
06-0977	3281.76	-1121.34	180.018	219.41
06-0978	2430.27	-701.08	206.865	261.31
06-0979	3182.74	-1287.86	184.307	193.85
06-0980	2944.6	-1305.54	194.1	245.07
06-0981	-66.61	577.75	197.843	170.8
06-0982	-0.45	585.01	195.053	172.1
06-0983	186.82	881.68	192.893	201.55
06-0984	145.34	556.07	161.717	144.7
06-0985	268.06	589.28	158.41	157.05
06-0986	348.96	636.18	156.03	161.25

Hole ID	Easting	Northing	El.	Total Depth
06-0987	425.02	730.93	154.5	166.55
06-0988	497.43	811.61	152.72	167.85
06-0989	3121.87	-530.46	183.21	260.9
06-0990	-127.15	1060.82	227.46	210.4
06-0991	-245.12	1167.57	228.85	209.3
06-0992	1564.43	-251.47	64.8	45
06-0993	1434.63	-395.35	70.18	41.5
06-0994	1483.44	-552.94	70.71	51.55
06-0995	1290.67	-544.41	73.36	48.3
06-0996	2561.59	-881.73	202.26	243.58
06-0997	2791.285	-1400.491	202.72	177.92
06-0998	3723.1	-715.651	61.24	96.1
06-0999	3481.557	-504.168	82.63	173.6
06-1000	3324.58	-405.978	92.88	157.9
06-1001	26.82	963.92	192.06	177.25
06-1002	-14.16	1213.95	226.92	231.95
06-1003	82.96	1401.86	226.73	276.2
06-1004	1886.62	-829.83	222.07	295.63
06-1005	-272.81	1322.82	209.85	209.85
06-1006	-158.6	1469.56	224.01	248.7
06-1007	-8.238	1545.81	229.3	275.1
06-1008	2567.85	-1485.04	202.7	183.63
06-1009	2434	-347.39	201.74	225.93
06-1010	3174.75	-307.66	104.06	222.2
06-1011	2830.13	-478.42	135.66	201
06-1012	3034.68	-411.87	118.27	135.75
06-1013	533.68	1289.7	78.32	107.35
06-1014	373	1130	110	124.75
06-1015	223.55	1036.84	129.37	144.3
06-1016	759.925	1159.34	78.75	128.7
06-1017	671.336	764.103	55.47	73.25
06-1018	697.127	653.225	57.18	63.65
06-1019	3894.46	-920.119	47.32	93.95
06-1020	2396.99	-1475.467	211.616	200.08
06-1021	446.387	1370.493	92.538	118.1
06-1022	2787.684	-1215.667	126.783	120.85
06-1023	2863.332	-1137.225	120.304	138.75
06-1024	2499.2	-1285.33	148	142.1
06-1025	3544.254	-1499.62	163.457	232.35
06-1026	3410.035	-1622.453	177.24	278.95
06-1027	3202.275	-1674.148	178.337	262.2
06-1028	3023.133	-1702.861	190.199	180.4
06-1029	2893.864	-1803.652	194.269	238.5
06-1030	850.086	439.985	59.168	63.8
06-1031	751.213	465.275	58.282	55.65
06-1032	686.001	386.568	60.575	54.55
06-1033	462.774	102.808	64.628	48.75
06-1034	295.766	227.733	68.068	54.9
06-1035	-8.957	222.214	72.649	43.95
06-1036	513.177	433.436	63.271	57.2
06-1037	2296.533	-1331.305	165	114
06-1038	2192.228	-1281.247	175.499	128.15
06-1039	2973.117	-1087.651	110	118.3
06-1040	2921.715	-2095.619	101.316	177.7
06-1041	3350.474	-1826.613	80.091	184.95
06-1042	3243.21	-1894.093	84.925	170.7
06-1043	2676.089	-2570.624	124.23	156.15
06-1044	3848.69	-1133.866	45.992	175.55
06-1045	3704.952	-1176.098	56.308	129.8
06-1046	2888.242	-1589.836	117.881	128.15

Hole ID	Easting	Northing	El.	Total Depth
06-1047	2764.409	-1660.283	126.838	151.45
06-1048	2536.325	-1667.435	141.525	149.26
06-1049	3534.311	-1840.815	71.921	190.55
06-1050	3651	-1776	66	178.2
06-1051	1730.157	-242.117	59.372	44.3
06-1052	1638.898	-405.448	65.267	47.88
06-1053	1713.964	-437.524	64.32	51.85
06-1054	1383.348	-575.701	73.639	51.85
06-1055	1365.22	-793.227	77.935	69.9
06-1056	1925.381	-240.29	83.373	94.55
06-1057	2191.465	-754.431	211.239	222.7
06-1058	2256.946	-894.332	213.516	274.5
06-1059	2399.632	-893.945	208.342	245.5
06-1060	2714.409	-828.027	202.535	262.95
06-1061	2749.111	-1875.109	198.11	213.65
06-1062	2584.629	-1943.91	205.96	174.95
06-1063	2422.407	-2010.307	210.66	118.85
06-1064	2519.457	-217.098	207.602	137.35
06-1065	2339.935	-1824.408	209.87	147.9
06-1066	2666.607	-2379.18	116.735	131.5
06-1067	2593.19	-2860.708	140.921	113.7
06-1068	2406.14	-2733.43	141.621	68.5
06-1069	2448.127	-2594.405	142.433	81
06-1070	2765.874	-2181.574	109.119	152.8
06-1071	2124.837	-998.746	213.4676	222.85
06-1072	1754.549	-1052.788	223.001	280.95
06-1073	2215.643	-350.787	206.548	216.25
06-1074	2395.16	-164.752	196.685	218.2
06-1075	2572.601	-410.941	207.362	179.5
06-1076	1542.991	-709.676	139.376	137.7
06-1077	1507.634	-847.329	147.345	153.15
06-1078	3383.144	-186.706	185.193	206.4
06-1079	2665.826	-278.444	193.13	224.55
06-1080	2837.633	-267.714	186.03	253.15
06-1081	3895.591	-1609.201	50.764	193.35
06-1082	3998	-1653	45	196.75

Table 6: Diamond Drill Hole Results – Current Programs (DDH928-1082)

Hole ID	From	To	Thickness	Manto	Cu%	Co%	Zn%	Mn%
04-0928	167.04	170.36	3.32	4	0.13	0.009	0.32	2.47
04-0929	147.4	151.3	3.9	3	0.02	0.006	5.12	1.52
04-0929	174.72	177.4	2.68	4	0.00	0.003	1.75	4.74
04-0929	189.4	193.22	3.82	5	0.28	0.006	0.61	2.51
05-0930	3.5	6.1	2.6	3	2.01	0.109	0.72	5.25
05-0930	55.05	60.98	5.93	4	0.50	0.033	0.32	0.69
05-0931	5.35	9.1	3.75	3	1.66	0.039	0.41	2.16
05-0931	49.34	54.76	5.42	4	0.47	0.038	0.44	0.59
05-0932	111.65	113.27	1.62	4a	2.24	0.092	0.82	0.84
05-0932	119.6	126.83	7.23	4	1.01	0.075	0.39	2.88
05-0933	6.7	13.3	6.6	3	0.28	0.040	0.62	2.85
05-0933	172.8	179.27	6.47	4	0.29	0.022	0.14	0.18
05-0934	114.65	152.1	37.45	4	0.46	0.028	0.20	1.00
05-0934a	116.1	147.16	31.06	4	0.28	0.030	0.25	0.94
05-0935	19	21.35	2.35	3	1.10	0.076	0.56	7.41
05-0935	97.4	102.61	5.21	4a	0.18	0.022	0.33	0.08
05-0935	108.4	117.45	9.05	4	0.08	0.059	0.31	0.90
05-0936	26.3	27.7	1.4	3a	0.11	0.043	1.15	4.14
05-0936	33.35	36.14	2.79	3	1.04	0.066	0.44	2.54
05-0936	128.1	150.26	22.16	4	0.07	0.032	0.17	0.17
05-0937	78.1	86.12	8.02	4	0.14	0.016	0.15	0.96
05-0938	48.05	54.5	6.45	3	0.48	0.009	0.17	0.52
05-0938	110.43	122.31	11.88	4a	0.08	0.021	0.15	0.37
05-0938	128.67	131.5	2.83	4	0.32	0.041	0.17	3.12
05-0939	43.65	47.7	4.05	3a	0.16	0.040	0.30	2.04

Hole ID	From	To	Thickness	Manto	Cu%	Co%	Zn%	Mn%
05-0939	51.6	54.58	2.98	3	2.57	0.081	0.55	4.89
05-0939	109.7	123.25	13.55	4	0.65	0.038	1.08	0.57
05-0940	48.8	61	12.2	3a	0.01	0.004	0.07	0.06
05-0940	117.13	157.74	40.61	4	0.09	0.015	0.17	0.44
05-0941	73.7	78.77	5.07	2	0.09	0.009	0.37	2.09
05-0941	145.8	147.29	1.49	3	2.21	0.122	0.56	7.20
05-0941	296.1	309.8	13.7	4a	0.74	0.030	0.34	0.58
05-0942	143.15	148.25	5.1	2	1.28	0.061	0.26	3.78
05-0942	201.75	203.21	1.46	3a	0.33	0.052	0.63	5.42
05-0943	146.05	149.7	3.65	2	1.75	0.041	0.68	2.35
05-0943	200.35	203.4	3.05	3	0.12	0.040	0.73	7.21
05-0943	328.4	332.55	4.15	4	0.14	0.025	0.18	0.88
05-0944	52.1	53.1	1	3a	0.06	0.042	1.43	6.08
05-0944	55.28	57.57	2.29	3	1.87	0.189	0.44	3.73
05-0944	205.52	215.85	10.33	4	0.05	0.046	0.26	1.59
05-0945	34.91	36	1.09	2a	0.40	0.017	0.20	0.45
05-0945	39.9	44.35	4.45	2	0.02	0.040	0.66	4.56
05-0945	73.7	78.42	4.72	3	2.20	0.028	0.20	0.91
05-0945	197.15	206.88	9.73	4	0.03	0.019	0.16	0.21
05-0946	113.05	116.91	3.86	3	2.43	0.039	0.10	0.33
05-0946	187	188.3	1.3	4	0.01	0.013	0.13	0.34
05-0947	73.5	78.41	4.91	3	0.02	0.003	0.71	1.60
05-0947	88.34	94.19	5.85	4	0.06	0.004	0.35	1.16
05-0948	97.35	104.27	6.92	1	0.00	0.002	0.08	0.30
05-0948	108.9	113.75	4.85	2	0.05	0.004	1.19	4.10
05-0948	119.6	122.33	2.73	3	0.17	0.005	0.75	4.61
05-0948	157.4	166.19	8.79	4a	0.14	0.008	1.14	3.54
05-0948	174.9	179.3	4.4	4	0.31	0.018	0.29	5.38
05-0949	185.42	187.1	1.68	0a	0.01	0.008	0.93	3.30
05-0949	190.48	191.4	0.92	0	0.01	0.006	0.72	1.00
05-0949	225.25	227.27	2.02	1	2.55	0.109	0.56	4.24
05-0950	63.7	64.16	0.46	0a	0.03	0.012	0.96	0.60
05-0950	66.44	68.03	1.59	0	0.01	0.008	0.42	0.70
05-0950	97.17	100.03	2.86	1	4.37	0.083	0.40	3.25
05-0951	47.21	49.7	2.49	0	0.07	0.011	0.45	0.44
05-0951	90.11	91.82	1.71	1	6.15	0.172	0.23	1.15
05-0952	108.75	111.7	2.95	1	0.36	0.036	0.42	4.53
05-0952	152.02	154.17	2.15	2	0.42	0.103	0.61	6.46
05-0952	160.6	169.9	9.3	3	0.05	0.062	0.82	7.10
05-0953	74.15	78	3.85	0	0.01	0.006	0.25	0.83
05-0953	98.87	100.75	1.88	1	2.16	0.107	0.64	3.41
05-0954	55.03	56.83	1.8	0a	0.02	0.006	0.42	1.01
05-0954	58.89	60.77	1.88	0	0.02	0.027	0.59	1.46
05-0954	94.55	97.06	2.51	1	1.73	0.085	0.59	4.40

Hole ID	From	To	Thickness	Manto	Cu%	Co%	Zn%	Mn%
05-0955	70.27	71.41	1.14	0a	0.03	0.007	0.80	2.40
05-0955	73.92	76.78	2.86	0	0.02	0.008	0.80	0.75
05-0955	121.45	123.74	2.29	1	3.17	0.090	0.98	4.45
05-0956	46.45	53.98	7.53	3	1.00	0.041	0.36	1.88
05-0957	110.13	116.27	6.14	2	0.42	0.044	0.38	1.82
05-0957	152	157.88	5.88	3	1.36	0.071	0.41	2.65
05-0958	113.28	114.73	1.45	2	1.94	0.143	0.39	7.50
05-0958	157.48	163.56	6.08	3	0.75	0.033	0.59	3.42
05-0959	126.5	130.52	4.02	2	0.65	0.077	0.50	5.44
05-0959	166.6	174.25	7.65	3	0.76	0.018	0.26	0.80
06-0960	73.11	80	6.89	0	0.02	0.008	0.27	0.51
06-0960	128.55	132.24	3.69	1	0.64	0.070	0.55	2.22
06-0961	52.46	54.55	2.09	0	0.01	0.019	0.44	0.84
06-0961	100.87	104.16	3.29	1	0.20	0.024	0.26	1.69
06-0961	163.78	164.38	0.6	2	0.01	0.014	0.42	2.19
06-0962	159.94	160.85	0.91	0a	0.02	0.009	0.98	1.43
06-0962	166.37	168.56	2.19	0	0.02	0.007	0.42	0.67
06-0962	199	201.69	2.69	1	2.19	0.069	0.60	3.24
06-0963	134.96	138.2	3.24	0a	0.01	0.004	0.30	1.37
06-0963	139.47	143.51	4.04	0	0.01	0.011	0.57	0.65
06-0963	175.37	180.48	5.11	1	0.50	0.032	0.52	1.76
06-0964	103.37	105.8	2.43	0a	0.01	0.004	0.31	1.40
06-0964	108	112.62	4.62	0	0.01	0.010	0.66	0.98
06-0964	135.72	145.65	9.93	1	0.06	0.024	0.47	2.36
06-0964	204.85	207.03	2.18	2	6.31	1.435	4.67	10.64
06-0965A	161.65	166.15	4.5	0	0.04	0.014	0.53	0.72
06-0965A	198.15	202.64	4.49	1	1.86	0.075	0.58	4.73
06-0966	152.27	153.48	1.21	0a	0.03	0.008	0.53	2.59
06-0966	155.5	158.6	3.1	0	0.02	0.013	0.52	0.82
06-0966	191.92	194.62	2.7	1	1.41	0.090	0.80	3.01
06-0967	202.41	208.2	5.79	0	0.01	0.004	0.18	0.63
06-0967	242.98	244.73	1.75	1	2.89	0.065	0.27	1.46
06-0968	145.71	150.2	4.49	1	0.58	0.041	0.63	1.74
06-0968	214.26	217.3	3.04	2	2.47	0.252	2.19	7.78
06-0969	156.1	159.46	3.36	1	1.31	0.058	0.38	1.67
06-0969	218.07	225.82	7.75	2	0.12	0.047	0.46	2.75
06-0970	84.65	89.8	5.15	1	0.01	0.008	1.10	2.73
06-0970	120.44	123.87	3.43	2a	0.01	0.004	0.22	1.07
06-0970	128.22	130.32	2.1	2	0.16	0.014	0.60	3.43
06-0970	193.65	195.81	2.16	3	1.66	0.065	0.80	3.34
06-0970	199.26	209.94	10.68	4	0.35	0.036	0.17	3.90
06-0971	72.56	74.43	1.87	0a	0.01	0.005	0.17	2.23
06-0971	77.48	81.19	3.71	0	0.01	0.013	1.18	2.09
06-0971	108.27	120.47	12.2	1	0.05	0.006	0.37	0.96

Hole ID	From	To	Thickness	Manto	Cu%	Co%	Zn%	Mn%
06-0971	173.57	180.42	6.85	2	0.14	0.019	0.46	1.87
06-0971	218.07	222.02	3.95	3	2.44	0.039	0.47	2.67
06-0972	119.91	121.11	1.2	0a	0.01	0.006	0.34	1.53
06-0972	123.96	127.34	3.38	0	0.05	0.007	0.29	0.45
06-0972	157.89	158.66	0.77	1	1.86	0.041	0.29	2.38
06-0972	205.2	207.2	2	2	0.89	0.066	0.43	4.42
06-0972	234.42	236.26	1.84	3	0.17	0.030	0.62	1.72
06-0973	125	127.1	2.1	2	0.15	0.013	1.49	4.58
06-0973	183.95	184.9	0.95	3a	0.21	0.068	1.18	5.03
06-0973	188.45	189.07	0.62	3	0.63	0.027	0.28	0.79
06-0974	123.4	128.15	4.75	2	0.16	0.014	0.92	1.60
06-0974	182.7	184	1.3	3a	1.47	0.138	0.85	6.15
06-0974	187	188.35	1.35	3	1.79	0.121	0.49	2.56
06-0975	137.05	141.72	4.67	2	0.06	0.019	0.72	5.66
06-0975	184.35	185.7	1.35	3a	0.18	0.057	1.51	7.52
06-0975	189.07	191	1.93	3	1.99	0.065	0.46	2.18
06-0976	130.85	134.05	3.2	2	2.21	0.070	0.99	5.90
06-0976	186.25	187.8	1.55	3a	2.53	0.067	1.06	9.41
06-0976	188.7	191.45	2.75	3	7.07	0.076	0.35	1.16
06-0977	154.49	155.89	1.4	0a	0.05	0.008	0.20	1.05
06-0977	158.22	161.48	3.26	0	0.01	0.007	0.27	0.48
06-0977	205.24	207.53	2.29	1	3.68	0.135	0.57	2.98
06-0978	123.83	125.39	1.56	0a	0.02	0.007	0.12	1.36
06-0978	129.72	131.75	2.03	0	0.04	0.011	0.55	0.71
06-0978	157.34	159.71	2.37	1	0.59	0.026	0.28	2.33
06-0978	211.31	213.71	2.4	2	0.90	0.020	0.35	3.31
06-0978	241.72	243.74	2.02	3	0.24	0.016	0.12	2.74
06-0978	252	253.68	1.68	4	0.22	0.017	0.19	1.92
06-0979	136.5	138.4	1.9	0a	0.02	0.007	0.53	1.67
06-0979	140.34	142.94	2.6	0	0.01	0.010	0.61	0.59
06-0979	189.5	190.15	0.65	1	1.86	0.036	0.17	2.71
06-0980	164.66	169.46	4.8	1	1.35	0.063	0.42	2.74
06-0980	233.78	235.46	1.68	2	0.48	0.088	1.37	5.65
06-0980	238.37	245.07	6.7	3	0.02	0.042	0.47	4.26
06-0981	105.35	107.15	1.8	2	0.11	0.015	0.36	2.07
06-0981	161.8	162.64	0.84	3a	0.12	0.046	2.31	12.58
06-0981	167.1	168.73	1.63	3	3.68	0.041	0.33	0.42
06-0982	105.6	109.82	4.22	2	1.53	0.043	0.69	2.77
06-0982	161.2	162.45	1.25	3a	0.84	0.110	0.46	8.80
06-0982	167.7	169.1	1.4	3	2.38	0.086	0.28	2.36
06-0983	138.8	141.07	2.27	2	1.63	0.097	0.38	6.60
06-0983	192.9	195.2	2.3	3a	0.05	0.026	0.35	2.95
06-0983	199.2	199.6	0.4	3	3.72	0.080	0.34	2.47
06-0984	135.95	137.52	1.57	3a	0.38	0.040	0.36	3.68

Hole ID	From	To	Thickness	Manto	Cu%	Co%	Zn%	Mn%
06-0984	140.55	142.1	1.55	3	3.23	0.032	0.21	1.76
06-0985	92.6	96.65	4.05	2	0.66	0.052	0.63	4.78
06-0985	148.9	149.2	0.3	3a	0.02	0.015	0.17	0.28
06-0985	151.45	153.5	2.05	3	0.56	0.019	0.38	1.89
06-0986	99.25	104.47	5.22	2	0.13	0.045	0.31	2.29
06-0986	154.75	157.25	2.5	3a	0.28	0.077	0.74	5.93
06-0986	158.93	160.05	1.12	3	3.68	0.121	0.49	4.32
06-0987	107.9	110.52	2.62	2	0.09	0.039	0.47	2.32
06-0987	160.35	161.1	0.75	3a	0.04	0.010	0.16	0.70
06-0987	162.3	164.4	2.1	3	4.82	0.167	0.28	1.61
06-0988	110.02	112.64	2.62	2	0.16	0.067	0.61	5.58
06-0988	159.6	161.45	1.85	3a	0.11	0.035	0.58	3.43
06-0988	163.7	163.82	0.12	3	8.15	0.026	0.24	3.34
06-0989	182.65	187.25	4.6	0	0.06	0.016	0.50	0.83
06-0989	217.44	220.19	2.75	1	0.75	0.044	0.16	2.42
06-0990	151.5	153.75	2.25	2	1.57	0.106	1.62	8.32
06-0990	201.8	202.8	1	3a	0.20	0.066	0.59	6.22
06-0990	204.52	206.25	1.73	3	2.92	0.048	0.37	1.46
06-0991	141.6	144.4	2.8	2	2.22	0.135	0.75	5.33
06-0991	200	204.58	4.58	3a	0.09	0.018	0.28	1.88
06-0991	206.39	207.69	1.3	3	1.57	0.065	0.51	2.42
06-0992	32.8	37.4	4.6	3	2.09	0.038	0.53	3.84
06-0993	34.1	40.03	5.93	3	0.89	0.036	0.24	2.19
06-0994	45.34	46.02	0.68	3a	0.01	0.015	0.18	0.24
06-0994	46.77	49.41	2.64	3	2.59	0.107	0.43	5.02
06-0995	38	38.35	0.35	3a	0.08	0.017	0.36	3.49
06-0995	38.8	40.8	2	3	2.30	0.013	0.22	0.17
06-0996	110.41	112.67	2.26	0a	0.01	0.003	0.14	0.93
06-0996	116.6	119.04	2.44	0	0.04	0.014	0.85	0.97
06-0996	152	159.56	7.56	1	1.34	0.081	0.64	3.71
06-0996	230.85	233.95	3.1	2	0.58	0.103	0.94	8.75
06-0997	125.36	128.1	2.74	1	0.15	0.037	0.74	4.36
06-0997	166.58	172.2	5.62	2	0.03	0.018	0.27	1.30
06-0998	51.35	52.9	1.55	0a	0.01	0.007	0.64	1.07
06-0998	54.35	56.45	2.1	0	0.01	0.010	0.66	0.97
06-0998	89.5	91.15	1.65	1	1.01	0.119	0.64	2.88
06-0999	116	116.8	0.8	0a	0.02	0.013	0.83	1.93
06-0999	119.5	121.55	2.05	0	0.01	0.008	0.34	0.58
06-0999	168.9	170.7	1.8	1	0.13	0.093	0.73	7.20
06-1000	111.75	113.2	1.45	0a	0.01	0.007	0.28	1.03
06-1000	115.45	117.3	1.85	0	0.06	0.009	0.61	0.62
06-1000	149.3	151.16	1.86	1	2.03	0.067	0.47	3.76
06-1001	114.89	118.04	3.15	2	0.87	0.108	0.51	2.97
06-1001	172	175.01	3.01	3	0.54	0.084	0.75	5.22

Hole ID	From	To	Thickness	Manto	Cu%	Co%	Zn%	Mn%
06-1002	173.56	175.39	1.83	2	0.32	0.058	0.57	4.73
06-1002	226.6	227.97	1.37	3a	0.02	0.023	0.66	4.04
06-1002	228.71	230.1	1.39	3	2.49	0.046	0.43	1.53
06-1003	139.45	141.26	1.81	1	0.02	0.016	0.86	3.42
06-1003	218.9	220.31	1.41	2	0.23	0.033	0.77	4.58
06-1003	268.44	269.78	1.34	3a	0.02	0.021	0.55	4.74
06-1003	272.6	273.75	1.15	3	3.11	0.051	0.36	1.77
06-1004	119.19	121.53	2.34	1	0.27	0.018	0.59	5.08
06-1004	190.69	194.55	3.86	2	0.09	0.020	0.48	1.05
06-1004	237.72	244.4	6.68	3	0.14	0.016	0.30	0.20
06-1004	284.54	293.58	9.04	4	0.18	0.023	0.06	2.84
06-1005	137.64	147.2	9.56	2	0.18	0.022	0.34	1.32
06-1005	202.74	205.11	2.37	3a	0.03	0.027	0.57	3.76
06-1006	184.65	187.7	3.05	2	0.13	0.043	0.49	3.13
06-1006	239.55	241.35	1.8	3a	0.06	0.025	0.42	3.31
06-1007	119.5	120.37	0.87	0	0.01	0.009	3.32	5.24
06-1007	145.2	150.38	5.18	1	0.02	0.012	0.40	3.01
06-1007	222.69	227.65	4.96	2	0.10	0.023	0.76	3.24
06-1007	265.92	266.67	0.75	3a	0.20	0.033	0.60	4.91
06-1007	268.5	270.87	2.37	3	0.27	0.027	0.21	0.68
06-1008	96.25	100.45	4.2	0	0.01	0.007	0.90	1.93
06-1008	137.31	140.15	2.84	1	0.08	0.032	1.09	3.37
06-1008	155.38	157.91	2.53	2	0.31	0.047	0.81	5.23
06-1008	175.05	177.94	2.89	3	0.00	0.018	0.67	3.82
06-1009	117.69	118.67	0.98	0a	0.01	0.006	0.33	2.03
06-1009	121.14	121.71	0.57	0	0.02	0.006	0.82	0.97
06-1009	149.03	152.68	3.65	1	0.72	0.048	0.56	1.20
06-1009	194.7	198.23	3.53	2	0.58	0.126	0.43	5.80
06-1009	220.16	223.12	2.96	3	0.22	0.021	0.43	0.43
06-1010	109.7	113	3.3	0	0.08	0.009	0.67	0.96
06-1010	146.9	148.4	1.5	1	1.23	0.061	0.35	2.28
06-1010	215.7	217.85	2.15	2	0.24	0.085	0.66	7.31
06-1011	125.53	126.67	1.14	1	0.11	0.007	0.32	0.81
06-1011	178.42	180.25	1.83	2	0.10	0.067	1.09	15.99
06-1011	191.95	193.8	1.85	3	0.21	0.028	0.21	0.57
06-1012	92.86	96.35	3.49	0	0.04	0.006	0.41	0.97
06-1012	128.1	131.7	3.6	1	1.35	0.071	0.31	2.71
06-1013	36.44	37.53	1.09	1	0.01	0.004	0.03	0.10
06-1013	76.86	78.4	1.54	2	0.45	0.041	0.61	2.93
06-1013	90.85	96.47	5.62	3a	0.17	0.031	0.20	0.98
06-1013	96.47	99.95	3.48	3	0.50	0.052	0.22	1.53
06-1014	74.79	77.56	2.77	2	0.27	0.058	1.20	8.34
06-1014	116.11	118.5	2.39	3a	0.05	0.031	0.38	2.68
06-1014	119.1	122.08	2.98	3	0.16	0.023	0.19	0.88

Hole ID	From	To	Thickness	Manto	Cu%	Co%	Zn%	Mn%
06-1015	84.7	89.45	4.75	2	0.09	0.026	0.34	1.51
06-1015	137.29	140.93	3.64	3a	0.17	0.017	0.28	1.90
06-1015	140.93	142.66	1.73	3	5.91	0.068	0.51	5.35
06-1016	60.35	63.4	3.05	2	0.30	0.074	2.22	11.05
06-1016	87.65	89.87	2.22	3	0.47	0.062	0.14	0.62
06-1016	119.22	121.06	1.84	4	0.55	0.021	0.14	0.43
06-1017	24.45	25.93	1.48	2	0.11	0.006	0.08	0.73
06-1017	60.4	61.74	1.34	3a	0.23	0.050	0.52	4.65
06-1017	62.32	65.68	3.36	3	1.27	0.095	0.26	2.17
06-1018	54.75	60.36	5.61	3	2.23	0.054	0.33	3.54
06-1019	66	70.05	4.05	2	0.10	0.080	0.68	9.07
06-1019	83.64	85.45	1.81	3	0.24	0.027	0.27	0.62
06-1019	87.3	87.5	0.2	4	3.25	0.352	0.01	0.89
06-1020	96.15	97.18	1.03	0	0.08	0.007	1.12	3.68
06-1020	111.67	112.68	1.01	1	0.23	0.008	0.54	4.21
06-1020	146.52	150.17	3.65	2	0.16	0.038	0.69	2.39
06-1020	175.6	178.75	3.15	3	0.01	0.016	0.78	5.99
06-1021	71.88	74.47	2.59	2	0.25	0.062	0.88	3.84
06-1021	94.35	95.35	1	3a	0.46	0.025	0.43	2.26
06-1021	96	99.8	3.8	3	1.06	0.088	0.32	1.81
06-1022	61.89	64.77	2.88	1	0.73	0.069	1.03	2.26
06-1022	111.44	116.18	4.74	2	1.08	0.140	1.65	5.29
06-1023	66.45	68.71	2.26	1	2.52	0.089	0.86	2.57
06-1023	132.96	134.19	1.23	2	0.13	0.080	1.24	8.93
06-1024	39.25	43.24	3.99	0	0.01	0.007	1.05	2.53
06-1024	88.8	92.14	3.34	1	2.15	0.063	0.73	1.79
06-1024	135.32	137.48	2.16	2	0.12	0.043	2.96	10.74
06-1025	176.65	178.56	1.91	0a	0.01	0.005	0.22	1.23
06-1025	180.98	182.3	1.32	0	0.08	0.019	0.69	1.47
06-1025	226.92	228.52	1.6	1	0.99	0.112	1.63	5.20
06-1026	169.65	170.77	1.12	0a	0.02	0.008	0.86	2.29
06-1026	171.65	176.46	4.81	0	0.01	0.007	0.24	0.44
06-1026	221.42	223.53	2.11	1	0.86	0.147	1.25	4.62
06-1027	129.73	131.25	1.52	0a	0.03	0.005	0.47	1.43
06-1027	133.73	136.44	2.71	0	0.01	0.008	0.63	0.54
06-1027	182.4	186	3.6	1	1.29	0.051	0.67	1.18
06-1027	253.1	254.85	1.75	2	0.11	0.070	0.58	5.74
06-1028	105.75	108.85	3.1	0	0.04	0.004	0.79	0.57
06-1028	158.08	159.8	1.72	1	1.93	0.153	0.52	0.37
06-1029	125.2	127.05	1.85	0a	0.02	0.004	0.09	0.86
06-1029	130.65	132	1.35	0	0.03	0.011	0.60	0.93
06-1029	184.75	186.06	1.31	1	1.44	0.141	0.42	0.84
06-1029	233.85	235	1.15	2	0.16	0.086	1.06	10.73
06-1030	15.3	17.76	2.46	2	0.34	0.052	0.72	6.11

Hole ID	From	To	Thickness	Manto	Cu%	Co%	Zn%	Mn%
06-1030	54.56	55.35	0.79	3a	0.08	0.067	0.76	5.40
06-1030	58.57	60.37	1.8	3	1.05	0.052	0.31	2.09
06-1031	45.83	46.63	0.8	3a	0.78	0.077	0.61	7.82
06-1031	49.26	52.08	2.82	3	2.65	0.069	0.20	1.59
06-1032	43.57	45.06	1.49	3a	0.05	0.014	0.30	2.02
06-1032	47.45	49.7	2.25	3	3.54	0.075	0.43	4.26
06-1033	34.29	41.4	7.11	3	0.32	0.020	0.25	0.64
06-1034	42.44	47.82	5.38	3	0.56	0.030	0.34	1.86
06-1035	30.5	31.83	1.33	3a	0.68	0.087	0.68	6.97
06-1035	35.23	41.1	5.87	3	1.73	0.073	0.17	0.04
06-1036	48.11	49.68	1.57	3a	0.08	0.038	2.40	6.93
06-1036	50.59	52.33	1.74	3	2.36	0.178	0.39	4.42
06-1037	35.9	40.15	4.25	0	0.05	0.007	0.91	3.07
06-1037	65.61	68.44	2.83	1	0.10	0.014	0.95	6.39
06-1037	99	101.25	2.25	2	0.35	0.059	1.16	5.75
06-1038	75.77	78.33	2.56	1	0.12	0.018	0.77	7.89
06-1038	89.1	92.14	3.04	2	0.78	0.070	1.34	6.97
06-1038	121.35	124.3	2.95	3	0.01	0.019	1.20	10.55
06-1039	108.5	110.64	2.14	1	1.11	0.062	0.57	3.69
06-1040	40.93	43.15	2.22	0a	0.01	0.005	0.25	1.58
06-1040	44.45	48.25	3.8	0	0.01	0.005	0.30	1.04
06-1040	100.45	103.3	2.85	1	0.77	0.025	0.60	1.61
06-1040	145.5	148.49	2.99	2	0.42	0.070	0.41	4.61
06-1041	61.87	63.98	2.11	0a	0.04	0.007	0.27	1.10
06-1041	66.85	70.1	3.25	0	0.03	0.008	0.26	0.91
06-1041	119.01	120.85	1.84	1	1.97	0.107	1.12	3.15
06-1041	177.53	178.95	1.42	2	0.10	0.083	0.87	6.30
06-1042	45.4	47.7	2.3	0a	0.01	0.004	0.23	0.57
06-1042	51.05	53.2	2.15	0	0.03	0.016	0.47	0.76
06-1042	103.23	105.4	2.17	1	2.00	0.160	0.72	3.33
06-1042	157.2	157.8	0.6	2	0.02	0.114	0.78	9.40
06-1043	54.64	55.99	1.35	0a	0.02	0.006	0.07	0.89
06-1043	57.15	61.08	3.93	0	0.01	0.005	0.52	0.47
06-1043	97.16	100.14	2.98	1	1.52	0.128	1.13	1.61
06-1043	114.15	117.57	3.42	2	1.02	0.099	2.88	8.52
06-1044	68.63	71.25	2.62	0a	0.01	0.007	0.53	1.21
06-1044	74.4	75.65	1.25	0	0.01	0.012	0.28	0.90
06-1044	107.7	109.15	1.45	1	2.48	0.106	0.65	5.89
06-1044	169.05	171.45	2.4	2	0.38	0.092	0.82	5.72
06-1045	78.7	80.05	1.35	0a	0.02	0.006	0.30	1.04
06-1045	81.65	84.53	2.88	0	0.02	0.006	0.41	0.88
06-1045	123.75	124.85	1.1	1	1.91	0.122	1.06	6.96
06-1046	39.88	42.1	2.22	1	0.08	0.043	1.24	7.84
06-1046	63.7	65.65	1.95	2	0.29	0.100	1.06	6.61

Hole ID	From	To	Thickness	Manto	Cu%	Co%	Zn%	Mn%
06-1046	93.1	94.5	1.4	3	0.01	0.024	0.47	0.16
06-1046	100	103.9	3.9	4	0.42	0.072	0.40	4.89
06-1047	35.89	37.53	1.64	0a	0.01	0.004	0.09	0.94
06-1047	40.65	42.9	2.25	0	0.02	0.010	0.77	1.43
06-1047	82.85	86	3.15	1	1.08	0.086	1.77	2.18
06-1048	36.3	47.55	11.25	0	0.02	0.006	0.41	1.42
06-1048	86.05	87.76	1.71	1	0.03	0.044	2.52	7.59
06-1048	129.32	131.25	1.93	2	0.60	0.055	0.99	5.35
06-1049	73.35	74.4	1.05	0a	0.01	0.004	0.18	0.46
06-1049	75.4	78.7	3.3	0	0.03	0.007	0.35	0.85
06-1049	122.7	125.55	2.85	1	0.10	0.037	0.59	1.33
06-1049	178.28	181.25	2.97	2	0.03	0.029	0.32	2.84
06-1050	70.1	71.2	1.1	0a	0.02	0.006	0.08	0.76
06-1050	72.35	74.85	2.5	0	0.03	0.009	0.58	0.84
06-1050	113.8	116.4	2.6	1	0.91	0.082	0.72	3.12
06-1050	169.65	172.9	3.25	2	0.42	0.091	0.50	7.09
06-1051	37	37.37	0.37	3a	0.01	0.018	0.49	0.10
06-1051	38.23	41.52	3.29	3	0.82	0.026	0.45	2.14
06-1052	40.55	41.3	0.75	3a	0.04	0.017	0.21	0.16
06-1052	42.23	42.7	0.47	3	0.18	0.018	0.28	0.53
06-1053	44.52	45.03	0.51	3a	0.01	0.016	0.29	0.16
06-1053	46.63	49	2.37	3	0.94	0.051	0.62	5.21
06-1054	45.4	46.71	1.31	3a	0.08	0.021	0.47	3.91
06-1054	47.8	50.15	2.35	3	3.50	0.064	0.21	1.33
06-1055	3.25	8.2	4.95	2	0.81	0.039	0.37	3.50
06-1055	64.78	66.55	1.77	3	0.37	0.023	0.26	0.16
06-1056	49.61	53.95	4.34	2	0.16	0.051	1.45	6.31
06-1056	85.4	90.28	4.88	3	0.09	0.024	0.44	1.32
06-1057	77.25	80.2	2.95	0	0.03	0.018	0.81	1.68
06-1057	109.66	113.66	4	1	0.20	0.019	0.76	1.73
06-1057	162.88	165.78	2.9	2	0.31	0.059	1.02	9.05
06-1057	198.77	200.6	1.83	3	0.44	0.042	0.56	6.36
06-1057	206.97	209.94	2.97	4	0.04	0.033	0.39	2.11
06-1058	91.5	92.95	1.45	0a	0.01	0.007	0.25	3.06
06-1058	97.1	98.9	1.8	0	0.01	0.014	1.33	2.77
06-1058	133.15	135.4	2.25	1	0.04	0.032	1.73	8.64
06-1058	199.88	203.19	3.31	2	0.22	0.042	3.05	7.00
06-1058	244.61	251.65	7.04	3	0.07	0.014	0.31	1.10
06-1058	259.65	263.2	3.55	4	0.06	0.018	0.15	2.80
06-1059	107.65	108.51	0.86	0a	0.01	0.008	0.38	2.27
06-1059	112.01	113.92	1.91	0	0.01	0.023	0.90	1.36
06-1059	145.06	151.81	6.75	1	0.04	0.040	1.22	4.24
06-1059	219.55	221.47	1.92	2	0.42	0.130	0.38	2.10
06-1059	240.81	243.3	2.49	3	0.10	0.017	0.23	0.52

Hole ID	From	To	Thickness	Manto	Cu%	Co%	Zn%	Mn%
06-1060	150.21	152.27	2.06	0a	0.01	0.003	0.54	0.33
06-1060	154.97	156.73	1.76	0	0.01	0.009	0.46	0.93
06-1060	190.33	193.17	2.84	1	0.53	0.026	0.20	1.16
06-1060	256.39	257.7	1.31	2	0.19	0.065	0.87	8.80
06-1061	160.53	164.06	3.53	1	0.41	0.033	0.79	1.65
06-1061	197.92	199.72	1.8	2	0.67	0.079	0.65	6.35
06-1062	107.05	109.31	2.26	0	0.09	0.016	1.38	1.98
06-1062	127.77	132.01	4.24	1	0.05	0.012	0.57	3.53
06-1062	166.95	169.24	2.29	2	0.31	0.060	1.50	10.46
06-1063	86.7	88.99	2.29	0	0.01	0.003	0.04	0.89
06-1063	93.95	96.66	2.71	1	0.05	0.008	1.09	4.24
06-1063	102.4	108.7	6.3	2	0.16	0.020	0.39	3.48
06-1064	100.29	101.42	1.13	0a	0.01	0.005	0.17	3.59
06-1064	103.7	105.45	1.75	0	0.02	0.008	1.81	3.13
06-1064	125.27	126.78	1.51	1	0.14	0.022	0.34	1.63
06-1065	85.63	86.97	1.34	0a	0.01	0.005	0.06	2.59
06-1065	93.5	96.3	2.8	0	0.01	0.007	1.20	3.45
06-1065	115.48	117.15	1.67	1	0.25	0.012	0.95	5.63
06-1065	142.21	145.58	3.37	2	3.24	0.045	0.65	5.23
06-1066	42.7	47.64	4.94	0	0.01	0.005	0.34	1.78
06-1066	80.67	85.28	4.61	1	2.07	0.057	0.23	0.83
06-1066	89.42	92.8	3.38	2	0.12	0.021	0.42	2.84
06-1066	117.67	121.84	4.17	3	0.32	0.031	0.48	2.85
06-1067	53.3	54.45	1.15	0a	0.01	0.002	0.19	0.85
06-1067	60.1	61.45	1.35	0	0.33	0.023	0.37	1.82
06-1067	66.7	69	2.3	1	0.00	0.001	0.06	2.01
06-1067	87	91	4	2	0.49	0.024	2.21	5.12
06-1067	102.9	109.7	6.8	3	0.22	0.040	0.70	5.99
06-1068	43	45.5	2.5	0	0.02	0.005	0.15	0.79
06-1068	46.62	50.8	4.18	1	0.01	0.011	0.59	0.72
06-1068	54.3	58.75	4.45	2	0.43	0.016	0.63	5.91
06-1069	42.3	45.4	3.1	0	0.01	0.003	0.07	0.60
06-1069	46.6	52.3	5.7	1	0.01	0.006	0.29	0.59
06-1069	71.7	73.75	2.05	2	0.56	0.027	1.16	2.16
06-1070	44.6	47.16	2.56	0a	0.01	0.007	0.32	2.14
06-1070	50.55	52.55	2	0	0.01	0.008	0.75	1.61
06-1070	98.5	102.57	4.07	1	0.15	0.034	0.44	1.42
06-1070	141.5	147.5	6	2	0.57	0.038	0.19	0.99
06-1071	109	116.2	7.2	1	0.03	0.021	0.97	2.79
06-1071	154.2	159.1	4.9	2	0.13	0.040	0.51	2.50
06-1071	195	202.45	7.45	3	0.03	0.027	0.41	2.31
06-1071	206.2	210.83	4.63	4	0.02	0.046	0.48	4.67
06-1072	115.9	127.7	11.8	1	0.02	0.008	0.29	1.34
06-1072	187.25	189.87	2.62	2	2.86	0.075	0.74	1.41

Hole ID	From	To	Thickness	Manto	Cu%	Co%	Zn%	Mn%
06-1072	248.9	255.9	7	3	1.81	0.078	0.57	1.95
06-1072	258.95	272.95	14	4	0.06	0.051	0.41	5.78
06-1073	121.85	122.93	1.08	0a	0.01	0.006	0.63	1.96
06-1073	128.14	129.39	1.25	0	0.01	0.009	0.88	0.73
06-1073	161.59	163.45	1.86	1	5.89	0.053	0.19	1.18
06-1073	184.15	184.35	0.2	2	0.12	0.032	0.53	6.32
06-1073	204.35	204.82	0.47	3a	0.01	0.025	0.36	0.50
06-1073	205.7	208.65	2.95	3	0.24	0.026	0.32	1.53
06-1073	210.3	214.8	4.5	4	0.03	0.042	0.30	3.66
06-1074	142.03	145	2.97	0a	0.00	0.003	0.03	0.19
06-1074	145.1	146.7	1.6	0	0.05	0.006	0.08	0.41
06-1074	161.55	163.7	2.15	1	0.01	0.004	0.04	0.28
06-1074	187.9	192.07	4.17	2	0.13	0.043	0.39	5.65
06-1074	210.9	214.57	3.67	3	1.73	0.157	0.29	4.64
06-1075	123.4	126.23	2.83	0	0.01	0.008	0.57	0.81
06-1075	156.5	159.4	2.9	1	0.18	0.020	0.27	1.60
06-1076	13.85	15.88	2.03	1	0.11	0.021	0.83	2.21
06-1076	80.4	83.05	2.65	2	0.11	0.032	0.69	2.01
06-1076	131.57	136.1	4.53	3	0.19	0.018	0.24	0.26
06-1077	22	30.75	8.75	1	0.04	0.012	0.92	3.41
06-1077	88.95	92.7	3.75	2	0.15	0.015	0.28	0.91
06-1077	144.78	148.5	3.72	3	0.55	0.010	0.20	0.31
06-1078	177.5	179.26	1.76	1a	0.01	0.008	0.45	0.80
06-1078	193.87	195.45	1.58	1	3.70	0.165	0.85	3.50
06-1079	133.25	136.1	2.85	1	0.83	0.071	0.37	1.26
06-1079	202.35	204.47	2.12	2	0.28	0.092	0.59	10.21
06-1079	218.45	220.8	2.35	3	1.74	0.042	0.21	1.94
06-1080	146.9	148.35	1.45	0a	0.02	0.006	0.25	0.95
06-1080	181.9	185.5	3.6	1	0.14	0.020	0.37	1.38
06-1080	243.7	247.2	3.5	3	0.63	0.020	0.21	0.77
06-1081	82.93	87.2	4.27	0	0.02	0.007	0.23	0.88
06-1081	122.17	124.64	2.47	1	1.22	0.087	0.60	3.22
06-1081	187.25	189.63	2.38	2	0.31	0.088	0.59	9.57
06-1082	83.36	87.25	3.89	0	0.01	0.006	0.35	0.69
06-1082	120.73	123.3	2.57	1	1.22	0.099	1.01	4.33
06-1082	185.6	191.15	5.55	2	0.02	0.021	0.36	3.06

12 SAMPLING METHODS AND APPROACH

All samples used in the resource estimation are from diamond drill holes. Down hole sample intervals vary as intervals were selected on the basis of geology. A total of 12,675 samples have been collected from the mineralized manto units and the mean sample interval was 0.95m.

Historical drilling activities prior to 2004 have not been observed. Sampling procedures (adopted by Curator) comprised:

- Core was transported from the drill site by either helpers or geologists to company warehouses in Santa Rosalia, where the boxes were labelled and core recoveries calculated.
- The core was then logged by a company geologist who simultaneously marked out all sample intervals.
- Core was split with a mechanical splitter (or a knife in poorly consolidated material) by a trained local helper.
- Core logging was based on geological intervals with detailed written descriptions for each interval. Mineralogical, structural and textural information was not recorded in dedicated fields, making it difficult to extract anything other than summary data from the logs. Logged geology intervals did not always correspond to assay intervals and in many cases overlaps exist between geology and assay intervals.

Drilling rigs were inspected during the recent drilling program in May 2006.

Procedures for drill site supervision and handling of core for this program were as follows:

- Drilling Supervision. At the time of manto intersection geologists were on-site to take notes on core recovery and the conditions of the core.
- Transport of the core. On completion of a hole the core boxes with lids are loaded into a pick up and secured with a rope to avoid core losses during transport.
- Layout of core for logging. In the core shack the core is placed in sequential order on benches. The benches are designed for ease of use and are well lit.
- Calculation of core recovery and RQD. Measurement of the length of recovered core between each drilling interval is used to calculate the percent recovery of the core. The total length of core that occurs in individual core pieces greater than 10 centimetres in length is measured in order to calculate the RQD.
- Logging procedures. The core is logged based on geological criteria such as lithology, formation number and members, paying special attention to subdivisions in the mineralized zones. Geologists complete a fully descriptive written log of each interval. This descriptive log is then computer coded for entry into a geological database. Each sample interval corresponds exactly to an assay interval.

- Sample selection. Sample selection is carried out in the mineralized manto intervals with a black marker pen indicating where each sample initiates and ends and showing arrows that indicate the interval to be sampled. The length of the samples varies depending on the type of material being sampled. The maximum length for individual samples does not exceed 1.5 m.
- Numbering of samples. The numbering sequence used for sampling is continuous and ascending with respect to depth in a specific drill hole. Numbering control is recorded in a sampling book, sample numbers are also recorded on the full log and the computer coded log sheets. Assay Quality Control items such as standards, blanks, and pulp duplicates are inserted at this stage.
- Sampling of the Core. Splitting of the core is completed using a mechanical splitter. One half of each sample interval is sent to the laboratory for assaying the other half is retained in the original core box.

The use of a saw is generally better than mechanical splitting in that it cuts through grains and particles, so that some material is collected in the sample whilst the rest is retained in the reject half. Where a splitter is used the split surface is typically uneven, with breaks that do not exactly honour the desired half core proportion. Particles along a break can be left intact and are either taken 100% into the sample or left behind in the reject. This can lead to increased sampling error and variability in the resulting assay data. In extreme circumstances this may result in sampling induced bias in the assay results.

During two site visits in 2006 Dr B Yeo was able to observe the core splitting process using the mechanical splitter as well as similar material being 'cut' using a core saw as described above.

It was concluded that the use of the core saw was inappropriate because typical manto ores are soft and friable. The cut core disintegrated immediately and created greater problems with regard to collection of representative material than caused by the mechanical splitter.

The mechanical splitter was quick and easy to use but comprised a very short cutting blade that required breaking of the core prior to splitting, and during use the material often crumbled making it difficult to take an accurate half core sample (see Figure 10).

Figure 10: Mechanical Core Splitter

The process of allocating material to the assay sample or to the retained portion at times appeared confusing and a little arbitrary. As a result a new mechanical splitter was designed by H&S and MMB personnel.

This initially comprised a wooden holder, fixed to a work bench, with a closing lid. An entire core box length of core was then placed in the holder, lid closed and a metal blade was guided through a slot in the lid. This blade was then hammered through the entire core interval. The blasé was left in place whilst all material from one side of the blade was removed to a sample bag, either by hand picking large fragments or brushing fines. This produced a very clean split, with all material from one half of core being easily collected. The concept can be appreciated by reference to Figure 11.

Figure 11: Prototype Mechanical Splitter

A more robust metal version of the device was subsequently built (see Figure 12) that was similar in concept but had the blade attached to a screw similar to the original splitter, rather than requiring hammering to break the core.

Figure 12: Final Mechanical Splitter Design

12.1 CORE RECOVERY

Core recovery from the historical drill holes was measured for each retrieved core run. Measured recoveries range from 82-90%, with a mean of 83% (Table 7). Where recovery was less than 100% no attempt was made to specifically identify location of lost core within the recovered interval.

Table 7: Diamond Core Recovery – Historical

Manto	Intervals	Recovery %
0	49	90
1	128	87
2	381	83
3	772	82
3a	564	84
3aa	50	88
4	141	84
Total	2,085	83%

Core Recovery for the current drill program (DDH928-1082) shows better overall values with a mean recovery of 94%. See Table 8 below.

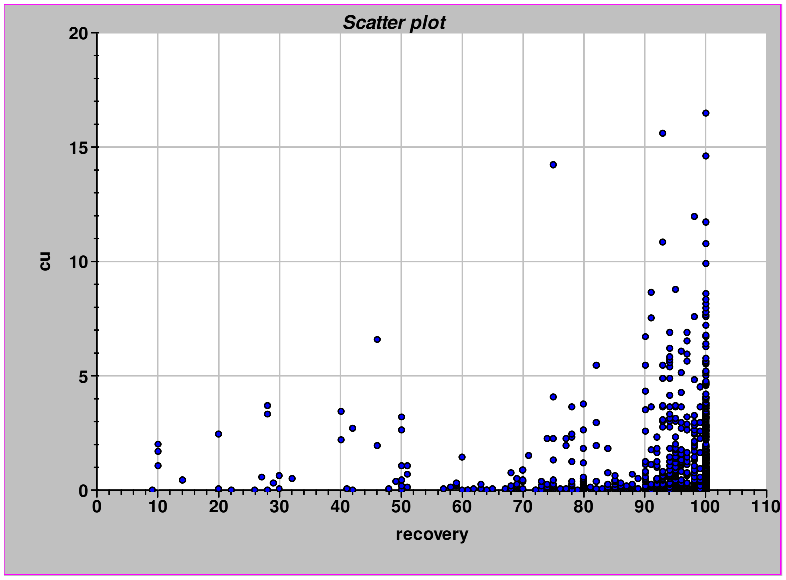
Table 8: Diamond Core Recovery - Current Programs (DDH928-1082)

Manto	Intervals	Recovery %
0	270	94
1	356	95
2	388	96
3	346	93
3a	75	95
3aa		
4	313	95
total	1,748	94%

For the historical holes, the core run intervals over which recoveries were determined do not correspond to specific assay intervals so it is not possible to evaluate whether a relationship, of any sort, exists between recovery and Cu or Co grade.

For the current drill holes recoveries have been determined for each sample interval. Figure 13 below shows core recovery against copper grade. There is no strong correlation between recovery and grade, although lower recoveries tend to be lower grade suggesting possible loss of friable mineralization.

Figure 13: Core Recovery vs. Copper Grade- Current Drill Program



Data Statistics

variable:	recovery	cu
weight by:	--	--
mean:	94.463	0.690
varnc:	139.449	2.464
coefvrn:	0.125	2.274
min:	9.000	0.002
q1:	94.000	0.018
median:	100.000	0.082
q3:	100.000	0.525
max:	100.000	16.500
iqr:	6.000	0.507

covarnc:	-0.167
Pearson:	-0.009
Spearman:	-0.016
no. data:	1748

(data set at full limits)

12.2 DENSITY DETERMINATIONS

Data and methodology used to determine density were taken from an internal company report (Felix 1996). Measurements of specific gravity were taken during the exploration drilling campaigns of 1995 and 1996. A total of 2,112 measurements were obtained of which 418 volumes were determined by a dimensional method (discussed below) and 1,694 by water immersion.

The dimensional method was used to obtain volume determinations in 1995 and the first weeks of 1996, after which only the water immersion method was used. Approximately a hundred samples were measured for specific gravity using both methods. The difference between the results averaged 0.4%. The samples were taken from both mantos (995 determinations) and non mineralized rocks (1,117 determinations).

Densities were determined on wet or undried samples. Dry density for mineralized Manto samples was calculated from the wet density by factoring the density value by the proportion of water content lost during sample drying. To determine water content, samples were dried at 110º C for a minimum of 6 hours. Water content was calculated, as the weight loss on drying, as a percentage of the original sample weight, by SGS-XRAL Laboratories in Hermosillo.

12.2.1 DIMENSIONAL (OR CALLIPER) METHOD

The core sample was cut with a saw perpendicular to its axis; samples were nominally 30cm in length. A tape was used to measure the diameter of the two ends, with two measurements taken at each end with the average of the four taken to establish the diameter of the sample. In the same way, the length of the core was measured with the same tape in two directions and the average taken as the length. The volume is calculated using the formula for the volume of a cylinder and bulk density determined dividing the weight by the volume after adjusting for core recovery.

12.2.2 IMMERSION METHOD

The immersion is based on the Archimedes Principle, which states that the density of an object is equal to the weight divided by the difference between the weight in air and the weight in water.

In the laboratory, the samples were sprayed with lacquer to form a thin and uniform coat. The lacquer is used to prevent water being absorbed by the sample. The sprayed sample was allowed to dry a few minutes and was weighed in normal (in air) conditions and then weighed submerged in water.

12.2.3 RESULTS

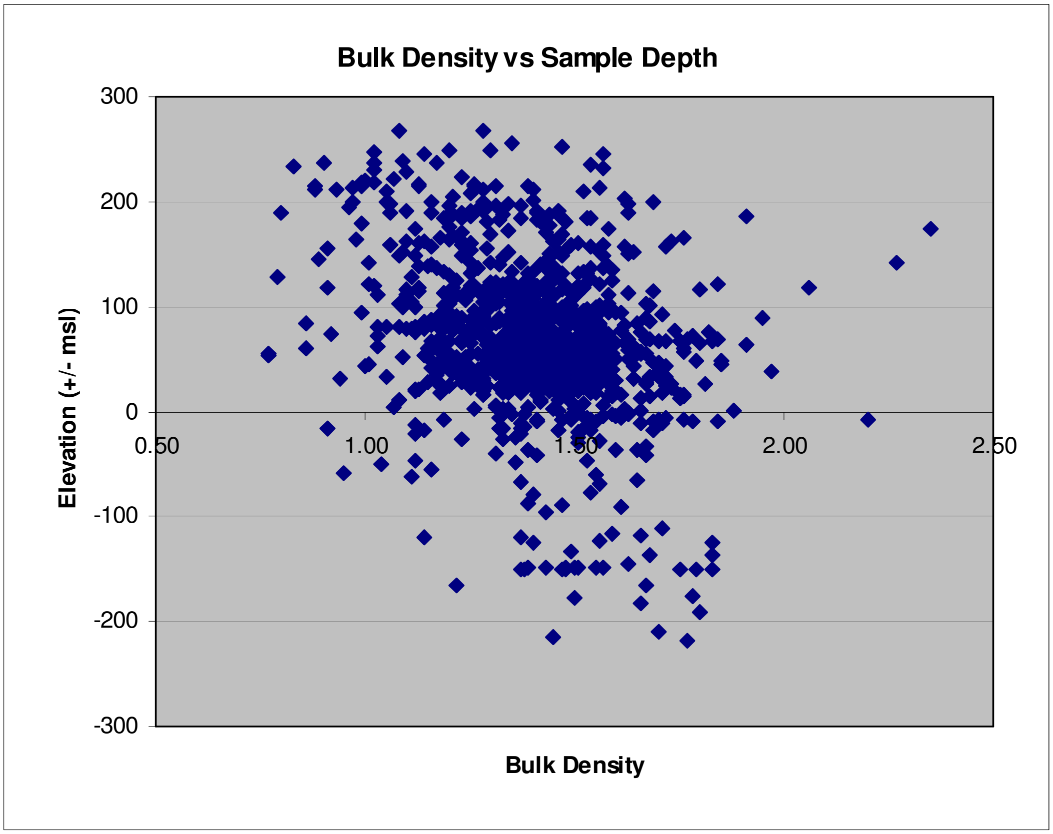
For the purposes of resource calculations, a global in situ dry bulk density of 1.41 tonnes per cubic meter has been used for manto material, based on an average calculated from 995 wet density measurements and the corresponding analysed water contents of the samples (Table 9).

Although there is some indication of increasing bulk density with sample depth, a graph of bulk density vs. depth (Figure 14) shows only a weak correlation.

Table 9: Bulk Density Summary by Manto

Manto	Wet (Natural)			Dry	
	Bulk Density	Data	H2O%	Bulk Density	Data
Tbcu0	1.84	53	26.75	1.36	53
Tbcu1	1.88	86	26.15	1.40	86
Tbcu1A	1.90	2	25.53	1.41	2
Tbcu2	1.87	136	26.93	1.38	136
Tbcu2A	1.86	11	26.86	1.36	11
Tbcu3	1.89	543	25.71	1.41	489
Tbcu3A	1.90	133	26.15	1.41	128
Tbcu3AA	1.88	9	26.89	1.38	9
Tbcu4	1.91	78	24.19	1.46	78
Tbcu4A	1.93	3	25.48	1.44	3
All Mantos	1.89	1,054	25.93	1.41	995

Figure 14: Bulk Density vs. Sample Depth



13 SAMPLE PREPARATION, ANALYSIS AND SECURITY

13.1 HISTORICAL DRILL HOLES – PRE 2004

Prior to 1997, assay samples were sent to the SGS-XRAL laboratory in Hermosillo where they were dried, crushed and pulverised. Analysis for Cu, Co, Zn, Fe and Mn was carried out at the same laboratory, with a perchloric acid digest and AAS finish. Samples that reported grades greater than 1% of Cu, Co or Zn were re-assayed using a method more appropriate for higher grade samples.

From 1997 onwards sample preparation was carried out at Chemex facilities in Hermosillo with pulps being forwarded to the Chemex laboratory in Vancouver, Canada for analysis. A four acid digest with AAS finish was used for Cu, Co, Zn, Fe and Mn.

The reject and remaining pulp material were returned to Santa Rosalia where they are securely and systematically stored in warehouses.

The reason for the change in assaying laboratory and technique was due to the identification of a systematic under-reporting of, primarily, Co by SGS-XRAL. The extent of the analytical problems and re medial measures taken are discussed in more detail in section 14.1 below.

All samples were stored, prior to shipping, in one of three locked warehouses in Santa Rosalia. When shipped, the samples were taken to the ferry by company personnel where they were put on the ferry and shipped across the Gulf of California to Guaymas. The representatives of the laboratory picked the samples up and delivered them directly to the laboratory in Hermosillo.

13.2 CURRENT DRILL HOLES

The samples for assaying are placed into 30 kilogram sacks and kept in locked premises until such time as they are transported to the laboratory. A company vehicle is used to transport the samples from Santa Rosalia to Guaymas via a ferry and then driven directly to Hermosillo, Sonora. The samples are driven and accompanied by a company employee.

Remaining core is stored in boxes in an underground storage area that is secured by a locked metal door.

The sample shipment is delivered to ALS Chemex located at:

Ignacio Salazar 688 Local 5, Fracc. Los Viñedos
C.P. 83147
Hermosillo, Sonora, México.

The sample preparation protocol used is as follows:

- Samples are oven dried at 110 degrees until dry
- Samples are fine crushed to >70% <2 mm (TM Rhino or Terminator crusher)
- Crushed sample is split using a riffle (Jones) splitter to 250 g
- 250 g of crushed material is pulverised (Labtech LM2)>85% <75 µm.

After sample preparation pulps are sent from Hermosillo to ALS Chemex in North Vancouver for assaying.

The assay method currently used for Copper, Cobalt, Zinc and Manganese is ME-ICP61a (four acid "near-total" digestion). Samples reporting high grades for Cu, Co, Zn or Mn are re-assayed for those elements by method AA62.

Total sulphur has been assayed for using ME-ICP61a with sulphide sulphur determined by either S-IR08, or S-IR07. Moisture content was determined by OA-GRA05s.

14 DATA VERIFICATION

Data verification can be considered as having four separate components:

- Are the assay results reported for each sample accurate?
- Are the samples, used in assaying process, representative of the sample interval?
- Are the reported assay values, which are identified by a unique sample number, assigned to the correct down hole sample intervals.
- Are the drill hole locations known accurately, and locations correctly entered in the database.

To evaluate accuracy and representativity, a number of quality control samples have to be inserted into the sample stream. These samples include Standard Reference Materials (SRMs or "standards") of known grade, blank samples with no grade, and possibly field duplicate samples. Field duplicates of diamond core samples should be the un-sampled half of core retained after the original round of sampling. To be most effective, the Quality Control samples need to be anonymous to the assaying laboratory.

14.1 HISTORICAL DATA – PRE 2004

14.1.1 PROBLEMS WITH THE SGS-XRAL ASSAY RESULTS

During the original round of assaying by SGS-XRAL, no SRMs or field duplicate samples were used. Some internal repeat assays were reported by the laboratory and on quick visual examination these appeared to be adequate, but no statistical analysis was completed to support this.

At a late stage in the process, a group of samples were retrieved and sent to an independent laboratory (Bondar-Clegg, Vancouver) for check analysis. The results of this check program revealed some considerable departures from the original values, though no standards were included with these samples, so the same samples were forwarded to Chemex Laboratories, also in Vancouver. The results of these second repeat check assays were again considerably different to both the original SGS-XRAL and Bondar-Clegg check assays, such that sufficient doubt was generated as to the validity of the entire assay database at Boléo.

Consequently a decision was taken by the company (International Curator Resources Ltd, "Curator") to review the entire assaying process used for the Boléo samples.

14.1.2 SUMMARY OF 1997 PROCESS REVIEW AND RE-ASSAYING PROGRAM

The review was carried out on behalf of Curator by consultants G. Peatfield and B. Smee. The process and subsequent re-assay program is reported in detail (Peatfield & Smee 1997, Peatfield 1997, Peatfield 1998). These reports are available and a limited synopsis is included in Sections 14.2.1 to 14.2.6.

Several problems were identified with respect to the original SGS database. These were:

- Despite repeated assurances from SGS that the perchloric acid digest would give a total metal extraction, data from check assaying indicated otherwise.
- Above 4%, Cu SGS assay values showed very poor correspondence with check assays from other laboratories. It was thought that this was to do with a dilution step in the process but SGS failed to provide a rational explanation.
- SGS reporting left much to be desired, with issues such as changes in reporting format and numerous data errors.
- Check assaying strongly suggested that Co values from SGS were reported systematically low, possibly by as much as 15%.

PREPARATION OF ASSAY STANDARDS

The use of standards to monitor assay accuracy was decided on immediately. It was also deemed essential that these standards should be prepared from Boléo material so that the matrix of the standards matched that of the samples.

Initially two standards (Interim Standards) were prepared from Boléo material (Boléo I, II). These were prepared at the SGS laboratory. Sub-samples were dispatched to 7 laboratories for round-robin analysis. The material was subjected to a 4 acid digest (nitric, hydrochloric, perchloric and hydrochloric), with an AAS finish. From the data received from each laboratory the mean and relative standard deviations were determined. The interim standards were used only as a stop-gap measure whilst the full review of the assay process and the preparation of formal standards were completed.

The formal standards (Boléo III, IV, V) were prepared at the Colorado Minerals Research Institute and CDN Resource Laboratories, Burnaby B.C. Canada. Composites of varying grade were combined to form three single 25kg samples. Samples were again dispatched to several laboratories in the USA, Canada and to SGS in Mexico for round-robin analysis. The same 4 acid digest method was used. The acceptable grade and limits for each standard (Peatfield 1997) are shown in Table 10.

Table 10: Boléo Assay Standards

	Copper			Cobalt			Zinc		
Standard	**mean**	**+2sd**	**-2sd**	**mean**	**+2sd**	**-2sd**	**mean**	**+2sd**	**-2sd**
Interim Standards									
Boléo I	1.546	1.682	1.410	0.0595	0.0640	0.0550	0.34	0.329	0.278
Boléo II	3.813	4.159	3.466	0.0527	0.5677	0.4377	0.355	0.389	0.321
Formal Standards									
Boléo III	0.514	0.555	0.472	0.0520	0.0630	0.0490	0.322	0.338	0.306
Boléo IV	1.124	1.202	1.047	0.0934	0.1082	0.0786	0.387	0.415	0.358
Boléo V	7.405	7.900	6.910	0.0986	0.1146	0.0827	0.459	0.459	0.434

Analytical Method and Assaying Laboratory

To determine the most appropriate analytical method to use, 12 samples were selected, prepared in replicate pulps and dispatched to several laboratories where they were assayed using four different techniques:

- aqua-regia digest
- sodium peroxide fusion
- perchloric acid/aqua-regia digest
- four acid digest (nitric, hydrochloric, perchloric, hydrofluoric).

The sodium fusion method tended to report substantially lower grades indicating less complete dissolution. The four acid method gave the most consistent results although slightly lower than the aqua-regia or perchloric/aqua-regia digests.

The four acid digest was chosen as the analytical technique and Chemex was chosen as the assaying laboratory. A second laboratory, Mineral Environments Laboratories, was chosen for check assaying.

Re-Assay Program

All pulp and reject material held by SGS in Hermosillo were returned to Santa Rosalia. Samples were selected for re-assay on the premise that any sample obviously or likely to be used in resource estimations would be re-assayed. When in doubt samples were included rather than omitted.

Thirty gram sub-samples of existing pulps were prepared and a completely new sample number sequence applied. Standards, blanks and duplicate pulps were inserted into the sample stream in sequence so that for every 40 samples sent to Chemex would include, two standards, two blanks and two duplicate pulps. In addition Chemex inserted two of their own standards and a sample blank. In all about 6,800 samples were re-assayed. To monitor quality 1,200 internal quality control samples (standards, blanks and duplicates) were inserted by Curator and an additional 800 laboratory samples were inserted by Chemex.

The results of the internal standards and blanks were monitored as data was received by means of control charts, which show the acceptable value and upper/lower limits (± 2s.d.). No concerning results or trends were identified. Blanks sample results were monitored by eye and no obvious contamination was seen. Chemex standards were monitored in a similar way.

Sample duplicate data was analysed using a mathematical technique (Thompson and Howarth 1978) to determine analytical precision. The results indicated that at grades likely to be mined, levels of precision were very good (± <5%) for both Cu and Co.

Second Laboratory Check Assaying

A total of 441 duplicate pulps were sent to a second laboratory (Mineral Environments Laboratories, “Min-En”) for check assays. Typically results were similar, there was however a clear laboratory bias, with Min-En results reporting lower than Chemex.

Similar results were seen in the data reported for the standards returned from Min-En. It was concluded that the check laboratory did not match their performance in the original round-robin tests.

Core Duplicate Assaying

As part of the re-assaying program, pulp duplicates were routinely inserted into the sample stream. However, because they are from the same original sample, they do not provide a means for determining overall precision resulting from all steps in the process, from sampling, sample preparation and assaying. To do this the remaining half-core has to be sampled and subjected to the same sample preparation and assaying regime.

One hundred samples were selected for core duplicate assaying. For each of these samples the remaining half-core was collected from the core tray and sent for assay. Half core was crushed and split into two equal samples. From each of these a 250 g sub-sample was taken and fine pulverized and a final 30 g sub-sample taken from each and sent to Chemex for assaying. As a result it was possible to compare the original assay against two identical assays of the remaining core, referred to as ‘A’ and ‘B’ core duplicates.

In addition, duplicate samples were also taken from the coarse residue of the original assay samples retained by the laboratory.

Standards were inserted into the sample stream at a rate of 1 in 20, as done previously.

The results showed, as expected, lower precision between the core duplicates and original samples than for the coarse residue duplicates and the original samples. Surprisingly though, there were also noticeable differences between the calculated precision from the ‘A’ and ‘B’ core duplicates. This lead to a decision to include core duplicate sampling programs as part of all future drilling and assaying programs.

Average grades from both the ‘A’ and ‘B’ core duplicates were lower than the original assays by ~10% for Cu, 5% for Co and 2% for Zn. No explanation was given for this difference.

Table 11 summarises the results of the ‘A’ and ‘B’ core duplicate sampling program.

Subsequently a further 55 core duplicate samples were assayed (Table 12). Levels of precision clearly improve from core duplicates to coarse residue duplicates to pulp duplicates and, as in the first program, the core duplicate assays show lower average grades than the original assays, although in the second program this difference is less.

Table 11: Boléo Core Duplicate Results – 1st Program

Core Duplicates	Range (%)	Ave. Precision	Original Assay Ave.	Dupl. Core Assay Ave.	% Diff.
Copper					
'A'	0.10 – 4.78	25.9	1.001	0.895.	-10.6
'B'	0.10 – 4.78	26.8	1.001	0.903	-9.8
Cobalt					
'A'	0.010 – 0.0525	20.2	0.063	0.060	-4.8
'B'	0.010 – 0.0525	20.1	0.063	0.060	-4.8
Zinc					
'A'	0.2 – 6.69	23.9	0.751	0.736	-2.0
'B'	0.2 – 6.69	14.2	0.751	0.736	-1.5

One reason for the improved result in the second program may be due to the fact that the duplicates and original samples were assayed in the same batch, whilst in the first program duplicates were submitted in two batches, one at a later date than the other.

Therefore any problem unique to a batch or prevalent at a laboratory for a limited time can impact the entire program. The average grades of both the coarse residue and pulp duplicates are almost identical to the original samples.

Table 12: Boléo Core Duplicate Results – 2nd Program

Duplicate Type	Range (%)	Ave. Precision	Original Assay Ave.	Dupl. Assay Ave.	% Diff.
Copper					
Core	0.10 – 4.38	14.4	1.020	0.957.	-6.2
Coarse Residue	0.10 – 3.34	5.7	0.816	0.811	-0.6
Pulp	0.10 – 3.72	1.7	0.595	0.587	-1.3
Cobalt					
Core	0.010 – 0.206	22.3	0.045	0.044	-2.2
Coarse Residue	0.010 – 0.335	8.8	0.051	0.051	0.0
Pulp	0.010 – 0.187	3.7	0.053	0.052	-1.9
Zinc					
Core	0.20 – 1.30	12.9	0.398	0.386	-3.0
Coarse Residue	0.20 – 0.65	1.5	0.326	0.325	-0.3
Pulp	0.20 – 1.48	1.8	0.441	0.440	-0.2

Conclusions Derived from the Re-Assay Program

The conclusions arrived at by Peatfield & Smee were:

- The initial assay method used by SGS was unreliable and uncontrolled, and therefore unacceptable.

- Tests carried out showed that a multi acid digestion provided the most consistent assay results.
- Round-Robin assaying showed the original laboratory had difficulty in generating statistically acceptable results.
- The re-assay program was completed under controlled QC conditions, and provided revised database assay information with acceptable accuracy.
- Subsequent core duplicate sampling produced relatively high but acceptable levels of precision.
- Precision based on reject duplicates showed little difference to pulp duplicate precision, indicating error attributable to sample preparation was minimal.
- The systematic decrease in metal in the core duplicates is not readily explicable. The improvement between the 1^{st} and 2^{nd} core duplicate sampling program may be due to the fact that in the 2^{nd} program duplicates were assayed in the same batch as the original samples.
- Routine monitoring of internal and Chemex standards indicates that there are no recognizable problems with the accuracy of Cu, Co, Zn and Mn assays in the Boléo database.

14.1.3 COMMENTS ON ASSAY QUALITY CONTROL RESULTS

As a result of the problems identified in the original Boléo assay database, the results from the re-assay program have been subjected to a high level of scrutiny.

The preparation of matrix matched standards is commendable and is the best means available to monitor and ensure assay accuracy.

The control charts for each standard (Peatfield & Smee 1997) show that although the Cu results do fall within the acceptable limits, they are mostly higher than the accepted value, rather then being scattered above and below. Conversely, the standards reported by the second laboratory were also within the acceptable limits but systematically low. These patterns are consistent, in all cases, with the Chemex value for each standard reported in the round-robin assaying relative to the accepted value (the mean of 7 participating laboratories). The implication of these results is that the Boléo assay data may be reported marginally high, in the order of 2% to 4%.

However, H&S consider that a more rigorous evaluation of the round-robin results used in determining the standard values standards should have been applied. Of the seven participating laboratories, results from six were used, in most cases, to determine the true values (Peatfield & Smee 1997). H&S believe that using results from fewer laboratories (3 or 4) of superior quality and consistency is best in determining the recommended values.

Figure 15 shows graphically the Cu data reported by each laboratory used for Boléo V, whilst Figure 16 shows the data used by H&S. A similar exercise was completed copper and cobalt data for Boléo III and IV the results are compared in Table 13.

Figure 15: Boléo Cu, Round-Robin Cu Assays used by Peatfield & Smee

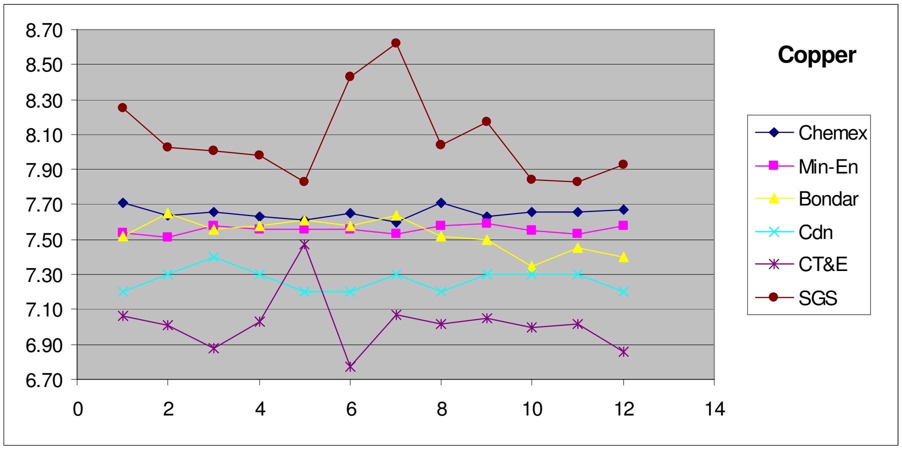


Figure 16: Boléo Cu, Round-Robin Cu Assays used by H&S

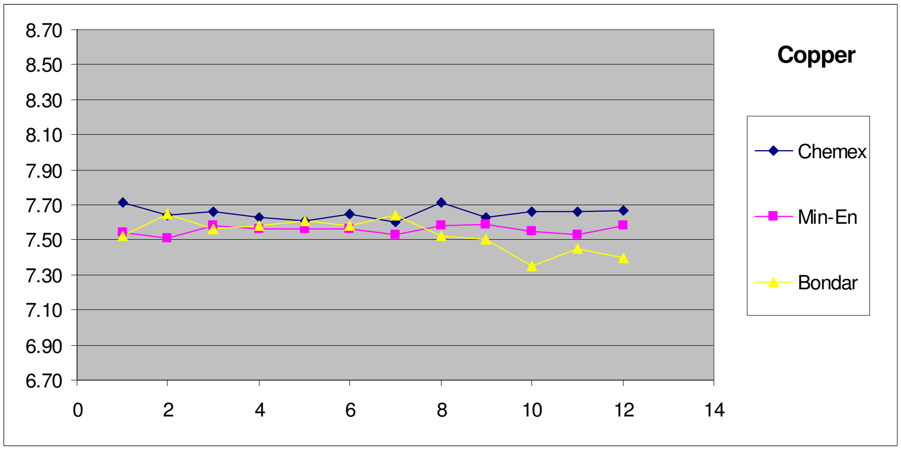


Table 13: Boléo Formal Standards – Revised Values

	Copper			Cobalt		
Standard	mean	+2sd	-2sd	mean	+2sd	-2sd
Formal Standards – Peatfield & Smee (1997)						
Boléo III	0.514	0.555	0.472	0.0520	0.0630	0.0490
Boléo IV	1.124	1.202	1.047	0.0934	0.1082	0.0786
Boléo V	7.405	7.900	6.910	0.0986	0.1146	0.0827
Revised Formal Standards – H&S						
Boléo III	0.525	0.530	0.519	0.0511	0.0567	0.0455
Boléo IV	1.135	1.159	1.111	0.0929	0.0988	0.0871
Boléo V	7.579	7.709	7.450	0.0986	0.1045	0.0927

Copper values determined by H&S are slightly higher than Peatfield & Smee, whilst cobalt values are slightly lower. If revised copper standard values were used, the potential assay bias of 2% to 4% would be reduced to less than 2%.

The core duplicate sampling programs show a consistent difference in the means of the original assays and the duplicate samples, by as much as 10% for copper. This may be in part attributable to batch issues but this has not been confirmed.

Another aspect that can cause differences between reported grades is the method and completeness of the collection of material from the core tray. It is unlikely that the fines in the bottom of a core tray are identical in make up as the larger segments of core. If, for example, fine grained chalcocite is dislodged, through handing of core, and collects in the tray it needs to be carefully collected, failure to do this may result in understating copper grade.

The use of a soft brush is often needed to ensure that all the fines are collected from the bottom of the tray. Unfortunately, sample collection was not observed. The 55 duplicate core samples from the second program were dispatched to the laboratory at the same time as the original samples, suggesting that they were bagged immediately on splitting and not returned to the core tray to be re-sampled at a later date. Yet results for these duplicate samples were still lower than for the original samples.

14.1.4 DATABASE VERIFICATION

To ensure that the database is accurate, in other words, that assays are assigned to the correct sampled interval, an audit of original assay certificates against the database files was carried out.

The assay data is kept in 10 Microsoft Excel spread sheets (a separate file for each hundred holes). Two holes were checked from each file (Table 14), with Cu, Co, and Zn assays checked. Geological logs and summary logs contained in the same Excel files were not audited.

Table 14: Drill Holes Audited for Data Entry Errors

Hole ID	Assay Job Num.	Hole ID	Assay Job Num.
22	A 9631180	491	A 9627709
57	A 9629734	526	A 9627711
115	A 9634051	592	A 9628562
173	A 9626536	656	A 9626533
226	A 9626538	695	A 9629737
287	A 9626541	746	A 9717119
324	A 9626542	765	A 9718436
361	A 9626544	818	A 9720883
410C	A 9626545	888	A 9737165

No data entry errors were detected.

14.1.5 DRILL HOLE SURVEYING

Drill hole collars at Boléo have not been surveyed. In place of conventional surveying, high resolution Orthophotos have been acquired from which drill hole coordinates have been calculated. To assist in identifying drill holes in the photos each collar was marked with a large white cross, with the hole at the centre. It is estimated that the accuracy using this method is to sub meter.

The drill hole spacing, at its closest, is in the order of 150 m x 150 m so sub meter accuracy for the easting and northing coordinates is adequate. Mantos are generally only a few metres thick so accuracy of the elevation coordinate is more critical. A tolerance of ±0.65 m has been quoted for this dimension (Albinson pers. comm. 2004). This has not been verified. The potential implications of an error in this regard are not considered serious.

In an open-pit environment, mining is carried out with strict grade control practices that are used to identify ore and waste prior to mining. In an underground situation geological control, particularly the different lithologies that define the base of each Manto should be adequate to avoid losses due to uncertainty in the collar surveys.

All the holes at Boléo are vertical, which is appropriate. No down hole surveys were conducted.

No drill holes were intersected during the mining trial.

14.2 CURRENT PROGRAM

Assay standards, duplicate sampling, blank samples and check assaying have been used during the 2006 drilling programs to validate the assay results reported by ALS.

14.2.1 ASSAY STANDARDS

The three Boléo standards (Boléo III, IV, V) discussed above have been used again during the current assaying program. Results are shown in the figures below. The reported results are combined and presented as a percentage difference from the accepted value of each standard.

Figure 17: Boléo Assay Standards – Cu

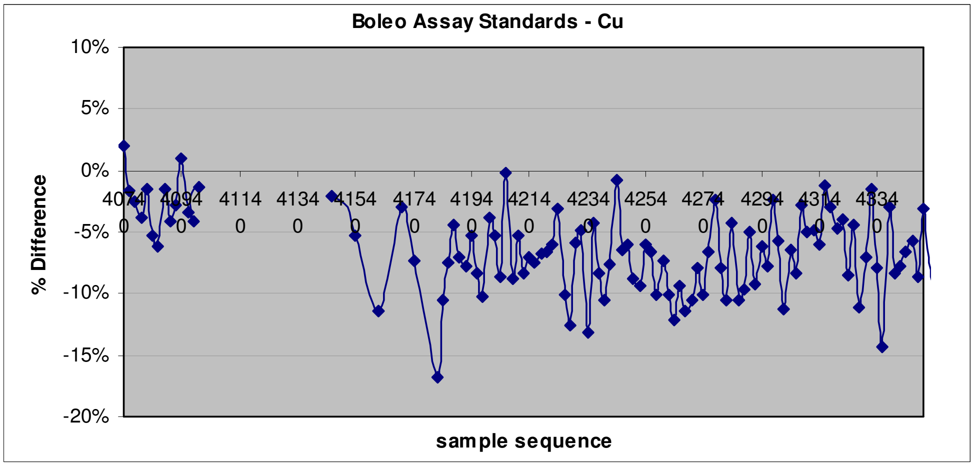


Figure 18: Boléo Assay Standards – Co

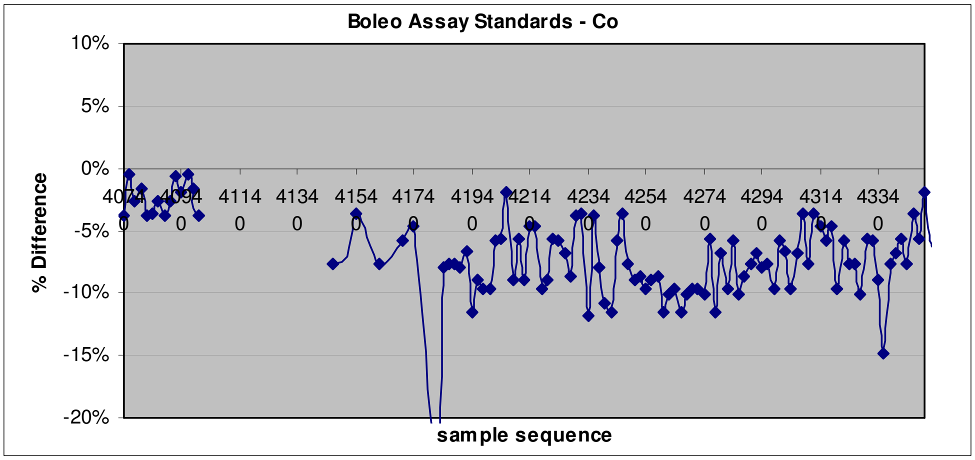
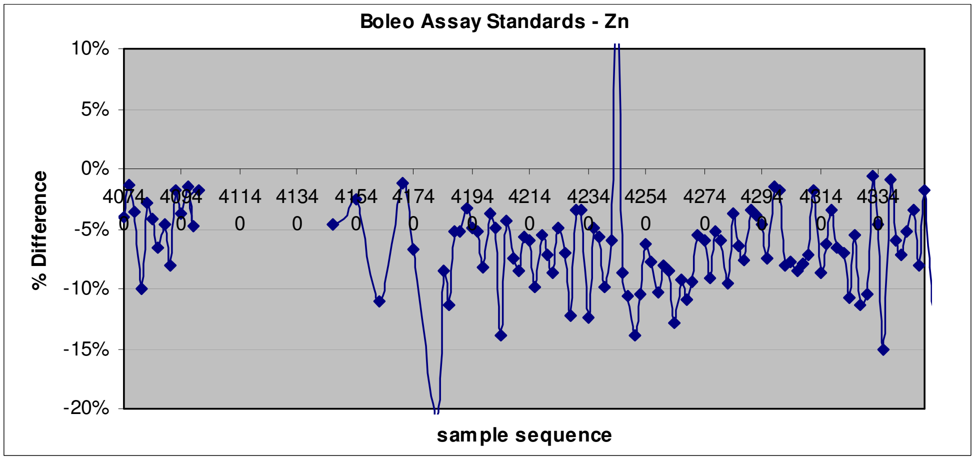


Figure 19: Boléo Assay Standards – Zn

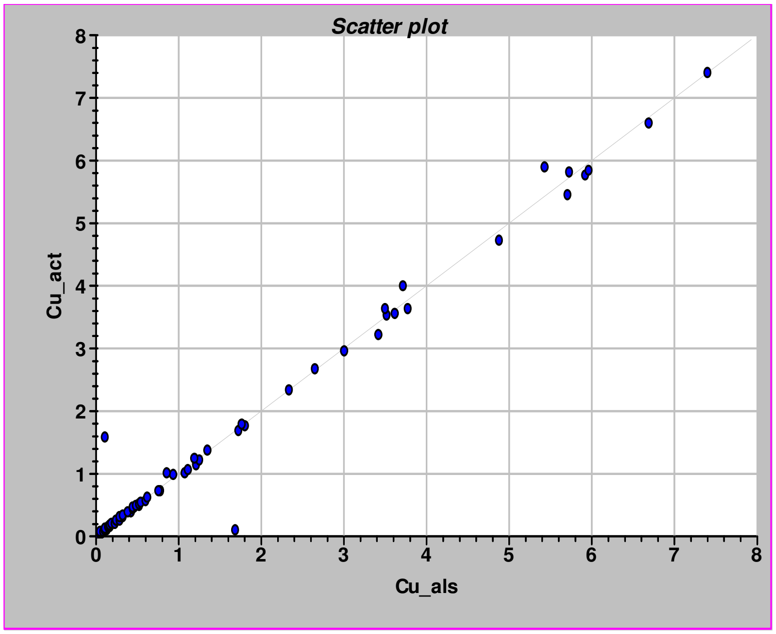


There is a persistent under-stating of standard grades by between 5% and 10% on the 2006 drill program. The reason for this is not at present well understood but the issue is currently under investigation and review.

14.2.2 CHECK ASSAYS

Check assays from ACTLabs in Phoenix however show very good correlation to the ALS-Chemex results (Figure 20). This suggests that the problem may be that the accepted values are not correct. Additional check assaying of the three Boléo standards is required to confirm this.

Figure 20: Boléo Check Assay Results – Cu

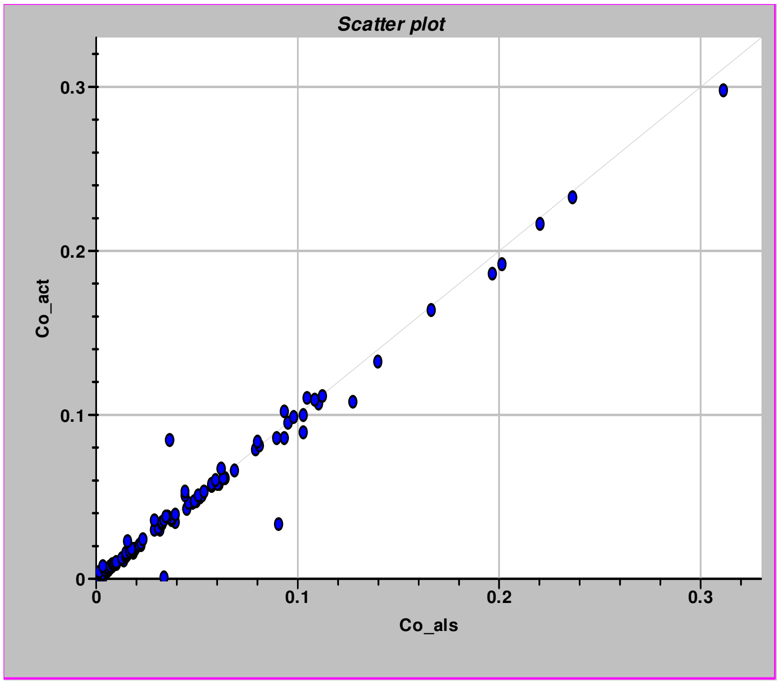


Data Statistics

variable:	Cu_als	Cu_act
weight by:	--	--
mean:	0.918	0.911
varnc:	2.712	2.693
coefvrn:	1.795	1.801
min:	0.003	0.005
q1:	0.029	0.030
median:	0.126	0.130
q3:	0.750	0.720
max:	7.410	7.400
iqr:	0.721	0.690
covarnc:	2.680	
Pearson:	0.992	
Spearman:	0.974	
no. data:	117	

(data set at full limits)

Figure 21: Boléo Check Assay Results – Co



Data Statistics

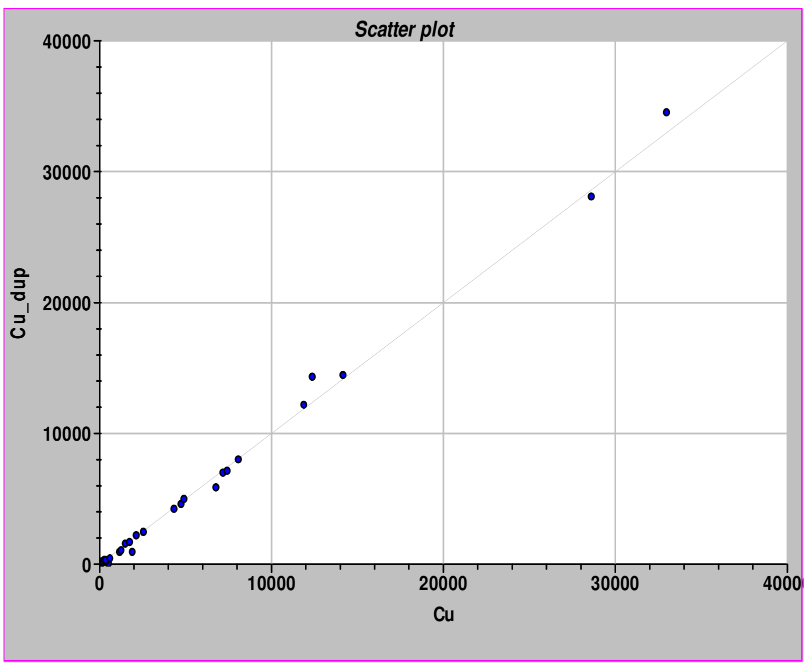
variable:	Co_als	Co_act
weight by :	--	--
mean:	0.045	0.045
v arnc:	0.003	0.003
coef v rn:	1.184	1.151
min:	0.001	0.001
q1:	0.008	0.009
median:	0.029	0.025
q3:	0.057	0.058
max:	0.311	0.299
iqr:	0.049	0.049
cov arnc:	0.003	
Pearson:	0.988	
Spearman:	0.971	
no. data:	117	

(data set at full limits)

14.2.3 DUPLICATE ASSAYS

Duplicate assays have been determined from crusher split reject material. Although the total number of duplicates is low (<50) the comparison is very good (Figures 22 and 23).

Figure 22: Boléo Duplicate Assay Results – Cu

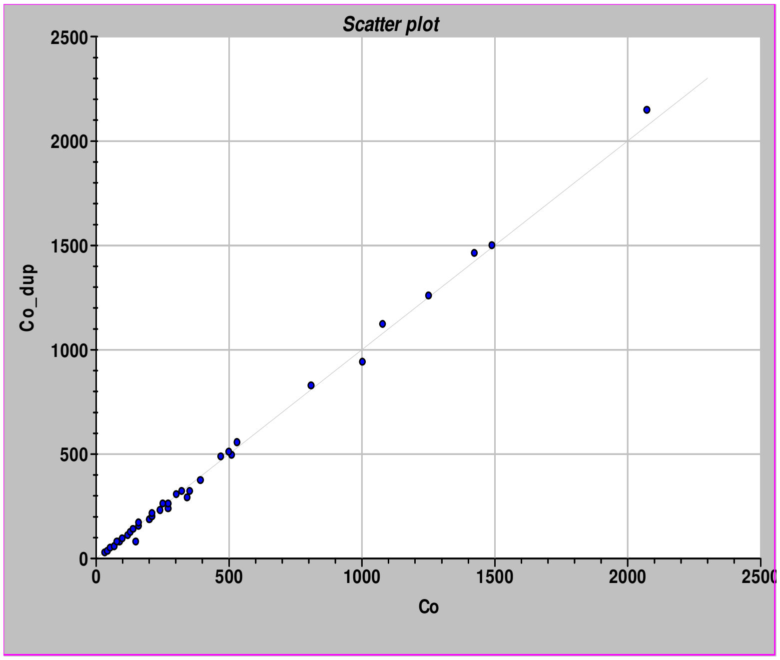


Data Statistics

variable:	Cu	Cu_dup
weight by:	--	--
mean:	7407.436	7576.923
varnc:	230195168.000	251677232.000
coefvrn:	2.048	2.094
min:	50.000	40.000
q1:	130.000	120.000
median:	1280.000	990.000
q3:	6750.000	5900.000
max:	75900.000	78800.000
iqr:	6620.000	5780.000
covarnc:	240580704.000	
Pearson:	1.000	
Spearman:	0.987	
no. data:	39	

(data set at full limits)

Figure 23: Boléo Duplicate Assay Results – Co



Data Statistics

variable:	Co	Co_dup
weight by:	--	--
mean:	419.231	417.949
varnc:	212904.531	225795.797
coefvrn:	1.101	1.137
min:	30.000	30.000
q1:	130.000	110.000
median:	210.000	220.000
q3:	470.000	490.000
max:	2070.000	2150.000
iqr:	340.000	380.000
covarnc:	219018.938	
Pearson:	0.999	
Spearman:	0.993	
no. data:	39	

(data set at full limits)

14.3 PROJECT DATABASE

14.3.1 DATA

A new Microsoft Access database was customised for the Boléo project. This database was set up to accommodate the historical data and the new data from the current programs.

Historical data exists in the form of 10 Excel spreadsheets, a single spread sheet for each sequential hundred holes. Each file contained several sheets including a drill hole coordinate sheet, assay sheet and summary geological logging sheet. The data is stored in a number of non-active tables, i.e., no new data is to be added to the tables. The data was taken as supplied and no modifications were made.

Data from the current drill programs is imported into a number of active tables to which the data is progressively added as it becomes available. These tables are:

- Assay Header Table – Dispatch number, sample number sequence, elements assayed, method of assay, detection limits.
- Assay Table – Sample number, reported results for each sample.
- Header Table – Drill hole coordinates, hole depth, hole azimuth, dip, drilling dates, drilling type, bit diameter etc.
- Geology Table – Logging and sampling intervals, sample number, dispatch number, geology logging data.

Assay files are loaded directly from digital files supplied by the laboratory without any modification or manipulation, therefore avoiding traditional errors from manual data entry.

Original geology logs are hand written in full descriptive text. These logs are then translated by the relevant geologist into computer coding on paper data entry forms. The data on these forms is then manually entered into Excel spreadsheets with identical column names and formats. Finally these Excel forms automatically uploaded into the database.

Data is verified by means of a number of database queries run that detect such errors as mismatched 'From' and ' To' intervals, drill hole depth greater than last 'To' depth, missing sample numbers, duplicated sample numbers, duplicated sample intervals, missing key geology fields such as formation and lithology. A list of acceptable codes for each geological field is used to prevent incorrect and inappropriate codes being used.

On completion of all data entry the assay data from the assay table is merged with the geology data from the geology table. Sample number and dispatch number are used to ensure correct data merging. Historical data and new data are also combined together at this time to produce two tables that contain data for all holes, which are:

- Collar and Survey Table – Easting, Northing, Elevation, Total depth, Azimuth and Dip
- Geology and Assay Table – H ole ID, Interval 'From and To', sample number, preferred assays fields, geology descriptors for each sample interval.

14.3.2 DRILL HOLE SURVEYING

Drill hole locations were surveyed using a Total Station instrument (model: Topcon – Hiper GD/Legacy H L1/L2 RTK). Accuracy is reported as ±5 mm in the horizontal and 10 mm vertically.

No down hole surveys were carried out.

15 MINERAL PROCESSING AND METALLURGICAL TESTING

15.1 METALLURGY

15.1.1 BACKGROUND

Treatment strategies for the Boléo polymetallic mixed oxide/sulphide ore were studied by Fluor in the mid 1990s during the Curator PFS development that resulted in a complex, high capacity flowsheet matching the requirements of a low-grade, 'super-pit' design. The flowsheet featured a combination of roasting, leaching, precipitation and metal refining.

Recently, in parallel with the adoption of selective mining of higher grades (and significantly reduced waste mining), Bateman sought to simplify the flowsheet via a more direct approach incorporating leaching, solid-liquid separation, solvent extraction and electrowinning.

Key to the revised process was the successful demonstration of the solid-liquid separation characteristics of the leached clay ore followed by an effective process for dealing with the manganese in the pregnant leach solution (PLS). These changes in processing strategy result in a more robust, operable flowsheet with reduced operating and capital cost.

Significantly, the flowsheet has been successfully tested in two separate pilot plant campaigns held at the SGS Lakefield facility in Ontario, Canada.

The metal recovery circuits are typical of those deployed in numerous operations world-wide. In developing the Boléo flowsheet, Bateman was able to incorporate the results of earlier testwork; supplementing these with recent bench scale and pilot plant testwork results, as well as information from recent Bateman projects featuring similar processes and unit operations.

The following summarises the metallurgical testwork history and goes on to describe the proposed Bateman process flowsheet.

15.1.2 PROOF OF CONCEPT PILOT CAMPAIGN

SOLID LIQUID SEPARATION TESTWORK

As a precursor to conducting a proof of concept pilot campaign a bench scale test work campaign was conducted at SGS Lakefield's facility in Ontario. Batch leach testing was carried out on six different ore samples from the Boléo Reserve. The samples varied widely in their copper and cobalt grades and were sourced from widely geographically dispersed outcrops of Manto 3.

The principal objective of the work was to generate leached pulp samples for subsequent settling test work by specialist vendors (Outokumpu) and consultant representatives (Pocock

Industrial). The samples were tested for their amenability to High Rate thickening in a series of tests, the success of which was key to the flowsheet development.

In essence the solid-liquid separation tests conducted at SGS Lakefield Research have demonstrated that Boléo ores can be settled and washed in a conventional CCD circuit utilizing high rate thickeners. Washing of settled solids to recover metal values from the PLS solution is fundamental to the economic success of the project.

The samples were found to settle best when diluted to between 2.0% and 3.5% solids. The optimum flocculant dosage for the leach varied from 3 ppm to 6 ppm. Underflow densities of 20% to 22% were achieved in test work with clear overflows being produced.

This test work was sufficiently successful to warrant taking the next step in the process development, namely the operation of a 'proof of concept' pilot plant campaign.

SOLVENT EXTRACTION TESTWORK

The SX Group of the Parker Centre, CSIRO Minerals in Perth, Australia have developed a process that allows for the separation of copper, cobalt and zinc from calcium, magnesium and manganese by solvent extraction with synergistic mixtures of commercially available organic extractants.

The combination of organic extractants results in a large 'synergistic' shift in the pH 50 for the system under consideration, allowing efficient separations of metals in solution to take place. This system of reagents allows for the separation of copper, cobalt and zinc from manganese, magnesium and calcium. This technology provides a means of dealing with the high levels of manganese in solution and is both fundamental to and appropriate for metals recovery from Boléo solutions.

In preliminary test work at pH 4.5, 100% of the copper was extracted, cobalt extraction was in the range of 94% to 98%; zinc extraction was in the range of 64% to 80% and manganese extraction in the range of 0.76% to 1.55%. In almost all cases, no calcium and magnesium were extracted at pH 4.5. The separation factors of copper and cobalt over manganese ranged in the ten thousands and thousands, respectively, suggesting an easy and complete separation of copper and cobalt from manganese for all the systems tested. The separation factors of zinc over manganese ranged in the hundreds, suggesting that a good separation of zinc from manganese can be obtained.

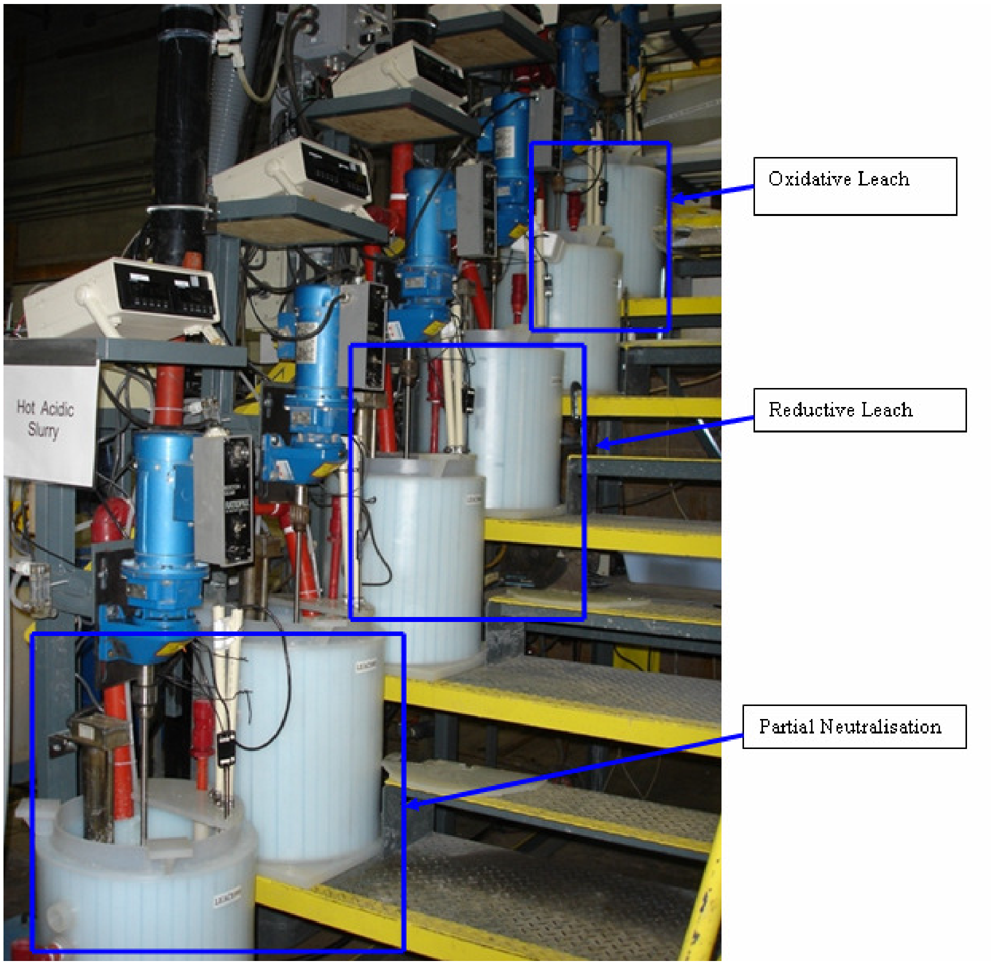
The success of this test work promoted further investigation and after some additional bench scale testing the concept was included in the flowsheet for 'proof of concept' piloting.

PROOF OF CONCEPT PILOT CAMPAIGN TESTWORK

A 'proof of concept' pilot plant campaign was conducted at the SGS Lakefield facility from November 16 to 28th , 2004 treating a bulk sample of Boléo ore grading 1.6% Cu, 0.087% Co, 0.58% Zn, 3.23% Mn and 8.71% Fe. The pilot plant flowsheet comprised:

- attritioning of the ore with grinding of coarse ore particles to form an ore slurry
- acid oxidation leaching of the ore with sulphuric acid in seawater
- acid reduction leaching of the ore with sulphur dioxide and sulphuric acid in seawater
- partial neutralization with limestone

The picture below shows the oxidative leach, the reductive leach and the partial neutralization steps undertaken at the SGS Lakefield facility in Ontario.



- Counter current decantation (CCD) washing of the leach residue in thickeners, to separate the metal rich aqueous solution from the clay waste

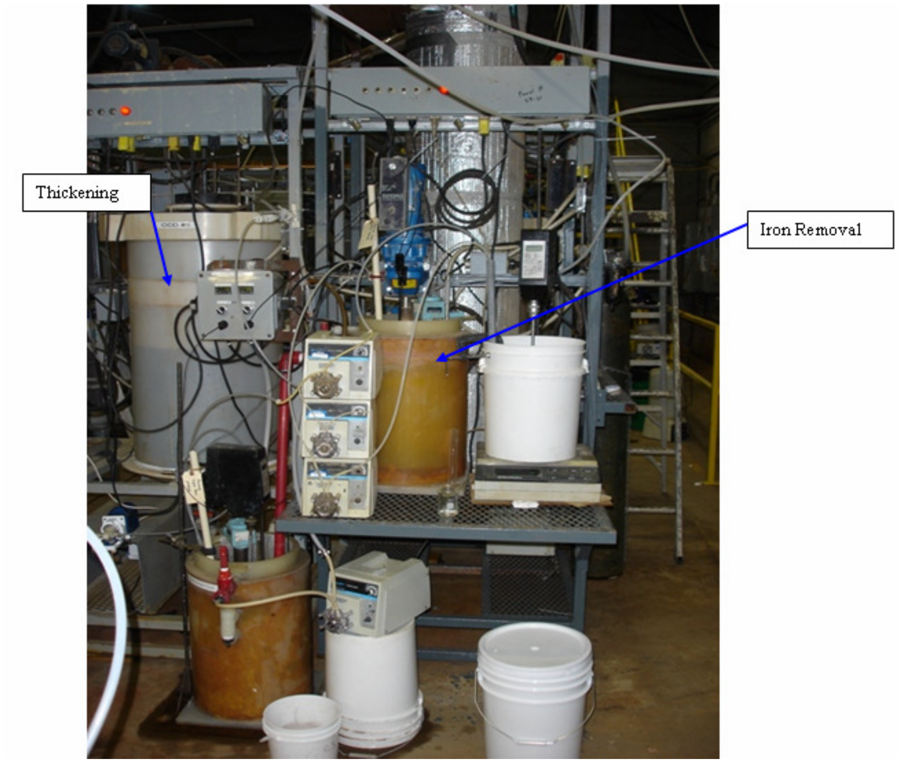
The picture below shows 4 of the 6 stage CCD thickeners.

- Copper solvent extraction/electrowinning

The picture below shows the copper solvent extraction mixers/settlers to the left and the electrowinning process to the right, under a fume extraction hood. The cathodes can be clearly seen suspended in the electrowinning cell.

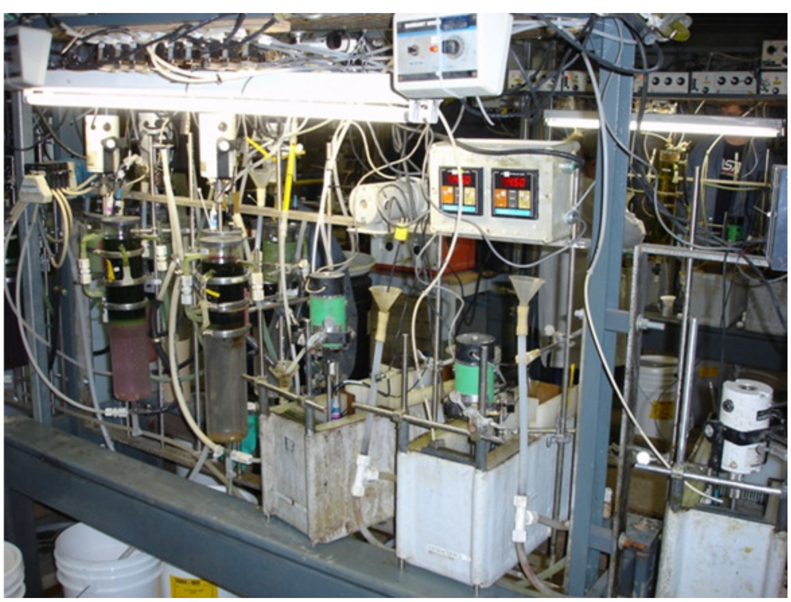
- Iron removal by pH adjustment and oxidation with air (and polish with hydrogen peroxide)

- Thickening of the iron residue prior to disposal;



- Direct Solvent Extraction technology ('DSX ®') for selective recovery of cobalt and zinc and small amounts of residual copper. As mentioned above DSX ® technology is the property of Commonwealth Scientific Industrial Research Organization (CSIRO), Perth, Australia.

The picture below shows the setup of the DSX circuit.



SGS Lakefield and Bateman Engineering have previously jointly reported the summary findings from the pilot plant. Highlights of the campaign included the following:

- A total of nearly 2 metric tonnes of ore were treated through the pilot plant.
- The pilot plant operated continuously for a total of 12 days in leaching, 11.5 days in CCD, 9.5 days in Copper SX/EW and 9 days in Cobalt and Zinc SX using the DSX ® technology.
- The oxidation, reduction leaching circuit gave excellent extractions of copper, cobalt and zinc. Copper extraction exceeded 90% during pilot operation. Cobalt extraction varied from 80% to as high as 90%. Zinc extraction was generally above 70%. These numbers are indicative of the potential of the Boléo process to extract the three pay metals copper, cobalt, and zinc.
- The CCD circuit was set up to simulate the use of the “high rate” type of thickeners with recirculation of overflow solution to dilute the feed slurry prior to flocculation. This method of settling and washing was based on recommendations from bench-scale testing by Outokumpu and Pocock Industrial and proved to be highly effective. The leach residue settled quickly, producing clear overflow solutions to advance to copper, cobalt and zinc recovery.
- 15.5 kg of copper metal were electrowon from the solvent extraction strip solutions at high efficiency. A picture of the first copper cathode produced during this campaign appears below. These cathode samples assayed at better than LME grade.

- The iron removal circuit was designed to remove iron, aluminium and other impurities from the solution prior to recovery of cobalt and zinc using DSX ® technology. The iron removal circuit consistently produced very low concentrations of key impurities in solution with negligible losses of cobalt and zinc.
- The DSX ® circuit for cobalt and zinc recovery performed very well. The advantage of the DSX ® circuit for Boléo plant design is that cobalt and zinc can be separated from manganese, magnesium and calcium. In the Lakefield pilot plant, cobalt and zinc were recovered with high overall extraction efficiency (+95%) to produce a concentrated zinc sulphate solution (for subsequent production of zinc sulphate monohydrate crystals) and a concentrated cobalt solution (for subsequent production of cobalt metal cathode).

Two further metallurgical tests were performed on the product streams from the pilot plant.

- Production of zinc sulphate monohydrate crystals. Zinc sulphate monohydrate was produced by evaporative crystallization of the zinc strip solution from the DSX ® circuit.
- Production of cobalt cathode. The cobalt strip solution from the DSX ® circuit containing cobalt along with small amounts of zinc and nickel was treated in a zinc solvent extraction circuit followed by a cobalt SX/EW circuit to purify the cobalt solution for electrolysis as high grade (+99.9%) cobalt cathode. Conventional solvent extraction reagents and process steps were utilized for this purpose.

In addition certain bench scale tests were conducted to obtain additional data for definitive feasibility study purposes including:

- Environmental testing of residues and solutions produced in the pilot plant program
- Characterization of High Acid Consuming (HAC) material from the Boléo site containing limestone and other alkali minerals. HAC material is intended to be used as a low cost neutralizing agent in the commercial Boléo plant
- Ore scrubbing and grinding testwork for developing final design for ore preparation circuit, including Bond Work Index determination for both ROM ore and HAC material
- Testwork on oxidation and precipitation of iron from the DSX ® circuit feed solution to ensure maximum removal of iron with minimum treatment time and reagent consumption
- Leach testing on 24 individual samples of ore that were composited to form the pilot plant feed. These tests were used to assess leach variability and acid consumption variability of the ore, and the production of an acid consumption model.

The successful completion of this pilot program was an important milestone in moving the Boléo project forward. For the first time Boléo ores had been treated in a continuous pilot plant program to leach, separate and recover pay metals in final commercial form.

15.1.3 *Fully Integrated Pilot Campaign*

Refinements to the process flowsheet followed in an attempt to optimise the capital and operating costs for the proposed operation as did targeted testwork initiatives designed to support the proposed flowsheet variations.

Targeted DSX system testwork conducted by Bateman's Solvent Extraction Group in late 2005 produced an optimum set of conditions for the operation of the DSX circuit, relating in particular to the relative proportions of the extractants and the operating pH.

Boléo limestone, available on the Boléo Reserve, was found to be effective in bulk neutralization duties and iron precipitation in the Iron Removal Stage during bench scale testing at SGS Lakefield. Soda ash was found to be effective at pH control. Milling in acidic raffinate was found to bolster circuit tenors and improve the overall 'water balance' for the circuit.

These various initiatives gave rise to the need to conduct a second pilot campaign with certain targeted objectives. The campaign was duly scheduled for July 2006 and the pilot plant constructed and commissioned to meet these dates.

Objectives for the 19 day pilot campaign included:

- Building on and verifying the “Proof of Concept” pilot plant results of 2004
- Demonstrating the flowsheet feasibility on a larger scale and to a greater extent of integration than in the previous campaign
- Assessing larger scale CCD settling behaviour and performance
- Testing the suitability of the DSX technology for the selective recovery of zinc and cobalt from a Mn matrix using a modified (optimised) mixture of the synergistic reagents
- Confirming key reagent consumptions, including the use of Boléo Limestone (HAC) for pH adjustment
- Demonstrating the use of soda ash in pH control in the various Solvent Extraction circuits as a lower cost replacement for sodium hydroxide
- Demonstrating the Cadmium Cementation step
- Demonstrating Manganese Carbonate production
- Confirming product quality
- Verifying existing design criteria and confirming current design assumptions
- Extracting engineering design data
- Testing the proposed plant control philosophy.

In addition to continuous piloting of the Boléo flowsheet, on-site bench-scale testwork was conducted by the following vendors and industry specialists:

- Outokumpu Technology – CCD & high rate thickening
- Pocock Industrial Inc – CCD & high rate thickening
- RPA Process – Filtration of iron residues, manganese carbonate product
- Mixtec – Agitation testing in oxidative and reductive leach, partial neutralization and tailings neutralization
- SGS Lakefield – Environmental characterisation of Boléo Pilot tailings
- SGS Lakefield – Production of purified cobalt carbonate
- SGS Lakefield – Cobalt removal from DSX zinc solution (precipitation by Caro’s acid).

Samples were also sent off-site for treatment by the following:

- GLV Pty Ltd – Paste Thickening options for the CCD Circuit
- Saskatchewan Research Council – Tailings pump loop testing
- Jenike & Johanson Ltd – Flowability testing of crushed ore.

Pilot Campaign Ore Feed

Approximately 10 tonnes of Boléo ore, composited from 143 channel samples of underground oxide Manto 3 ore from the test mining campaign, were homogenised, and crushed to produce the feed to the pilot plant. A 200 kg composite sample was prepared and used for preliminary grinding testwork.

Of the initial 10 tonnes prepared approximately 8 tonnes were milled during the campaign, and approximately 5 tonnes were consumed in the pilot campaign. The average grade of the elements of interest for the duration of the campaign is shown in Table 15.

Table 15: Comparison of As Received, 200 kg Composite and Scrubber Feed Assays

	H_2O %	Cu %	Co g/t	Zn g/t	Mn %	Fe g/t
Baja Assays – By Chemex (As Received from Site)						
Average	27.39	2.0078	1252	4753	3.9601	8.2383
Length Weighted Average		1.9928	1235	4757	3.9464	8.1712
Boléo Bench Testwork - 200 kg SGS Composite Sample						
Calculated	27.81	2.1388	1177	4820	3.6855	8.3434
Assayed		1.8200	1130	4020	4.8200	8.0000
Composite Samples Taken off the Feed Belt to Scrubber During Milling Campaigns						
Weighted Average		2.1700	1320	4800	5.0400	8.1400

Grade variation of the prepared feed slurry over the 19-day campaign is shown in Table 16:

Table 16: Leach Feed Slurry Assays

Sample	Cu %	Co g/t	Zn g/t	Mn %	Si %	Fe %
Feed Ore 1A	2.16	1020	4780	4.14	18.7	8.04
Feed Ore 1B	2.25	1230	4650	5.58	19.4	8.15
Feed Ore 2A	2.26	1410	4810	5.99	18.5	8.22
Feed Ore 2B	2.26	1240	5270	4.27	20.4	8.90
Feed Ore 2C	2.27	1570	4990	5.21	19.0	7.90
Feed Ore 3A	2.18	1370	5130	5.25	20.1	8.59
Feed Ore 3B	2.04	1550	4640	5.45	20.6	7.71
Feed Ore 3C	2.04	1370	4990	4.16	21.2	8.58
Minimum Feed Ore	2.04	1020	4640	4.14	18.5	7.71
Maximum Feed Ore	2.27	1570	5270	5.99	21.2	8.59
Average Feed Ore	2.18	1350	4910	5.01	19.7	8.26

Oxidative, Reductive and Partial Neutralisation Leach Stages

Recovery of Cu, Co and Zn from Boléo ore requires three atmospheric leach stages.

The first stage is an oxidative leach with sulphuric acid (pH 1.2) at 70°C to 80°C for three hours. No reagent addition was required to maintain a redox potential of 600 mV to 950 mV due to the natural occurrence of MnO_2 within the Boléo ore body.

A second stage reductive leach with sulphuric acid (pH 1.5) at 70℃ to 80℃ for three hours, ensures dissolution of the manganese minerals. A redox potential of 400 mV is maintained by addition of SO_2 gas. This reductive step releases additional copper, cobalt, and zinc locked within the manganese mineralization.

A partial neutralization stage, conducted at pH 2.0 at 70℃ 80℃ for one hour follows prior to copper solvent extraction to ensure high extraction efficiencies in copper SX by subtle pH adjustment. The pH modification is also carried out to minimize gel formation, especially silica gel.

Impurities leached include calcium, iron, magnesium, and aluminum, contributing to the overall acid consumption of the ore.

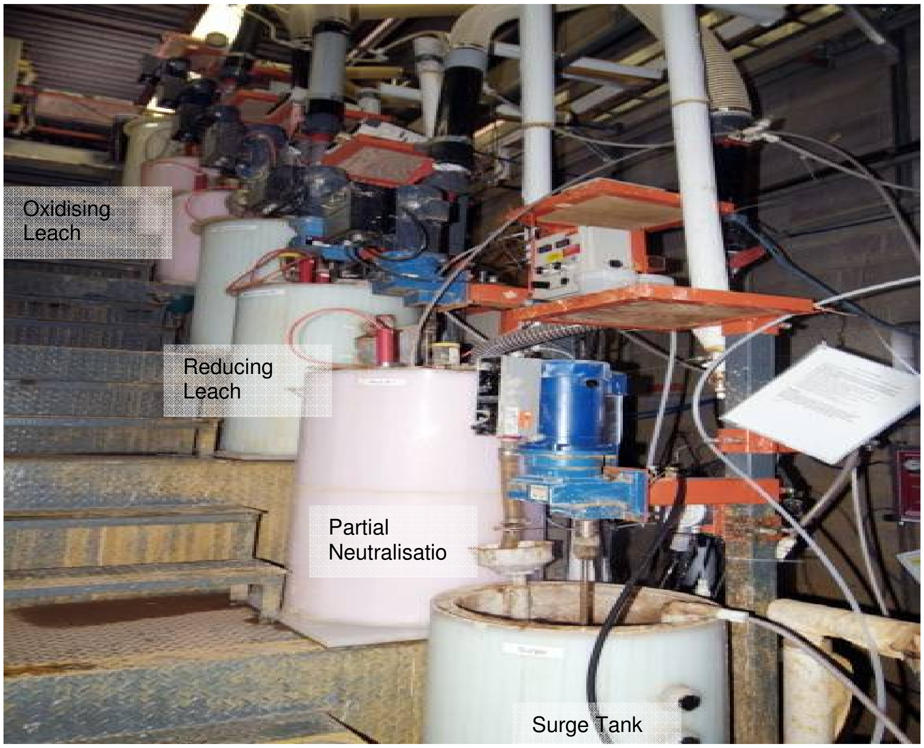


Photo Overview of the Leach Circuit.

COUNTER CURRENT DECANTATION (CCD)

Due to the fine nature of the feed slurry, counter current decantation was selected as the residue wash step in order to reduce soluble losses. The CCD circuit setup included six stages of thickeners, feed dilution to 2% to 3% solids, underflow washing with brine and manganese carbonate precipitation thickener overflow at a wash ratio of 1.75 m^3/t and flocculation with Hychem 301.

Photo Overview of the piloting Counter Current Decantation (CCD) equipment setup.

The CCD circuit was operated at 40℃ as per the requirements of the downstream solvent extraction circuit. The final tailings were drummed, with a portion blended with iron removal circuit residue, neutralised with HAC and lime to pH 7, and sent for pump loop testing by the Saskatchewan Research Council and environmental characterisation testwork by SGS Lakefields' Environmental Group.

COPPER SOLVENT EXTRACTION (CUSX)

The Cu SX process utilized 20%v/v LIX 664N extractant and Orfom SX 80 CT diluent. The circuit consisted of two extraction, one scrub and two strip stages operated at 40℃.

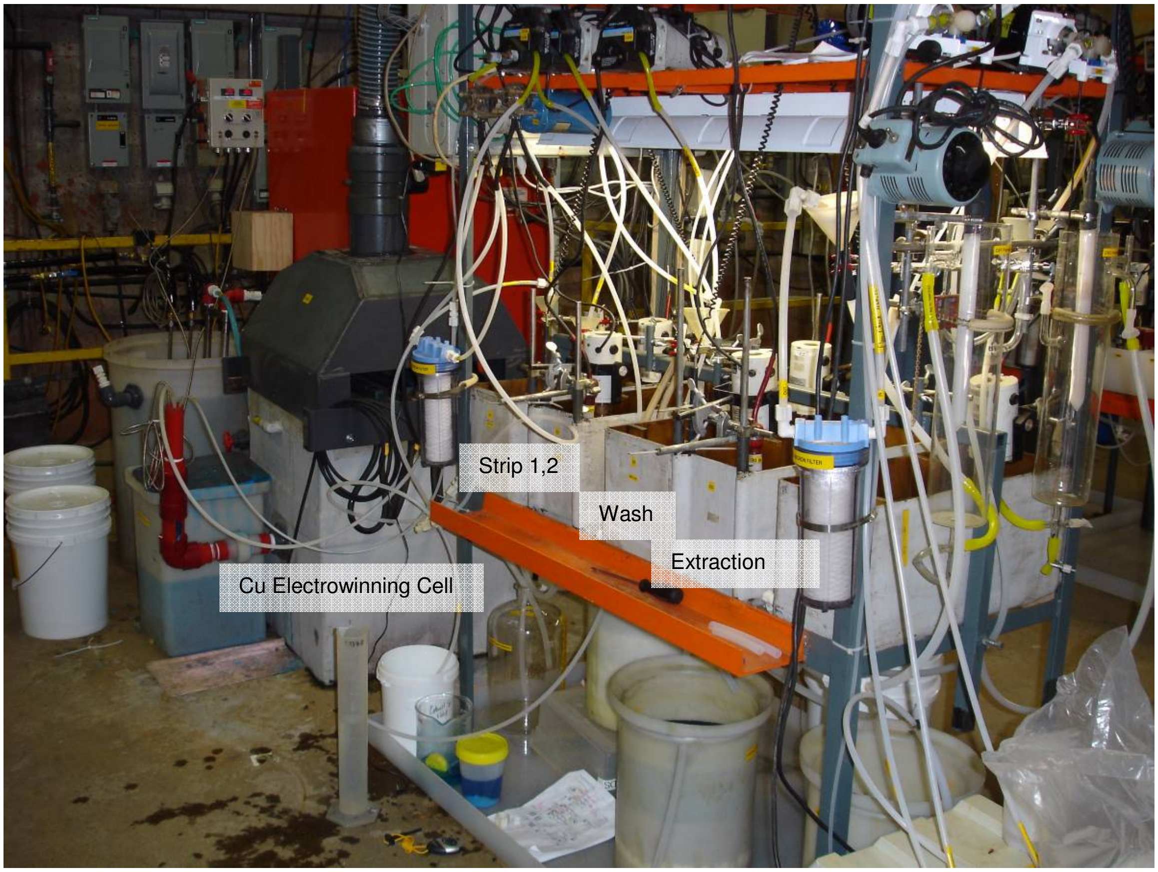


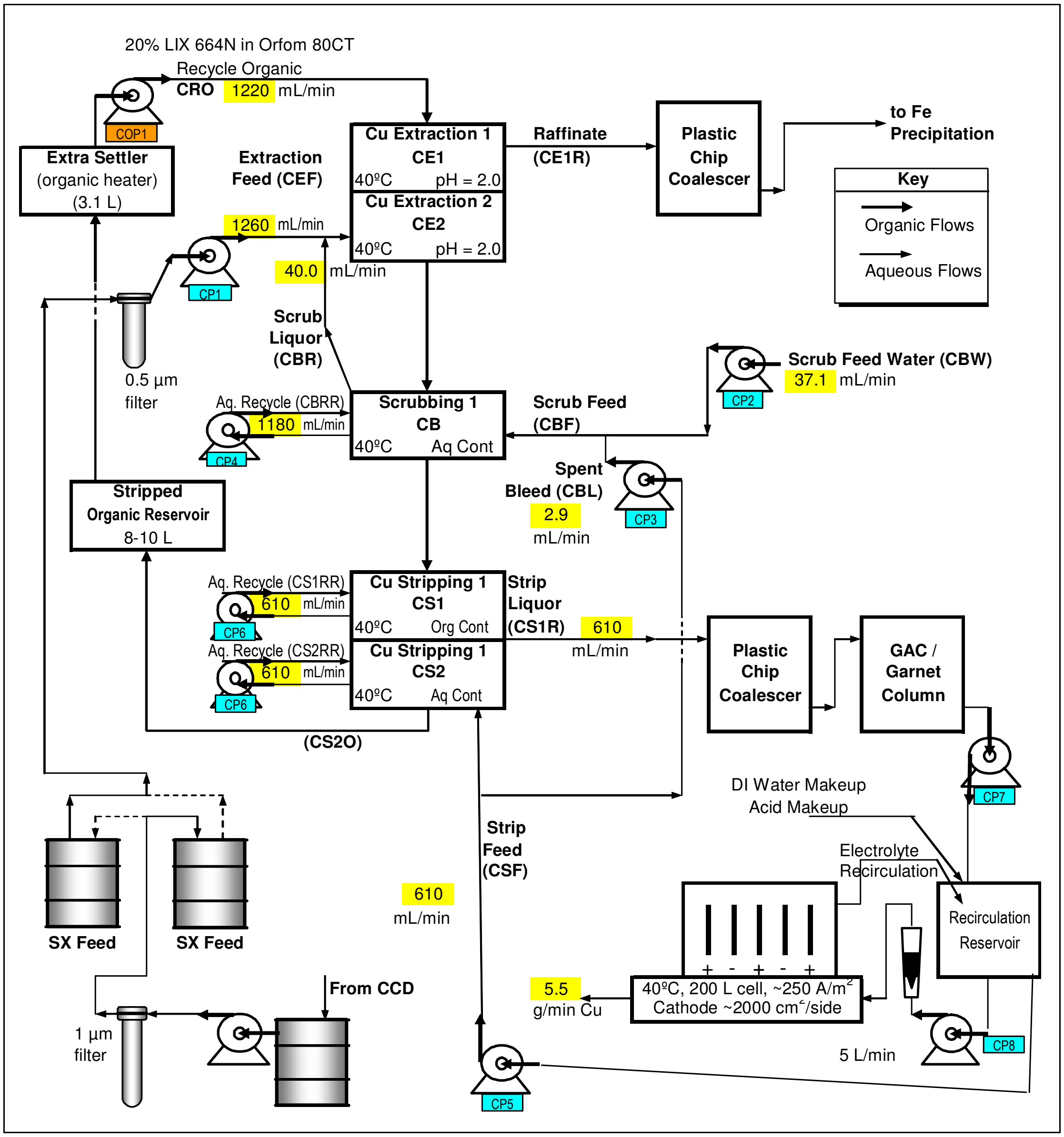
Photo Overview of the piloting Copper Solvent Extraction equipment setup.

Pregnant Leach Solution (PLS) from CCD 1 overflow was collected and filtered to remove suspended solids using a 1 µm in-line cartridge filter prior to being pumped to a storage container ahead of the CuSX circuit. The solution was filtered with a 0.5 µm filter prior to entering the first extraction stage. No pH adjustment was done in the scrub and strip stages. Organic entrainment removal was via a plastic chip coalescer for the raffinate stream and a combination of a plastic chip coalescer and a multimedia activated carbon filter for the loaded electrolyte stream.

Copper Electrowinning (CuEW)

The CuEW process utilized a conventional circuit with lead-calcium-tin anodes and stainless steel starter cathode sheets, operated at 40°C and a target current density of 250 A/m^2 over a nominal four day strip cycle. The lead anodes were conditioned prior to the start of the campaign to reduce the potential for lead contamination of the copper product.

Figure 20: Cu Solvent Extraction and Electrowinning Flowsheet, and Operating Parameters

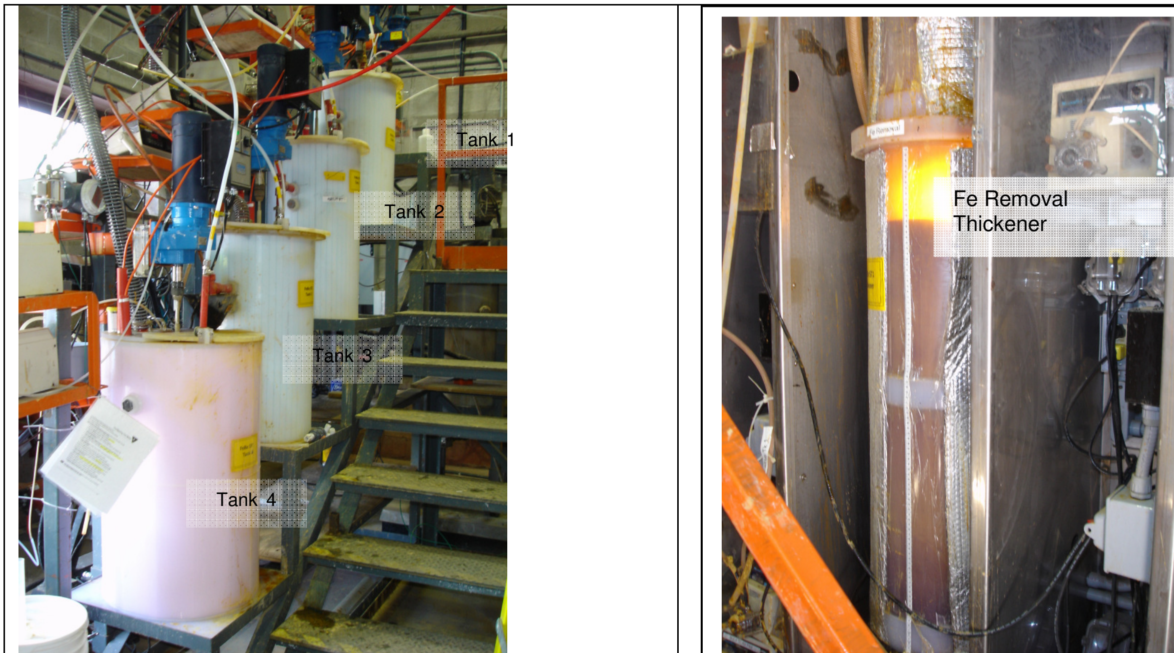


Iron Removal

The objective of this step is to precipitate the iron, aluminium and residual copper as hydroxides by adjusting the pH of the process feed stream (CuSX raffinate) with HAC to pH4, effectively oxidizing Fe^{+2} to Fe^{+3}.

Initially, the iron was removed in two stages with HAC and air being fed to the first stage, and lime and oxygen fed to the second stage. The circuit was modified during the campaign by removing one of the original thickeners giving a circuit configuration consisting of a feed tank and four reaction tanks followed by one thickener.

The iron precipitate was thickened using Hychem 301. The thickener overflow proceeded to the Co Zn DSX circuit and the underflow was filtered. The filter cake was bagged and drummed. The filtrate was returned to the second reaction tank. The circuit was operated at 50°C during the campaign to be representative of the operating temperature of the industrial plant.



Photos: Pilot Plant Iron Removal Reactors and Iron Removal Thickener

CSIRO Direct Solvent Extraction (DSX)

The DSX process utilized a 6.25% Versatic 10 (0.33M) and 13.2% LIX 63 (0.3M) organic mix in Orfom 80 SX CT diluent. This organic mixture was developed by the Commonwealth Scientific & Industrial Research Organisation (CSIRO) in Australia for the selective recovery of Co and Zn in a matrix containing high manganese concentrations (typically 20 g/L to 50 g/L Mn).

Extensive bench-scale testing by CSIRO indicated selective recovery of Co and Zn from Boléo solutions and this was successfully demonstrated in the previous pilot plant campaign. At that time, however, the process suffered the disadvantage of organic degeneration, thought to be caused by manganese oxidation, acerbated by elevated temperature (>30°C).

Subsequent testwork by CSIRO and Bateman Advanced Technologies (BAT, Israel) confirmed the effect of manganese on the organic stability and lead to a series of tests by CSIRO to optimise both the circuit conditions and the relative proportion of the synergistic reagents. Testwork successfully demonstrated that degradation can be all but eliminated by ensuring that manganese does not load onto the Lix 63 extractant.

The key process parameter in this regard is to ensure that the pH is maintained below 4.5 pH units.

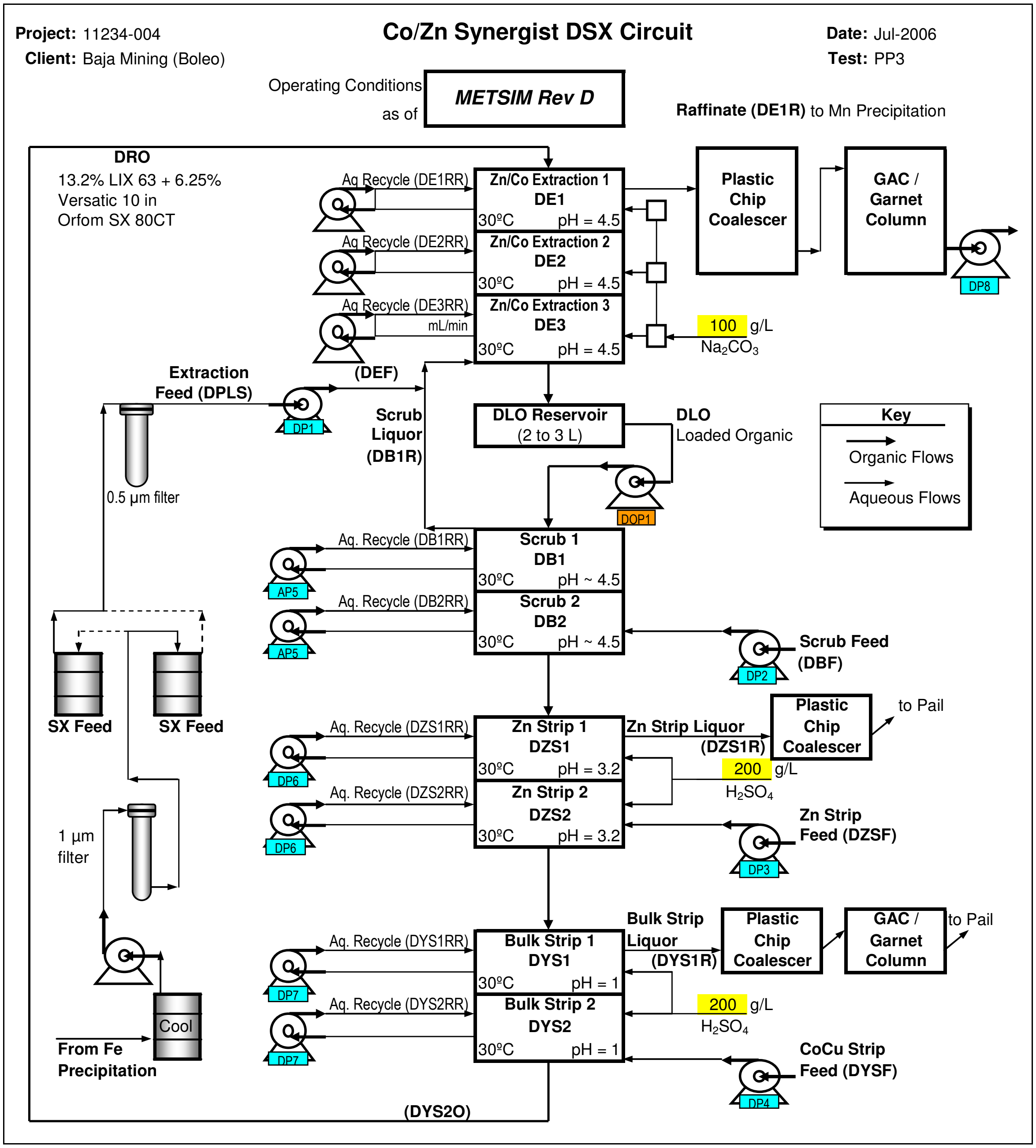
The circuit consisted of three extraction, two scrub, two Zn strip and two bulk (Co, Zn) strip stages operated at 30℃.

Photo Overview of the Pilot Plant Direct Solvent Extraction (DSX) Circuit

Iron free solution was collected and filtered to remove suspended solids with an in-line cartridge filter prior to being fed to the DSX circuit. pH in extraction was controlled by dosing a 100 g/L sodium carbonate solution, pH in Zn strip and bulk strip stages was controlled by dosing 200 g/L sulphuric acid. Zn and Co strip solutions were collected for further processing to zinc sulphate and cobalt metal respectively in the second phase of the testwork campaign.

The DSX operating parameters were as follows:

Figure 21: Direct Solvent Extraction (DSX) Flowsheet & Operating Parameters



MANGANESE CARBONATE

The objective of this circuit is to precipitate manganese from the DSX raffinate stream. The raffinate from the Co Zn DSX circuit is fed to the manganese precipitation tank, which is maintained under an inert atmosphere with nitrogen gas.

Sodium carbonate (Na_2CO_3) is used to precipitate manganese as its carbonate. The treated stream is processed in a thickener; the overflow is recycled to the CCD circuit to be used as wash water and the underflow is filtered.

Photo Overview of The Manganese Carbonate Precipitation Circuit (Reaction Tanks and Thickener)

Photos: Manganese Carbonate Thickener Underflow And Filter Cake

Pilot Plant Controls

Pilot plant controls included regular flowrate/mass measurements, on-line process variable trending (temperature, redox potential and pH), analytical laboratory assay trending, bench titrations, reagent consumption monitoring (flowmetres, load cell measurements, containers) and log sheets. Actual flowrates were cross-checked with the load cell data to produce a full mass balance with the following accuracies:

- solids within 5 % deviation
- solution within 1 % deviation.

A high level process review was performed daily by the management team on site.

Analytical methods utilized by SGS Minerals Services Analytical Group included:

Table 17: SGS Minerals Services Analytical Methods Employed

Area	Sample Type	Method
Batch Grinding	Pulp	SG
	Solids	Malvern
	Liquids	SG, ICPLa, Cl, Fe^{2+}, FAT
Leach and Partial Neutralization	Pulp	SG
	Solids	ICPS
	Liquids	SG, ICPLa, Cl, Fe^{2+}, FAT
CCD	Pulp	SG, TSS
	Solids	ICPS
	Liquids	SG, ICPLb
Iron Removal	Pulp	SG, TSS
	Solids	ICPS
	Liquids	SG, ICPLa, Cl, Fe
Cu SXEW	Aqueous	SG, ICPLa, Cu, Ge/In/Ga, Cl, FAT
	Organic	SG, ICPO
	Cathode	ICPC
DSX	Aqueous	SG, ICPLa, Fe, Ge/In/Ga, Cl
	Organic	SG, ICPO

Special sample preparation for both analytical and SG determination samples included the multiple washing (two-stage repulp) of solids to minimize errors from soluble metal contributions.

Commissioning and Campaign Duration

Pilot plant commissioning commenced on 5th June 2006. The integrated pilot plant campaign ran until 24th June. The zinc solvent extraction and cobalt solvent extraction and electrowinning circuits were run during the period of 4th to 15th July in a separate campaign. The start-up times for the various sections are shown in Table 18.

Table 18: Boléo Pilot Plant Timeline

Section	Time	Date
Milling	Batch	5 June
Leach	23h00	6 June
CCD1 (Solids feed)	10h30	7 June
CuSX	10h45	8 June
Fe Removal	19h30	8 June
CuEW	09h00	10 June
DSX	01h00	10 June
$MnCO_3$	10h30	11 June
First Cu Cathode	15h00	13 June
End of Campaign		
Leach	06h00	24 June
CuSX	23h00	25 June
Fe Removal	07h00	27 June
DSX	07h00	27 June
$MnCO_3$	07h00	27 June

SUMMARY OF PILOTING RESULTS

LEACH AND PARTIAL NEUTRALISATION

The Leach and Partial Neutralization operational KPIs were as follows:

Table 19: Leach and Partial Neutralisation Operating Result

Parameter	Unit	Oxidative Leach Tank 1	Oxidative Leach Tank 2	Reductive Leach Tank 1	Reductive Leach Tank 2	Partial Neutralization
Temperature	ºC	80	80	80	80	80
pH (Ag/AgCl – sat'd KCL)		1.5	1.4-1.7	1.2-1.5	1.2 -1.5	2.06
Redox Potential	mV	>800	>800	400	400	~400

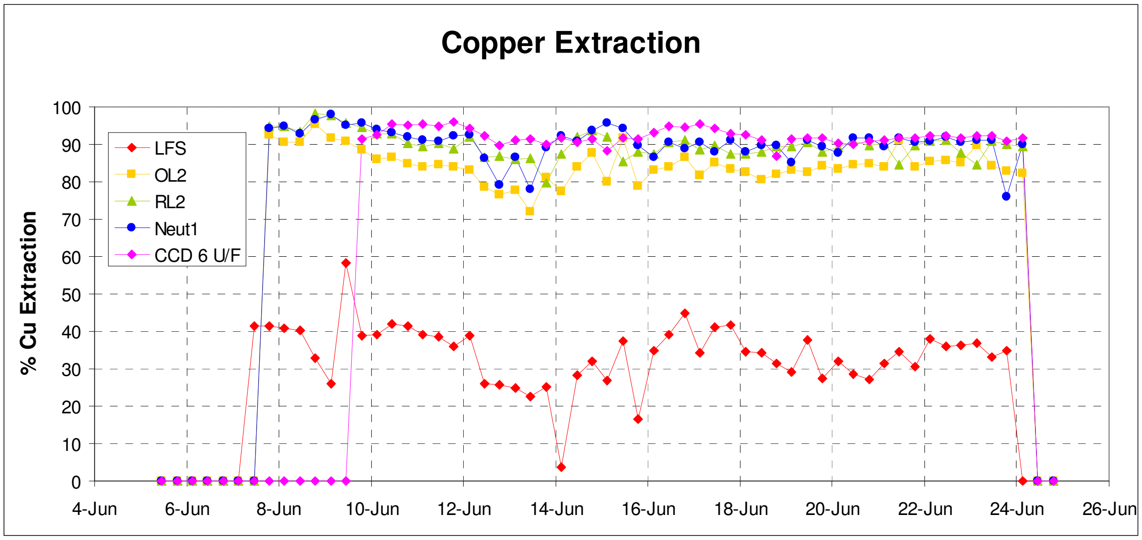
The average leach extractions for the various pH conditions of the campaign in Oxidative Leach Tank 2 are shown below. The figures were calculated using a silica tie method to compare the metals values in the feed with the values in the washed residue from CCD.

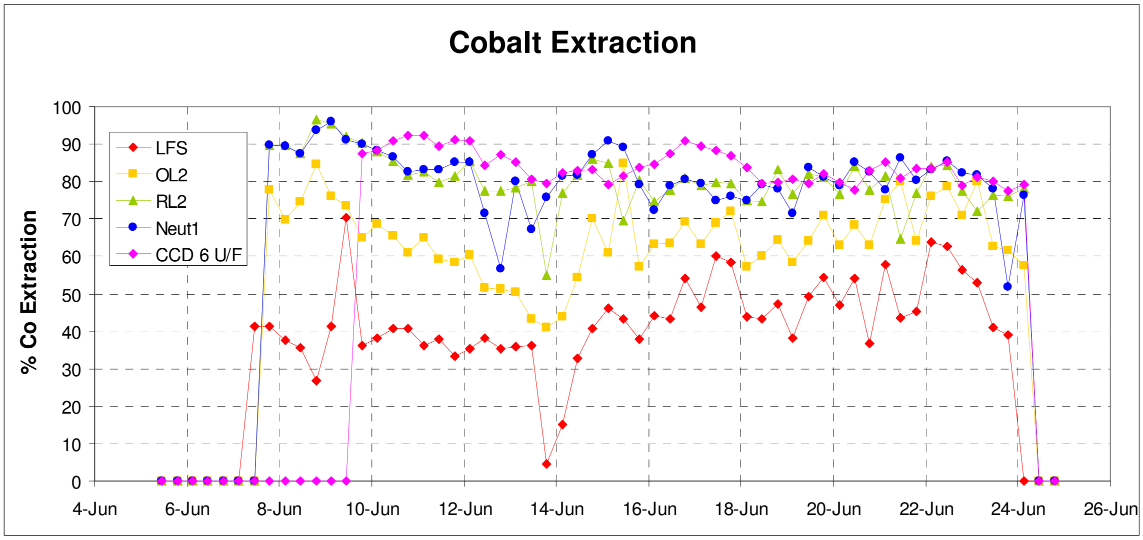
Table 20: Leach Efficiencies

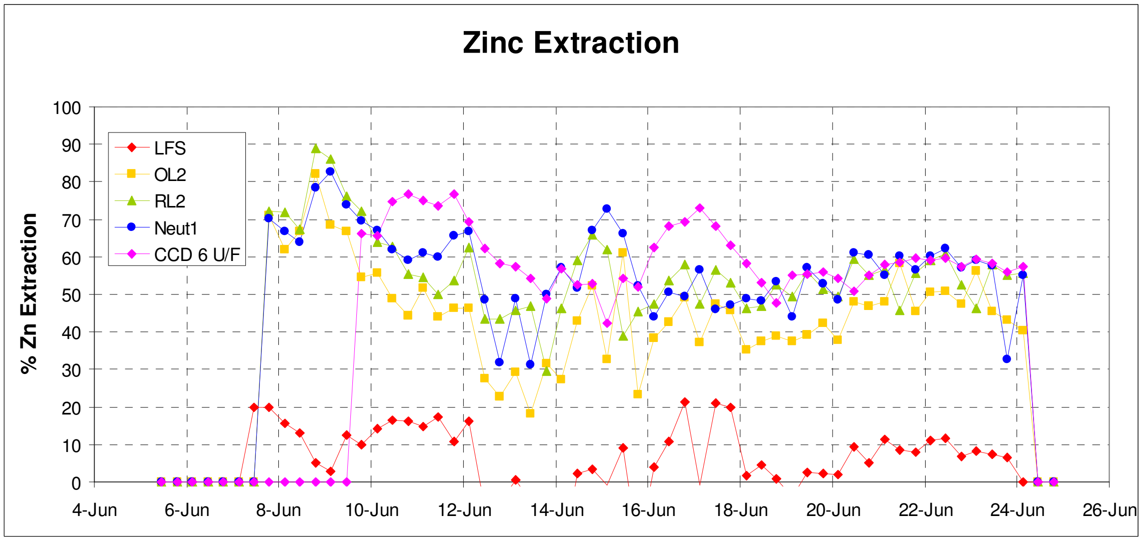
Condition	OL2 pH	% Extraction Cu	% Extraction Co	% Extraction Zn	% Extraction Mn
OL2 pH = 1.7	1.7	90.5	80.8	52.9	96.6
OL2 pH = 1.5	1.5	90.5	79.4	54.4	95.8
OL2 pH = 1.4	1.4	92.4	82.1	60.1	97.7
Start-up	1.2	94.1	89.3	71.6	94.9
Overall average	1.4	91.8	82.4	59.1	96.5

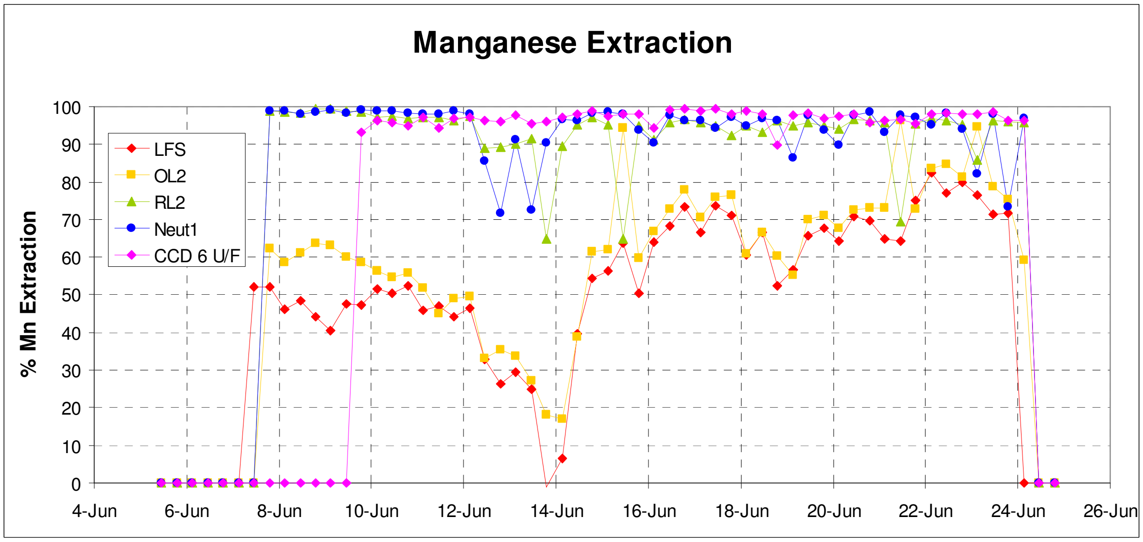
Graphs of the metals extractions for the duration of the pilot plant campaign are shown below.

Figure 24: Boléo Metal Extractions









Note: The following abbreviations apply - Leach Feed (LFS), Oxidative Leach (OL2), Reductive Leach (RL2), Partial Neutralisation (Neut 1) and CCD6 Underflow (CCD 6 U/F).

COUNTER CURRENT DECANTATION (CCD)

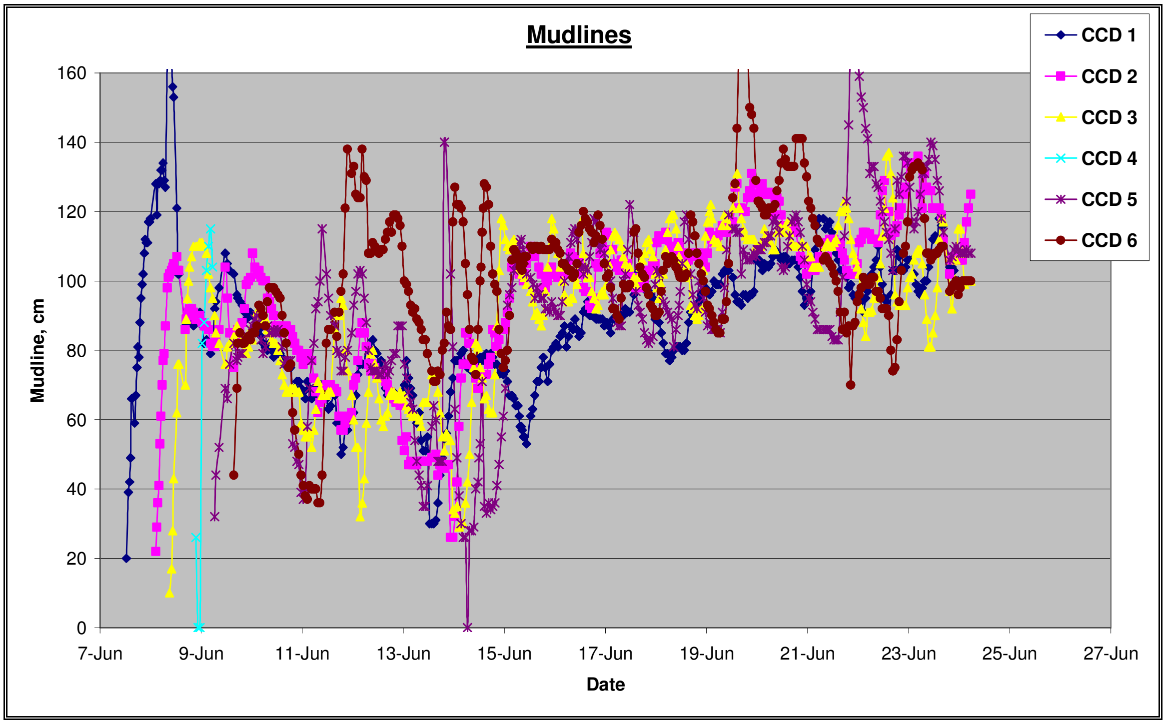
The CCD circuit average operational KPIs were as follows:

Table 21: CCD Operating Results

Parameter	Unit	CCD 1	CCD 2	CCD 3	CCD 4	CCD 5	CCD 6
U/F Density	% solids	10	12	14	15	16	15
Wash Ratio	m^3/t	1.75					
Wash Efficiency	%	91					
Flocculant Addition - Total	g/t	470					

The graph below shows the improved operational performance of the CCD circuit as the campaign progressed.

Figure 25: Boléo Pilot Plant CCD Thickener Mud Levels



Observations:

- The CCD circuit produced underflow densities ranging from 10% to 24 % solids as reported on a shift basis. However, some of the results, particularly the higher ones, could be suspect as the calculation depended on a fixed value for the SG of the dry solids. The underflow densities obtained during vendor testwork (shown below) give a more accurate indication of the values attainable.
- Wash efficiencies as high as 99% were achieved but only at wash ratios in excess of 4 m^3/t.
- CCD 1 overflow clarities were generally good with clarity wedge readings of 47 on a 0 to 47 scale.
- There is potential for two-stage flocculation to reduce flocculant consumption.
- Analysis of the CCD6 underflow assays revealed that additional leaching was taking place in the CCD circuit.

Vendor Testwork

Thickening testwork was carried out by Outokumpu Technology and Pocock Industrial Inc during the campaign. In addition, samples were sent to GL&V for paste thickening testwork. The table below summarises the vendor results.

Table 22: Vendor Testwork – CCD Parameters

Parameter	Outokumpu	Pocock	GLV
Diluted feed, % solids	2-3	2-3	1.5
Max underflow, % solids	16.1	16.1	21-22
Overflow clarity – TSS, ppm	60-85	80-130	-
Optimum rise rate, m/h	1.64	0.87	-
Typical bed height, mm	180-230	1000	300
Flocculant	Hychem NF301	Hychem NF301	Hychem 302

It was noted that underflow densities were as much as 6% (absolute) lower at similar rise rates achieved in comparative tests conducted by Outokumpu in 2004. The reasons for the difference in performance are not well understood but it is postulated that they result from the combined effects of milling in acidic raffinate and the use of Boléo limestone for neutralization purposes.

COPPER SOLVENT EXTRACTION (CUSX)

The CuSX circuit extracted an average of 98.6% Cu. The average pregnant leach solution and raffinate assay values are shown below.

Table 23: Copper Solvent Extraction Solution Assays

Element	Unit	PLS	Raffinate
Cu	mg/L	2515	34.5
Co	mg/L	265	257
Zn	mg/L	1023	976
Mn	mg/L	11710	11601
Fe_{Total}	mg/L	5916	5837
Ca	mg/L	601	589
Mg	mg/L	7573	7442
Al	mg/L	3369	2901
Ni	mg/L	28	28
Si	mg/L	50-170	<140
Cl	mg/L	14535	13570

Only minor amounts of manganese and iron were co-extracted with the copper. The average organic assays are listed in Table 24.

Table 24: Copper Solvent Extraction Organic Assays

Element	Unit	Organic Assays		
		Loaded	Scrubbed	Stripped
Cu	mg/L	5620	5417	3200
Mn	mg/L	0.10	0.07	<0.05
Fe_{Total}	mg/L	36	8	0.9
Ni	mg/L	< 2	< 2	< 2

COPPER ELECTROWINNING (CUEW)

Four plating cycles were carried out during the campaign. The conditions are shown in Table 25.

Table 25: Plating Cycle Times & Operating Conditions

Cycle Number	1		2		3		4	
Start Time	June 9 17:50		June 13 15:33		June 16 18:43		June 21 15:18	
End Time	June 13 15:06		June 16 11:51		June 21 15:18		June 25 23:18	
Cathode Mass (kg)	A	B	A	B	A	B	A	B
	8.78	8.88	4.41	4.52	5.74	5.88	4.52	4.61
Av. Current Density,A/m^2	183		162		155		167	
Current Efficiency, %	94.4		97.3		99.5		99.6	

In spite of the difficulties encountered in copper solvent extraction all four cycles produced copper of the following quality. All exceeded the LME grade-A specification.

Table 26: Cathode Quality

Sample Number	Cathode, Cu%	
1	>99.996	>99.996
2	>99.993	>99.995
3	>99.995	>99.995
4	>99.996	>99.996

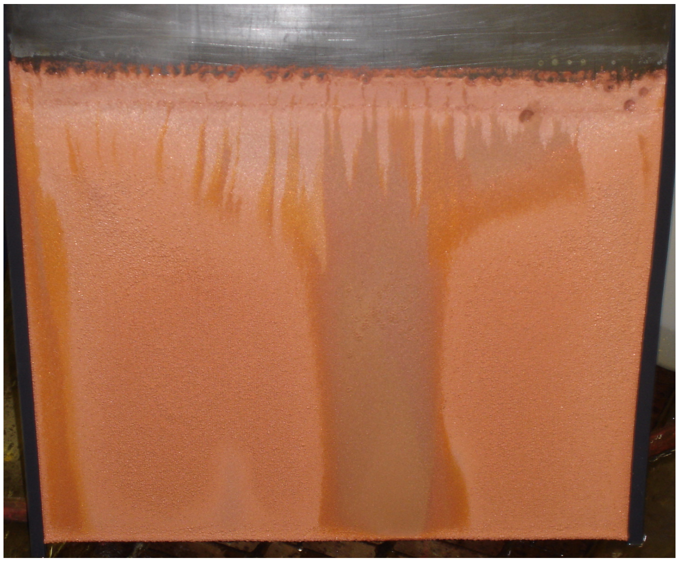


Photo Cathode From Cycle No.1 – 13 June 2006

Iron Removal

The occasionally erratic performance of the iron removal circuit did not adversely affect the downstream DSX operation. Many of the operational problems resulted from the relatively small size of the pilot plant equipment. The blockage of spargers and the poor oxygen utilization are not expected to occur in the industrial plant with the frequency that they did in the pilot plant.

The average concentration of iron in the feedstream was 5454 mg/L and in the thickener overflow (feed to DSX) it was 17 mg/L giving an average iron rejection for the campaign of 99.7%.

It must be noted that the iron levels in the thickener overflow were generally less than 1 mg/L and the average quoted above is a result of excursions resulting from blockage of oxygen spargers and / or limestone (HAC) addition.

The iron removal thickener in the pilot plant gave an average percent solids in the underflow of 49%. This was supported by Outokumpu testwork results which reported a maximum underflow density of 46% solids, at a rise rate 3.81 m/h and an overflow clarity of 72 ppm solids.

Observations

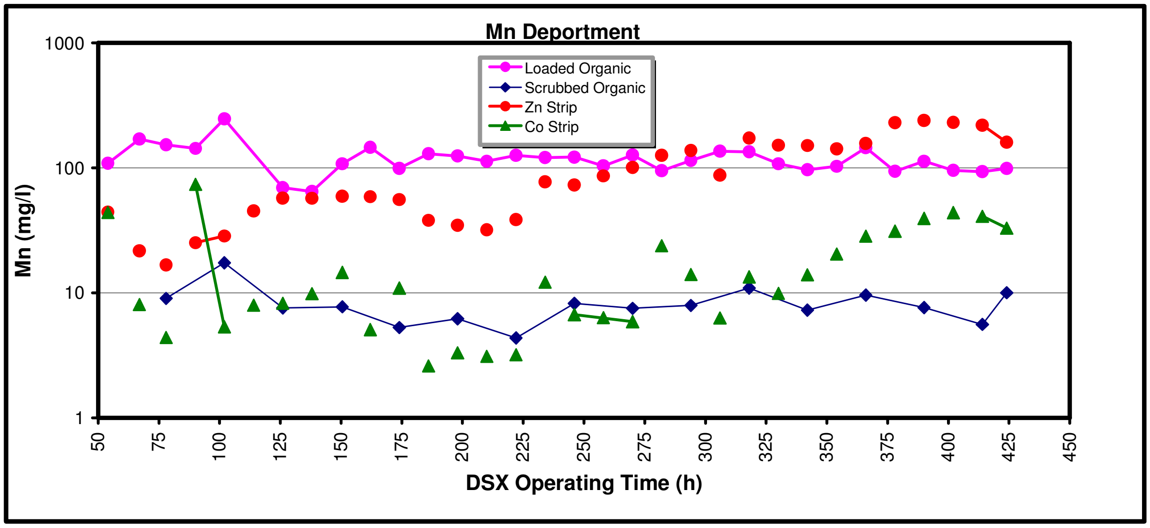
- In general the iron precipitate settled and filtered easily and gave a clearer overflow at higher temperatures. The initial operating temperature of 60°C was dropped incrementally to 45°C over a period of two days from 15th June. However, this resulted in poorer settling performance, reduced overflow clarity and poorer filtration characteristics and a decision was taken to run the circuit at 50°C for the remainder of the campaign.
- The occasional use of hydrogen peroxide to “trim” the remaining ferrous iron did not cause problems with the DSX extractant during the campaign.
- A combination of coagulant (Magnafloc 368) and flocculant (Magnafloc 155) was found to best flocculate the iron removal thickener feed stream.

CSIRO Direct Solvent Extraction (DSX)

The DSX circuit ran trouble-free for the 425 hours of the campaign. Crud was formed in the extraction stages but did not adversely affect the physical or chemical behaviour of the circuit. The crud is likely to have been caused by particulates or precipitates in the feed stream and was not formed by products of organic degradation as was the case in the first pilot plant campaign in November 2004.

None of the black “manganese crud” of 2004 was evident and this is attributed to the excellent control of pH that ensured low manganese loading on the organic at approximately 100 mg/L throughout the campaign.

Figure 26: Manganese Loading On DSX Organic Phase



METAL EXTRACTION

The DSX extractions for copper, cobalt, and zinc are shown in Table 27.

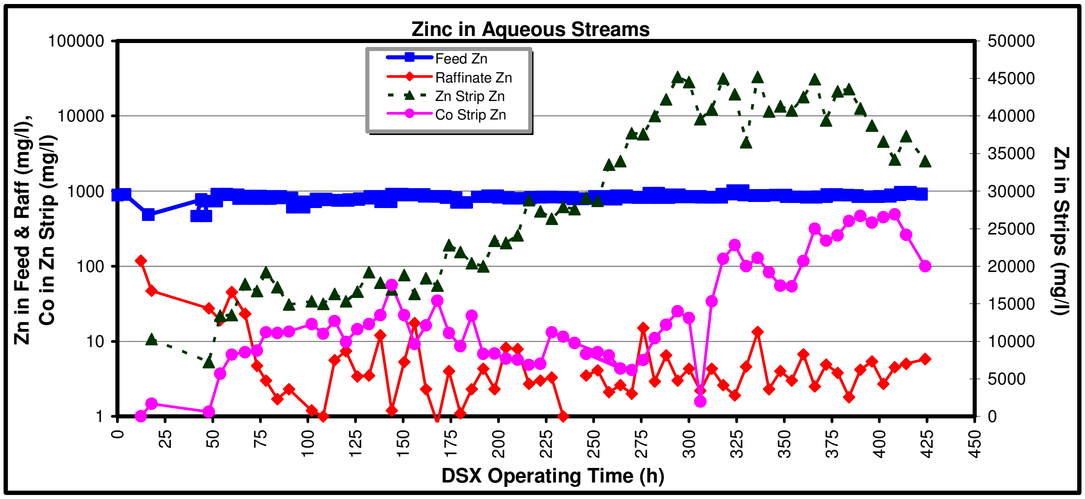
Table 27: DSX Metals Extractions

Description	Unit	Cu	Co	Zn
Extraction	%	99.57	99.48	99.03

ZINC SELECTIVE STRIP

The graph below shows that a zinc tenor in the zinc strip liquor of 40 g/L to 45 g/L can be maintained while keeping a Zn:Co ratio of approximately 50:1.

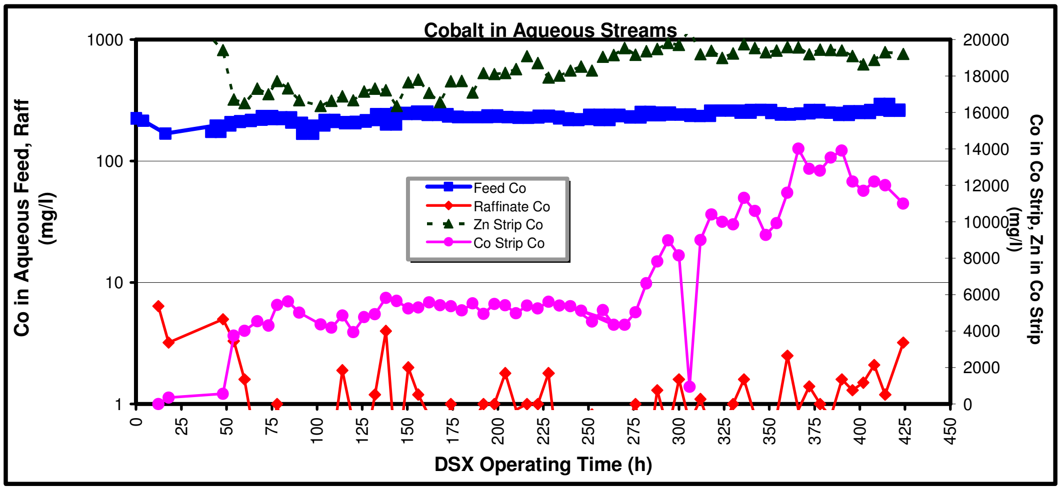
Figure 27: Zinc in Solution vs. DSX Operating Time



BULK STRIP

The graph below shows the cobalt and zinc tenors in the bulk strip liquor. Cobalt concentrations in excess of 10 g/L were attained while maintaining a Zn:Co ratio of 1.6:1.

Figure 28: Cobalt in Solution vs. DSX Operating Time



Observations

- Excellent pH control in extraction was achieved with 100 g/L sodium carbonate.

MANGANESE CARBONATE

Manganese carbonate was precipitated from the DSX raffinate by the addition of 150 g/L sodium carbonate slurry. The sodium carbonate was added at 80% of stoichiometric requirements to minimize the co-precipitation of calcium and magnesium. The table below shows the average results of 19 of the 21 batches produced. Intermittent overdosing of the sodium carbonate resulted in some low manganese values but in general values of 45% or greater were achieved.

Table 28: Manganese Carbonate Product Average Assay Values

Element	Value	Units
Mn	44.2	%
Ca	1.1	%
Mg	0.2	%
Al	0.4	%
Fe	173	g/t
Co	29	g/t
Zn	201	g/t
Al	0.4	g/t
Ni	365	g/t
Si	443	g/t
Cu	< 5	g/t
Cd	< 5	g/t

Observations:

- The pale pink manganese carbonate precipitate settled, filtered and washed well.
- Seeding of the feed from 1.4% to 5.8% solids with recycled underflow led to better flocculation, improved underflow percent solids (62% to 67%) and clearer overflows (47 on the clarity wedge).

Reagent Consumption

Consumption (kg of reagent per tonne of dry feed) of the key reagents in the leach and CCD circuits for various pH conditions is shown below.

Table 29: Summary of Leach and CCD Reagent Consumptions

Condition	RL2 ORP	H_2SO_4 Rate (kg/t)	SO_2 Rate (kg/t)	HAC Solids Rate (kg/t)	Flocculant Rate (kg/t)
OL2 pH = 1.7	399	228	81	69	0.53
OL2 pH = 1.5	397	228	71	73	0.40
OL2 pH = 1.4	399	330	129	116	0.46
Start up	427	496	144	362	0.55
Overall average	404	311	106	138	0.47

15.2 Process Plant Design

This section remains largely unchanged from the previous document but does contain some fundamental changes to the flowsheet and the process design as a result of the knowledge and experience gained from the Fully Integrated Pilot Campaign.

15.2.1 Introduction

The proposed treatment route for the Boléo ore consists of a two-stage, whole of ore, sulphuric acid and sulphur dioxide leach followed by solid-liquid separation in a counter current decantation circuit prior to solution purification and metal concentration using solvent extraction technologies. Cobalt and copper will be electrowon to produce high purity products for export to global metal markets. Zinc will be recovered in a granulator as a zinc sulphate crystal suitable for incorporation in animal feed.

Sulphuric acid is manufactured on the site in a stand-alone acid producing facility, employing sulphur as the main feedstock. Power is produced by harnessing the energy in the steam produced during the acid production process to generate power in a so-called 'co-generation' facility consisting of a steam turbine and generator. Limestone, which is available on the mine site, is crushed in the field and milled on site to provide for plant bulk neutralization duties.

Revision I of the schematic flow diagrams summarizing the proposed plant processing route is included below.

Figure 29: Schematic Flow Diagram Sheet 1

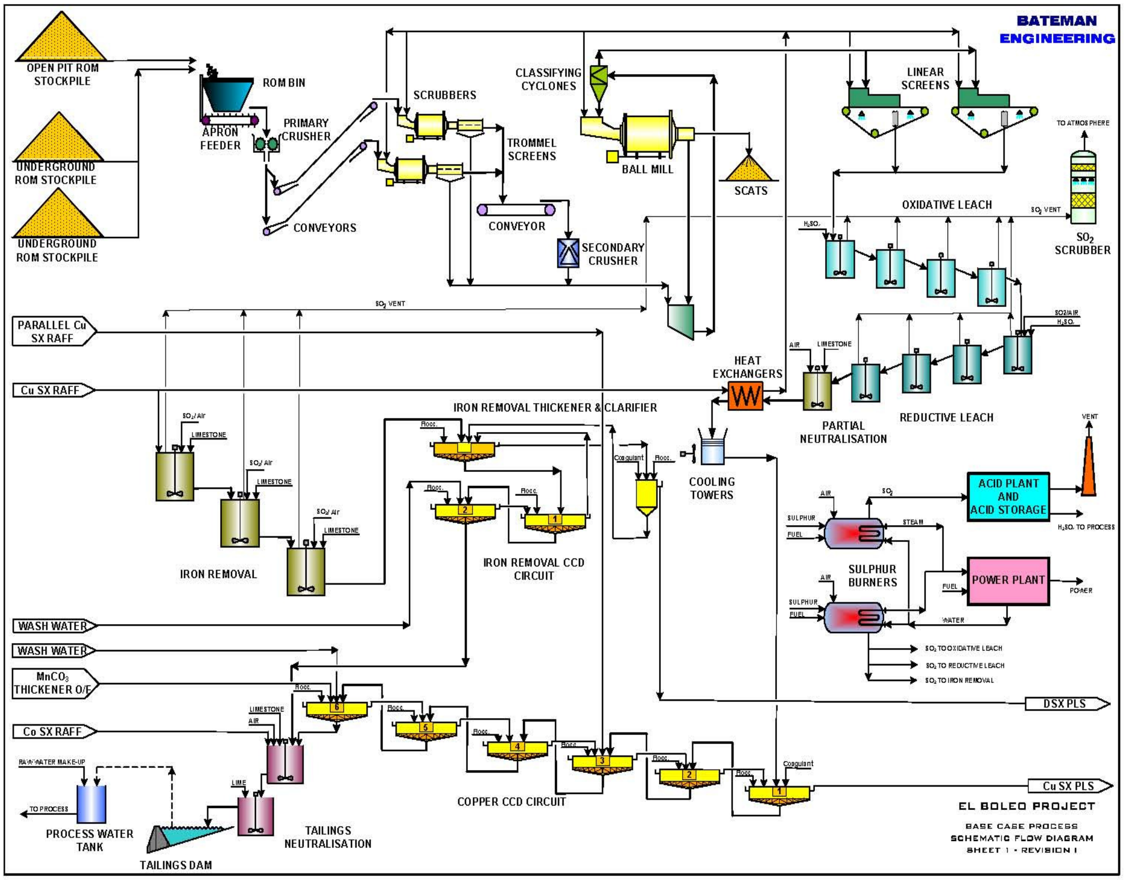


Figure 30: Schematic Flow Diagram Sheet 2

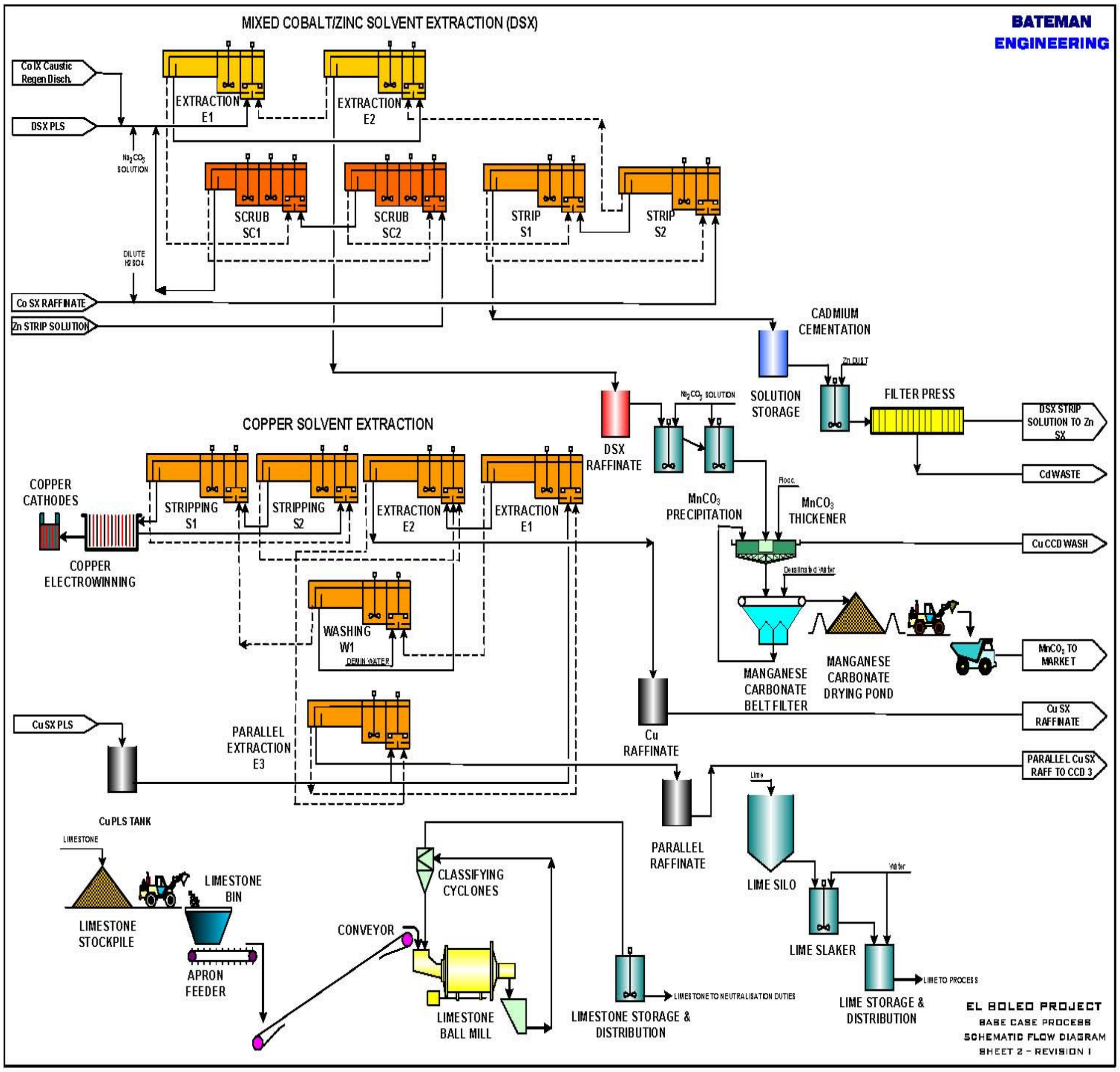
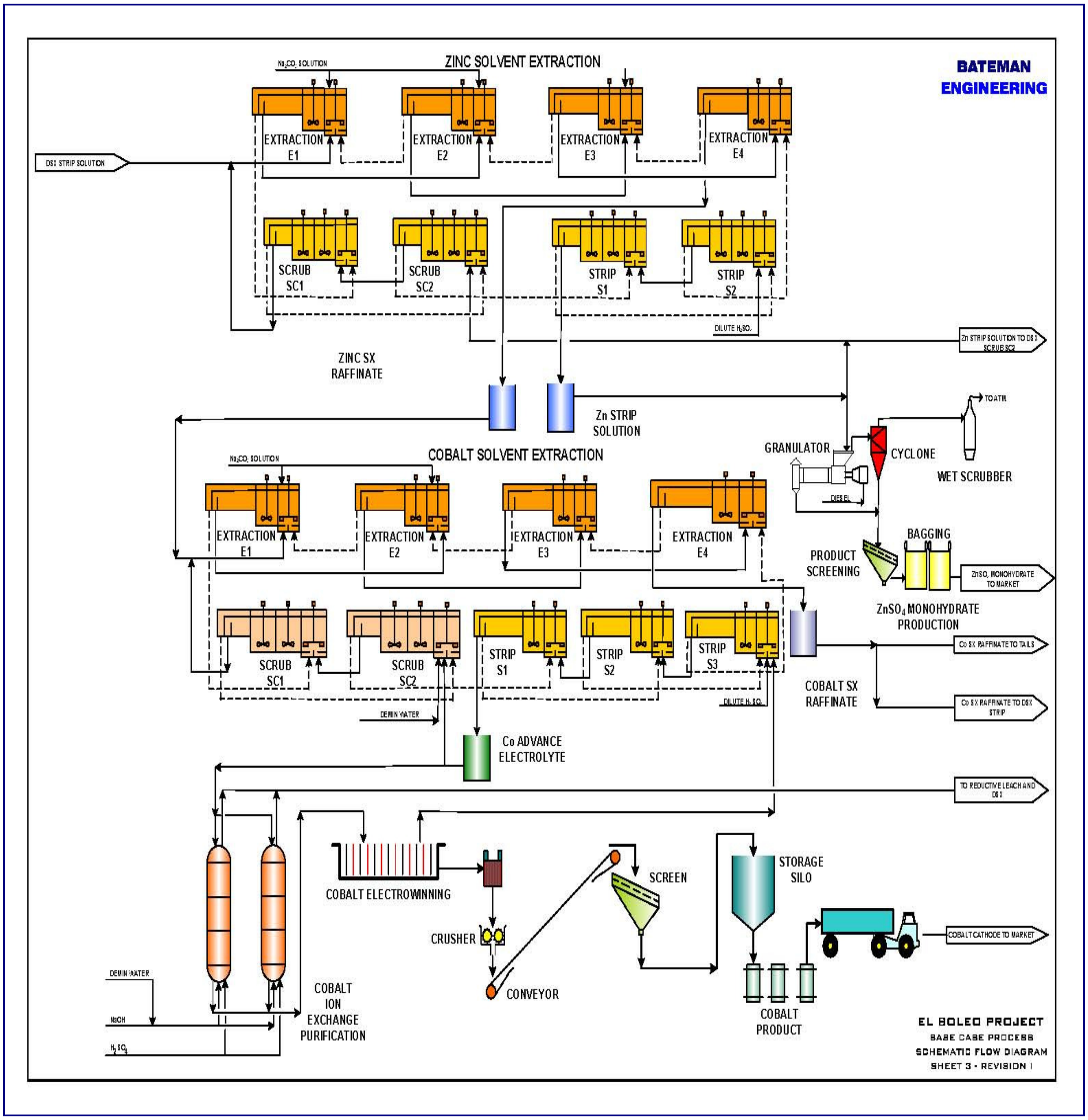


Figure 31: Schematic Flow Diagram Sheet 3



15.2.2 *PROCESS PLANT DESCRIPTION*

The proposed process plant consists of the following main areas:

Primary Crushing – ROM ore is crushed through a single-stage, toothed roll crusher selected specifically to maintain high availabilities on the high clay containing Boléo ore types.

Scrubbing, Screening, Secondary Crushing and Milling – The crushed ROM is treated in two scrubbers designed to slurry the high clay component of the ore. The slurry exiting the scrubber is screened. The remaining coarse ore fraction is crushed in a secondary crushing operation and reports via a pump box to a cyclone cluster for classification. Screen underflow is also cycloned to remove the fine sized material. The coarse fraction from the cyclone is returned to the ball mill for further size reduction. Slurrying and milling take place in acidic raffinate returned from copper solvent extraction. Ceramic balls are used as grinding media.

Oxidative Leach – The ground ore is leached in sulphuric acid and recycled copper SX raffinate to leach (dissolve) the copper, manganese and zinc containing minerals and produce a solution rich in copper and zinc sulphate solution. A small portion of the cobalt is leached at this stage too.

Reductive Leach – The slurry is subjected to a reductive leach where SO_2 gas is bubbled through the slurry to facilitate the leaching of the remaining cobalt minerals. At the end of the reductive leach typically 90% of the copper and 80% of the cobalt has been leached into solution.

Partial Neutralisation – The slurry overflowing the reductive leaching circuit will contain residual acid which will be partially neutralised to ensure optimal copper recovery in the downstream copper solvent extraction circuit. Partial neutralization is achieved via the oxidation of ferrous to ferric ions using blower air and the addition of a relatively small quantity of Boléo Carbonate.

Slurry Cooling – Partially neutralised slurry cooling takes place in a heat exchanger step followed by further cooling in a forced draft cooling tower. Cooling of the slurry is required to facilitate the performance of downstream operations. Heat exchangers are used to recover heat into raffinate for subsequent use elsewhere in the circuit.

Counter-Current Decantation (CCD) Circuit – Cooled leach residue slurry discharged from the cooling tower is washed with sea water in a six stage CCD circuit to recover soluble copper, zinc and cobalt. Deep bed, high rate thickeners are utilized. Thickener underflow is pumped from the last CCD stage to a tailings neutralization facility. Thickener overflow from the first CCD stage contains the bulk of the leached copper, zinc and cobalt and is termed pregnant leach solution or PLS. The PLS is transferred to the copper SX circuit for copper extraction and concentration prior to electrowinning.

Copper Solvent Extraction (SX) – The copper solvent extraction circuit consists of a 2-stage extract, 1-stage wash, 1-stage parallel extract and 2-stage strip operation. Copper is recovered from the PLS into an organic liquor known as an extractant. The PLS aqueous solution, now stripped of copper and known as raffinate (containing residual copper, zinc, cobalt, iron and manganese, all as sulphate species) is split between the scrubbing and milling circuit, (where it is used as dilution and make-up solution in the comminution circuit) and the iron removal process where levels of certain metallic species, principally iron and aluminium, are reduced prior to further concentration of the zinc and the cobalt in the DSX ® operation.

The loaded organic stream from the copper extraction circuit is 'stripped' with spent copper electrolyte from the copper electrowinning operation, producing loaded copper electrolyte and stripped organic, the latter being recycled to copper SX. Loaded copper electrolyte is pumped, via multi media filters to remove carry-over organic and entrained solids, to Copper Electrowinning where copper is recovered as LME grade cathode.

Electrowinning (EW) – Copper metal is electro-deposited from filtered loaded electrolyte, onto stainless steel blanks, known as cathodes, over a nominal 7-day cycle. Cathodes are harvested via an automated stripping machine on a semi-continuous basis.

An overhead crane will lift out sets of cathodes from the EW cells. The cathodes are then washed, and fed on a chain conveyor type system to the automated stripping machine. Copper cathode is automatically 'stripped' from the stainless steel blanks, sampled, weighed and packaged for sale. Spent copper electrolyte is eventually returned to the copper SX circuit.

Iron Removal – The copper raffinate from the copper SX will contain residual copper, zinc, cobalt, iron and manganese. This step has been designed to remove residual acid and iron from solution. Air is sparged through the solution to convert the ferrous ion to ferric ion and, by increasing the pH using limestone, encourage the formation of iron hydroxide precipitates. The products of the iron removal process, primarily goethite, will be thickened and washed in a small CCD circuit in order to recover as much process liquor and dissolved metal values as possible before being diverted to tailings neutralization.

Thickener overflow reports to a pinned bed clarifier for final polishing ahead of the DSX ® circuit. This solution is at a pH of 4.5 in preparation for the DSX ® extraction step.

Tailings Neutralization and Disposal – Thickened tailings from the main CCD circuit and thickened underflow from the iron removal CCD circuit are treated in a neutralization operation where limestone is added to neutralise excess acid precipitate metal salts contained in the tailings solution. Subsequently lime will be added to the slurry in a second tank to raise the pH to that required for disposal.

The slurry discharged from this process is pumped to a tailings dam situated to the west of the plant site known as the Curuglu area.

DSX ® – Copper, zinc and cobalt, contained in solution from the iron removal clarifier overflow, are separated from manganese, magnesium and calcium into an organic extractant phase via the DSX® process. The DSX process employs two extractants known as LIX63i and Versatic 10 to effect this separation. The zinc solvent extraction circuit consists of a 2-stage extract, 2-stage scrub and 2-stage strip operation. The organic phase is scrubbed with aqueous solutions to remove small quantities of impurity from the loaded organic.

The loaded organic – containing the cobalt and the zinc - is then stripped and concentrated into an acidic stream for further treatment. This stream reports to the zinc solvent extraction circuit after a cadmium reduction step.

Cadmium Removal – the DSX strip solution is diverted to a cadmium removal step where zinc dust is used to purify the solution of excess cadmium at a pH of 3.3, enabling the product to meet the quality specifications required for use as an additive to animal feed. The solution is filtered with the filter cake being diverted to tailings and the filtrate being passed to a zinc SX step.

Zinc Solvent Extraction – The purified bulk strip solution from DSX ® becomes the feed to a zinc removal solvent extraction operation, prior to cobalt SX and cobalt EW. The zinc solvent extraction circuit consists of a conventional 4-stage extract, 2-stage scrub and 2-stage strip operation. Strip liquor, rich in zinc, reports to two places in the flowsheet. A small proportion is returned to the DSX ® as scrub liquor and the remainder the feed to a facility producing zinc sulphate monohydrate crystals in a fluidized bed granulator.

Cobalt Solvent Extraction, Ion-Exchange and Electrowinning – Raffinate from the zinc secondary SX circuit, now containing only cobalt and very small quantities of zinc, nickel, and iron, will report to the cobalt SX operation. The circuit consists of a 4-stage extract, 2-stage scrub and 3-stage strip operation. Strip liquor from the cobalt SX operation is further purified through ion exchange where highly selective resins remove extraneous metal ions prior to electrowinning to produce cobalt metal. Cobalt electrowinning requires high purity feed solutions; the IX circuit provides a final 'catch-all' scavenging process. Electrowon cobalt is crushed, screened and dispatched in drums as a premium metal product.

Zinc Sulphate Monohydrate Production – Zinc strip solution reports to a fluidised bed granulator where heat is applied to the solution via the use of hot air to affect the crystallisation process. Particles are 'grown' to the appropriate size and shape, cooled, screened and packaged in a stand-alone production facility. The air used in the process is heated by combustion of diesel.

Sulphuric Acid Generation and SO_2 Production – Sulphuric acid and sulphur dioxide are produced in a 'stand alone' facility which consists of a number of unit operations including:

- liquid sulphur storage
- air drying
- sulphur burning
- gas conversion (employing catalysts)
- economizers and heat exchangers
- gas scrubbing and acid storage
- liquid SO_2 production and storage.

The SO_2 is sparged directly into the slurry in the reductive leach operation.

Power Production – Steam exported from the sulphuric acid and SO_2 gas plant is used to generate electricity via the use of a gas driven turbine and power generation system. The power generated via this co-generation system exceeds the overall mining and plant operational requirements at design throughputs, making the Boléo Operation self sufficient in electricity. Any power shortfall experienced at elevated throughputs can be made up by the use of standby diesel powered generating sets.

Limestone Milling – A limestone milling circuit is required to produce ground limestone slurry for various bulk neutralization duties throughout the circuit namely Partial Neutralisation, Tailings Neutralisation and Iron Removal. Limestone is mined on the Boléo Reserve, crushed in the field, stockpiled, milled and then circulated through the plant on a ring main system.

Process Water – Seawater will be employed throughout the circuit as process water.

Cooling Water – Seawater will be employed throughout the circuit as cooling water.

Desalinated Water – Desalinated water, required for particular duties throughout the plant, will be produced in a Multiple Effect Distillation type desalination facility. A small quantity of this water production will be diverted for use throughout the plant and mining operations as potable water.

Tailings Dam – The Boléo Creek tailings dam, located in the Curuglu area, is designed to accommodate approximately 20 years worth of plant tailings. The dam wall will be raised in 3 stages over the life of the mine. The first dam wall will accommodate approximately 5 to 7 years worth of plant tailings production at design feedrates.

16 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

16.1 3D GEOLOGICAL INTERPRETATION

The geology of the Boléo deposit is well known (see sections 7, 8, and 9 above) and has been extensively studied. Mineralization is effectively confined to seven shallowly dipping layers, referred to as 'mantos'. A large number of faults have been identified from either surface mapping or mapping historical workings. These faults play an important role with respect to local continuity of both geology and grade.

To enable the resource estimates to be accommodated accurately within the geological framework the existing geological interpretation was used to construct a 3D framework comprising faults and Manto footwall surfaces.

Digital geological data supplied by MMB comprises:

- Manto footwall contours (for manto 1, 2, 3aa, and 3), in plan as *.dxf files
- Traces of faults in plan, effectively as lines of intersection on Manto 1 and 3 footwall surfaces, as *.dxf files
- Erosional windows of mantos, as *.dxf files
- Approximately 230 cross-sections (each 2km in width), amounting to 460 line kilometres.

The geological interpretation was built in the software package MineSight 3D.

16.1.1 FAULT SURFACES

To build the fault framework, faults were firstly digitised as 'polylines' from each cross-section. No nomenclature or labelling exists to correlate faults from section to section, so this was achieved by using the plans of fault line of intersections with Mantos 1 and 3 as guide to connect the digitised polylines correctly. The process and results are depicted below in Figures 32 to 35.

Figure 32: Fault Framework in Section – Detail of Cross-Sections (900N) as Supplied by MMB

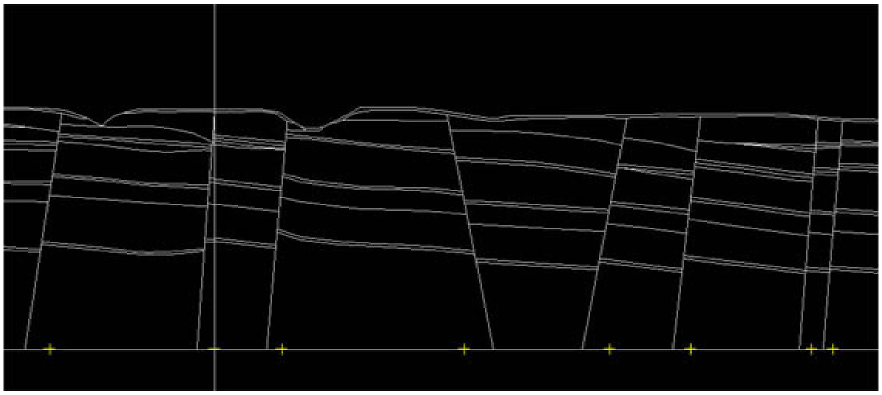


Figure 33: Digitised Fault Traces – Faults Digitized (dashed red lines) Following Cross-Sectional Interpretation (900N)

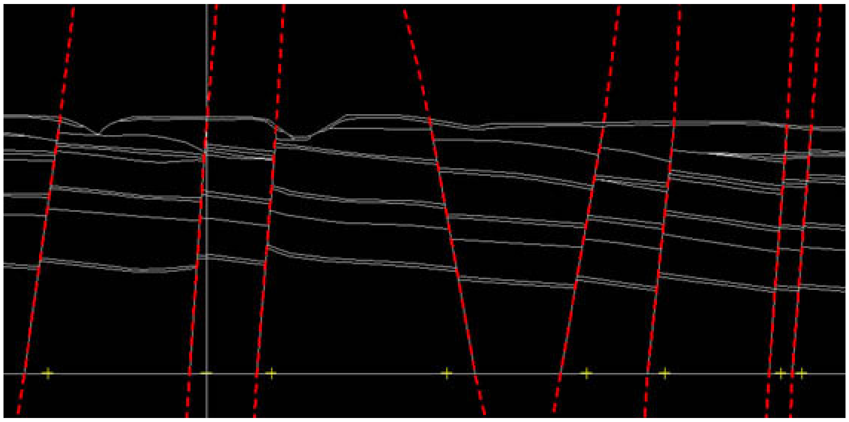
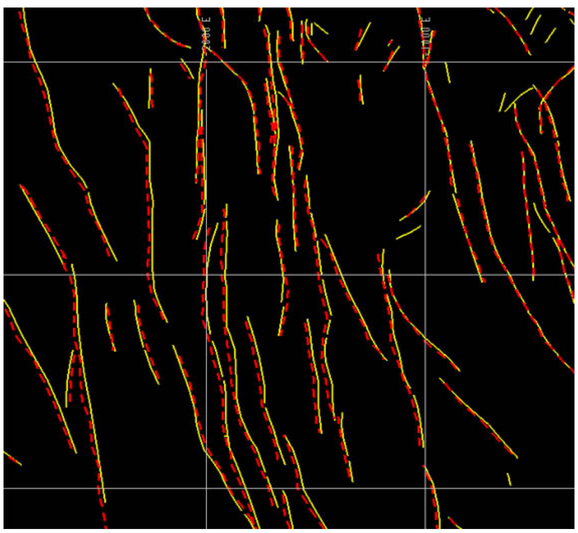
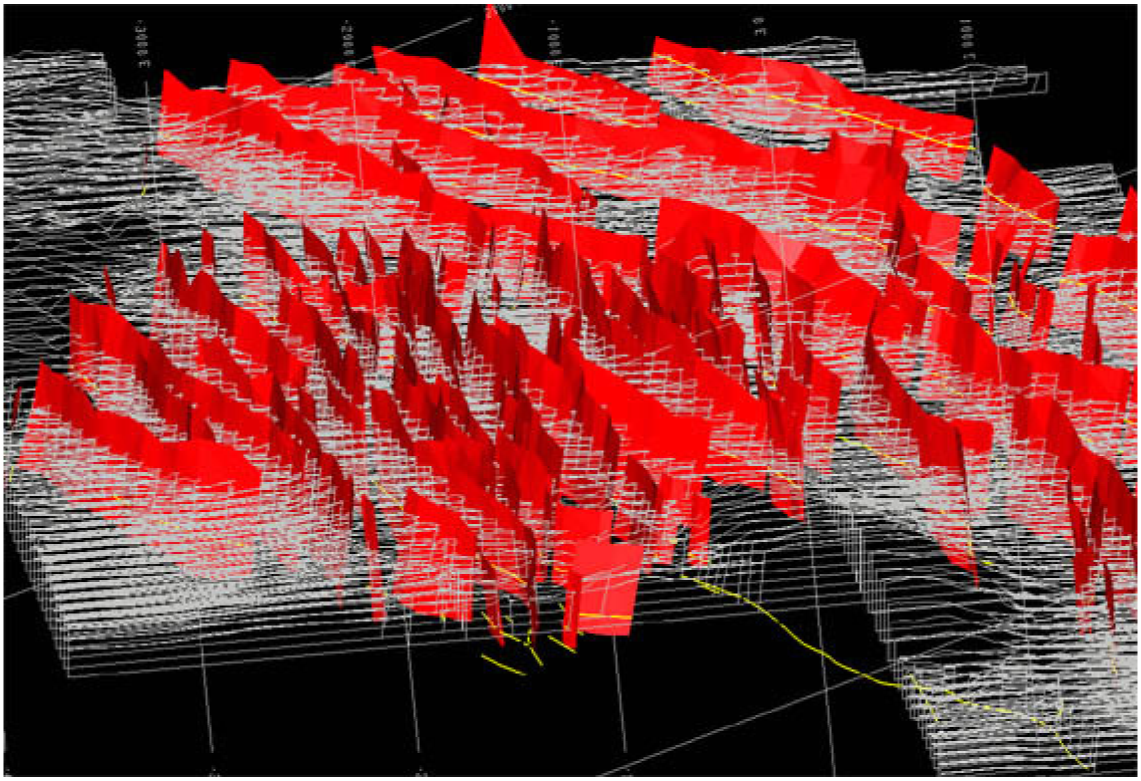


Figure 34: Correlation of Fault Traces



Note: Digitized Polylines of Faults (red) Joined from Section to Section using Fault Lines of Intersection on Manto 3 (yellow) as a guide

Figure 35: Final 3D Fault Framework with Mintec Cross-Sections



16.1.2 *MANTO FOOTWALL SURFACES*

The Manto footwall surfaces were more complicated to define as they had to fit with the fault framework described above.

To create these footwall surfaces, each drill hole was tagged with a set of points corresponding to the lowest logged occurrence of each manto. For each Manto, these points were triangulated to form a continuous 3D surface (Figure 36).

The resulting 3D surface was sliced along the same 100 section lines as the Mintec geological interpretation (Figure 37), these slice lines were then superimposed over the Mintec sections and 'dragged' and 'snapped' to the digitised fault framework (Figure 38).

Figure 36: Triangulated Manto Surface – (Manto 3, oblique 60°)

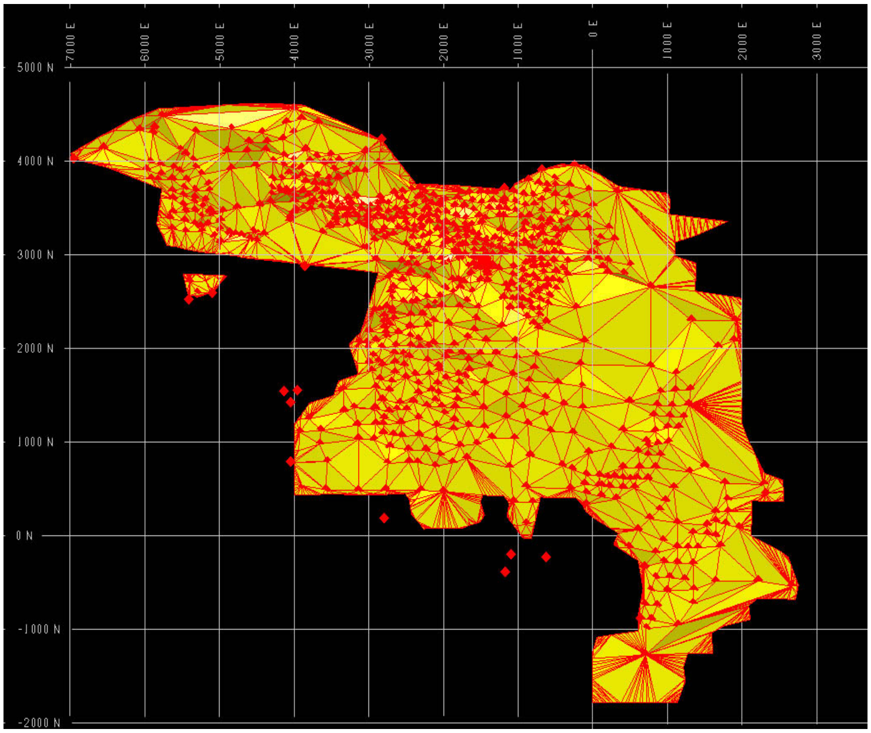


Figure 37: Triangulated Manto Surface Sliced Along Section Lines – (Manto 3, oblique 60°)

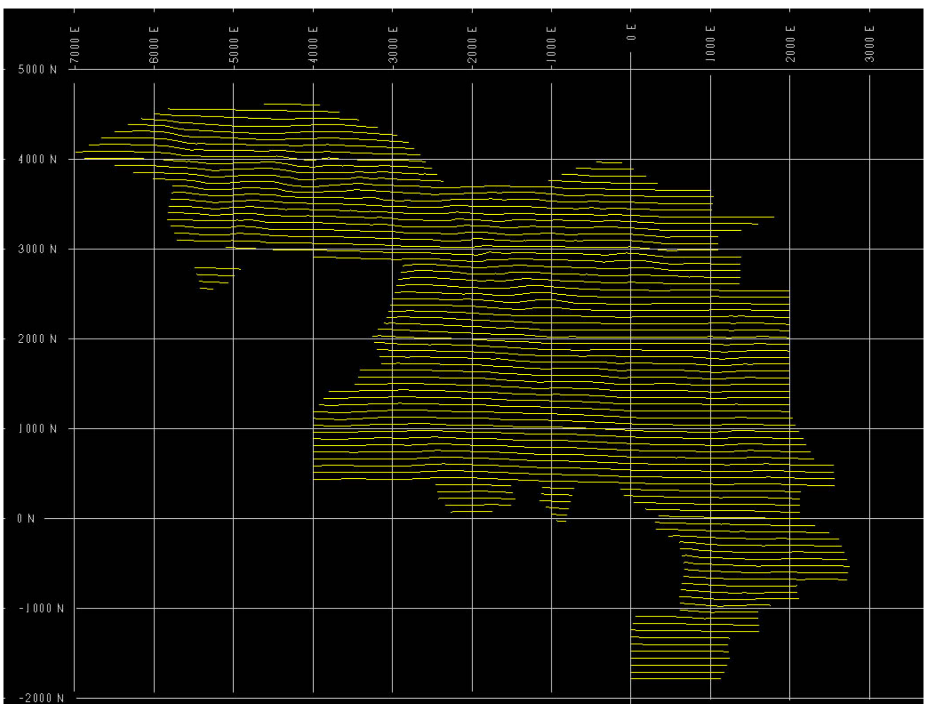
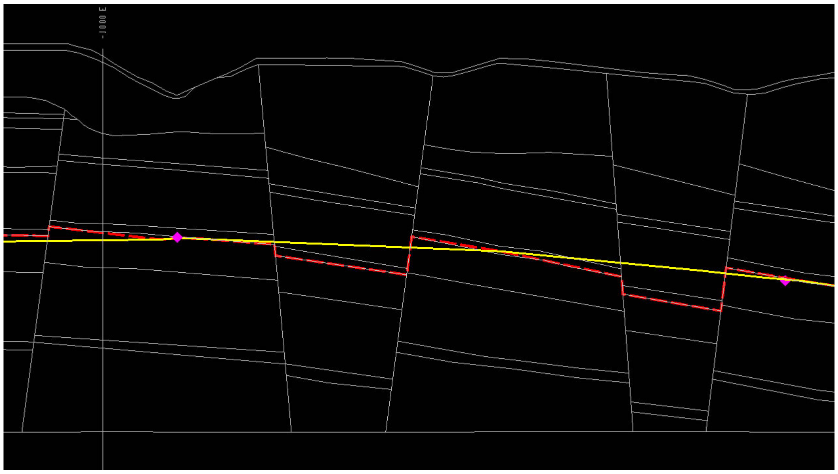


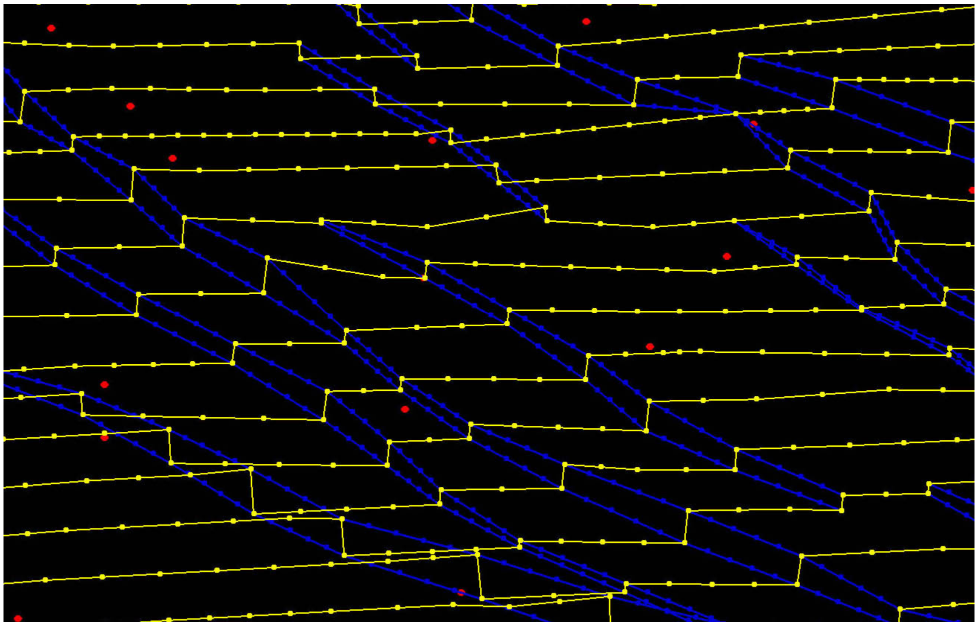
Figure 38: Snapping Footwall Surface to Faults



Note: Triangulated Surface Slices (yellow), New Surface Slices after Snapping to Faults (dashed Red)

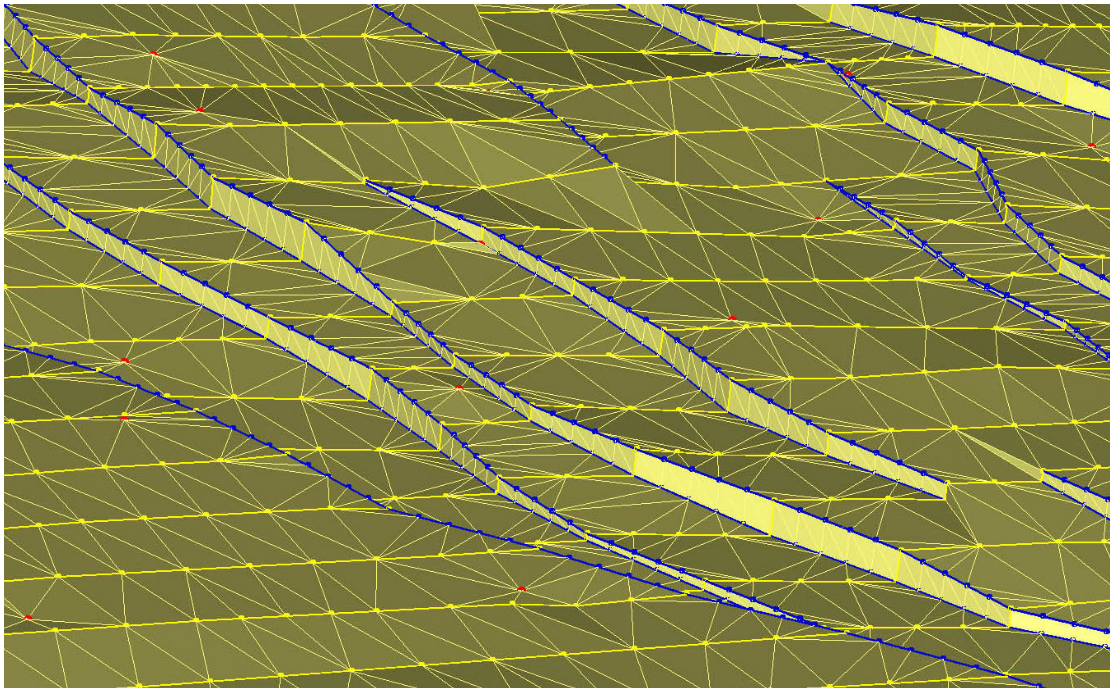
A new manto surface was then created from the resulting profiles that honour the faults, the original drill hole tag points and tie lines along the upper and lower intersections of each fault with the manto surface. The latter are required to force the triangulation to precisely honour the fault offsets (Figures 39 and 40).

Figure 39: Re-Triangulation of Manto Surface Components



Note: New Fault Snapped Section Lines (yellow), Fault Tie Lines (blue) and DH tag Points (red)

Figure 40: Re-Triangulation of Manto Surface – Detail of Final Result



The resulting triangulation is a new manto footwall surface (Figure 41) that honours the base of the manto as defined by each drill hole intersection but also honours the faults as defined on the geological cross-sections (Figure 42).

Figure 41: Re-Triangulation of Manto Surface – Full View (Manto 3, Oblique 60°)

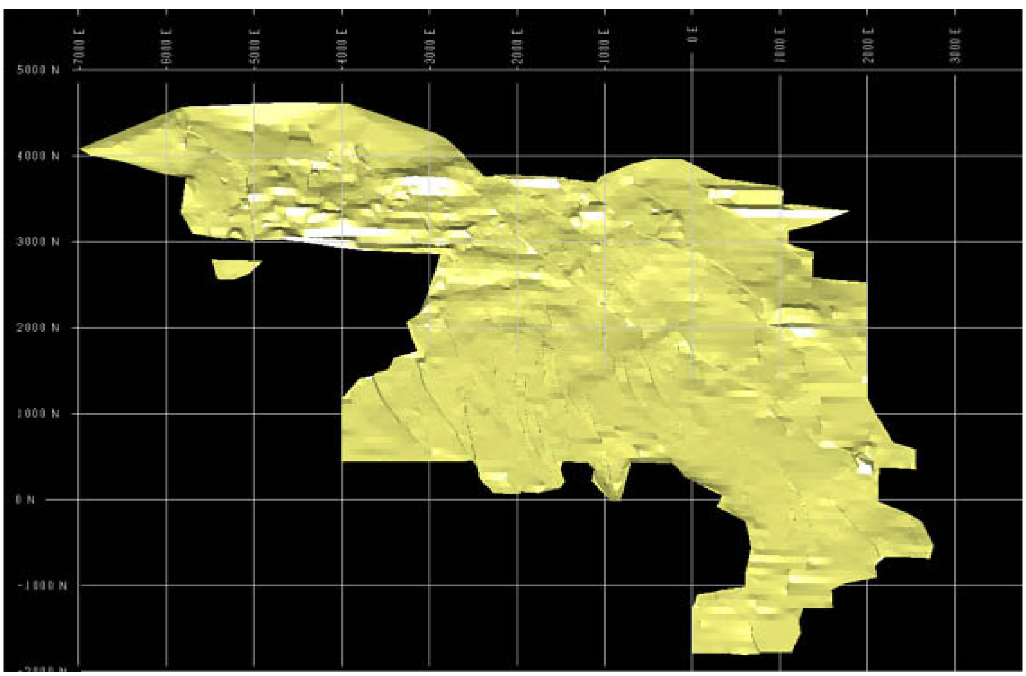
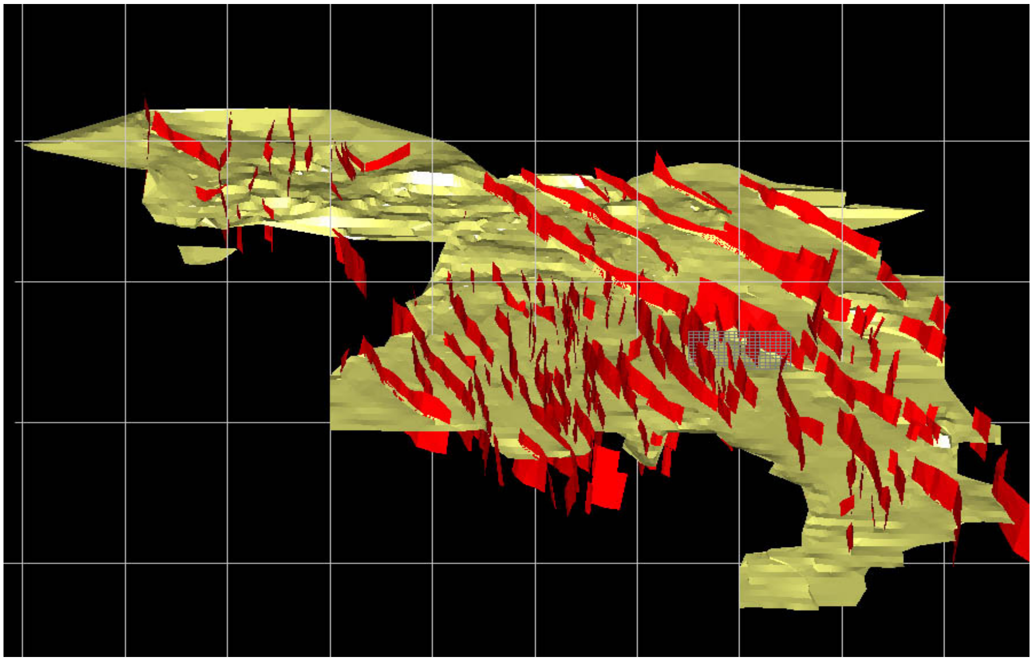


Figure 42: Re-Triangulation of Manto Surface – Full View with Faults (Manto 3, Oblique 60°)



The same process was followed for all seven Manto footwall surfaces.

16.2 RESOURCE ESTIMATION TECHNIQUE

16.2.1 BLOCK MODEL VS. GRIDDED SEAM MODEL

The Boléo copper-cobalt resource will be mined by a combination of open cut and underground methods. Consequently, different resource models need to be built that reflect these different requirements.

In 2005 3D block models were produced for the purposes of open cut optimisation. This approach was appropriate and has been used once again in the latest studies to produce models suitable for open cut optimisation.

As predicted the 3D block models proved to be less useful for underground mine design. The planned underground mining method is constrained by minimum and maximum height parameters and it is not possible to exercise the same degree of mining selectivity using a 3D block model. Consequently a seam model approach to resource modelling has been used to produce models for underground mine design and scheduling.

The predominant mining method used for each manto will vary. Mantos 1, 2 and 4 will be mined by underground methods only. Manto 0, 3aa, 3a will be mined by open cut methods and Manto 3 will be mined by a combination of both open cut and underground methods.

16.2.2 DATA DOMAINS AND FLAT MODELS

The definition of data domains is required to limit data used for grade estimation to the data within the area being considered. The most obvious control that needs to be applied at Boléo is to keep separate the data and estimation process for each Manto. Data for each manto was loaded into different data files and individual models were created and the estimations process was carried out separately for each manto.

The second controlling influence on the data that needs to be considered is the effect of the faulting. Faulting is predominantly post mineralization and as such has resulted in fragmentation of once continuous Manto units into numerous blocks, each bounded by faults. In plan, the continuity of the mantos remains. In section, continuity before a fault termination is significantly reduced to distances as short as 100 m.

Previous resource estimates have used faults as domain boundaries with ore blocks and data restricted to specific fault blocks. The effect of this, where faulting is dense, is to severely limit data available for use in the grade estimation process. To estimate grade of a particular block, Kriging techniques capture data by means of a search ellipse (with dimensions defined by the user) with its centre at the block mid-point. All data that falls with this search ellipse are used in the estimation process.

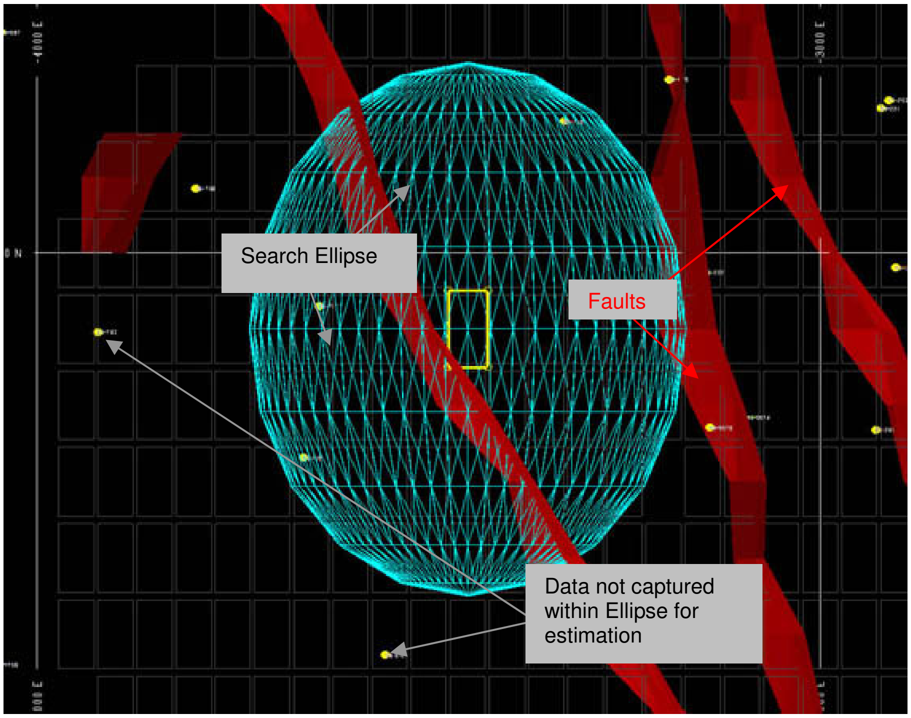
In Figure 43, a search ellipse is shown in plan and in section. In plan the ellipse, centred over a highlighted block, appears to capture three drill holes that fall within it (yellow points). However, when viewed in section, the block, which is adjacent to but on the up-throw side of a fault, has its search ellipse well above the data on the down-throw side and consequently no data will be captured from that side of the fault.

Because the faulting at Boléo is predominantly post mineralization, the two segments of Manto now off-set by faulting would originally have been juxtaposed. This being the case, the inclusion of the holes on the down-throw side of the fault would be justified. It is not practical to capture data significantly off-set by faults simply by means of modifying the search ellipse parameters (i.e., increasing the vertical search distance) as this would require very large vertical searches that may also result in too much data from higher levels in the manto being captured as well.

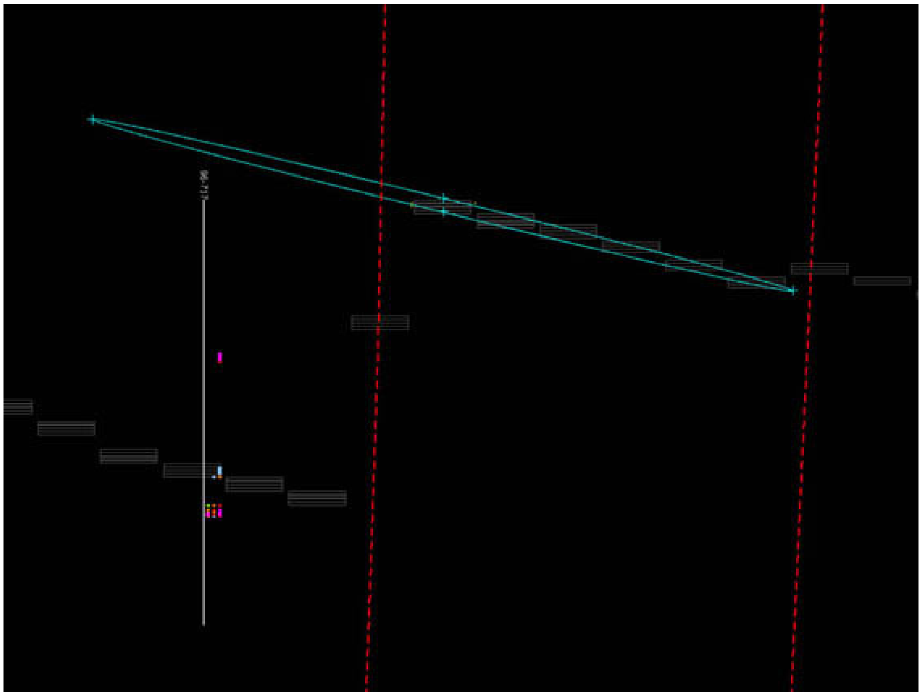
To achieve the desired effect the base of each manto intersection in each drill hole was set at the same (fixed) elevation. The base of the model was also set at the same elevation. This step removes the effect of the faulting and returns drill hole intersections to their pre-faulting position, relative to adjacent holes. It is then possible to consider all the data for each Manto as a single mineralized domain.

The effect of this is to create a resource model that has the same EW and NS lateral extents but exists in a reduced or flat vertical space. Aligning each hole at the same level not only removes the faulting off-sets but also the easterly dip of the mantos. This has an additional advantage in that each block model has a significantly smaller vertical extent as each model only has to be slightly thicker than the maximum Manto thickness. These models are referred to as flat models.

Figure 43: Search Ellipses – Plan & Section Views



Plan View



Section View

All grade modelling steps for both 3D block models and seam models was carried out using the flat models. A final step was required to convert the flat model block elevations back into true space.

This process was achieved by:

- Generating a 2D gridded surface from the already created Manto footwall surfaces. These surfaces were gridded on 25 m x 25 m centres such that a sub-set of grid points were coincident with block mid-points.
- Deleting all points from the gridded surface file that were not coincident with block centroid points.
- Further editing the reduced gridded surface file, so that the elevation was set as the mid-point elevation of the lowest blocks in the flat model. The base of the flat block models is set at zero so the mid-point elevation is 0.5 m.
- Retain the true footwall elevation values to be simply imported into the flat model as a separate model item.
- Exporting (after grade estimation was completed) block coordinates and the item containing the true footwall elevation, from all blocks with estimates, into a spreadsheet where the true mid-point elevation of each block stacked above a footwall block was calculated by adding 1m progressively to the footwall elevation for each higher block.
- Importing true model elevations for each block as a new item in the flat models.

All blocks now have both flat and true mid-point elevations.

16.3 3D Block Models

16.3.1 *Composite Length and Block Dimensions*

In the resource estimation process when original assay samples are of varying lengths it is necessary to composite these into equal lengths, because each original assay value is representative of different proportions of a mineralized interval. Compositing results in an assay population where each value or composite has equal weight.

At Boléo original sample lengths vary considerably, from as little as 10 cm to >1.5 m. The average sample length for all mantos is about 0.95 m. A composite of 1m was considered appropriate.

After determining a 1 m composite length, a similar 1m block height for the 3D model is sensible. Ideally, model blocks should be greater than the composite interval, but thicker blocks would not be suitable as the manto thickness is generally <5 m.

The drill hole spacing and, therefore, data density, in plan, is used to decide the block dimensions. The closest drill hole spacing at Boléo, over a meaningful area, is approximately 140 m x 140 m in the Saturno – Arroyo Boléo area. Blocks 100 m x 100 m, or larger would be best from a modelling perspective, but large blocks would not fit readily into the framework of faults. Consequently blocks 50 m EW by 100 m NS were chosen.

16.3.2 UNIVARIATE STATISTICS OF DATA COMPOSITES

Univariate statistics and histograms of grade for Cu, Co, and Zn for each Manto are shown in Tables 30, 31, and 32.

Although new 3D block models have not been produced for Manto 1 and 2 the 1m composite statistics are shown for completeness and comparison. Resource models for manto 0, 3aa and 4 have not been updated at this stage, so composite statistics are the same as reported previously.

Coefficients of Variations (CV) are generally less than 2, which indicate the data is not highly skewed and that non-linear estimation techniques, such as Multiple Indicator Kriging, are not warranted. Exceptions to this are the Copper distributions for Mantos 1 and 2, which have CV values of 2.29 and 2.55 respectively, whilst Manto 3a and 4 have CV values close to 2. Cobalt and Zinc CV values are uniformly low.

Ordinary Kriging is considered an appropriate method to use for Cu, Co and Zn, for Manto 0, 3a, 3a, 3 and 4 because:

- Manto 1 and 2 are not reported as 3D block models based on the 1 m composites
- The copper resource is dominated by Manto 3 which has a CV well below 2
- Co and Zn CV values are uniformly low.

Table 30: Univariate Statistics of Assay Composites – Copper

	Manto						
Copper	**0**	**1**	**2**	**3aa**	**3a**	**3**	**4**
Mean	0.02	0.55	0.27	0.53	0.35	.84	0.32
CV	1.9	2.29	2.55	1.3	2.04	1.46	2.0
min	0.002	0.001	0.001	0.001	0.001	0.001	0.001
Q1	0.01	0.01	0.02	0.09	0.02	0.09	0.05
Median	0.01	0.04	0.08	0.20	0.07	0.342	0.14
Q3	0.02	0.43	0.25	0.66	0.34	1.14	0.32
Max	0.29	15.35	13.45	3.96	7.45	14.60	8.53
IQR	0.01	0.42	0.23	0.58	0.32	1.05	0.28
Data	237	775	1,859	127	1,781	4,283	1,312

Table 31: Univariate Statistics of Assay Composites – Cobalt

	Manto						
Cobalt	0	1	2	3aa	3a	3	4
Mean	0.008	0.0375	0.043	0.082	0.068	0.069	0.028
CV	1.0	1.29	1.82	0.8	1.04	0.97	1.0
min	0.001	0.001	0.001	0.007	0.001	0.002	0.001
Q1	0.004	0.007	0.013	0.028	0.023	0.027	0.013
Median	0.006	0.017	0.026	0.064	0.045	0.049	0.020
Q3	0.009	0.047	0.058	0.11	0.083	0.089	0.034
Max	0.079	0.475	2.82	0.281	0.630	1.510	0.464
IQR	0.005	0.040	0.045	0.082	0.060	0.062	0.464
Data	237	775	1,859	127	1,781	4,283	1,312

Table 32: Univariate Statistics of Assay Composites – Zinc

	Manto						
Zinc	0	1	2	3aa	3a	3	4
Mean	0.42	0.81	0.87	0.73	0.56	0.35	0.22
CV	0.9	1.71	1.2	0.9	0.8	1.26	0.8
min	0.01	0.003	0.025	0.06	0.00	0.00	0.00
Q1	0.12	0.27	0.3	0.30	0.26	0.18	0.11
Median	0.29	0.49	0.57	0.61	0.43	0.27	0.16
Q3	0.60	0.86	1.01	0.92	0.71	0.41	0.27
Max	1.73	20	9.28	4.99	5.11	13.76	1.56
IQR	0.49	0.59	0.71	0.62	0.45	0.23	0.17
Data	237	775	1,859	127	1,781	4,283	1,312

16.3.3 SPATIAL CONTINUITY OF GRADE

Variograms of Copper, Cobalt and Zinc were created and modelled using Hellman & Schofield proprietary software "GS3". Variogram analysis utilized data re-aligned for flat models.

Variogram maps for each metal showed very poor structure with no strong directional controls (Figure 44). A weak anisotropy in a NE-SW direction is apparent, particularly for Copper. The range, or distances over which there is a spatial relationship between the grades at two points appear to be relatively short, a few hundred metres, compared to the full extent of the mantos (i.e., several kilometres).

Directional variograms (using trigonometric rather than grid coordinate conventions, i.e., 000 = East, 090 = North) were generated for all data. Modelled variograms for Manto 3 are shown below (Figures 45 to 48).

Variograms were modelled with a relatively low nugget as determined from the well structure down hole (Z direction) variograms. Variogram models for Mantos 3a, 3 and 4 are shown in Table 33. Note that no variograms were possible for Manto 0 and 3aa due to lack of data. Variograms for Manto 1 were used when estimating grade in Manto 0 and similarly variograms for Manto3a used for estimating grade in Manto 3aa.

Figure 44: Manto 3 Variogram Maps

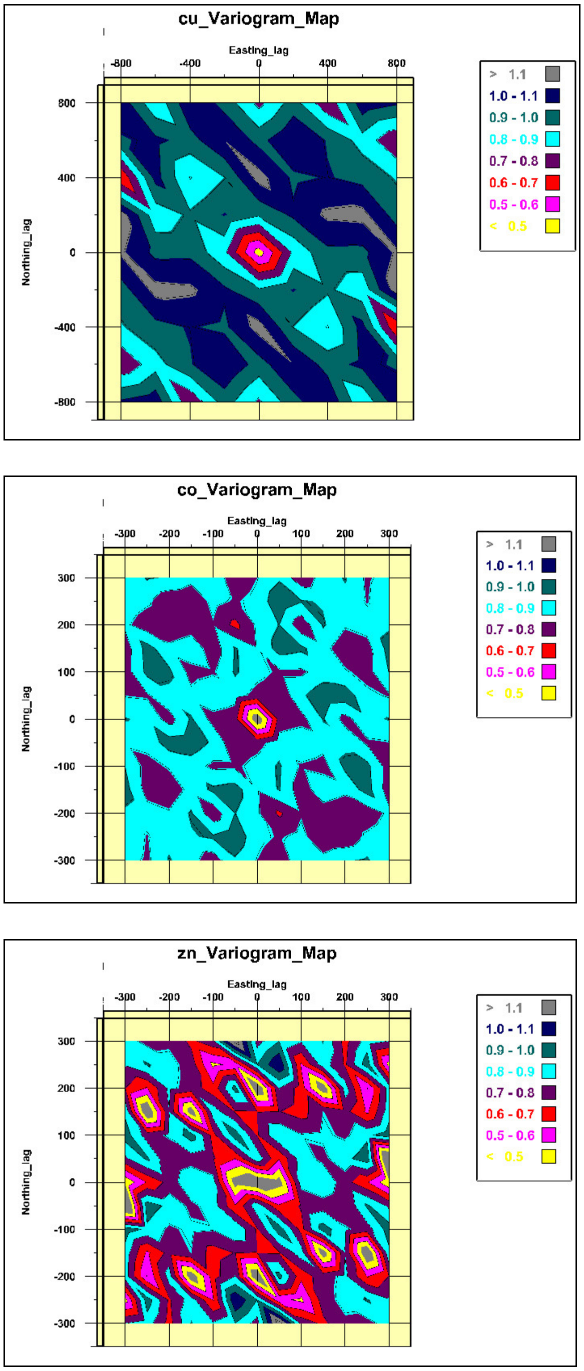


Figure 45: Variograms of Copper – Manto 3 (X, Y, Z directions)

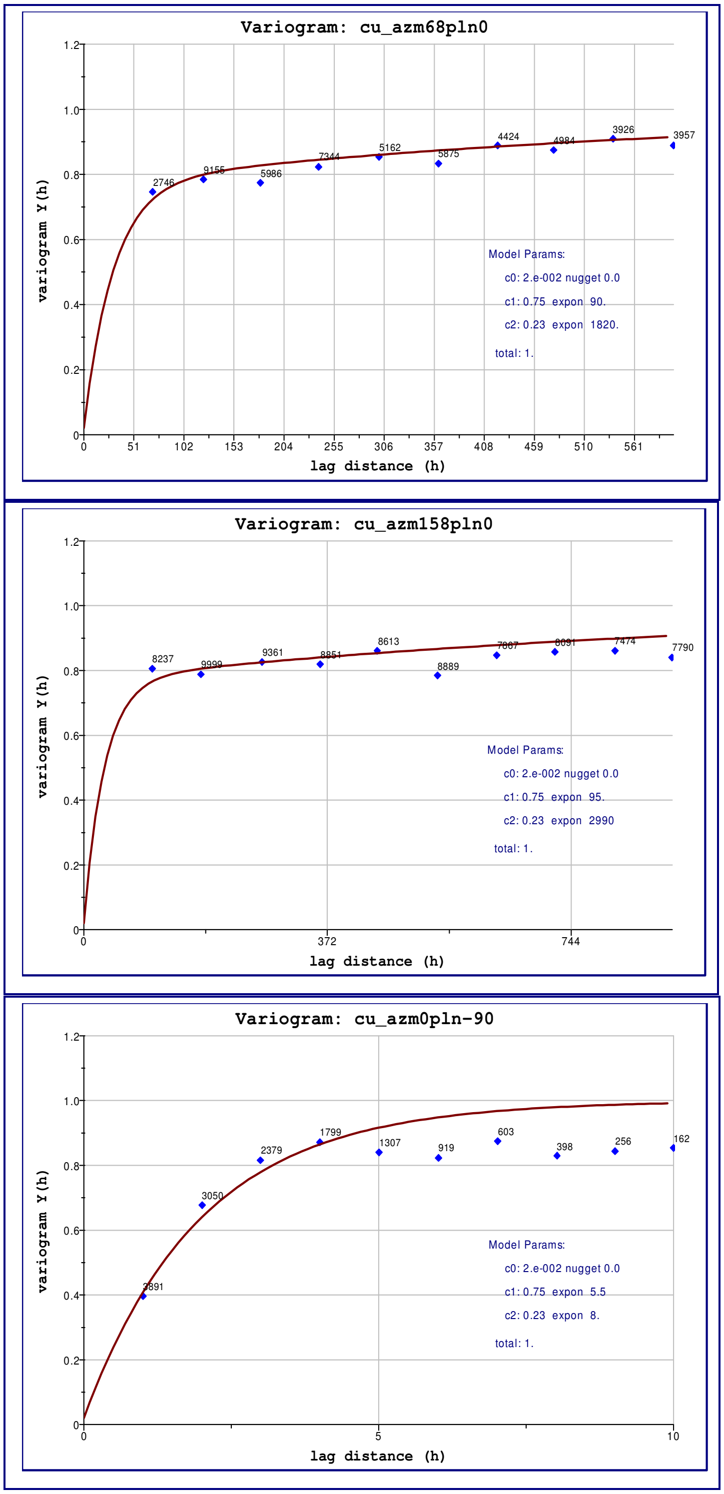


Figure 46: Variograms of Cobalt – Manto 3 (X, Y, Z directions)

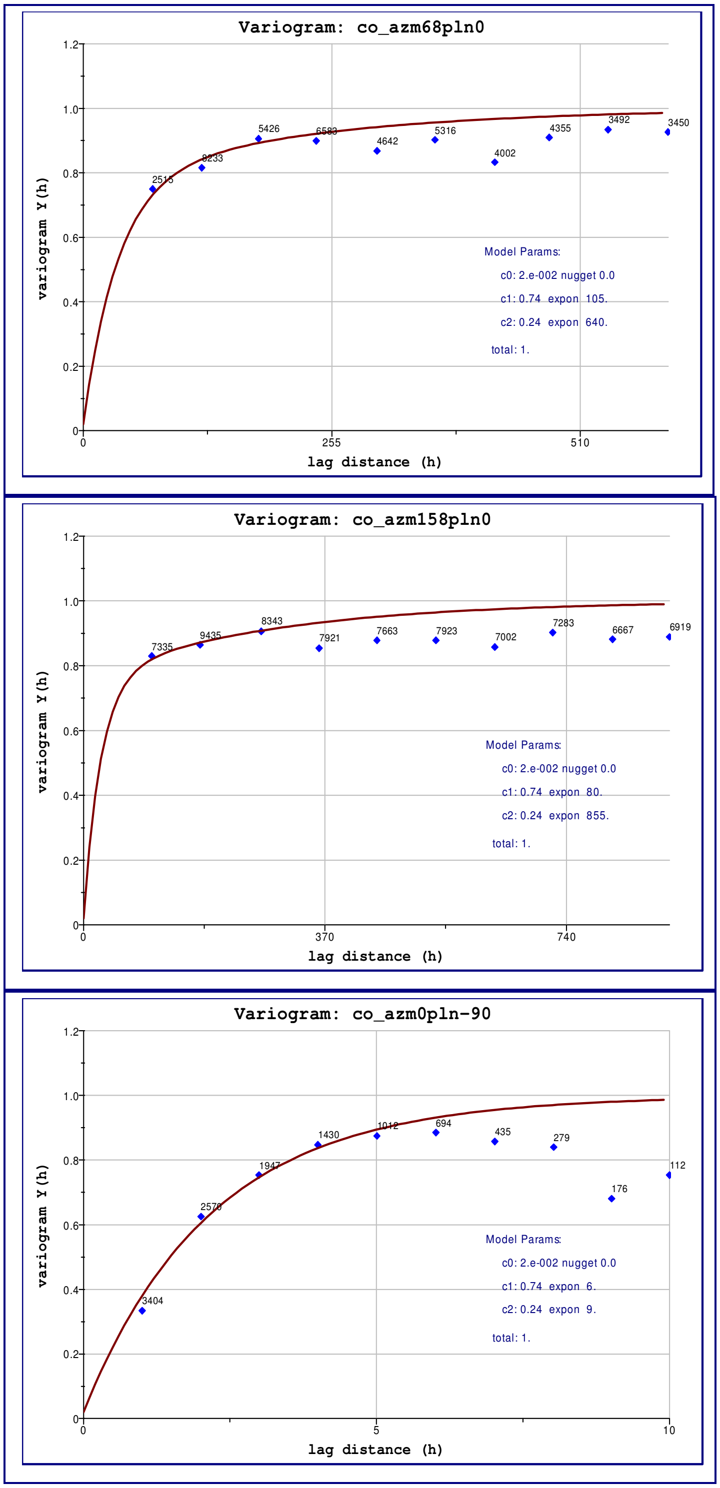


Figure 47: Variograms of Zinc – Manto 3 (X, Y, Z directions)

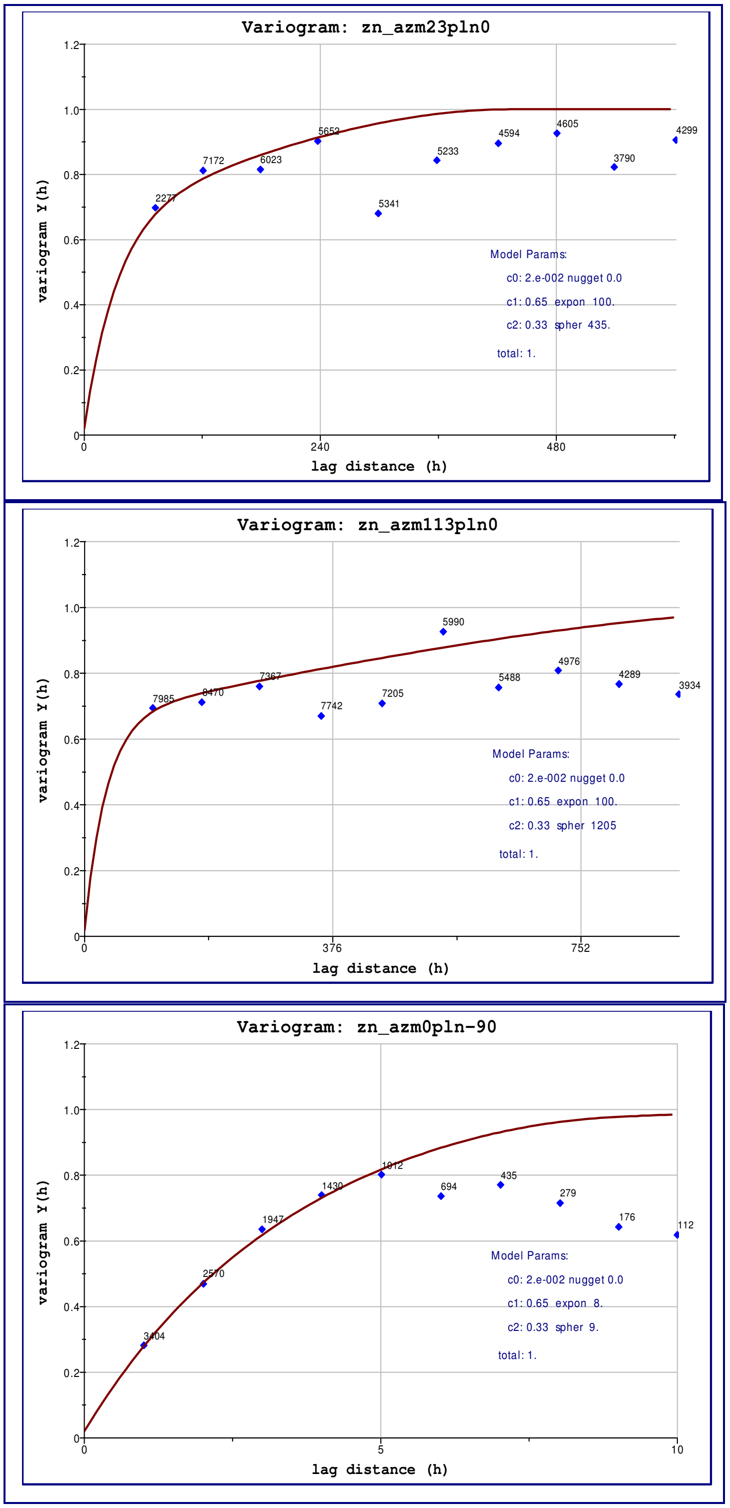


Table 33: Variogram Models

Metal	Structure	Manto 3a			Manto 3			Manto 4		
		Nugget	c1	c2	Nugget	c1	c2	Nugget	c1	c2
Copper	type		exp	sph		exp	exp		exp	sph
	variance	0.02	0.68	0.3	0.02	0.75	0.23	0.05	0.2	0.75
	range - X	-	100	1600	-	90	1820	-	155	180
	range - Y	-	170	1080	-	95	2290	-	180	240
	range - Z	-	5	6	-	5.5	8	-	5.5	8.5
	azimuth	292			292			330		
Cobalt	type		exp	sph		exp	exp		exp	sph
	variance	0.02	0.73	0.25	0.02	0.74	0.24	0.1	0.46	0.44
	range - X	-	100	4000	-	105	640	-	90	350
	range - Y	-	110	4000	-	80	855	-	175	570
	range - Z	-	4	6	-	6	9	-	15	7.5
	azimuth	315			292			360		
Zinc	type		exp	sph		exp	sph		exp	sph
	variance	0.02	0.7	0.28	0.02	0.65	0.33	0.05	0.1	0.85
	range - X	-	200	300	-	100	435	-	85	105
	range - Y	-	80	600	-	100	1205	-	90	110
	range - Z	-	5	2.5	-	8	9	-	7	15
	azimuth	360			337			360		

16.3.4 SEARCH PARAMETERS AND DATA CRITERIA

For each metal, three estimation passes were completed. Each pass progressively reduced the extent of the search radii or increased the number of data required before a grade estimate was calculated for each block. Therefore each pass progressively improves the accuracy or confidence of the block estimates.

In the previous resource estimates reported in 2005, search and data parameters shown in Table 34, were used.

Table 34: Search Parameters – 2005 3D Block Models

Parameter	Manto 2, 3aa,3a,3, 4			Manto 0, 1		
Pass	1	2	3	1	2	3
Search Radii (m)						
X – direction	200	280	400	500	750	1000
Y – direction	250	350	500	500	750	1000
Z – direction	2	2	4	2	2	4
Data Criteria						
Min Data	18	8	6	18	8	6
Max Data	32	32	32	32	32	32

For the current models these search parameters were modified in an attempt to improve the confidence in the grade estimates (Table 35).

The first pass remains very similar in terms of both radii and data. There is a small reduction in data required but tighter controls on the configuration of data around the block. This is achieved but specifying the number of search octants that must have at least one data composite used in the grade estimation.

For the second pass the search radii have been expanded by a factor of 2, but much the same data and configuration as used in pass one has been applied. The third pass uses the same search radii but slightly stricter data configuration. Only the pass 2 parameters are significantly different. The purpose of making these changes was to increase the data number used and to improve the configuration of these data around a block. This could only be achieved by expanding the search radii.

Table 35: Search Parameters – Current 3D Block Models

Parameter	Manto 3a,3		
Pass	1	2	3
Search Radii (m)			
X – direction	200	400	400
Y – direction	250	500	500
Z – direction	2	2	4
Data Criteria			
Min Data	14	14	7
Octants	4	4	2
Max Data	32	32	32

Resource estimates for Manto 0, 3aa and 4 have not been updated at this stage so the original search and data parameters still apply to these models.

Block discretization has been changed from 5 x 5 x 1 used in 2005 to 3 x 8 x 1. This simply results in a more even distribution of estimation nodes throughout a block.

16.3.5 MODEL CODING

The block models have been coded as follows:

- The proportion of a block below the topographic surface – Topo item
- The proportion of a block below the upper surface of the Manto – ore percent item
- A flag to specify whether a block is oxide or sulphide
- A flag to specify whether a block is within the land held by the company – claim item
- A flag to specify whether a block is within the areas affected by historic workings – mined item.

Only those blocks that are below both the topographic surface, the upper manto surface and within the land boundary held by the company have a metal grade estimate.

The oxide – sulphide distinction was based on polygons provided by MMB.

16.3.6 RESOURCE CLASSIFICATION – 3D BLOCK MODELS

The resources have been classified on the basis of the 3 estimation passes as Measured, Indicated and Inferred.

A block can only be classified as:

- Measured if 14 individual assay composites from at least 4 search octants are located within a search ellipse with radii of 200 m x 250 m x 2 m.
- Indicated if 14 individual assay composites from at least 4 search octants are located within a search ellipse with radii of 400 m x 500 m x 2 m.
- Inferred if 7 individual assay composites from at least 2 search octants are located within a search ellipse with radii of 400 m x 500 m x 4 m.

Manto 0 is classified only as inferred because of the significantly wider drill hole spacing, approximately 500 m x 450 m.

Mantos 1 and 3 have been extensively mined in the past. The location of the old mines are well known (Wilson 1955), however due to the method of mining adopted the precise location and extent of voids, pillars and back-filled excavations are not known.

Ore was mined using a shortwall mining method, however, only high grade ore (>3.5% Cu) was taken to the surface for processing. It is estimated that the ore processed amounted to only 40% of mined material, the remaining 60%, referred to as ‘Retaque’, was side-cast into old excavations as mining progressed. Exploration drill holes have intersected voids in about 50 instances and a similar number of Retaque intervals have been identified. The estimation process has been carried out effectively ignoring this issue other than treating voids as ‘missing’ or ‘no data'.

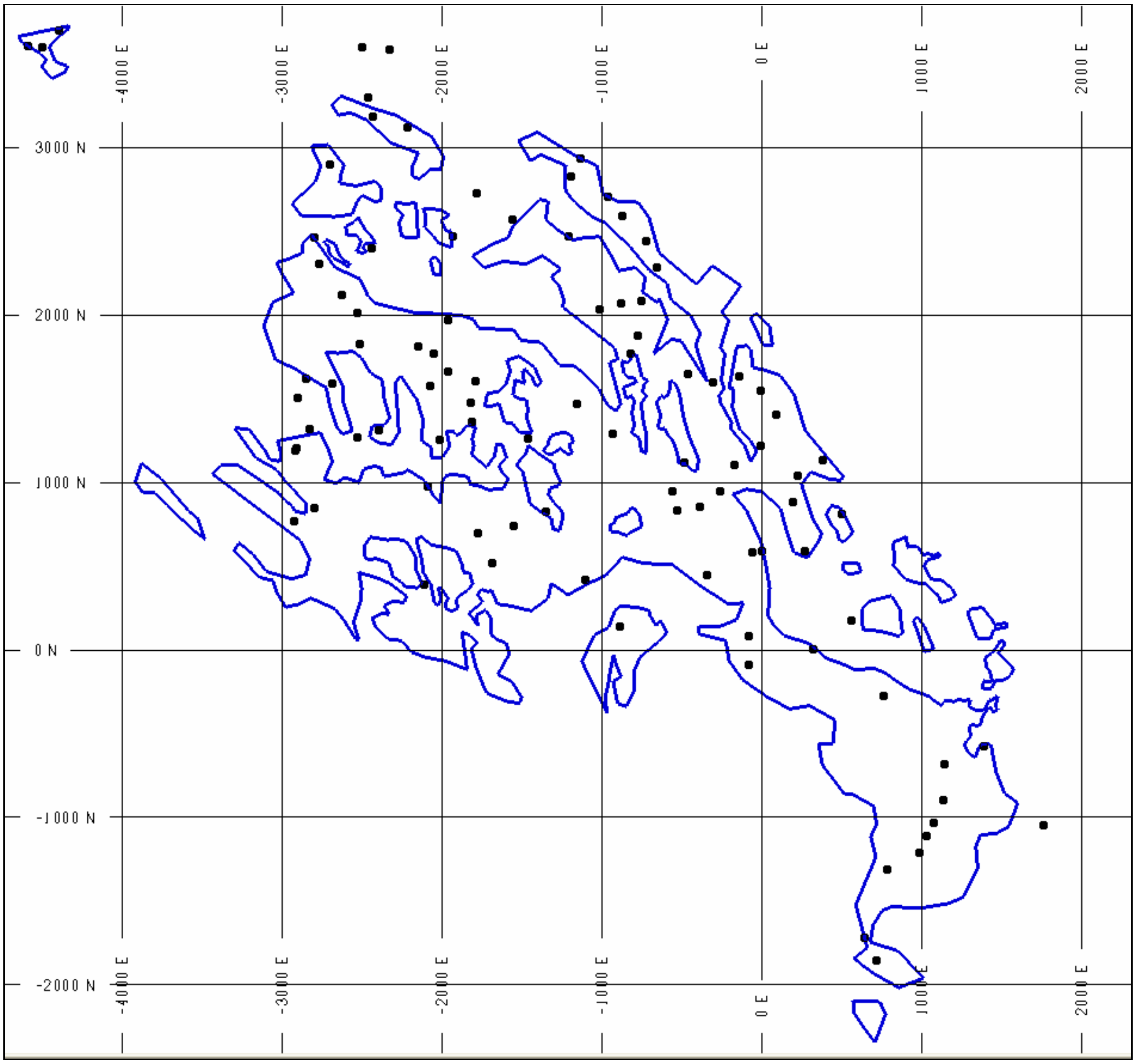
The resource, however, should be modified to account for the previous mining activities. This could be achieved by simply removing an amount of ore equal to the known treated ore tonnage, 13.6Mt. However, this may be overly conservative since many of the workings have collapsed and the void volume is now lower than was originally the case.

The lower grade clayey breccias, that typically overlie the higher grade laminated manto material that was mined, have collapsed or sagged into the original voids and areas of back-fill, prior to the resource drilling programs of the 1990s. The higher grade ore mined and processed has effectively been ‘replaced’ by lower grade material, firstly due to back-filling as mining progressed and later by the collapse of the workings where voids were left. This replaced material has been sampled and can be considered in situ. Voids still remain though and these have been accounted for by:

- digitising a simplified footprint shape around the area of historic workings (Figure 48)
- the footprint area was extended up 2 m to form a 3D object that enveloped the workings
- all model blocks that fell within this 'mined envelope' are flagged as 'mined'
- determining the proportion of sample intervals within the area affected by past mining that intersected voids
- factoring the tonnes by the proportion of sample interval that were voids.

For manto 3 a total of 335 m of drill hole intersection occur within the mined volume of which 12% are voids (see Figure 48). It is appropriate therefore, to factor the tonnes down by 12%.

Figure 48: Manto 3 Old Mine Areas – Location Drill Holes Which Encountered Voids



16.3.7 RESOURCE ESTIMATES – 3D BLOCK MODELS

Grade estimation in 2005 was completed using a specialist mining software package known as "Minesight-3D" (v 3.2), available from Metech Pty Ltd.

For the current resource models a new Ordinary Kriging routine added to the GS3 software used for data analysis and variogram estimation has been used. The benefit of using this new software option is that now the entire data analysis and resource modelling process can be completed within one software package.

The Boléo Resource includes Measured, Indicated and Inferred Categories.

Resources are quoted at copper equivalent cutoff grades of 0.5% and 1.0%. Copper Equivalent is a means of describing the effective grade of a polymetallic resource by bringing the value of all the pay metals (copper, cobalt and zinc in this instance) to bear. The proportion of each metal in the copper equivalent is weighted in the formula according to its 'value' i.e., its price. The prices used in the formula below for each of the metals are based on long term average prices as follows:

- copper – US$0.95/lb
- cobalt – US$12/lb
- zinc – US$0.45/lb.

The copper equivalent formula used is therefore:

- CuEq% = Cu% + 12Co%/0.95 + 0.45Zn%/0.95.

The resource models for mantos 0, 3aa and 4 are unchanged from the 2005 resource estimates. Manto 3 and 3a are new resource estimates, incorporating the latest drilling results. Manto 1 and 2 are not reported as 3D block models and have been removed from the following tables. These models are discussed and reported as seam models in a following section.

MEASURED AND INDICATED RESOURCE

Table 36: Measured and Indicated Resource at 0.5% CuEq. Cutoff

0.5% CuEq Cutoff	Manto	0	3aa	3a	3	4
Measured	Tonnes (10^6)		1.6	10.4	39.4	2.4
	CuEq.%		2.289	2.03	2.24	1.82
	Cu%		0.647	0.43	1.02	0.96
	Co%		0.096	0.101	0.089	0.056
	Zn%		0.91	0.67	0.30	0.33
Indicated	Tonnes (10^6)		1.8	35.7	75.5	27.6
	CuEq.%		2.039	1.36	2.03	1.10
	Cu%		0.55	0.36	1.09	0.50
	Co%		0.087	0.058	0.059	0.037
	Zn%		0.816	0.57	0.42	0.29

0.5% CuEq Cutoff	Manto	0	3aa	3a	3	4
Total	Tonnes (10^6)		3.4	46.1	114.9	30.0
	CuEq.%		2.16	1.51	2.10	1.16
	Cu%		0.60	0.38	1.07	0.53
	Co%		0.091	0.068	0.069	0.039
	Zn%		0.86	0.59	0.38	0.29

Table 37: Measured and Indicated Resource at 1.0% CuEq. Cutoff

0.5% CuEq Cutoff	Manto	0	3aa	3a	3	4
Measured	Tonnes (10^6)		1.6	9.0	36.5	1.4
	CuEq.%		2.318	2.20	2.35	2.57
	Cu%		0.656	0.49	1.09	1.42
	Co%		0.097	0.110	0.089	0.075
	Zn%		0.92	0.70	0.31	0.44
Indicated	Tonnes (10^6)		1.7	24.2	65.8	10.6
	CuEq.%		2.093	1.63	2.21	1.77
	Cu%		0.565	0.48	1.22	0.89
	Co%		0.090	0.067	0.062	0.054
	Zn%		0.839	0.63	0.43	0.41
Total	Tonnes (10^6)		3.3	33.2	102.3	12.0
	CuEq.%		2.20	1.78	2.26	1.86
	Cu%		0.61	0.48	1.17	0.95
	Co%		0.093	0.079	0.072	0.057
	Zn%		0.88	0.65	0.39	0.41

INFERRED RESOURCE

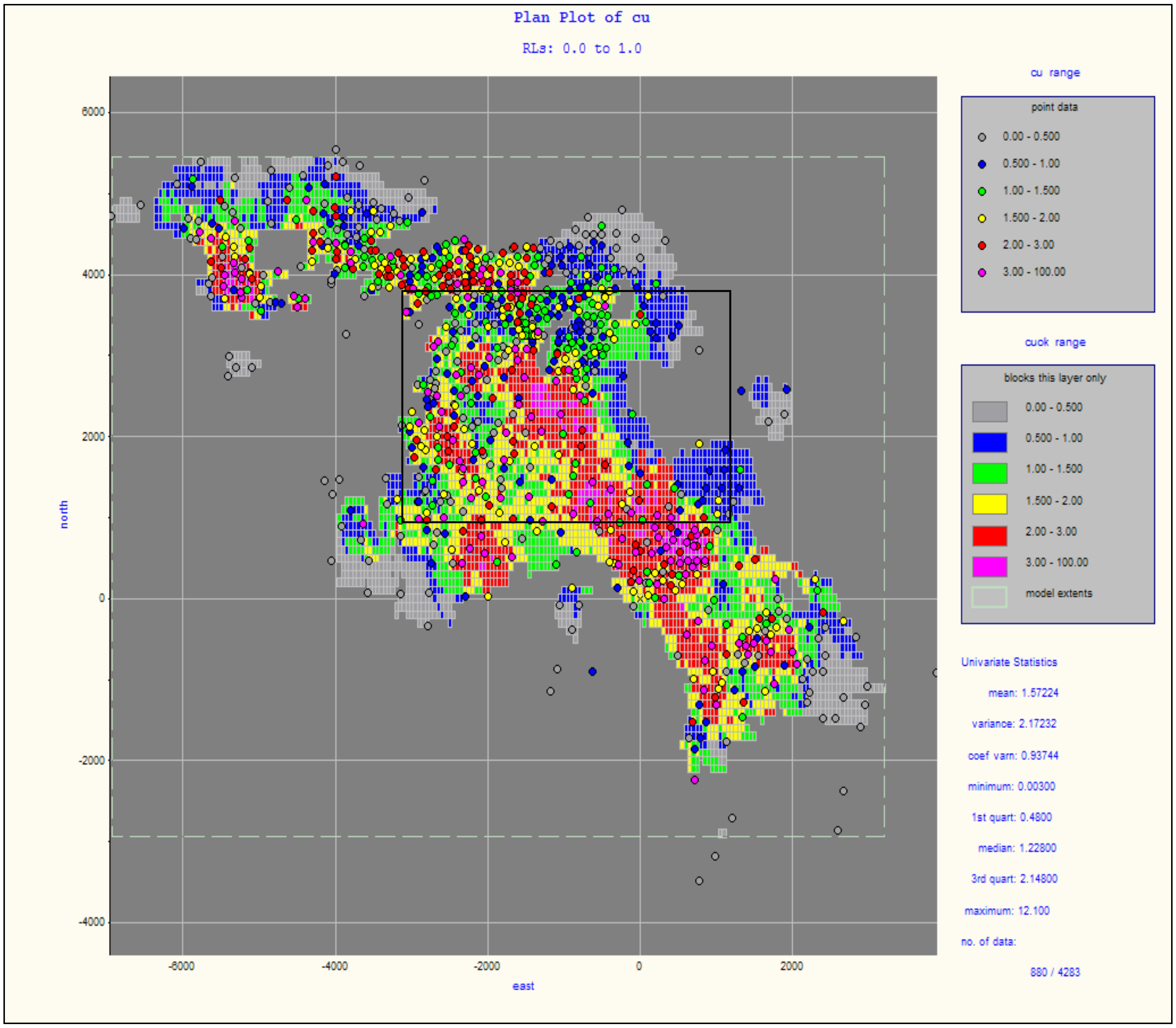
Table 38: Inferred Resource

Cutoff	Manto	0	3aa	3a	3	4
0.5% CuEq.	Tonnes (10^6)	9.7	0.5	25.2	58.5	56.9
	CuEq.%	0.56	1.92	1.19	1.51	0.87
	Cu%	0.04	0.58	0.34	0.63	0.38
	Co%	0.012	0.083	0.046	0.048	0.030
	Zn%	0.77	0.64	0.58	0.58	0.23
1.0% CuEq.	Tonnes (10^6)	0.0	0.4	14.3	40.8	12.7
	CuEq.%	1.11	2.03	1.50	1.81	1.47
	Cu%	0.02	0.63	0.49	0.83	0.79
	Co%	0.039	0.086	0.056	0.054	0.041
	Zn%	1.26	0.66	0.64	0.65	0.32

16.3.8 MODEL VERIFICATION

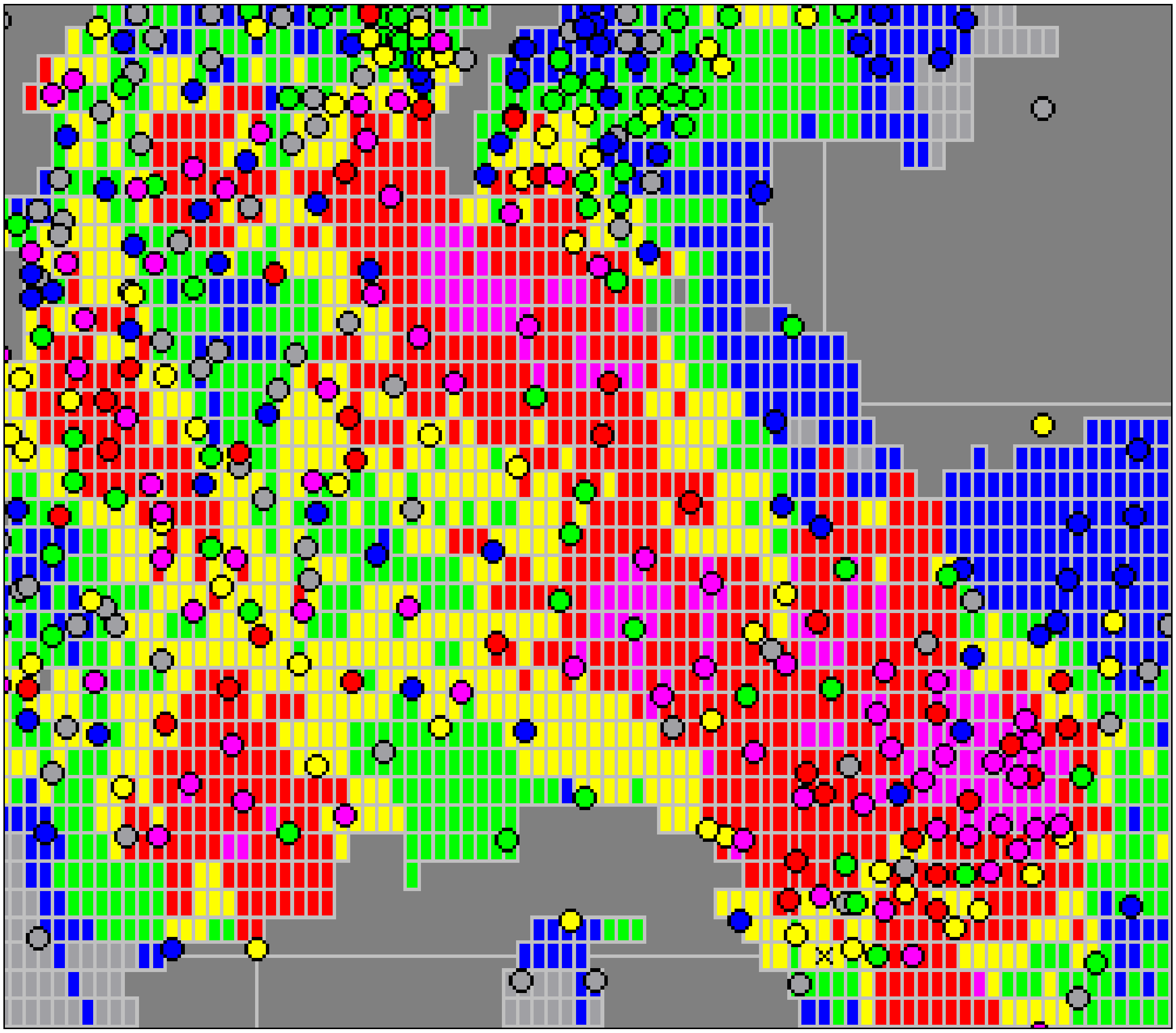
The model grade estimates have been verified by plotting block grades against assay composite grades, in plan, for different levels of the model. Plans of the basal level of Manto 3 are shown for copper, cobalt, and zinc (Figures 49 to 56).

Figure 49: Manto 3 Copper – Block Model with Assay Composites



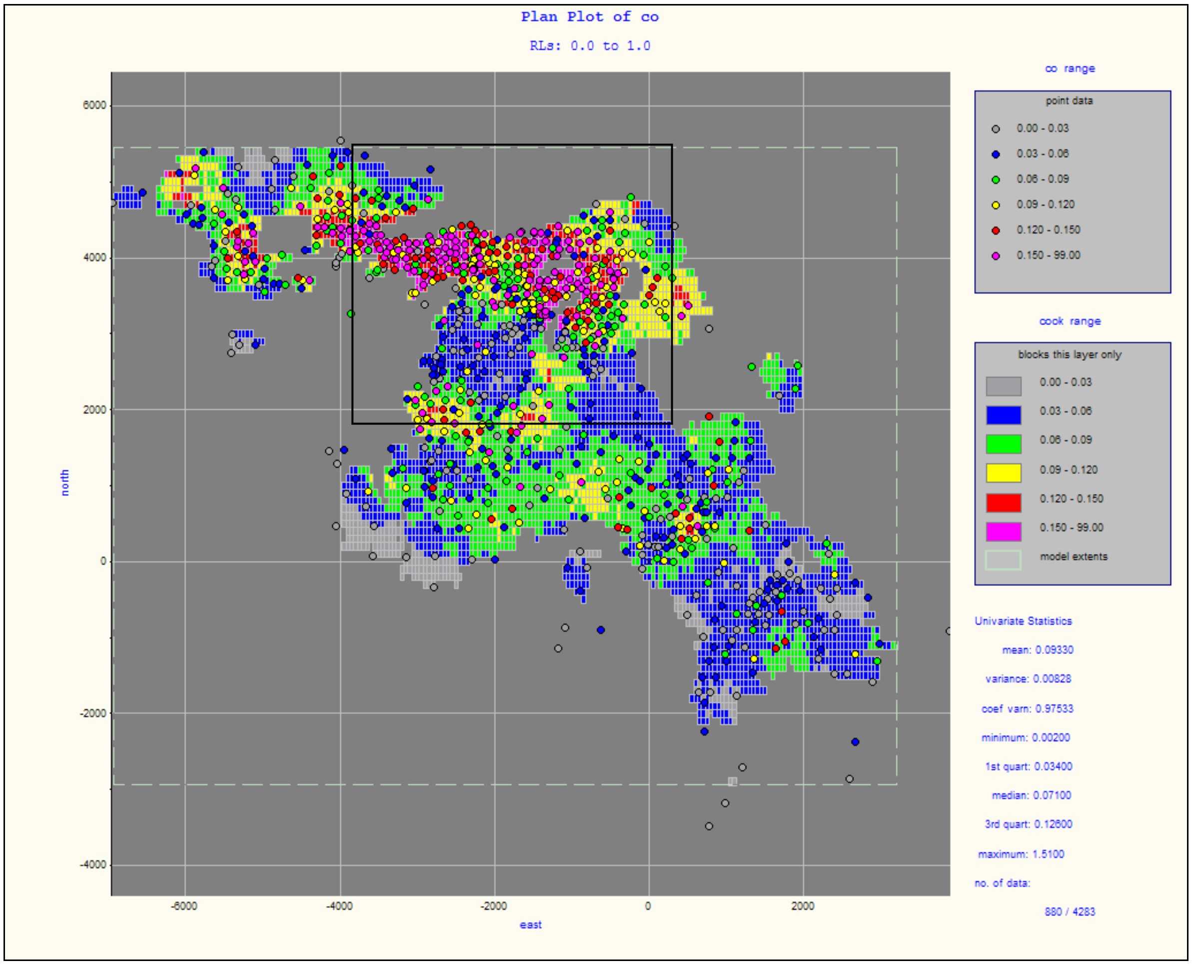
Notes: Total Resource (basal layer 0 m to 1 m). Box highlights area shown in detail in Figure 50.

Figure 50: Detail of Manto 3 Copper – Block Model with Assay Composites



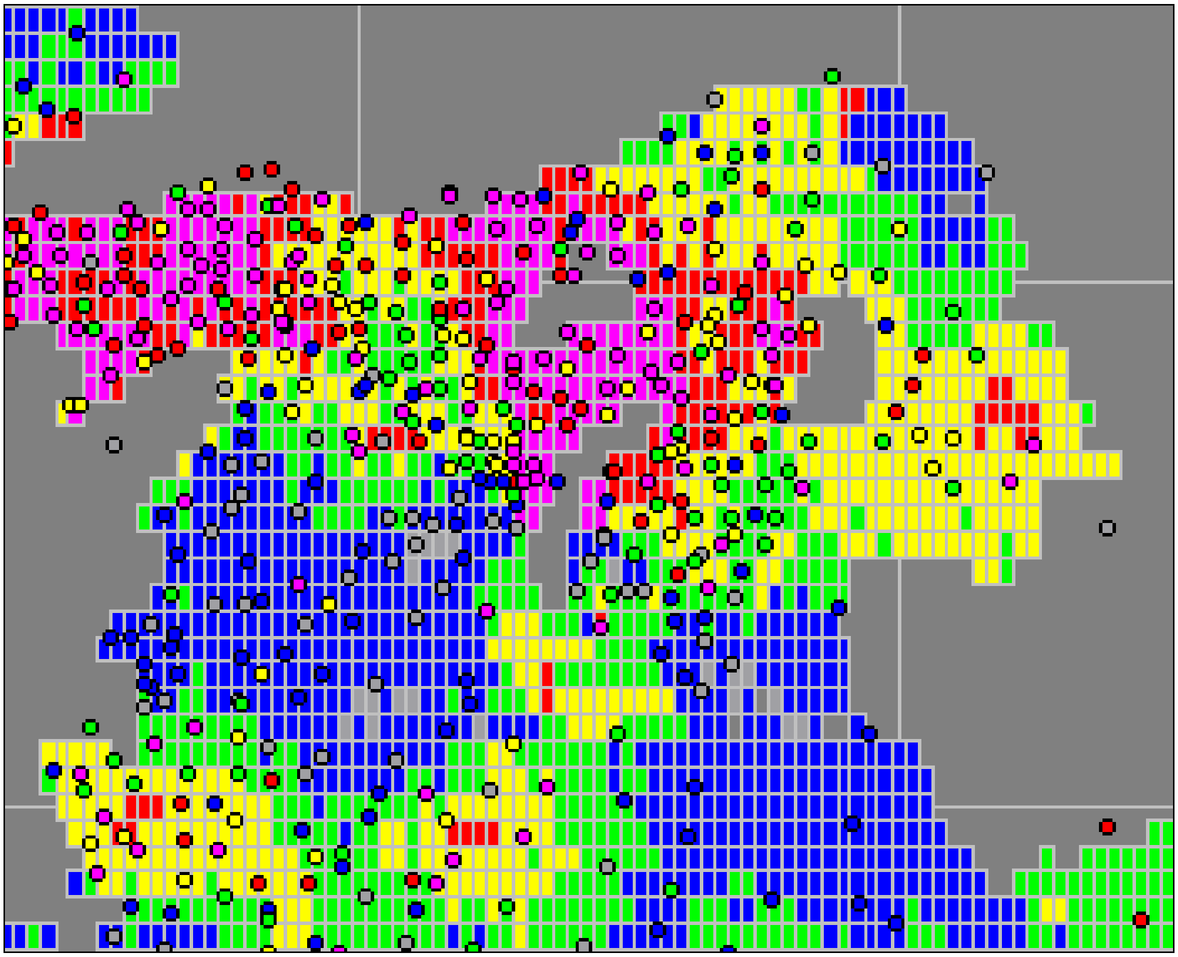
Note: the legend for Fig 50 is as per Figure 49.

Figure 51: Manto 3 Cobalt – Block Model with Assay Composites



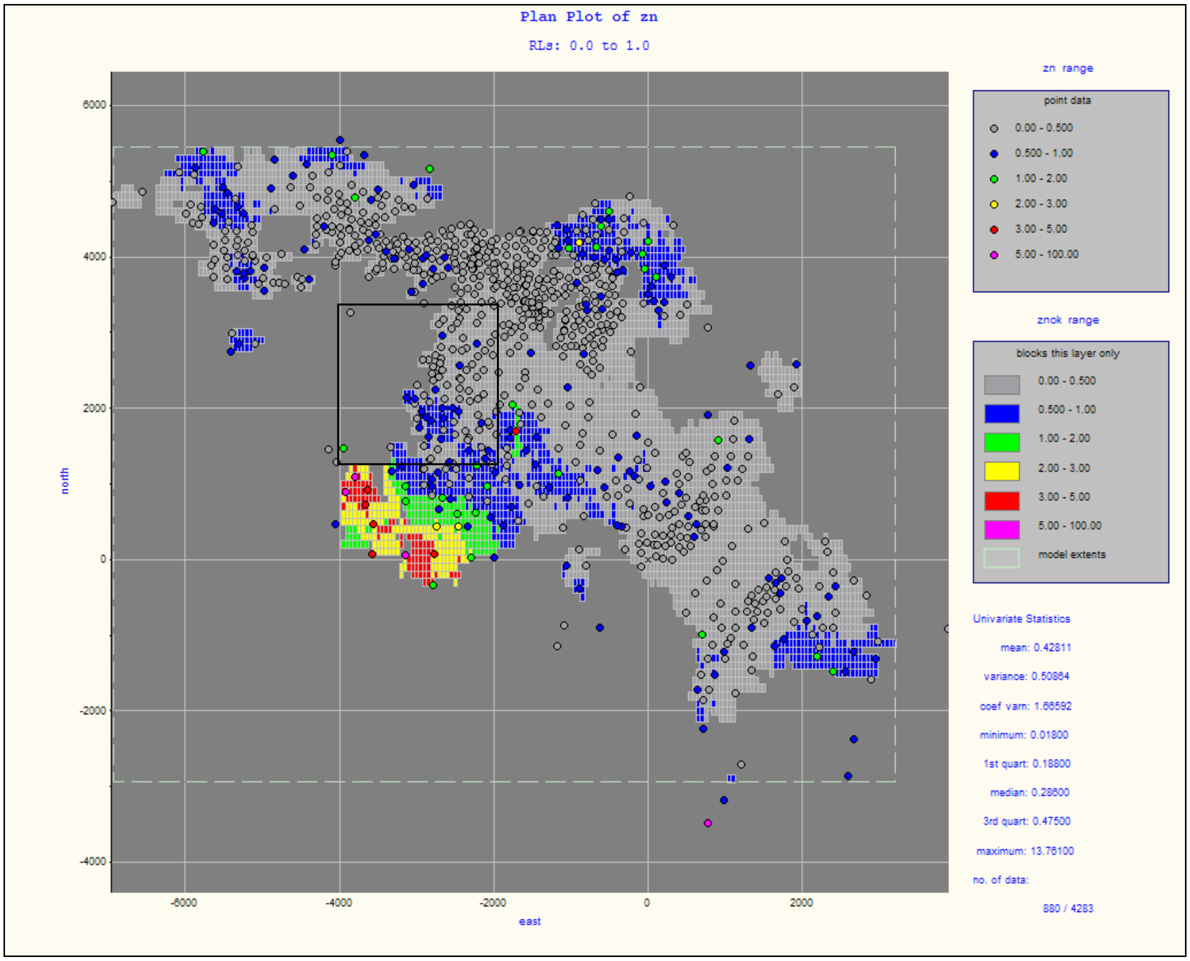
Notes: Total Resource (basal layer 0-1 m). Box highlights area shown in detail in Figure 52.

Figure 52: Detail of Manto 3 Cobalt – Block Model with Assay Composites



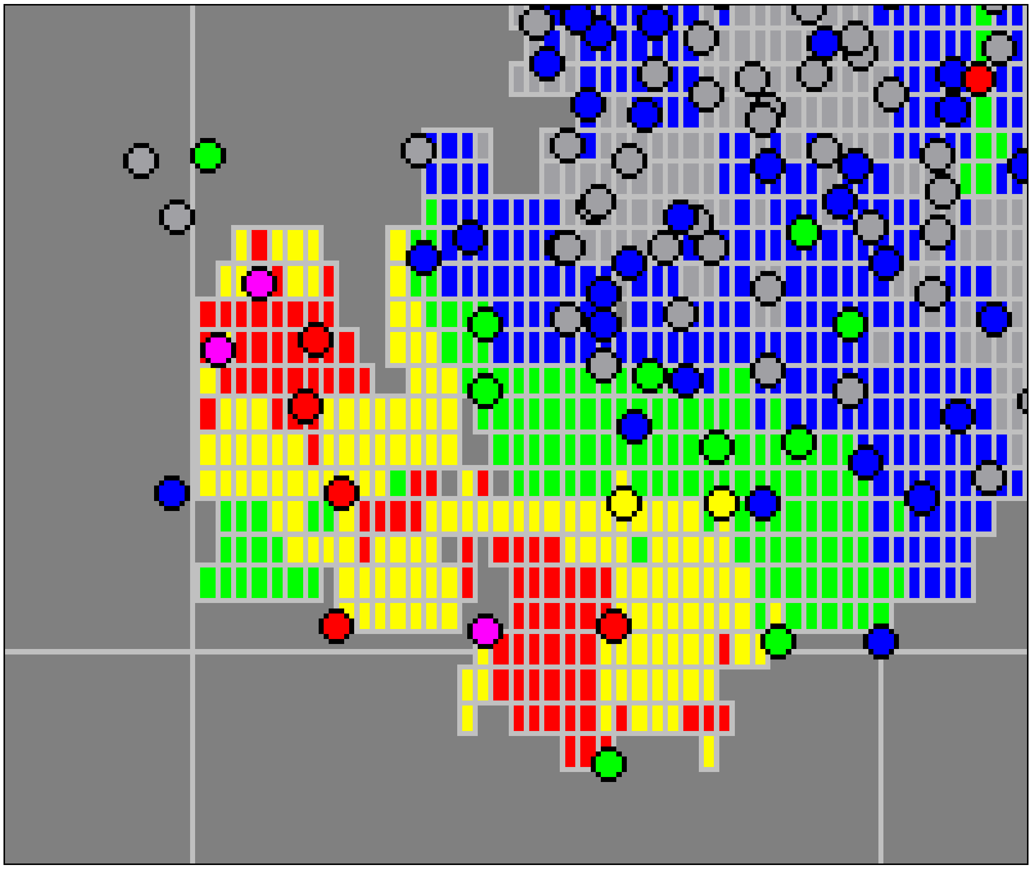
Note: the legend for Fig 52 is as per Figure 51.

Figure 53: Manto 3 Zinc – Block Model with Assay Composites



Notes: Total Resource (basal layer 0-1 m). Box highlights area shown in detail in Figure 54.

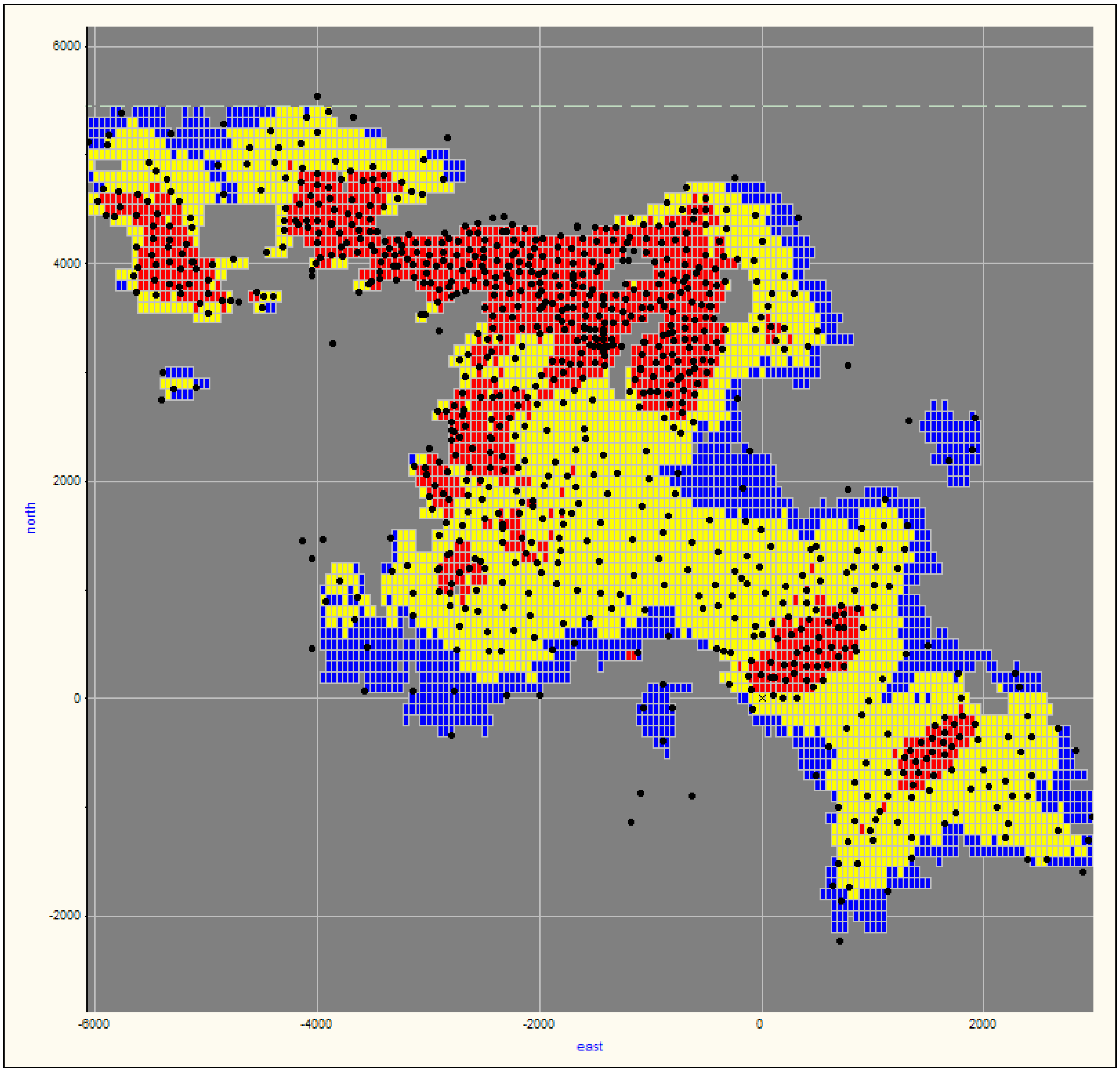
Figure 54: Detail of Manto 3 Zinc – Block Model with Assay Composites



Note: the legend for Fig 54 is as per Figure 53.

The appropriateness of the resource classification has been assessed by plotting the classification against the drill hole collar locations (Figure 55). The pattern shows a tight concentric zonation around areas of closer spaced drilling. Note that the occurrence of a drill hole either in or very close to a block does not guarantee a high level of resource classification.

Figure 55: Manto 3 Resource Classification – Block Model with Drill Hole Locations



Note: Measured – Red, Indicated – Yellow, Inferred – Blue (basal layer 0 to 1 m).

16.3.9 Comparison with 2005 3D Block Models

The reported resource estimates for Manto 3a and 3 are compared against the 2005 resource estimates.

For Manto 3 the differences between the reported resource from 2005 and the new resource are attributable to three causes:

- addition of new drill hole data and re-interpretation of the manto footwall
- changes to the kriging search parameters

- re-classification of the retaque proportion of the 2005 resource as Measured, Indicated or Inferred.

In the table below the new Manto 3 resource is reported in four columns with a different label in the row headed "Year of Estimate". The data reported in each of these columns is:

- 2005 – As estimated and reported in the year 2005
- 2007 A – New estimate, new drill hole data, Retaque tonnes segregated out
- 2007 B – New estimate, new drill hole data. Retaque tonnes allocated to Measured, Indicated or Inferred resource based on kriging search and data parameters.

The differences between 2005 data and 2007A are due to the new data, the changes made to the kriging search parameters and the identification of "false bottom" manto footwalls. The false bottom footwalls have allowed low grade or barren basal intervals to be removed from the grade estimation. When the effect of the false bottom is included in the resource calculations it allows for increased mining heights at similar grades. This results in a small net increase in tonnage.

The differences between 2007A and 2007B are only due to the removal of the Retaque category and re-allocation of this material to either Measured, Indicated, or Inferred resource according to kriging search and data parameters. This step adds approximate 10 Mt to the combined measured and indicated resource and elevates the grade from 1.97% CuEq. to 2.10%. This happens because the Retaque material represents the old mined areas which are generally the higher grade areas.

Table 39: Manto 3 – Current vs. 2005 Resource Model, Measured and Indicated Resources

		Manto 3					
Cutoff Grade		0.5% CuEq.			1.0% CuEq.		
Year of Estimate		2005	2007 A	2007 B	2005	2007 A	2007 B
Measured	Tonnes (10^6)	34.4	37.5	39.4	31.5	34.6	36.5
	CuEq%	2.14	2.21	2.24	2.27	2.33	2.35
	Cu%	0.88	0.98	1.02	0.95	1.05	1.09
	Co%	0.089	0.086	0.089	0.094	0.090	0.089
	Zn%	0.29	0.30	0.30	0.30	0.30	0.31
Indicated	Tonnes (10^6)	62.5	63.0	75.5	51.7	53.3	65.8
	CuEq%	1.87	1.84	2.03	2.10	2.02	2.21
	Cu%	0.93	0.92	1.09	1.08	1.05	1.22
	Co%	0.059	0.058	0.059	0.064	0.061	0.062
	Zn%	0.41	0.40	0.42	0.43	0.41	0.43
Total	Tonnes (10^6)	96.9	100.5	114.9	83.2	87.9	102.3
	CuEq%	1.97	1.98	2.10	2.16	2.14	2.26
	Cu%	0.91	0.94	1.07	1.03	1.05	1.17
	Co%	0.070	0.068	0.069	0.075	0.072	0.072
	Zn%	0.37	0.36	0.38	0.38	0.37	0.39

Table 40: Manto 3 - Current vs. 2005 Resource Model, Inferred Resources

		Manto 3					
Cutoff Grade		0.5% CuEq.			1.0% CuEq.		
Year of Estimate		2005	2007 A	2007 B	2005	2007 A	2007 B
Inferred	Tonnes (10^6)	54.9	55.5	58.5	39.7	37.8	40.8
	CuEq%	1.51	1.46	1.51	1.79	1.77	1.81
	Cu%	0.65	0.59	0.63	0.83	0.78	0.83
	Co%	0.049	0.047	0.048	0.054	0.054	0.054
	Zn%	0.52	0.58	0.58	0.58	0.65	0.65
"Retaque"	Tonnes (10^6)	18.2	17.4		17.8	17.4	
	CuEq%	2.57	2.86		2.60	2.87	
	Cu%	1.60	1.83		1.62	1.84	
	Co%	0.058	0.062		0.058	0.062	
	Zn%	0.52	0.52		0.52	0.52	
Total	Tonnes (10^6)	73.1	72.9	58.5	57.5	55.2	40.8
	CuEq%	1.77	1.79	1.51	2.04	2.12	1.81
	Cu%	0.88	0.88	0.63	1.07	1.11	0.83
	Co%	0.051	0.051	0.048	0.056	0.057	0.054
	Zn%	0.52	0.57	0.58	0.56	0.61	0.65

Manto 3a shows an overall drop in tonnes at each cutoff grade although the amount and grade of Measured and Indicated material remains very similar. Infill drill holes for Manto 3a generally reported low grades and where located outside the main area of mineralization for this Manto. The differences between the two models are mostly attributable to the change in kriging parameters.

Table 41: Comparison between Current and 2005 Resource Models, Manto 3a

		Manto 3a			
CuEq. Cutoff %		0.5		1.0	
Year of Estimate		2005	2007	2005	2007
Measured	Tonnes (10^6)	9.7	10.4	8.4	9.0
	CuEq%	2.12	2.03	2.31	2.20
	Cu%	0.45	0.43	0.51	0.49
	Co%	0.106	0.101	0.115	0.110
	Zn%	0.71	0.67	0.74	0.70
Indicated	Tonnes (10^6)	38.1	35.7	23.8	24.2
	CuEq%	1.32	1.36	1.63	1.63
	Cu%	0.35	0.36	0.47	0.48
	Co%	0.055	0.058	0.067	0.067
	Zn%	0.59	0.57	0.66	0.63

		Manto 3a			
CuEq. Cutoff %		0.5		1.0	
Year of Estimate		2005	2007	2005	2007
Measured & Indicated	Tonnes (10^6)	47.8	46.1	32.2	33.2
	CuEq%	1.48	1.51	1.81	1.78
	Cu%	0.37	0.38	0.48	0.48
	Co%	0.065	0.068	0.080	0.079
	Zn%	0.61	0.59	0.68	0.65
Inferred	Tonnes (10^6)	29.6	25.2	17.8	14.3
	CuEq%	1.25	1.19	1.55	1.50
	Cu%	0.38	0.34	0.54	0.49
	Co%	0.047	0.046	0.057	0.056
	Zn%	0.58	0.58	0.63	0.64

16.4 SEAM MODELS

Seam models have been produced for Manto 1, 2 and 3. For Mantos 1 and 2 these are the only models reported as these mantos will only be mined from underground. Manto 3 however will be mined from both open cuts and under ground developments. Therefore Manto 3 seam models reported here are not in addition to, but are a sub-set of and are included in the reported open cut 3D block models.

16.4.1 COMPOSITE LENGTH AND BLOCK DIMENSIONS

The type of the seam composite used from each drill hole has a big impact on the resource estimate. Three alternative strategies for seam definition are:

- A geological seam. The full logged manto interval is composited into a single seam, from the footwall to the top of the last identified manto interval.
- A grade seam. A grade cutoff is used to define the top and bottom of the seam.
- A mining seam. The base of the seam is defined by the manto footwall whilst the top is defined by mining constraints.

The grade of copper mineralization generally decreases from higher grade towards the base to lower grades at the top of each manto. If a geological seam is used significant lengths of material below economic cutoff grades can be included in the composite. If the thickness of the manto is greater than the minimum mining height then the inclusion of sub-economic low grade material will result in an unrealistic lower grade resource than bears no relation to the anticipated mining method. Therefore geological composites have not been used.

Definition composites by grade alone can be problematical. There is not obvious 'natural' cutoff in the data that can be consistently applied throughout. The main problem with grade defined seams is the confidence in the lateral continuity of the seam. Seams defined this way typically show quite variable heights above the known manto floor and many intervals would include a certain amount of internal waste. Both of these issues show that the lateral continuity of grade seams cannot be demonstrated. Variogram analysis which shows relatively short ranges also supports this contention. The grade seam method can only be used if the seam can be referenced against a defined geological horizon that has demonstrated continuity.

The manto footwall is a well defined geological entity with well established and demonstrated continuity. Over much of Manto 3 for example a boulder conglomerate defines the floor. Elsewhere though, other consistent lithologies are present, such as the calcareous sandstone unit that occurs beneath parts of Manto 1 and the barren clayey unit that overlies the conglomerate in certain parts of Manto3.

To ensure continuity of the seams the manto footwall or floor has to be used to define the base of each composite. The thickness of the seam can then be based on any other criteria.

The most sensible method to determine the seam thickness is to use a combination of grade and mining constraints. Mining studies at Boléo have determined a minimum height of 1.8 m and a maximum height of 4.2 m. Consequently it was decided to produce seam composites that reflect this. For each hole two different seam composites would be produced, both with a base defined by the manto footwall:

- A 1.8 m seam. Where the full natural manto interval in a hole is actually less than 1.8 m a certain amount of barren material is added to bring the thickness up to 1.8 m. Material greater than 1.8 m above the Manto floor is ignored and not used in the resource estimates.
- A variable thickness seam from 1.8 m to 4.2 m. As for the 1.8 m seam where the full natural manto interval in a hole is less than 1.8 m a certain amount of barren material is added to bring the thickness up to 1.8 m. Where the full natural thickness of the seam is greater than 1.8 m then additional increments of 0.6 m, from 1.8 m to 4.2 m were added to the seam. If the grade of these increments were greater than a specified grade the seam thickness would be increased to include these increments. If the grade of additional increments was not above the specified grade then the minimum 1.8 m composite would be used. Material greater than 4.2 m above the Manto floor is ignored and not used in the resource estimates.

The grade threshold used justify inclusion of additional increments was 1% Cu. Increments were determined at 2.4 m, 3.0 m, 3.6 m, and 4.2 m.

Model block dimensions were the same in terms of X (50m) and Y (100m) as used in the 3D block models. The minimum thickness models would have a fixed vertical height of 1.8 m, whilst models produced with seam composites up to 4.2 m would have variable height as determined by kriging seam thickness.

After producing the seam composites using the method described above it was clear that for Mantos 1 and 2 that only the minimum 1.8 m seams were appropriate. Consequently only minimum thickness resource estimates would be produced.

16.4.2 *Univariate Statistics of Data Composites*

Univariate statistics of Copper, Cobalt and Zinc for Manto 1, 2 and 3 are shown below (Tables 42 to 44). Manto 1 has been separated into two geological domains (D1, D2) based on interpretation of palaeo geography at the time of deposition. Domain 2 is a waste domain overlying a rise in the submerged basement floor, while D1 is the mineralized domain distal to the basement rise.

Generally the means of the seam composites are higher than the individual 1m composites and the coefficients of variation are lower.

Table 42: Univariate Statistics of Assay Composites – Copper

Copper	Manto					
	1	1	1	2	3	3
Seam/Domain	1.8	1.8/D1	1.8/D2	1.8	1.8	4.2m
Mean	0.872	2.132	0.309	0.421	1.406	1.388
CV	1.612	0.79	2.428	1.976	0.936	0.894
min	0.003	0.13	0.003	0.001	0.004	0.004
Q1	0.016	0.927	0.012	0.056	0.473	0.506
Median	0.133	1.757	0.03	0.164	1.114	1.141
Q3	1.155	2.538	0.19	0.403	1.893	1.91
Max	8.744	8.744	4.928	7.585	11.241	11.241
IQR	1.139	1.611	0.178	0.347	1.42	1.404
Data	214	66	148	477	873	873

Table 43: Univariate Statistics of Assay Composites – Cobalt

Copper	Manto					
	1	1	1	2	3	3
Seam/Domain	1.8	1.8/D1	1.8/D2	1.8	1.8	4.2 m
Mean	0.049	0.089	0.031	0.056	0.088	0.088
CV	1.097	0.515	1.51	1.571	0.895	0.839
min	0	0.013	0	0.001	0.002	0.002
Q1	0.01	0.054	0.008	0.018	0.034	0.034
Median	0.025	0.085	0.014	0.041	0.069	0.07
Q3	0.075	0.113	0.032	0.074	0.119	0.123
Max	0.372	0.201	0.372	1.683	1.215	0.95
IQR	0.065	0.059	0.024	0.056	0.085	0.089
Data	214	66	148	477	873	873

Table 44: Univariate Statistics of Assay Composites – Zinc

Copper	Manto					
	1	1	1	2	3	3
Seam/Domain	1.8	1.8/D1	1.8/D2	1.8	1.8	4.2m
Mean	0.995	0.638	1.154	1.029	0.405	0.405
CV	1.666	0.635	1.692	0.979	1.372	1.34
min	0.016	0.062	0.016	0.018	0.008	0.008
Q1	0.353	0.322	0.387	0.463	0.192	0.197
Median	0.637	0.568	0.691	0.716	0.29	0.29
Q3	0.895	0.794	0.945	1.203	0.46	0.46
Max	14.265	2.446	14.265	7.628	9.094	9.094
IQR	0.542	0.472	0.558	0.74	0.268	0.263
Data	214	66	148	477	873	873

16.4.3 SPATIAL CONTINUITY OF GRADE

Despite the reduction in sample variance the continuity determined from the experimental variograms is not significantly improved compared to the variograms determined from the 1 m composite data.

Variograms of seam composites for Manto 3 are shown below for copper, cobalt and zinc.

Figure 56: Variograms of Copper – Manto 3 (X, Y, directions)

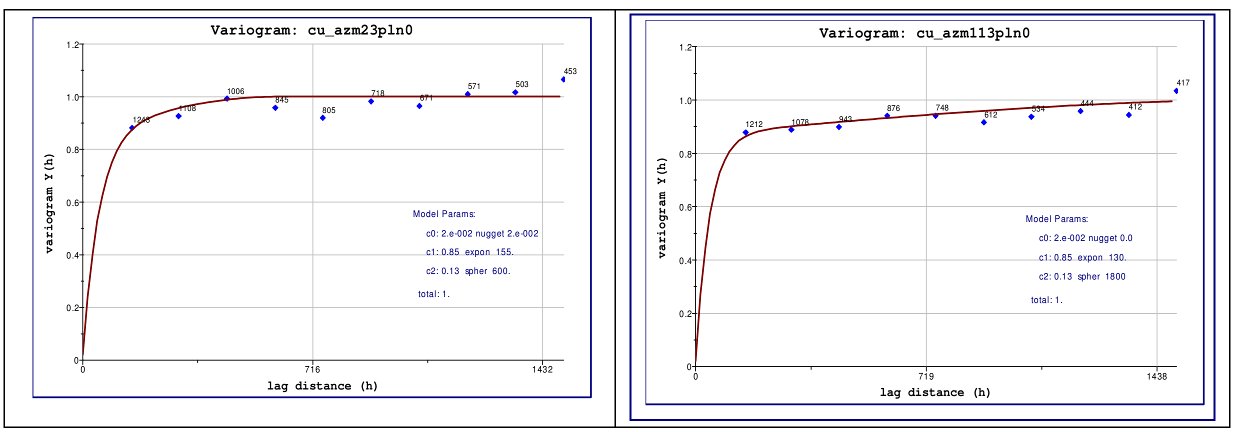


Figure 57: Variograms of Cobalt – Manto 3 (X, Y, directions)

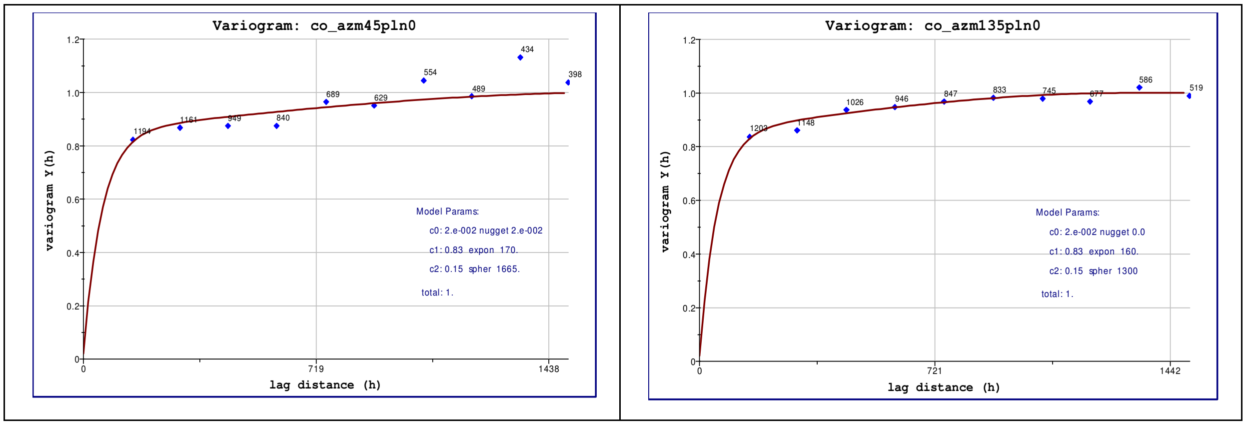
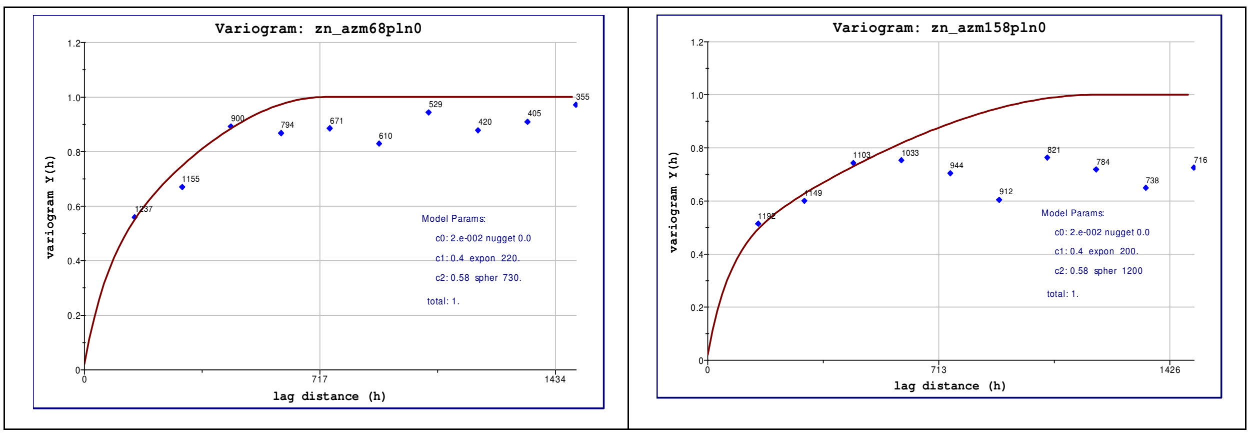


Figure 58: Variograms of Zinc – Manto 3 (X, Y, directions)



Full variogram models are shown in Table 45.

Table 45: Variogram Models – Seam Composites

		Manto 1			Manto 2			Manto 3		
Metal	**Structure**	**Nugget**	**c1**	**c2**	**Nugget**	**c1**	**c2**	**Nugget**	**c1**	**c2**
Copper	type		Exp	sph		exp	sph		exp	sph
	variance	0.02	0.7	0.28	0.02	0.58	0.4	0.02	0.85	0.13
	range – X	-	200	130	-	110	130	-	155	600
	range – Y	-	360	1150	-	130	280	-	130	1800
	azimuth	330			315			330		
Cobalt	type		Exp	sph		exp	exp		exp	sph
	variance	0.02	0.69	0.29	0.02	0.52	0.46	0.02	0.83	0.15
	range – X	-	100	500	-	120	210	-	170	1665
	range – Y	-	175	2805	-	120	350	-	160	1300
	azimuth	330			315			360		

Metal	Structure	Manto 1			Manto 2			Manto 3		
		Nugget	c1	c2	Nugget	c1	c2	Nugget	c1	c2
Zinc	type		Exp	sph		exp	sph		exp	sph
	variance	0.02	0.66	0.32	0.02	0.65	0.33	0.02	0.4	0.58
	range – X	-	185	200	-	245	270	-	220	730
	range – Z	-	460	940	-	240	340	-	200	1200
	azimuth	330			315			360		

16.4.4 SEARCH PARAMETERS AND DATA CRITERIA

The same search strategy with 3 consecutive estimation passes has been used for the seam models as used for the 3D block models. The only difference being that the data requirements are modified to take into account the single seam composites rather than several individual 1m composites for each hole.

Table 46: Search Parameters – Current 3D Block Models

Parameter	Manto 1, 2, 3		
Pass	1	2	3
Search Radii (m)			
X – direction	200	400	400
Y – direction	250	500	500
Data Criteria			
Min Data	4	4	2
Octants	4	4	2
Max Data	8	8	8

16.4.5 RESOURCE CLASSIFICATION – SEAM MODELS

The resources have been classified on the basis of the three estimation passes as Measured, Indicated and Inferred. A block can only be classified as:

- Measured if 4 seam composites from at least 4 search octants are located within a search ellipse with radii of 200 m x 250 m.
- Indicated if 4 seam composites from at least 4 search octants are located within a search ellipse with radii of 400 m x 500 m.
- Inferred if 2 seam composites from at least 2 search octants are located within a search ellipse with radii of 400 m x 500 m.

No measured resource is reported for Manto 1. Blocks that fell in this category based on the criteria defined above did not form coherent domains. Additional drilling currently being completed will help resolve this.

16.4.6 RESOURCE ESTIMATES – SEAM MODELS

The Boléo seam models include Measured, Indicated and Inferred Categories.

Resources are quoted at the 0.5% and 1.0% CuEq. cutoff grades.

- CuEq.%= Cu % + 12Co%/0.95 + 0.45Zn%/0.95.

For Manto 3 both 3D block model and seam models are presented. Manto 3 is generally thicker than the minimum mining height of 1.8 m so both a 1.8 m and variable (1.8 - 4.2 m) seam model is reported. The seam estimates for Manto 3 are not in addition to the 3D block model report previously in this document. The seam models are a sub-set of the 3D block model.

Table 47: Seam Models – Measured and Indicated Resource at 0.50% CuEq.

0.5% CuEq	Manto	1	2	3	3
Seam Height		1.8	1.8	1.8	1.8 - 4.2
Measured	Tonnes (10^6)		5.6	17.4	20.0
	CuEq%		1.88	2.93	2.93
	Cu%		0.51	1.56	1.56
	Co%		0.067	0.096	0.096
	Zn%		1.12	0.34	0.34
Indicated	Tonnes (10^6)	8.3	24.5	25.0	27.7
	CuEq%	2.61	1.64	2.62	2.63
	Cu%	1.34	0.46	1.54	1.54
	Co%	0.067	0.056	0.067	0.067
	Zn%	0.90	1.01	0.50	0.50
Total	Tonnes (10^6)	8.3	30.1	42.4	47.7
	CuEq%	2.61	1.68	2.75	2.76
	Cu%	1.34	0.47	1.55	1.55
	Co%	0.067	0.058	0.079	0.079
	Zn%	0.90	1.03	0.43	0.43

Table 48: Seam Models – Measured and Indicated Resource at 1.0% CuEq.

1.0% CuEq	Manto	1	2	3	3
Seam Height		1.8	1.8	1.8	1.8 - 4.2
Measured	Tonnes (10^6)		4.7	17.1	19.8
	CuEq%		2.07	2.97	2.96
	Cu%		0.57	1.58	1.58
	Co%		0.074	0.097	0.096
	Zn%		1.21	0.34	0.34
Indicated	Tonnes (10^6)	7.1	18.9	23.6	26.3
	CuEq%	2.92	1.89	2.72	2.73
	Cu%	1.54	0.55	1.62	1.61

1.0% CuEq	Manto	1	2	3	3
	Co%	0.075	0.065	0.069	0.069
	Zn%	0.91	1.10	0.50	0.51
Total	Tonnes (10^6)	7.1	23.6	40.7	46.1
	CuEq%	2.92	1.93	2.83	2.83
	Cu%	1.54	0.55	1.60	1.60
	Co%	0.075	0.067	0.081	0.081
	Zn%	0.91	1.12	0.43	0.44

Note: Manto 3 resource quoted above is a sub-set of NOT additional to the 3D block model.

Table 49: Seam Models - Inferred Resource

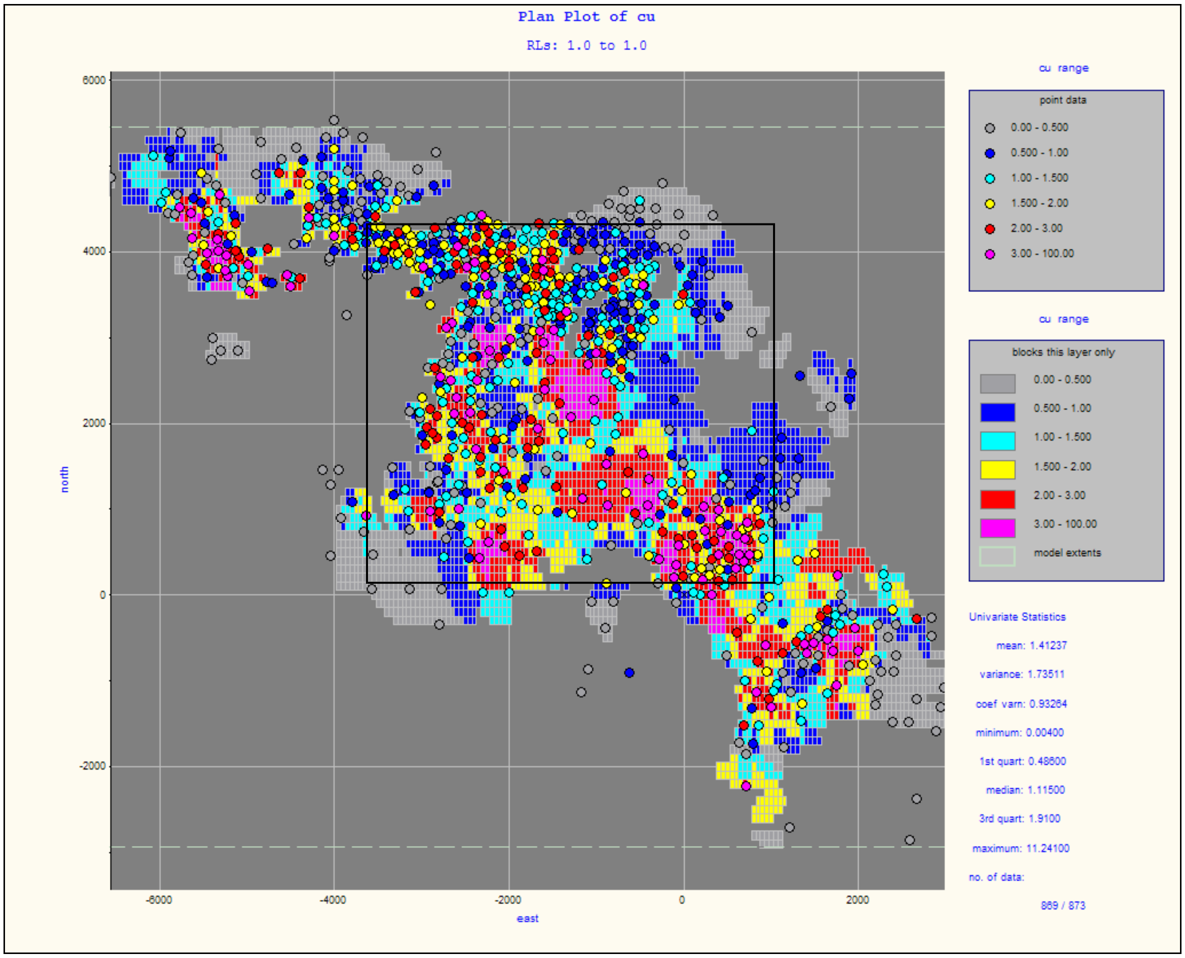
Cutoff	Manto	1	2	3	3
Seam Height		1.8	1.8	1.8	1.8 - 4.2
0.5% CuEq.	Tonnes (10^6)	10.2	51.3	26.4	28.4
	CuEq%	2.19	1.48	1.95	2.01
	Cu%	0.87	0.34	0.93	0.97
	Co%	0.063	0.048	0.053	0.055
	Zn%	1.13	1.13	0.73	0.73
1.0% CuEq.	Tonnes (10^6)	8.6	37.5	22.7	24.8
	CuEq%	2.47	1.75	2.14	2.19
	Cu%	0.99	0.41	1.06	1.09
	Co%	0.071	0.056	0.057	0.059
	Zn%	1.22	1.32	0.78	0.77

Note: Manto 3 resource quoted above is a sub-set of, NOT additional to, the 3D block model.

16.4.7 MODEL VERIFICATION

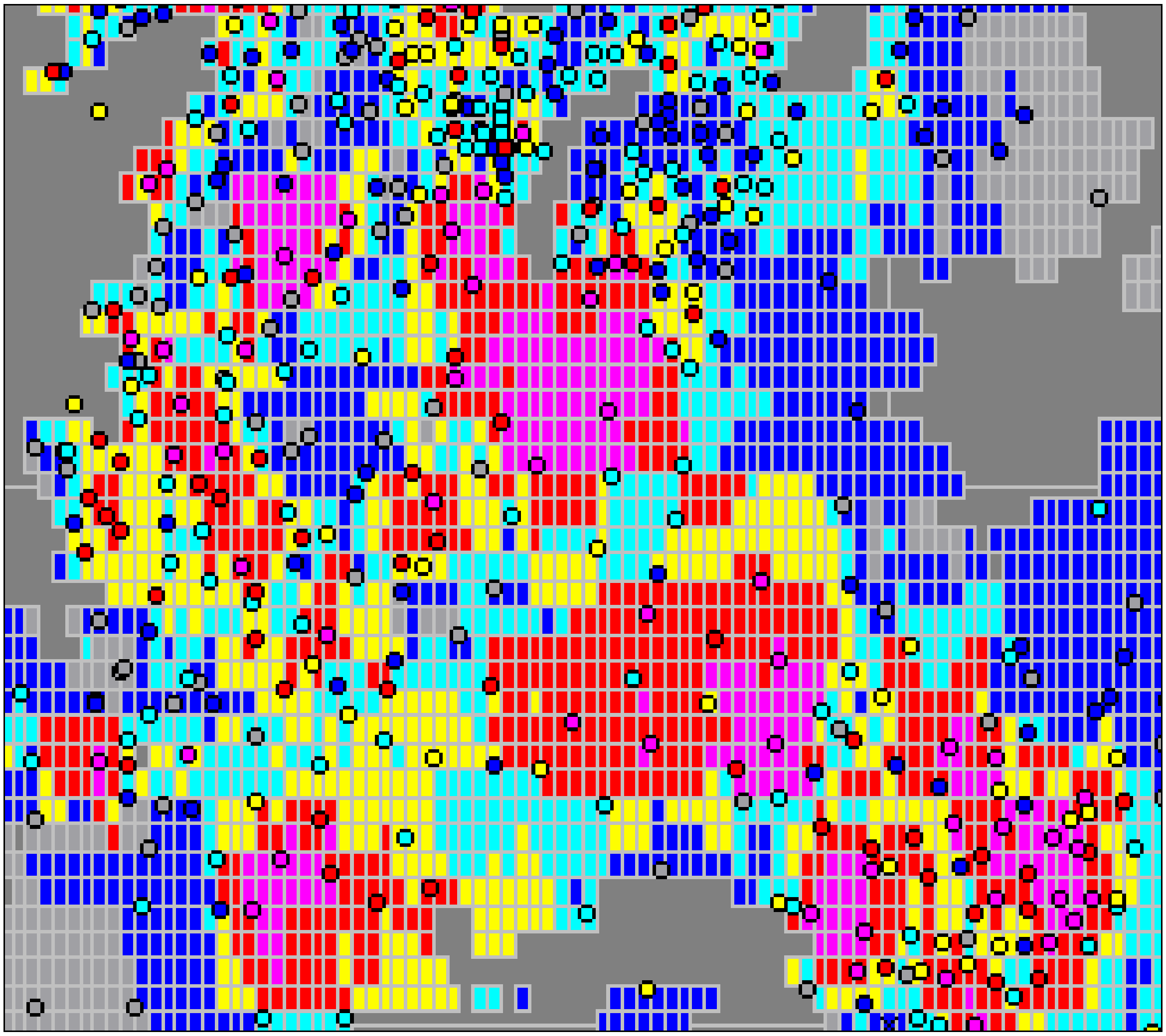
The model grade estimates have been verified by plotting block grades against assay composite grades, in plan, for different levels of the model. Plans of Manto 1 and 3 are shown for copper (Figures 59 to 64).

Figure 59: Manto 3 Copper – Block Model with Assay Composites



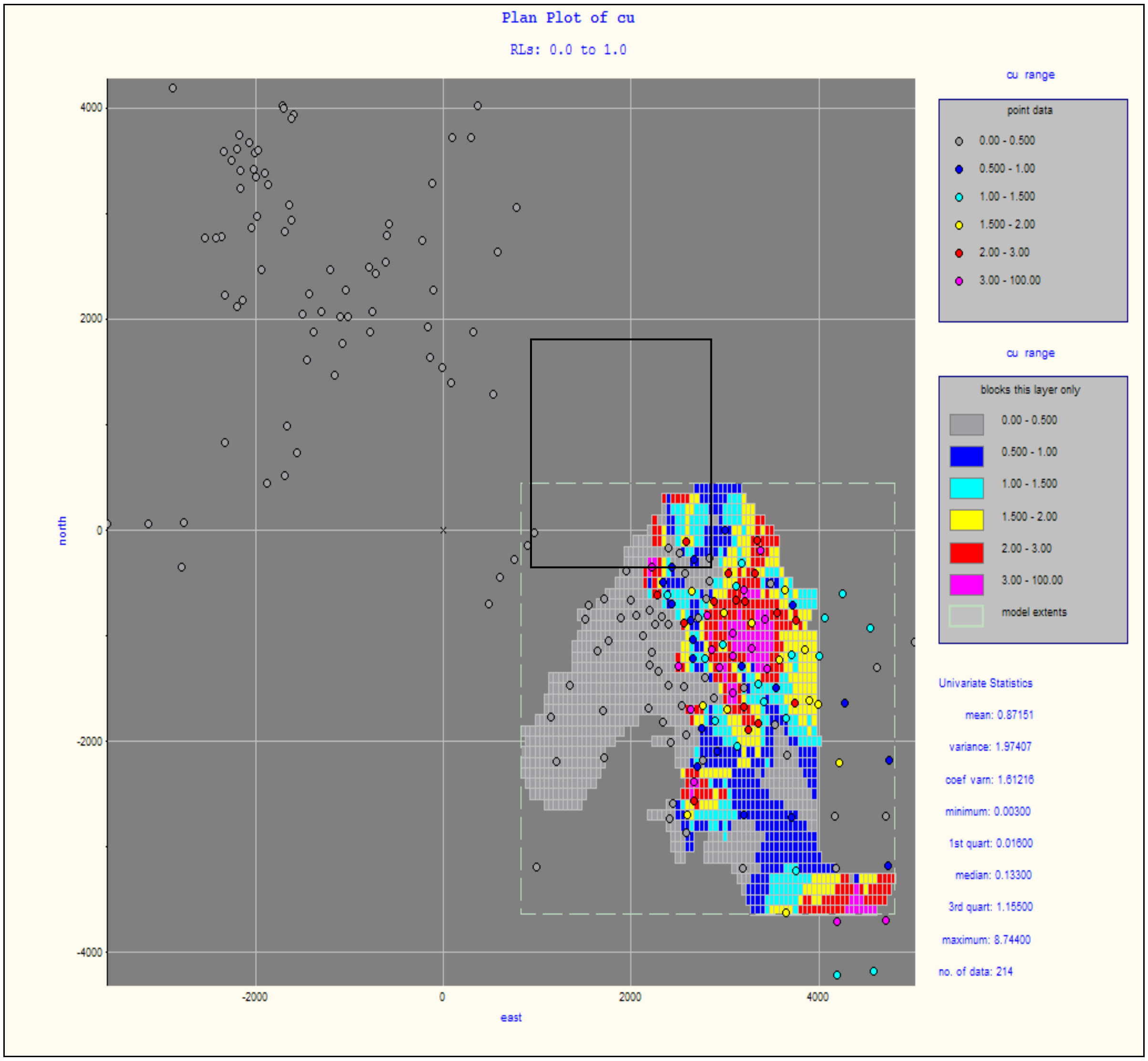
Note: Box highlights area shown in detail in Figure 60.

Figure 60: Detail of Manto 3 Copper



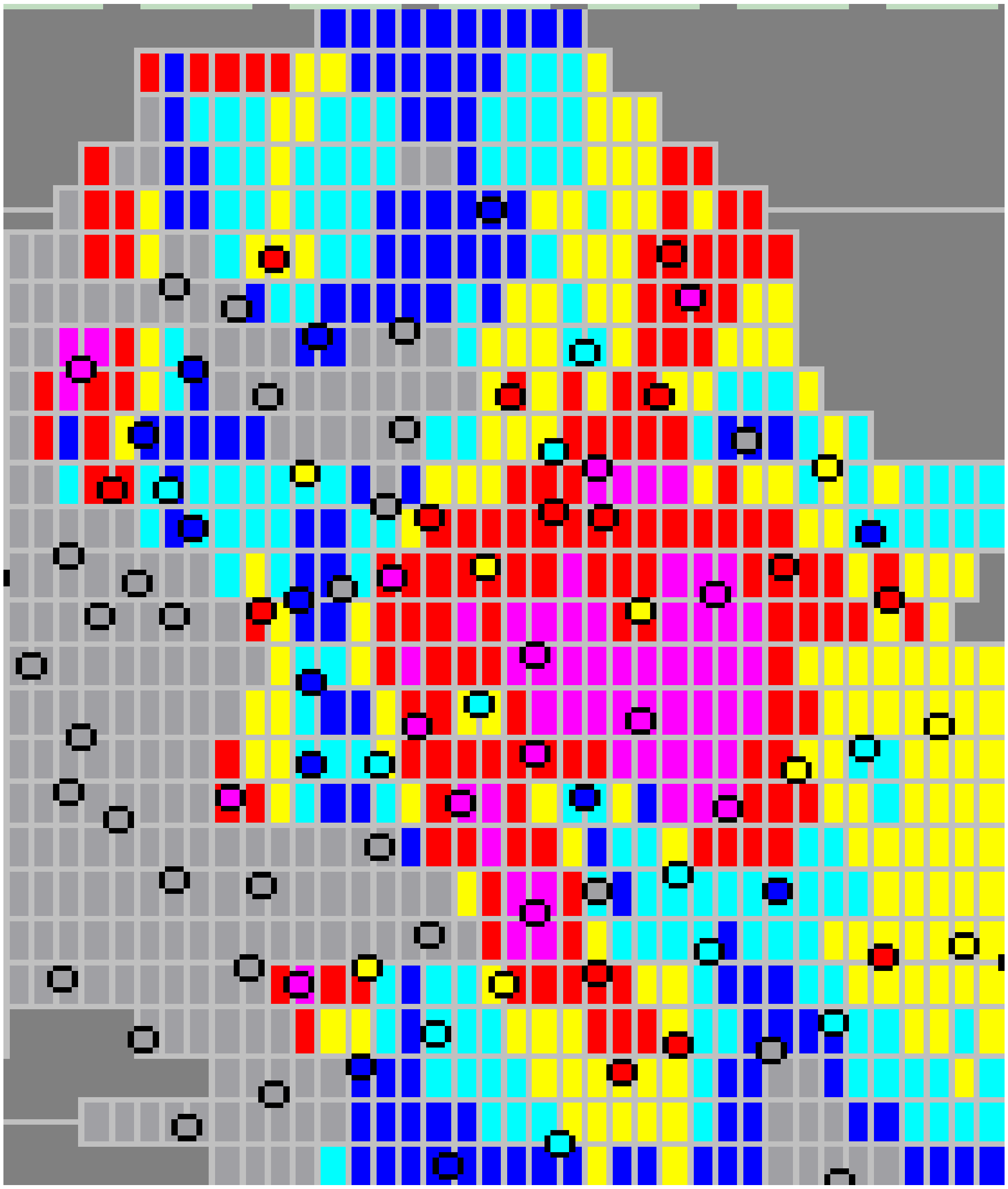
Note: The legend for Fig. 60 is as per Fig. 59.

Figure 61: Manto 1 Copper – Block Model with Assay Composites



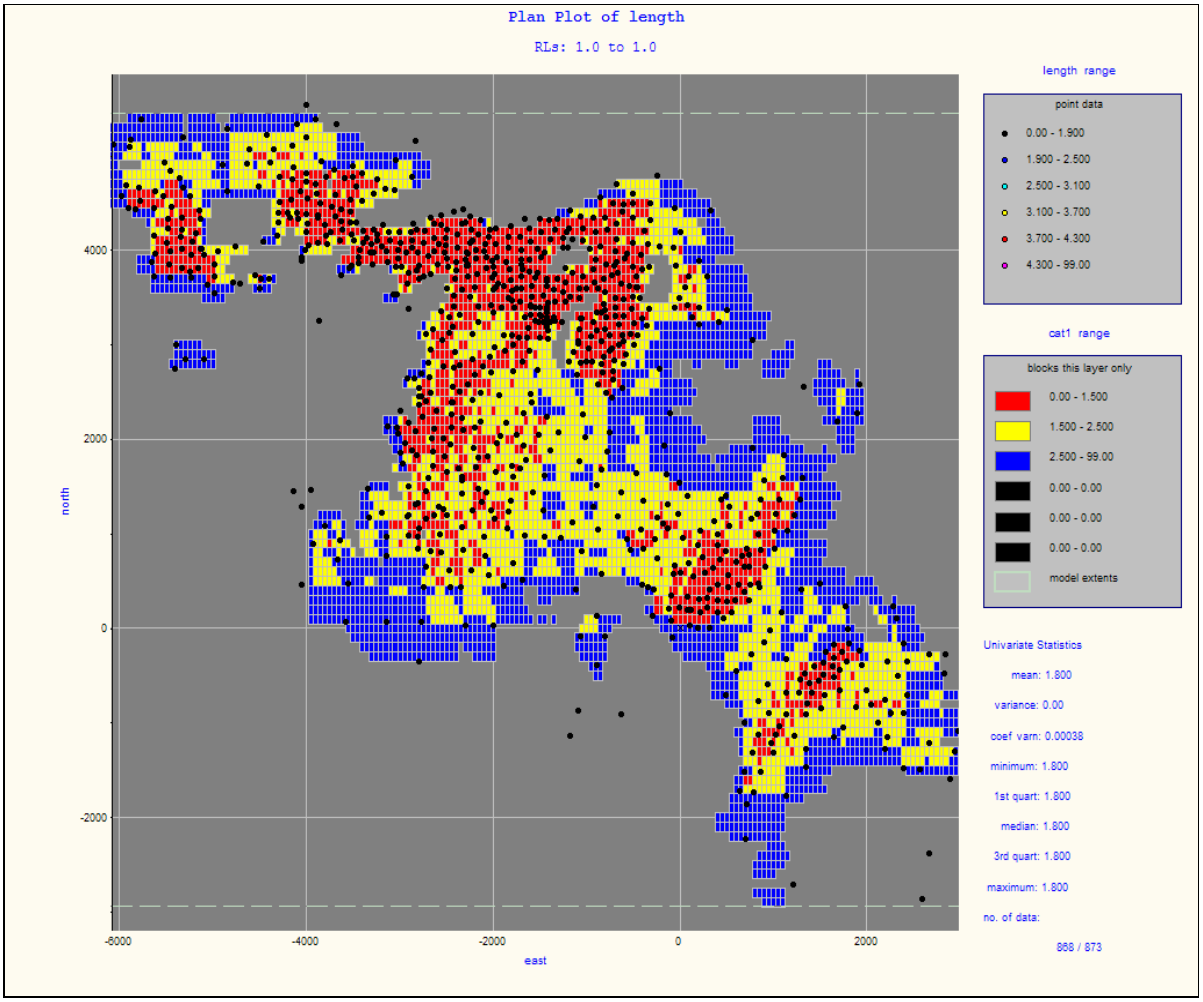
Note: Box highlights area shown in detail in Figure 62

Figure 62: Detail of Manto 3 Copper



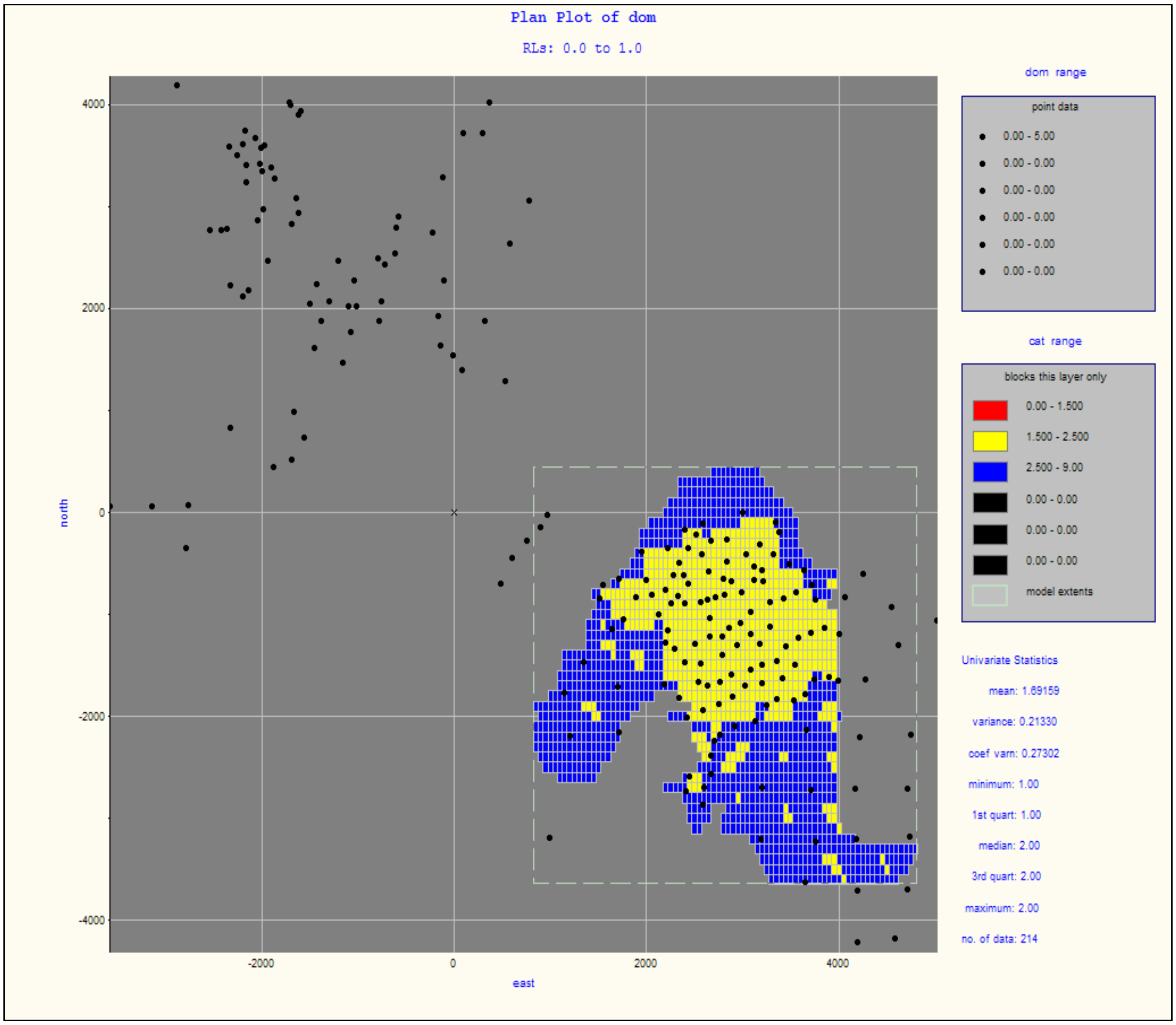
Note: The legend for Fig. 60 is as per Fig. 59.

Figure 63: Manto 3 Resource Classification – Block Model with Drill Hole Locations



Note: Measured – Red, Indicated – Yellow, Inferred – Blue (basal layer 0-1 m)

Figure 64: Manto 3 Resource Classification – Block Model with Drill Hole Locations



Note: Measured – Red, Indicated – Yellow, Inferred – Blue. (basal layer 0-1 m)

16.5 TOTAL RESOURCE

The tables below combine the 3D block models and the seam models in a single table for each cutoff grade.

- Manto 1 and 2 are seam models
- Manto 3aa, 3a, 3 and 4 are the 3D block models.

Table 50: Combined Resource at 0.50% CuEq.

Cutoff CuEq	0.50%	Manto						
		0	1	2	3aa	3a	3	4
Model Type		3D blk	Seam	Seam	3D blk	3D blk	3D blk	3D blk
Measured	Tonnes (10^6)			5.6	1.6	10.4	39.4	2.4
	CuEq%			1.88	2.29	2.03	2.24	1.82
	Cu%			0.51	0.65	0.43	1.02	0.96
	Co%			0.067	0.096	0.101	0.089	0.056
	Zn%			1.12	0.91	0.67	0.30	0.33
Indicated	Tonnes (10^6)		8.3	24.5	1.8	35.7	75.5	27.6
	CuEq%		2.61	1.64	2.04	1.36	2.03	1.10
	Cu%		1.34	0.46	0.55	0.36	1.09	0.50
	Co%		0.067	0.056	0.087	0.058	0.059	0.037
	Zn%		0.90	1.01	0.82	0.57	0.42	0.29
Measured & Indicated	Tonnes (10^6)		8.3	30.1	3.4	46.1	114.9	30.0
	CuEq%		2.61	1.68	2.16	1.51	2.10	1.16
	Cu%		1.34	0.47	0.60	0.38	1.07	0.53
	Co%		0.067	0.058	0.091	0.068	0.069	0.039
	Zn%		0.90	1.03	0.86	0.59	0.38	0.29
Inferred	Tonnes (10^6)	9.7	10.2	51.3	0.5	25.2	58.5	56.9
	CuEq%	0.56	2.20	1.48	1.92	1.19	1.51	0.87
	Cu%	0.04	0.87	0.34	0.58	0.34	0.63	0.38
	Co%	0.012	0.063	0.048	0.083	0.046	0.048	0.030
	Zn%	0.77	1.13	1.13	0.64	0.58	0.58	0.23

Table 51: Combined Resource at 1.0% CuEq.

Cutoff CuEq	1.0%	Manto						
		0	1	2	3aa	3a	3	4
Model Type		3D blk	Seam	Seam	3D blk	3D blk	3D blk	3D blk
Measured	Tonnes (10^6)			4.7	1.6	9.0	36.5	1.4
	CuEq%			2.07	2.32	2.20	2.35	2.57
	Cu%			0.57	0.66	0.49	1.09	1.42
	Co%			0.074	0.097	0.110	0.089	0.075
	Zn%			1.21	0.92	0.70	0.31	0.44
Indicated	Tonnes (10^6)		7.1	18.9	1.7	24.2	65.8	10.6
	CuEq%		2.92	1.89	2.09	1.63	2.21	1.77
	Cu%		1.54	0.55	0.57	0.48	1.22	0.89
	Co%		0.075	0.065	0.090	0.067	0.062	0.054
	Zn%		0.91	1.10	0.84	0.63	0.43	0.41
Measured & Indicated	Tonnes (10^6)		7.1	23.6	3.3	33.2	102.3	12.0
	CuEq%		2.92	1.93	2.20	1.78	2.26	1.86
	Cu%		1.54	0.55	0.61	0.48	1.17	0.95

Cutoff CuEq	1.0%	Manto						
		0	1	2	3aa	3a	3	4
Model Type		3D blk	Seam	Seam	3D blk	3D blk	3D blk	3D blk
	Co%		0.075	0.067	0.093	0.079	0.072	0.057
	Zn%		0.91	1.12	0.88	0.65	0.39	0.41
Inferred	Tonnes (10^6)	0.03	8.6	37.5	0.4	14.3	40.8	12.7
	CuEq%	1.11	2.47	1.75	2.03	1.50	1.81	1.47
	Cu%	0.02	0.99	0.41	0.63	0.49	0.83	0.79
	Co%	0.039	0.071	0.056	0.086	0.056	0.054	0.041
	Zn%	1.26	1.22	1.32	0.66	0.64	0.65	0.32

16.6 MINERAL RESERVE ESTIMATES

16.6.1 REPORTING STATUS

Mineral Resource and Reserve estimates for the Boléo Copper Project are being prepared to comply with the requirements of the Canadian National Instrument 43-101 (NI 43-101). Mine planning is well advanced on the open cut and underground sections of the orebody but the resource drilling conducted in late 2006 is yet to be incorporated in the resource model.

Mineral Reserve Estimates will not be stated until the final resource models, including the mineral resource categories, are available for inclusion in the mine plan. This is planned to occur in March 2007.

Any mining quantities or grades expressed in this Updated Preliminary Economic Assessment are subject to verification and amendment against the final resource model and as such they do not constitute a Mineral Reserve Estimate.

17 OTHER RELEVANT DATA AND INFORMATION

17.1 MINING AND MINE DESIGN

17.1.1 *OPEN CUT MINING*

Open cut operations at Boléo will consist of ore mining from the shallow sections of Manto 3 and overlying mantos in the northern half of the deposit and quarrying of limestone from the Dos de Abril area in the north western corner of the mining area.

17.1.2 *COPPER PITS*

Curator undertook extensive close spaced drilling over the northern section of the deposit during the 1990s to define an Open cut resource, mainly on Mantos 3 and 3A. Although the current mine plan focuses on extraction of the deeper, higher grade resources in Mantos 1, 2 and 3 the close drilling means that the areas most suitable for open cut mining are at a higher level of resource confidence and it is anticipated that they will be classed as either Indicated or Measured Mineral Resources and thus be available for definition of Mineral Reserves under NI 43-101.

RESOURCE MODEL

Since open cut operations allow selective mining based on close spaced grade control sampling in the cuts, a block model was used as opposed to the single layer seam models used for underground mining. Block grades were estimated as whole block values using ordinary kriging with a block size of:

- 100 m north south
- 50 m east west
- 1 m vertically.

This allows for selecting a particular mining block vertically as well as laterally.

MINING LOSS AND DILUTION

Grade control drilling is allowed for on a 20 m x 20 m pattern with samples every 0.5 m vertically through the mantos. The open cut mine plan in this report uses whole block grade estimates. The increased definition provided by the grade control should allow exclusion of lower grade portions of the large resource blocks. For this reason it is assumed that the whole block estimates represent a diluted resource and no further dilution adjustments are made. This assumption will be reviewed further prior to issue of the Mineral Reserves estimate.

Pit Wall Slopes

A series of geotechnical core holes were drilled during the mid 1990s. Geotechnical analysis of the data at that time suggested an average pit wall slope of 45°. This slope was adopted for current open cut mine planning.

Mining Methods

Four mining methods are proposed for the ore cuts:

- Conventional mining by 110 tonne hydraulic excavator and 50 tonne trucks will be used to open the initial voids in most of the cuts. Waste from this opening will be placed in dumps outside the pit, although in some cases it can placed adjacent to the pit to be pushed back into the pit void at a later date. In the case of Pit 4 in the Texcoco area, there is an opportunity for the waste to be hauled a short distance for use as bulk fill on the downstream side of the tailings dam wall.
- Once a void of 100 m to 200 m width is formed, bulldozers will be used to push overburden from above the manto into the mined out areas. Experience in many coal mines across the world has shown dozer pushing to be a cheap and effective stripping method. The Boléo overburden is well suited to dozer pushing because is relatively weak and does not contain large boulders. In most cases the push directions will be planned to keep the waste within the pit limits thus minimizing the total area of disturbed land. In some areas, such as Pits 3 and 4 at Texcoco and Pit 2 in Boléo Arroyo, Manto 3 outcrops high on the valley wall creating excellent conditions for dozer push to form out of pit waste dumps off the valley side. Any such areas will require careful planning to minimize the disturbed area and ensure long term stability of the pushed out of pit dumps.
- Overburden stripping will be stopped 5 m above the estimated position of the target mantos. Grade control drilling will then be undertaken to define the target mining surfaces and ore grades. The final 5 m will then be mined by the excavator and trucks.
- Ore zones from each manto will be mined by the 110 tonne excavator and the 50 tonne trucks will haul the ore over purpose built haul roads to the run of mine (ROM) stockpile area at the plant site at the mouth of the Soledad arroyo.

Light blasting will be used in the manto overburden to promote productivity of the excavator and dozer fleets.

Pit Selection

Whittle and Minex pit optimization software tools were applied to the Boléo deposit to guide pit selection. Both systems showed that very large pits can be economically mined. However, when the cost of overburden removal is included, open cut ore costs 3 to 4 times more to deliver to the ROM stockpile than the estimated cost of underground mine production.

The best copper grades are in the deeper sections of the mantos so the project is committed to underground mining. Given these facts, the open cuts were defined by selecting the sections of the optimized pit shells which are not well suited to underground mining because they are too shallow, too steep or too faulted or in locations where the open cut wall provides a good entry point for underground mining.

The defined pits contain just over 10 Mt of ore grade manto at an average ratio of 12 bcm waste per tonne of ore. The current mine plan averages 590,000 tonnes of potential ore per year, although this schedule requires further smoothing and matching against the underground mining schedule.

17.1.3 LIMESTONE QUARRY

The ore treatment process requires an average of 718,000 tonnes of limestone grading 65% $CaCO_3$ per year which will be sourced from a section of the Coquina fossiliferous limestone bed in the north western corner of the mining area. The Coquina bed is part of the Gloria formation which occurs high in the Boléo sequence and can be seen outcropping along many ridge lines. This section of the Coquina bed averages just over 64% $CaCO_3$.

Conventional mining with a 180 tonne hydraulic excavator matched with 90 tonne trucks is planned for both overburden and limestone. All material is blasted. The trucks haul waste to an out of pit dump in the valley immediately east of the quarry and limestone to a stockpile at the southern end of the quarry.

The 6.5 km haul from the quarry to the plant site is too long for the 90 tonne off-highway dump trucks so a wheeled loader will rehandle limestone from the quarry stockpile into purpose built 50 tonne road haulers for transport to the plant site.

Haul roads have been designed to take advantage of the natural topography wherever possible and their costing is included as part of the surface infrastructure capital cost estimate. In the case of an occasional heavy rainfall event brought about by cyclonic activity the roads will be repaired and re-built on an as needed basis. In addition, there are allowances in the capital cost estimate for pit dewatering equipment.

Part of the limestone deposit overlies potential ore grade Manto 3. This area was considered too deep to mine initially but after the southern portion of the quarry is removed, the reduced overburden depth makes it possible to extend copper Pit 1 further north.

17.1.4 OPEN CUT DEVELOPMENT SEQUENCE

Excavator mining of copper Pits 3, 4 and 5 starts in Year 1 to provide early highwall access to underground mining blocks on the down throw side of the San Antonio fault south west of Pits 6 and 7 and in the Texcoco area.

The excavator moves into the higher sections of Pit 2 in Year 2 and the dozers commence stripping in Pits 3 and 4 before moving into Pit 2 in Year 4. Pit 1 is commenced in Year 7 when the limestone quarry has mined out the sections overlying the copper pit. Pit 8 in Purgatorio Arroyo is commenced Year 11 once underground mining is finished in that area. Finally, Pits 6 and 7 are commenced in Year 14 and 15. Even though they are closest to the plant site their average grade is lower than the other pits.

Although the schedule will require smoothing it achieves its basic aims of supplementing high grade underground feed over the first five to eight years then ramping up the production rate to make up for decreasing underground tonnes and grades in the latter years of the mine life.

17.1.5 OPEN CUT MINING COSTS

Mine operating costs were estimated from first principles by sizing the mining fleet, workforce and consumables (mainly explosives) to match the production schedule.

Operating hours were estimated for each machine each year and hourly operating costs were applied for overhauls, parts, fuel, lube, tyres and wear parts.

Lease costs were estimated for all mining equipment assuming a 10% lease rate on rolling 5 year terms. This was done in preference to estimating an equipment capital schedule because it spreads the capital costs over time and it makes the overall mining costs closer to a contract mining cost which is one of the options under consideration.

Explosives consumption was estimated against the mining volumes and estimated powder factors. Costs per tonne were applied against ANFO and emulsion usage and a percentage allowance was made for blasting accessories such as detonators, primers and detonating cord.

Total annual costs were estimated for each employee classification and applied against the workforce schedule.

Total grade control drilling metres and numbers of samples were estimated against the ore mining schedule.

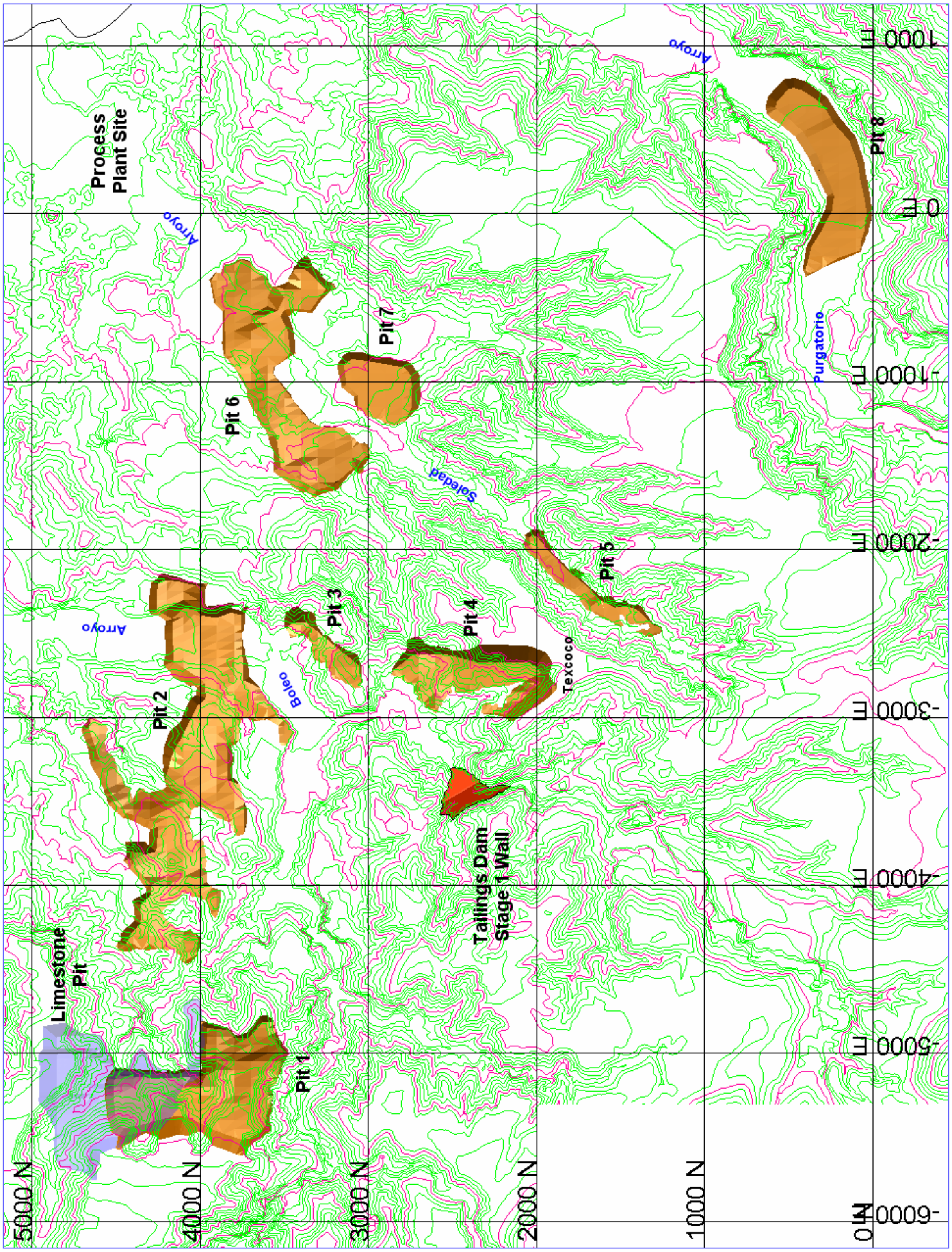
A contingency was not applied to the total estimated cost to cover items, but this will be examined in further detail as the final feasibility study is prepared during the first quarter of 2007.

Total open cut mining costs averaged:

- Copper pits ore and waste combined — US$2.15/bcm or US$1.14/t (wet)
- Limestone quarry ore and waste combined — US$3.55/bcm or US$1.64/t
- Copper pits, total cost ore to ROM — US$19.79/t (dry)
- Quarry, total cost limestone to plant — US$10.65/t.

The open pit locations are shown below in Figure 65.

Figure 65: Open Cut Pit Locations



17.2 UNDERGROUND MINING

Underground mining operations at Boléo will consist of room-and-pillar mining with pillar extraction. The initial mining approach is to target and sequence mining extraction to take advantage of the higher copper (Cu) grade “bull’s eyes” that exist in each of the mantos during the early production years. The project life is a minimum of 20 years. Preliminary mine planning has been based on a geologic model provided by Dr W Yeo of Hellman & Schofield Pty Ltd.

Underground mining will take place in Mantos 1, 2, and 3 where (1) there are reasonably sized, contiguous mineralized zones expected to be classified as either measured, indicated or inferred in the current resource model, (2) the mantos have a Cu content greater than 0.2% at a minimum mining height of 1.8 m, and (3) the mantos have an overburden thickness of at least 20 m. Approximately 70% of the underground mining is in Manto 3, 16% in Manto 2, and 14% in Manto 1.

Of the approximately 48.8 million underground dry tonnes initially modelled for the Preliminary Economic Assessment (PEA), none are measured, approximately 38.5 Mt are indicated, and approximately 10.3 Mt are inferred. An additional 10 Mt of manto ore are required to be mined during the last 4 years of the initial mine plan and were not modelled as of this publication date.

The underground mining resource was divided into independent mines, each of which can be accessed from logical portal locations (manto outcrops, box cuts and/or shallow slopes). To the extent practical, the mines segregate the resource by copper grade. The primary objectives for sequencing the mines were winning the higher copper grade ore first, while optimizing the initial plant feed grade at 2.3% Cu and then maintaining a higher feed grade for as many years as practical. The copper grade gradually decreases by year for the life of the project. Production and mined ore grades for the first 16 years were modelled using SurvCADD™ software, with production and grades for the remaining 4 years of the project life estimated by percent aerial recovery.

Underground mine production starts with one mining unit approximately 9 months prior to plant commissioning. Production is systematically ramped up over the next 21 months to five mining units operating with three production crews each and a targeted annual production rate of approximately 2.6 million dry tonnes per year, coinciding with the start of the second year of plant operation. Underground production is maintained at 2.6 Mt through Year 5, when a sixth underground mining unit is added to compensate for declining ore grade, increasing raw ore production by 20% to approximately 3.1 Mdmt/a.

Beginning in Year 15, underground production is gradually decreased to approximately 1.5 Mmt/a and maintained at that rate through Year 19. In Year 20, the production is reduced to approximately 0.75 Mt. The reduction in underground production between years 15 through 20 is expected to be offset by equivalent production from open-cut mining.

UNDERGROUND MINING METHOD

As a consequence of the Boléo mantos depositional and geotechnical similarities with coal seam deposits (bedded and relatively soft ground), the recommended underground mining method is similar to those successfully used for coal, trona, potash, and salt seam room-and-pillar mining with pillar removal in North America, Australia, and South Africa.

Underground mining will utilize (1) a combination of hydraulic excavators, continuous miners, rubber-tired batch haulage, and mobile roof-bolting equipment for access drift and pillaring crosscut development and (2) remote-controlled continuous miners, continuous haulage, and mobile roof supports (MRS) for pillar extraction. Development mining and pillar extraction may be independent activities with different equipment and crews; however, mine scheduling and synchronizing mine production with plant feed requirements dictates some units conducting both development and pillar mining operations. For both development and pillar removal, ore haulage from the producing units to the portal stockpile will be by belt conveyor.

Ore from the underground mines will be transported to the plant either by haulage truck from the surface stockpiles at the mine portal or via an overland conveying system. Both systems are expected to work on a 24/7 basis.

Depending on mine size, three or four parallel drifts (access mains), connected at regular intervals by crosscuts, will be excavated from the mine portal through the manto to access mining districts. All mine portals will be established above the typical food levels and where this is not possible flood protection measures will be taken to minimize the possibility of flooding the workings. Emergency drainage equipment is included in the capital cost estimate for the underground mine.

Where mine conditions and production scheduling allow, development work for an entire district will be completed prior to initiating pillar removal. Multiple, independent districts will be simultaneously mined in various stages of development and pillar removal. Equipment will be moved between districts as needed. A conceptual mine plan for a mining district is shown in Figure 66 and a larger area map is shown in Figure 67.

UNDERGROUND RESOURCE RECOVERY

Underground resource recovery will be impacted by extensive faulting, steep manto dip, mining method/equipment limitations, and historical mine workings. Except where access drifts cross faults, a 10 m buffer is left unmined on both sides of faults. Areas where the manto dip exceeds 25% are not mined. To account for mining method/equipment limitations and historical mine workings, it is assumed that only 95% of the development excavations and 75% of pillared areas will be recovered. These assumptions will be reviewed prior to issuing the mineral reserves estimate.

UNDERGROUND RESOURCE GRADE DILUTION

The underground resource model for each manto initially calculates ore grade for a 1.8 m interval. Where total manto thickness is less than 1.8 m, the grade is diluted to account for a minimum 1.8 m mining height. Where the manto thickness is greater than 1.8 m, the grade of each adjacent 0.6 m intervals are sequentially checked. If the immediately adjacent 0.6 m interval exceeds 1% Cu, 0.6 m is added to the resource height and average grade is adjusted to include that additional increment, and only then is the next 0.6 m interval evaluated for inclusion in the resource model.

The mining equipment selected for Boléo can adapt to infinitely variable mining heights between 1.8 and 4.2 m. Within those mining limitations, the excavation height will approximate the grade interval, not the 0.6 m increments of the resource model. Consequently, the resource model already accounts for reasonable grade dilution and no additional dilution adjustments are appropriate. This mechanism for addressing grade dilution during the mining operation will be reviewed prior to issuing the final mining plan.

Underground Geotechnical Mine Design

All underground pillar and excavated opening dimensions used for mine layout and design are based on (1) observations made and documented[6] during test mining at Boleo's Texcoco mine during the last quarter of 2005 and 2006, and (2) a geotechnical study conducted by AAI.[7]

Underground Productivity

Productivity at the underground mines is based on observations made during test mining at the Texcoco Mine including a roof-bolting productivity test conducted in November of 2006 and AAI's extensive experience with similar production equipment in coal, potash, and trona mines. It is assumed that developing mining units will produce 490 dt per unit shift and pillaring units will produce 2,536 dt per unit shift.

Underground Mining Costs

Underground mine costs were based on the following:

- Capital/leasing costs were estimated on an annual basis for the equipment and infrastructure required to produce the required ore tonnage and grade from the proposed underground mine sequencing plan.
- Labour costs were estimated for each employee classification and applied against a detailed workforce schedule developed to produce the required ore tonnage and grade from the proposed underground mine sequencing plan.
- Maintenance cost per dry tonne of ore produced was estimated by averaging actual maintenance costs from five US underground coal mines utilizing similar equipment as proposed for Boléo.
- Supply cost per dry tonne of ore produced was estimated by calculating the anticipated installed roof support cost per ton of ore produced and adding an estimated cost per ton of

[6] Agapito Associates, Inc. (2006), Preliminary Geotechnical Performance Study for Underground Mining of El Boléo Copper Cobalt Project, Texcoco Test Mine Including Operations Observations and Recommendations, draft report to Baja Mining Corp, July.
[7] Agapito Associates, Inc. (2007), "Geotechnical Evaluation for Underground Mine Design," draft report to Baja Mining Corp, February.

other supplies based on experience from US underground coal mines utilizing similar mining methods as proposed for Boléo.

Figure 66: Conceptual Mine Layout for a District

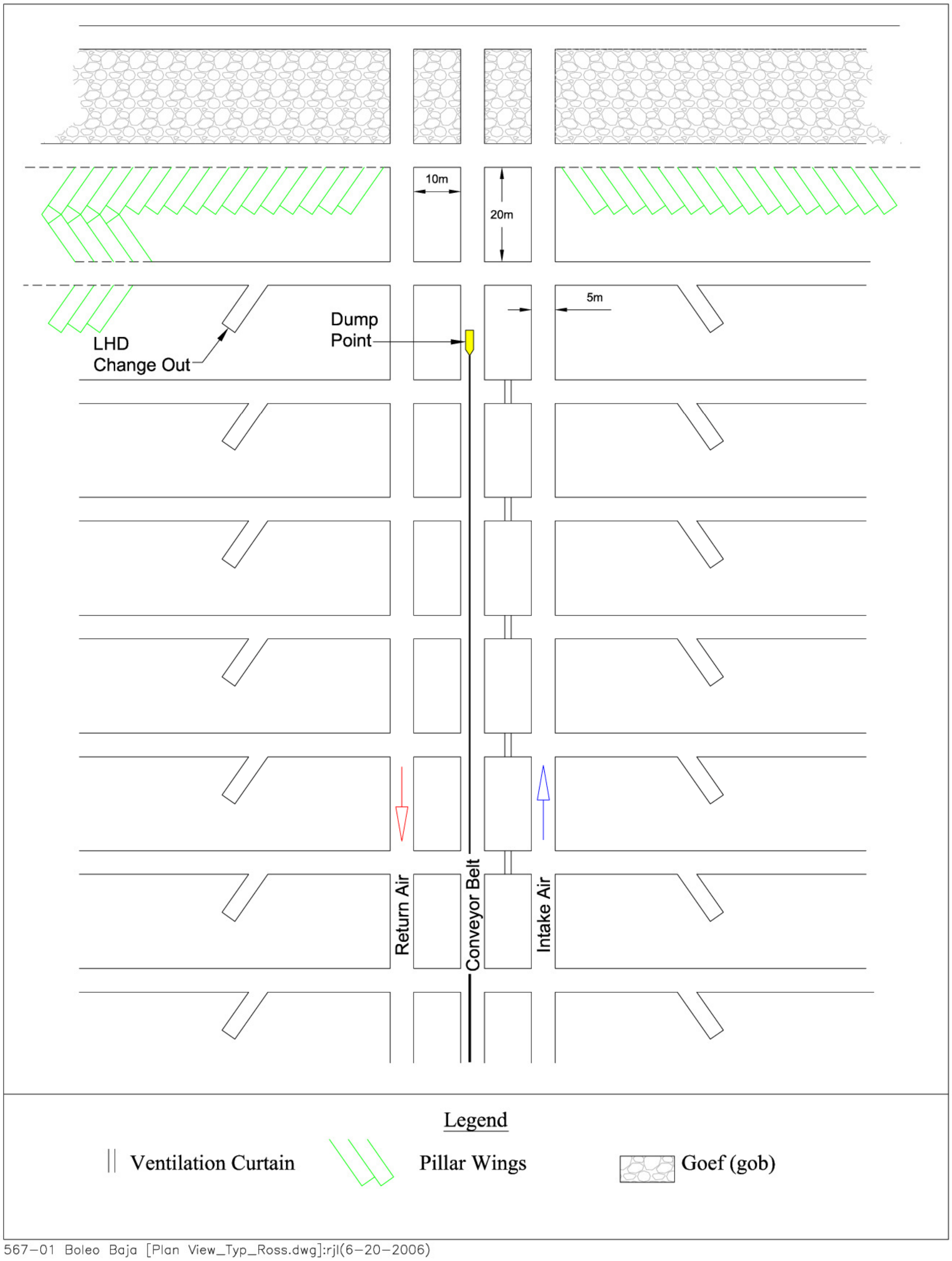
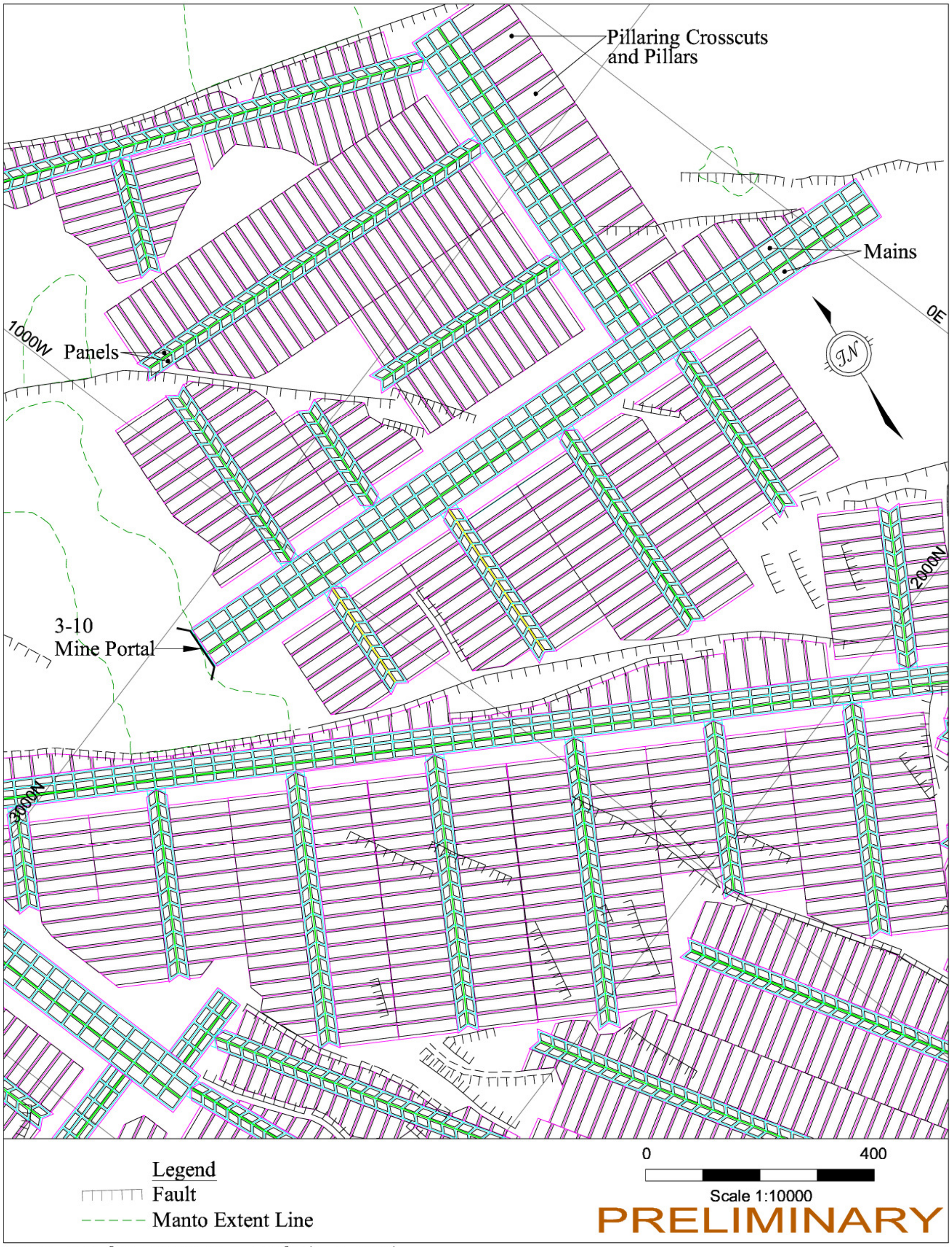


Figure 67: Mining Area 3-10 Preliminary Layout



567-01 Boleo [Typ Blocking TR_1-2007]:rjl(1-11-2007)

18 PRELIMINARY ECONOMIC ASSESSMENT

18.1.1 SUMMARY

This Preliminary Economic Assessment ("PEA") of El Boléo project is based upon:

- The mineral resource estimate for copper, cobalt and zinc prepared by Hellman and Schofield discussed in Section 15
- The process flowsheet developed by Bateman and recoveries of copper, cobalt and zinc achieved during the Fully Integrated Pilot Plant testing program at SGS Lakefield Research Ltd., Lakefield, Ontario, conducted under the joint guidance of Baja Mining Corp. & Bateman, as discussed in Section 14
- The open cut mine design developed by Australian Mine Development and Design, as discussed in Section 16.1
- The underground mine design developed by Agapito Associates, Inc., as discussed in Section 16.2.

Reference is made to sections 14, 15 and 16.1 and 16.2 for a detailed discussion of the Process Flowsheet, Mineral Resource Estimate and Mine Design.

US$ and Metric measures are used throughout this section.

The current base-case is for annual mine production to deliver 3.5 Mwt (2.6 Mdt) per year to the process facility; with maximum annual metal production of 50,000 tonnes of copper, 2,600 tonnes of cobalt and 30,000 tonnes of zinc sulphate monohydrate.

Capital cost of the construction of the mine and process plant complex is currently estimated at US$540 million (including Owner's Costs and Contingencies). Total Operating costs are estimated at US$31.62/dt of ore treated, averaged over the 20 year project life being modelled. During the initial 20 years 58.5 Mt of ore will be processed. The Total Measured and Indicated Resource is 232.8 dmt, potentially leaving a large amount of ore to be processed after the initial 20 years.

Preliminary mine scheduling was based upon the scheduling of resource blocks which include Inferred material, using copper equivalent grades and process recoveries. A copper equivalent grade was determined using the following base-case metal prices:

- Copper – US$0.95/lb
- Cobalt – US$12.00/lb
- Zinc – US$0.45/lb
- Copper recovery – 91.2%
- Cobalt recovery – 78.2%

Base Case financial modelling of the project was done using conservative long term metal prices. The assumed Base Case long term copper price is $1.25, starting in 2013, with step down pricing between current levels and long term according to the LME 5 year forward price curve published as of January 25, 2007. Those prices are $2.20 in 2009, $1.95 in 2010, $1.75 in 2011, and $1.50 in 2012.

Expressions of interest have been received from potential offtake partners indicating that net-back price of copper would include a slight premium (assumed to be $0.02/lb) above the LME, or COMEX, price after adjusting for freight. The Base Case price for cobalt is $12.00/lb flat over the life of the project. The Base Case price for zinc sulphate is $950/mt flat over the life of the project. Provision is made for delivery of the products to the end markets, including packaging, freight, and freight insurance.

The modelling, based on the current un-optimised preliminary mine schedule, indicates that the project is financially attractive at base-case metal prices. Financial modelling, using the base case prices and only 20 years for the project life, shows that the project could generate a net after- tax Internal Rate of Return (IRR) of 19.0% with a discounted present value, at an 8% discount rate, of US$333.4 million.

Using a 6% discount rate generates an NPV, after tax, of $US445.9 million.

Cash flow analysis was also conducted using the following metal price assumptions to test the project's economic robustness and sensitivity to changes in those prices, as shown in Table 51:

- The SEC approved 5 year average prices, comprised of the weighted average of the 3-year trailing price and 2 year leading price. This calculates to be $2.20/lb for copper and $16.00/lb for cobalt.
- The current prices which, as of the end of January 2007 are $2.50/lb of copper and $26.00/lb of cobalt and $1,500/ts of zinc sulphate monohydrate.
- An "Opportunity Case" is also shown to demonstrate the potential impact of adding the recovery of Manganese (as Manganese Carbonate) to the basic project.

Table 52: Preliminary Economic Assessments – Base-Case Highlights

Description	Quantity or Value
Preliminary Mine Production Schedule	2,600,000 dmt/a, increasing to 3,100,000 dmt/a in Year 6
Metal Production	Up to 60,000 t/a Cu cathode Up to 3,100 t/a Co cathode Up to 36,000 t/a $ZnSO_4$ salt
Capital Cost	US$540 million
19 year average Operating Cost, excluding start-up year	US$31.62/t of ore
Long term metal prices	Copper – US$1.25/lb Cobalt – US$12.00/lb ZSM – US$950/t
(After tax) Internal rate of return (IRR)	19.0 %

The project is sensitive to 4 key variables: Copper price; cobalt price; the capital cost and to operating costs. The sensitivity of the After-Tax IRR and NPV (at 8% Discount Rate) relative to the Base Case is shown in the table below to indicate the effect of plus or minus 10% changes in the key variables. Note that the changes to the Copper price apply to all of the annual prices, starting in 2009, and not just the long-term price.

Table 53: Sensitivity to Key Variables

	After Tax IRR				After Tax NPV at 8% ($Millions)		
	-10%	Base Case	+10%		-10%	Base Case	+10%
Copper Price	16.0%	19.0%	21.7%		$236	$333	$420
Cobalt Price	18.1%	19.0%	19.9%		$298	$333	$367
Capital Cost	21.5%	19.0%	16.6%		$375	$333	$283
Operating Cost	20.7%	19.0%	17.1%		$396	$333	$266

18.1.2 *Product Marketing*

The processing facility at El Boléo property will produce London Metal Exchange (LME) grade copper cathode (metal), and high purity cobalt metal. It is not proposed to produce zinc metal but rather to evaporate the zinc sulphate stripped in the zinc solvent extraction circuit to produce zinc sulphate monohydrate for subsequent sale into the soil micro-nutrient market, animal feed market, or delivery to a zinc refinery.

Off-take agreements with respect to the above products have not yet been negotiated, but “Expressions of Interest” have been received from potential offtake candidates.

The copper cathode produced on site is expected to exceed LME purity specifications for sale and may command a premium on LME pricing. Indicative terms from offtake parties indicate that a premium of $0.02/lb above LME could be expected and this assumption has been built into the pricing assumptions. If an off-take agreement is not negotiated for the copper it could be offered for sale on the London or Comex metal markets.

The current process flowsheet provides for the production of cobalt metal on-site. For purposes of this study it is presumed cobalt metal will be marketed through an off-take agreement that will be negotiated prior to the commencement of production. Costs for packaging (drums), freight insurance, and freight to market will be deducted from the indicated selling prices.

The zinc sulphate monohydrate is considered to be a “value added” product and should command a price equivalent to the value of contained zinc metal, or higher. It is assumed that the material would be sold through an agent and that the capital cost of the production facility would be paid for by the agent’s company and recovered through a processing fee. The net price received by Baja is also adjusted for costs of packaging (jumbo bags), freight to market, and the agent’s selling commission.

18.1.3 CAPITAL COST ESTIMATE

The capital cost for development of the Boléo Project has been developed by a number of specialist organizations. These organizations are listed below in a table that summarises major areas of significant capital cost and the organizations responsible for development of capital costs for these respective areas. The Capital Cost Estimate for the project development has been co-ordinated and integrated by Wardrop Engineering on behalf of Bateman Engineering Canada Corp.

Table 54: Capital Cost Areas of Responsibility

Major Cost Area	Consultant	Location
Open Pit Mining	AMDAD	Sydney, Australia
Underground Mining	Agapito Associates, Inc	Golden, Colorado
Mining Surface Infrastructure	Wardrop Engineering	Vancouver, Canada
Process Plant and General Infrastructure	Wardrop Engineering	Vancouver, Canada
Tailings Dam	Arcadis Geotechnica	Santiago, Chile
Co-Generation Plant	Fransen Engineering Ltd	Vancouver, Canada
Acid Plant	Fenco Pty Ltd	Toronto, Canada
SO_2 Gas Production Facility	Noram Engineering & Constructors Ltd	Vancouver, Canada
Barging Facility	ATI	Vancouver, Canada
Liquid Sulphur Infrastructure	ICEC Canada Ltd	Calgary, Canada
Mexican Construction Labour Rates	UHDE Jacobs	Mexico City

The overall capital cost is estimated at US$540 million.

This number includes the capital cost components of all infrastructure, the process plant, and the mining operation, including the mining fleet, mining infrastructure, tailings dam, haul road construction, electrical power and water reticulation to the various mine sites, waste disposal, construction camp and various community initiatives.

Note that although these capital cost estimate numbers are still under review by both Bateman Engineering Canada Corp and Baja Mining Corp. and have not yet been signed off internally by either organisation it is believed that there are no significant omissions in the analysis.

Table 55 provides a summary breakdown of the estimated total project capital costs. The base date of the cost estimate is July 31, 2006. All costs are listed in US$. The numbers are rounded to reflect that they are approximations.

Table 55: Capital Cost Estimate Summary

Project Area	Estimate Capital Cost US$
Mining & Tailings	45,563,000
Process Plant	181,950,000
Services & Infrastructure	116,493,000
Buildings (incl. Construction Camp)	15,316,000
Construction Indirects & Freight	38,171,000
Sub-total – Direct Field Costs	397,493,000

Project Area	Estimate Capital Cost US$
EPCM	45,433,000
Owners Costs	35,681,000
Contingency	62,041,000
Total – Project Capital Cost	540,648,000

The Capital Costs set out in the above table exclude:

- mine rehabilitation costs
- mine closure and environmental costs
- working capital
- capital spares
- first fills of reagents
- sustaining capital.

Provision for the above costs was made in the economic model as individual line items. Attempts will be made to minimize Working Capital requirements through the involvement of off-take agreements offering prompt payment for metals following transfer of title as the products leave the plant gate.

The project evaluation has been presented for a 20 year life, but it is expected that the life of the project will be considerably longer. There is no credit given for recapture of the salvage value of the equipment after the initial 20 years of operation. It is assumed that any costs for closure and reclamation of the process plant would be adequately covered by the salvage value of the plant. Reclamation of the open cut mines is expensed as it is incurred during the initial 20 year life on an ongoing basis, so that there will not be a significant closure and reclamation cost associated with mining.

18.1.4 OPERATING COSTS

Operating costs were developed by Bateman and Baja from the following sources:

- Extensive bench scale metallurgical and pilot plant test work data
- Quoted budget prices for reagents and consumables, typically from North American suppliers
- Appropriate labour costs for Expatriates and Mexican nationals for project development in the Baja California area of Mexico, drawing on remuneration experience from the local Gypsum Project, adjacent to the Boléo Resource on the Baja Peninsula and other Mexican Operations
- Maintenance costs based on other plant operations of a similar nature
- Estimates of open cut mining costs from AMDAD Pty Ltd

- Estimates of underground mining costs from Agapito Associates, Inc based on the Boléo test mine experience as well their operating costing data base of similar operations in North America in coal, potash and trona
- A factored approach to product marketing and product freight costs based on discussions with freight forwarders, shipping agents and interested off take parties.

Note that although these operating cost estimate numbers are still under review by both Bateman Engineering Canada Corp and Baja Mining Corp. and have not yet been signed off internally by either organisation it is believed that there are no significant omissions in the analysis.

18.1.5 PRELIMINARY ECONOMIC ASSESSMENT

SUMMARY

The financial model utilizes the current preliminary mine production scheduled over a 20 year mine life, the associated diluted metal grades based on the H&S geological resource and AMDAD and Agapito mine plans, the capital and operating costs as submitted by Bateman Engineering, and projected metal prices. The Economic Analysis of the Boléo project has been done on an all equity basis (with no financial leveraging) using the Discounted Cash Flows, after taxes, for the construction period and first 20 years of the project life. The project is not limited to 20 years by the orebody, and it can be expected that the project will continue beyond that length of time, adding to the 20 year value of the project. The projected cash flows allow for all capital expenditures, including construction, working capital and sustaining capital. Since this is an unleveraged model no financing costs or interest were accounted for in the Project's costs.

This Preliminary Economic Assessment includes the use of some inferred resources to enable a 20 year project life to be modelled. Approximately 81% of the ore mined in this model is quantified as Measured or Indicated. Some Inferred resources were scheduled, primarily in the last 5 years of the life of the project to complete the desired 20 year schedule but conservative estimates have been used for the grades mined in those years. It is expected that the incorporation of the results from the 18,000 m of drilling that was completed between October and December, 2006 will upgrade these resources into the Indicated or Measured categories, enabling a full 20 year project to be modelled.

On an all-equity basis the PEA indicates that El Boléo project has a payback period for invested capital within 42 months of start-up of commercial production of copper (on an after-tax basis). The potential net present value of the project utilizing base-case prices at a 6% discount rate is US$445.9 million, or $333.3 million at 8%, and the internal rate of return on investment is 19.0% on an after tax basis.

Using the assumed "Base Case" metal prices the annual revenue from cobalt and zinc sulphate sales is potentially adequate to cover almost all of the operating costs. This will result in a cash cost of copper production, net of by-product credits just slightly above zero cents per pound of LME grade copper production. Using higher by-product prices, but still below current levels, will drop the net cash cost of production of copper below zero. The addition of a Manganese product could also drop the net cash cost of copper to less than zero.

BOLÉO PROJECT ECONOMIC ANALYSIS ASSUMPTIONS AND DISCUSSION

Capital Cost and Expenditure Timing

The total project capital costs for the construction of the mine and process plant are detailed in Sections 17.1.3 and 17.1.4 of this report. The timing of expenditures is based on an accelerated project schedule that enables the start-up of the copper production circuit in 2 years, with construction continuing after that time on the DSX circuit for the recovery of cobalt and zinc. At this time there is no allowance for the capital cost for the construction of the manganese carbonate circuit, and it remains an opportunity for future production.

It is expected that there will be an offset of approximately 8 months between the start-up of copper production, and that of cobalt and zinc. This philosophy was adopted to allow the operating team to focus on the start-up of the main revenue generator (copper) without the distraction of simultaneously starting up the more complex DSX circuit. This should result in a faster ramp up rate for copper which will more that offset the loss of revenue from the other two products, and make the most efficient use of the technical resources. This schedule will also enable more efficient use of a smaller construction crew and minimize the size of construction camp required, along with potential impact on the infrastructure of the neighbouring town of Santa Rosalia and its residents.

Although the projected construction schedule indicates that the DSX circuit can potentially be built in 11 months, the economic model has allowed for a full year of stagger between the two start-ups.

A summary of the Boléo Project development timing is shown in Table 56. The timing of these activities forms the basis of timing of the cash flows in the economic model.

Table 56: Summary of Boléo Project Development Timing

Year	Activity	Percentage Expenditure
-2	Mine pre-development Site preparation for process plant Commencement of construction of acid plant	10%
-1	Construction of Site and Process Plant Underground development Construction of permanent wharf	70%
1	Start-up of copper production Construction of DSX circuit, cobalt EW, ZSM granulation	20%
2	Start-up of cobalt and zinc production	

Sustaining Capital

In the economic analysis provision is made each year for sustaining capital. This allowance covers cost associated with capitalized rebuilds, refurbishment rebuilds, and replacement of equipment and major spares to maintain the operation at the design capacity.

The provision for sustaining capital includes:

- General allowance based on 3% of the direct capital costs of the process plant and infrastructure
- Mine mobile equipment rebuilds and replacements from Year 6 onwards
- Development of underground mine portals
- Lifts of the tailings dam in years 4 and 9.

Working Capital

In the cash flow analysis an estimate of annual working capital requirements is made, and includes:

- The net balance of accounts payable and receivable
- Inventories, including first fills of reagents and capital spares
- In process inventories
- Finished product inventories up to the point of transfer to the purchaser. This is expected to be a warehouse at the port of Guaymas as has been discussed with interested offtake parties.

Revenue

- Net Sales Revenue from the sale of copper is calculated on the basis of the LME (or COMEX) price, plus a premium, and minus the total cost of delivery to the buyer including freight and insurance. It is assumed that copper will be delivered FOB to a warehouse at the port of Guaymas. The premium received depends on quality and market conditions, which may vary. It is assumed that the quality will be LME Grade A. An average premium of $0.02/lb is assumed. A long term LME price of $1.25/lb is assumed for copper. As the project will be starting up in 2009 the published forward pricing curve is used for the first four years of production (2009 to 2012), and the long term price is used thereafter.
- A long term price of $12.00/lb is assumed for cobalt, relative to the current level of $26, or the 5 year average of $16. Deductions are made for packaging (drums) and freight. The quality of cobalt cathode produced during the pilot plant campaign was very high and would indicate that a premium may be available, but this is not assumed in this analysis.
- Zinc Sulphate Monohydrate is assumed to be sold into the US fertilizer and animal feed markets, using agents. Provision is made for freight into the core area of use and for a selling commission to the agents. A long term selling price of $950/t of ZSM is assumed, before the deductions for freight and commissions.

Depreciation and Taxation

- Mexican tax laws provide for 2 methods of calculation of depreciation. Assets can either be depreciated at a rate of 12% for 8 years, with the remainder taken in the ninth year or at the rate of 87% in the first year of operation with the remaining 13% taken at the end of the project life. For the purposes of evaluation of this project, under the all equity funding assumption, it has been determined that it is tax effective to claim the 87% depreciation in the first year, and the remainder in the final year. This assumption could change when the project financing is put into place.

- Mexican federal income tax rates are currently 28%. There is no state income tax. Tax rates have been decreasing at a rate of 1% per year and it has been stated that the intent is to continue decreasing the rate until it reaches 25%. However, the legislation enabling this has not been passed by the current congress and until it is in place the project has assumed that the rate remains at 28%.
- There is a Value Added Tax in Mexico and the applicable rate is 10% for Baja California. The tax is paid on purchases and recovered on sales, so that the full amount is normally recoverable by mining projects, which has been assumed for this project.

Manganese Opportunity Case

- The Boléo orebody contains a substantial amount of Manganese (Mn) which could easily be recovered, with a minor capital cost increment. The technology was tested and proven during the fully integrated pilot plant campaign conducted at the SGS Lakefield facility in June, 2006.
- There is a substantial operating cost increase, however, due to the consumption of soda ash to produce the manganese as a carbonate.
- The quality and purity of the manganese carbonate is high, however market studies have indicated that there is a limited market for this product per se, and it would likely be used as an intermediate product for the production of higher value forms of manganese compounds.
- Manganese sulphate could easily be produced at site, or at other locations closer to the markets, for use in fertilizer and animal feeds. This would be synergistic with the planned production of zinc sulphate.
- This market could absorb approximately one third of the anticipated production of manganese from Boléo.
- Another potential end product for the conversion of manganese carbonate is electrolytic manganese dioxide. The current estimated price of manganese carbonate is $500/t ($1,100/t of contained Mn) and that of manganese sulphate is $550/t ($1,212/t of contained Mn).
- The addition of a large quantity of manganese carbonate into the market could however negatively affect pricing. An opportunity case is included in the scenarios below, which assumes a selling price of $400/t for manganese carbonate.

Tables 57 to 64 contain outputs from detailed calculations of the project economics at various metal prices.

Table 57: Boléo Preliminary Economic Assessment – Base Case Summary

		Pre-Start-up	Yr 1	Yr 2 - 5	Yr 5-10	Yr 10-15	Yr 15-20	Yr 1 - 20
Capital (total)		($432,000)	($108,000)	Average	Average	Average	Average	Average
Ore grades:	Cu	%	2.10	2.19	1.54	1.33	0.83	1.47
	Co	%	0.08	0.12	0.09	0.09	0.07	0.09
	Zn	%	0.62	0.74	0.63	0.58	0.33	0.57
	Mn	%	3.14	2.31	2.50	2.88	1.86	2.43
Ore treated		t/a	1,170	2,633	3,120	3,120	3,120	2,925
Production:	Cu	t/a	22,408	52,682	43,934	37,901	23,560	38,006
	Co		0	2,251	2,298	2,074	1,610	2,048
	ZSM		0	33,607	37,075	34,151	19,415	30,928
	$MnCO_3$		0	0	0	0	0	0
Revenue:	Cu	$000/a	$109,664	$188,418	$123,002	$106,113	$65,962	$116,936
	Co		$0	$59,550	$60,800	$54,862	$42,599	$51,475
	ZSM		$0	$31,926	$35,221	$32,443	$18,444	$27,912
	$MnCO_3$		$0	$0	$0	$0	$0	$0
	Total		$109,664	$279,894	$219,023	$193,418	$127,005	$196,323
Op. Costs:	Mining	$000/a	$26,635	$28,520	$26,687	$26,213	$25,470	$26,628
	Process		$22,186	$59,949	$67,492	$67,446	$56,153	$60,872
	G & A		$1,876	$2,396	$1,916	$1,926	$1,911	$1,915
	Sales, Dist'n		$199	$4,422	$4,718	$4,324	$2,515	$3,784
	Total		$50,896	$95,287	$100,813	$99,909	$86,049	$84,622
			Yr 1	Yr 2 - 5	Yr 5-10	Yr 11-15	Yr 15-20	19 Yr Avg.
Unit Op Costs:	Mining	$/t ore	$22.77	$10.83	$8.55	$8.40	$8.16	$8.82
(wtd averages)	Process		$18.96	$22.77	$21.63	$21.62	$18.00	$20.85
	G & A		$1.60	$0.72	$0.61	$0.62	$0.61	$0.63
	Sales, Dist'n		$0.17	$1.68	$1.51	$1.39	$0.81	$1.32
	Total		$43.50	$36.01	$32.31	$32.02	$27.58	$31.62
Before Tax Cash Flow		$000/a	($73,754)	$176,337	$109,717	$87,760	$41,213	$103,757
After Tax Cash Flow		$000/a	($73,754)	$156,299	$78,645	$62,986	$34,900	$83,208
Earnings		$000/a	($422,791)	$159,097	$79,900	$63,704	$16,233	$79,733
Cash Cost/lb Cu:								
	Gross	$/lb CuEq.	$1.19	$0.53	$0.59	$0.65	$0.91	$0.67
	Net of by-product credits		$1.19	$0.06	$0.05	$0.15	$0.48	$0.15

Note: Long Term Prices: Copper $1.25/lb, Cobalt $12.00/lb, Zinc Sulphate $950/t

After Tax 20 year IRR:		19.04%
NPV at	0%	$1,002,100
	6%	$445,883
	8%	$333,377

Table 58: Boléo Preliminary Economic Assessment – 5 Year Price Case Summary

Long Term Metal Prices: Co: \$2.20, Co: \$16.00, $ZnSO_4$: \$950

		Pre-Start-up	Yr 1	Yr 2 - 5	Yr 5 - 10	Yr 10-15	Yr 15 - 20	Yr 1 - 20
Capital (total)		(\$432,000)	(\$108,000)	Average	Average	Average	Average	Average
Ore grades:	Cu	%	2.10	2.19	1.54	1.33	0.83	1.47
	Co	%	0.08	0.12	0.09	0.09	0.07	0.09
	Zn	%	0.62	0.74	0.63	0.58	0.33	0.57
	Mn	%	3.14	2.31	2.50	2.88	1.86	2.43
Ore treated		t/a	1,170	2,633	3,120	3,120	3,120	2,925
Production:	Cu	t/a	22,408	52,682	43,934	37,901	23,560	38,006
	Co		0	2,251	2,298	2,074	1,610	2,048
	ZSM		0	33,607	37,075	34,151	19,415	30,928
	$MnCO_3$		0	0	0	0	0	0
Revenue:	Cu	\$000/a	\$109,664	\$257,826	\$215,011	\$185,488	\$115,304	\$185,999
	Co		\$0	\$79,400	\$81,067	\$73,149	\$56,798	\$68,634
	ZSM		\$0	\$31,926	\$35,221	\$32,443	\$18,444	\$27,912
	$MnCO_3$		\$0	\$0	\$0	\$0	\$0	\$0
	Total		\$109,664	\$369,152	\$331,298	\$291,081	\$190,546	\$282,545
Op. Costs:	Mining	\$000/a	\$26,635	\$28,520	\$26,687	\$26,213	\$25,470	\$26,628
	Process		\$22,186	\$59,949	\$67,492	\$67,446	\$56,153	\$60,872
	G&A		\$1,876	\$2,396	\$1,916	\$1,926	\$1,911	\$1,915
	Sales, Dist'n		\$199	\$4,491	\$4,810	\$4,404	\$2,564	\$3,853
	Total		\$50,896	\$95,357	\$100,905	\$99,988	\$86,098	\$84,665
			Yr 1	Yr 2 - 5	Yr 5-10	Yr 11-15	Yr 15 - 20	19 Yr Avg.
Unit Op Costs	Mining	\$/t ore	\$22.77	\$10.83	\$8.55	\$8.40	\$8.16	\$8.82
Weighted Averages	Process		\$18.96	\$22.77	\$21.63	\$21.62	\$18.00	\$20.85
	G&A		\$1.60	\$0.72	\$0.61	\$0.62	\$0.61	\$0.63
	Sales, Dist'n		\$0.17	\$1.71	\$1.54	\$1.41	\$0.82	\$1.34
	Total		\$43.50	\$36.04	\$32.34	\$32.05	\$27.60	\$31.65
Before Tax Cash Flow		\$000/a	(\$73,754)	\$265,525	\$221,901	\$185,344	\$104,704	\$194,369
After Tax Cash Flow		\$000/a	(\$73,754)	\$220,515	\$159,417	\$133,247	\$80,614	\$148,448
Earnings		\$000/a	(\$422,791)	\$223,313	\$160,672	\$133,965	\$61,947	\$144,974
Cash Cost/lb Cu:	Gross	\$/lb CuEq.	\$1.19	\$0.60	\$0.68	\$0.76	\$1.07	\$0.78
	Net of by-product credits		\$1.19	(\$0.11)	(\$0.16)	(\$0.07)	\$0.21	(\$0.06)

After Tax 20 year IRR:		30.22%
NPV at	0%	\$2,242,695
	6%	\$1,114,821
	8%	\$890,983

Table 59 : Boléo Preliminary Economic Assessment – Current Price Case Summary

Current Metal Prices Cu: \$2.50, Co: \$26.00, $ZnSO_4$: \$1,500

		Pre-Start-up	Yr 1	Yr 2 - 5	Yr 5 - 10	Yr 10 - 15	Yr 15 - 20	Yr 1 - 20
Capital (total)		(\$432,000)	(\$108,000)	Average	Average	Average	Average	Average
Ore grades:	Cu	%	2.10	2.19	1.54	1.33	0.83	1.47
	Co	%	0.08	0.12	0.09	0.09	0.07	0.09
	Zn	%	0.62	0.74	0.63	0.58	0.33	0.57
	Mn	%	3.14	2.31	2.50	2.88	1.86	2.43
Ore treated		t/a	1,170	2,633	3,120	3,120	3,120	2,925
Production:	Cu	t/a	22,408	52,682	43,934	37,901	23,560	38,006
	Co		0	2,251	2,298	2,074	1,610	2,048
	ZSM		0	33,607	37,075	34,151	19,415	30,928
	MnCO3		0	0	0	0	0	0
Revenue:	Cu	\$000/a	\$124,483	\$292,667	\$244,066	\$210,554	\$130,885	\$211,134
	Co		\$0	\$129,025	\$131,733	\$118,868	\$92,297	\$111,530
	ZSM		\$0	\$50,410	\$55,612	\$51,226	\$29,122	\$44,072
	MnCO3		\$0	\$0	\$0	\$0	\$0	\$0
	Total		\$124,483	\$472,102	\$431,412	\$380,649	\$252,304	\$366,736
Op. Costs:	Mining	\$000/a	\$26,635	\$28,520	\$26,687	\$26,213	\$25,470	\$26,628
	Process		\$22,186	\$59,949	\$67,492	\$67,446	\$56,153	\$60,872
	G & A		\$1,876	\$2,396	\$1,916	\$1,926	\$1,911	\$1,915
	Sales, Dist'n		\$214	\$4,526	\$4,839	\$4,429	\$2,580	\$3,878
	Total		\$50,911	\$95,391	\$100,934	\$100,013	\$86,114	\$84,679
			Yr 1	Yr 2 - 5	Yr 5-10	Yr 11-15	Yr 15-20	19 Yr Avg.
Unit Op Costs:	Mining	\$/t ore	\$22.77	\$10.83	\$8.55	\$8.40	\$8.16	\$8.82
(weighted averages)	Process		\$18.96	\$22.77	\$21.63	\$21.62	\$18.00	\$20.85
	G & A		\$1.60	\$0.72	\$0.61	\$0.62	\$0.61	\$0.63
	Sales, Dist'n		\$0.18	\$1.72	\$1.55	\$1.42	\$0.83	\$1.35
	Total		\$43.51	\$36.05	\$32.35	\$32.06	\$27.60	\$31.66
Before Tax Cash Flow		\$000/a	(\$58,950)	\$368,440	\$321,985	\$274,886	\$166,447	\$282,940
After Tax Cash Flow		\$000/a	(\$58,950)	\$293,577	\$231,478	\$197,717	\$125,077	\$211,962
Earnings		\$000/a	(\$407,986)	\$296,375	\$232,733	\$198,435	\$106,410	\$208,488
Cash Cost/lb Cu:	Gross	\$/lb CuEq.	\$1.19	\$0.53	\$0.59	\$0.66	\$0.91	\$0.68
	Net of by-product credits		\$1.19	(\$0.70)	(\$0.89)	(\$0.84)	(\$0.68)	(\$0.79)

After Tax 20 year IRR:		40.63%
NPV at	0%	\$3,454,721
	6%	\$1,786,587
	8%	\$1,457,416

Table 60: Boléo PEA – Opportunity: Base Case with Manganese Production

Long Term Metal Prices: Cu: $1.25, Co: $12.00, $MnCO_3$: $400, $ZnSO_4$: $950

			Pre-Start-up	Yr 1	Yr 2 - 5	Yr 5 - 10	Yr 10 -15	Yr 15 - 20	Yr 1 - 20
Capital (total):			($448,000)	($112,000)	Average	Average	Average	Average	Average
Ore grades:	Cu	%		2.10	2.19	1.54	1.33	0.83	1.47
	Co	%		0.08	0.12	0.09	0.09	0.07	0.09
	Zn	%		0.62	0.74	0.63	0.58	0.33	0.57
	Mn	%		3.14	2.31	2.50	2.88	1.86	2.43
Ore treated		t/y		1,170	2,633	3,120	3,120	3,120	2,925
Production:	Cu	t/a		22,408	52,682	43,934	37,901	23,560	38,006
	Co			0	2,251	2,298	2,074	1,610	2,048
	ZSM			0	33,607	37,075	34,151	19,415	30,928
	$MnCO_3$			0	102,351	134,533	155,073	100,013	117,875
Revenue:	Cu	$000/a		$109,664	$188,418	$123,002	$106,113	$65,962	$116,936
	Co			$0	$59,550	$60,800	$54,862	$42,599	$51,475
	ZSM			$0	$31,926	$35,221	$32,443	$18,444	$27,912
	$MnCO_3$			$0	$40,940	$53,813	$62,029	$40,005	$47,150
	Total			$109,664	$320,835	$272,836	$255,447	$167,010	$243,473
Op. Costs:	Mining	$000/a		$26,635	$28,520	$26,687	$26,213	$25,470	$26,628
	Process			$22,186	$78,217	$91,504	$95,123	$74,003	$81,910
	G&A			$1,876	$2,396	$1,916	$1,926	$1,911	$1,915
	Sales, Dist'n			$199	$9,539	$11,445	$12,078	$7,516	$9,677
	Total			$50,896	$118,672	$131,551	$135,340	$108,900	$103,050
				Yr 1	Yr 2 - 5	Yr 5 - 10	Yr 11-15	Yr 15-20	19 Yr Avg.
Unit Op Costs:	Mining	$/t ore		$22.77	$10.83	$8.55	$8.40	$8.16	$8.82
(wtd averages)	Process			$18.96	$29.71	$29.33	$30.49	$23.72	$28.19
	G & A			$1.60	$0.72	$0.61	$0.62	$0.61	$0.63
	Sales, Dist'n			$0.17	$3.62	$3.67	$3.87	$2.41	$3.37
	Total			$43.50	$44.89	$42.16	$43.38	$34.90	$41.02
Before Tax Cash Flow		$000/a		($77,914)	$193,348	$132,638	$114,123	$58,880	$124,747
After tax Cash Flow		$000/a		($77,914)	$169,643	$95,129	$81,926	$47,943	$98,660
Earnings		$000/a		($440,330)	$172,914	$96,451	$82,792	$28,124	$95,070
Cash Cost/lb Cu:	Gross	$/lb CuEq.		$1.19	$0.65	$0.77	$0.88	$1.14	$0.86
	Net of by-product credits			$1.19	($0.09)	($0.19)	($0.17)	$0.15	($0.10)

After Tax 20 year IRR:		20.93%
NPV at	0%	$1,277,645
	6%	$580,218
	8%	$441,228

Table 61: 20 Year Detailed Cash Flow – Base Case

		2007	2008	2009	2010	2011	2012	2013	2014
Throughput:	dmt/a			1,170	2,470	2,600	2,600	2,860	3,120
Grades:	%Cu			2.10	2.10	2.49	1.96	2.22	1.62
	%Co			0.08	0.15	0.15	0.12	0.06	0.06
	%Zn			0.62	0.81	1.00	0.71	0.42	0.84
Recoveries:	Cu			91.2%	91.2%	91.2%	91.2%	91.2%	91.2%
	Co			0.0%	60.0%	78.2%	78.2%	78.2%	78.2%
	Zn			0.0%	50.0%	65.6%	65.6%	65.6%	65.6%
Production:	t/a Cu			22,408	47,305	59,043	46,476	57,905	46,096
	t/a Co			0	2,238	2,989	2,481	1,297	1,440
	t/a ZSM			0	28,581	48,731	34,599	22,514	49,121
Prices:	Cu			$2.20	$1.95	$1.75	$1.50	$1.25	$1.25
	Co			$12.00	$12.00	$12.00	$12.00	$12.00	$12.00
	ZSM			$950	$950	$950	$950	$950	$950
Revenue:	Cu			$109,664	$205,441	$230,383	$155,732	$162,117	$129,056
($000's)	Co			$0	$59,199	$79,066	$65,619	$34,316	$38,081
	Zn			$0	$27,152	$46,295	$32,869	$21,388	$46,665
	Total			$109,664	$291,793	$355,744	$254,221	$217,821	$213,802
Total Op Cost $000's				$58,896	$106,402	$104,388	$95,192	$88,193	$106,736
Initial Capital		$54,000	$378,000	$108,000					
Sustaining Capital				$0	$1,350	$2,430	$3,980	$2,430	$18,915
Working Capital at YE				$8,522	$11,591	$12,471	$11,853	$11,388	$12,688
Working Capital Change				$8,522	$3,069	$880	-$619	-$465	$1,300
Income Taxes				$0	$0	$887	$43,559	$35,704	$25,371
Net Cash Flow		($54,000)	($378,000)	($73,754)	$173,971	$247,159	$112,109	$91,958	$61,480
		2015	2016	2017	2018	2019	2020	2021	2022
Throughput:	dmt/a	3,120	3,120	3,120	3,120	3,120	3,120	3,120	3,120
Grades:	%Cu	1.53	1.64	1.60	1.33	1.69	1.33	1.29	1.31
	%Co	0.06	0.13	0.12	0.09	0.08	0.09	0.08	0.09
	%Zn	0.55	0.63	0.56	0.59	0.38	0.47	0.51	0.79
Recoveries:	Cu	91.2%	91.2%	91.2%	91.2%	91.2%	91.2%	91.2%	91.2%
	Co	78.2%	78.2%	78.2%	78.2%	78.2%	78.2%	78.2%	78.2%
	Zn	65.6%	65.6%	65.6%	65.6%	65.6%	65.6%	65.6%	65.6%
Production:	t/a Cu	43,535	46,665	45,527	37,844	48,088	37,844	36,706	37,275
	t/a Co	1,537	3,196	3,025	2,293	1,952	2,171	1,830	2,245
	t/a ZSM	32,163	36,841	32,748	34,502	22,222	27,485	29,824	46,197
Prices:	Cu	$1.25	$1.25	$1.25	$1.25	$1.25	$1.25	$1.25	$1.25
	Co	$12.00	$12.00	$12.00	$12.00	$12.00	$12.00	$12.00	$12.00
	ZSM	$950	$950	$950	$950	$950	$950	$950	$950
Revenue:	Cu	$121,886	$130,649	$127,463	$105,953	$134,633	$105,953	$102,767	$104,360
($000's)	Co	$40,662	$84,552	$80,034	$60,671	$51,635	$57,444	$48,408	$59,380
	Zn	$30,555	$34,999	$31,110	$32,777	$21,110	$26,110	$28,332	$43,888
	Total	$193,103	$250,200	$238,607	$199,401	$207,378	$189,507	$179,507	$207,628
Total Op Cost $000's		$98,638	$98,449	$100,324	$99,919	$95,322	$101,245	$96,418	$107,545
Initial Capital									
Sustaining Capital		$2,700	$14,580	$2,700	$2,700	$6,382	$14,432	$2,700	$2,700
Working Capital at YE		$12,173	$12,098	$12,294	$12,254	$11,984	$12,432	$12,011	$12,780
Working Capital Change		-$515	-$76	$196	-$40	-$270	$448	-$421	$769
Income Taxes		$25,793	$38,939	$38,061	$27,197	$29,821	$21,198	$22,607	$27,366
Net Cash Flow		$66,487	$98,308	$97,325	$69,625	$76,124	$52,185	$58,203	$69,248

			2023	2024	2025	2026	2027	2028
Throughput:	dmt/a		3,120	3,120	3,120	3,120	3,120	3,120
Grades:	%Cu		1.04	1.23	0.94	0.74	0.71	0.52
	%Co		0.09	0.10	0.06	0.06	0.07	0.05
	%Zn		0.77	0.62	0.38	0.26	0.23	0.17
Recoveries:	Cu		91.2%	91.2%	91.2%	91.2%	91.2%	91.2%
	Co		78.2%	78.2%	78.2%	78.2%	78.2%	78.2%
	Zn		65.6%	65.6%	65.6%	65.6%	65.6%	65.6%
Production:	t/a Cu		29,593	34,999	26,747	21,056	20,203	14,796
	t/a Co		2,171	2,342	1,561	1,440	1,610	1,098
	t/a ZSM		45,028	36,256	22,222	15,204	13,450	9,941
Prices:	Cu		$1.25	$1.25	$1.25	$1.25	$1.25	$1.25
	Co		$12.00	$12.00	$12.00	$12.00	$12.00	$12.00
	ZSM		$950	$950	$950	$950	$950	$950
Revenue:	Cu		$82,851	$97,987	$74,884	$58,952	$56,562	$41,425
($000's)	Co		$57,444	$61,962	$41,308	$38,081	$42,599	$29,045
	Zn		$42,776	$34,443	$21,110	$14,444	$12,777	$9,444
	Total		$183,071	$194,392	$137,303	$111,476	$111,938	$79,914
Total Op Cost $000's			$99,016	$91,756	$89,385	$85,937	$85,970	$77,196
Initial Capital								
Sustaining Capital			$2,700	$2,700	$2,700	$2,700	$2,700	$0
Working Capital at YE			$12,085	$11,559	$11,505	$11,289	$11,307	$0
Working Capital Change			-$695	-$526	-$54	-$216	$19	-$11,307
Income Taxes			$22,878	$28,080	$12,759	$6,493	$6,613	-$22,383
Net Cash Flow			$59,173	$72,381	$32,512	$16,562	$16,636	$36,408

Table 62: 20 Year Detailed Cash Flow – 5 Year Prices Case – (3 year trailing + 2 year leading)

		2007	2008	2009	2010	2011	2012	2013	2014
Throughput:	t/a			1,170	2,470	2,600	2,600	2,860	3,120
Grades:	%Cu			2.10	2.10	2.49	1.96	2.22	1.62
	%Co			0.08	0.15	0.15	0.12	0.06	0.06
	%Zn			0.62	0.81	1.00	0.71	0.42	0.84
Recoveries:	Cu			91.2%	91.2%	91.2%	91.2%	91.2%	91.2%
	Co			0.0%	60.0%	78.2%	78.2%	78.2%	78.2%
	Zn			0.0%	50.0%	65.6%	65.6%	65.6%	65.6%
Production:	t/a Cu			22,408	47,305	59,043	46,476	57,905	46,096
	t/a Co			0	2,238	2,989	2,481	1,297	1,440
	t/a ZSM			0	28,581	48,731	34,599	22,514	49,121
Prices:	Cu			$2.20	$2.20	$2.20	$2.20	$2.20	$2.20
	Co			$16.00	$16.00	$16.00	$16.00	$16.00	$16.00
	ZSM			$950	$950	$950	$950	$950	$950
Revenue:	Cu			$109,664	$231,512	$288,955	$227,451	$283,385	$225,594
($000's)	Co			$0	$78,932	$105,421	$87,492	$45,754	$50,774
	Zn			$0	$27,152	$46,295	$32,869	$21,388	$46,665
	Total			$109,664	$337,597	$440,671	$347,812	$350,527	$323,033
Total Op Cost $000's				$58,896	$106,428	$104,447	$95,263	$88,314	$106,833
Initial Capital		$54,000	$378,000	$108,000					
Sustaining Capital				$0	$1,350	$2,430	$3,980	$2,430	$18,915
Working Capital at YE				$8,522	$11,591	$12,471	$11,853	$11,388	$12,688
Working Capital Change				$8,522	$3,069	$880	-$619	-$465	$1,300
Income Taxes				$0	$0	$37,468	$69,744	$72,828	$55,929
Net Cash Flow		($54,000)	($378,000)	($73,754)	$219,749	$295,446	$179,443	$187,420	$140,057

		2015	2016	2017	2018	2019	2020	2021	2022
Throughput:	dmt/a	3,120	3,120	3,120	3,120	3,120	3,120	3,120	3,120
Grades:	%Cu	1.53	1.64	1.60	1.33	1.69	1.33	1.29	1.31
	%Co	0.06	0.13	0.12	0.09	0.08	0.09	0.08	0.09
	%Zn	0.55	0.63	0.56	0.59	0.38	0.47	0.51	0.79
Recoveries:	Cu	91.2%	91.2%	91.2%	91.2%	91.2%	91.2%	91.2%	91.2%
	Co	78.2%	78.2%	78.2%	78.2%	78.2%	78.2%	78.2%	78.2%
	Zn	65.6%	65.6%	65.6%	65.6%	65.6%	65.6%	65.6%	65.6%
Production:	t/a Cu	43,535	46,665	45,527	37,844	48,088	37,844	36,706	37,275
	t/a Co	1,537	3,196	3,025	2,293	1,952	2,171	1,830	2,245
	t/a ZSM	32,163	36,841	32,748	34,502	22,222	27,485	29,824	46,197
Prices:	Cu	$2.20	$2.20	$2.20	$2.20	$2.20	$2.20	$2.20	$2.20
	Co	$16.00	$16.00	$16.00	$16.00	$16.00	$16.00	$16.00	$16.00
	ZSM	$950	$950	$950	$950	$950	$950	$950	$950
Revenue:	Cu	$213,061	$228,379	$222,809	$185,210	$235,342	$185,210	$179,640	$182,425
($000's)	Co	$54,217	$112,736	$106,712	$80,895	$68,846	$76,592	$64,544	$79,173
	Zn	$30,555	$34,999	$31,110	$32,777	$21,110	$26,110	$28,332	$43,888
	Total	$297,832	$376,114	$360,631	$298,881	$325,299	$287,912	$272,516	$305,486
Total Op Cost $000's		$98,729	$98,547	$100,420	$99,998	$95,422	$101,325	$96,494	$107,623
Initial Capital									
Sustaining Capital		$2,700	$14,580	$2,700	$2,700	$6,382	$14,432	$2,700	$2,700
Working Capital at YE		$12,173	$12,098	$12,294	$12,254	$11,984	$12,432	$12,011	$12,780
Working Capital Change		-$515	-$76	$196	-$40	-$270	$448	-$421	$769
Income Taxes		$55,091	$74,167	$72,201	$55,030	$62,811	$48,729	$48,628	$54,744
Net Cash Flow		$141,826	$188,896	$185,114	$141,194	$160,954	$122,979	$125,114	$139,650

		2023	2024	2025	2026	2027	2028		
Throughput:	dmt/a	3,120	3,120	3,120	3,120	3,120	3,120		
Grades:	%Cu	1.04	1.23	0.94	0.74	0.71	0.52		
	%Co	0.09	0.10	0.06	0.06	0.07	0.05		
	%Zn	0.77	0.62	0.38	0.26	0.23	0.17		
Recoveries:	Cu	91.2%	91.2%	91.2%	91.2%	91.2%	91.2%		
	Co	78.2%	78.2%	78.2%	78.2%	78.2%	78.2%		
	Zn	65.6%	65.6%	65.6%	65.6%	65.6%	65.6%		
Production:	t/a Cu	29,593	34,999	26,747	21,056	20,203	14,796		
	t/a Co	2,171	2,342	1,561	1,440	1,610	1,098		
	t/a ZSM	45,028	36,256	22,222	15,204	13,450	9,941		
Prices:	Cu	$2.20	$2.20	$2.20	$2.20	$2.20	$2.20		
	Co	$16.00	$16.00	$16.00	$16.00	$16.00	$16.00		
	ZSM	$950	$950	$950	$950	$950	$950		
Revenue:	Cu	$144,826	$171,284	$130,900	$103,049	$98,871	$72,413		
($000's)	Co	$76,592	$82,616	$55,077	$50,774	$56,798	$38,726		
	Zn	$42,776	$34,443	$21,110	$14,444	$12,777	$9,444		
	Total	$264,194	$288,343	$207,088	$168,267	$168,447	$120,583		
Total Op Cost $000's	$99,078	$91,830	$89,441	$85,981	$86,012	$77,227			
Initial Capital									
Sustaining Capital	$2,700	$2,700	$2,700	$2,700	$2,700	$0			
Working Capital at YE	$12,085	$11,559	$11,505	$11,289	$11,307	$0			
Working Capital Change	-$695	-$526	-$54	-$216	$19	-$11,307			
Income Taxes	$45,575	$54,366	$32,283	$22,382	$22,424	-$11,004			
Net Cash Flow	$117,537	$139,974	$82,717	$57,420	$57,292	$65,667			

Table 63: 20 Year Detailed Cash Flow – Current Prices Case

		2007	2008	2009	2010	2011	2012	2013	2014
Throughput:	dmt/a			1,170	2,470	2,600	2,600	2,860	3,120
Grades:	%Cu			2.10	2.10	2.49	1.96	2.22	1.62
	%Co			0.08	0.15	0.15	0.12	0.06	0.06
	%Zn			0.62	0.81	1.00	0.71	0.42	0.84
Recoveries:	Cu			91.2%	91.2%	91.2%	91.2%	91.2%	91.2%
	Co			0.0%	60.0%	78.2%	78.2%	78.2%	78.2%
	Zn			0.0%	50.0%	65.6%	65.6%	65.6%	65.6%
Production:	t/a Cu			22,408	47,305	59,043	46,476	57,905	46,096
	t/a Co			0	2,238	2,989	2,481	1,297	1,440
	t/a ZSM			0	28,581	48,731	34,599	22,514	49,121
Prices:	Cu			$2.50	$2.50	$2.50	$2.50	$2.50	$2.50
	Co			$26.00	$26.00	$26.00	$26.00	$26.00	$26.00
	ZSM			$1,500	$1,500	$1,500	$1,500	$1,500	$1,500
Revenue:	Cu			$124,483	$262,798	$328,003	$258,187	$321,680	$256,080
($000's)	Co			$0	$128,265	$171,309	$142,175	$74,351	$82,508
	Zn			$0	$42,872	$73,097	$51,899	$33,771	$73,682
	Total			$124,483	$433,935	$572,410	$452,261	$429,802	$412,270
Total Op Cost $000's				$58,911	$106,460	$104,486	$95,294	$88,353	$106,863
Initial Capital		$54,000	$378,000	$108,000					
Sustaining Capital				$0	$1,350	$2,430	$3,980	$2,430	$18,915
Working Capital at YE				$8,522	$11,591	$12,471	$11,853	$11,388	$12,688
Working Capital Change				$8,522	$3,069	$880	-$619	-$465	$1,300
Income Taxes				$0	$0	$105,455	$98,981	$95,014	$80,906
Net Cash Flow		($54,000)	($378,000)	($58,950)	$316,056	$359,159	$254,625	$244,470	$204,285
		2015	2016	2017	2018	2019	2020	2021	2022
Throughput:	dmt/a	3,120	3,120	3,120	3,120	3,120	3,120	3,120	3,120
Grades:	%Cu	1.53	1.64	1.60	1.33	1.69	1.33	1.29	1.31
	%Co	0.06	0.13	0.12	0.09	0.08	0.09	0.08	0.09
	%Zn	0.55	0.63	0.56	0.59	0.38	0.47	0.51	0.79
Recoveries:	Cu	91.2%	91.2%	91.2%	91.2%	91.2%	91.2%	91.2%	91.2%
	Co	78.2%	78.2%	78.2%	78.2%	78.2%	78.2%	78.2%	78.2%
	Zn	65.6%	65.6%	65.6%	65.6%	65.6%	65.6%	65.6%	65.6%
Production:	t/a Cu	43,535	46,665	45,527	37,844	48,088	37,844	36,706	37,275
	t/a Co	1,537	3,196	3,025	2,293	1,952	2,171	1,830	2,245
	t/a ZSM	32,163	36,841	32,748	34,502	22,222	27,485	29,824	46,197
Prices:	Cu	$2.50	$2.50	$2.50	$2.50	$2.50	$2.50	$2.50	$2.50
	Co	$26.00	$26.00	$26.00	$26.00	$26.00	$26.00	$26.00	$26.00
	ZSM	$1,500	$1,500	$1,500	$1,500	$1,500	$1,500	$1,500	$1,500
Revenue:	Cu	$241,853	$259,241	$252,918	$210,238	$267,145	$210,238	$203,915	$207,077
($000's)	Co	$88,102	$183,196	$173,407	$131,454	$111,875	$124,461	$104,883	$128,657
	Zn	$48,244	$55,261	$49,121	$51,753	$33,332	$41,227	$44,735	$69,296
	Total	$378,199	$497,699	$475,446	$393,445	$412,353	$375,926	$353,534	$405,030
Total Op Cost $000's		$98,758	$98,578	$100,450	$100,023	$95,454	$101,350	$96,519	$107,647
Initial Capital									
Sustaining Capital		$2,700	$14,580	$2,700	$2,700	$6,382	$14,432	$2,700	$2,700
Working Capital at YE		$12,173	$12,098	$12,294	$12,254	$11,984	$12,432	$12,011	$12,780
Working Capital Change		-$515	-$76	$196	-$40	-$270	$448	-$421	$769
Income Taxes		$77,586	$108,202	$104,341	$81,500	$87,177	$73,366	$71,307	$82,609
Net Cash Flow		$199,670	$276,414	$267,759	$209,261	$223,610	$186,331	$183,430	$211,304
			2023	2024	2025	2026	2027	2028	
Throughput:		t/a	3,120	3,120	3,120	3,120	3,120	3,120	

19 CONCLUSIONS

19.1 GEOLOGY AND MINERAL RESOURCE MODELLING

A Measured and Indicated Resource has been defined, amounting to:

- 233 Mt at 1.83% CuEq. at a cutoff grade of 0.5% CuEq.
- 182 Mt at 2.13% CuEq. at a cutoff grade of 1.0% CuEq.

In addition an Inferred Resource has been defined, amounting to:

- 203 Mt at 1.32% CuEq. at a cutoff grade of 0.5% CuEq.
- 114 Mt at 1.76% CuEq. at a cutoff grade of 1.0% CuEq.

19.2 METALLURGY AND PROCESS DESIGN

A 'Fully Integrated' Pilot Plant Campaign was conducted at the SGS Lakefield facility from June 23 rd to July 20th, 2006 treating a bulk sample of Manto 3 oxide ore from the Boléo Property.

Highlights of the campaign include the following:

- Pilot plant commissioning commenced on 5th June 2006. The integrated pilot plant campaign ran until 24th June
- A total of nearly 5 mt of ore were treated through the pilot plant
- The zinc solvent extraction and cobalt solvent extraction and electrowinning circuits were run during the period of 4th to 15th July in a separate campaign using the solutions collected in the earlier phase.
- The oxidation and reductive leaching circuit once more gave excellent extractions of copper, cobalt and zinc. Copper extraction exceeded 90% during pilot operation. Cobalt extraction varied from 80% to as high as 90%. Zinc extraction was generally above 70%. These numbers are confirmation of the earlier proof of concept flowsheet test performance.
- The iron removal circuit consistently produced very low concentrations of key impurities in solution with negligible losses of cobalt and zinc.
- The CSIRO DSX ® circuit for cobalt and zinc recovery performed extremely well. In the Lakefield pilot plant, cobalt and zinc were recovered with high overall extraction efficiency (+95%) to produce a concentrated zinc sulphate solution (for production of zinc sulphate monohydrate crystals for sale) and a concentrated cobalt solution (for production of cobalt metal cathode).
- The successful completion of this pilot program is an important milestone in advancing the design of the hydrometallurgical facility and moving the Boléo project forward. Once again Boléo ores have been treated in a continuous pilot plant program to leach, separate and recover pay metals in final commercial form. Importantly the use of soda ash for pH control

and the use of Boléo carbonate for neutralization duties was successfully demonstrated, as was the production of manganese carbonate and zinc sulphate monohydrate.

- Design criteria have been confirmed and data for the purposes of ultimately formulating process guarantees for the Boléo flowsheet have been gathered.

19.3 MINING

- The potential for surface and underground mining were assessed for the H&S resource model. A series of underground mining operations, supported by small open-cut surface mines, delivering targeted high-grade copper-cobalt-zinc manganese ore (0.5% to 2.5% Cu) to the process facility was selected as the best alternative for the scale of operation envisaged by Baja Mining Corp.
- The mine modelling process has not been optimized in this preliminary economic assessment. Additional modelling and cost estimating work is required to fully develop the Boléo property. It is expected this effort will produce a more efficient mine plan in the future.
- A suitable limestone source was located on the property and will be mined by surface methods to provide calcium carbonate for plant process needs.
- A tailings dam and associated tailings disposal facility has been designed with the capacity to support the life-of-mine projections.

19.4 ENVIRONMENTAL

An environmental plan was submitted to Mexican Federal authorities and the basic Environmental Impact Manifest (EIM) permit was approved and issued to the Company in December, 2006. This allows the company to begin submitting additional specific permitting requests for construction and operational activities in accordance with the EIM provisions.

19.5 PRELIMINARY ECONOMIC ASSESSMENT

- Financial modelling based on the current, un-optimized preliminary mine schedule indicates that the project is attractive using conservative (base-case) metal prices while recovering copper, cobalt and zinc.
- Modelling of these base-case metal prices over a projected 20 year mine life shows that the project could generate a cumulative after tax cash flow of US$1,434,100 million with a discounted present value of US$445.9 million at a 6% discount rate, or US$333.4 million using a 8% discount rate.
- The cash cost of production of copper, net of by-product prices, is close to zero using conservative pricing and becomes significantly less than zero if by-product prices approach current, or recent average, price levels.
- The addition of manganese production could add extra value to the project.

20 RECOMMENDATIONS

20.1 GEOLOGY AND MINERAL RESOURCE MODELLING

H&S made the following recommendations in the 2005 PEA, all of which have been adopted. For ease of reference these are re-quoted below.

"…The drilling data is currently kept in a series of Excel spread sheets. These spread sheets need to be replaced by a formal database. H&S suggest the use of Microsoft Access. The advantages a database has over spread sheets include:

- improved data security
- automated data entry
- automated data checking
- data easily exported in correct format use in other software packages
- data is easily interrogated
- activity reporting
- intercept reporting
- automated monitoring of QC data
- checking of QC data during data entry and automated alerts for 'out of spec' values.

Drill hole logging has not been carried out in a way that is amenable for use in any mining or geological software processing package. Descriptions of geological intervals are completed as full text rather than coded to allow for computer processing. H&S recommend that new logging sheets be drafted to facilitate recording the data in a way that allows all the pertinent geological characteristics to be recorded, whilst in a format that can be uploaded directly into a database, without manual data entry. This typically results in a logging form with dedicated data fields that allows the geologist to record quantitative as well as qualitative observations. This can add a lot of value to the data and can be extremely useful during resource modelling. It is also important that future assaying and logging is completed over the same down hole intervals. Matching of grade and geological items in this way facilitates statistical analysis of grade according the relevant geological criteria. It would also enable accurate modelling of ore type characteristics.

Sampling practices were not observed during the present study. It is recommended that these be reviewed by H&S at an early opportunity during the next drilling program…"

There are no further recommendations.

20.2 METALLURGY AND PROCESS DESIGN

Bateman made a small number of recommendations in the 2005 PEA, most of which related to the upcoming 2006 pilot campaign, all of which were implemented at that time. For ease of reference these are re-quoted below.

- …"That every effort be made to ensure that the sample(s) utililized in the proposed confirmatory pilot plant campaign represent the diversity of ore type anticipated in the first 5 years of the mine life as per the preliminary mining schedule
- That Boléo HAC be employed in all neutralization operations during the pilot campaign
- That the pilot campaign be fully integrated – including zinc salt production, cobalt metal production and all recycle streams – to verify the overall recovery of copper, cobalt & zinc from the circuit
- That the pilot campaign be undertaken at a sufficiently large scale that regular sampling does not unduly influence the overall process performance
- That appropriate vendors be contracted to conduct bench scale testwork during the campaign in the areas of agitation and solid-liquid separation with the ultimate objective of obtaining process guarantees from the successful vendors for these process functions..."

Further recommendations include:

- Getting better definition of the acid consumption variability of the ore for the first five years of operation via testwork. This acid consumption distribution will serve to provide confidence in the ability of the processing plant to operate at design values for a given mining plan and blending strategy.
- Getting improved definition of the proposed process flowsheets ability to successfully process small quantities of sulphide ore as part of the ore feed mix.

Work in both these important areas has commenced in late January 2007 and results will be incorporated in the plant design as appropriate.

20.3 MINING

AMDAD and H&S made the following recommendations in the 2005 PEA, all of which have been adopted. For ease of reference these are re-quoted below.

- …"The trial mining operation should be kept as close as possible to anticipated operating conditions so that it can be reliably calibrated against the design and production parameters to be used in the feasibility study.
- An infill drilling program should be conducted to bring the first two to five years of production up to at least Indicated Resource status. A range of drill hole spacings should be planned in virgin and previously mined areas to assess the continuity of grade over the anticipated mining heights under both conditions. This may allow a review of the overall deposit to re-assess the resource categories.

- A “mining” resource model should be constructed using economic mining height thickness calculated at each drill hole and including roof and floor data for this height. This could be used to prepare hazard plans for roof and floor conditions covering information such as the thickness of clay breccia below the overlying sandstone or the presence of false sandstone floors above the conglomerate. It may also assist in defining the extent of voids or stope fill.
- Preliminary development layouts should be designed to assess the mining recovery and to define/assist definition of ore haulage and ventilation options.
- More detailed analysis of mining methods can be deferred until the data from the mining trial is available…"

At this stage of the project development there are no further recommendations with respect to Mining.

20.4 Updated Preliminary Economic Assessment

There are no recommendations pertinent to the economic assessment:

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Agapito Associates, Inc. (2007), "Geotechnical Evaluation for Underground Mine Design," draft report to Baja Mining Corp, February 2006.

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CONSENT OF AUTHOR

TO: The Toronto Stock Exchange
The British Columbia Securities Commission
The Alberta Securities Commission
The Ontario Securities Commission
The United States Securities and Exchange Commission (SEC)

I, Donald J. E. Hunter, do hereby consent to the filing, with the regulatory authorities referred to above, of the report titled, *"An Updated Preliminary Assessment of the El Boléo Copper Cobalt Project"*, dated the 31st January, 2007 (the "Preliminary Assessment") and to extracts from or a summary of the Preliminary Assessment in the written disclosure in the news release and material change report of Baja Mining Corp. being filed.

I also certify that I have read the written disclosure being filed and such written disclosure fairly and accurately represents the information in the Preliminary Assessment that supports the disclosure.

Dated this 16th day of March, 2007

/s/ Donald J. E. Hunter____________
Donald J. E. Hunter

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CONSENT OF AUTHOR

TO: The British Columbia Securities Commission
The Alberta Securities Commission
The Ontario Securities Commission
The Toronto Stock Exchange (TSX)
United States Securities & Exchange Commission (SEC)

I, Eric Norton, P.Eng., do hereby consent to the filing, with the regulatory authorities referred to above, of the technical report titled, "*Baja Mining Corp. – An Updated Preliminary Assessment of the El Boleo Copper Cobalt Project***",** of the El Boleo Project located in Santa Rosalia, Baja California Sur, Mexico and dated January 31, 2007 (the "Technical Report") and filed in March 2007. In addition, I do hereby consent that any extracts from, or a summary of, the Technical Report may be filed with any stock exchange, any other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the publicly accessible company files on their websites.

Dated this 10th day of March, 2007

/s/ Eric Norton___________________
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CONSENT OF AUTHOR

TO: The British Columbia Securities Commission
The Alberta Securities Commission
The Ontario Securities Commission
The Toronto Stock Exchange (TSX)
United States Securities & Exchange Commission (SEC)

I, Scott Britton, P.Eng., do hereby consent to the filing, with the regulatory authorities referred to above, of the technical report titled, "*Baja Mining Corp. – An Updated Preliminary Assessment of the El Boleo Copper Cobalt Project***",** of the El Boleo Project located in Santa Rosalia, Baja California Sur, Mexico and dated January 31, 2007 (the "Technical Report") and filed in March 2007. In addition, I do hereby consent that any extracts from, or a summary of, the Technical Report may be filed with any stock exchange, any other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the publicly accessible company files on their websites.

Dated this 10th day of March, 2007

/s/ Scott Britton___________
Scott Britton, P.Eng.

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CONSENT OF AUTHOR

TO: The British Columbia Securities Commission
The Alberta Securities Commission
The Ontario Securities Commission
The Toronto Stock Exchange (TSX)
United States Securities & Exchange Commission (SEC)

I, Grant Bryan Bosworth, P.Eng., do hereby consent to the filing, with the regulatory authorities referred to above, of the technical report titled, "*Baja Mining Corp. – An Updated Preliminary Assessment of the El Boleo Copper Cobalt Project*"**,** of the El Boleo Project located in Santa Rosalia, Baja California Sur, Mexico and dated January 31, 2007 (the "Technical Report") and filed in March 2007. In addition, I do hereby consent that any extracts from, or a summary of, the Technical Report may be filed with any stock exchange, any other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the publicly accessible company files on their websites.

Dated this 10th day of March, 2007

/s/ Grant B. Bosworth_______
Grant Bryan Bosworth, P.Eng.

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CONSENT OF AUTHOR

TO: The British Columbia Securities Commission
The Alberta Securities Commission
The Ontario Securities Commission
The Toronto Stock Exchange (TSX)
United States Securities & Exchange Commission (SEC)

I, William Yeo, PhD, MAusIMM, do hereby consent to the filing, with the regulatory authorities referred to above, of the technical report titled, "*Baja Mining Corp. – An Updated Preliminary Assessment of the El Boleo Copper Cobalt Project***",** of the El Boleo Project located in Santa Rosalia, Baja California Sur, Mexico and dated January 31, 2007 (the "Technical Report") and filed in March 2007. In addition, I do hereby consent that any extracts from, or a summary of, the Technical Report may be filed with any stock exchange, any other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the publicly accessible company files on their websites.

Dated this 10th day of March, 2007

/s/ William J. A. Yeo______________
William J. A. Yeo

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CONSENT OF AUTHOR

**TO: The British Columbia Securities Commission
The Alberta Securities Commission
The Ontario Securities Commission
The Toronto Stock Exchange (TSX)
United States Securities & Exchange Commission (SEC)**

I, Michael Richard Holmes, Boleo Study Manager, do hereby consent to the filing, with the regulatory authorities referred to above, of the technical report titled, "*Baja Mining Corp. – An Updated Preliminary Assessment of the El Boleo Copper Cobalt Project***",** of the El Boleo Project located in Santa Rosalia, Baja California Sur, Mexico and dated January 31, 2007 (the "Technical Report") and filed in March 2007. In addition, I do hereby consent that any extracts from, or a summary of, the Technical Report may be filed with any stock exchange, any other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the publicly accessible company files on their websites.

Dated this 10th day of March, 2007

/s/ Michael R. Holmes_______________
Michael Richard Holmes

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CONSENT OF AUTHOR

TO: The British Columbia Securities Commission
The Alberta Securities Commission
The Ontario Securities Commission
The Toronto Stock Exchange (TSX)
United States Securities & Exchange Commission (SEC)

I, Timothy A. Ross, PE, SME Registered Member, do hereby consent to the filing, with the regulatory authorities referred to above, of the technical report titled, "*Baja Mining Corp. – An Updated Preliminary Assessment of the El Boleo Copper Cobalt Project***",** of the El Boleo Project located in Santa Rosalia, Baja California Sur, Mexico and dated January 31, 2007 (the "Technical Report") and filed in March 2007. In addition, I do hereby consent that any extracts from, or a summary of, the Technical Report may be filed with any stock exchange, any other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the publicly accessible company files on their websites.

Dated this 10th day of March, 2007

/s/ Timothy A. Ross_______________
Timothy A. Ross

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CONSENT OF AUTHOR

TO: The British Columbia Securities Commission
The Alberta Securities Commission
The Ontario Securities Commission
The Toronto Stock Exchange (TSX)
United States Securities & Exchange Commission (SEC)

I, John Wyche, MAusIMM, MMICA, CPMin, do hereby consent to the filing, with the regulatory authorities referred to above, of the technical report titled, "*Baja Mining Corp. – An Updated Preliminary Assessment of the El Boleo Copper Cobalt Project***",** of the El Boleo Project located in Santa Rosalia, Baja California Sur, Mexico and dated January 31, 2007 (the "Technical Report") and filed in March 2007. In addition, I do hereby consent that any extracts from, or a summary of, the Technical Report may be filed with any stock exchange, any other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the publicly accessible company files on their websites.

Dated this 10th day of March, 2007

/s/ *John Wyche*______________
John Wyche

Certificate of Qualifications

I, Donald Hunter, FAusIMM, do hereby certify that:

1. I am a consulting mining engineer with an office at 9 Wildwood Street, Kenmore Hills, Queensland, 4069, Australia.

2. I graduated from the Royal School of Mines, Imperial College of Science & Technology, University of London, with a Bachelor of Science Degree in Mining Engineering in 1973.

3. I am a Fellow of the Australasian Institute of Mining and Metallurgy and a Member of the Institution of Materials, Metallurgy and Mining, an Associate of the Royal School of Mines, a Chartered Professional (Mining) as recognized by the AusIMM and a Chartered Engineer as recognized by the IOMMM.

4. I have worked as a mining engineer for a total of 33 years since graduation from university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI-43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I have reviewed the *Updated Preliminary Assessment of the El Boléo Copper Cobalt Project* dated 31st January, 2007 (the "Preliminary Assessment") relating to the Boléo property located in Baja California Sur, Mexico. I have been remunerated for preparing this report on the basis of a fee for services.

7. Since July 2004 I have acted as a consulting mining engineer to Baja Mining Corporation of Vancouver, Canada in matters relating to the Boléo Project. I visited the property that is the subject of this Preliminary Assessment on three occasions; from 27th – 29th August 2004 and from 14th – 19th November 2004. A visit from 5th – 9th February 2006 was made specifically to observe part of the underground mining trial.

8. I am not aware of any material fact or material change with respect to the subject matter of the Preliminary Assessment that is not reflected in the Preliminary Assessment, the omission to disclose which makes the Preliminary Assessment misleading.

9. I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Preliminary Assessment has been prepared in compliance with that instrument and form.

11. As of the date of this certificate, to the best of my knowledge, information and belief, the Preliminary Assessment contains all scientific and technical information that is required to be disclosed to make the Preliminary Assessment not misleading.

12. I consent to the filing of the Preliminary Assessment with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Preliminary Assessment.

Dated this 16th day of March, 2007.

/s/ Donald J.E. Hunter
Donald J. E. Hunter

Certificate of Qualifications

I, Eric William Norton, do hereby certify that:

1. I am an employee of:

   Baja Mining Corp.
   2350 – 1177 West Hastings Street
   Vancouver, B.C.
   Canada

2. I graduated with a B.A. Sc. (Hons) degree in Engineering (Metallurgy and Materials Science) from the University of Toronto, Canada in 1974.

3. I am a Member of the Association of Professional Engineers and Geoscientists of BC.

4. I have worked as an engineer for a total of 33 years since my graduation from university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the section of the technical report titled Updated Preliminary Assessment of the El Boleo Copper Cobalt Project, (the "Technical Report"), dated January 2007, relating to the economic evaluation of the project. I visited the Property six times in 2006 between the months of April and December.

7. I have read NI 43-101 and certify that the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

8. I have been involved with the property since April 2006. This involvement takes the form of being the Director of Project Development for Baja Mining Corp., the owners of the property.

9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

10. I am not independent of the issuer and as a result have had the report (and in particular the Preliminary Economic Assessment) reviewed by Mr. Don Hunter, an independent qualified person.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated 31 January, 2007

Signature of Qualified Person

/s/ Eric Norton

Eric William Norton – B.A.Sc (Hons)
Print name of Qualified Person

Certificate of Qualifications

I, Scott Gokey Britton, do hereby certify that:

1. I am an employee of:

   Baja Mining Corp.
   2350 – 1177 West Hastings Street
   Vancouver, B.C.
   Canada

2. I graduated with a B. Sc. degree in Mining Engineering from the Virginia Polytechnic Institute & State University, Blacksburg, Virginia, USA in 1977.

3. I am a Member of the Association of Professional Engineers of Wyoming (active) and Pennsylvania (Inactive). I am a Registered Member of the Professional Society of Mining Engineers (SME-AIME).

4. I have worked as an engineer for a total of 30 years since my graduation from university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the section of the technical report titled Updated Preliminary Assessment of the El Boleo Copper Cobalt Project, (the "Technical Report"), dated January 2007, relating to the economic evaluation of the project. I visited the Property six times in 2006 between the months of April and December.

7. I have read NI 43-101 and certify that the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

8. I have been involved with the property since May 2006. This involvement takes the form of being the General Manager – Mining for Baja Mining Corp., the owners of the property.

9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

10. I am not independent of the issuer and as a result have had the report (and in particular the Underground Mining) reviewed by Mr. Timothy Ross, an independent qualified person.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated 31 January, 2007

Signature of Qualified Person

/s/ Scott Britton

Scott Gokey Britton
Print name of Qualified Person

Certificate of Qualifications

I, Grant Bryan Bosworth, P.Eng., do hereby certify that:

1. At the time of the preparation of the technical report titled Updated Preliminary Assessment of the El Boleo Copper Cobalt Project, (the "Technical Report") dated January 2007, I was employed as Project Manager and that I am employed by and carried out this assignment for:

   Wardrop Engineering Inc.
   #800 – 555 West Hastings Street
   Vancouver, B.C., Canada, V6B 1M1

2. I graduated with a degree in B. Sc. (Mineral Engineering) from the University of Alberta in 1981.

3. I am a registered Professional Engineer in the Province of British Columbia and the Province of Alberta.

4. I have worked in engineering, construction management and project management in the minerals industry for 25 years.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for certain inputs to the technical report titled Updated Preliminary Assessment of the El Boleo Copper Cobalt Project, (the "Technical Report") dated January 2007 relating to the Property, specifically the plant capital cost projections and certain surface infrastructure costs. I have visited the Property on one occasion in August 2006.

7. I have had an involvement in the Property since March 2006. The nature of this involvement includes managing sundry engineering services to the feasibility study as a sub-consultant to Bateman Engineering Canada Corp.

8. I have not had prior involvement with the property that is the subject of the Feasibility.

9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

10. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 42-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Feasibility with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Feasibility.

Dated 2 February, 2007

Signature of Qualified Person

/s/ Grant Bryan Bosworth

Grant Bryan Bosworth, P.Eng.
Print name of Qualified Person

Certificate of Qualifications

I, William Yeo, MAusIMM, do hereby certify that:

1. I am an employee of:

   Hellman & Schofield Pty Ltd
   Level 4, 46 Edward Street
   BRISBANE QLD 4000
   AUSTRALIA

2. I graduated with a BSc(Hons) degree in geology from Oxford Polytechnic, UK in 1979. In addition I have obtained a PhD in geochemistry and petrology from University Bristol, UK in 1984.

3. I am a Member of the Australasian Institute of Mining and Metallurgy.

4. I have worked as a geologist for a total of 18 years since my graduation from university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the Resource aspects of the technical report titled Updated Preliminary Assessment of the El Boleo Copper Cobalt Project, (the "Technical Report") and dated January 2007 relating to the Property. I have visited the Property for three weeks, in August and 2 weeks in November 2004 and again for 5 weeks in September and October 2006 and 2 weeks in January 2007.

7. I have had an involvement in the Property since August 2004. The nature of this involvement includes resource estimation and general consulting in relation to QA/QC, geological logging.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

9. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated 31 January, 2007

Signature of Qualified Person

/s/ William J. A. Yeo

William J. A. Yeo, MAusIMM PhD
Print name of Qualified Person

Certificate of Qualifications

I, Michael Richard Holmes, MSAIMM, do hereby certify that:

1. I am an employee of:

   Bateman Engineering Pty Ltd.
   Level 8, 301 Coronation Drive
   Milton, QLD 4064
   AUSTRALIA

2. I graduated with a BSc Engineering degree in Metallurgy from the University of the Witwatersrand, RSA in 1985. In addition I have obtained a BComm, from UNISA in 1994 and a MBA from the Wits Business School in Johannesburg in 1997.

3. I am a Member of the South African Institute of Mining and Metallurgy (Membership No. 40334) and am a registered, Professional Engineer with ECSA – the Engineering Council of South Africa (Pr. Eng No. 910429).

4. I have worked in the metallurgical field for a total of 19 years.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for various inputs to the technical report titled Updated Preliminary Assessment of the El Boleo Copper Cobalt Project, (the "Technical Report") and dated January 2007 relating to the Property, including all metallurgical input, certain capital cost projections and the process plant operating cost projections. I have visited the Property on four occasions in the past 32 months, namely May 2004, August 2004, February 2005 and September 2006.

7. I have had an involvement in the Property since May 2004. The nature of this involvement includes the co-ordination of the feasibility study as the designated Bateman Project Manager

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

9. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated 31 January, 2007

Signature of Qualified Person

/s/ Michael R. Holmes

Michael Richard Holmes BSc (Eng), BComm, MBA, MSAIMM, Pr Eng
Print name of Qualified Person

Certificate of Qualifications

I, Timothy A. Ross, do hereby certify that:

1. I am an employee of:

   Agapito Associates, Inc.
   1726 Cole Blvd. Bldg. 22 Ste. 130
   Golden, CO 80401
   USA

2. I graduated with a BS degree in Mining Engineering from Virginia Polytechnic and State University in 1977.

3. I am a Member of the Society of Mining Engineers and authorized to use the title "SME Founding Registered Member".

4. I have worked as an engineer for a total of 30 years since my graduation from university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the section of the technical report titled Updated Preliminary Assessment of the El Boleo Copper Cobalt Project, (the "Technical Report"), dated January 2007, relating to underground mining of the deposit. I have visited the Property six times since August of 2005.

7. I have had an involvement in the Property since July 2005. The nature of this involvement includes geotechnical evaluation, underground mine design and underground mine planning.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

9. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated 11 January, 2007

Signature of Qualified Person

/s/ Timothy A. Ross

Timothy A. Ross
Print name of Qualified Person

Certificate of Qualifications

I, John Wyche, MAusIMM, MMICA, CPMin, do hereby certify that:

1. I am an employee of:

   Australian Mine Design and Development Pty Ltd
   Level 4, 46 Edward Street,
   BRISBANE QLD 4000
   AUSTRALIA

2. I graduated with a BE(Hons) degree in mining from Queensland University, Australia, in 1981. In addition I have obtained a BComm from Queensland University in 1990.

3. I am a Member of the Australasian Institute of Mining and Metallurgy, and member of Mineral Industry Consultants Association (Aust) and a Chartered Practicing Engineer (Mining).

4. I have worked as a mining engineer for a total of 25 years since my graduation from university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am partly responsible for the preparation of the Mining aspects of technical report titled Updated Preliminary Assessment of the El Boleo Copper Cobalt Project, (the "Technical Report") and dated January 2007 relating to the Property. I have visited the Property for 4 days in January 2005, 1 week in December 2005 and again for 4 days in February 2006 during the mining trial.

7. I have had an involvement in the Property since June 2004. The nature of this involvement includes general consulting in relation to mine planning and preparation of an underground mining trial.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

9. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated 31 January, 2007

Signature of Qualified Person

/s/ John Wyche

John Wyche MAusIMM, MMICA, CPMin
Print name of Qualified Person