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 July 2007

Commission File Number 000-51690

Baja Mining Corp.

(Translation of registrant's name into English)

2350 – 1177 West Hastings Street,
Vancouver, British Columbia V6E 2K3

(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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1. **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: June 21, 2007

By: /s/ John Greenslade

John Greenslade
President

BAJA MINING CORP.

EL BOLEO PROJECT

FEASIBILITY STUDY SUMMARY REPORT

BAJA CALIFORNIA SOUTH, MEXICO

Prepared for

Baja Mining Corp.

by

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11 July 2007

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1 SUMMARY

1.1 GEOLOGY & MINERAL RESOURCE MODELLING

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

The deposit, which occurs in the Boleo 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 continuously mined predominantly by underground methods from 1886 to 1972, during which time an estimated 18 Mt of ore were treated. After 1972 both underground and open pit mining was carried out sporadically until the copper smelter at Santa Rosalía closed in 1985. Since cessation of mining operations International Curator Resources (Curator) carried out exploration between the years 1993 to 1998 and Minera y Metalúrgica del Boleo S.A. de C.V. (MMB) has been carrying out exploration and project development activities from 2004 to the present. When MMB acquired the property in 2004 it also acquired the Curator drill hole assay database which has been used in the current resource study.

Independent geologic consultants, Hellman & Schofield (H&S) of Sydney, Australia were engaged in 2004 to oversee the geologic modelling of the Boleo deposit and since then have produced a number of resource estimates in accordance with NI 43-101 standards. In addition to updating resource estimates as new drilling data became available, H&S also reviewed previous estimates.

The appropriateness and success of analytical processes and quality control methods used by previous explorers of the property have been studied at length and as a result, since 2004 new quality assurance procedures have been developed for MMB's exploration activities. These included the preparation of matrix matched assay standards 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.

The deposit will be mined by a combination of underground and open pit mining methods and since planning and operating requirements are quite different separate block models were developed for each mining method.

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 incorporating minimum and maximum mining height considerations have been developed for Mantos 1, 2, 3, and 4 for underground mine planning while a 3-D block model was developed for Mantos 0, 3aa and 3a which will be mined by open pit methods.

A 3D digital model was developed from geological interpretation, data and geologic information developed by MMB which included 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 sets available for grade estimation, from increasingly localised data search regimes within search ellipses of varying dimensions.

For 3D Block and Seam models the resource categories were determined using search ellipses with the following dimensions:

- 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 the resource category was based on the number of composites contained within the search ellipses:

- 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 the resource category was based on the number of composites contained within the search ellipse:

- 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 grade of 0.5% Cu can be found in Table 1.

The Inferred Resource can be found in Table 2.

An additional model was created to account for historic underground workings in Mantos 1 and 3. Tonnes were factored down by 12% to account for material extracted and processed.

Table 1: Measured and Indicated Resources

Final Models Cutoff: CuEq = 0.5%			Manto 0	1	2	3AA	3A	3	4	Total
Measured and Indicated Total	3D Block *Open pit*	tonnes $x10^6$	9.9	19.5	61.9	3.2	46.3	116.0	20.4	277.1
		CuEq%	0.69	2.24	1.66	1.99	1.49	2.06	1.11	1.77
		Cu%	0.03	0.96	0.37	0.53	0.38	1.06	0.51	0.70
		Co%	0.010	0.055	0.050	0.082	0.063	0.066	0.038	0.057
		Zn%	0.71	0.92	0.98	0.81	0.61	0.42	0.27	0.62
		Mn%	1.28	2.99	4.30	5.39	4.07	2.18	1.75	3.00
	Seam Underground	*tonnes $x10^6$*		10.8	30.5	2.9	33.7	62.9	6.2	147.0
		CuEq%		2.81	1.96	2.07	1.43	2.63	1.55	2.17
		Cu%		1.37	0.50	0.58	0.38	1.48	0.87	0.97
		Co%		0.071	0.060	0.082	0.059	0.076	0.044	0.067
		Zn%		0.91	1.07	0.84	0.58	0.49	0.30	0.66
		Mn%		3.28	4.56	5.33	2.18	2.38	2.18	2.90

Table 2: Inferred Resources

Final Models Cutoff: CuEq = 0.5%			Manto 0	1	2	3AA	3A	3	4	Total
Inferred	3D Block *Open pit*	tonnes $x10^6$	7.4	49.2	52.5	0.6	23.8	56.4	63.3	253.2
		CuEq%	0.66	1.72	1.27	1.45	1.20	1.48	0.90	1.29
		Cu%	0.02	0.48	0.22	0.51	0.30	0.54	0.39	0.39
		Co%	0.008	0.044	0.042	0.053	0.044	0.047	0.032	0.040
		Zn%	0.69	1.01	0.79	0.51	0.58	0.59	0.25	0.63
		Mn%	1.77	2.93	3.60	5.09	3.40	2.53	1.50	2.64
	Seam Underground	*tonnes $x10^6$*		8.0	21.8	0.2	18.5	23.5	13.6	85.6
		CuEq%		2.38	1.72	2.80	1.31	2.03	1.05	1.67
		Cu%		0.65	0.28	1.40	0.29	0.87	0.55	0.52
		Co%		0.053	0.053	0.084	0.049	0.057	0.032	0.050
		Zn%		1.49	1.14	0.70	0.66	0.74	0.22	0.81
		Mn%		3.92	4.70	8.18	2.69	2.50	1.56	3.10

1.2 METALLURGY & PROCESS DESIGN

Treatment strategies for the Boleo 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.

In parallel with the adoption of selective mining of higher grades (and significantly reduced waste mining), Bateman have 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 via a solvent extraction process known as the Direct Solvent Extraction (DSX) process. This process allows for the removal proposed zinc and cobalt away from manganese, magnesium, and calcium.

These testwork breakthroughs, coupled with changes in processing strategy, result in a more robust, operable flowsheet with reduced operating and capital cost.

More significantly, the propose flowsheet has been successfully tested in extensive testwork campaigns over the past three years. A "Proof of Concept" pilot plant campaign was held at SGS's facilities in Ontario in November 2004 and another, more comprehensive "Fully Integrated" pilot plant campaign was conducted as the same institution in June and July of 2006. The success of these two pilot campaigns provides a high level of confidence in the proposed flowsheet, confirming that the process development initiative undertaken over the past two years has taken the right direction.

The proposed process flowsheet now consists of a two stage atmospheric leach of a whole ore stream in an acidic, chloride environment. The leach circuit consists of both an oxidative and a reductive leach, processes that has been well proven in extensive batch and pilot level testing over the years.

The leached metals are separated from the leach slurry in a counter current decantation (CCD) washing circuit. Experience in the nickel laterite plants in Western Australia and the Sepon Copper project in Laos, has demonstrated that large CCD circuits can operate effectively on difficult-to-handle clay ores to recover dissolved metal values. This key process step has been extensively tested at bench scale and at pilot plant level and found to be successful in the recovery of metals in solution from leached Boleo solid residues.

Finally, the dissolved metals are recovered from the wash solution and concentrated employing four separate solvent extraction circuits, two electrowinning circuits and a fluid bed drying operation to produce high-quality copper and cobalt metal cathode and zinc sulphate monohydrate crystals.

The metal recovery circuits are typical of those deployed in numerous operations worldwide. In developing the Boleo flowsheet, Bateman was able to incorporate the results of the earlier Curator and Mintec driven testwork, supplementing these with additional bench scale and extensive pilot plant testwork results, underpinned by information from recent Bateman's projects featuring similar processes and unit operations.

This following listing summarises the metallurgical testwork undertaken in the development of the Boleo process flowsheet to date:

- Pre-feasibility Testwork Review by Ken Baxter – Bateman's Technical Head of Copper Processing – in 3Q 2001
- Bench Scale Solid-Liquid Separation Testwork on Leached Boleo Ore Samples at SGS in Ontario – in 4Q 2004 by Outokumpu and Pocock Industrial Inc.
- Commencement of DSX testwork by the CSIRO in Perth, Australia – in 4Q 2004
- "Proof of Concept" Pilot Campaign at SGS – in 4Q 2004
- Boleo Tailings Characterization by the SGS's Environmental Group – in 4Q 2004
- Manganese Carbonate Production Testing at SGS – in 3Q and 4Q 2005
- DSX Cycle Testwork by the Bateman Solvent Extraction Group – in 4Q 2005
- Boleo Carbonate Ore Testing at SGS – in 1Q 2006
- Fully Integrated Pilot Campaign Testing – in 3Q 2006
- Material Handling Testwork by Jenike & Johanson – in 3Q 2006
- DSX Optimization Testwork by the Bateman Solvent Extraction Group – in 4Q 2006
- Tailings Pumping Testwork – in 4Q 2006
- Boleo Tailings Characterization by the SGS's Environmental Group – in 4Q 2006
- Ore Variability Testwork – in 2Q 2007.

The results of the various testwork campaigns have been employed Bateman Engineering Canada Corp. to generate flowsheet designs for the purposes of advancing the feasibility study for a planned greenfield development consisting of open pit and underground mining operations, a hydrometallurgical processing plant and all the associated facilities and infrastructure necessary to produce copper and cobalt metal and a zinc sulphate monohydrate salt on site.

The process plant is being designed to produce and treat 3.1 Mt/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, there is the potential to produce up to 100,000 t/a of a manganese carbonate product via the relatively simple addition of a small number of unit operations to the current flowsheet.

It is MMB's intention to "de-bottleneck" the plant operation as part of a continuous improvement programme to ensure the product production levels remain at target values as ROM head grade begins to decrease over the LOM. De-bottlenecking of the production facility requires modest capital investments at the appropriate time. Capital costs associated with de-bottlenecking and improving operational efficiency have been allowed for in the financial analysis of all of the base case scenarios.

1.3 MINING

Based on resource models by H&S, mine engineering consultant, Australian Mine Development and Design, (AMDAD) prepared a 25 year life of mine plan and schedule using mainly underground mining methods for the first 20 years of production followed by 5 years of surface mining. A number of different mining scenarios were tested before this scenario using predominantly underground mining operations supported by a number of small, open pit mines was selected as that which would best deliver the targeted plant feed grade at the required rate.

In addition to carrying out several mine planning and scheduling iterations AMDAD also prepared pit designs, evaluated equipment requirements, estimated production rates and prepared operating and capital cost estimates for the Feasibility Study.

The seam formation and low material strength of the mantos suggested conventional "soft rock" underground mining methods such as used in coal, potash, or salt mining would be appropriate for Boleo underground conditions and accordingly a number of methods were examined for their suitability. 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 and accommodated within the complex, fault disturbed structure of mantos. After extensive study, room-and-pillar mining using continuous miners was selected because the method requires relatively low capital cost and can accommodate variations and undulations in the footwall as well as variations in mining height to an extent that would be impossible with longwall or shortwall methods. Moreover, room and pillar mining can achieve relatively high productivity and panels can be layout out in such a way that the mineral recovery from relatively small, fault bounded mining blocks can be maximized. In the Boleo environment room and pillar mining will provide a degree of planning and operational flexibility that neither long- or short-wall mining methods can.

The resource seam model was used to define areas where room and pillar mining could be carried out. The criteria used for this determination were the following;

- 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 1.8 m height.
- Maximum mining height of 4.2 m, matching the designed 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 using '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 to 2006 under the supervision of consultants, Australian Mine Design & Development (AMDAD), Sydney, Australia, and Agapito Associates, Inc. (AAI), Grand Junction, Colorado, USA. These tests confirmed the suitability of the proposed continuous mining method and provided valuable field data and information concerning ground behaviour during mining and methods of ground support.

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 Boleo property, has also been modelled as the main source of calcium carbonate needed for process plant operations. This limestone will be quarried at a rate of approximately 800,000 t/a for the first 20 years of operation. A small amount of additional limestone will need to be imported after Year 5 of operations to supplement the locally available limestone. After Year 20 the project limestone requirement will be met entirely by the imported product. The operating cost of such an initiative has been allowed for in the economic model for the project.

1.4 ECONOMIC ASSESSMENT

A financial model was created utilizing the current mine production schedule over an initial 25 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$1.50/lb, cobalt US$15.00/lb and zinc sulphate US$1,200/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 Table 3 to indicate the effect of a plus or minus 10% change in the key variables.

Table 3: Project Economic Sensitivities Summary

	After Tax IRR (%)				After Tax NPV at 8% ($ millions)		
	-10%	Base Case	+10%		-10%	Base Case	+10%
Copper Price	21.6	24.7	27.5		571	700	822
Cobalt Price	24.1	24.7	25.3		663	700	738
Capital Cost	27.3	24.7	22.3		744	700	652
Operating Cost	26.0	24.7	23.4		765	700	635

This modelling, based on the current mine schedule, indicates that the project is financially attractive at base-case metal prices. Financial modelling, using the base case prices and 25 years for the project life, shows that the project could generate a net after- tax Internal Rate of Return (IRR) of 24.7%.

At an 8% discount rate, the project generates an NPV, after tax value of US$700.3 million. Using a 6% discount rate to repeat the above analysis generates an NPV, after tax, of $US924.0 million.

2 INTRODUCTION

H&S were commissioned in 2004 and were retained throughout the feasibility study to carry out resource studies which included;

- production of 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
- production of a digital grade model, for all mantos, for Cu, Co, Zn, and Mn
- production of an underground void model
- provision of a Confidence Classification, taking into account drill spacing, structural and other controls on mineralization, suitable for quotation to NI 43-101 standards
- review the QA/QC data as relevant to sampling, assaying precision and accuracy
- assessment of 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 prior to MMB's direct involvement in the project were sourced from various reports by consultants G. Peatfield, B. Smee and D. Mehner (Peatfield and Smee 1997, Peatfield 1997, Peatfield 1998, Mehner 2003).

Between 1993 and 1998 drilling, sampling and assaying activities by Curator provided a significant part of the data used in the current resource study. 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 prior to that time.

Three infill drill programs were carried out by MMB, 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 March 31, 2007 (DDHs 04-928 to 07-1233).

These programs were observed and monitored by H&S's representative Dr. B Yeo who also prepared the current resource estimate.

3 RELIANCE ON OTHER EXPERTS

The capital cost for development of the Boleo 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 coordinated 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 Geotecnica	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 / operations Terminal	Oemsa	Mexico City, Mexico
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 Inc. have relied on the consultants listed above for the generation of capital and operating cost estimates in their particular areas of expertise. Neither Bateman Engineering Canada Corp nor Wardrop Engineering has attempted to formally verify the accuracy or sufficiency of the cost estimates provided by these consultants.

The portions of the report to which the above disclaimer applies are Section 18.1.3, Capital Cost Estimate and Section 18.1.4, Operating Cost Estimate.

4 PROPERTY DESCRIPTION & LOCATION[1]

4.1 LOCATION

The Boleo project is located along the east coast of the Baja peninsula centred on the port town of Santa Rosalía 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 Boleo 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 Boleo 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. Rosalía in the San Bruno basin area. The titled concessions are listed in Table 4 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 4: Boleo Property, Sole Concessions, January 2007

Claim	Title No.	Surface Area (ha)	Date Initiated	Expiry Date	Annual Taxes
El Boleo	218082	4,975.6132	Sept. 29-2000	Sept. 28-2050	285,004
El Boleo I	218092	72.4463	Aug. 31-2000	Aug. 30-2050	4,150
El Boleo19, 256.1872 II fracc. Uno	218179	1,296.6156	Sept. 29-2000	Sept. 28-2050	74,270
El Boleo II fracc. Uno A	218180	507.2841	Sept. 29-2000	Sept. 28-2050	29,058
Boleo 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
Boleo II fracc. IV	218975	267.1579	Jan. 28-2003	Jan. 27-2053	7,608
Boleo X fracc. 5	211055	1.3829	Mar. 24-2000	Mar. 23-2050	80
Boleo X fracc. 8	211058	3.9486	Mar. 24-2000	Mar. 23-2050	226
Boleo X fracc. 9	211059	9.9612	Mar. 24-2000	Mar. 23-2050	570
Boleo X fracc 12	211062	3.1241	Mar. 24-2000	Mar. 23-2050	178
Boleo 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**

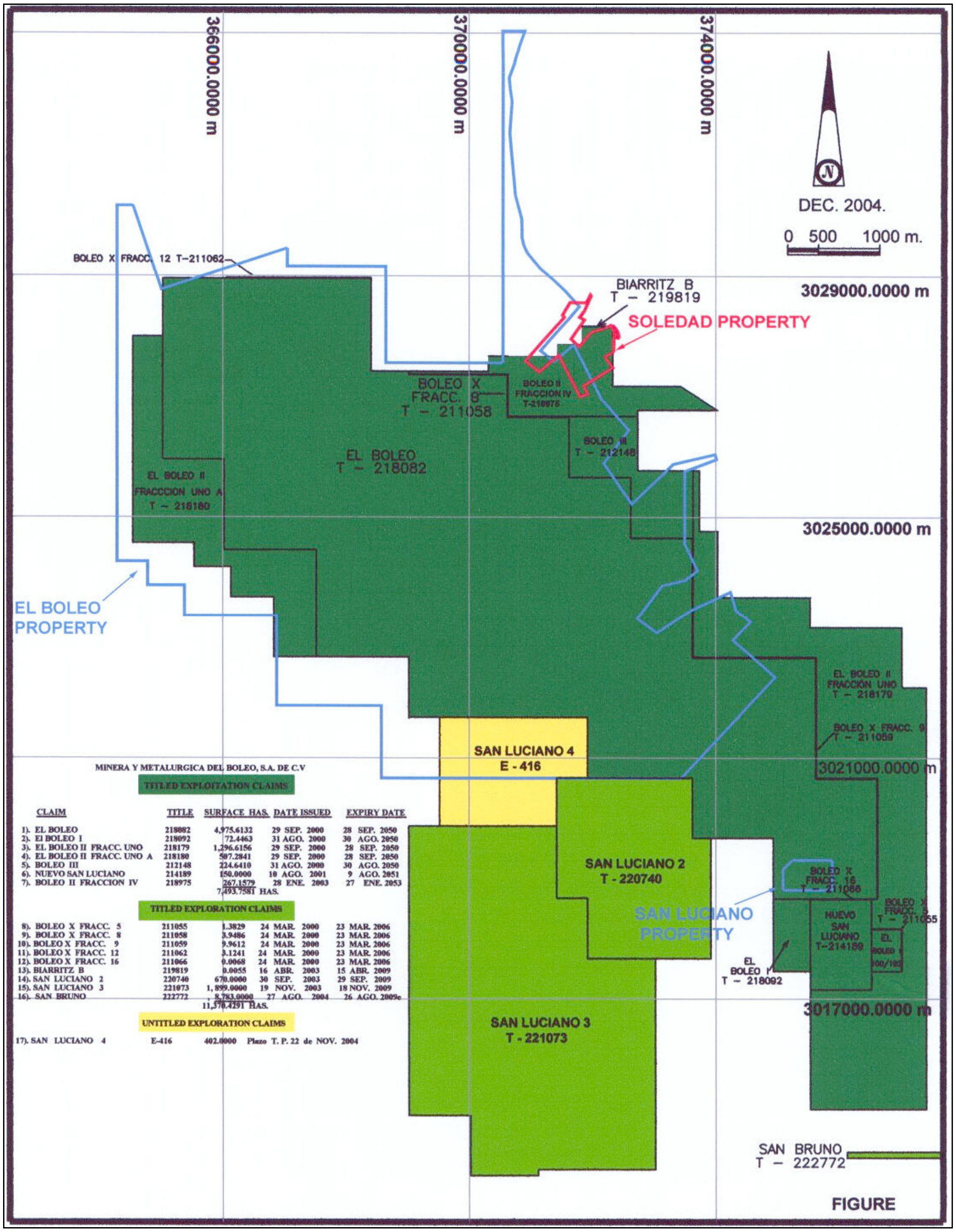
Note: the exchange rate used is Pesos 10.8116 =US$1

The project includes three surface lots currently totalling 6,692.58 ha as shown in Table 5 and Figure 2. It should be noted that Baja Mining Corp. is currently consolidating the various surface claims into a single master deed and as part of this process, is streamlining its final boundaries with other property owners and expects the final surface property size to change slightly from the above mentioned amount.

Table 5: Boleo Project, Surface Property, and Annual Taxes

Surface Lot	Size (ha)	Annual Taxes
El Boleo	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 Boleo Property Map



The surface property also contains one rented tract totalling 539 ha. This tract is Ejido land located on the western edge of the Project Area and covers a portion of the anticipated maximum extent of the tailings disposal impoundment. The rented period is 35 years with both a renewal option and the obligation of both parties to continue to work towards an outright purchase.

4.3 OWNERSHIP

The mineral concessions covering El Boleo copper-cobalt-zinc manganese deposit are 100% owned by Minera y Metalúrgica del Boleo 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 & 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 4 and 5). 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 Comprobaciones 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 & 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 Boleo 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 Boleo 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 centered on the Desierto de Vizcaino on the west central coast of the Baja. The Biosphere extends south to encompass the historic Boleo Mining District and to protect certain environmental and cultural features in the town of Santa Rosalía. It is believed the Biosphere intended to protect the historic buildings dating from the late 1800s associated with the early mining of the Boleo district. It should be noted these buildings are outside the Boleo project and study area boundary and will not be directly affected by project development.

Since the Boleo 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 Boleo property is located on tidewater on the east coast of the State of Baja California Sur, Mexico, adjacent to the town of Santa Rosalía. 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 Rosalía. 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 Rosalía 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 Sonora 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. Historically, and as recently as late 2006 the area has been subject to flash flooding caused by cyclonic conditions in the Gulf of California. Although comparatively rare, these storm events have been destructive and suitable allowance have been made in the engineering and design work to account for and minimize the impact of such events so that Mining Operations can be scheduled 365 days per year, 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 Rosalía, 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 Rosalía has a population of 10,000 people and services

[2] Information derived from Mehner, 2003

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 Rosalía. Other items including machinery and trained personnel are readily available from 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 Boleo 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, cap lamp 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 (blowing pressure)
- steel tunnel cowling
- mine supplies consisting of wooden props, 8 ft, 7/8" headed rebar roof bolts, plates, wire mesh and steel strapping
- several pieces of underground mine equipment, including a “roadheader” style of 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 Boleo 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 Rosalía (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 short 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 Boleo and after recognizing the vast potential recommended a significant investment to develop the district. On May 16, 1885, the Compagnie du Boleo (later to be known in Mexico as the Compañía del Boleo, S.A. – “the Boleo Company”) was formed in Paris, backed mainly by the banking interests of the French House of Rothschild.

On July 7, 1885, the Boleo 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 Boleo 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 Boleo 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 Compañía Minera Santa Rosalía, 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 offshore, until final closure in 1985. Table 6 summarizes the historical mining activities during the periods of production.

Table 6: Historical Mining Activities at Boleo

Period of Activity	Metric Tonnes Mined	Avg. Copper Grade	Tonnes Cu 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%	Unk.

During the latter years of operation at Boleo, 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 higher than 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 Compañía Minera Santa Rosalía 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.

After cessation of operations at Boleo 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 a 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 works completed since 1993 was prepared by Fluor Daniel Wright and presented to the underlying owners.

In summary, Fluor Daniel Wright estimated proven reserves at Boleo to be 71.2 Mt grading 1.44% Cu, 0.092% Co and 0.55% Zn with further probable reserves of 13.1 Mt grading 1.57% Cu, 0.065% Co and 0.81% Zn. These were deemed sufficient to support an 11,500 tonnes per day 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 Boleo concessions to Terratech.

After re-gaining control of Boleo, 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 reorganization. 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 Metalúrgica del Boleo S.A. de C.V.

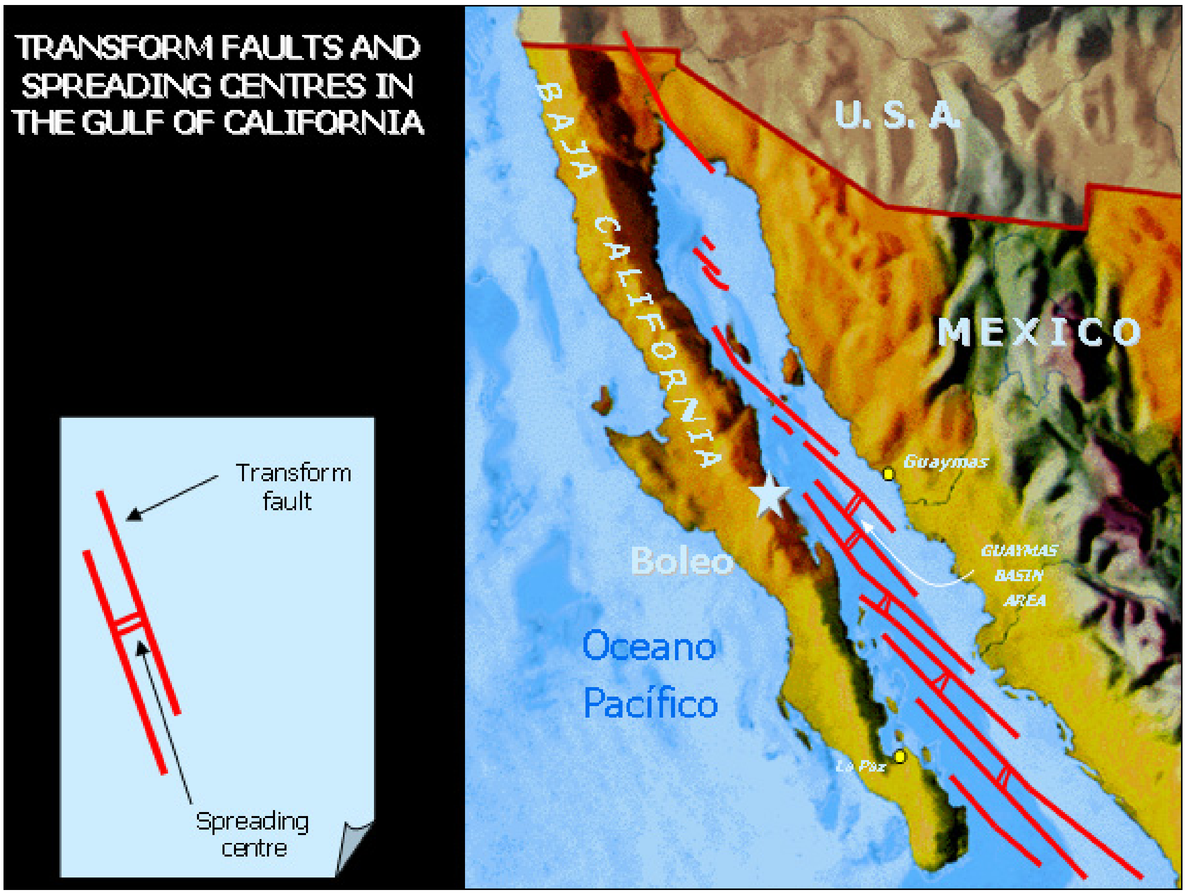
In November 2003, the shareholders of Mintec entered into an agreement with First Goldwater Resources Inc., a TSX-Venture Exchange listed public company, to exchange all the issued shares of Mintec for shares in First Goldwater (now Baja Mining Corp) and effectively purchased the Boleo copper-cobalt-zinc 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 Boleo deposits occur within the Boleo sub-basin of the Santa Rosalía basin. This basin formed because 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 Boleo Geological Setting



The timing of initial rifting varies from 13 Ma to 8 Ma. In the Boleo 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).

[3] Information from Mehner, 2003

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 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).

Stratigraphically the Boleo copper-cobalt-zinc manganese deposits occur within the late Miocene age succession of fine to coarse clastic sedimentary rocks of the Boleo Formation, lying unconformably on andesitic rocks of early to middle Miocene age called the Comondú Volcanics. The Boleo 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 Boleo and overlying formations collectively make up the so-called Boleo Basin. Locally, the entire succession is capped by Pleistocene to recent flows and pyroclastic rocks of the Tres Vírgenes 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.

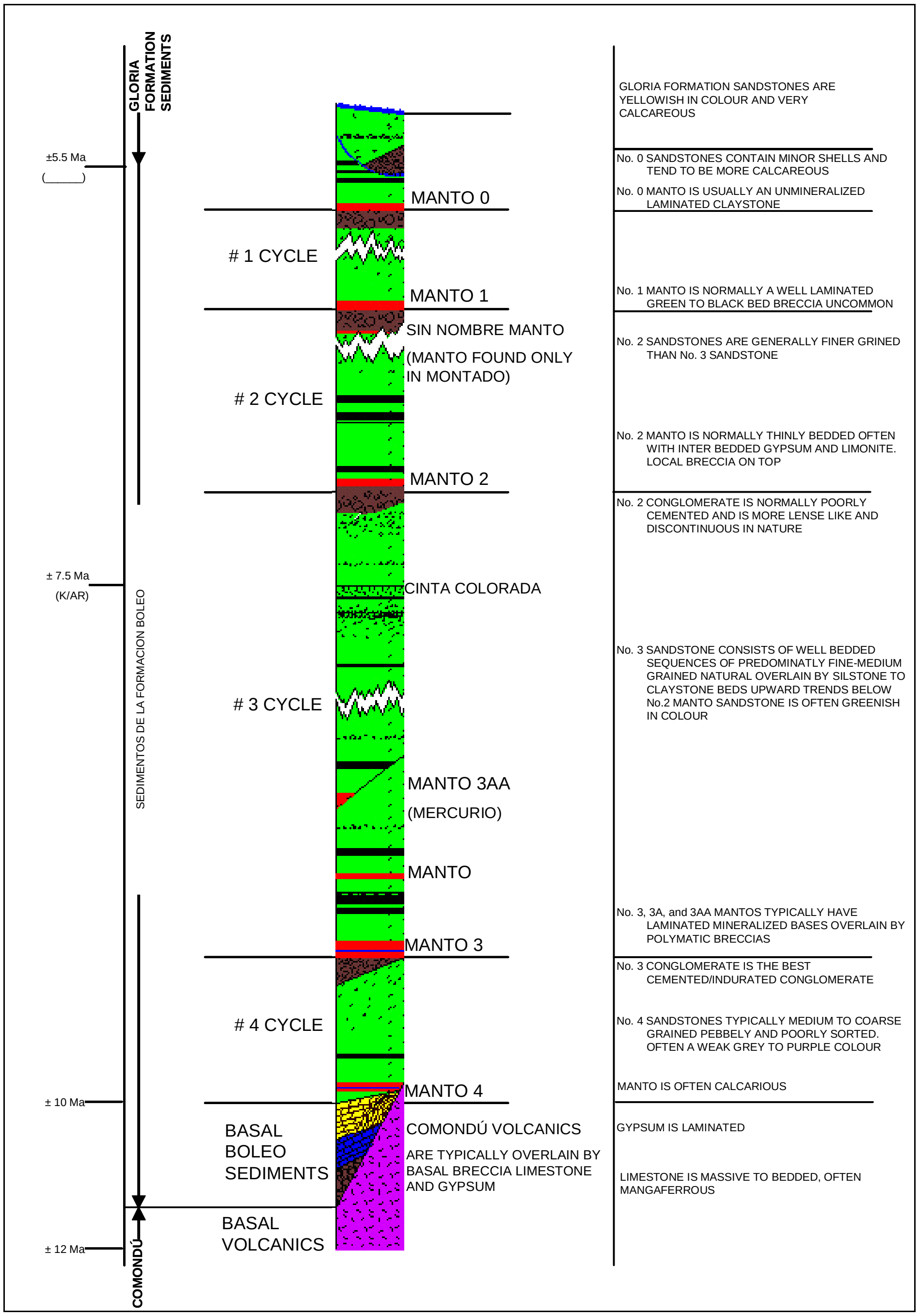
7.2 PROPERTY GEOLOGY

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 Ma to 11 Ma are underlain by Cretaceous granodiorite (Schmidt, 1975).

The overlying Boleo 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 Boleo Formation is a 1 m to 5 m 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.

Figure 4: Boleo Formation Stratigraphic Column



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.

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 m 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 the 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 Boleo formation rocks.

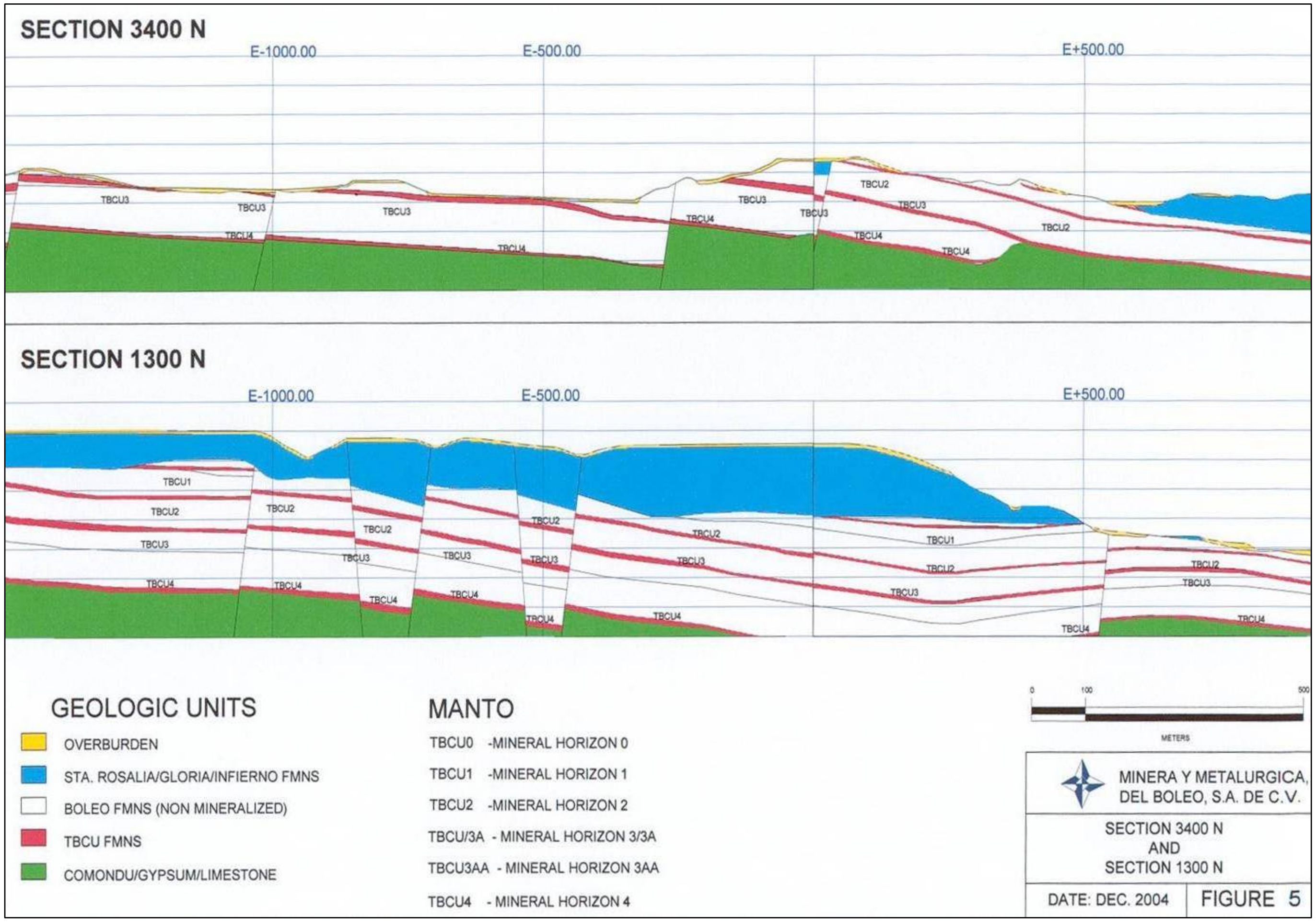
Unconformably overlying the Boleo clastics are fossiliferous marine sandstones and conglomerates of the lower Pliocene (about 5.3 Ma.) Gloria formation (Bailes et al., 2001). These in turn are overlain by slight unconformity with a sequence of fossiliferous marine sandstones and conglomerates of the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

7.3 STRUCTURAL GEOLOGY

The Boleo Formation rests on an irregular volcanic basement, with several distinct basement highs and intervening troughs. In places, these basement highs were so pronounced that they 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 on the major faults, with much lesser movements, 1 m to 5 m, 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.

Figure 5: Geological Cross-Sections



Major faults at Boleo are, in most cases, laterally separated by several hundred, to in some cases, over a thousand metres. 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.

8 DEPOSIT TYPES

The Boleo 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 main subject of this report - are the manto hosted copper-cobalt-zinc manganese deposits, which occur in Boleo formation clastic sediments.

In addition limestone will be mined in large quantities from the Boleo District over the life of the mine. These limestone deposits formed from calcareous mineral matter occur over portions of the property, and assume significance in the Arroyo Boleo and Arroyo Infierno areas of the district. The basal unit in the Boleo Formation for example is a 1 m to 5 m 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.

Other possible mineral deposit 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 Boleo and 20 km southeast on San Marcos Island.

9 MINERALIZATION

Deposits of copper-cobalt-zinc-manganese mineralization in the Boleo 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 Boleo 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, Mantos 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 Boleo 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 Boleo 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.

MMB geological staff have outlined the presence of a thin layer of claystone within a manto creating a “false bottom” over large areas of Mantos 1 and 3. 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” refer to Section 9.1.1.

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 Boleo 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 Boleo 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.

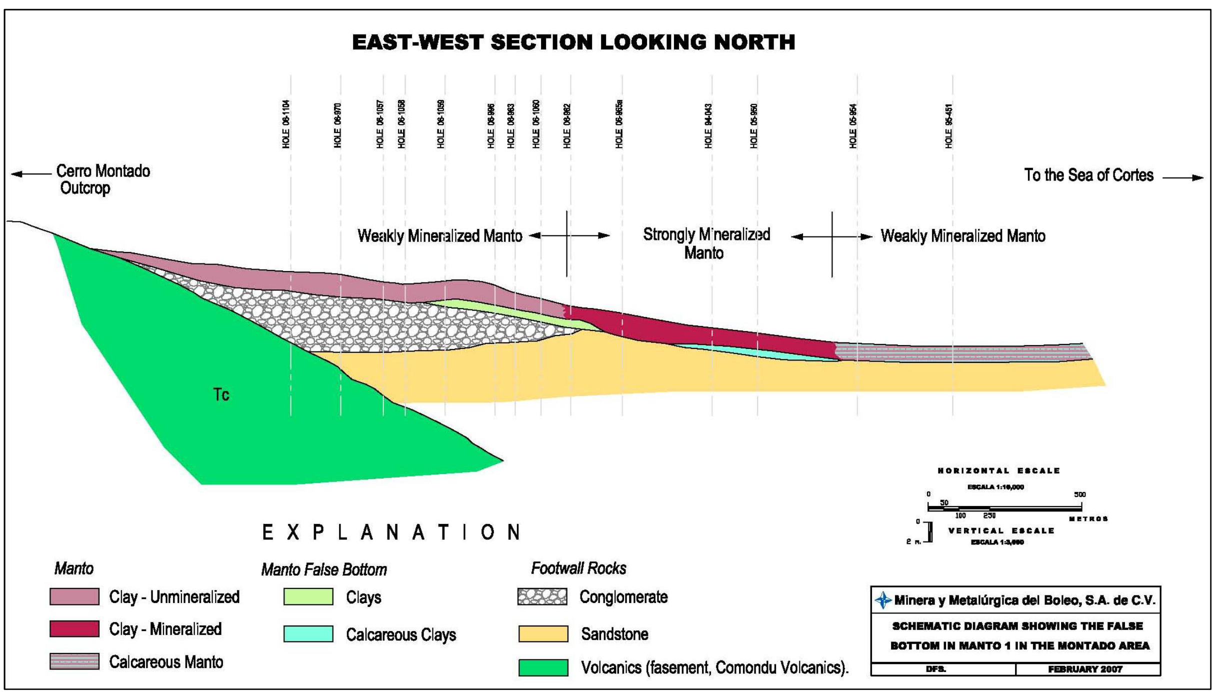
Manto 1 makes up the bulk of the mineral resource in the southeast portion of the Boleo Property, where the lower mantos (3 and 4) were for the most part not deposited. The beds dip to the southeast and as a consequence, Manto 1 lies deeply buried in this area. To the northwest, Manto 1 overlies the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

9.1.1 Boleo 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 2004 to 2007, 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 defined if two or more adjacent diamond drill holes exhibit similar and co-relatable footwall stratigraphy (Figure 6 and Figure 7).

Figure 6: Manto 1 Hole DDH955 – False Bottom Geology

Figure 7: Manto 1 Schematic Cross Section of False Bottom Geology



False bottoms have been identified to date through drill programs and the position of old stopes in Mantos 1, 3 and 4. The location of false bottoms in Manto 3 is shown in Figure 8 and Figure 9. Thickness of these false bottom units at the base of the mantos ranges from less than a metre to several metres wide.

Figure 8: Manto 3 Location of False Bottoms

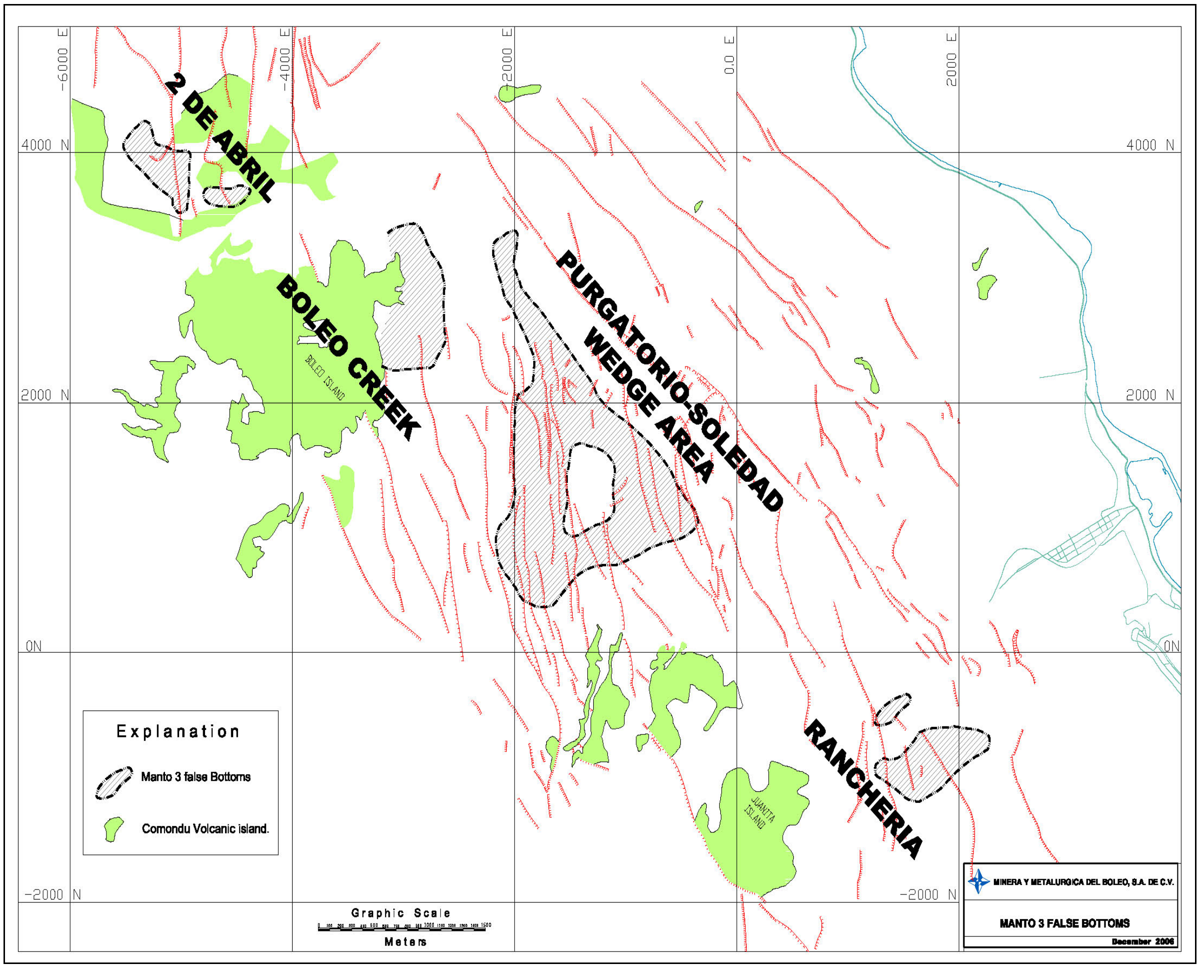
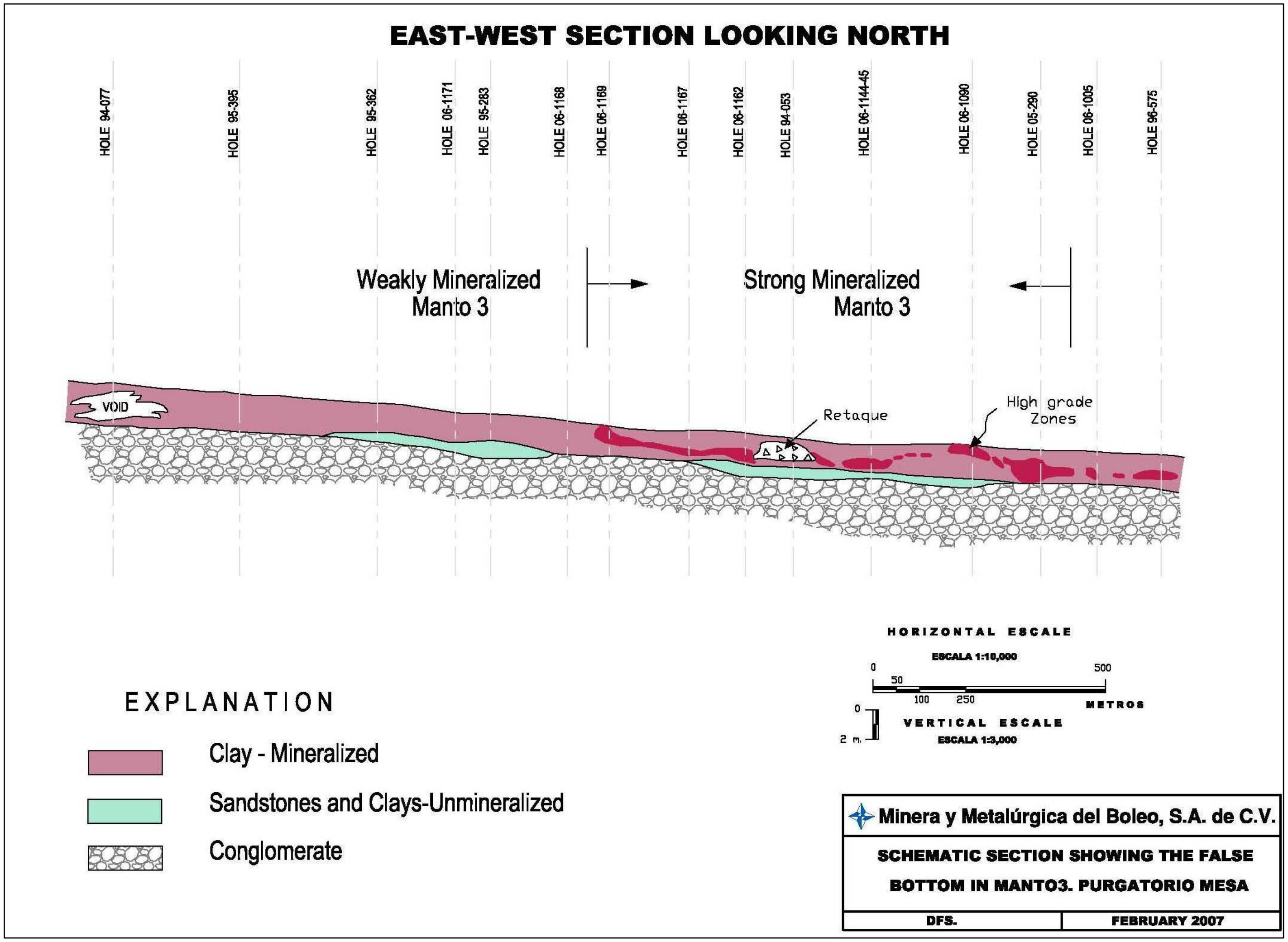


Figure 9: Manto 3 Location of False Bottoms



Both, vertical grade distribution in the metals, as well as the position of old stopes provide additional evidence the well-mineralized horizon occurs at the top of the manto. The example from three diamond holes (Table 7) demonstrates vertical change in grade between the claystone false bottom and the well-mineralized top of Manto 3.

Table 7: 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.

10 EXPLORATION

Since acquiring control of the Boleo Project in 2001, and prior to December 2004, MMB has concentrated on carrying out a complete geological, mining, and processing review of the Boleo Property in place of a straight field exploration work. This includes an independent assessment by Hellman and Schofield Pty Ltd. (H&S) of the open pit copper-cobalt-zinc manganese resources. Bateman Engineering Pty Ltd. completed an in-depth review of alternate flow sheets for the processing of ore from the Boleo Property and Agapito Associates Ltd. and Australian Mine Design and Development Ltd. carried out investigations 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 Boleo Property.

The latest drilling activities commenced on the property in December 2004 and are discussed further in Section 11.

11 DRILLING

11.1 GENERAL

The current resource study uses all diamond drill holes (DDHs) completed by previous owners of the Boleo Property, plus the additional holes of the four infill drill programs carried out by MMB between December 2004 and March 2007 (DDHs 04-928 to 07-1233).

11.2 HISTORICAL

The oldest recorded drilling program was carried out between 1927 and 1940 when 10,237 m 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 never kept or have been lost or destroyed.

The most extensive drilling program ever conducted in the Boleo District was that of International Curator Resources Ltd. between 1993 and 1998. 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 Rosalía formations) was expected, the upper portion of the hole was triconed before coring commenced near the target horizon. A total of 969 holes for 73,473 m were completed by International Curator.

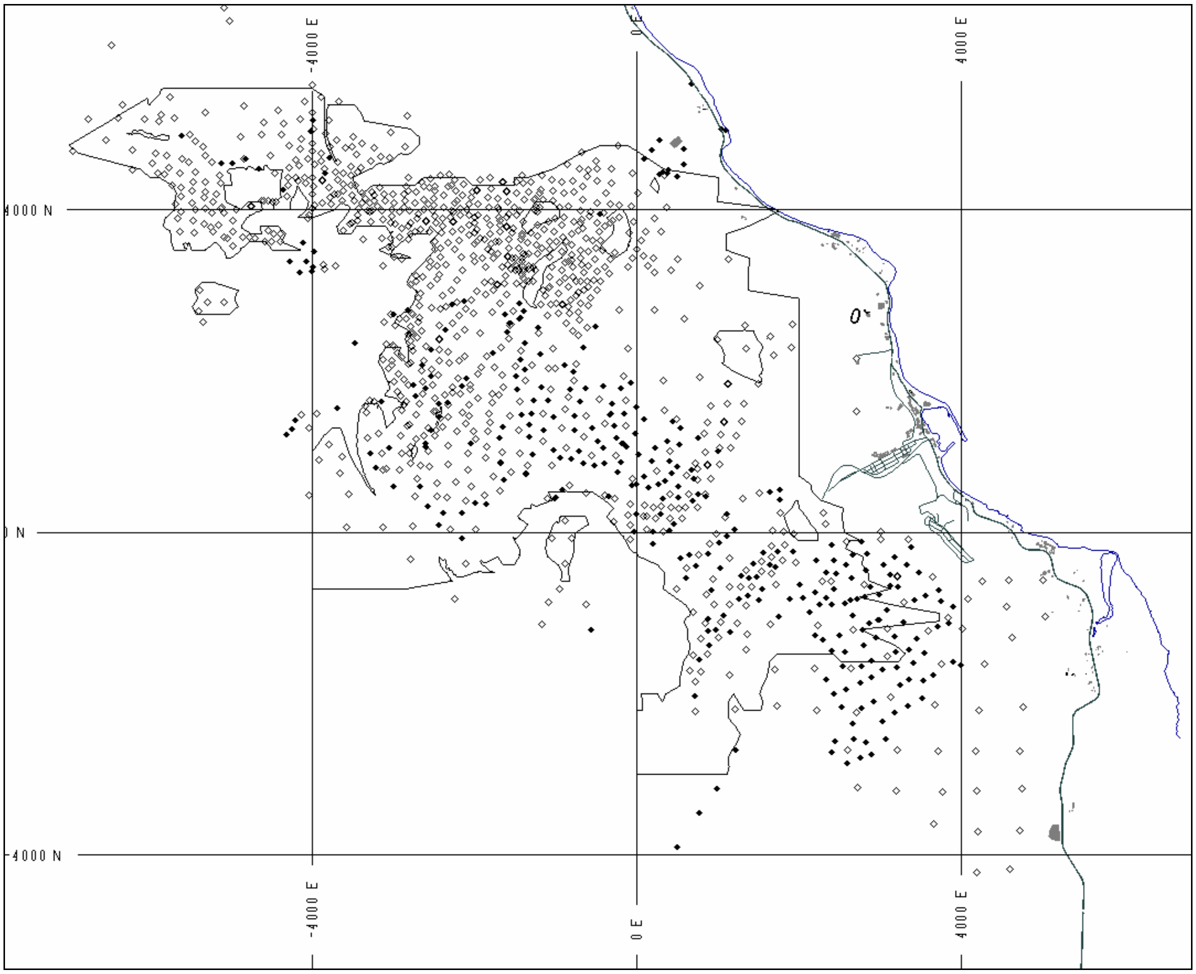
The need to increase the proportion of the Boleo resource classified as measured and indicated led to the execution of four drill programs between December 2004 and March 2007.

11.3 2004 TO 2007

Four infill drill programs totalling 310 DDH's and 46,617 m were carried out by MMB on the property between December 2004 and March 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 10.

Appendix II.3.1 contains a full listing of all new drill hole locations and the results from diamond drill holes DDH 928 to 1233. Results are reported for the logged manto intersections only from each hole and are quoted as single composite intervals for each manto.

Figure 10: Drill Hole Locations – Historical and Infill



Note: Pre-2004 drill holes are shown as open symbols. MMB Drill holes shown as black solid symbols (DDH 928 to DDH 1233).

12 SAMPLING METHODS & 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 12,666 samples have been collected from the mineralized manto units and the mean sample interval was 0.95 m.

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

- Core was transported from the drill site by either helpers or geologists to company warehouses in Santa Rosalía, where the boxes were labelled and core recoveries calculated.
- A company geologist, who simultaneously marked out all sample intervals, then logged the core.
- A trained local helper split core with a mechanical splitter (or a knife in poorly consolidated material).
- 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 cm 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 Figure 11.

Figure 11: Mechanical Splitter (H&S Modified Design)

12.1 CORE RECOVERY

Core recovery from the historical drill holes was measured for each retrieved core run. Measured recoveries range from 82% to 90%, with a mean of 83% (Table 8). Where recovery was less than 100% no attempt was made to specifically identify location of lost core within the recovered interval.

Table 8: 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**

Table 9 shows the core recovery for the current drill programs (DDH928-1233) shows better overall values with a mean recovery of 94.4%.

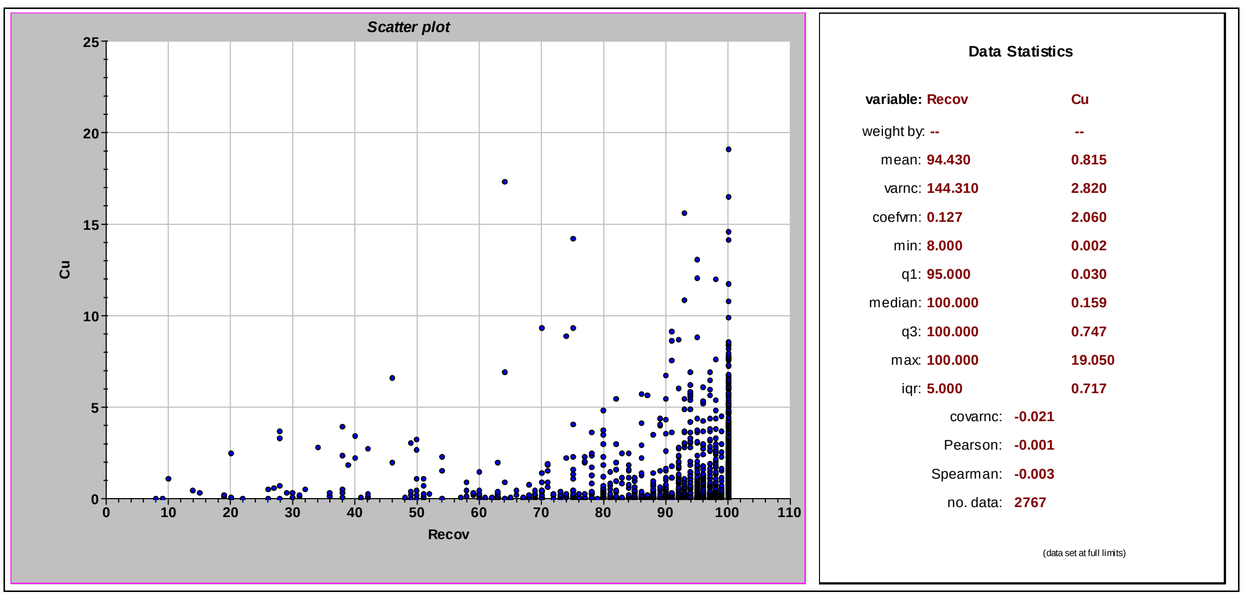
Table 9: Diamond Core Recovery – Current Programs (DDH928-1233)

Manto	Intervals	Recovery %
0	208	95.4
1	400	95.0
2	680	95.6
3	780	92.9
3a	277	94.9
3aa	22	89.5
4	396	94.3
Total	**2,767**	**94.4**

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 copper or cobalt grade.

For the current drill holes, recoveries have been determined for each sample interval. Figure 12 shows core recovery against copper grade. There is no correlation between recovery and grade, although lower recoveries tend to be lower grade suggesting possible loss of friable mineralization.

Figure 12: Core Recovery vs. Copper Grade – Current Drill Programs



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 2,110 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 (993 determinations) and non-mineralized rocks (1,117 determinations).

Densities were determined on un-dried 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 110°C for a minimum of six hours. SGS-XRAL Laboratories in Hermosillo calculated the water content, as the weight loss on drying and as a percentage of the original sample weight.

12.2.1 *DIMENSIONAL (OR CALLIPER) METHOD*

The core sample was cut with a saw perpendicular to its axis; samples were nominally 30 cm 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 t/m^3 has been used for manto material, based on an average calculated from 993 wet density measurements and the corresponding analysed water contents of the samples (Table 10).

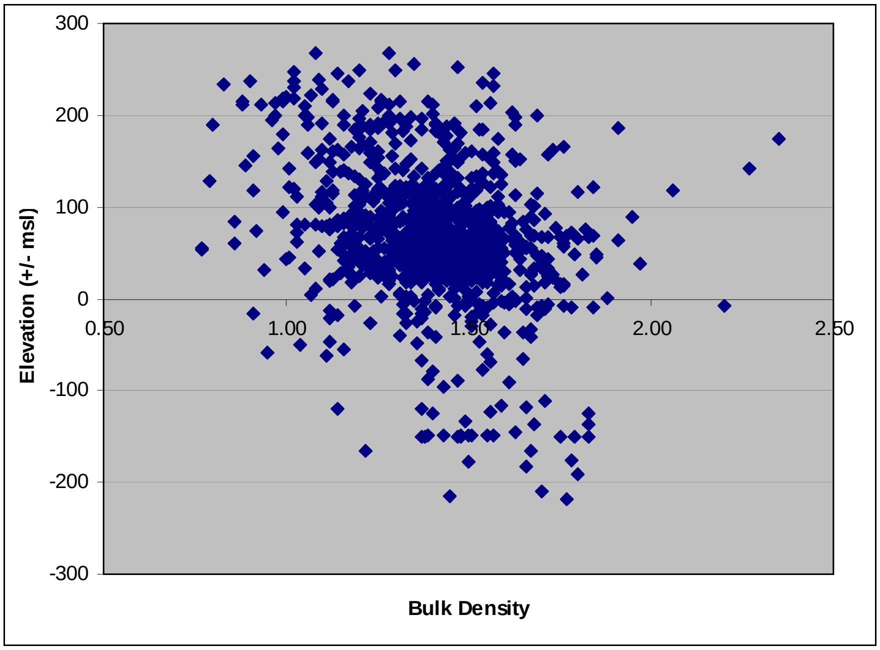
Although there is some indication of increasing bulk density with sample depth, a graph of bulk density vs. depth (

Figure 12) shows only a weak correlation.

Table 10: 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
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	1,052	25.93	1.41	993

Figure 12: Bulk Density vs. Sample Depth



13 SAMPLE PREPARATION, ANALYSIS & 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 pulverized. Analysis for copper, cobalt, zinc, iron, and manganese 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 Rosalía 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 cobalt by SGS-XRAL. The extent of the analytical problems and remedial measures taken are discussed in more detail in Section 14.1.

All samples were stored, prior to shipping, in one of three locked warehouses in Santa Rosalía. 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 POST 2004 DRILL HOLES

The samples for assaying are placed into 30 kg sacks and kept in locked premises until they are transported to the laboratory. A company vehicle is used to transport the samples from Santa Rosalía 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° 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 pulverized (Labtech LM2)> 85% < 75 µm.

After sample preparation, pulps are transported for assaying from Hermosillo to ALS Chemex in North Vancouver, British Columbia, Canada.

The assay method currently used for copper, cobalt, zinc and manganese is ME-ICP61a (four acid "near-total" digestion).

Sample Decomposition: HNO3-HClO4-HF-HCl digestion (ASY-4A02)

Analytical Method: Inductively Coupled Plasma – Atomic Emission Spectroscopy (ICP – AES)

The sample is digested in a mixture of nitric, hydrochloric and hydrofluoric acids. Perchloric acid is added to assist oxidation of the sample and to reduce the possibility of mechanical loss of sample as the solution is evaporated to moist salts. Elements are determined by inductively coupled plasma – atomic emission spectroscopy (ICP-AES).

Samples reporting high grades (>100,000 ppm) for copper, cobalt, zinc or manganese are re-assayed for those elements by method AA62.

Sample Decomposition: HNO_3-$HClO_4$-HF-HCl digestion (ASY-4ACID)

Analytical Method: Atomic Absorption Spectroscopy (AAS)

A prepared sample (0.4) g is digested with nitric, perchloric, and hydrofluoric acids, and then evaporated to dryness. Hydrochloric acid is added for further digestion, and the sample is again taken to dryness. The residue is dissolved in nitric and hydrochloric acids and transferred to a volumetric flask (100 or 250) mL. The resulting solution is diluted to volume with de-mineralized water, mixed, and then analyzed by atomic absorption spectrometry against matrix-matched standards.

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 Pre-2204 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 pre-1997 assay database at Boleo.

Consequently, International Curator Resources Ltd. (Curator) made the decision to review the entire assaying process used for the Boleo 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 below.

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

- Despite repeated assurances from SGS that the perchloric acid digest would give a total metal extraction, data from check assaying indicated otherwise.
- Above 4%, copper 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; however, 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 copper 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 Boleo material so that the matrix of the standards matched that of the samples.

Initially two standards (Interim Standards) were prepared from Boleo material (Boleo I, II). These were prepared at the SGS laboratory. Sub-samples were dispatched to seven laboratories for round-robin analysis. The material was subjected to a four-acid digest (nitric, hydrofluoric, 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 (Boleo III, IV, V) were prepared at the Colorado Minerals Research Institute and CDN Resource Laboratories, Burnaby, BC, Canada. Composites of varying grade were combined to form three single 25 kg samples. Samples were again dispatched to several laboratories in the USA, Canada, and to SGS in Mexico for round-robin analysis. The same four-acid digest method was used. The recommended grade and limits for each standard (Peatfield, 1997) are shown in Table 11.

Table 11: Boleo Assay Standards

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

Analytical Method & 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 Rosalía. 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 Chemex inserted an additional 800 laboratory samples.

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 (±2sd). 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 copper and cobalt.

Second Laboratory Check Assaying

A total of 441 duplicate pulps were sent to a second laboratory (Mineral Environments Laboratories, “Min-En”) for check assays. The results were typically similar; however, there was 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, which lead to the 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 copper, 5% for cobalt and 2% for zinc. No explanation was given for this difference.

Table 12 summarises the results of the ‘A’ and ‘B’ core duplicate sampling program.

Subsequently a further 55 core duplicate samples were assayed (Table 13). 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 12: Boleo Core Duplicate Results – 1st Program

Core Duplicates	Range (%)	Avg. Precision	Original Assay Avg.	Dupl. Core Assay Avg.	% 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

Table 13: Boleo Core Duplicate Results – 2nd Program

Duplicate Type	Range (%)	Avg. Precision	Original Assay Avg.	Dupl. Assay Avg.	% 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

One reason for the improved result in the second program may be that the duplicates and original samples were assayed in the same batch, while 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 affect the entire program. The average grades of both the coarse residue and pulp duplicates are almost identical to the original samples.

Conclusions Derived from the Pre-2004 Re-Assay Program

Peatfield & Smee arrived to the following conclusions:

- 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 1st and 2nd core duplicate sampling program may be due to the fact that in the 2nd 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 copper, cobalt, zinc, and manganese assays in the Boleo database.

14.1.3 Comments on Assay Quality Control Results

Because of the problems identified in the original Boleo 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 copper 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 seven participating laboratories).

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 13 shows graphically the copper data reported by each laboratory used for Boleo V, while Figure 14 shows the data used by H&S. A similar exercise was completed copper and cobalt data for Boleo III and IV the results are compared in Table 14. 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%.

Figure 13: Boleo V Cu, Round-Robin Cu Assays used by Peatfield & Smee

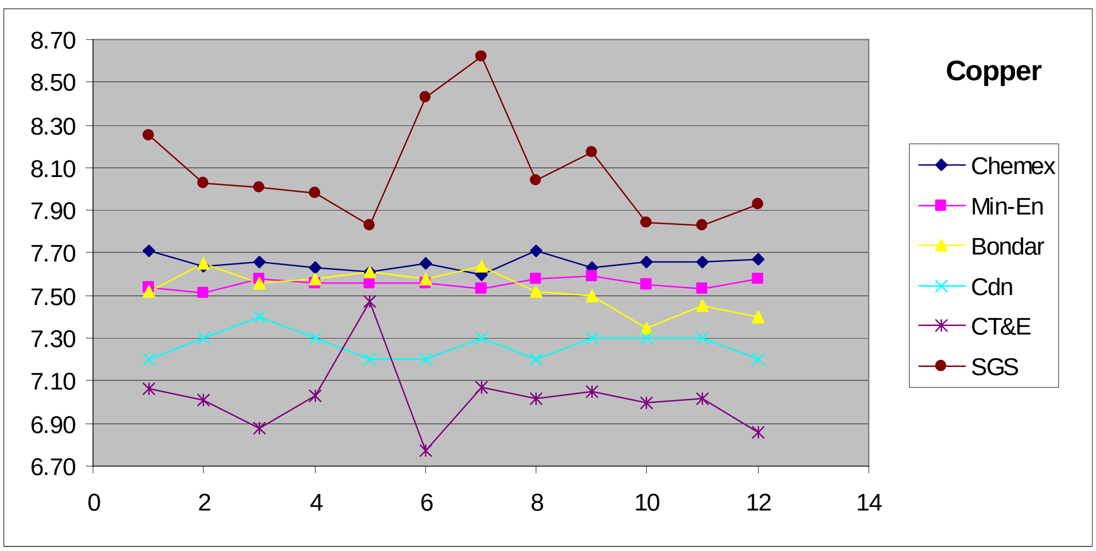


Figure 14: Boleo V Cu, Round-Robin Cu Assays used by H&S

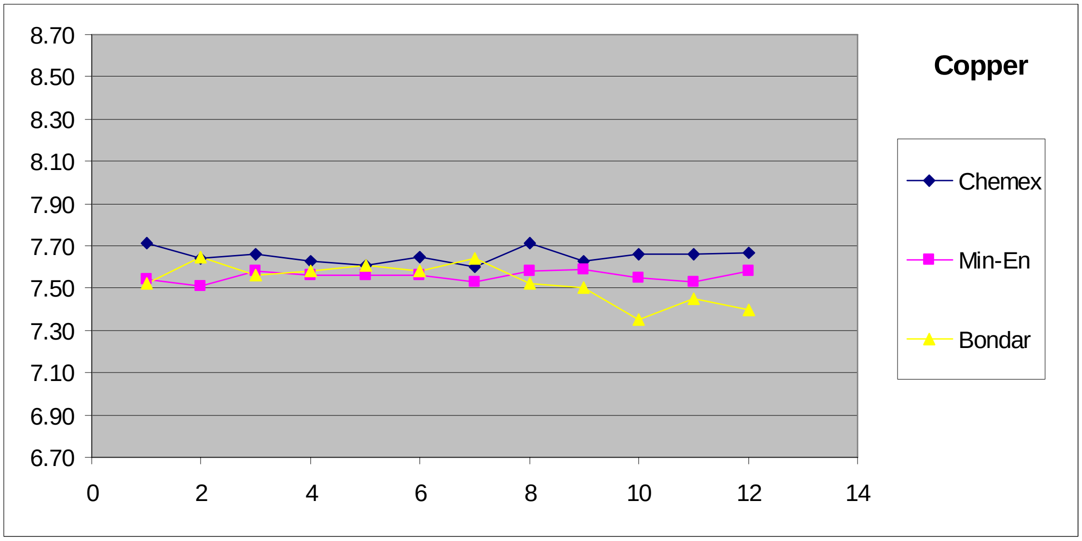


Table 14: Boleo Formal Standards – Revised Values

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

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 located in 10 Microsoft Excel spreadsheets (a separate file for each hundred holes). Two holes were checked from each file (Table 15 shows nine of the drill holes audited), with copper, cobalt, and zinc assays checked. Geological logs and summary logs contained in the same Excel files were not audited.

Table 15: Drill Holes Audited for Data Entry Errors

Hole ID	Assay Job No.	Hole ID	Assay Job No.
022	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

Pre-2004 drill hole collars at Boleo 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. Fifty pre-2004 drill holes were checked using high precision GPS to verify the collar coordinates. Average results of this survey show easting, northing and elevation to be within 0.4, 1.1 and 1.2 m respectively of the orthophoto coordinate.

The drill hole spacing, at its closest, is in the order of 150 m x 150 m so ~1 m 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 ±1.2 m that was obtained for this dimension is 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 Boleo are vertical, which is appropriate. No down hole surveys were conducted.

14.2 THE MMB DRILLING PROGRAM

Assay standards, duplicate sampling, blank samples and check assaying have been used during the 2004-07 MMB initiated drilling programs to validate the assay results reported by ALS.

14.2.1 ASSAY STANDARDS

The three Boleo standards (Boleo III, IV, V) discussed above have been used again during the current assaying program (see Figure 15 to Figure 17). The reported results are combined and presented as a percentage difference from the revised accepted values of each standard (see Section 3.13.3).

There is a persistent under-stating of standard grades by between 5% and 11% for the 2006/2007 drill programs. The reason for this is not at present well understood but the issue is currently under investigation and review by MMB staff. Initial thoughts were that the sample preparation laboratory failed to adequately dry the standard samples thereby overstating the true weight of the sample aliquot taken for analysis. A small batch of 18 standard samples was sent to the laboratory for check assay, with specific instructions to completely dry the pulps. The samples returned similarly low values ranging from -6% to -9%. A larger number of standard pulps (90) were sent to ALS and two other laboratories, ACME and Global Discovery, in an attempt to establish whether or not the recommended values are still applicable or whether the material has degraded over time.

The results are very similar to the routine standard results reported by ALS. All laboratories consistently under reported copper (see Figure 18 to Figure 20). It therefore appears that there is a problem with the Boleo assay standards.

Comparison between check assays completed at ACTLabs Skyline is very good (see Section 14.2.3) which supports the interpretation that the poor results of the standards does not indicate that the routine sample data reported by ALS are under stated for the three commodity elements.

Figure 15: Boleo Assay Standards – Cu

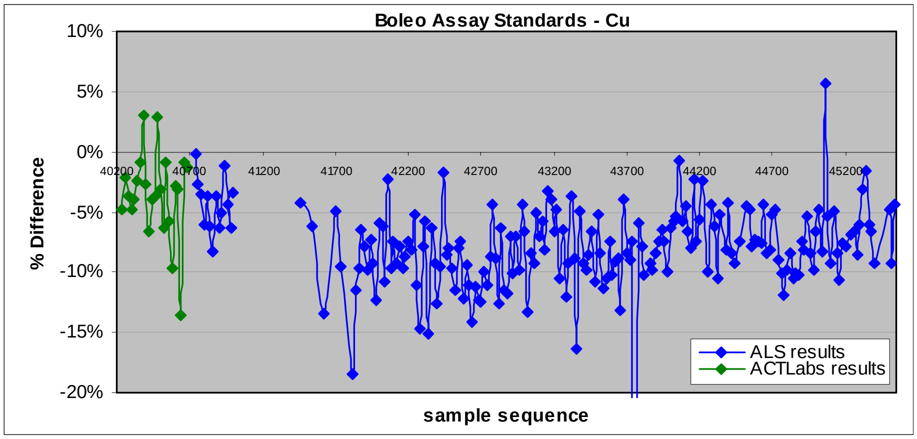


Figure 16: Boleo Assay Standards – Co

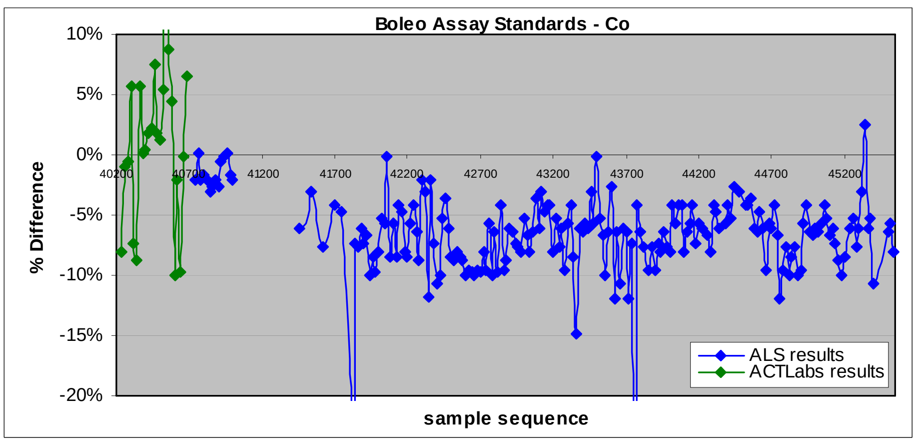


Figure 17: Boleo Assay Standards – Zn

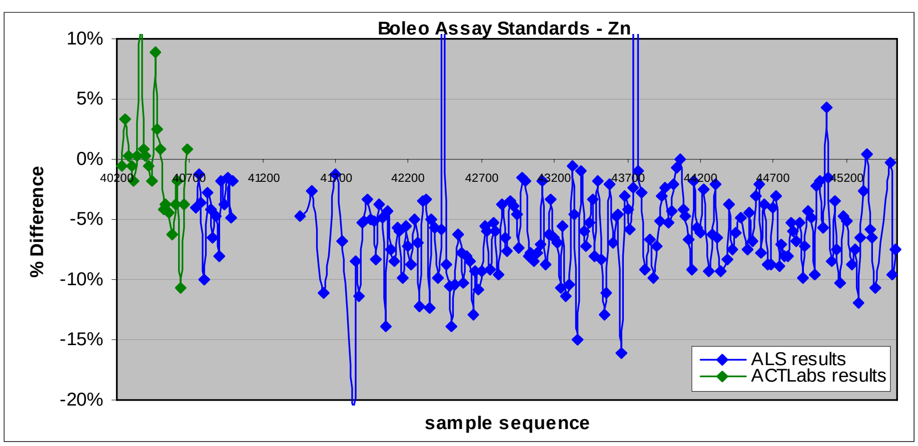


Figure 18: Check Assay Boleo III – Cu

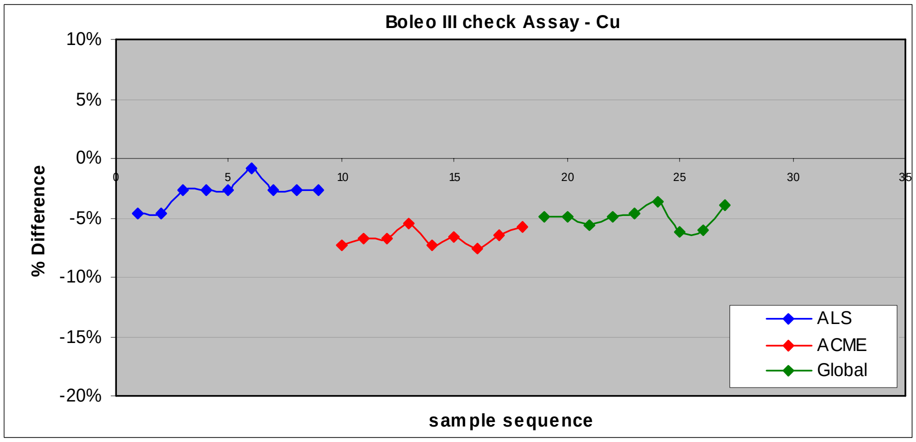


Figure 19: Check Assay Boleo IV – Cu

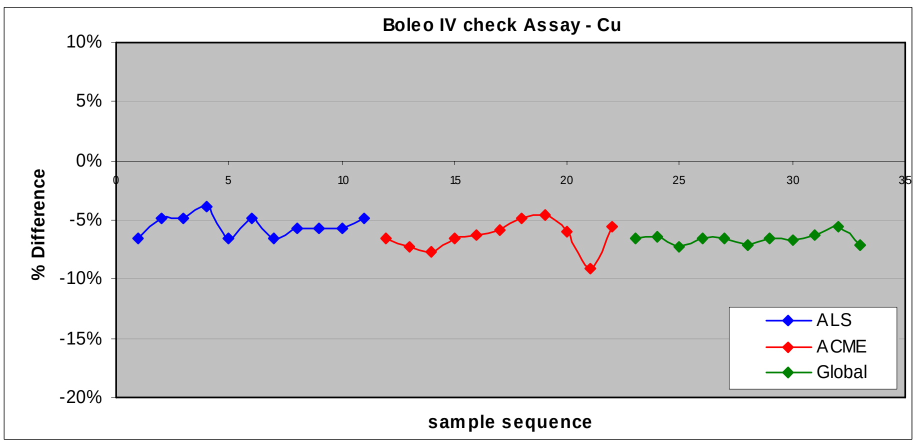
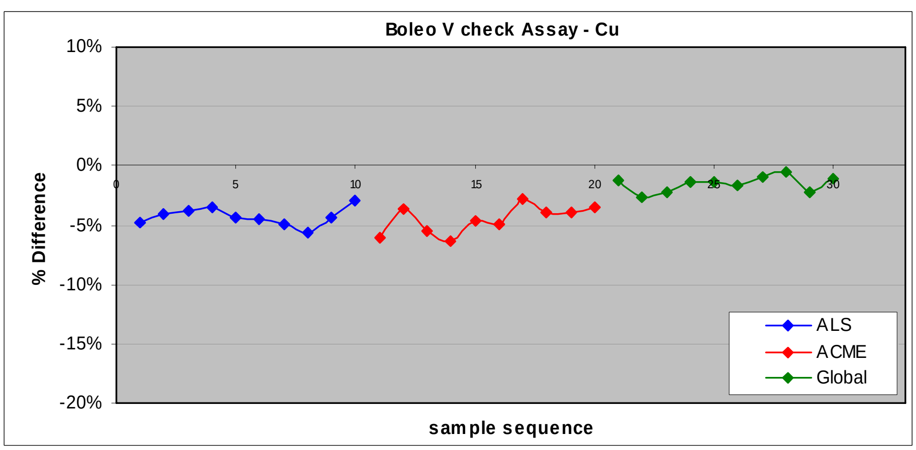


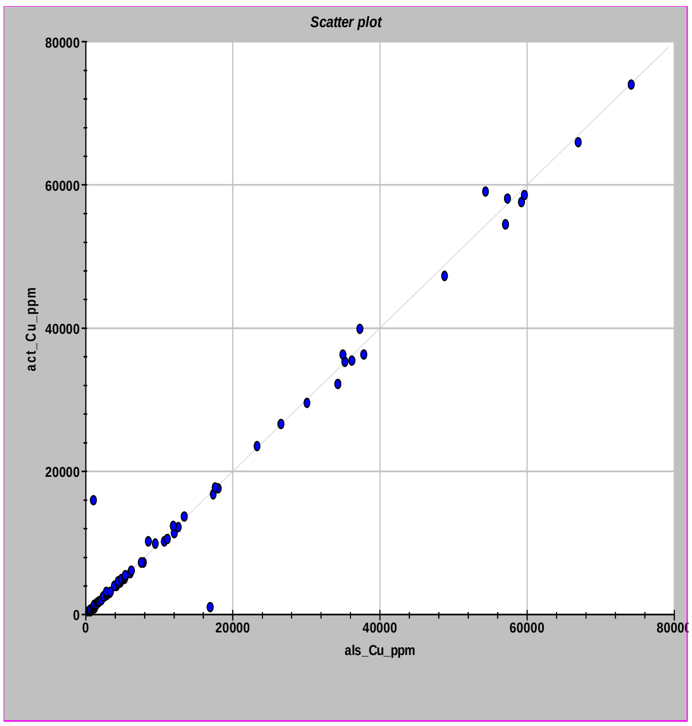
Figure 20: Check Assay Boleo V – Cu



14.2.2 CHECK ASSAYS

Check assays from ACTLabs in Tuscon; however, show very good correlation to the ALS-Chemex results (see **Error! Reference source not found.** to **Error! Reference source not found.**). These results can also be used to make some comments regarding the problem with the assay standards reported above. It would appear unlikely that both laboratories produced similarly poor results, to the extent that the inter laboratory correlation was as good as reported.

Figure 21: Boleo Check Assay Results – Cu

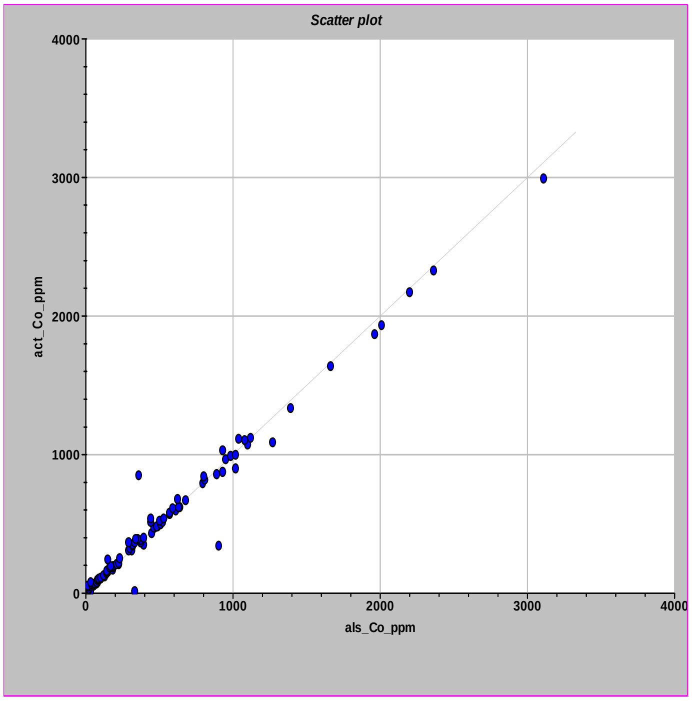


Data Statistics

variable:	als_Cu_ppm	act_Cu_ppm
weight by:	--	--
mean:	9157.606	9090.599
varnc:	271510656.000	269567680.000
coefvrn:	1.799	1.806
min:	30.000	-100.000
q1:	280.000	300.000
median:	1220.000	1200.000
q3:	7500.000	7200.000
max:	74100.000	74000.000
iqr:	7220.000	6900.000
covarnc:	268270880.000	
Pearson:	0.992	
Spearman:	0.980	
no. data:	117	

(data set at full limits)

Figure 22: Boleo Check Assay Results – Co



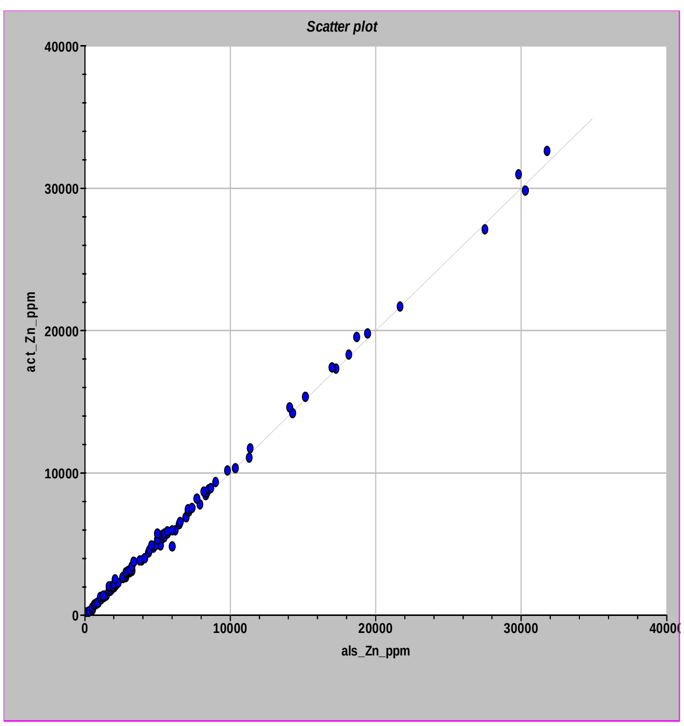
Data Statistics

variable:	als_Co_ppm	act_Co_ppm
weight by:	--	--
mean:	445.897	445.299
varnc:	286494.281	269513.813
coefvrn:	1.200	1.166
min:	10.000	10.000
q1:	80.000	80.000
median:	230.000	240.000
q3:	570.000	570.000
max:	3110.000	2990.000
iqr:	490.000	490.000

covarnc:	274524.313
Pearson:	0.988
Spearman:	0.970
no. data:	117

(data set at full limits)

Figure 23: Boleo Check Assay Results – Zn



Data Statistics

variable:	als_Zn_ppm	act_Zn_ppm
weight by:	--	--
mean:	5762.832	5842.478
varnc:	42157400.000	43144748.000
coefvrn:	1.127	1.124
min:	170.000	100.000
q1:	1690.000	1700.000
median:	3210.000	3200.000
q3:	6980.000	6900.000
max:	31800.000	32600.000
iqr:	5290.000	5200.000

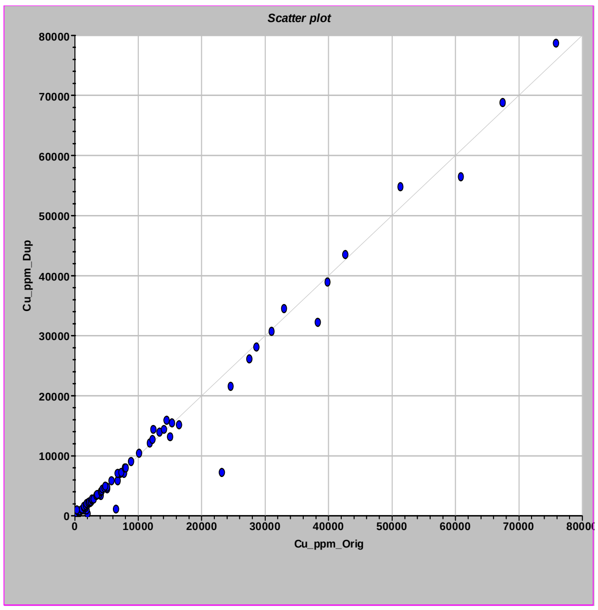
covarnc:	42617692.000
Pearson:	0.999
Spearman:	0.997
no. data:	113 / 117

(data is sub-setted)

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 (see Figure 24 to Figure 26).

Figure 24: Boleo Duplicate Assay Results – Cu

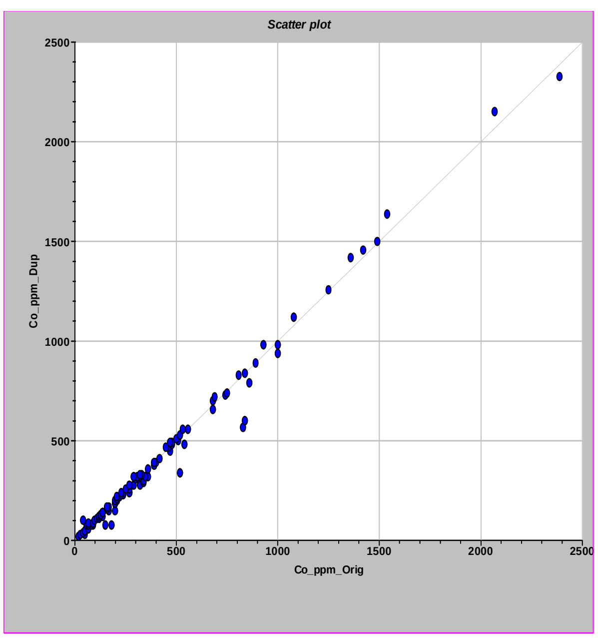


Data Statistics

variable:	Cu_ppm_Orig	Cu_ppm_Dup
weight by:	--	--
mean:	8123.238	7803.714
varnc:	210480592.000	209644752.000
coefvn:	1.786	1.855
min:	50.000	40.000
q1:	330.000	260.000
median:	2010.000	1750.000
q3:	7380.000	7170.000
max:	75900.000	78800.000
iqr:	7050.000	6910.000
covarnc:	208226304.000	
Pearson:	0.991	
Spearman:	0.981	
no. data:	105 / 106	

(data is sub-setted)

Figure 25: Boleo Duplicate Assay Results – Co

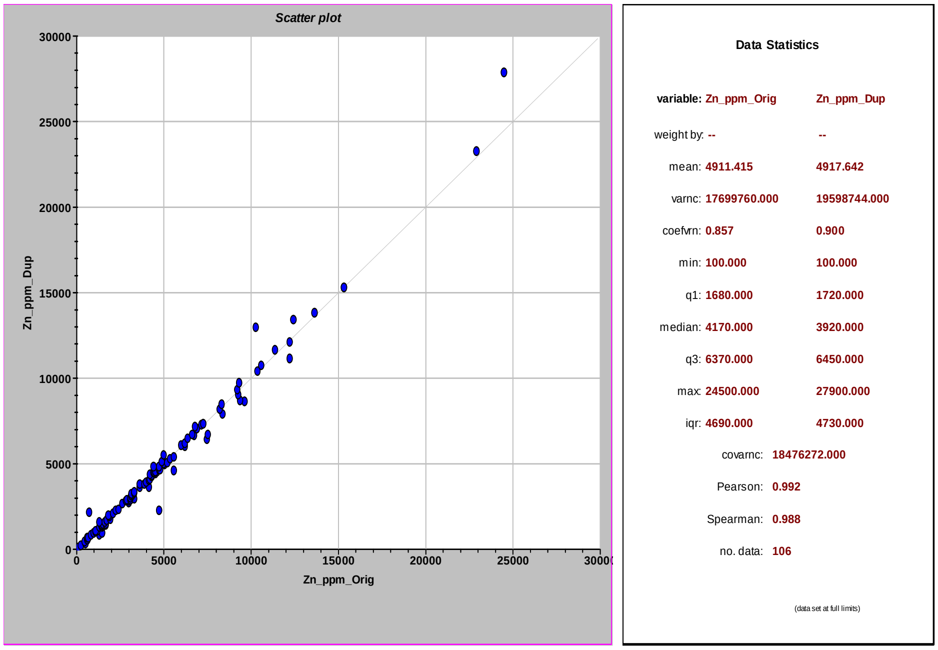


Data Statistics

variable:	Co_ppm_Orig	Co_ppm_Dup
weight by:	--	--
mean:	422.925	414.528
varnc:	176758.422	180143.641
coefvn:	0.994	1.024
min:	20.000	20.000
q1:	140.000	140.000
median:	270.000	280.000
q3:	510.000	490.000
max:	2390.000	2330.000
iqr:	370.000	350.000
covarnc:	177353.734	
Pearson:	0.994	
Spearman:	0.989	
no. data:	106	

(data set at full limits)

Figure 26: Boleo Duplicate Assay Results – Zn



14.2.4 BLANK SAMPLE ASSAYS

Blank samples, derived from the unmineralized Gloria Formation, were inserted into the sample stream at regular intervals. These samples are designed to test for cross-contamination in the laboratory.

Figure 27 to Figure 29 shows the results of consistent low levels of Cu, Co and Zn, with occasional elevated values. The increased spread of Cu values (on the right hand side of the graph) may be indicative of less thorough laboratory hygiene between samples at a time when reduced sample turn around time was required. There is however, no corresponding elevation in either Co or Zn.

Figure 27: Boleo Blank Assay Results – Cu

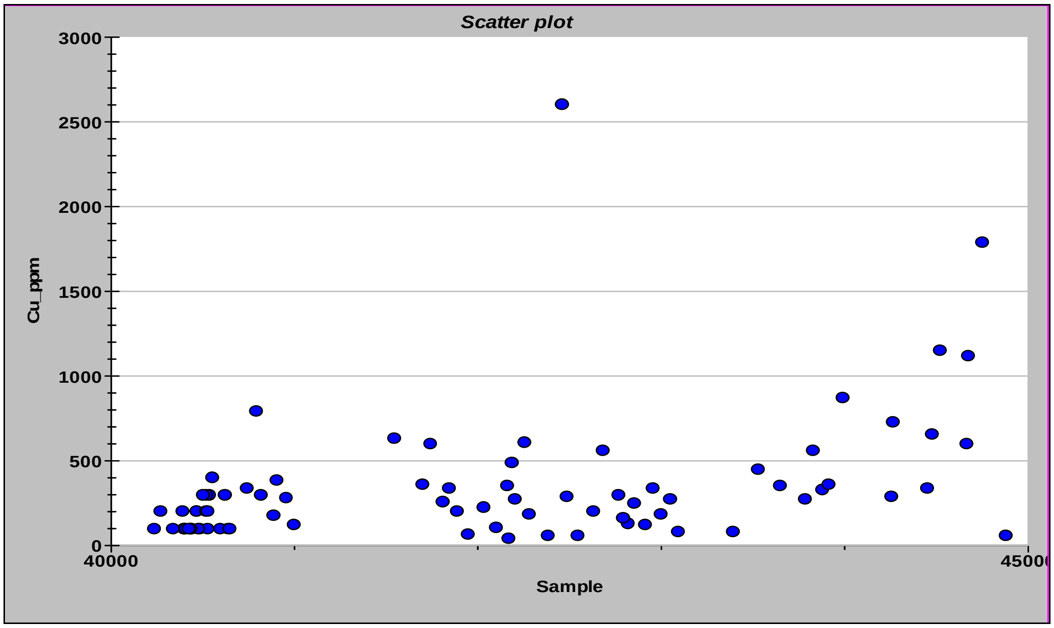


Figure 28: Boleo Blank Assay Results – Co

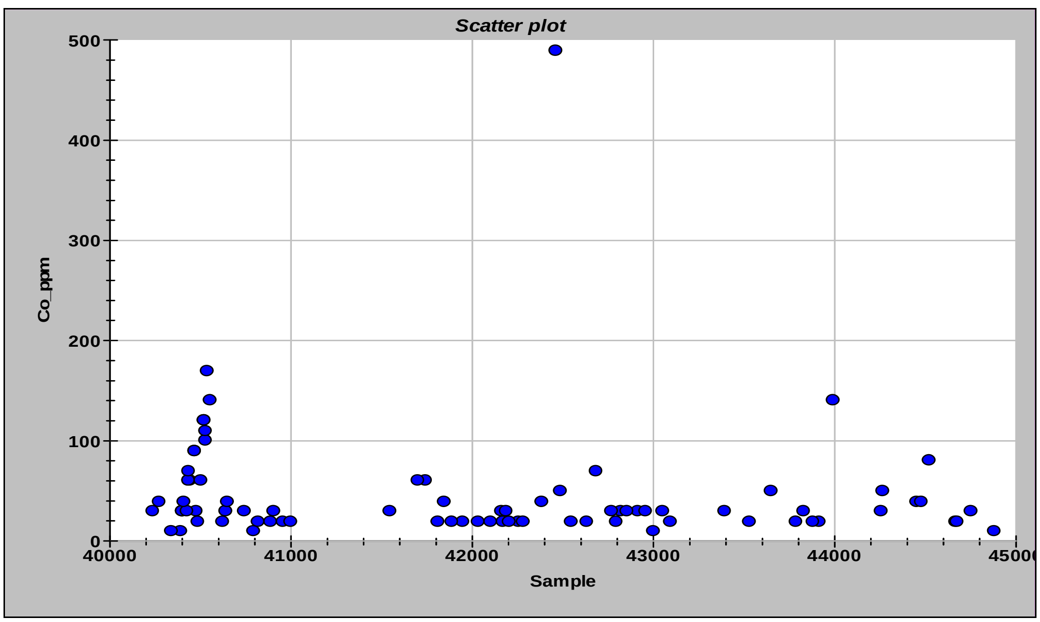
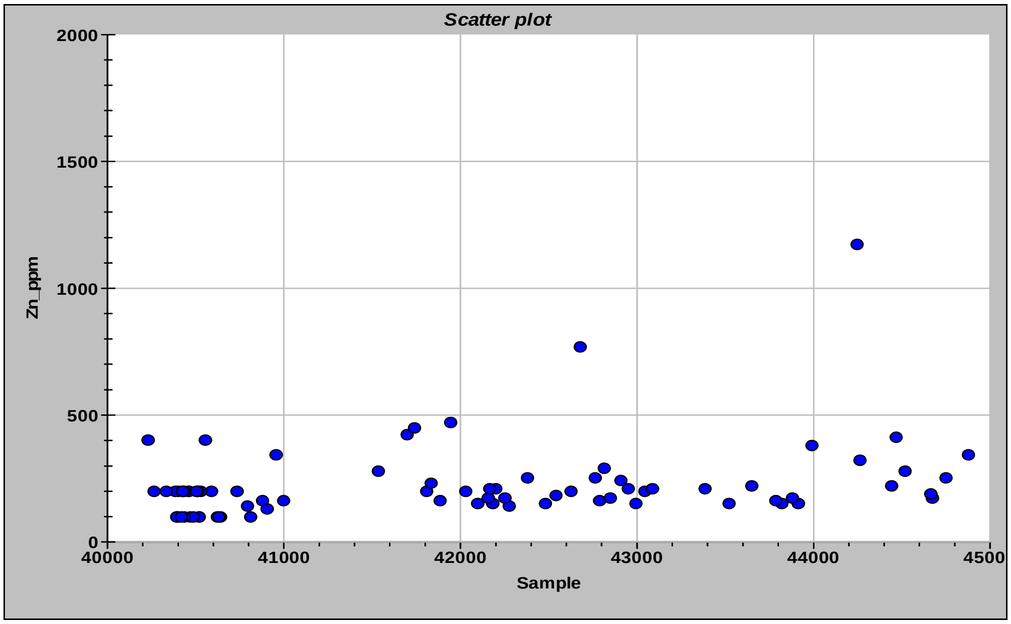


Figure 29: Boleo Blank Assay Results – Zn



14.3 PROJECT DATABASE

14.3.1 DATA

A new Microsoft Access database was customized for the Boleo Project. This database was set up to accommodate the historical data and the new data from the current programs.

Historical data exists in 10 Excel spreadsheets, a single spreadsheet 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. The relevant geologist then translates these logs 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*: Hole 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 high precision GPS. Twenty-seven drill holes were re-surveyed by high precision GPS to verify the original collar coordinates. Easting and northing coordinates were within an average of ±10 mm of the original survey. Elevation variation fell within two groups, with half returning values averaging ±0.3 m of the original and the other half returning values averaging ±2.7 m of the original. The later group were surveyed using a different survey base station which resulted in the larger variation measured. A re-survey of these points will be carried out.

Based of the above results (and ignoring the elevation results from the questionable survey point) the tolerances obtained for all dimensions are not considered serious.

No down hole surveys were carried out.

15 MINERAL PROCESSING & METALLURGICAL TESTING

15.1 METALLURGY

15.1.1 BACKGROUND

Treatment strategies for the Boleo 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.

In parallel with the adoption of selective mining of higher grade ore (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. These changes in processing strategy result in a more robust, operable flowsheet with reduced operating and capital cost.

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

The metal recovery circuits are typical of those deployed in numerous operations worldwide. In developing the Boleo flowsheet, Bateman was able to incorporate the results of earlier testwork initiatives – some of which date back by 15 years; 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 major metallurgical testwork history of the past 3 years 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 testwork campaign was conducted at SGS Lakefield's facility in Ontario. Batch leach testing was carried out on six different ore samples from the Boleo 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 testwork 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 Boleo 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 testwork with clear overflows being produced.

This testwork 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

At the time of the above testwork the SX Group of the Parker Centre, CSIRO Minerals in Perth, Australia developed a process that allowed 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 the organic extractants results in an enhanced chemical effect (essentially a 'synergistic' shift in the pH 50 of separation) 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 conventional means of dealing with the high levels of manganese in solution and is both fundamental to and appropriate for metals recovery from Boleo solutions.

In preliminary testwork 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 a complete separation of copper and cobalt from manganese. The separation factors of zinc over manganese ranged in the hundreds, suggesting that a good separation of zinc from manganese can be obtained.

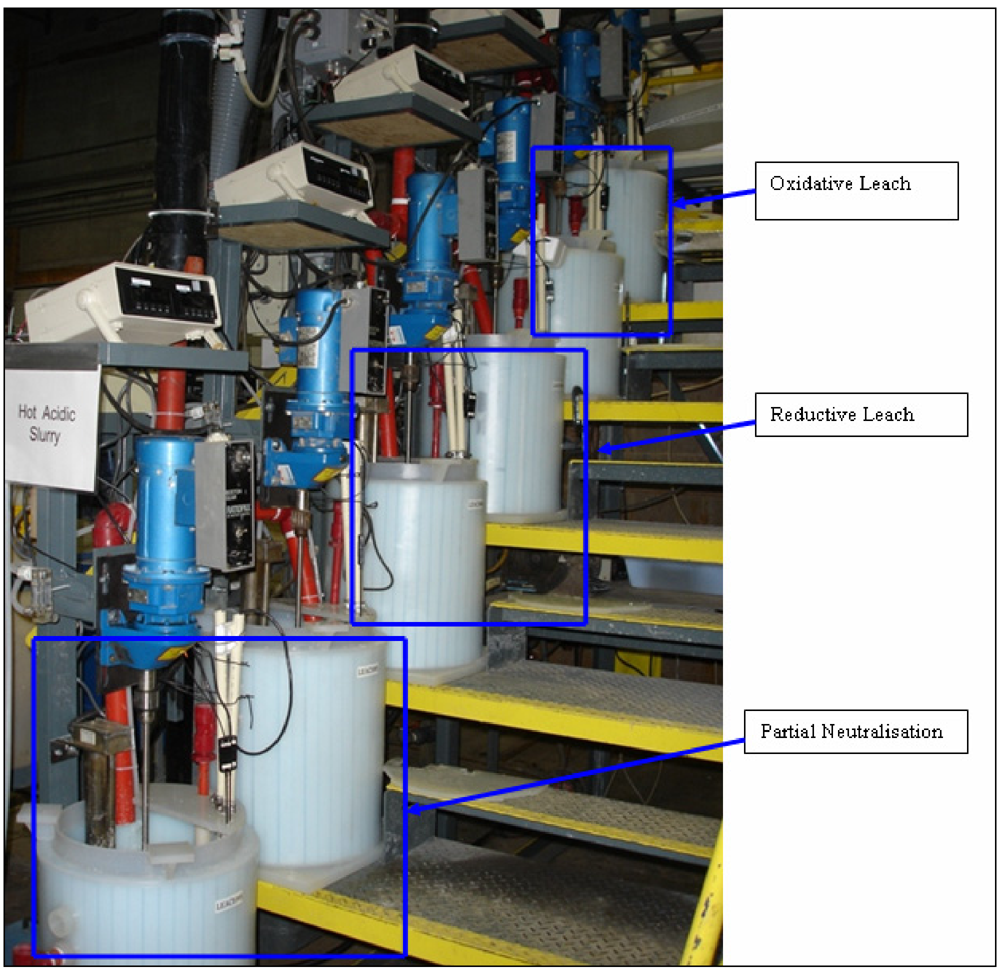
The success of this testwork promoted further investigation and after some additional bench scale testing the concept was included in the flowsheet for 'Proof of Concept' piloting.

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 Boleo oxide ore from manto 3 grading 1.6% Cu, 0.087% Co, 0.58% Zn, 3.23% Mn and 8.71% Fe. The pilot plant flowsheet comprised:

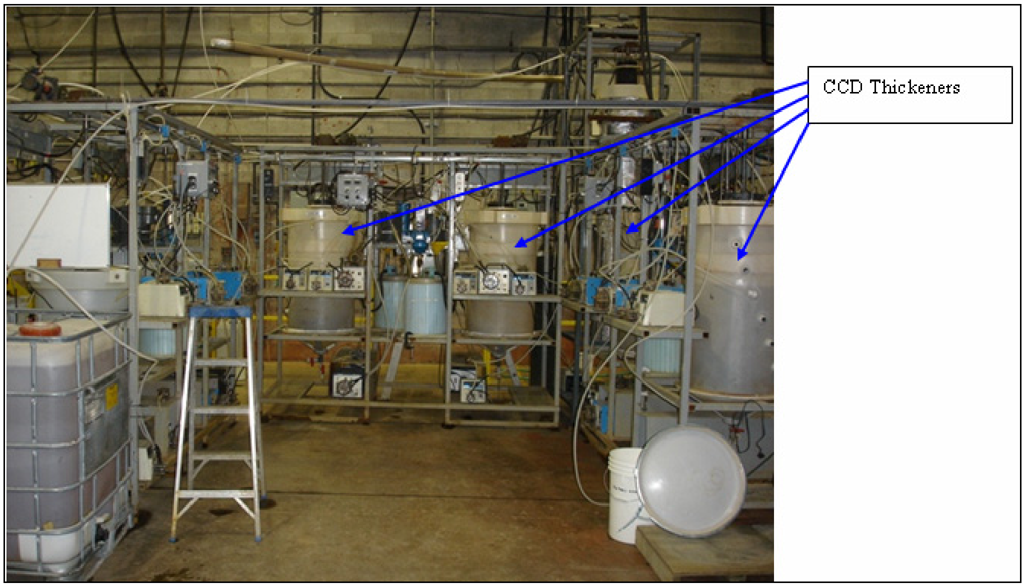
- attritioning of the ore with grinding of coarse ore particles to form an ore slurry
- acid oxidation leaching of the ore with sulphuric acid in seawater
- acid reduction leaching of the ore with sulphur dioxide and sulphuric acid in seawater
- partial neutralization with limestone

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

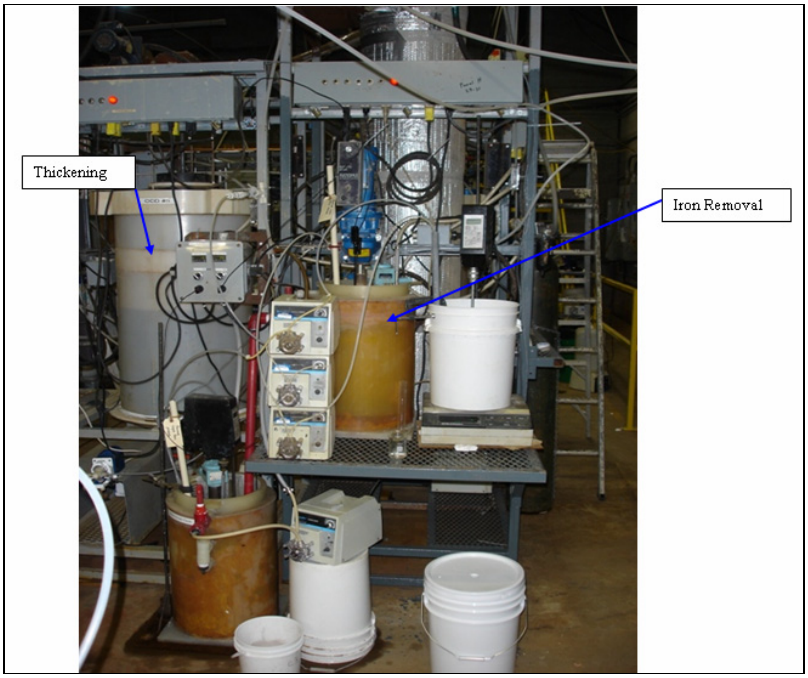
  The picture below shows four of the six stage CCD thickeners.



- Copper solvent extraction and electrowinning.

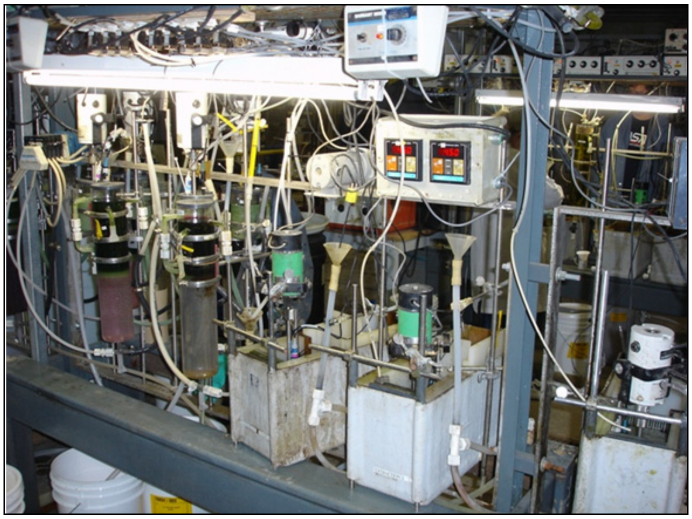
  The picture below shows the copper solvent extraction mixers/settlers to the left and the electrowinning process to the right, under a fume extraction hood. The cathodes can be clearly seen suspended in the electrowinning cell.

- Iron removal by pH adjustment and oxidation with air (and polish with hydrogen peroxide if required)
- Thickening of the iron residue prior to disposal



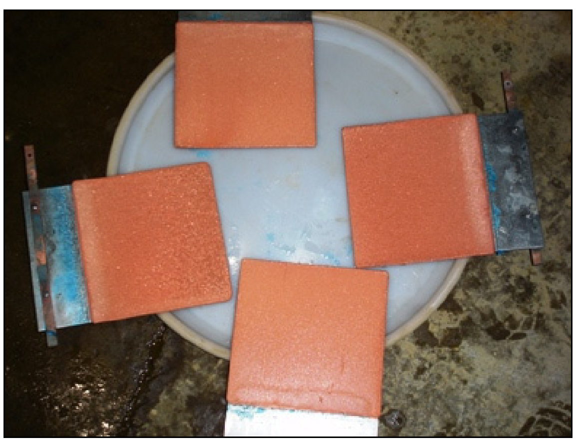
- Direct Solvent Extraction technology for selective recovery of cobalt and zinc and small amounts of residual copper. As mentioned above DSX ® technology is the property of Commonwealth Scientific Industrial Research Organization (CSIRO), Perth, Australia.

  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 mt 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 90%. Zinc extraction was generally above 70%. These numbers are indicative of the potential of the Boleo 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. 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 Boleo site containing limestone and other alkali minerals. HAC material is intended to be used as a low cost bulk neutralizing agent in the commercial Boleo 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 Boleo project forward. For the first time Boleo 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 optimize 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 two extractants and the operating pH.

Boleo limestone, available on the Boleo Reserve, was found to be effective in bulk tailings 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 an optimised mixture of the synergistic reagents
- Confirming key reagent consumptions, including the use of Boleo 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 Boleo 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 characterization of Boleo Pilot tailings
- SGS Lakefield – Production of purified cobalt carbonate
- SGS Lakefield – Cobalt removal from DSX zinc solution.

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, materials handling testwork.

Pilot Campaign Ore Feed

Approximately 10 tonnes of Boleo ore, composited from 143 channel samples of underground oxide Manto 3 ore from the test mining exercise conducted by AMDAD and Agapito in late 2005/early 2006, 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 16.

Table 16: Comparison of As Received, 200 kg Composite and Scrubber Feed Assays

	H_2O %	Cu %	Co g/t	Zn g/t	Mn %	Fe %
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
Boleo 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 17.

Table 17: Leach Feed Solids 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 Grade	2.04	1020	4640	4.14	18.5	7.71
Maximum Feed Ore Grade	2.27	1570	5270	5.99	21.2	8.59
Average Feed Ore Grade	2.18	1350	4910	5.01	19.7	8.26

Oxidative, Reductive & Partial Neutralization Leach Stages

Recovery of Cu, Co and Zn from Boleo 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 Boleo ore body.

A second stage reductive leach with sulphuric acid (pH 1.5) at 70°C to 80°C 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°C 80°C 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 aluminium, contributing to the overall acid consumption of the ore.

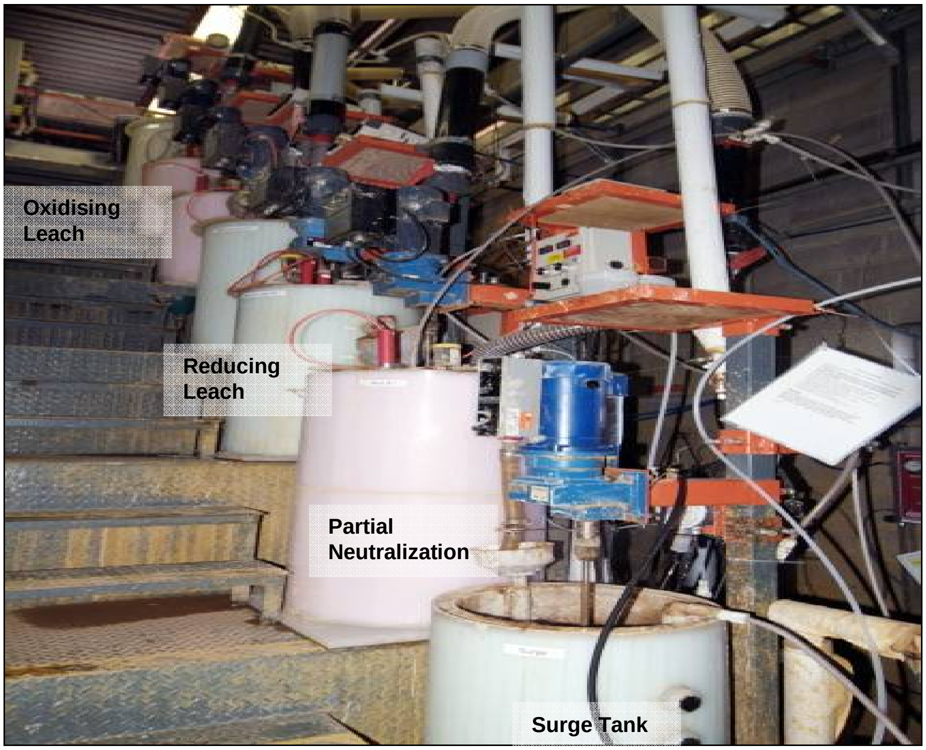


Photo 1 Overview of the Leach Circuit.

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.

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

Photo 2 Overview of the piloting Counter Current Decantation (CCD) equipment setup.

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 extractions, one wash, and two strip stages operated at 40°C.

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 wash 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.

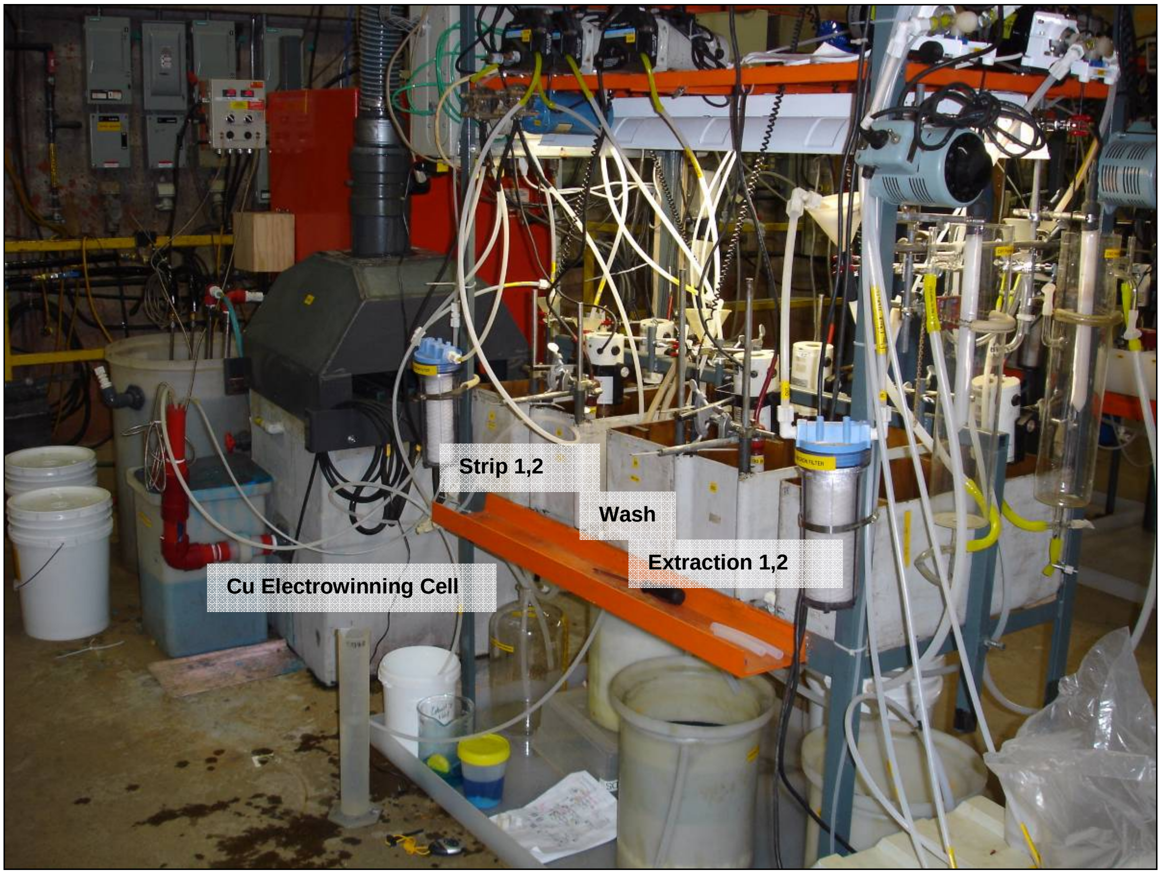
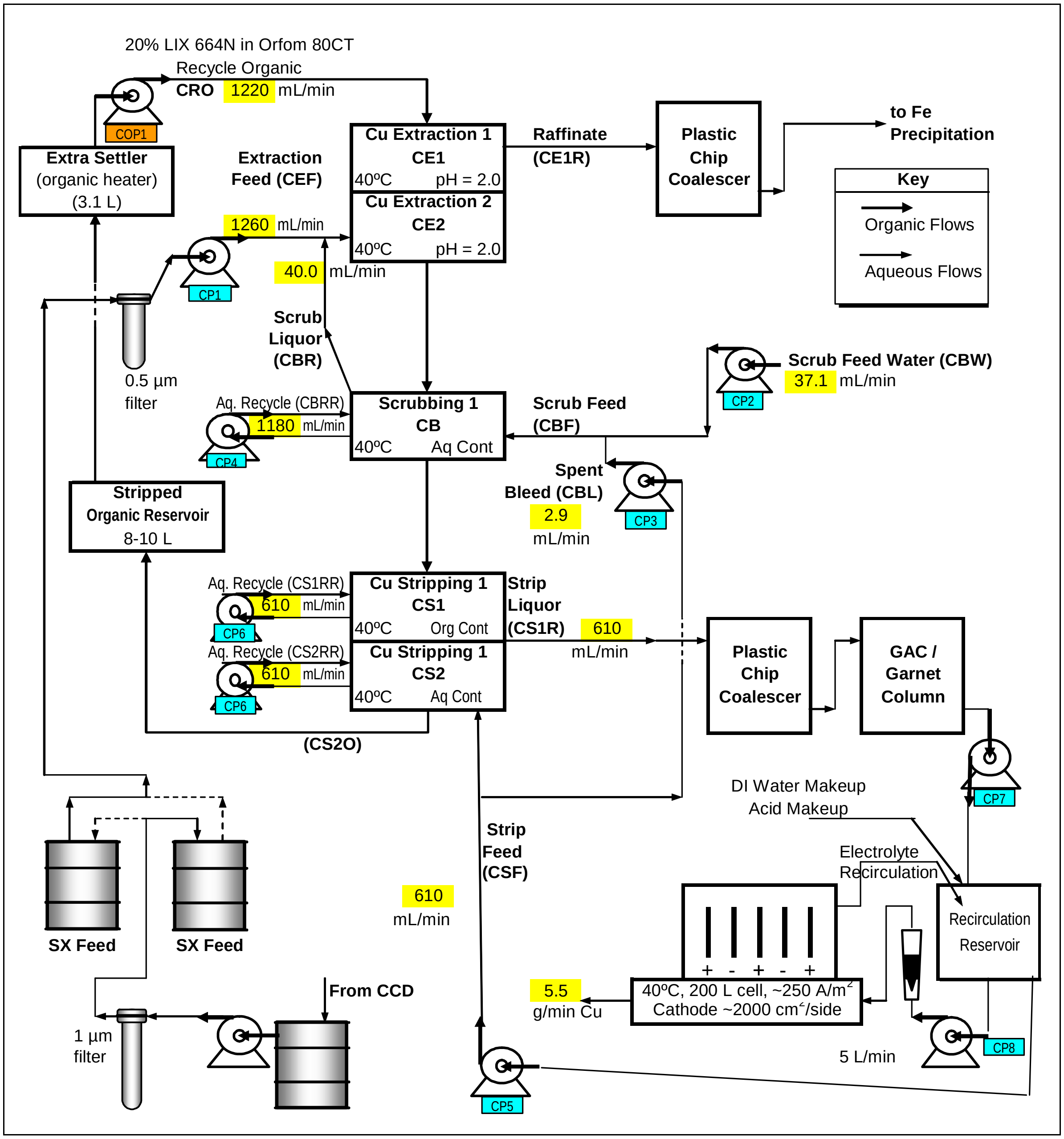


Photo 3 Overview of the piloting Copper Solvent Extraction equipment setup.

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 30: 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.

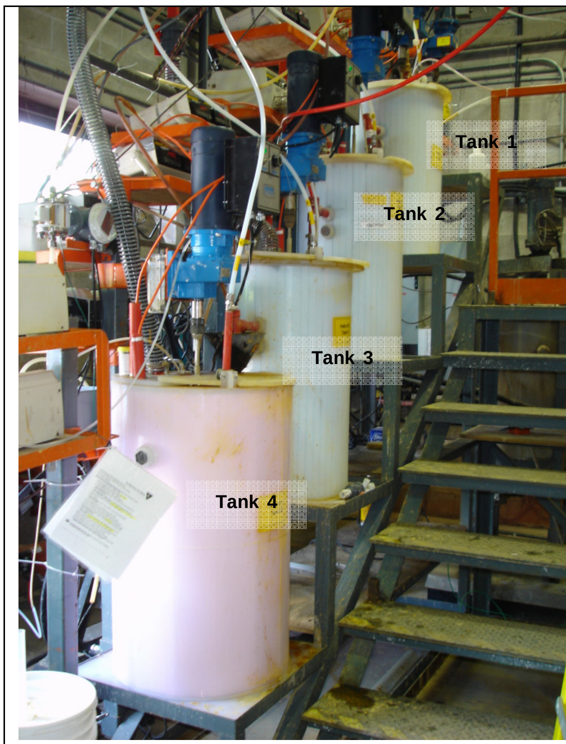


Photo 4 Pilot Plant Iron Removal Reactors and Iron Removal Thickener

CSIRO Direct Solvent Extraction (DSX)

The DSX process utilized a 6.25% Versatic 10 (0.33M) and 13.2% LIX 63 (0.3M) v/v organic mix in Orfom 80 SX CT diluent. This organic mixture was developed 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 the CSIRO had indicated selective recovery of Co and Zn from Boleo solutions and this had also been 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 the CSIRO and the Bateman Solvent Extraction Group (also known as Bateman Advanced Technologies or BAT, a specialist solvent extraction division within the Bateman Group) confirmed the effect of manganese on the organic stability and lead to a series of tests by CSIRO to optimize 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.

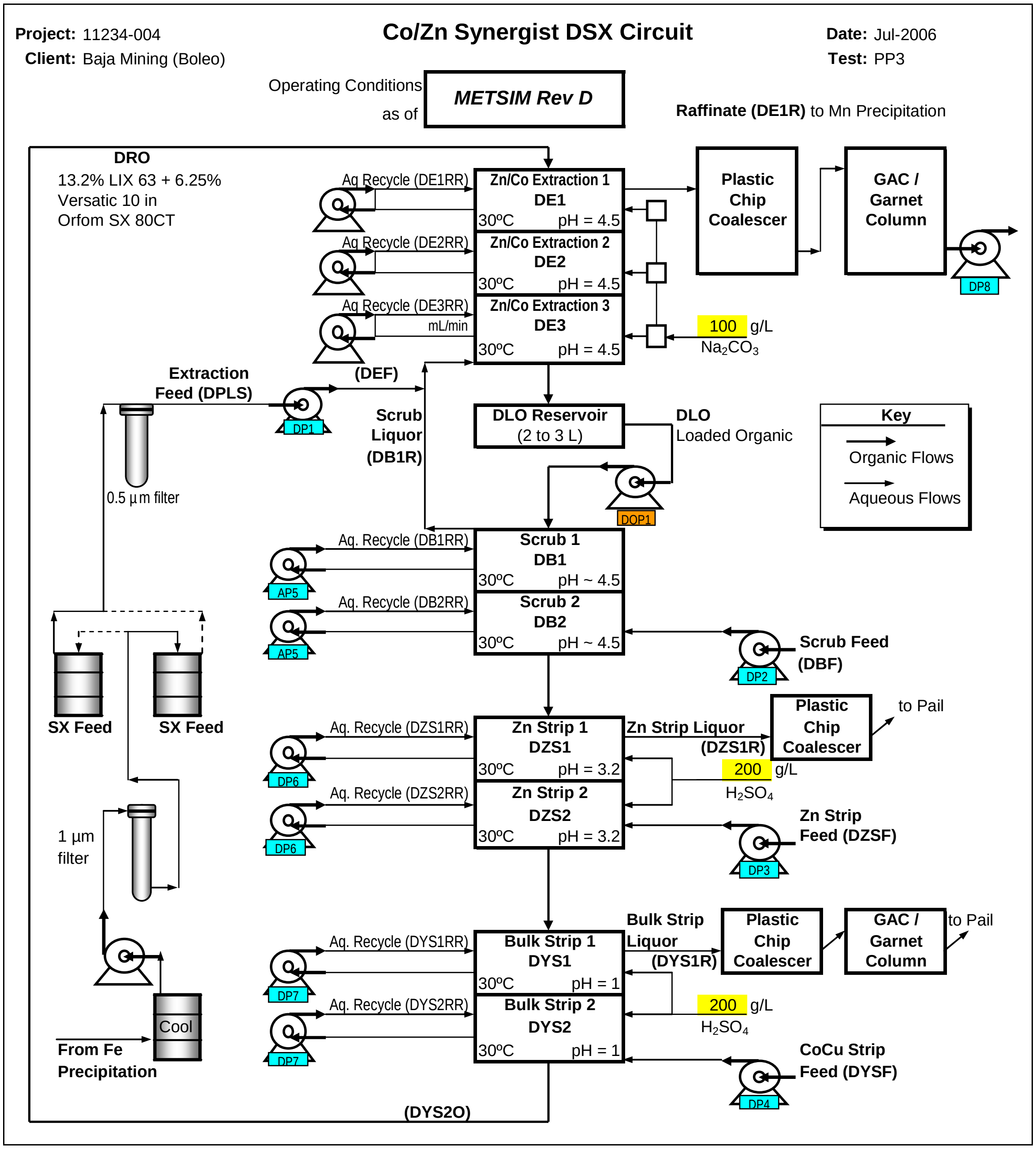
The circuit consisted of three extraction, two scrub, two Zn strip and two bulk (Co, Zn) strip stages operated at 30°C.

Photo 5 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 31: Direct Solvent Extraction (DSX) Flowsheet and 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.

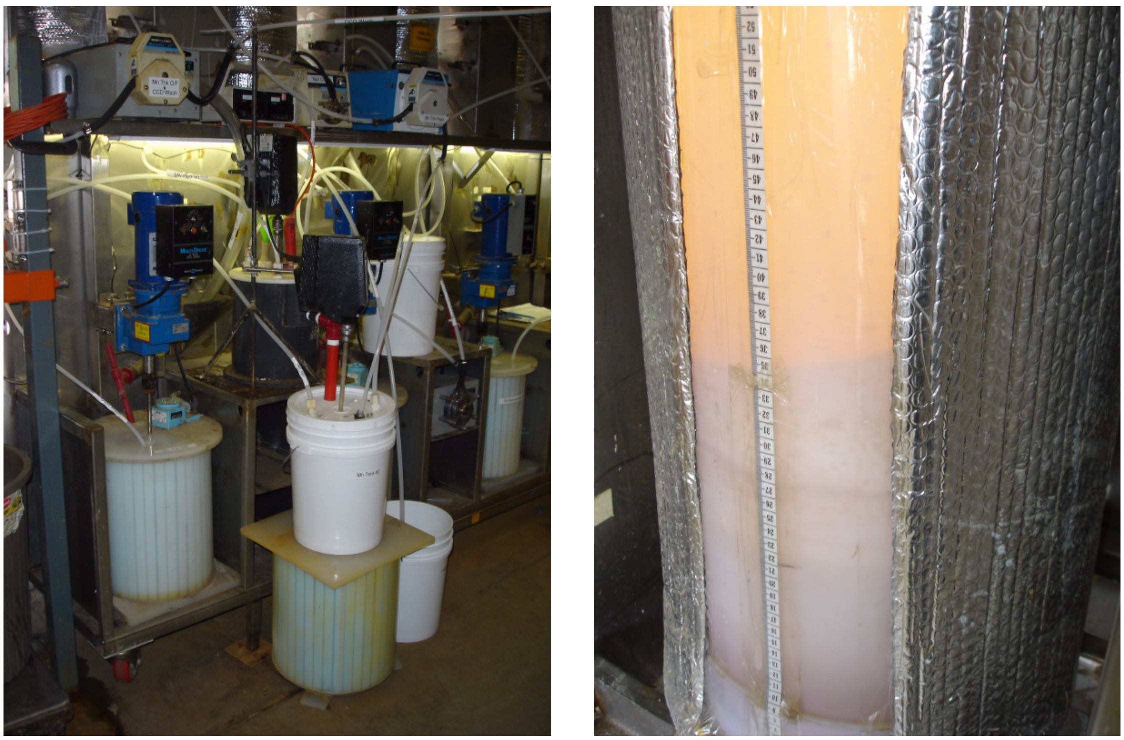


Photo 6 Overview of The Manganese Carbonate Precipitation Circuit (Reaction Tanks and Thickener)

Photo 7 Manganese Carbonate Thickener Underflow And Filter Cake

Pilot Plant Controls

Pilot plant controls included regular flow rate/mass measurements, on-line process variable trending (temperature, redox potential and pH), analytical laboratory assay trending, bench titrations, reagent consumption monitoring (flow metres, load cell measurements, containers) and log sheets. Actual flow rates 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.

Table 18 shows the analytical methods utilized by SGS Minerals Services Analytical Group.

Table 18: 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 re-pulp) of solids to minimize errors from soluble metal contributions.

Commissioning & 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 startup times for the various sections are shown in Table 19.

Table 19: Boleo Pilot Plant Timeline

Section	Time	Date
Start of Campaign		
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 Neutralization

Table 20 shows the leach and partial neutralization operational KPIs.

Table 20: Leach and Partial Neutralization 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 in Table 21. 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.

Graphs of the metals extractions for the duration of the pilot plant campaign are shown in Figure 32 to Figure 35.

Table 21: 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

Figure 32: Boleo Metal Extractions – Cu

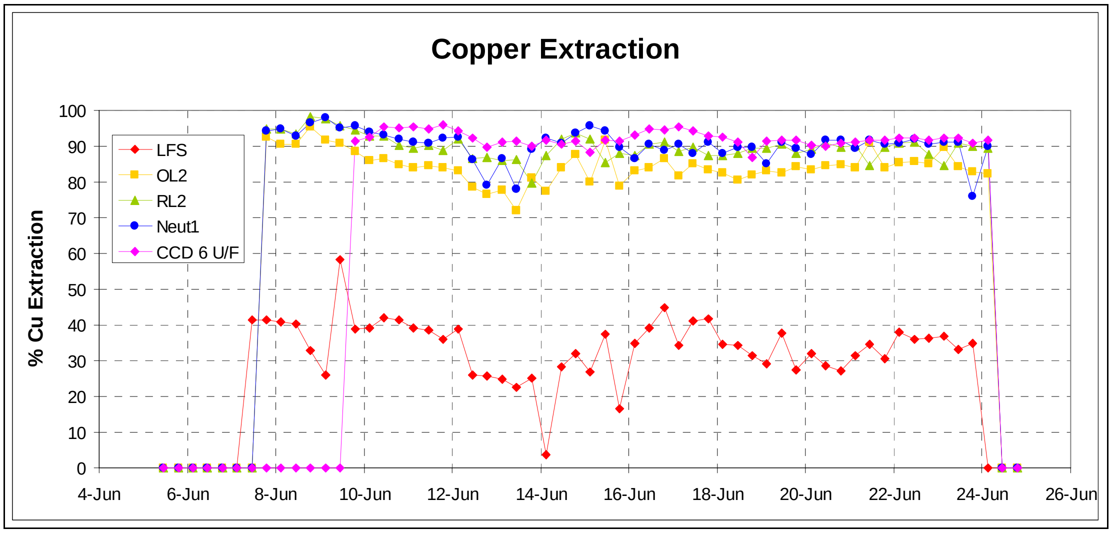


Figure 33: Boleo Metal Extractions – Co

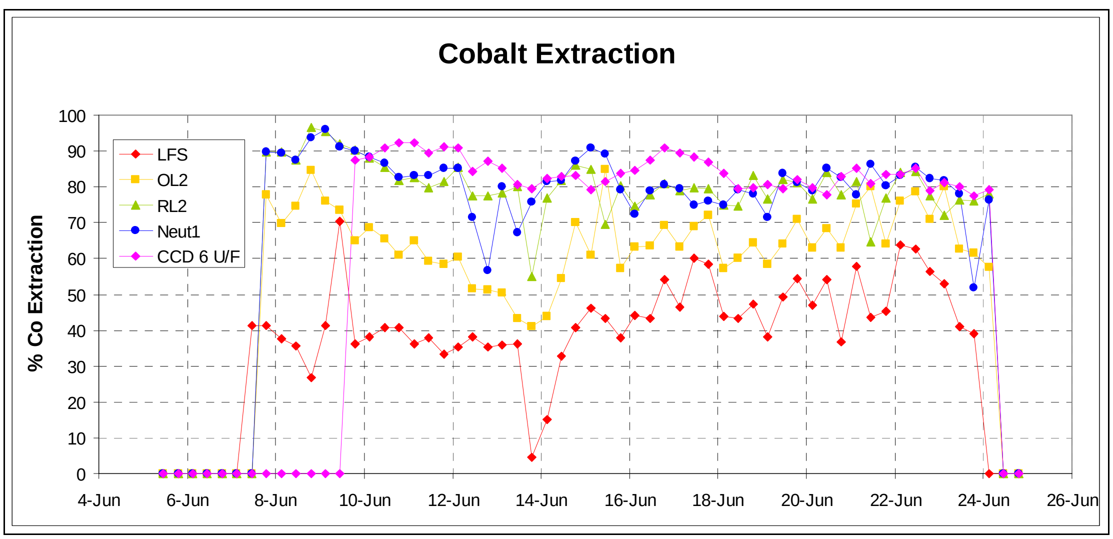


Figure 34: Boleo Metal Extractions – Zn

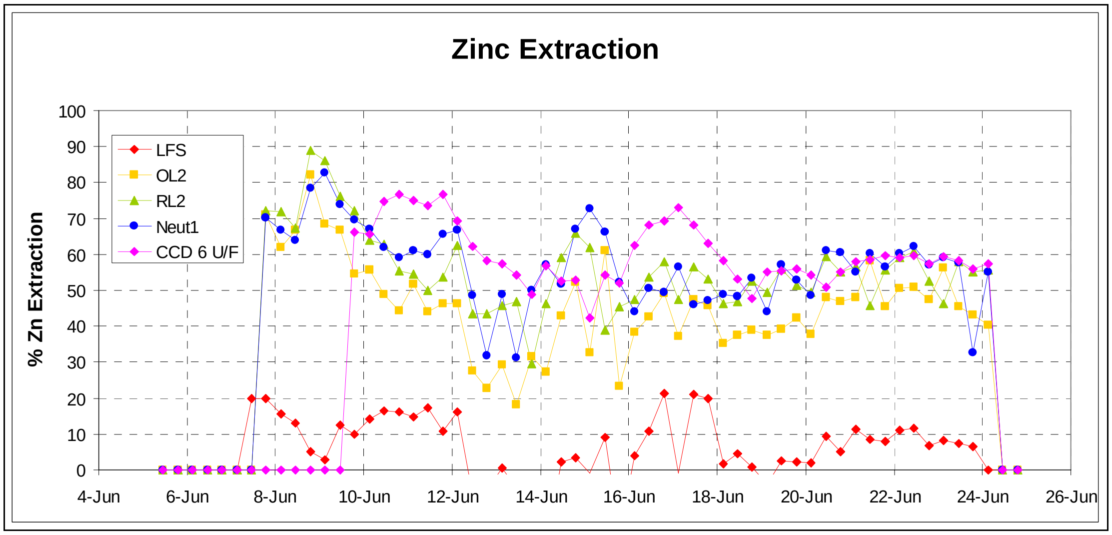
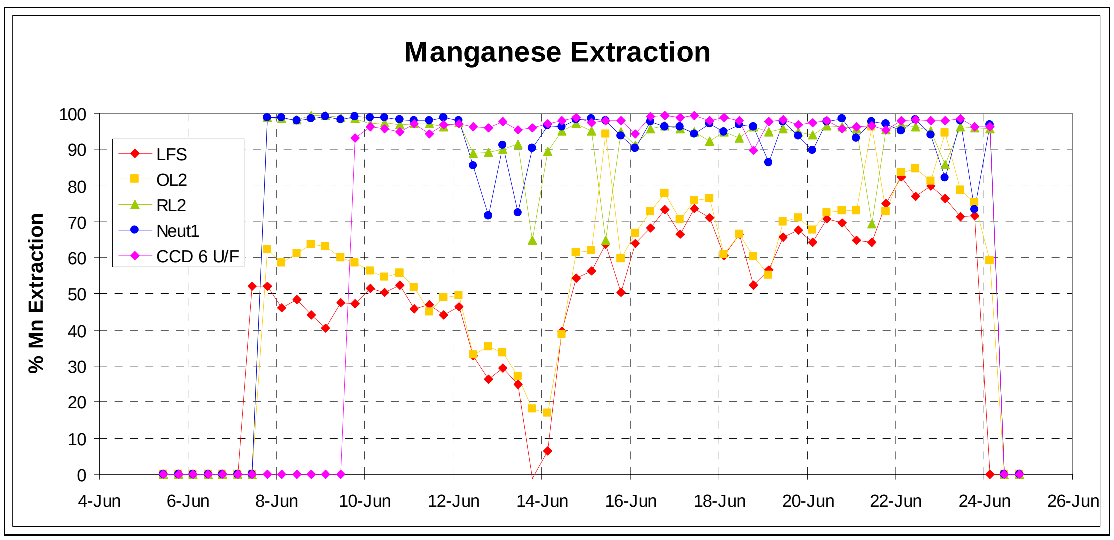


Figure 35: Boleo Metal Extractions – Mg



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

COUNTER CURRENT DECANTATION (CCD)

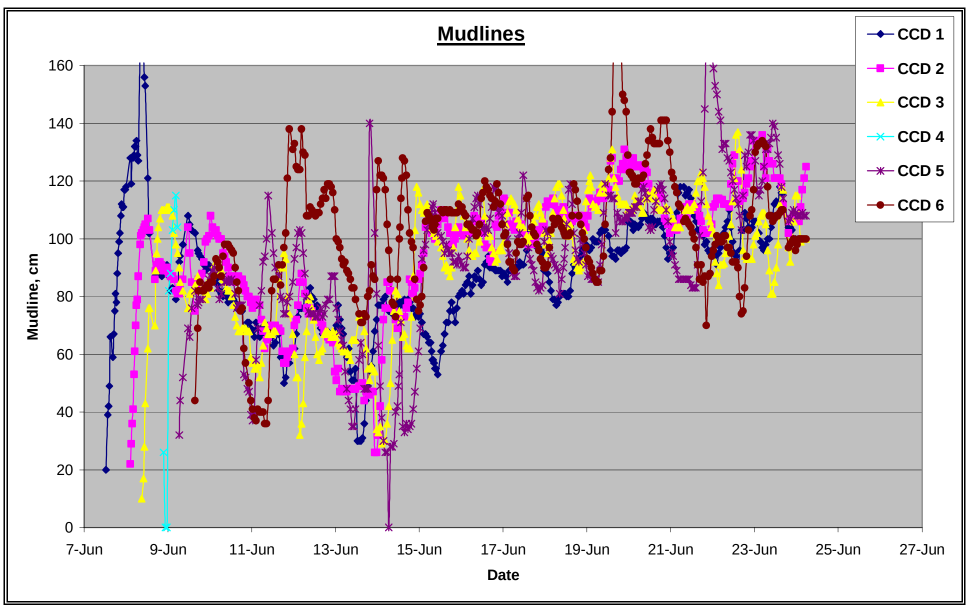
The CCD circuit average operational KPIs were as shown in Table 22.

Table 22: 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					

Figure 36 shows the improved operational performance of the CCD circuit as the campaign progressed.

Figure 36: Boleo 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, the underflow density calculation is dependent on using a fixed value for the SG of the dry solids and small variations in the SG will effect the calculation. As such the underflow densities obtained during vendor testwork (shown in Table 23) 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

Outokumpu Technology and Pocock Industrial Inc. carried out thickening testwork during the campaign. In addition, samples were sent to GL&V for paste thickening testwork. Table 23 shows a summary of the vendor results.

Table 23: 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 to those achieved in comparative tests conducted by Outokumpu in 2004. The reasons for the difference in settling 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 Boleo 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 in Table 24.

Only minor amounts of manganese and iron were co-extracted with the copper. The average organic assays are listed in Table 25.

Table 24: 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

Table 25: Copper Solvent Extraction Organic Assays

		Organic Assays		
Element	Unit	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 26.

Table 26: Plating Cycle Times and 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	

All four cycles produced copper quality exceeding the LME grade-A specification. (see Table 27).

Table 27: 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

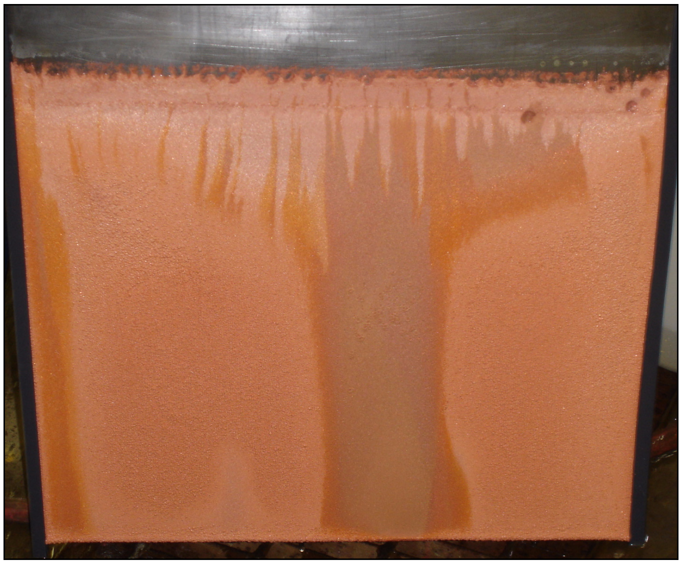


Photo 8 Cathode from Electrowinning Cycle No.1 – 13 June 2006

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 feed stream was 5,454 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:

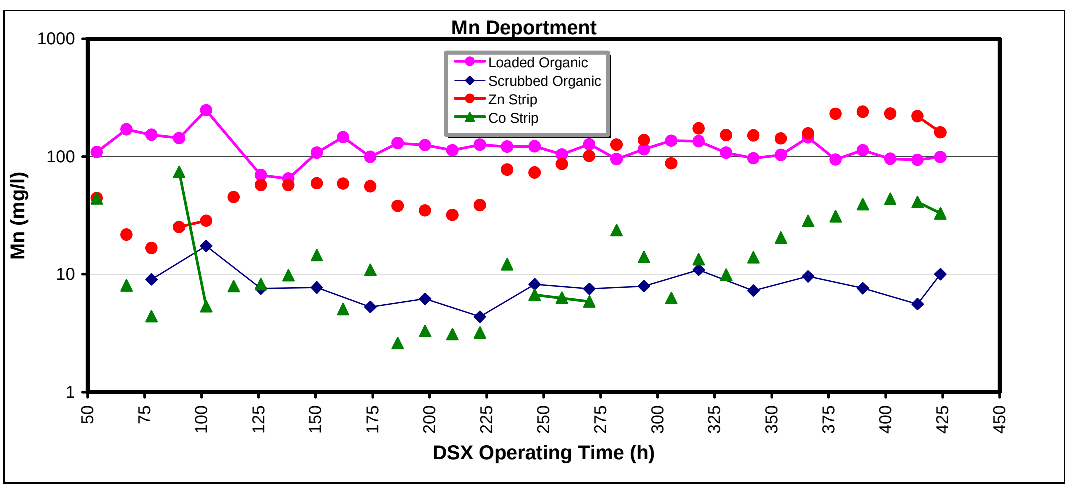
- In general the iron precipitate settled and filtered easily and gave a clearer overflow at higher temperatures. The initial operating temperature of 60°C was dropped incrementally to 45°C over a period of two days from 15th June. However, this resulted in poorer settling performance, reduced overflow 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 37: Manganese Loading On DSX Organic Phase



METAL EXTRACTION

The DSX extractions for copper, cobalt, and zinc are shown in Table 28.

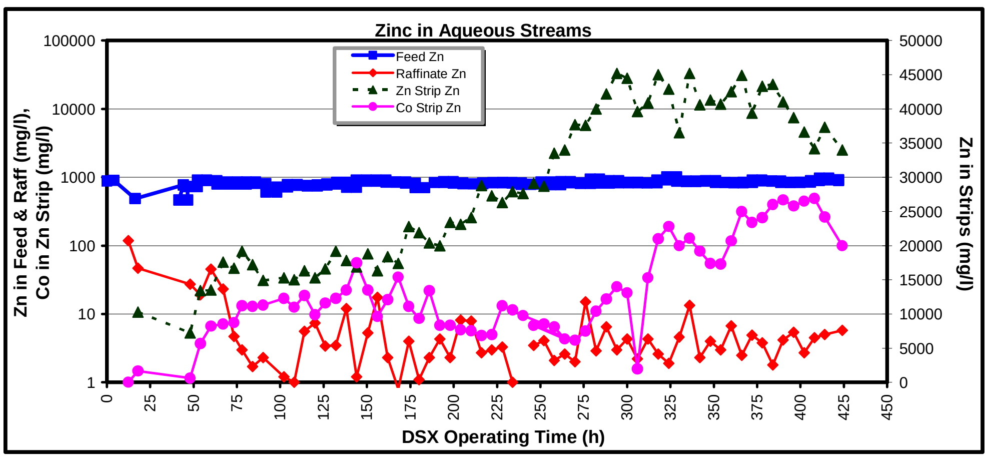
Table 28: DSX Metals Extractions

Description	Unit	Cu	Co	Zn
Extraction	%	99.57	99.48	99.03

ZINC SELECTIVE STRIP

Figure 38 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.

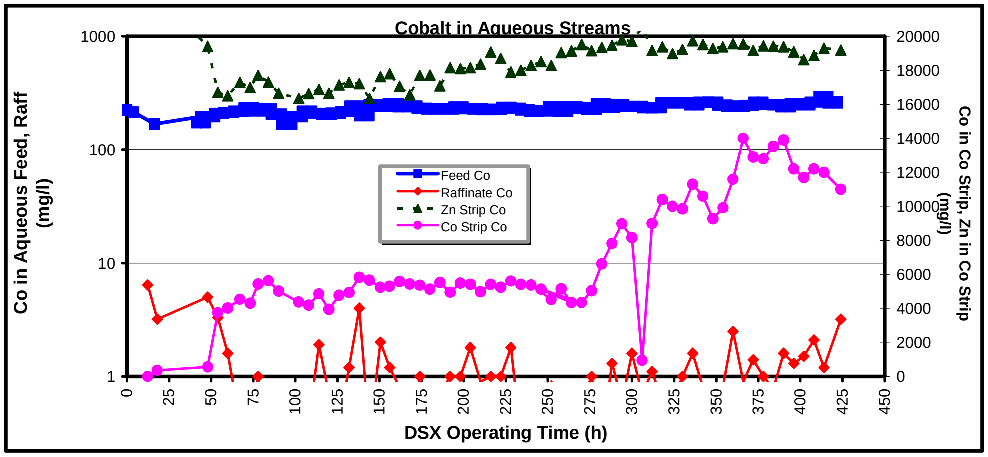
Figure 38: Zinc in Solution vs. DSX Operating Time



BULK STRIP

Figure 39 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 39: 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. Table 29 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 29: Manganese Carbonate Product 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 in Table 30.

Table 30: 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

15.1.4 VARIABILITY TESTWORK

In March and April of 2007 a bench scale testwork programme was initiated at SGS's Lakefield Research laboratories with a number of objectives:

- to increase the level of confidence in the design acid consumption values which in turn affect the size of the acid plant needed for the project
- to assess leach recovery variability across the orebody
- to establish and increase confidence in the grade-recovery relationships for each of the paymetals.

To this end a total of twenty composite samples were produced from approximately 350 individual drill core samples. The drill core samples originate from across the Boleo Reserve, covering an area of some 10 km^2, representing some 25 to 30% of possible resource. The 350 individual core samples were predominantly manto 3 oxide material with about 15% of the samples being sourced from manto 2.

The samples were composited to reflect the first five years of feed to the plant according to the [then] mining plan. Each of the samples was therefore designed to represent 4 months worth of plant feed.

Twenty individual bench scale leach tests were carried out, simulating the proposed process flowsheet, to generate 20 data sets of grade-recovery relationships for each of the copper, cobalt, zinc and manganese elements as well as the acid consumption levels required to achieve the leach recovery.

Two additional leach tests were added to the series where leach testing was conducted on feed ore from the Fully Integrated Pilot Plant campaign so as to allow a direct comparison to be made with pilot campaign results. Intermittent duplicate analyses were also undertaken to provide additional confidence in the results.

The results of this programme are currently under review by Baja Mining Corp. and cannot be reported at this time but will be incorporated in the final feasibility study documentation. Initial feedback is that the results fall well within the current set of design assumptions for both acid consumption and metal recovery.

15.2 PROCESS PLANT DESIGN

This section remains largely unchanged from the previous document.

15.2.1 INTRODUCTION

The proposed treatment route for the Boleo 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 in Figure 40 to Figure 42.

Figure 40: Schematic Flow Diagram Sheet 1

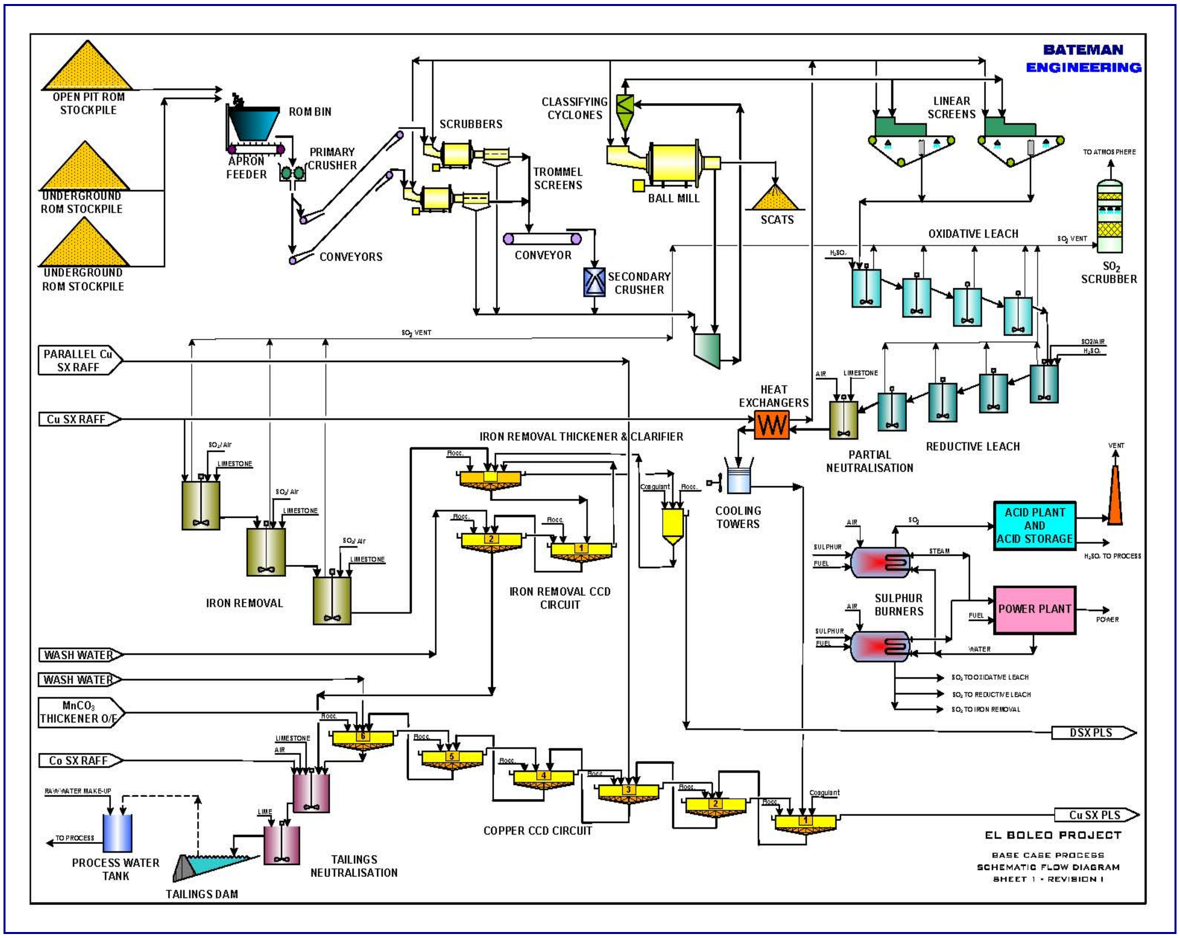


Figure 41: Schematic Flow Diagram Sheet 2

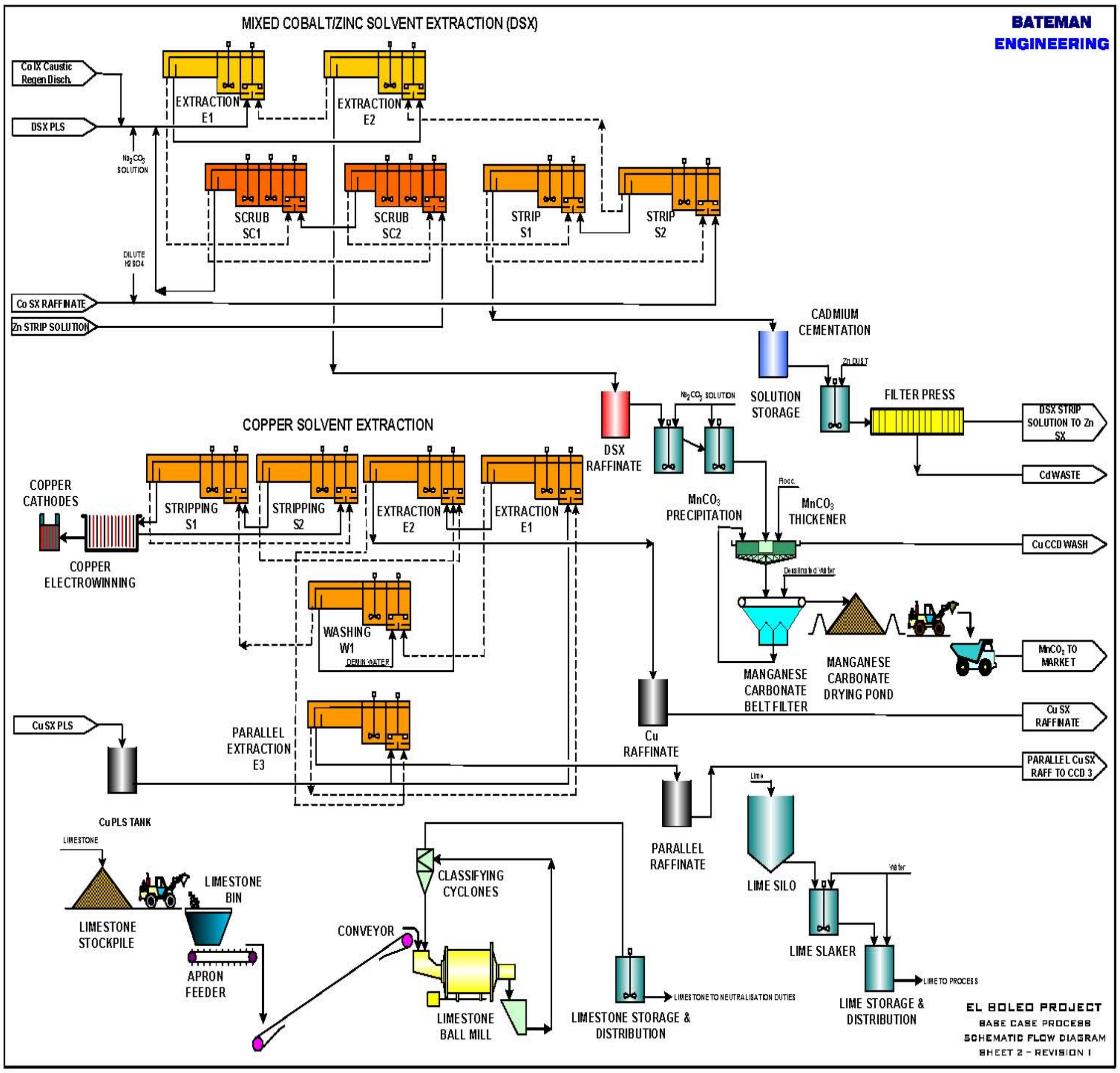
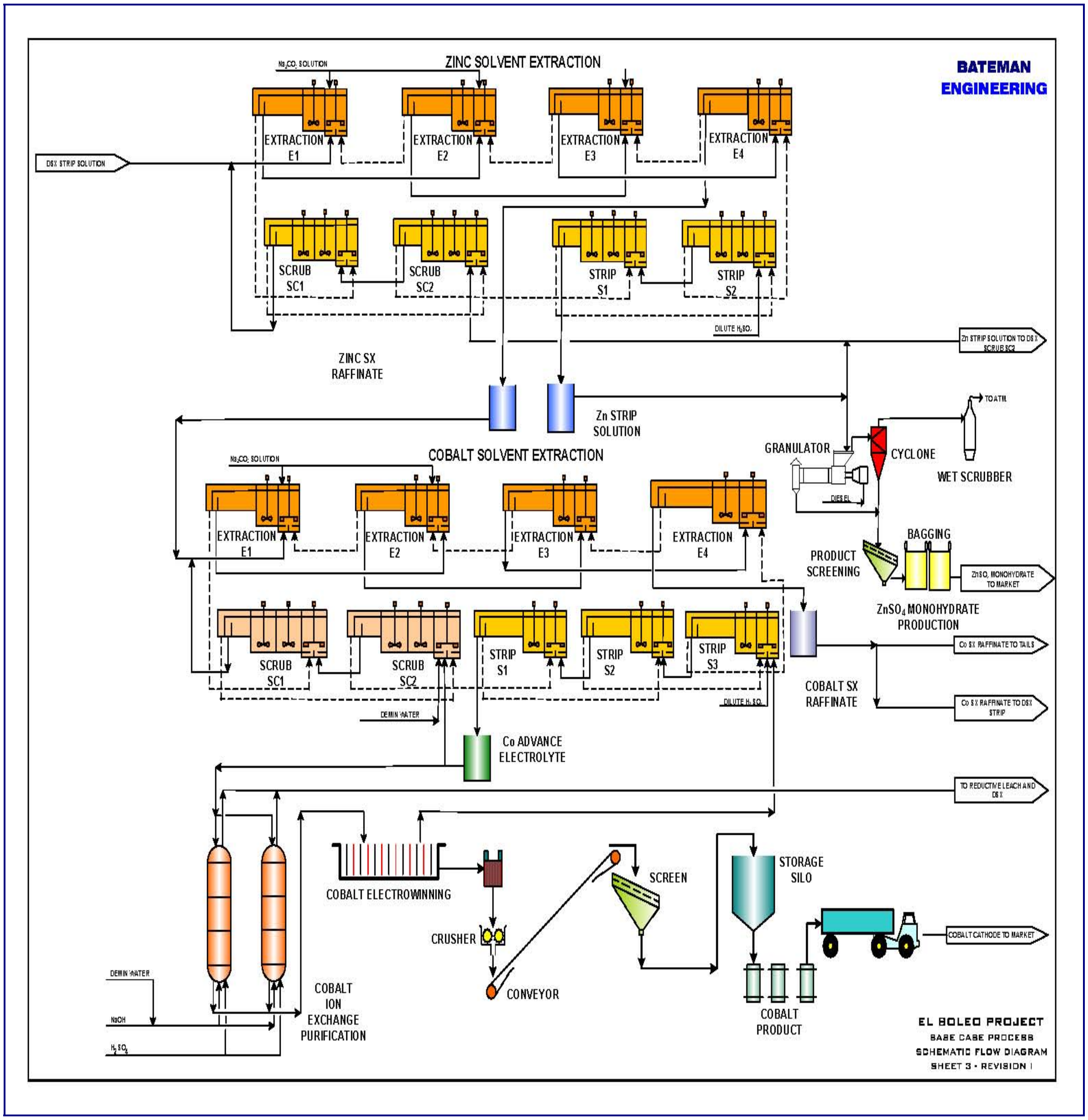


Figure 42: Schematic Flow Diagram Sheet 3



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 Boleo 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 Neutralization – 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 Boleo 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 to assist with maintaining temperature in the leach circuit.

Counter-Current Decantation (CCD) Circuit – Cooled leach residue slurry discharged from the cooling tower is washed with seawater 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. Some additional leaching has been shown to take place in the CCD circuit. Thickener overflow from CCD stage 1 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 is split between the scrubbing and milling circuit, (where it is used as dilution and make-up solution in the comminution circuit), 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) and the CCD circuit (where it is used as additional wash solution to improve CCD wash efficiency).

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 6-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 circuit 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 Boleo carbonate, 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 solids removal ahead of the DSX circuit. This solution is maintained 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 Boleo carbonate is added to neutralise excess acid and 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 between 7 and 8 to meet local environmental disposal requirements.

The slurry discharged from this process is pumped to a tailings dam situated to the west of the plant site known as the Curuglú 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 – Neutralised zinc strip solution reports to a fluidised bed granulator where heat is applied to the neutralised solution via the use of hot air in the granulation process. Particles are 'grown' to the appropriate size and shape, cooled, screened and packaged in a stand-alone production facility. The hot 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
- SO_2 gas production.

The SO_2 is sparged into the slurry in the reductive leach operation. Sulphuric acid is dosed according to the numerous needs of the 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 overall mining and plant operational power requirements are largely met by this co-generation system, making the Boleo Operation virtually self sufficient in electricity. The power shortfall experienced at elevated throughputs will 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 Neutralization, Tailings Neutralization and Iron Removal. Limestone is mined on the Boleo 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 proportion of this water production will be diverted for use throughout the plant and mining operations as potable water.

Tailings Dam – The Boleo Creek tailings dam, located in the Curuglú area, is designed to accommodate approximately 20 years worth of plant tailings. The dam wall will be raised in 4 stages over the life of the mine. The first dam wall will accommodate approximately 4 years worth of plant tailings production at design feed rates.

16 MINERAL RESOURCE & MINERAL RESERVE ESTIMATES

16.1 3D GEOLOGICAL INTERPRETATION

The geology of the Boleo deposit is well known (see Sections 3.7, 3.8, and 3.9) 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 and to a lesser extent from drilling. 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 by Curator and MMB was used by H&S 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 2 km in width), amounting to 460 line km.

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

16.1.1 FAULT SURFACES

To build the fault framework, faults were firstly digitized 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 digitized polylines correctly. The process and results are shown in Figure 43 through Figure 46.

Figure 43: Fault Framework in Section – Detail of Cross-Sections (900N) as Supplied by MMB

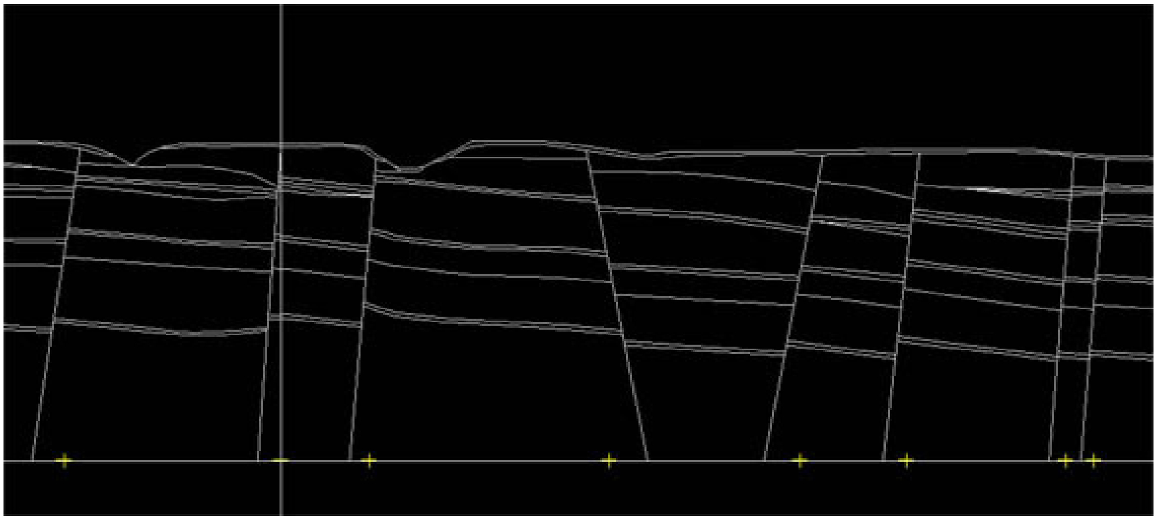


Figure 44: Digitized Fault Traces – Faults Digitized (dashed red lines) Following Cross-Sectional Interpretation (900N)

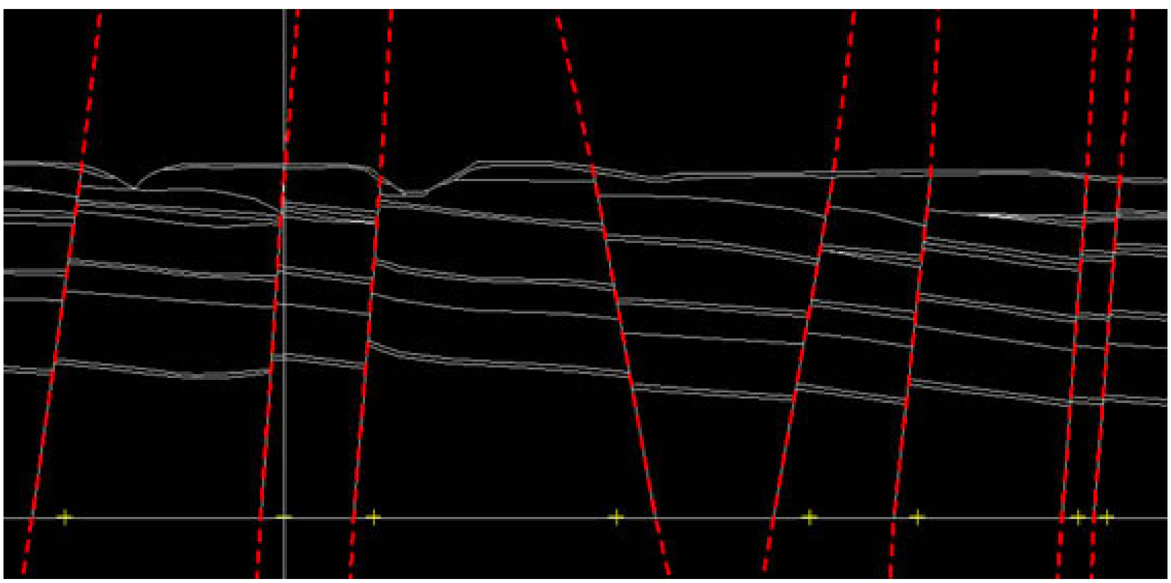
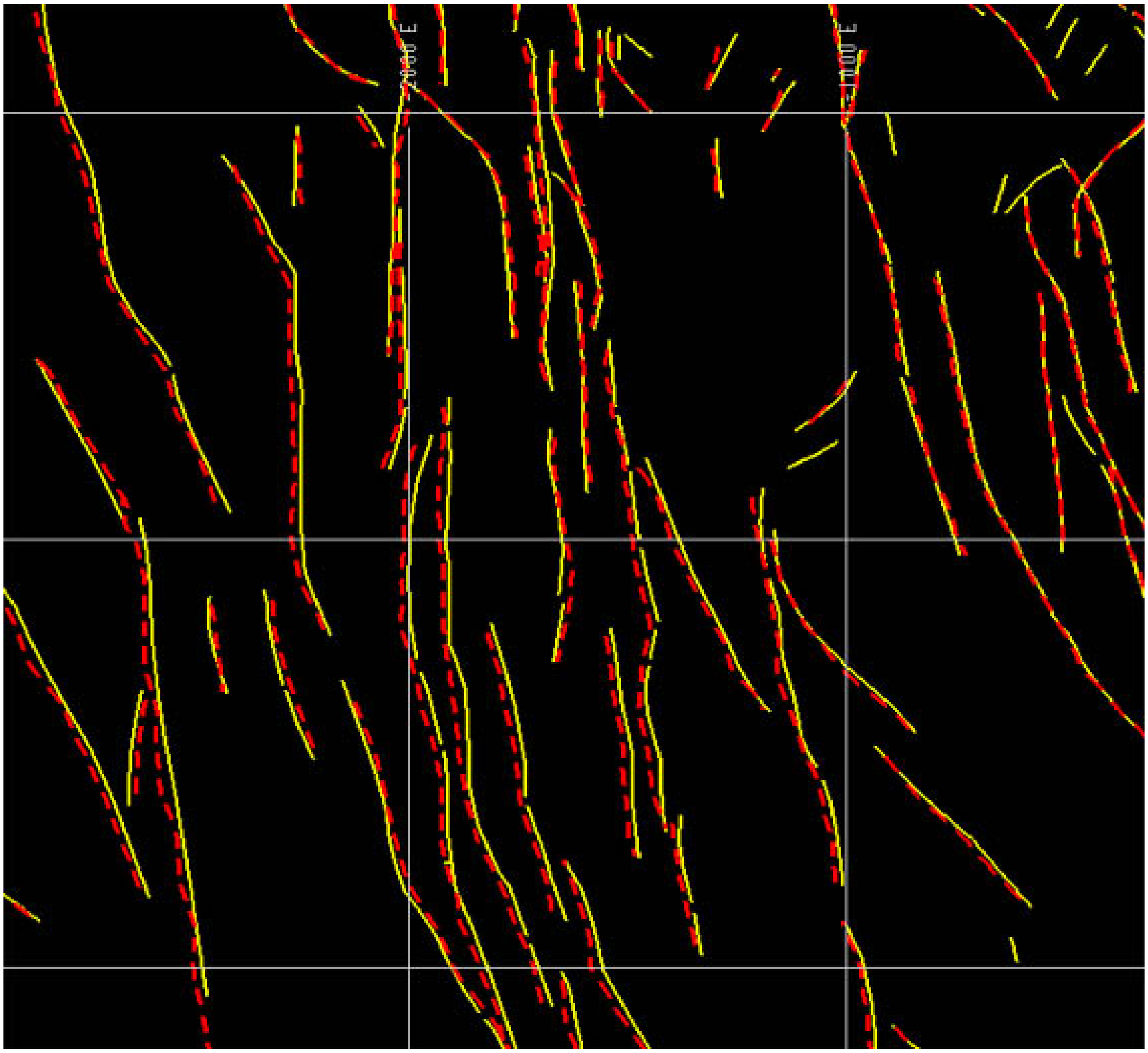
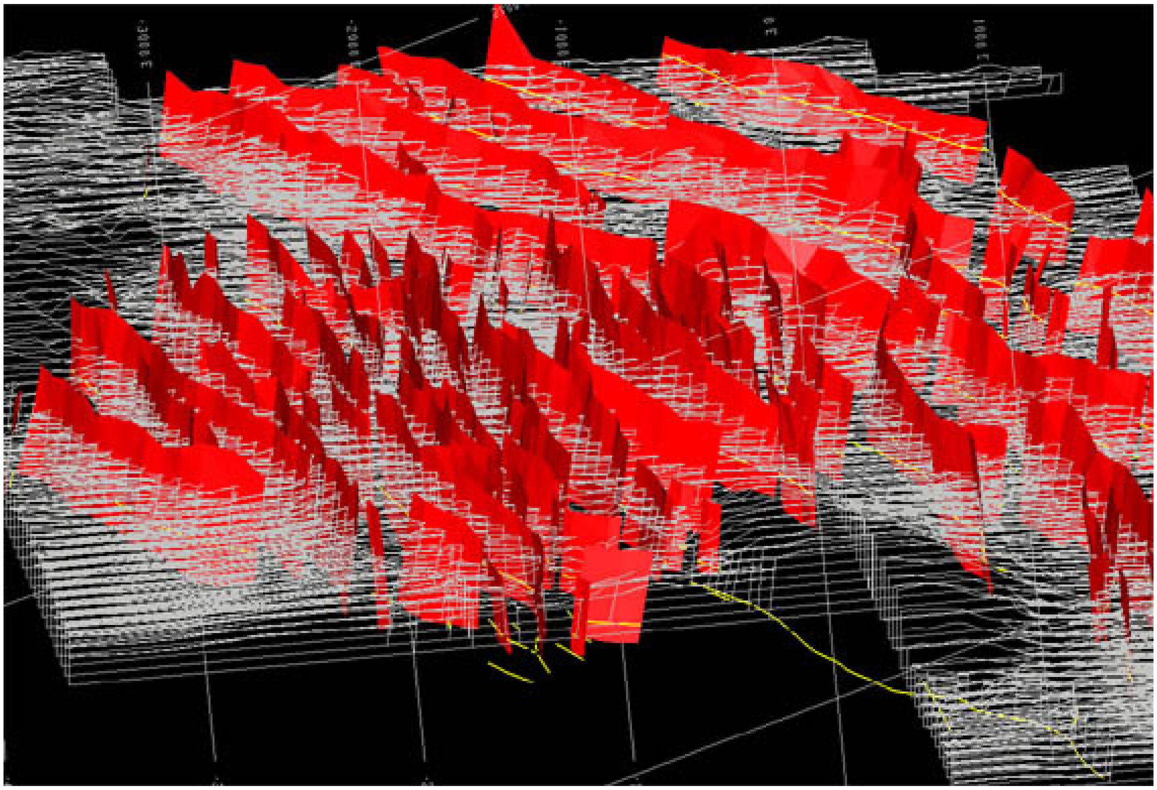


Figure 45: Correlation of Fault Traces



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

Figure 46: 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 47).

The resulting 3D surface was sliced along the same 100 section lines as the Curator geological interpretation (Figure 48), these slice lines were then superimposed back over the Curator sections and 'dragged' and 'snapped' to the digitized fault framework (Figure 49).

Figure 47: Triangulated Manto Surface – (Manto 3, oblique view)

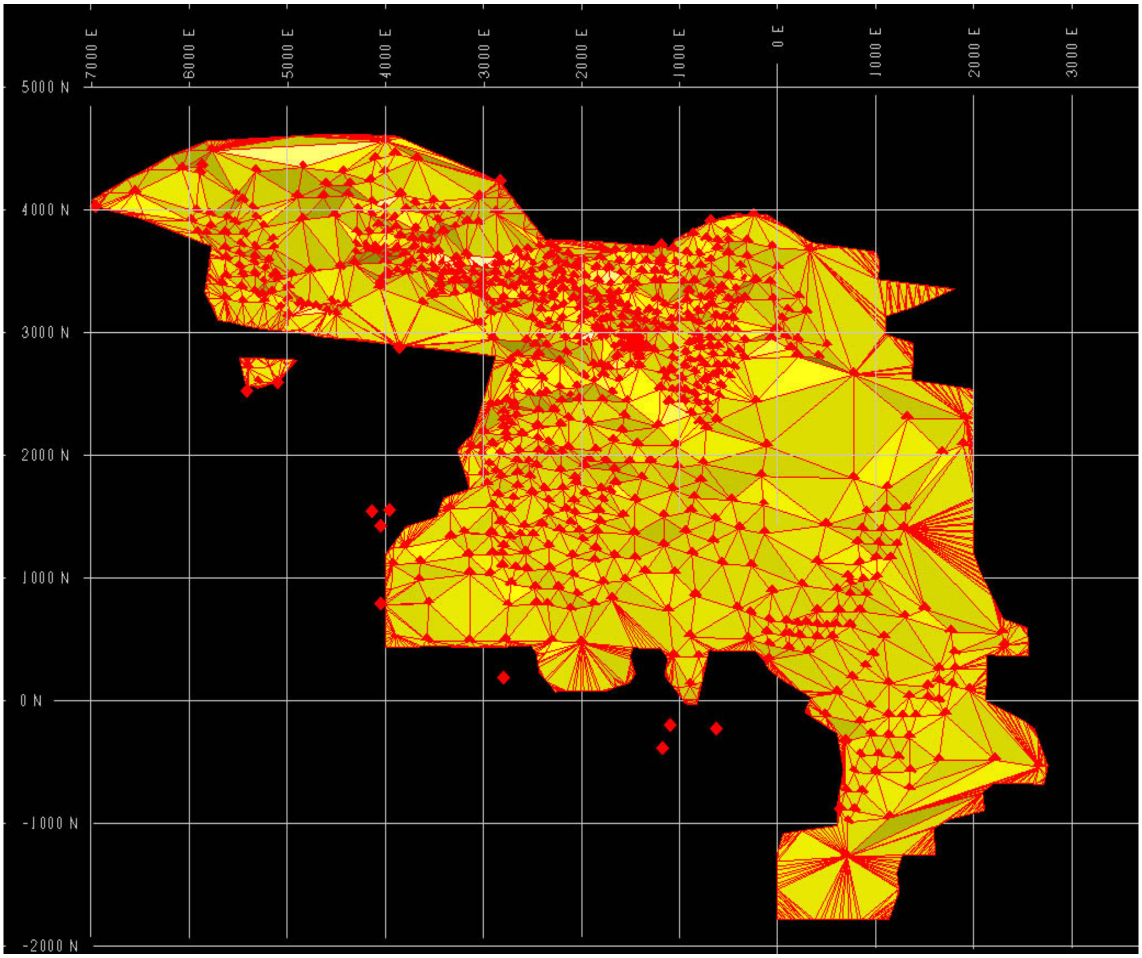


Figure 48: Triangulated Manto Surface Sliced Along Section Lines – (Manto 3, oblique view)

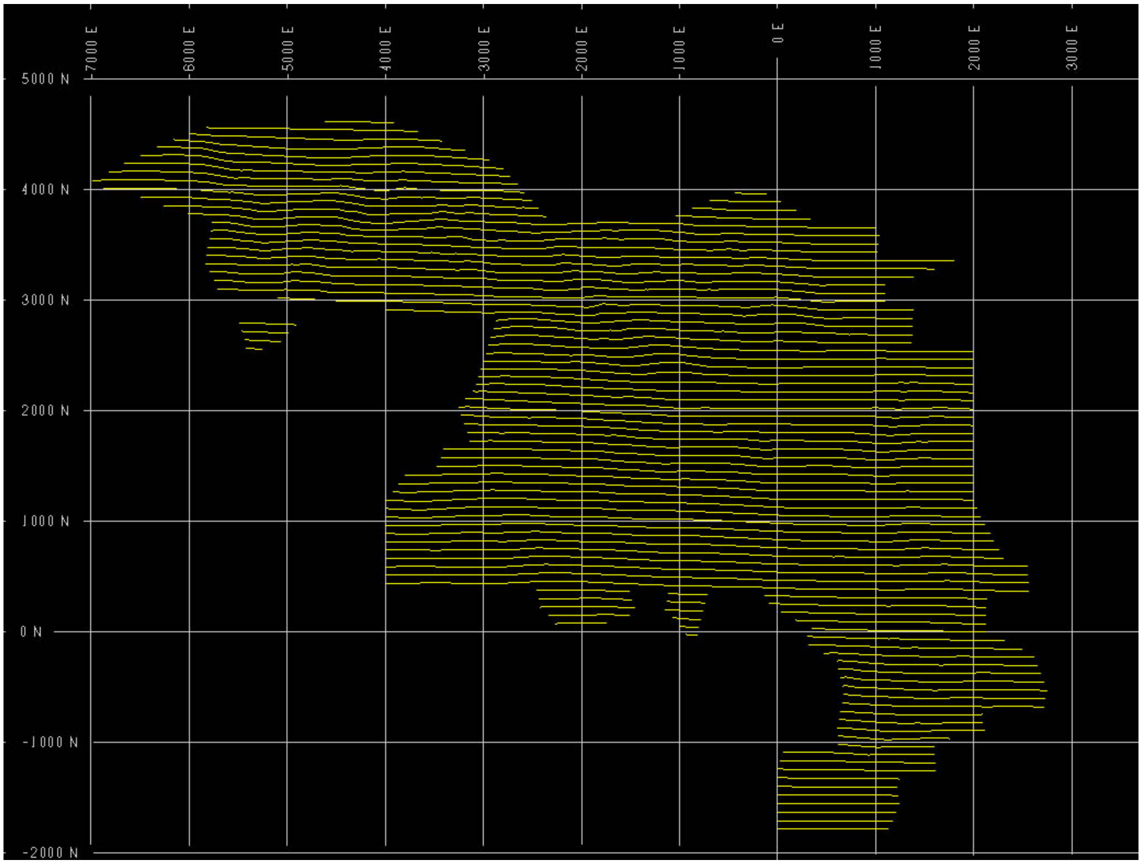
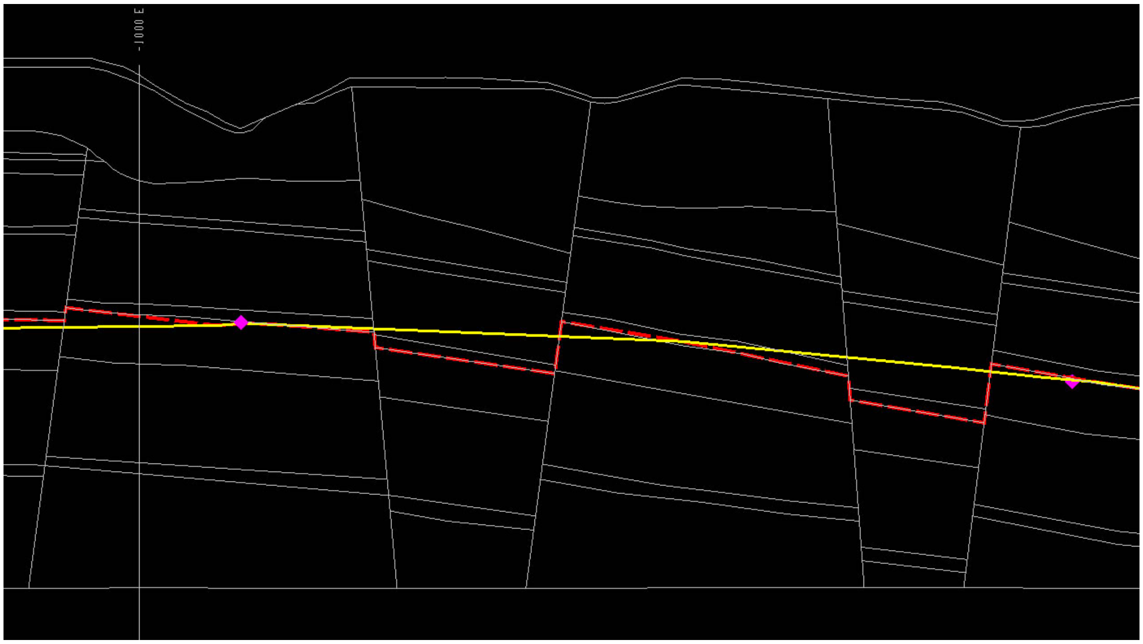


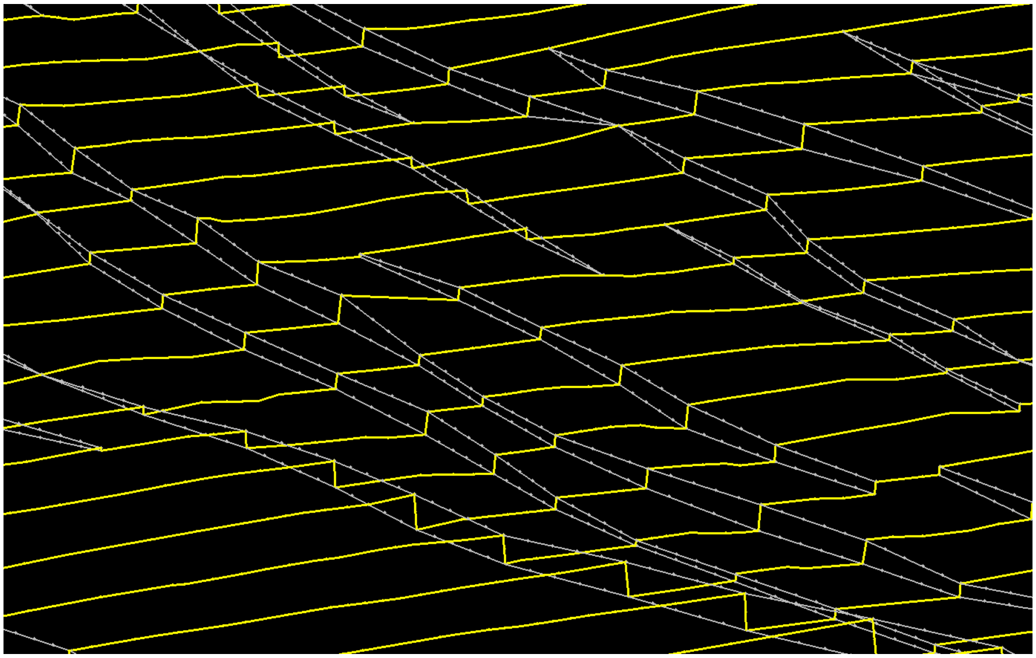
Figure 49: Cross Section Showing Snapping Footwall Surface to Faults (Manto 3)



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; by linking each profile together progressively ensuring the correct fault displacements are maintained, to form a triangulated surface (Figure 50 and Figure 51). Final surfaces are demonstrated in Figure 52 and Figure 53.

Figure 50: Re-Triangulation of Manto Surface – Section Lines and Fault Tie Lines (Manto 3, oblique view)



Note: New Fault Snapped Section Lines (yellow), Fault Tie Lines (blue)

Figure 51: Re-Triangulation of Manto Surface – Detail of Final Result (Manto 3,oblique view)

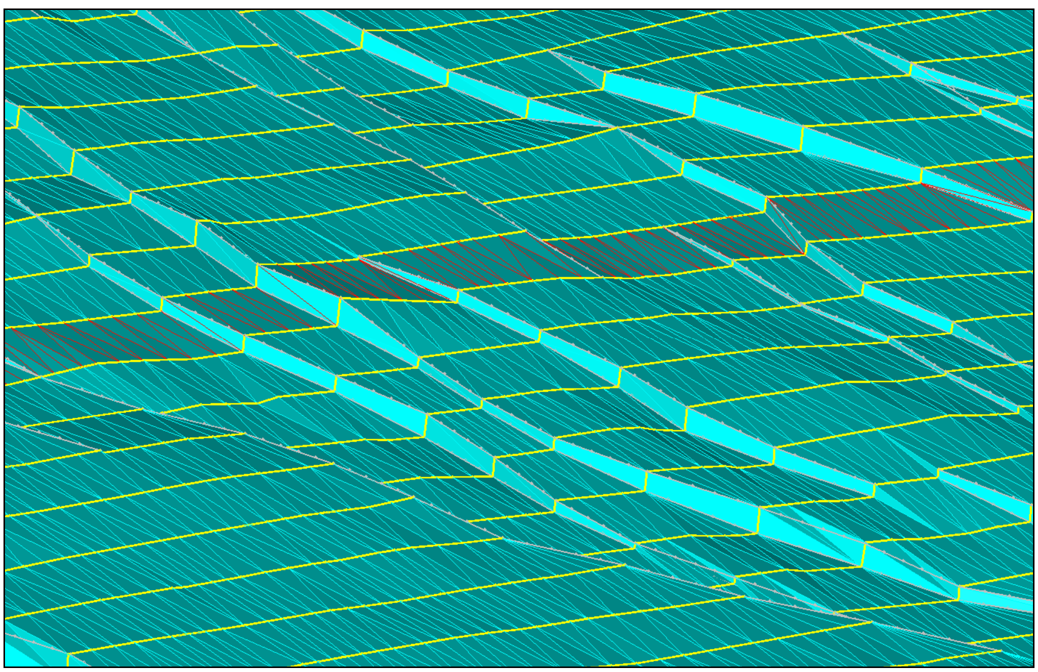


Figure 52: Final Triangulation of Manto Surface – Full View (Manto 3, oblique view)

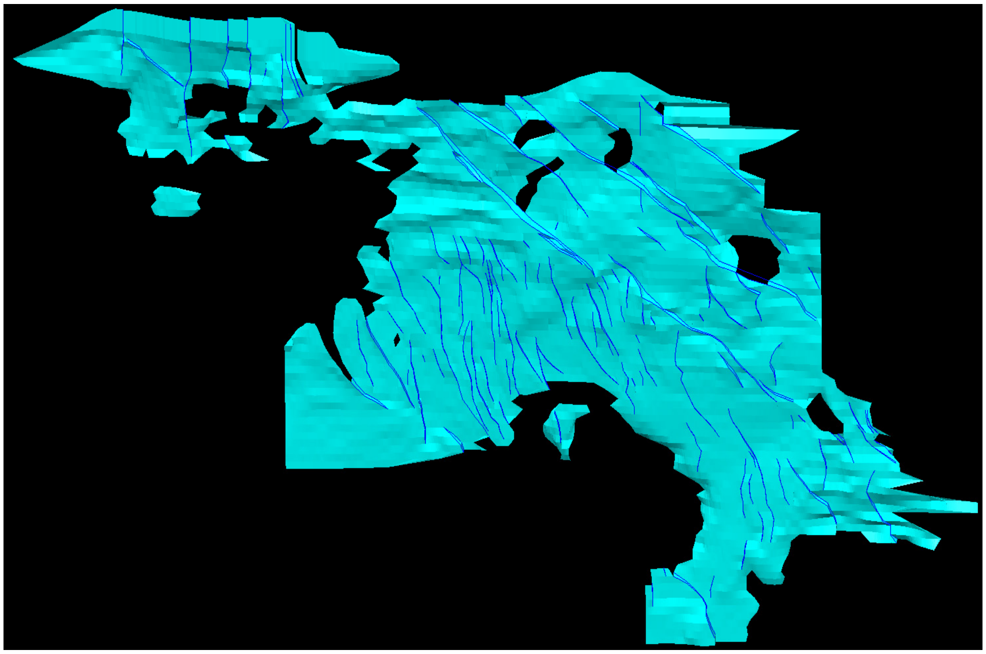
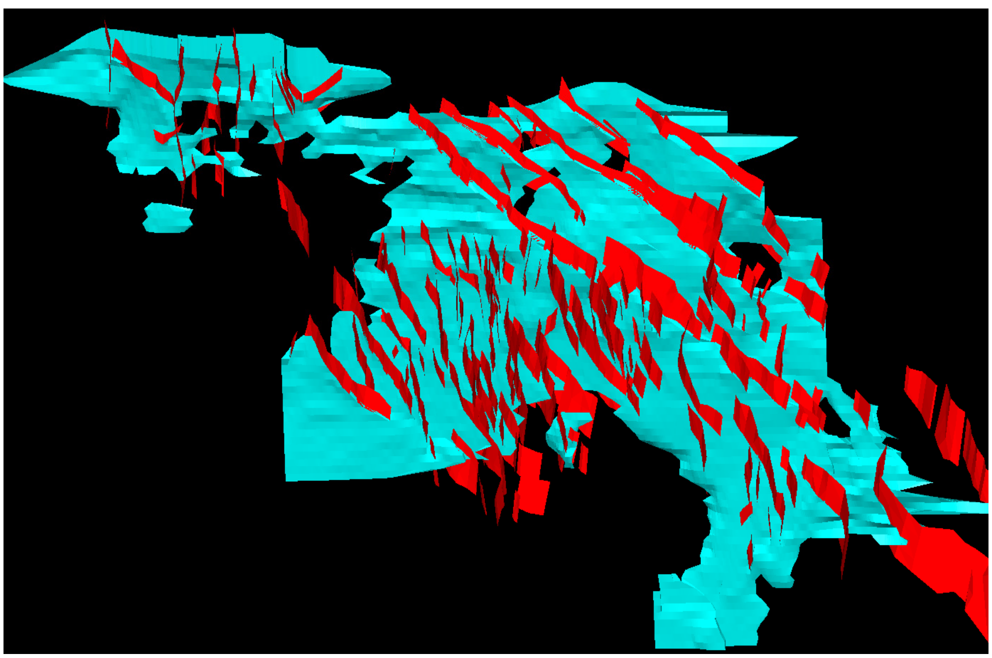


Figure 53: Re-Triangulation of Manto Surface – Full View with Faults (Manto 3, oblique view)



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 Boleo copper-cobalt-zinc manganese resource will be mined by a combination of open cut and underground methods. Consequently, different types of resource model have been constructed that reflect the different requirements of open cut and underground mine planning.

In 2004, resource estimates were completed by H&S as 3D block models. These models represented the total resource within the property and were used for open cut mine planning. This approach was appropriate and has been used again in the latest studies (2006/2007) to produce resource models suitable for open cut mine planning.

The 3D block models proved to be less useful for underground mine design. The planned underground mechanised room and pillar mining method is constrained by minimum and maximum height parameters and it is not possible to exercise the same degree of mining selectivity as in an open pit environment. To provide resource estimates more suitable for underground mine design and planning, a portion of the total resources have been re-estimated using a gridded seam model approach. Although the area covered by the gridded seam models is constrained within the larger block model, the methodology used to construct the gridded seam model differs from that used for the block model.

The predominant mining method used for each manto will vary. Mantos 1, 2, and 4 will be mined by underground methods. 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 & 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 Boleo 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 fault-bounded blocks. In plan, the mantos retain their continuity but in section, continuity between faults 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. This approach is now considered unrealistic since mineralization predominantly pre-dates faulting. The effect of constraining grade estimation within fault blocks, particularly where faulting is dense, is to limit data available for use in the grade estimation process. To estimate grade of a particular block, estimation techniques, such as kriging, capture data by means of a search ellipse (with dimensions defined by the user) with its centre at the block mid-point. Data that falls within this search ellipse are used in the estimation process.

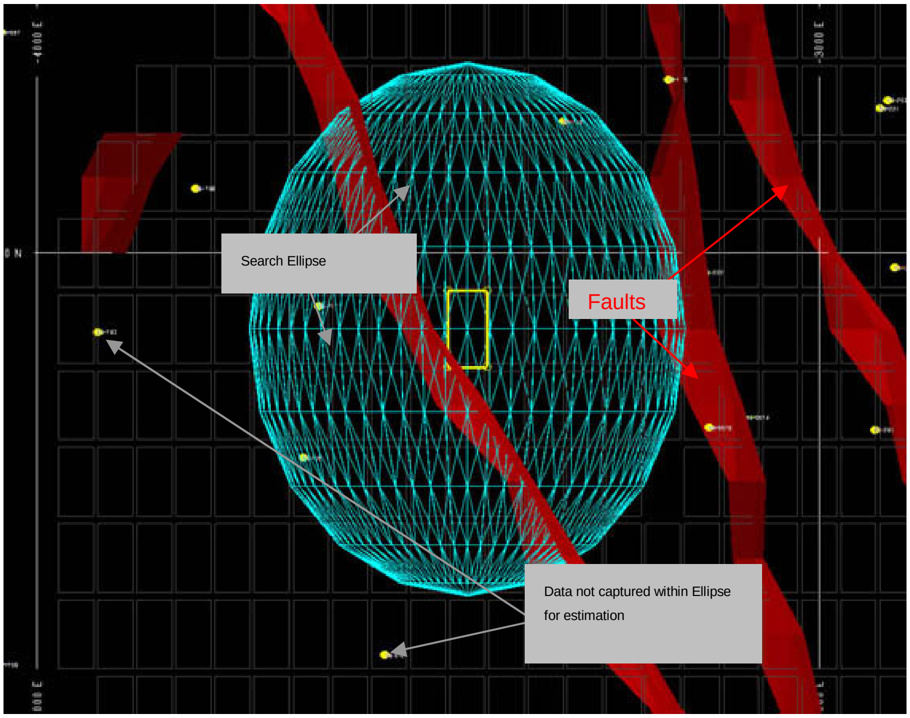
In Figure 54, a search ellipse is shown in plan and in section. In plan, the ellipse centred over a highlighted block appears to capture three drill holes that fall within it (yellow points). However, when viewed in section, the block, which is adjacent to but on the up-throw side of a fault, has its 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 Boleo 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 offset 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.

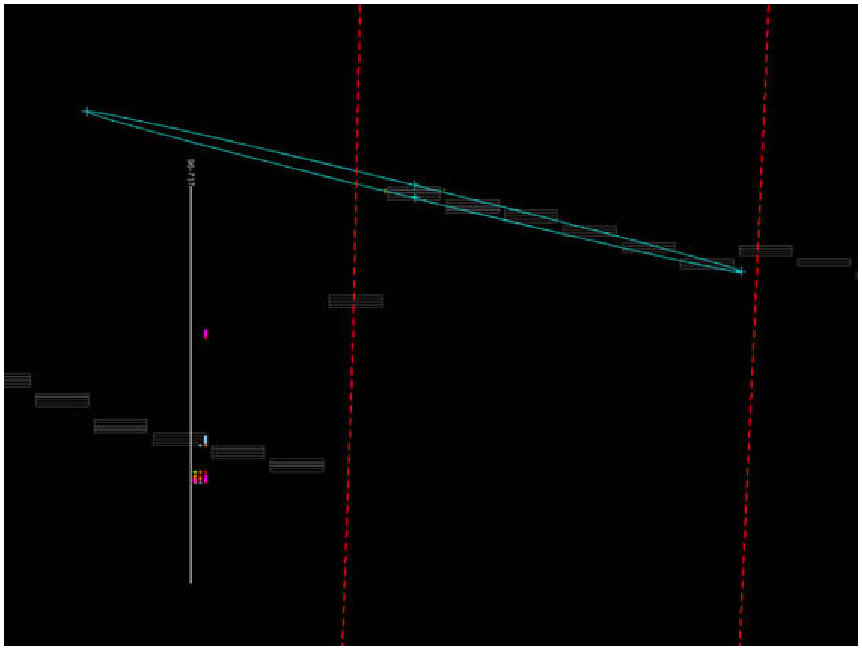
To achieve the desired effect the base of each manto intersection in each drill hole was set at the same (fixed) elevation. The base of the model was also set at the same elevation. This step removes the effect of the faulting and returns drill hole intersections to their pre-faulting position, relative 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 it only has to be slightly thicker than the maximum Manto thickness. These models are referred to as ‘flat’ models.

Figure 54: 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. The final part of the modelling process was 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 data, so that the elevation was set as the Z mid-point elevation of the lowest blocks in the flat model (0.5 m) and the gridded true footwall elevation values were set as integer values to be imported into the flat model as a separate model item.
- Import the true footwall elevation values into the lowest/basal level of the flat 3D block models (Z mid-point equals 0.5 m). Progressively increment the footwall elevation level by 1 m for each block model layer above the basal layer. So that model blocks with a Z mid-point value of 1.5 (the second layer of blocks) have a true elevation of the footwall elevation plus one; where Z mid-point equals 2.5 the true elevation equals the footwall elevation plus two and so on.
- The flat model data can now be exported from flat space and re-imported into a true elevation block model using the incremented footwall elevations Z mid-point values.

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

16.3 3D BLOCK MODELS

16.3.1 COMPOSITE LENGTH & 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 produces an assay population where each value or composite has equal weight.

At Boleo 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 1 m was considered appropriate.

After determining a 1 m composite length, a similar 1 m 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 Boleo, over a meaningful area, is approximately 140 m x 140 m in the Saturno – Arroyo Boleo 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 x 100 m NS were chosen.

16.3.2 UNIVARIATE STATISTICS OF DATA COMPOSITES

Univariate statistics and histograms of grade for copper, cobalt, and zinc for each manto are shown in Tables 31 to 33.

Coefficients of Variation (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 2.18 and 2.42 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 copper, cobalt, and zinc because:

- the copper resource is dominated by Manto 3 which has a CV well below 2
- cobalt and zinc CV values are uniformly low.

Table 31: Univariate Statistics of Assay Composites – Cu

Copper	Manto						
	0	1	2	3aa	3a	3	4
Mean	0.024	0.581	0.312	0.492	0.359	0.881	0.328
CV	1.784	2.176	2.42	1.471	1.99	1.447	2.043
Min	0.002	0.001	0.002	0.002	0.002	0.003	0.003
Q1	0.007	0.011	0.023	0.071	0.02	0.093	0.042
Median	0.011	0.044	0.094	0.185	0.075	0.368	0.131
Q3	0.020	0.483	0.280	0.606	0.359	1.195	0.330
Max		12.654	13.451	4.859	7.448	13.584	10.211
IQR	0.014	0.472	0.257	0.535	0.339	1.102	0.288
Data	476	799	1,915	161	1,858	4,542	1,850

CV = Coefficient of Variation, Q1,3 = Quartile Values, IQR = Interquartile range or midspread; Data = the number of data sets used in the statistical analysis

Table 32: Univariate Statistics of Assay Composites – Co

Cobalt	Manto 0	1	2	3aa	3a	3	4
Mean	0.009	0.038	0.044	0.072	0.065	0.068	0.03
CV	0.925	1.285	1.768	0.865	1.053	0.986	0.91
Min	0.001	0.001	0	0.007	0.002	0.002	0.002
Q1	0.005	0.007	0.013	0.022	0.022	0.027	0.014
Median	0.007	0.017	0.027	0.05	0.042	0.049	0.022
Q3	0.01	0.047	0.059	0.102	0.079	0.087	0.037
Max	0.079	0.475	2.823	0.281	0.625	1.51	0.464
IQR	0.005	0.04	0.046	0.08	0.057	0.06	0.023
Data	476	799	1,915	161	1,858	4,542	1,850

CV = Coefficient of Variation, Q1,3 = Quartile Values, IQR = Interquartile range or midspread; Data = the number of data sets used in the statistical analysis

Table 33: Univariate Statistics of Assay Composites – Zn

Zinc	Manto 0	1	2	3aa	3a	3	4
Mean	0.473	0.797	0.891	0.667	0.563	0.372	0.23
CV	0.867	1.712	1.194	0.96	0.805	1.312	1.271
Min	0.009	0.016	0.003	0.045	-1	0.012	0.002
Q1	0.171	0.257	0.31	0.238	0.263	0.186	0.11
Median	0.387	0.49	0.587	0.529	0.43	0.272	0.164
Q3	0.671	0.822	1.016	0.84	0.712	0.42	0.28
Max	3.32	20	9.28	4.987	5.112	13.761	9.395
IQR	0.5	0.565	0.706	0.602	0.449	0.234	0.17
Data	476	799	1,915	161	1,858	4,542	1,850

CV = Coefficient of Variation, Q1,3 = Quartile Values, IQR = Interquartile range or midspread; Data = the number of data sets used in the statistical analysis

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 55). 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 (Figure 55 to Figure 69).

Variograms were modelled with a relatively low nugget as determined from the well structured, down hole (Z direction) variograms. Variogram models for Mantos 3a, 3, and 4 are shown in Table 34. Note that no variograms were produced for 3aa due to lack of data, so the variograms for Manto 3a were used when estimating grade in Manto 3aa.

Figure 55: Manto 3 Variogram Maps

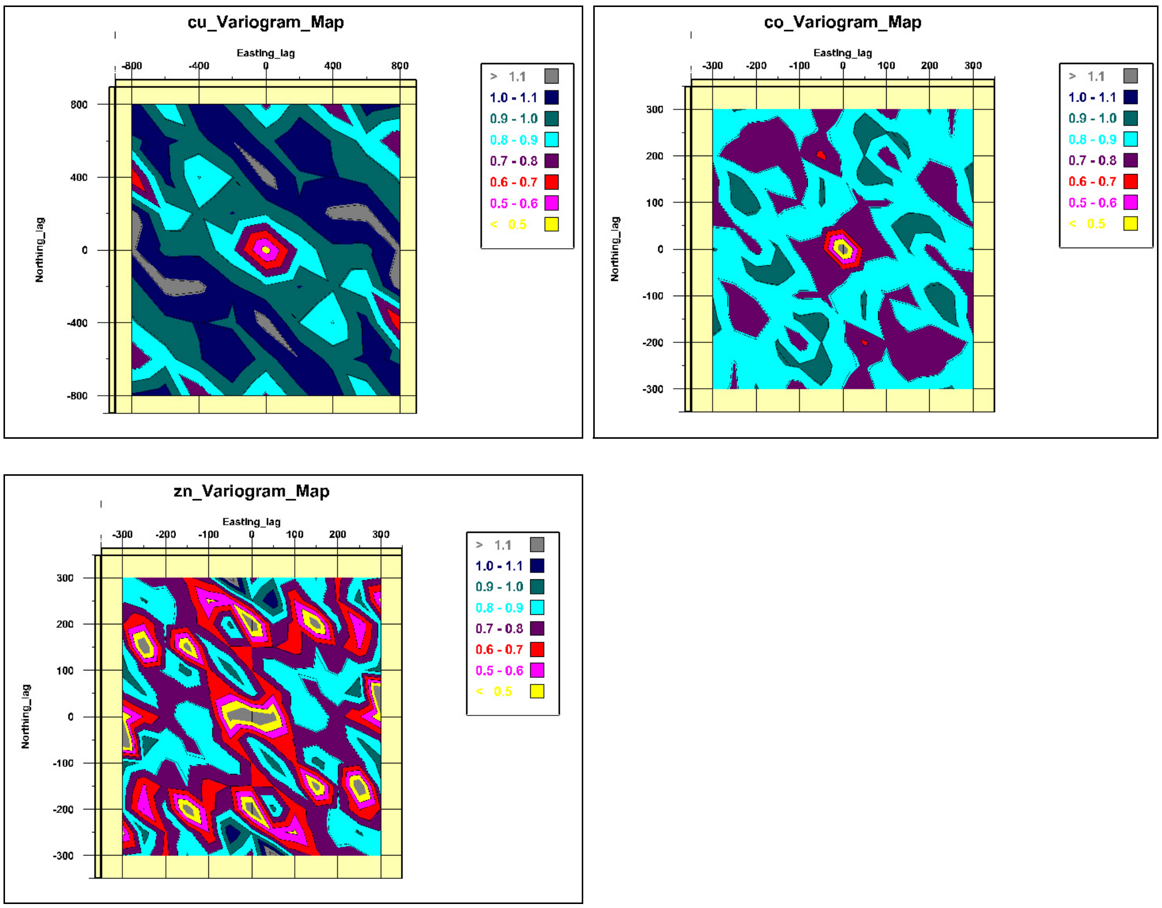


Figure 56: Variograms of Cu – Manto 3 (X, Y, Z directions)

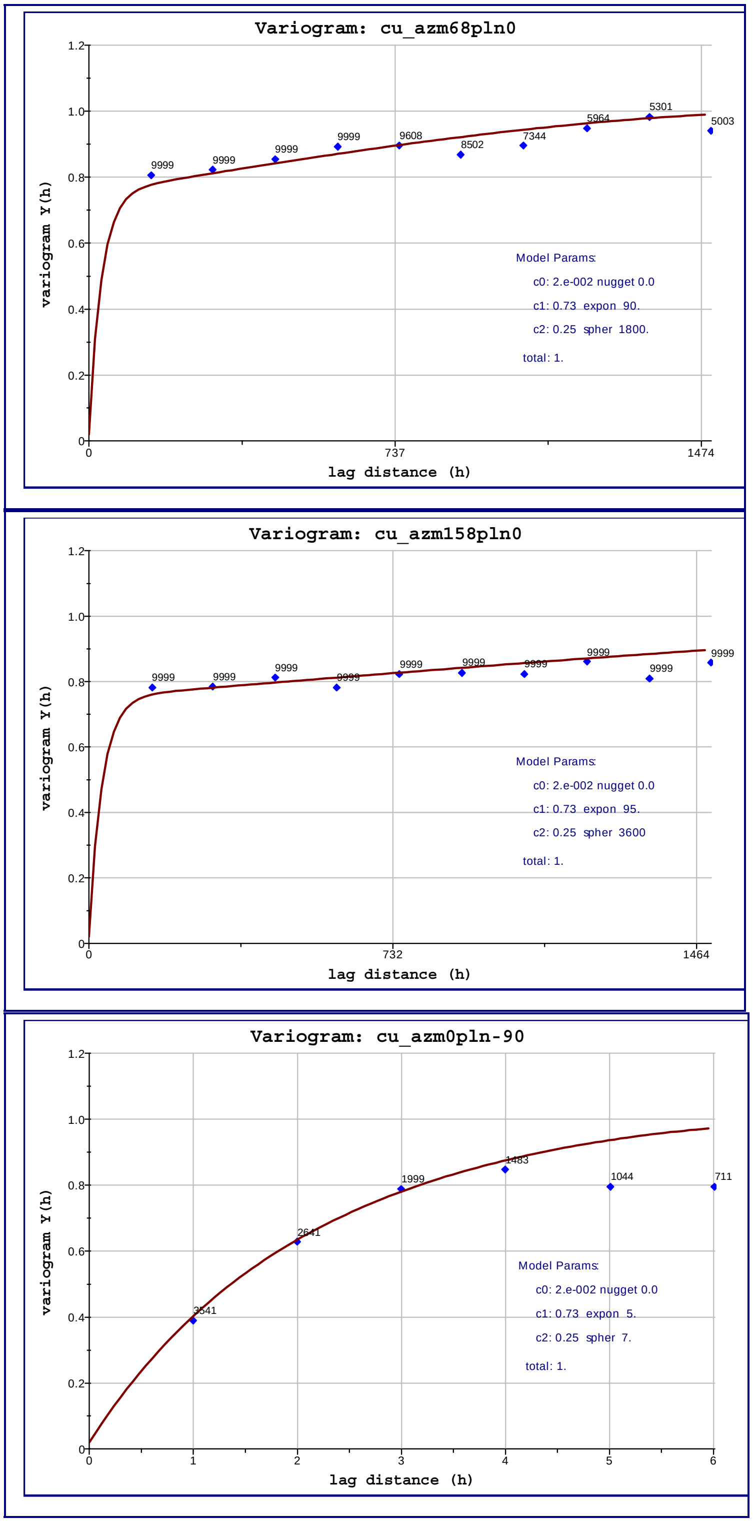


Figure 57: Variograms of Co – Manto 3 (X, Y, Z directions)

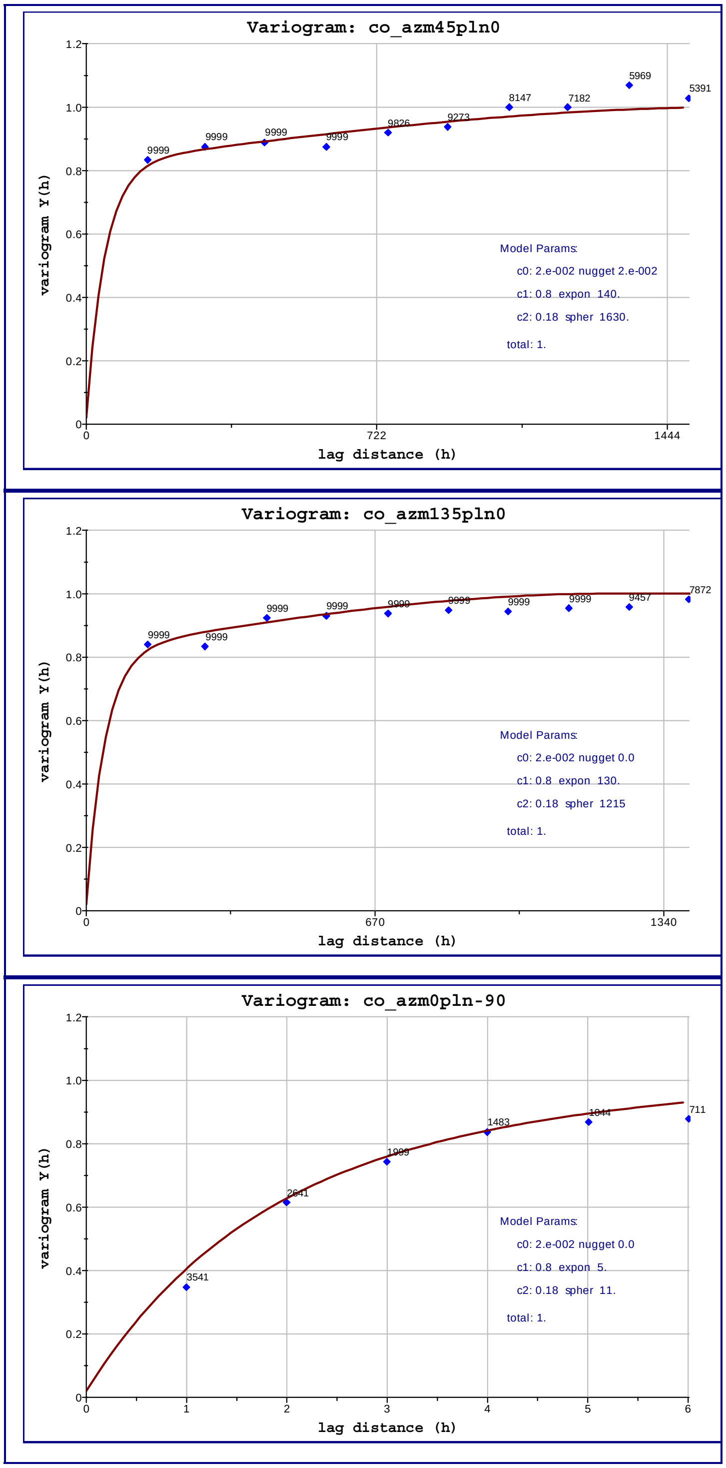
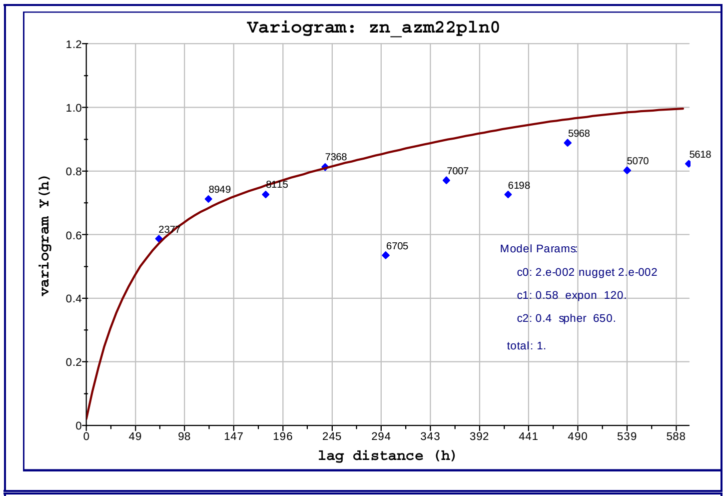
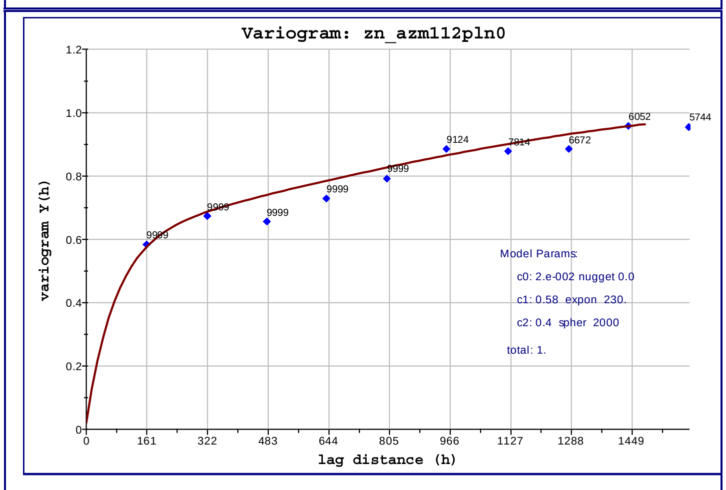


Figure 58: Variograms of Zn – Manto 3 (X, Y, Z directions)





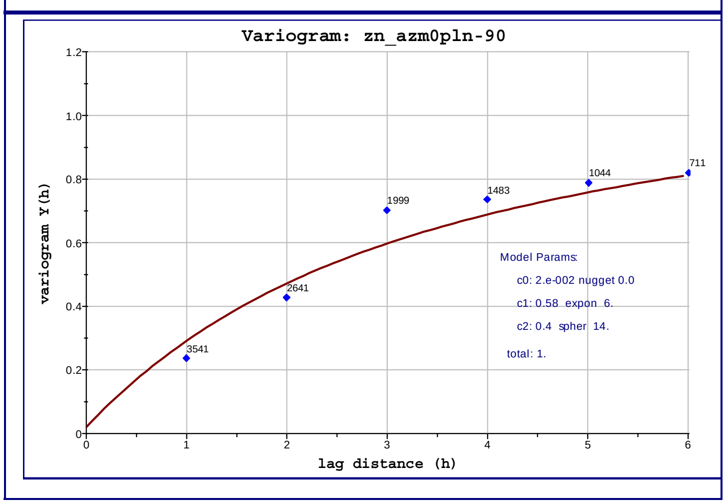


Table 34: Variogram Models

		Manto 0			Manto 1			Manto 2		
Metal	**Structure**	**Nugget**	**c1**	**c2**	**Nugget**	**c1**	**c2**	**Nugget**	**c1**	**c2**
Copper	type		exp	sph		exp	sph		exp	sph
	variance	0.05	0.7	0.25	0.05	0.65	0.3	0.05	0.8	0.15
	range - X	-	120	420	-	150	360	-	180	510
	range - Y	-	200	265	-	200	380	-	190	930
	range - Z	-	2	2.5	-	4	5	-	3.5	4
	azimuth	315	-	-	330	-	-	315	-	-
Cobalt	type		exp	sph		exp	sph		exp	sph
	variance	0.05	0.7	0.25	0.05	0.55	0.4	0.05	0.75	0.2
	range - X	-	140	430	-	170	540	-	120	250
	range - Y	-	120	570	-	340	400	-	100	700
	range - Z	-	6	9	-	3.5	10	-	3.5	4
	azimuth	315	-	-	330	-	-	315	-	-
Zinc	type		exp	sph		exp	sph		exp	sph
	variance	0.05	0.7	0.25	0.05	0.75	0.2	0.05	0.5	0.45
	range - X	-	170	260	-	160	190	-	240	460
	range - Y	-	280	340	-	340	400	-	100	525
	range - Z	-	4	5	-	5	5	-	5	9
	azimuth	330	-	-	330	-	-	315	-	-
		Manto 3a			**Manto 3**			**Manto 4**		
Metal	**Structure**	**Nugget**	**c1**	**c2**	**Nugget**	**c1**	**c2**	**Nugget**	**c1**	**c2**
Copper	type		exp	sph		exp	sph		exp	sph
	variance	0.05	0.64	0.31	0.02	0.73	0.25	0.05	0.72	0.23
	range - X	-	120	720	-	90	1800	-	120	600
	range - Y	-	150	1540	-	95	3600	-	250	760
	range - Z	-	3.5	9	-	5	7	-	6	7
	Azimuth	360	-	-	292	-	-	338	-	-
Cobalt	Type		exp	sph		exp	sph		exp	sph
	Variance	0.05	0.66	0.29	0.02	0.8	0.18	0.05	0.68	0.27
	range – X	-	210	1180	-	140	1630	-	280	500
	range – Y	-	145	2400	-	130	1215	-	110	740
	range – Z	-	4	5	-	5	11	-	7.5	18
	Azimuth	360	-	-	315	-	-	315	-	-
Zinc	Type		exp	sph		exp	sph		exp	sph
	Variance	0.05	0.84	0.11	0.02	0.58	0.4	0.05	0.65	0.3
	range – X	-	160	220	-	120	650	-	160	435
	range – Y	-	120	590	-	230	2000	-	290	375
	range – Z	-	4	5	-	6	14	-	6	45
	Azimuth	360	-	-	338	-	-	315	-	-

16.3.4 Search Parameters & Data Criteria

For each metal species, 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 and confidence of the block estimates (see Table 35).

Table 35: Search Parameters – 3D Block Models

Parameter	Manto 3a,3		
Pass	3	2	1
Search Radii (m)			
X – direction	200	400	400
Y – direction	250	500	500
Z – direction	2	2	4
Long axis azimuth	315		
Data Criteria			
Min Data	14	14	7
Octants	4	4	2
Max Data	32	32	32

16.3.5 MODEL CODING

The block models have been coded as follows:

- proportion of a block below the upper surface of the manto – ore percent item
- flag to specify whether a block is within the land held by the company – claim item
- 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.

16.3.6 RESOURCE CLASSIFICATION – 3D BLOCK MODELS

The resources have been classified based on the three 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.

16.3.7 ACCOUNTING FOR PAST MINING ACTIVITIES

Mantos 1 and 3 have been extensively mined in the past. The locations 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 with any confidence.

Ore was mined using a short wall 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 67 instances and a similar number of Retaque intervals have been identified. The estimation process has been carried out effectively ignoring this issue other than by treating voids as 'missing' or 'no data'.

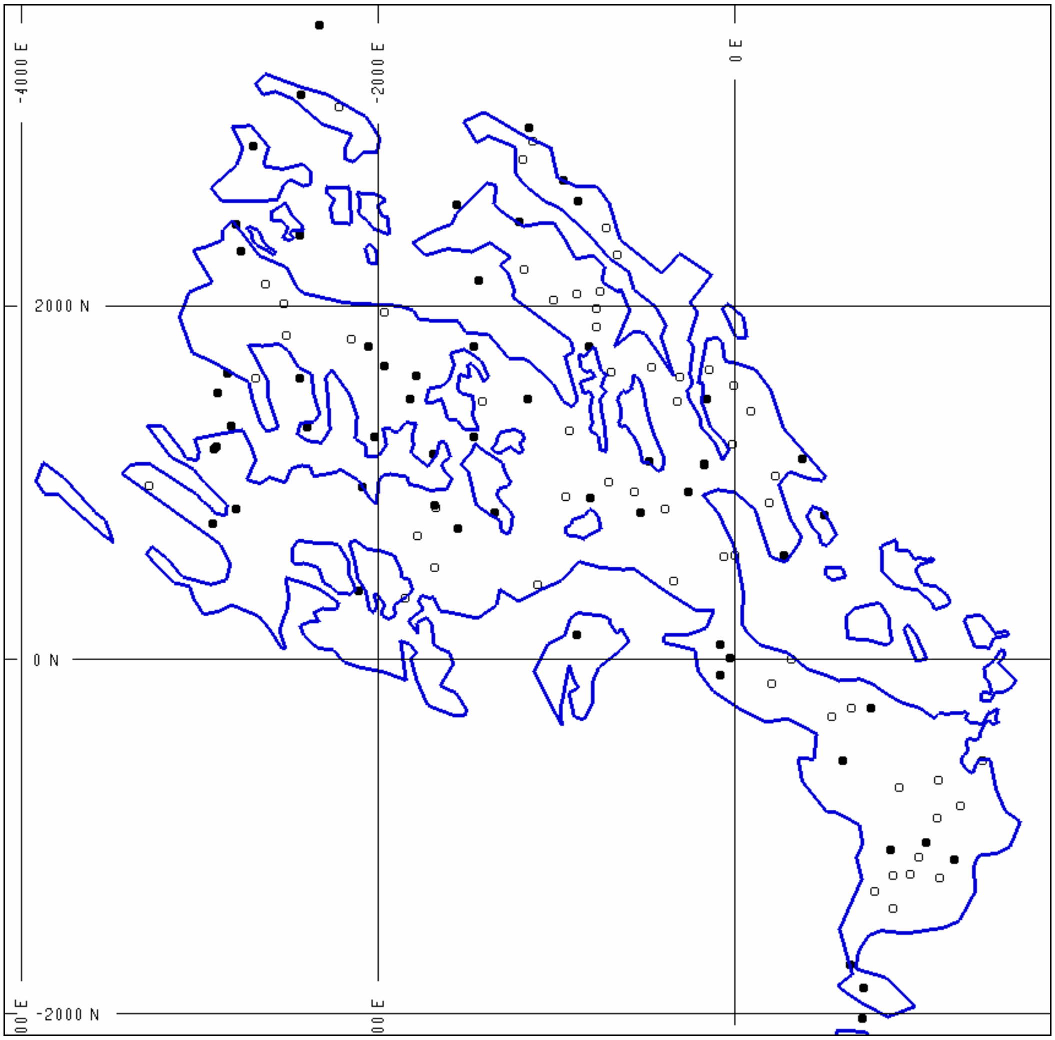
The resource, however, was modified to account for the previous mining activities. One way this could have been achieved was by simply removing an amount of ore equal to the known treated ore tonnage, 13.6 Mt. However, this may be overly conservative since many of the workings collapsed and the void 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. A number of previously mined areas were intersected during the 2005-2006 underground mining trial. In some areas where full closure was observed, it was difficult to distinguish between in-situ manto and "retaque". This "replaced" material was sampled and for the purposes of the DFS resource estimate, was considered as in-situ material while mined voids were accounted for as follows:

- digitizing a simplified footprint shape around the area of historic workings
- 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 443 m of drill hole intersection occur within the mined volume shape of which 12% are voids (see Figure 59). It is appropriate therefore, to factor the tonnes down by 12%.

Figure 59: Plan View of Manto 3 Old Mine Areas Showing Location of Drill Holes Which Encountered Voids (solid) and Retaque (open)



Drill hole information showed a smaller percentage of voids encountered in Manto 1 than Manto 3, but it was decided to reduce Manto 1 tonnage by the same 12% factor for the historically mined area.

16.3.8 RESOURCE ESTIMATES – 3D BLOCK MODELS

Grade estimation was completed by H&S using a specialist mining software package known as "MineSight 3D" (v 3.2), developed by Metech Pty. Ltd.

The Boleo 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 the following long-term average prices:

- copper – US$1.50/lb
- cobalt – US$15/lb
- zinc – US$1.20/lb.

The copper equivalent formula is: **CuEq%** = Cu% + 15 x Co%/1.50 + 1.20 x Zn%/1.50

(Note that manganese is not included in the copper equivalency formula as the production of a manganese product such as a carbonate or a sulphate is being handled as an 'opportunity' for the purposes of the feasibility study.)

The 3D block model resources are reported in Table 36 to Table 38.

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

0.5% CuEq Cutoff	Manto	0	1	2	3AA	3A	3	4
Measured	Tonnes (10^6)	2.3	4.6	9.0	0.8	10.0	44.6	3.3
	CuEq%	0.63	2.76	2.04	2.10	1.94	2.17	1.64
	Cu%	0.03	1.51	0.65	0.51	0.43	1.10	0.85
	Co%	0.010	0.069	0.064	0.083	0.098	0.082	0.050
	Zn%	0.62	0.71	0.94	0.96	0.66	0.32	0.35
Indicated	Tonnes (10^6)	7.5	14.9	52.9	2.4	36.4	71.4	17.0
	CuEq%	0.71	2.08	1.60	1.95	1.36	1.99	1.00
	Cu%	0.03	0.79	0.32	0.53	0.37	1.04	0.44
	Co%	0.010	0.050	0.048	0.082	0.053	0.057	0.035
	Zn%	0.73	0.98	0.99	0.76	0.59	0.48	0.26
Total	Tonnes (10^6)	9.9	19.5	61.9	3.2	46.3	116.0	20.4
	CuEq%	0.69	2.24	1.66	1.99	1.49	2.06	1.11
	Cu%	0.03	0.96	0.37	0.53	0.38	1.06	0.51
	Co%	0.010	0.055	0.050	0.082	0.063	0.066	0.038
	Zn%	0.71	0.92	0.98	0.81	0.61	0.42	0.27

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

1.0% CuEq Cutoff	Manto	0	1	2	3AA	3A	3	4
Measured	Tonnes (10^6)	0.04	4.4	7.8	0.8	8.6	39.8	2.0
	CuEq%	1.09	2.84	2.23	2.13	2.12	2.33	2.29
	Cu%	0.05	1.57	0.73	0.51	0.49	1.20	1.26
	Co%	0.012	0.071	0.071	0.084	0.107	0.086	0.066
	Zn%	1.15	0.71	1.00	0.97	0.69	0.33	0.46
Indicated	Tonnes (10^6)	0.9	12.3	42	2.2	26.0	61.0	6.3
	CuEq%	1.12	2.36	1.81	2.06	1.59	2.20	1.52
	Cu%	0.03	0.94	0.37	0.57	0.46	1.19	0.74
	Co%	0.011	0.057	0.054	0.086	0.061	0.060	0.048
	Zn%	1.23	1.06	1.12	0.79	0.65	0.52	0.37
Total	Tonnes (10^6)	0.9	16.7	49.8	3.0	34.6	100.8	8.3
	CuEq%	1.11	2.49	1.88	2.08	1.72	2.25	1.70
	Cu%	0.03	1.11	0.43	0.55	0.47	1.19	0.86
	Co%	0.011	0.061	0.057	0.086	0.073	0.07	0.052
	Zn%	1.23	0.97	1.10	0.84	0.66	0.44	0.39

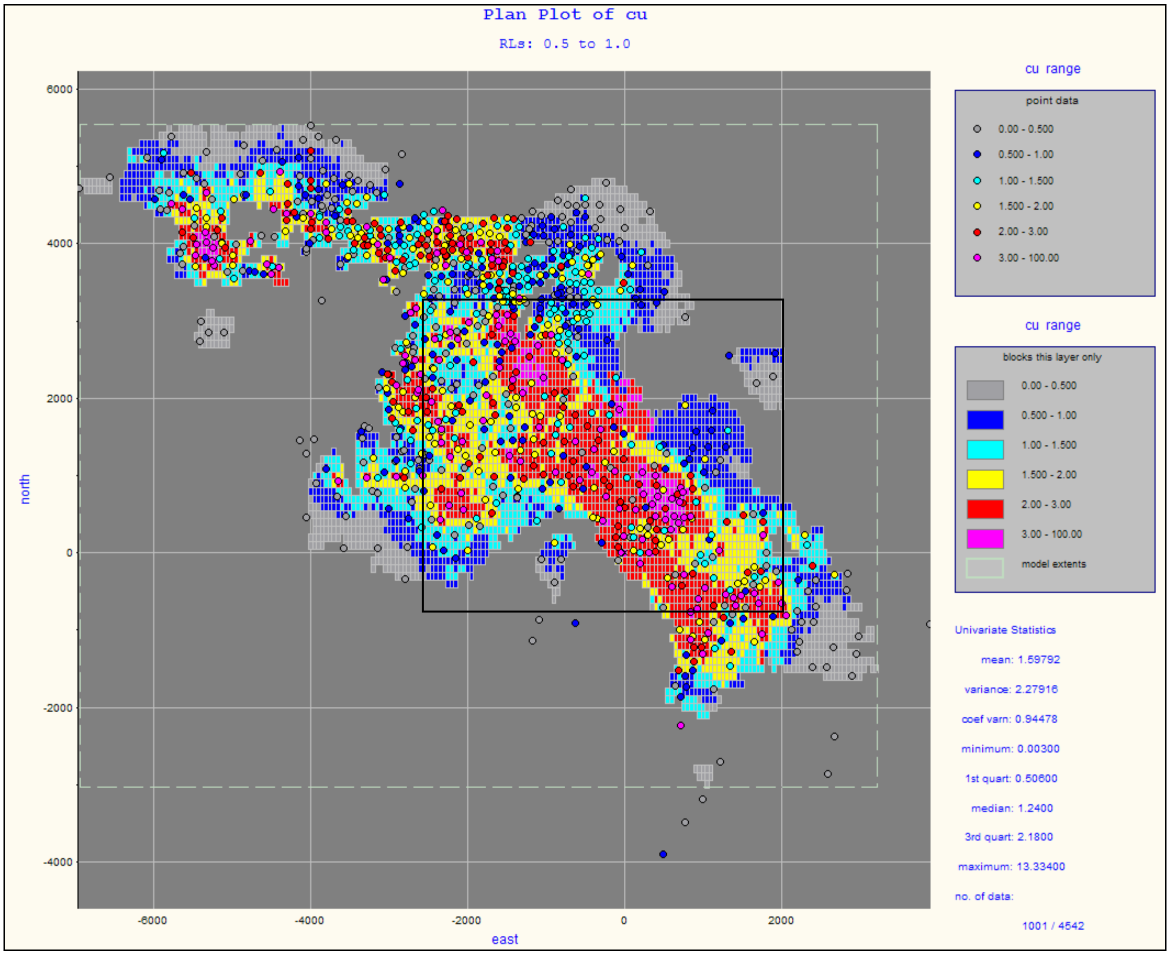
Table 38: Inferred Resources

Cutoff	Manto	0	1	2	3AA	3A	3	4
0.5% CuEq	Tonnes (10^6)	7.4	49.2	52.5	0.6	23.8	56.4	63.3
	CuEq%	0.66	1.72	1.27	1.45	1.20	1.48	0.90
	Cu%	0.02	0.48	0.22	0.51	0.30	0.54	0.39
	Co%	0.008	0.044	0.042	0.053	0.044	0.047	0.032
	Zn%	0.69	1.00	0.79	0.51	0.58	0.59	0.25
1.0% CuEq	Tonnes (10^6)	0.3	34.9	34.2	0.4	13.6	40.3	18.2
	CuEq%	1.06	2.13	1.54	1.86	1.49	1.75	1.39
	Cu%	0.02	0.65	0.25	0.71	0.43	0.69	0.72
	Co%	0.008	0.056	0.049	0.067	0.055	0.053	0.042
	Zn%	1.20	1.15	0.99	0.60	0.65	0.67	0.31

16.3.9 MODEL VERIFICATION

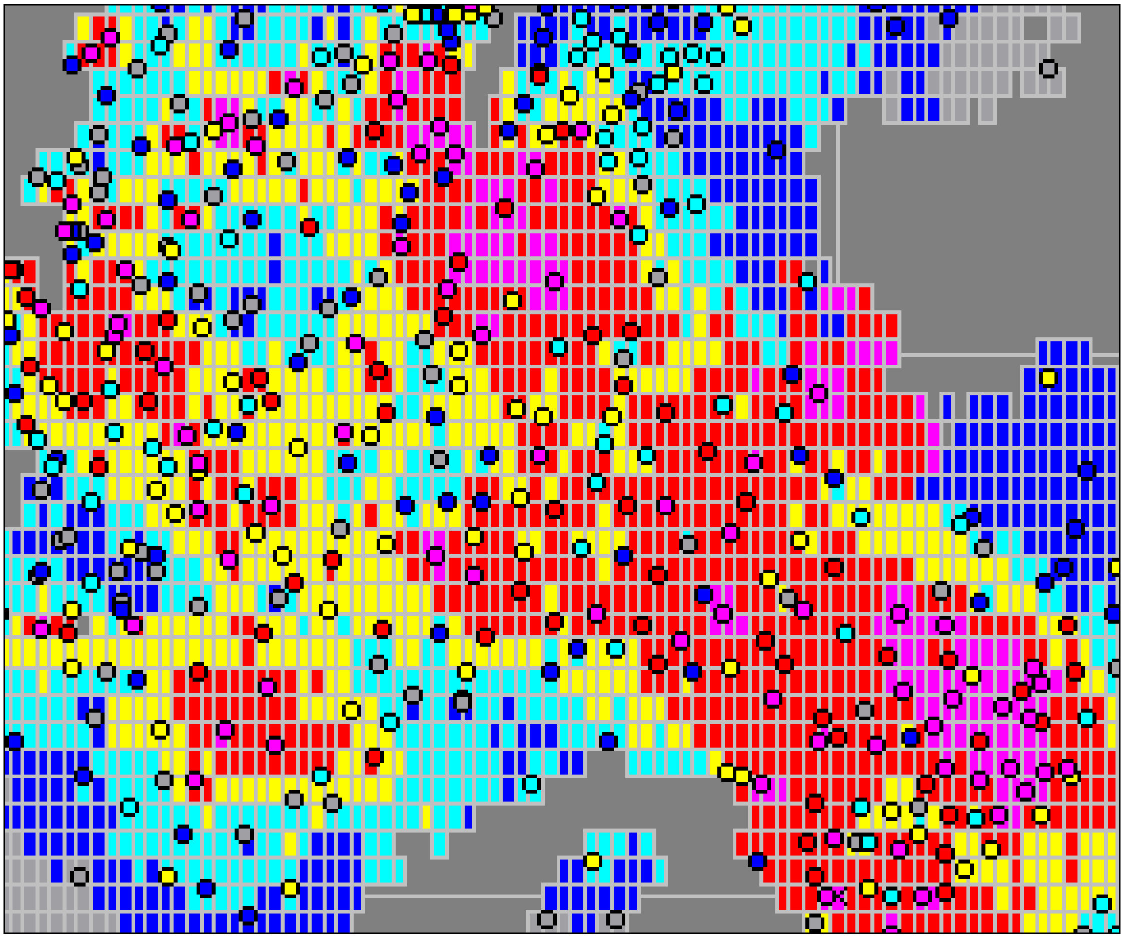
The model grade estimates have been verified by H&S by plotting block grades against assay composite grades, in plan, for different levels of the model. Plans of the basal level of Manto 3 are shown for copper, cobalt, and zinc (Figure 60 to Figure 65).

Figure 60: Plan View of Manto 3 Copper Distribution Block Model with Assay Composites



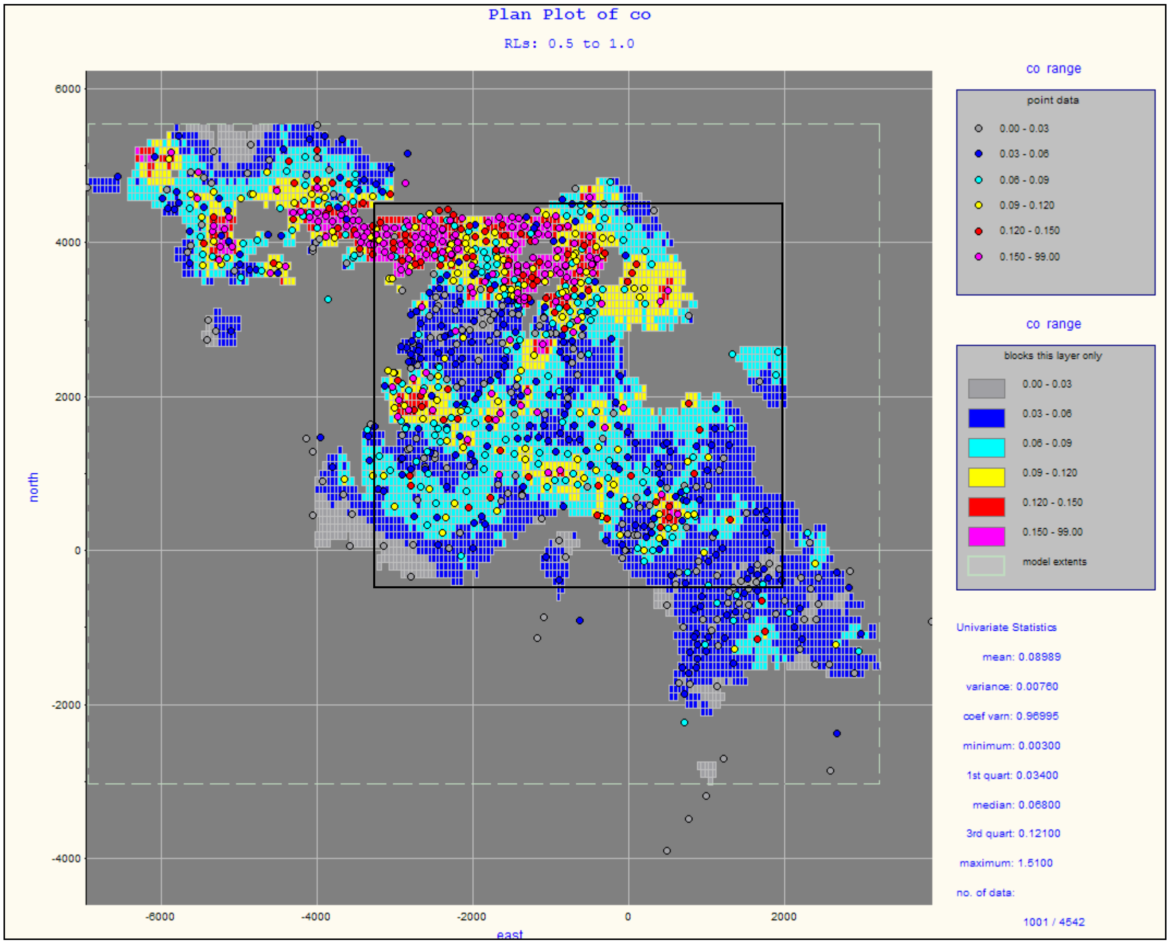
Notes: Total Resource (basal layer 0 m to 1 m). Box highlights area shown in detail in Figure 61.

Figure 61: Detail of Manto 3 Copper Block Model with Assay Composites



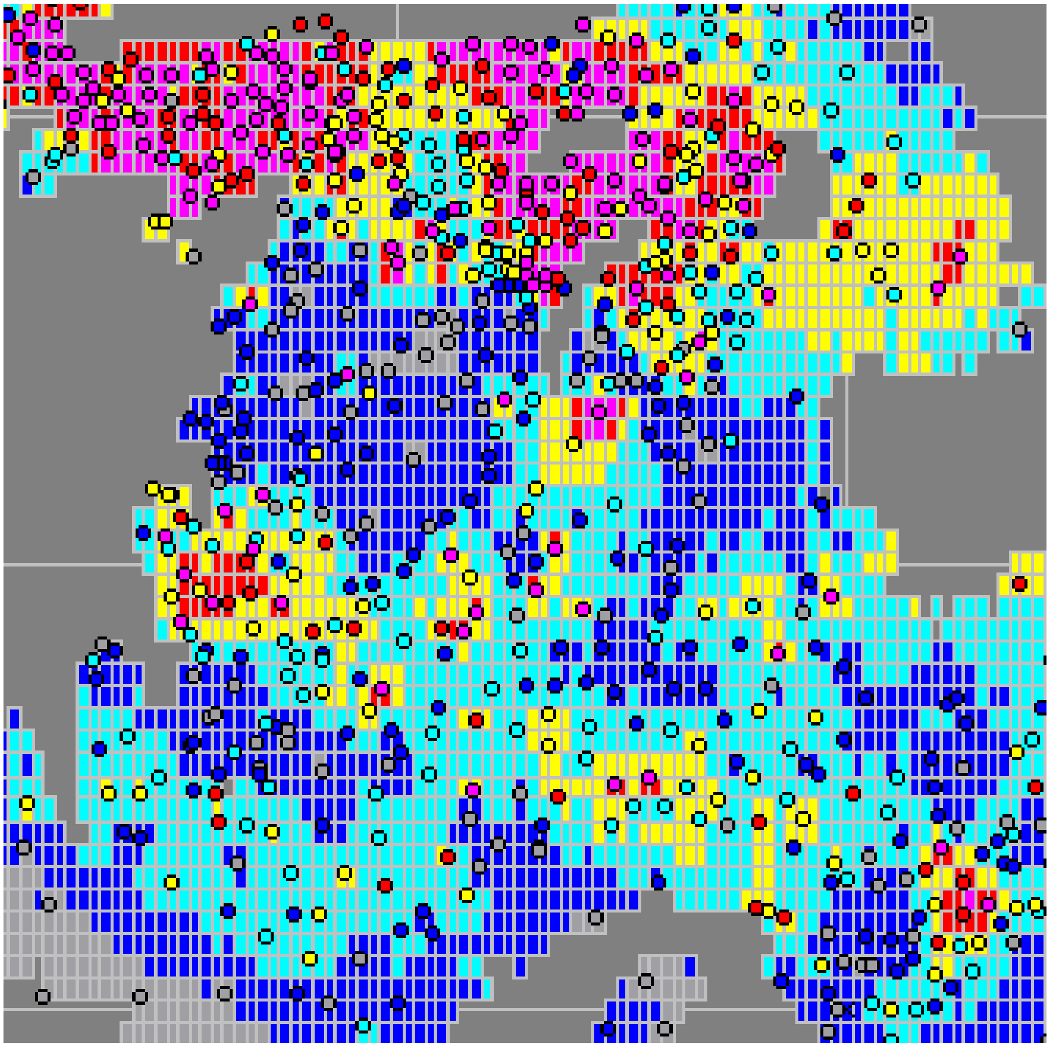
Note: the legend for Figure 61 is as per Figure 60.

Figure 62: Plan View of Manto 3 Cobalt Distribution Block Model with Assay Composites



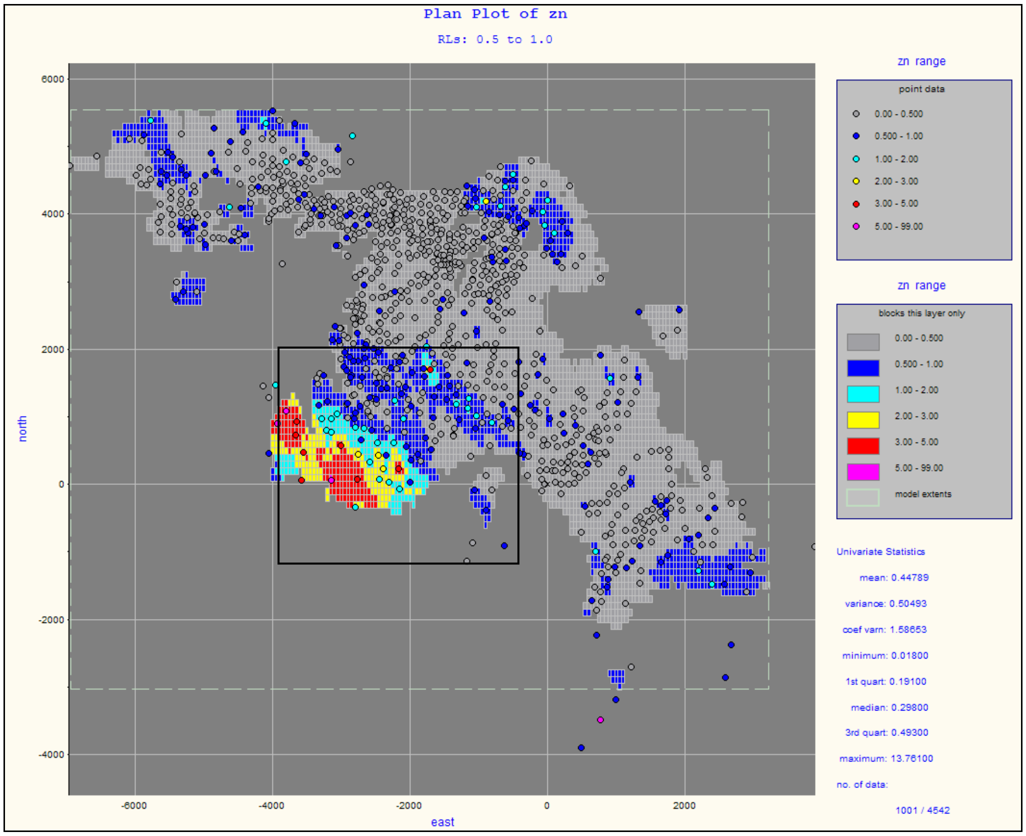
Notes: Total Resource (basal layer 0 m to 1 m). Box highlights area shown in detail in Figure 63.

Figure 63: Detail of Manto 3 Cobalt – Block Model with Assay Composites



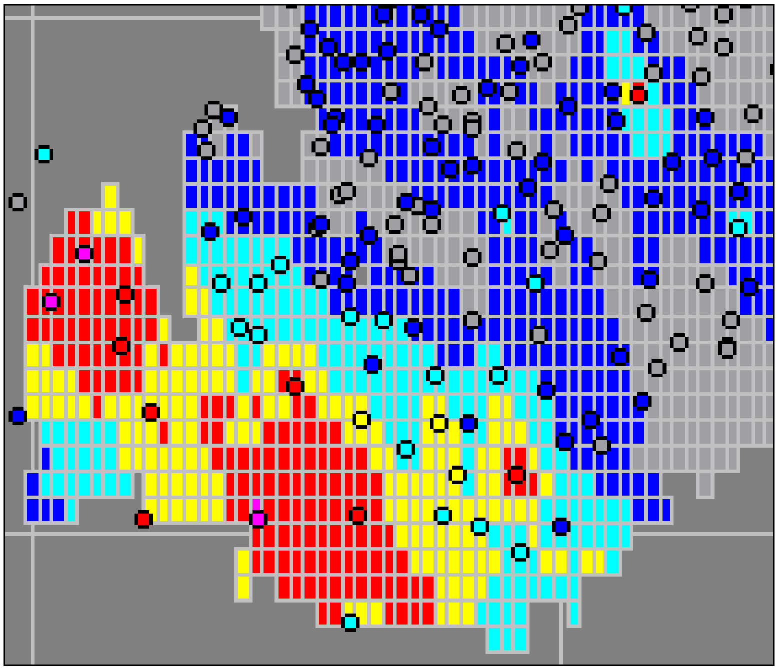
Note: the legend for Figure 63 is as per Figure 62.

Figure 64: Plan View of Manto 3 Zinc distribution Block Model with Assay Composites



Notes: Total Resource (basal layer 0 m to 1 m). Box highlights area shown in detail in Figure 65.

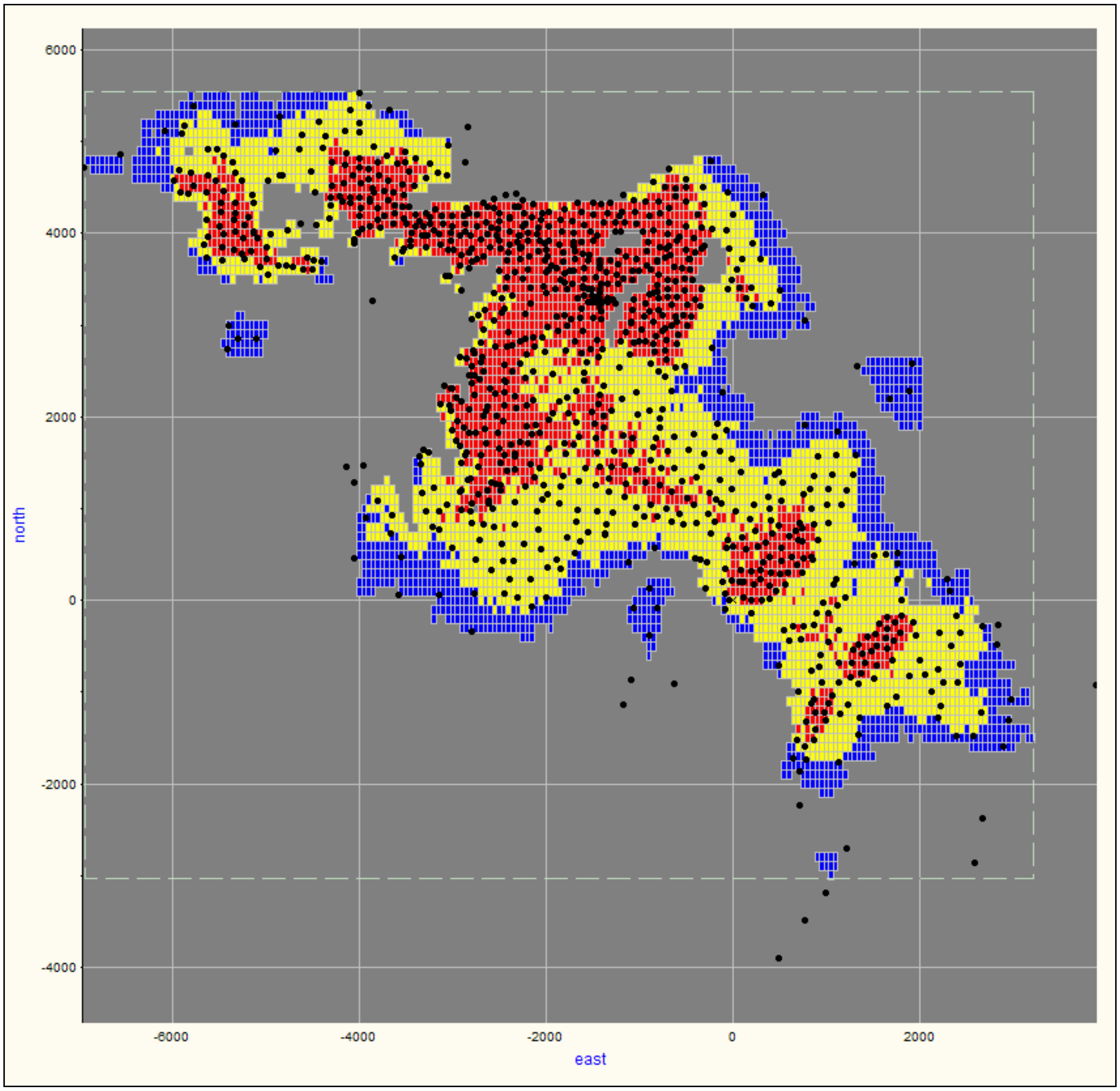
Figure 65: Detail of Manto 3 Zinc – Block Model with Assay Composites



Note: the legend for Figure 65 is as per Figure 64.

The appropriateness of the resource classification has been assessed by plotting the classification against the one-metre data composite density (Figure 66). 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 66: Plan View of Manto 3 Resource Classification – Block Model with Drill Hole Locations



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

16.4 Seam Models

Seam models used for underground mine design have been produced for all Mantos except Manto 0.

16.4.1 COMPOSITE LENGTH & BLOCK DIMENSIONS

The type of 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 that bears no relation to the anticipated mining method. Therefore, geological composites have not been used.

Definition of composites by grade alone can be problematical. There is not an 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 manto floor and many intervals would include a certain amount of internal waste. Both of these phenomena show that the lateral continuity of grade seams cannot be demonstrated. Variogram analysis, which shows relatively short ranges, also supports this notion. 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 Boleo have determined a minimum mining 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 0.3 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. Material greater than 4.2 m above the Manto floor is ignored and not used in the resource estimates.

The grade threshold used to decide on the inclusion of additional increments was 0.5% Cu.

Model block dimensions were the same in terms of X (50 m) and Y (100 m) 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, 2, 3AA and 3 that only the minimum 1.8 m seams were appropriate. Consequently only minimum thickness resource estimates were produced. Variable seam models were produced for Manto 3 and 4.

16.4.2 UNIVARIATE STATISTICS OF DATA COMPOSITES

Univariate statistics of copper, cobalt, and zinc are shown in Tables 39 to 41. Manto 1 has been separated into two geological domains (D1, D2) based on interpretation of paleo 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 mean values of the seam composites are higher than the individual 1m composites and the coefficients of variation are lower.

Table 39: Univariate Statistics of Seam Composites – Copper

	Manto						
Copper	1 Dom1	1 Dom2	2	3aa	3a	3	4
Mean	2.179	0.641	0.443	0.598	0.38	1.452	0.474
CV	0.814	1.494	1.862	1.058	1.594	0.878	1.724
min	0.047	0.004	0	0.004	0	0.004	0.003
Q1	0.927	0.068	0.063	0.123	0.035	0.553	0.055
Median	1.757	0.237	0.17	0.394	0.121	1.191	0.201
Q3	2.559	0.881	0.425	0.877	0.429	1.967	0.44
Max	8.744	4.928	7.585	2.977	4.59	11.241	5.245
IQR	1.632	0.813	0.362	0.754	0.394	1.414	0.385
Data	68	82	556	59	702	990	195

Table 40: Univariate Statistics of Seam Composites – Cobalt

	Manto						
Cobalt	1 Dom1	1 Dom2	2	3aa	3a	3	4
Mean	0.087	0.056	0.056	0.086	0.067	0.086	0.032
CV	0.541	1.084	1.501	0.646	0.927	0.819	1.018
min	0.001	0.001	0	0.008	0	0.002	0.002
Q1	0.054	0.013	0.018	0.031	0.024	0.035	0.014
Median	0.081	0.032	0.042	0.067	0.047	0.069	0.023
Q3	0.113	0.079	0.074	0.119	0.086	0.12	0.037
Max	0.201	0.372	1.683	0.244	0.482	0.862	0.317
IQR	0.059	0.066	0.056	0.088	0.062	0.085	0.023
Data	68	82	556	59	702	990	195

Table 41: Univariate Statistics of Seam Composites – Zinc

	Manto						
Zinc	1 Dom1	1 Dom2	2	3aa	3a	3	4
Mean	0.623	1.129	1.037	0.741	0.596	0.431	0.247
CV	0.656	1.56	0.995	0.766	0.661	1.341	0.984
min	0.021	0.035	0	0.013	0	0.008	0.001
Q1	0.322	0.466	0.461	0.383	0.321	0.201	0.095
Median	0.534	0.655	0.72	0.61	0.497	0.302	0.175
Q3	0.794	1.075	1.186	0.886	0.776	0.481	0.306
Max	2.446	13.35	7.628	3.152	3.685	9.094	2.029
IQR	0.472	0.609	0.725	0.503	0.455	0.28	0.211
Data	68	82	556	59	702	990	195

16.4.3 SPATIAL CONTINUITY OF GRADE

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

Variograms of seam composites for Manto 3 are shown in Figure 67 to Figure 69 for copper, cobalt, and zinc.

Figure 67: Variograms of Copper – Manto 3 (X, Y, directions)

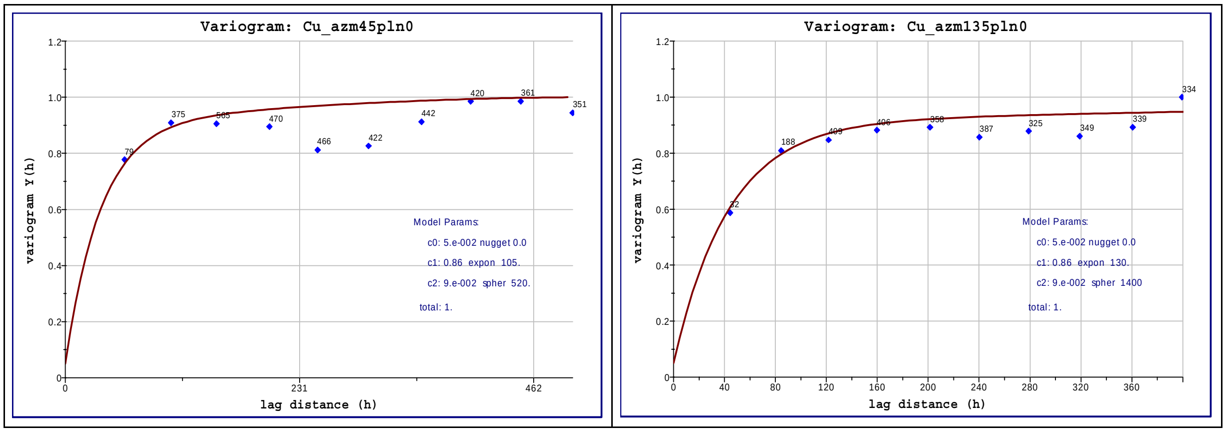


Figure 68: Variograms of Cobalt – Manto 3 (X, Y, directions)

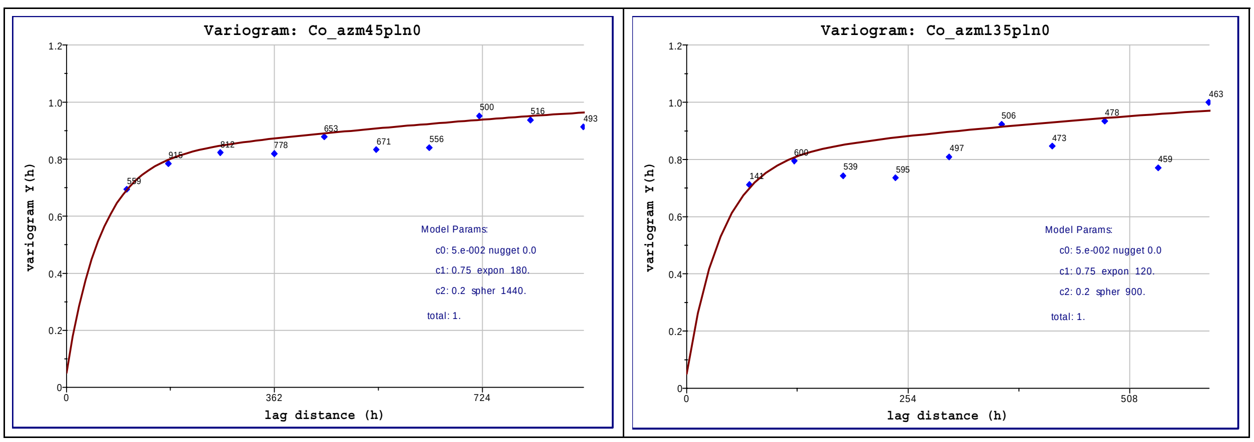


Figure 69: Variograms of Zinc – Manto 3 (X, Y, directions)

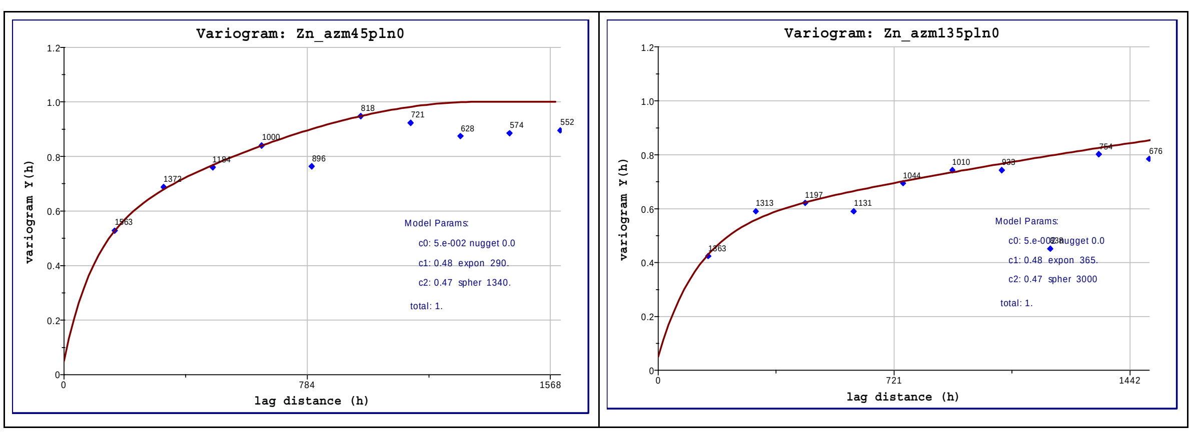


Table 42 shows the full variogram models.

Table 42: Variogram Models – Seam Composites

		Manto 1			Manto 2			Manto 3A			Manto 3		
Metal	**Structure**	**C0**	**c1**	**c2**	**C0**	**c1**	**c2**	**C0**	**c1**	**c2**	**C0**	**c1**	**c2**
Copper	type		exp	sph		exp	sph		exp	sph		exp	sph
	variance	0.02	0.65	0.33	0.05	0.65	0.3	0.05	0.54	0.41	0.05	0.86	0.09
	range - X	-	260	380	-	220	380	-	150	960	-	105	520
	range - Y	-	180	1200	-	190	620	-	220	835	-	130	1400
	range - Z	-	5	5	-	5	5	-	5	5	-	5	5
	azimuth	330	-	-	315	-	-	315	-	-	315	-	-
Cobalt	type		exp	sph		exp	sph		exp	sph		exp	sph
	variance	0.02	0.2	0.78	0.05	0.85	0.1	0.05	0.5	0.45	0.05	0.75	0.2
	range - X	-	280	770	-	110	460	-	170	1410	-	180	1440
	range - Y	-	180	800	-	200	260	-	160	2080	-	120	900
	range - Z	-	5	5	-	5	5	-	5	5	-	5	5
	azimuth	315	-	-	315	-	-	315	-	-	315	-	-
Zinc	type		exp	sph		exp	sph		exp	sph		exp	sph
	variance	0.1	0.26	0.64	0.05	0.66	0.29	0.05	0.48	0.47	0.05	0.48	0.47
	range - X	-	160	1000	-	270	420	-	290	1340	-	290	1340
	range - Y	-	230	370	-	120	730	-	365	3000	-	365	3000
	range - Z	-	5	5	-	5	5	-	5	5	-	5	5
	azimuth	315	-	-	315	-	-	315	-	-	315	-	-

Due to the lack of data no variograms were produced for manto 3AA. Whilst for Manto 4, despite a reasonable number of data composites in total, the mineralised portion of the manto is of very limited extent incorporating very few data and similarly no variograms were produced. For both these two mantos grade estimation was by Inverse Distance Squared (ID2), rather than Ordinary Kriging.

16.4.4 SEARCH PARAMETERS & DATA CRITERIA

The same search strategy with three 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 1 m composites for each hole (Table 43).

Table 43: Search Parameters – Current 3D Seam Models

Parameter	Manto 1, 2, 3		
Pass	**3**	**2**	**1**
Search Radii (m)			
X – direction	200	400	400
Y – direction	250	500	500
Long axis azimuth	315		
Data Criteria			
Min Data	4	4	2
Octants	4	4	2
Max Data	8	8	8

16.4.5 Resource Classification – Seam Models

As it was necessary to produce a block model for open cut mine planning and seam models for underground mine planning and as these two model types were constructed using different estimation techniques and parameters, the seam model resource classification differs slightly from the block model classification.

The seam 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.

16.4.6 Resource Estimates – Seam Models

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

Resource estimates are quoted at the 0.5% and 1.0% CuEq Cutoff grades (Table 44 to Table 46).

- **CuEq%** = Cu% + 15 x Co%/1.50 + 1.20 x Zn%/1.50

The seam models **are not** additional to the 3D block models discussed previously they are effectively sub-sets of these models.

Table 44: Seam Models – Measured and Indicated Resource at 0.50% CuEq

0.5% CuEq Cutoff	Manto	1	2	3AA	3A	3	4
Seam Height (m)		1.8	1.8	1.8	1.8	1.8 - 4.2	1.8 - 4.2
Measured	Tonnes (10^6)	4.0	9.2	-	14.9	31.9	1.1
	CuEq%	2.89	2.18	-	1.55	2.77	2.91
	Cu%	1.59	0.67	-	0.40	1.59	1.83
	Co%	0.072	0.067	-	0.069	0.087	0.076
	Zn%	0.73	1.05	-	0.58	0.38	0.39
Indicated	Tonnes (10^6)	6.8	21.2	2.9	18.7	31.1	5.1
	CuEq%	2.76	1.87	2.07	1.33	2.49	1.26
	Cu%	1.23	0.42	0.58	0.36	1.37	0.66
	Co%	0.071	0.058	0.082	0.052	0.065	0.037
	Zn%	1.02	1.08	0.84	0.57	0.59	0.28
Total	Tonnes (10^6)	10.8	30.5	2.9	33.7	62.9	6.2
	CuEq%	2.81	1.96	2.07	1.43	2.63	1.55
	Cu%	1.37	0.50	0.58	0.38	1.48	0.87
	Co%	0.071	0.060	0.082	0.059	0.076	0.044
	Zn%	0.91	1.07	0.84	0.58	0.49	0.30

Table 45: Seam Models – Measured and Indicated Resource at 1.0% CuEq

1.0% CuEq Cutoff	Manto	1	2	3AA	3A	3	4
Seam Height (m)		1.8	1.8	1.8	1.8	1.8 - 4.2	1.8 - 4.2
Measured	Tonnes (10^6)	3.8	8.3		10.9	31.6	0.7
	CuEq%	3.00	2.34		1.83	2.78	3.95
	Cu%	1.66	0.73		0.48	1.60	2.55
	Co%	0.075	0.072		0.084	0.087	0.099
	Zn%	0.73	1.11		0.64	0.38	0.51
Indicated	Tonnes (10^6)	6.3	18.8	2.5	12.8	30.3	1.9
	CuEq%	2.91	2.00	2.28	1.57	2.54	2.23
	Cu%	1.32	0.46	0.65	0.46	1.40	1.29
	Co%	0.075	0.062	0.089	0.062	0.066	0.058
	Zn%	1.05	1.15	0.93	0.62	0.60	0.44
Total	Tonnes (10^6)	10.1	27.1	2.5	23.7	61.9	2.6
	CuEq%	2.95	2.10	2.28	1.69	2.66	2.71
	Cu%	1.45	0.54	0.65	0.47	1.50	1.65
	Co%	0.075	0.065	0.089	0.072	0.077	0.070
	Zn%	0.93	1.14	0.93	0.63	0.49	0.46

Table 46: Seam Models – Inferred Resource

Cutoff	Manto	1	2	3A	3A	3	4
Seam Height (m)		1.8	1.8	1.8	1.8	1.8 - 4.2	1.8 - 4.2
0.5% CuEq.	Tonnes (10^6)	8.0	21.8	0.2	18.5	23.5	13.6
	CuEq%	2.38	1.72	2.80	1.31	2.03	1.05
	Cu%	0.65	0.28	1.40	0.29	0.87	0.55
	Co%	0.053	0.053	0.084	0.049	0.057	0.032
	Zn%	1.49	1.14	0.70	0.66	0.74	0.22
1.0% CuEq.	Tonnes (10^6)	6.9	18.7	0.1	13.6	20.9	5.4
	CuEq%	2.63	1.88	3.12	1.49	2.17	1.58
	Cu%	0.73	0.31	1.61	0.34	0.96	0.92
	Co%	0.060	0.058	0.090	0.057	0.060	0.044
	Zn%	1.62	1.24	0.77	0.73	0.78	0.28

16.4.7 MODEL VERIFICATION

The model grade estimates have been verified by plotting block grades against assay composite grades in plan. Plan views of Manto 1, 2 and 3 are shown for copper, cobalt, and zinc and resource classification (Figure 70 to Figure 81).

Figure 70: Plan View of Manto 1 Copper Distribution – Seam Model with Seam Composites

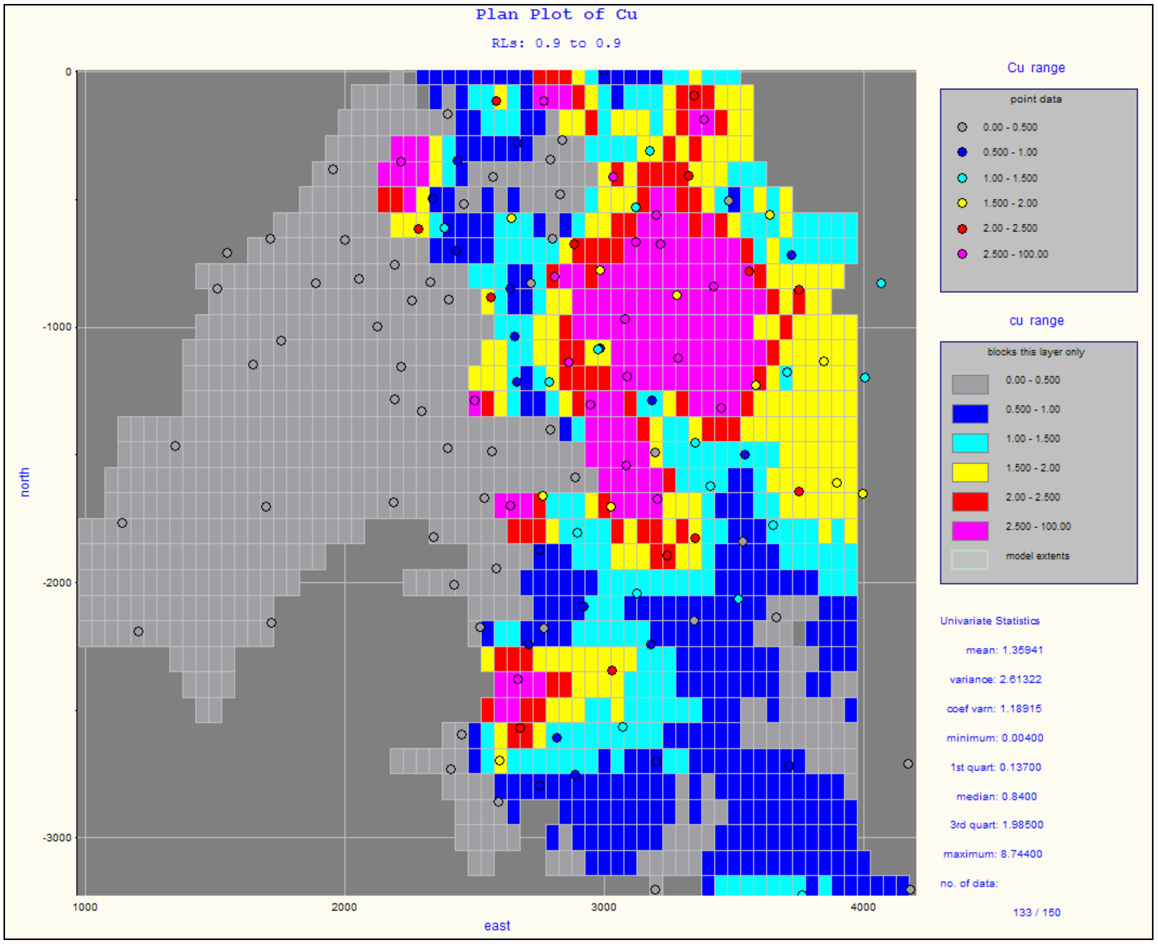


Figure 71: Plan View of Manto 1 Cobalt Distribution - Seam Model with Seam Composites

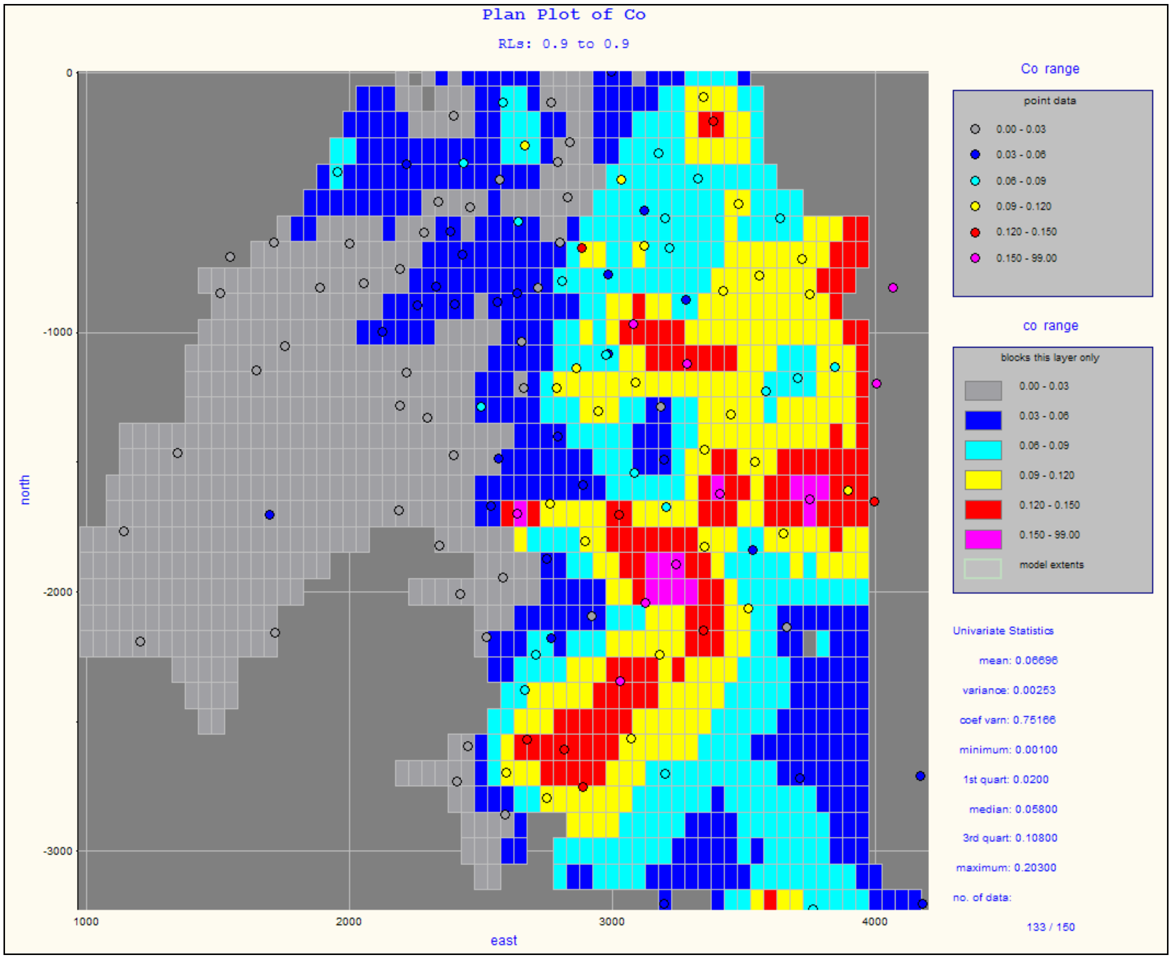


Figure 72: Plan View of Manto 1 Zinc Distribution – Seam Model with Seam Composites

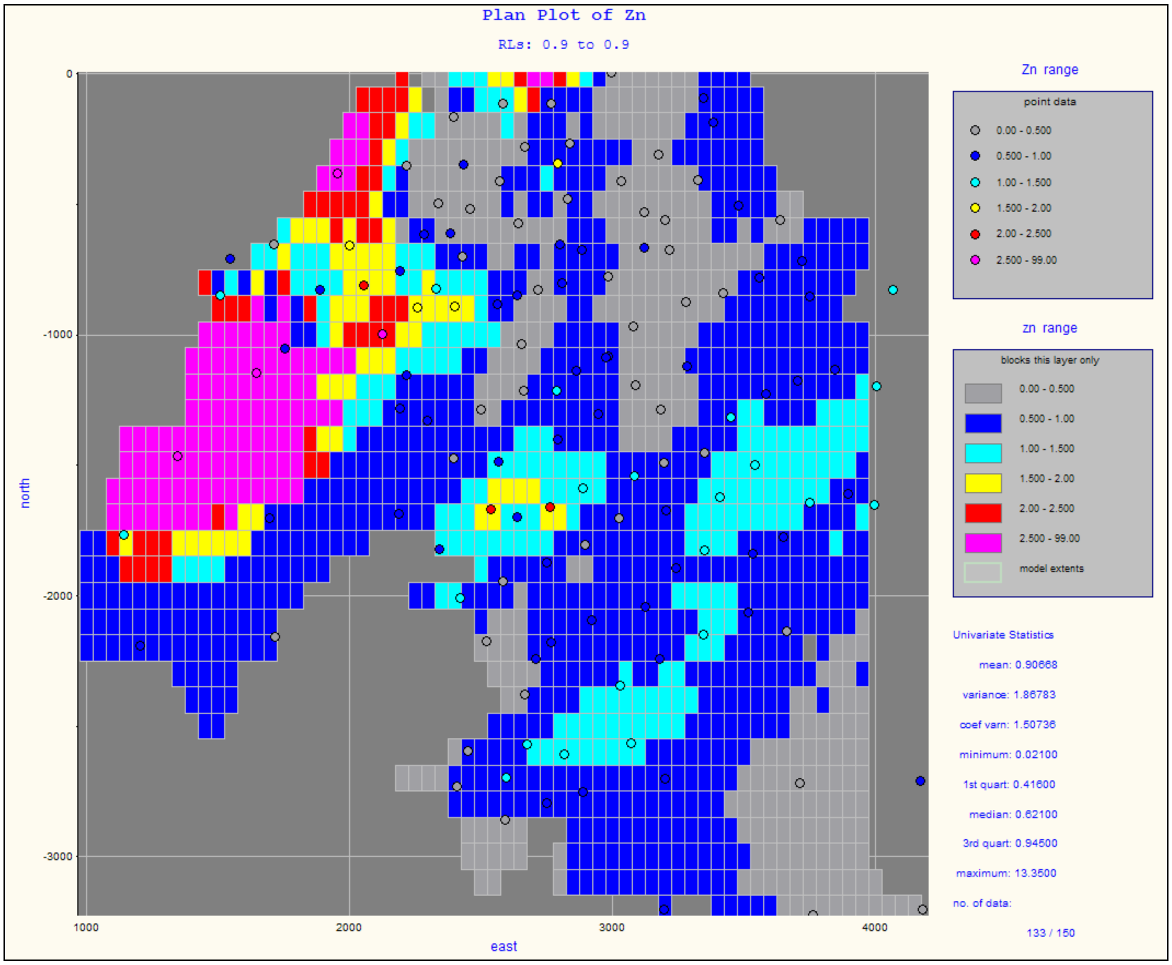
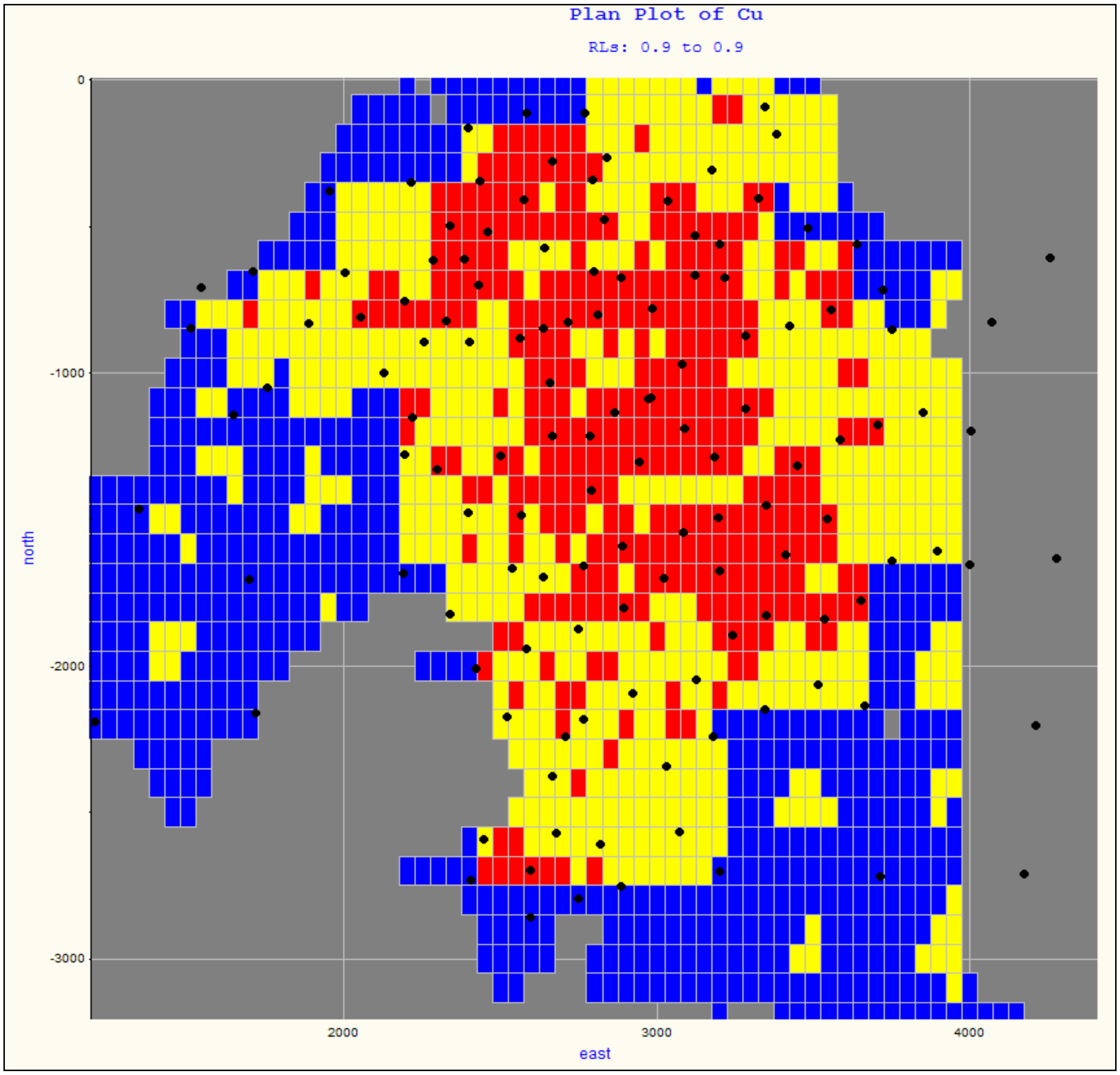


Figure 73: Plan View of Manto 1 Resource Classification – Seam Model with Drill Hole Locations



Note: Measured = Red, Indicated = Yellow, Inferred = Blue

Figure 74: Plan View of Manto 2 Copper Distribution – Seam Model with Seam Composites

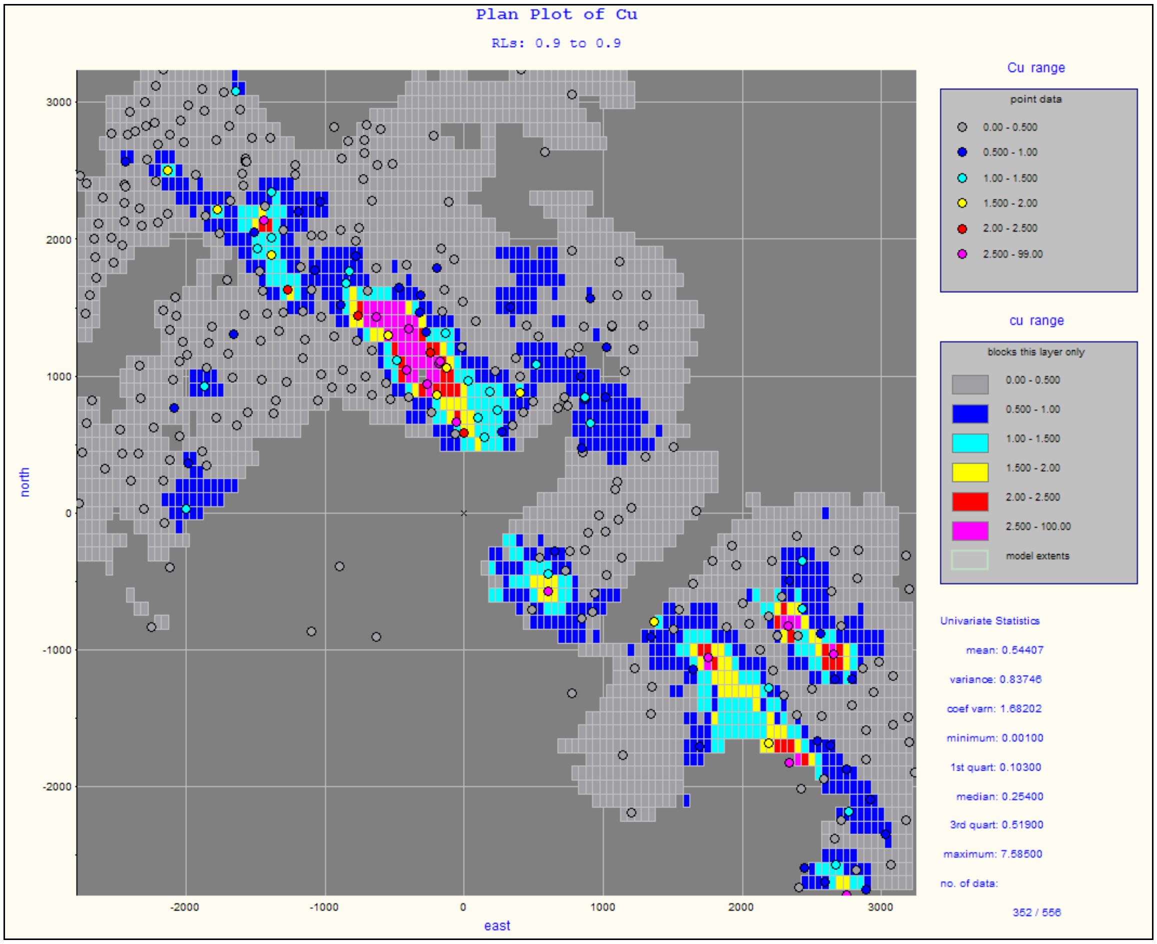


Figure 75: Plan View of Manto 2 Cobalt Distribution – Seam Model with Seam Composites

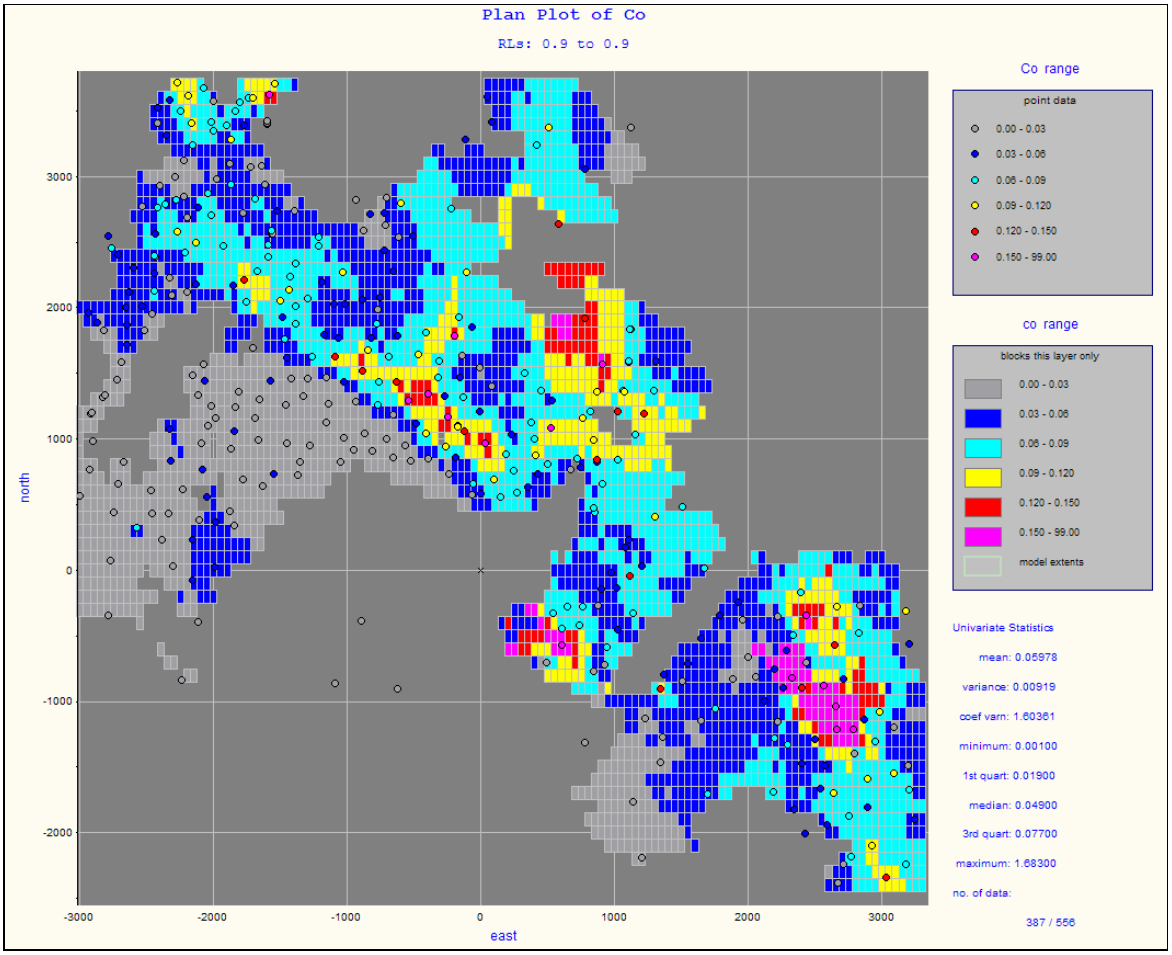


Figure 76: Plan View of Manto 2 Zinc distribution – Seam Model with Seam Composites

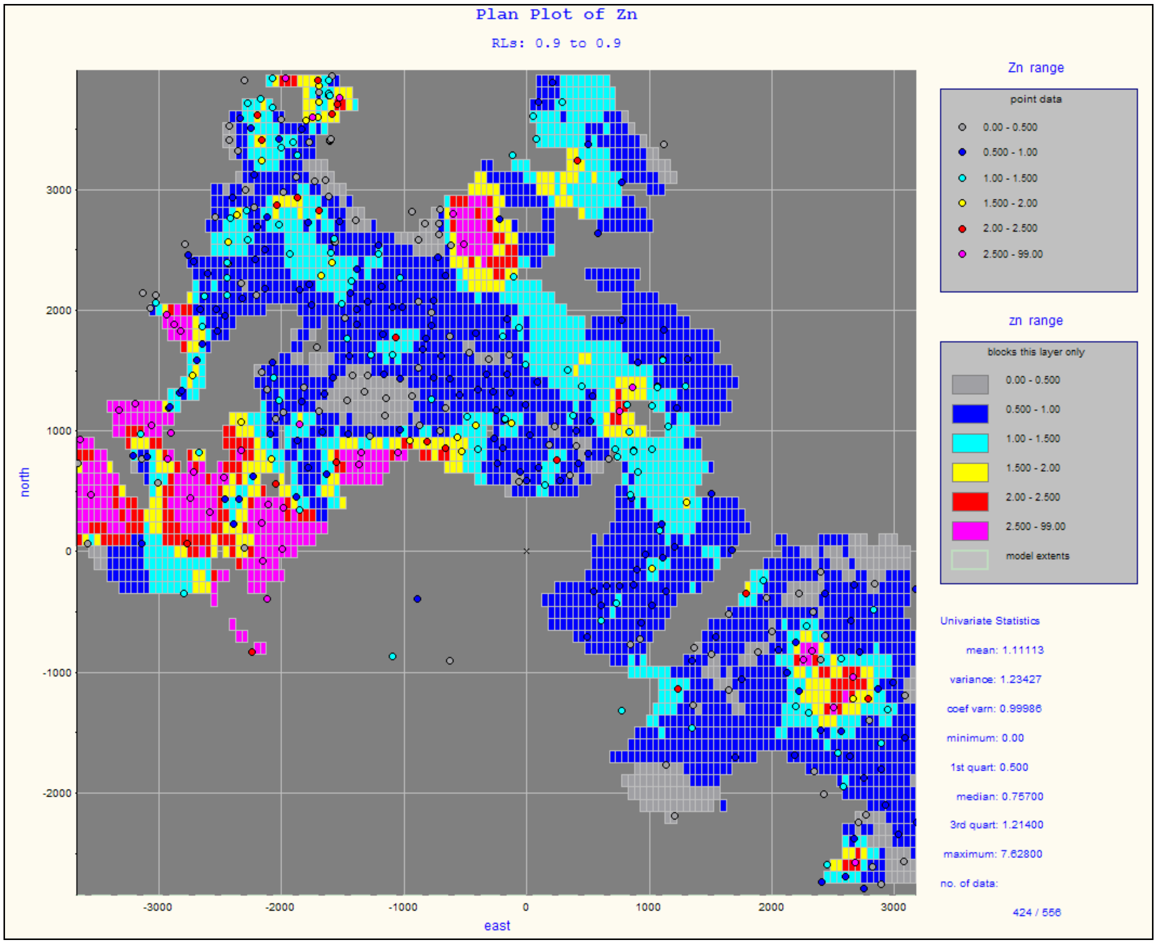
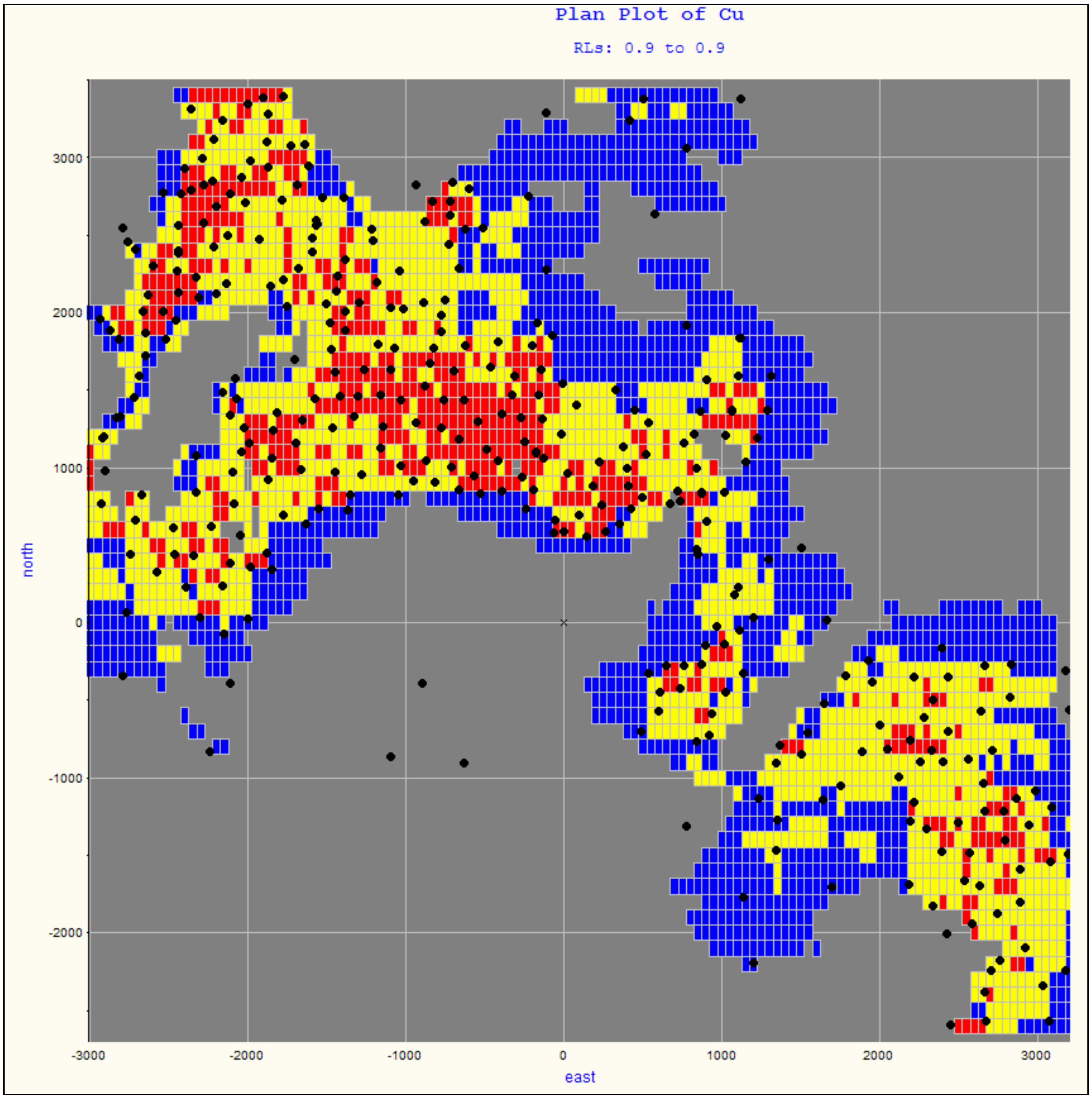


Figure 77: Plan View of Manto 2 Resource Classification – Seam Model with Drill Hole Locations



Note: Measured = Red, Indicated = Yellow, Inferred = Blue.

Figure 78: Plan View of Manto 3 Copper Distribution – Seam Model with Seam Composites

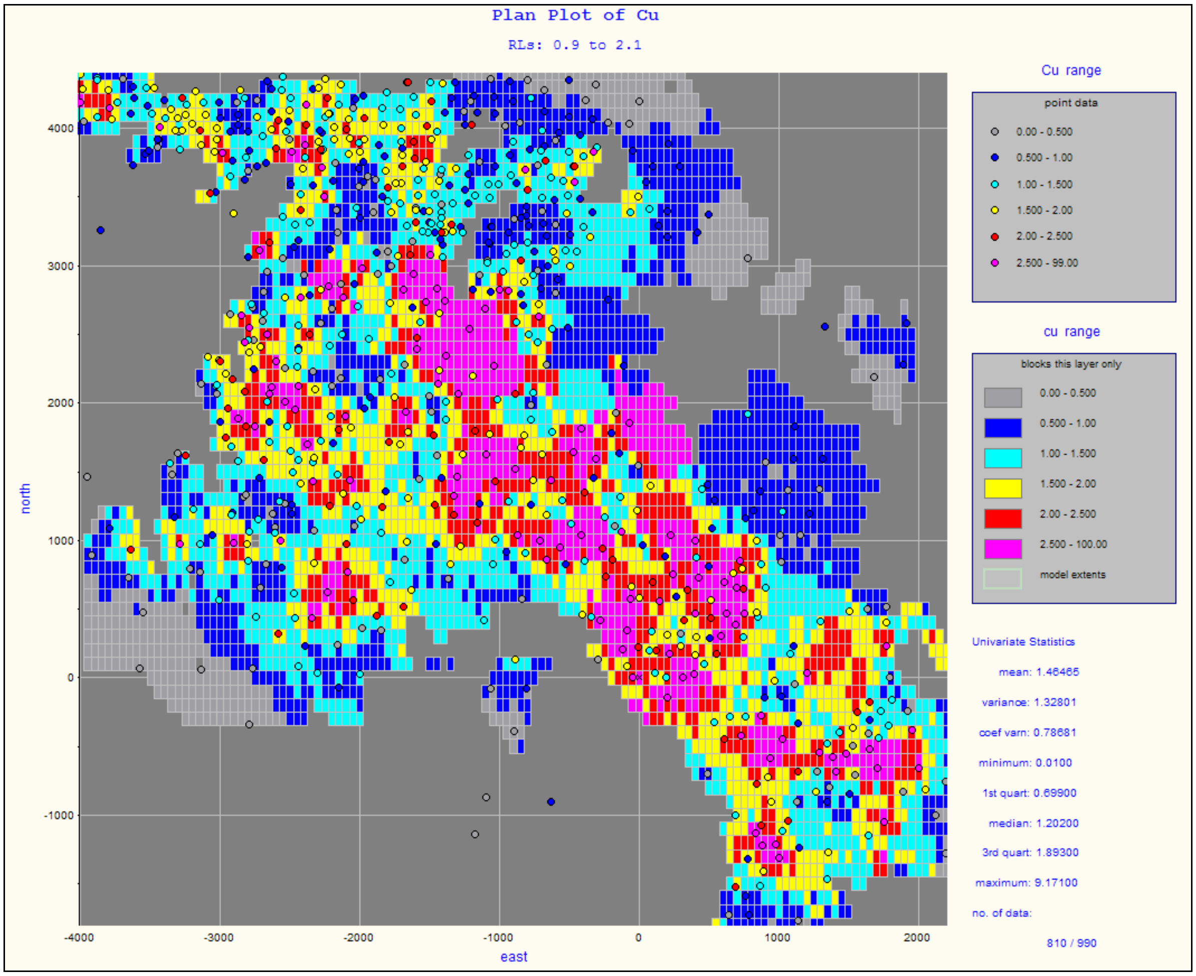


Figure 79: Plan View of Manto 3 Cobalt distribution – Seam Model with Seam Composites

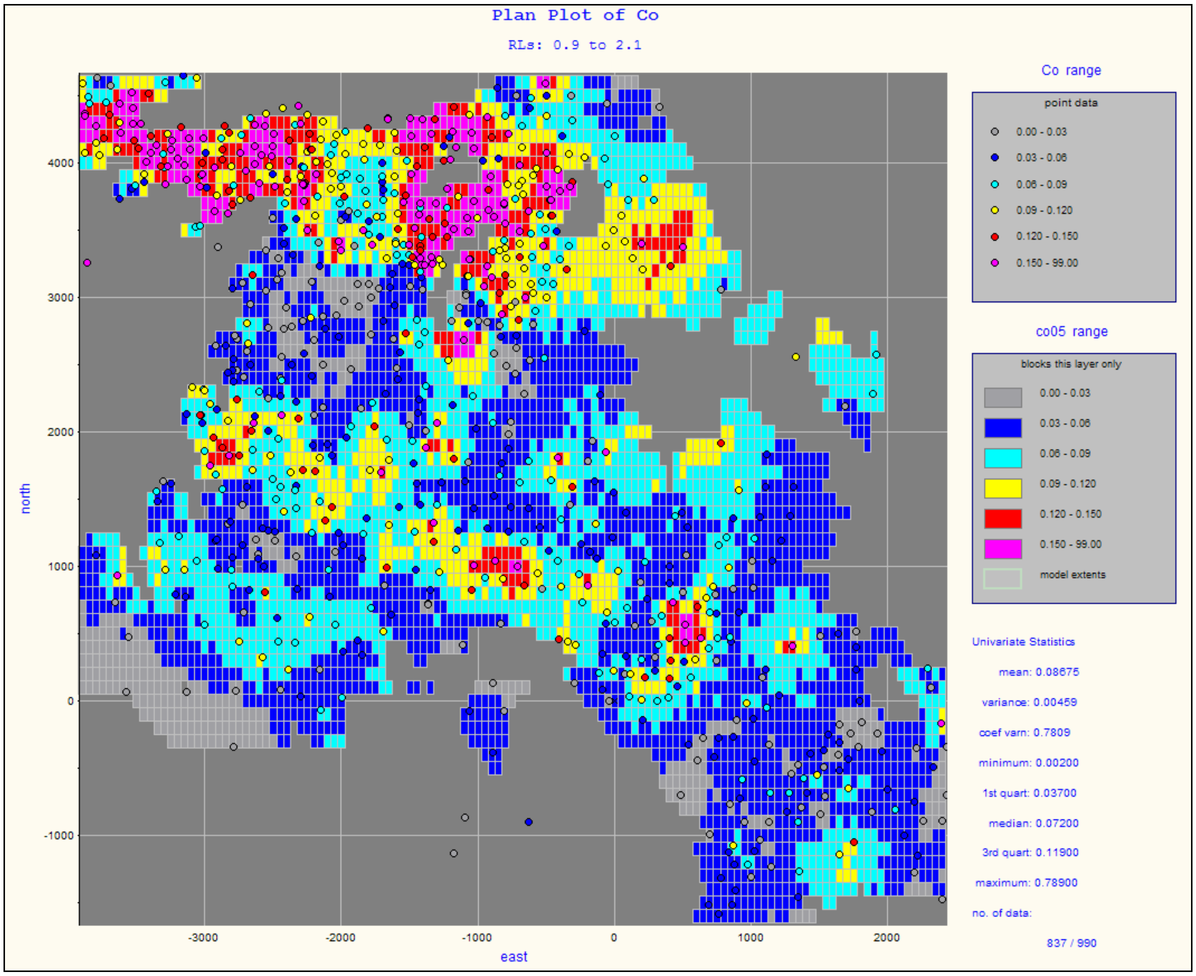


Figure 80: Plan View of Manto 3 Zinc distribution – Seam Model with Seam Composites

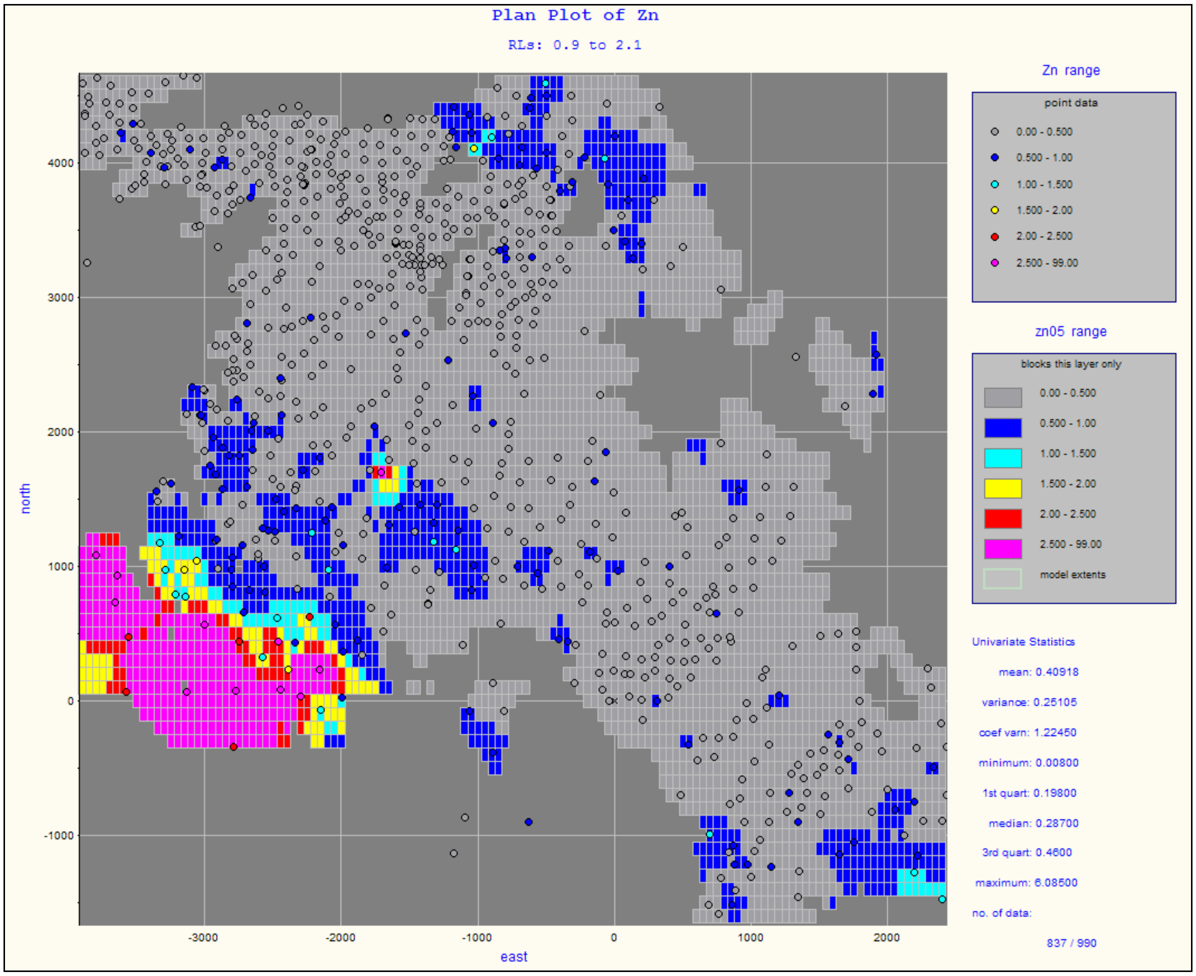
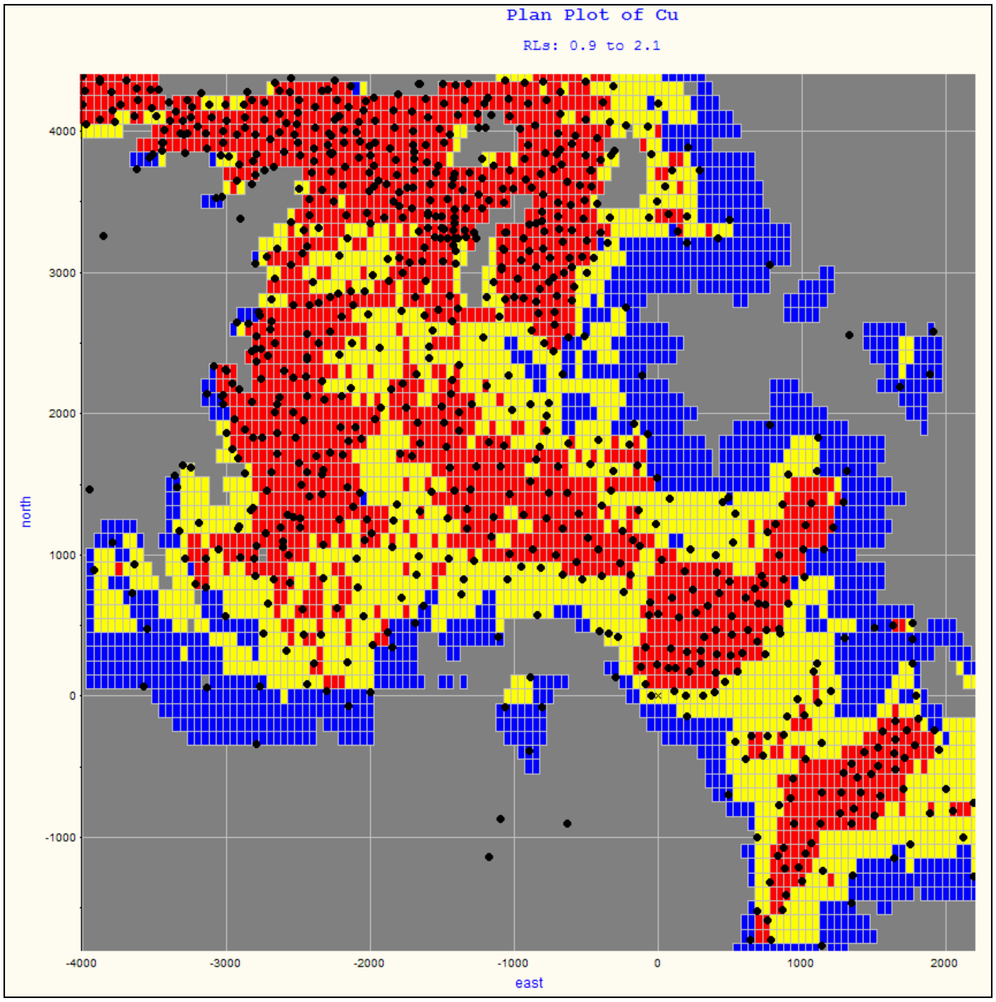


Figure 81: Plan View of Manto 3 Resource Classification – Seam Model with Drill Hole Locations



Note: Measured = Red, Indicated = Yellow, Inferred = Blue.

16.5 Total Resource

Table 47 through Table 52 combine the 3D block models and the seam models in a single table for both the 0.5% and 1.0% CuEq Cutoff grade.

Table 47: Measured and Indicated Resources at 0.50% CuEq

Final Models			Manto							
Cutoff: CuEq = 0.5%			0	1	2	3AA	3A	3	4	Total
Measured	3D Block	tonnes x10^6	2.3	4.6	9.0	0.8	10.0	44.6	3.3	74.6
		CuEq%	0.63	2.76	2.04	2.10	1.94	2.17	1.64	2.09
		Cu%	0.03	1.51	0.65	0.51	0.43	1.10	0.85	0.93
		Co%	0.010	0.069	0.064	0.083	0.098	0.082	0.050	0.078
		Zn%	0.62	0.71	0.94	0.96	0.66	0.32	0.35	0.48
		Mn%	0.91	2.74	4.09	5.47	5.10	1.98	2.36	2.72
	Seam	*tonnes x10^6*		4.0	9.2		14.9	31.9	1.1	61.1
		CuEq%		2.89	2.18		1.55	2.77	2.91	2.39
		Cu%		1.59	0.67		0.40	1.59	1.83	1.17
		Co%		0.072	0.067		0.069	0.087	0.076	0.078
		Zn%		0.73	1.05		0.58	0.38	0.39	0.55
		Mn%		2.90	4.28		2.29	2.17	3.00	2.58
Indicated	3D Block	tonnes x10^6	7.5	14.9	52.9	2.4	36.4	71.4	17.0	202.5
		CuEq%	0.71a	2.08	1.60	1.95	1.36	1.99	1.00	1.65
		Cu%	0.03	0.79	0.32	0.53	0.37	1.04	0.44	0.62
		Co%	0.010	0.050	0.048	0.082	0.053	0.057	0.035	0.050
		Zn%	0.73	0.98	0.99	0.76	0.59	0.48	0.26	0.66
		Mn%	1.39	3.07	4.34	5.37	3.79	2.30	1.63	3.10
	Seam	*tonnes x10^6*		6.8	21.2	2.9	18.7	31.1	5.1	85.8
		CuEq%		2.76	1.87	2.07	1.33	2.49	1.26	2.02
		Cu%		1.23	0.42	0.58	0.36	1.37	0.66	0.84
		Co%		0.071	0.058	0.082	0.052	0.065	0.037	0.060
		Zn%		1.02	1.08	0.84	0.57	0.59	0.28	0.73
		Mn%		3.50	4.68	5.33	2.09	2.59	2.00	3.13

Table 48: Measured and Indicated Total Resource at 0.50% CuEq

Final Models			Manto							
Cutoff: CuEq = 0.5%			0	1	2	3AA	3A	3	4	Total
Measured and Indicated Total	3D Block	tonnes x10^6	9.9	19.5	61.9	3.2	46.3	116.0	20.4	277.1
		CuEq%	0.69	2.24	1.66	1.99	1.49	2.06	1.11	1.77
		Cu%	0.03	0.96	0.37	0.53	0.38	1.06	0.51	0.70
		Co%	0.010	0.055	0.050	0.082	0.063	0.066	0.038	0.057
		Zn%	0.71	0.92	0.98	0.81	0.61	0.42	0.27	0.62
		Mn%	1.28	2.99	4.30	5.39	4.07	2.18	1.75	3.00
	Seam	*tonnes x10^6*		10.8	30.5	2.9	33.7	62.9	6.2	147.0
		CuEq%		2.81	1.96	2.07	1.43	2.63	1.55	2.17
		Cu%		1.37	0.50	0.58	0.38	1.48	0.87	0.97
		Co%		0.071	0.060	0.082	0.059	0.076	0.044	0.067
		Zn%		0.91	1.07	0.84	0.58	0.49	0.30	0.66
		Mn%		3.28	4.56	5.33	2.18	2.38	2.18	2.90

Table 49: Inferred Resources at 0.50% CuEq

Final Models			Manto							
Cutoff: CuEq = 0.5%			0	1	2	3AA	3A	3	4	Total
Inferred	3D Block	tonnes x10^6	7.4	49.2	52.5	0.6	23.8	56.4	63.3	253.2
		CuEq%	0.66	1.72	1.27	1.45	1.20	1.48	0.90	1.29
		Cu%	0.02	0.48	0.22	0.51	0.30	0.54	0.39	0.39
		Co%	0.008	0.044	0.042	0.053	0.044	0.047	0.032	0.040
		Zn%	0.69	1.01	0.79	0.51	0.58	0.59	0.25	0.63
		Mn%	1.77	2.93	3.60	5.09	3.40	2.53	1.50	2.64
	Seam	*tonnes x10^6*		8.0	21.8	0.2	18.5	23.5	13.6	85.6
		CuEq%		2.38	1.72	2.80	1.31	2.03	1.05	1.67
		Cu%		0.65	0.28	1.40	0.29	0.87	0.55	0.52
		Co%		0.053	0.053	0.084	0.049	0.057	0.032	0.050
		Zn%		1.49	1.14	0.70	0.66	0.74	0.22	0.81
		Mn%		3.92	4.70	8.18	2.69	2.50	1.56	3.10

Table 50: Measured and Indicated Resources at 1.0% CuEq

Final Models			Manto							
Cutoff: CuEq = 1.0%			0	1	2	3AA	3A	3	4	Total
Measured	3D Block	tonnes x10^6	0.04	4.4	7.8	0.8	8.6	39.8	2.0	63.4
		CuEq%	1.09	2.84	2.23	2.13	2.12	2.33	2.29	2.32
		Cu%	0.05	1.57	0.73	0.51	0.49	1.20	1.26	1.06
		Co%	0.012	0.071	0.071	0.084	0.107	0.086	0.066	0.085
		Zn%	1.15	0.71	1.00	0.97	0.69	0.33	0.46	0.50
		Mn%	1.96	2.77	4.34	5.49	5.46	2.05	3.17	2.92
	Seam	tonnes x10^6		3.8	8.3		10.9	31.6	0.7	55.3
		CuEq%		3.00	2.34		1.83	2.78	3.95	2.56
		Cu%		1.66	0.73		0.48	1.60	2.55	1.26
		Co%		0.075	0.072		0.084	0.087	0.099	0.083
		Zn%		0.73	1.11		0.64	0.38	0.51	0.57
		Mn%		2.89	4.56		2.20	2.18	3.91	2.61
Indicated	3D Block	tonnes x10^6	0.9	12.3	42.0	2.2	26.0	61.0	6.3	150.7
		CuEq%	1.12	2.36	1.81	2.06	1.59	2.20	1.52	1.96
		Cu%	0.03	0.94	0.37	0.57	0.46	1.19	0.74	0.78
		Co%	0.011	0.057	0.054	0.086	0.061	0.060	0.048	0.058
		Zn%	1.23	1.06	1.12	0.79	0.65	0.52	0.37	0.76
		Mn%	2.70	3.25	4.75	5.52	4.22	2.43	2.08	3.48
	Seam	tonnes x10^6		6.3	18.8	2.5	12.8	30.3	1.9	72.6
		CuEq%		2.91	2.00	2.28	1.57	2.54	2.23	2.24
		Cu%		1.32	0.46	0.65	0.46	1.40	1.29	0.96
		Co%		0.075	0.062	0.089	0.062	0.066	0.058	0.066
		Zn%		1.05	1.15	0.93	0.62	0.60	0.44	0.79
		Mn%		3.46	5.02	5.79	2.11	2.59	3.52	3.34

Table 51: Measured and Indicated Total Resources at 1.0% CuEq

Final Models			Manto							
Cutoff: CuEq = 1.0%			0	1	2	3AA	3A	3	4	Total
Measured and Indicated Total	3D Block	tonnes x10^6	0.9	16.7	49.8	3.0	34.6	100.8	8.3	214.1
		CuEq%	1.11	2.49	1.88	2.08	1.72	2.25	1.70	2.07
		Cu%	0.03	1.11	0.43	0.55	0.47	1.19	0.86	0.86
		Co%	0.011	0.061	0.057	0.086	0.073	0.070	0.052	0.066
		Zn%	1.23	0.97	1.10	0.84	0.66	0.44	0.39	0.68
		Mn%	2.67	3.13	4.69	5.51	4.53	2.28	2.34	3.32
	Seam	tonnes x10^6		10.1	27.1	2.5	23.7	61.9	2.6	127.9
		CuEq%		2.95	2.10	2.28	1.69	2.66	2.71	2.38
		Cu%		1.45	0.54	0.65	0.47	1.50	1.65	1.09
		Co%		0.075	0.065	0.089	0.072	0.077	0.070	0.073
		Zn%		0.93	1.14	0.93	0.63	0.49	0.46	0.70
		Mn%		3.24	4.88	5.79	2.15	2.38	3.63	3.03

Table 52: Inferred Resources at 1.0% CuEq

Final Models			Manto							
Cutoff: CuEq = 0.5%			0	1	2	3AA	3A	3	4	Total
Inferred	3D Block	tonnes x10^6	0.3	34.9	34.2	0.4	13.6	40.3	18.2	141.9
		CuEq%	1.06	2.13	1.54	1.86	1.49	1.75	1.39	1.72
		Cu%	0.02	0.65	0.25	0.71	0.43	0.69	0.72	0.55
		Co%	0.008	0.056	0.049	0.067	0.055	0.053	0.042	0.052
		Zn%	1.20	1.15	0.99	0.60	0.65	0.67	0.31	0.82
		Mn%	2.84	3.29	4.29	6.14	4.06	2.73	1.73	3.25
	Seam	tonnes x10^6		6.9	18.7	0.1	13.6	20.9	5.4	65.6
		CuEq%		2.63	1.88	3.12	1.49	2.17	1.58	1.95
		Cu%		0.73	0.31	1.61	0.34	0.96	0.92	0.62
		Co%		0.060	0.058	0.090	0.057	0.060	0.044	0.058
		Zn%		1.62	1.24	0.77	0.73	0.78	0.28	0.95
		Mn%		4.24	5.14	8.93	2.65	2.55	2.57	3.50

16.6 MINERAL RESERVES

The initial 25 years of production and all mine planning were based on the measured and indicated resource models provided by Hellman & Schofield Pty Ltd. The open cut mine plan, completed by AMDAD, used the block model at 0.5% CuEg Cutoff and the underground mine plan, completed by Agapito, used the seam models at 0.5% CuEq Cutoff.

This has resulted in the outlining of sufficient Proven and Probable reserves to cover the initial 25-year mine plan of a mineable 74.36 Mt at 1.47% Cu, 0.08% Co and 0.57% Zn. The first 20 years of production will come solely from underground sources. Years 20 to 25 production will come from a combination of underground and open cut sources. A minor quantity of Inferred resource is also included within the 74.4 Mt planned as this material is in areas through which principle accesses must proceed.

Underground mining will take place in Mantos 1, 2, 3 and 4 in areas where drilling has shown grade continuity and thicknesses to be sufficiently established to classify these areas as Proven and Probable Reserves (Table 53).

Table 53: Underground Proven and Probable Reserves

Underground Reserves		Cu%	Co%	Zn%	Mn%
Class	tonnes x10^6				
Proven	29.8	1.69	0.08	0.46	2.38
Probable	37.6	1.34	0.07	0.69	3.36
Total	67.4	1.49	0.07	0.59	2.93

Open-cut mining will take place in the Proven and Probable defined areas of Mantos 2, 3a, 3aa, and 3 and is introduced into the production stream beginning in Year 21 and increases in volume as the underground mining is reduced (Table 54).

Table 54: Open Cut Proven and Probable Reserves

Open Cut Reserves		Cu%	Co%	Zn%	Mn%
Class	tonnes x10^6				
Proven	11.1	0.75	0.10	0.37	2.74
Probable	6.5	0.70	0.07	0.48	3.08
Total	17.6	0.73	0.09	0.41	2.87

Reserves shown are included within the resources defined at the 0.5% CuEq Cutoff and are not additional to them.

17 OTHER RELEVANT DATA & INFORMATION

17.1 MINING

The mining plan supporting the economics of El Boleo has been scheduled for an initial 25 years of plant operation. The plan consists of a series of underground and surface mines scheduled in such a manner as to provide the highest grades of copper ore for the first 8 years of operation and then the highest grades of copper equivalent ore for the remaining years of the plan.

The nominal production rates for all mining operations is 3.1 Mdt/a. At full production, the mining workforce totals 282 salaried and hourly employees. Skilled and experienced "Ex-pat" labour, used in the start-up years for training and skill development needs, are essentially phased out by Year 6.

The surface mining plan incorporates all the surface mining and surface infrastructure activities required for executing both the underground mine plan (Section 17.2), the surface production schedule, the limestone quarry, tailings dam, roads, utilities, and ROM ore haulage. The surface plans were prepared by AMDAD and Wardrop Engineering.

The underground plan was prepared by AAI and schedules mining activities for the initial 20 years of plant operation. The underground plan is based on detailed geotechnical analysis of:

- the ground conditions observed and measured in the test mine
- the geomechanics testing of core
- the experience of AAI in room and pillar mining operations and the predicted behaviour of the ground.

17.1.1 SURFACE MINING & MINE DESIGN

Surface mining activities at the Boleo Project will consist of:

- mining of copper cobalt ore from a series of shallow pits towards the end of the mine life
- a quarry to supply calcium carbonate in the form of limestone for use in the ore processing plant.

The mining fleet and personnel will also be used to assist other areas of the operation including:

- excavation and haulage of rock fill for use in the tailings dam wall construction
- haulage of ore from underground portal sites not serviced by conveyors
- general earthworks such as road maintenance.

Although all of the ore for the first 20 years will be sourced from the underground mines, the surface mine fleet is an essential part of the project because it allows calcium carbonate, which is major input cost for ore processing, to be sourced at less than half the cost of purchasing limestone from the nearest supplier.

The open cut copper pits add 14% to the underground ore reserves and allow the project to continue operating for at least six more years past project Year 20 when the underground mine is nearing depletion. The shallow pits and the quarry mining fleet also provide an easily accessible alternative ore source if the underground mine suffers any disruption during the first 20 years.

Open Cut Mining

Open cut operations at Boleo 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 2 de Abril area in the north western corner of the mining area.

The open cuts will be phased in at the end of production year 20 to compliment the declining underground mining operations, allowing full production feedrate to the plant to be maintained.

Open Cut Mining Analyses

All of the open cut targets mine to Manto 3 as the base unit. Some pits include areas of Mantos 3A and 3AA above Manto 3. No Manto 1, 2 or 4 ore is included in the open cut targets.

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 & 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.

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. Geotechnical analysis undertaken as part of the feasibility study is considered adequate for initial level of confidence slope design purposes but an on-going process of data collection and slope monitoring during operations will be required to allow pit slopes to be optimized.

Mining Method

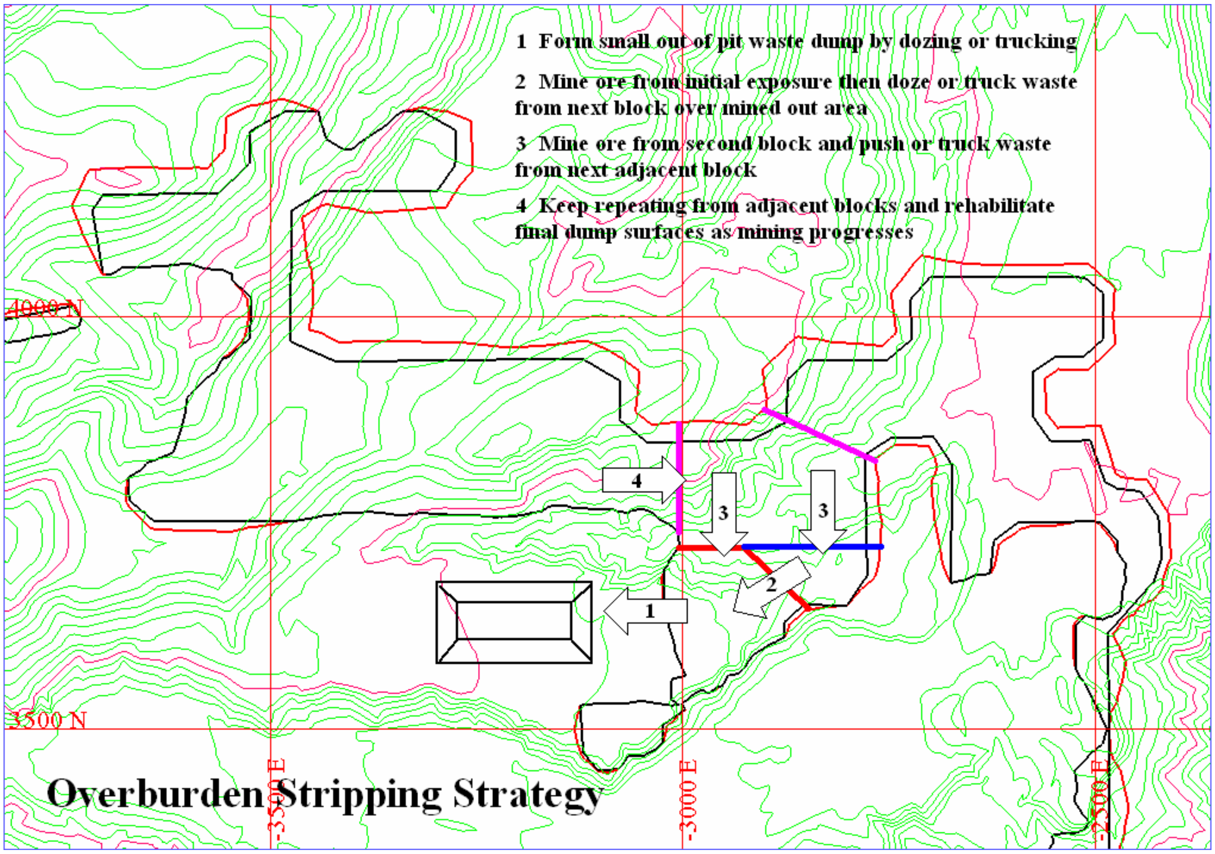
The open cut copper cobalt pits will use the same fleet as the limestone quarry but the mining method places more emphasis on the bulldozers as primary waste mining units. All of the pits start at outcrop and in many cases; this outcrop line is in an arroyo wall so that there is open space below the pit start position. Almost all of the overburden consists of weak sandstones, siltstones and clay and the mantos themselves are free digging. Overburden is 0 m to 40 m thick and the economic manto thickness is mostly 1 m to 4 m. The floor of Manto 3 in the open cut areas is hard conglomerate.

The overburden stripping strategy is illustrated in Figure 82.

The general mining sequence is:

- Drill and blast all overburden. It could be ripped but light blasting at an average powder factor of 0.32 kg of explosive per cubic metre will greatly improve productivity of the bulldozers and excavators.
- In the first block of each pit, the overburden is moved to an out of pit waste dump adjacent to the pit. Where the elevation of the pit start point allows, this initial block overburden can be moved by dozer push. In flatter areas it will be mined by a 120 tonne hydraulic excavator loading 55 tonne trucks.
- In subsequent blocks the overburden will be mostly dozer pushed over the mined out area in the previous block. In some cases, the pit geometry or manto dip will require some portion of the overburden to be mined by the excavators. Overburden for each block was assigned dozer and excavator percentages based on an assessment of the block geometry in cross-section.

Figure 82: Overburden Stripping Strategy



- In all blocks overburden removal will cease 5 m above the expected roof position of the economic manto as defined from the previous block and the resource model. The manto will be grade control drilled from this bench to determine the ore horizon and average grades for the block.
- The excavator will mine the last 5 m of waste and the trucks will place the waste on the dozer push waste in the previous block.
- The excavators will mine the manto ore and the trucks will haul it to the nearest conveyor station set up for the underground mines. In the case of the pits in the east end of Soledad Arroyo, the trucks will haul directly to the process plant stockpiles.

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

The detailed shape of the mining blocks within each pit will depend on the thickness and dip of the manto and the overburden depth and topography across the pit. However, in most cases each block will be mined a series of strips no more than 70 m wide to accommodate efficient dozer stripping.

PIT SELECTION

Prior to settling on the combination of underground and small scale, open pit mining adopted for this DFS, the potential for a large scale, low grade open pit operation with limited selectivity was assessed as a possible method of exploiting multiple mantos. This initial analysis of open cut potential was performed in 2005 using standard Whittle pit and Minex seam optimization software. The resultant open cut operation was extensive, large scale and low grade and characterized by high stripping ratios. While the potential for a very large scale, low grade operation was demonstrated from a technical point of view without regard to cost, this proposed mining method created a number of issues:

- Large scale open cut mining would deliver average mill head grades of around 0.9% Cu. To achieve the desired level of copper metal production and sufficient cashflow for economic viability the metallurgical process plant would need to operate at between 6 and 10 Mt/a (dry).
- The extensive mining operation would create major changes to the landscape over a large area with correspondingly significant environmental impacts which MMB wished to avoid, if possible.
- To access the relatively narrow but extensive mineralized mantos large volumes of overlying waste would have to be removed resulting in waste to ore stripping ratios of up to 20 :1
- The initial capital cost in terms of pre-production waste stripping and mining equipment fleet acquisition combined with the capital cost of a metallurgical processing plant with a capacity of 6 to 10 Mdt/a was thought to be beyond Baja Mining's capability to finance.
- The distribution of relatively thin mineralized mantos and significant thicknesses of totally barren overburden resulted in an uneven production schedule. Attempts to maintain a steady, year on year plant feed rate resulted in significant spikes in overburden stripping requirements with corresponding inefficient equipment fleet utilization.

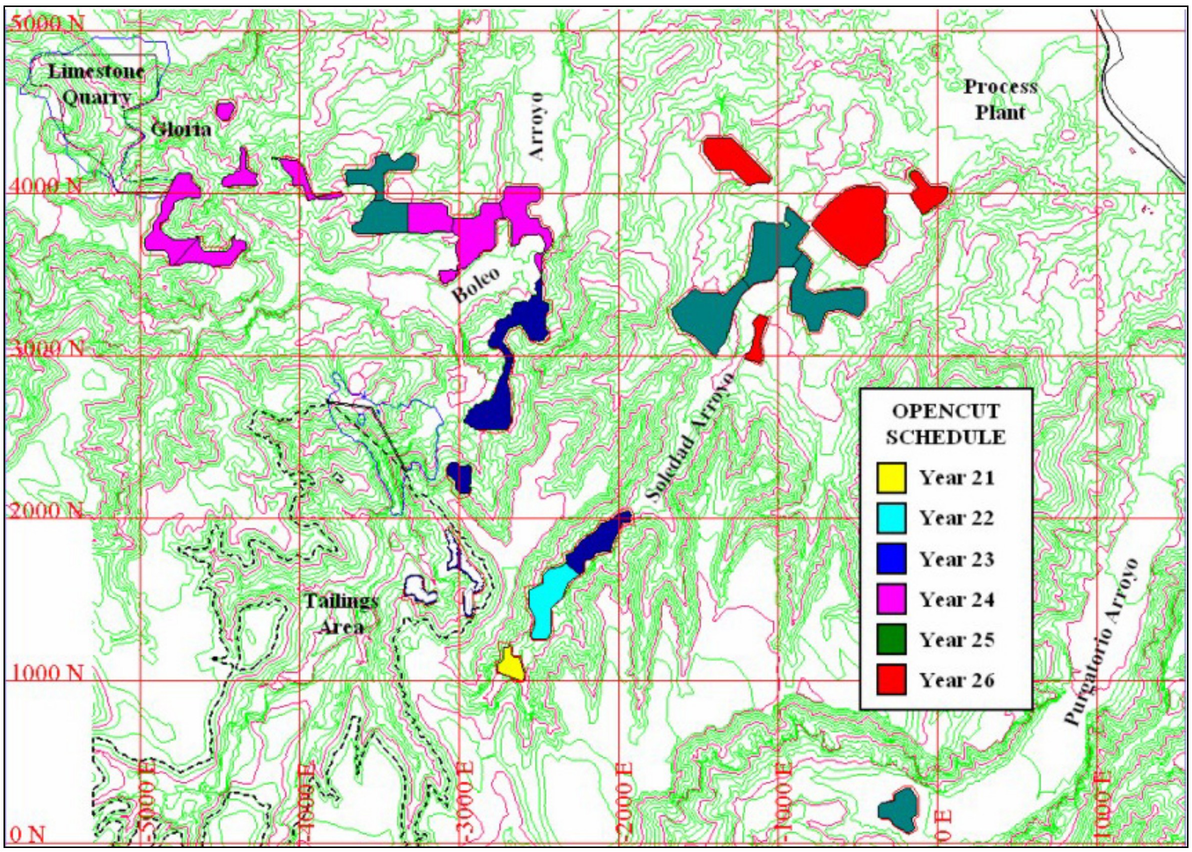
Work to date on open cut and underground operating costs shows that underground mining can deliver ore at US$9.00 to US$12.00/dt. Open cut mining costs US$1.80 to US$2.20/t. This means that underground mining can deliver ore more cheaply once the waste to ore ratio exceeds 6 to 1. If the average payable manto thickness is 3 m, the pits should give way to underground mining at depths in excess of 18 m.

This finding caused a reassessment of underground mining potential and the eventual development of the combined underground and small scale open pit mining method that forms the basis of this DFS which assumes open cut mining will not commence until Project Year 21.

At that time, the underground mine production starts to decline and limestone quarrying is complete. This makes it possible to transfer the limestone quarrying fleet to the open cuts and mine sufficient ore to maintain 3.1 Mdt/a feed for the process plant. As underground production falls, open cut production will increase correspondingly such that by Year 25 the open cuts are supplying the entire ore feed. The feasibility study mining schedule shows the final open cut pit is depleted in the third quarter of Year 26.

The open cut mining sequence shown in Figure 83 and is designed to mine from highest to lowest copper equivalent grade while maintaining efficient access from the pits to the conveyor loading points developed for the underground mines.

Figure 83: Open Cut Mining Schedule



17.1.2 LIMESTONE QUARRY

The ore treatment process requires in excess of 750,000 tonnes of limestone grading 65% $CaCO_3$ per year for various neutralisation duties. This tonnage requirement varies according the ore grade and pay metal mix. This limestone 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 Boleo sequence and can be seen outcropping along many ridge lines. This section of the Coquina bed averages just over 64% $CaCO_3$.

The quarry mining method is based around two 120 tonne hydraulic excavators loading 55 tonne rigid body dump trucks. All material is blasted using an average powder factor of 0.45 kg of explosives per cubic metre of rock.

As the quarry advances into the arroyo walls, some parts of the pit can reach up to 100 m height from the pit floor to the pit crest. There is limited room to develop haul roads over this height and the cost of hauling down large elevation changes is prohibitive. To overcome the elevation differences, two large bulldozers will be used to push the blasted waste rock down to the excavator working level, which will generally be no more than 30 m above the final pit floor.

The hydraulic excavators are in backhoe configuration to allow greater flexibility in mining irregular bench shapes and forming access ramps onto the benches and across the waste dumps. The dump trucks haul ore to either the crushing station at the head of the Boleo conveyor or the low grade stockpile and waste either to the out of pit waste dump or to a final in pit waste position above one of the mined out areas.

Limestone hauled to the crusher will be held in stockpile and rehandled into the crusher by a wheeled loader. The conveyor also transports copper ore from several of the underground mines so the limestone can only be conveyed on one eight hour shift per day.

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.

PURCHASED LIMESTONE

The hydrometallurgical copper cobalt recovery process requires approximately 161 kg of pure $CaCO_3$/dt of ore or 250 kg of 65% strength $CaCO_3$/dt of ore. As the grade of limestone from the quarry falls over the LOM, more tonnes of limestone are required to maintain the $CaCO_3$ tonnage fed to the plant. The quarry schedule assumes that limestone feed production rate will be capped at 1 Mt/a. At full ore production of 3.1 Mdt/a, the 1 Mt/a limestone limit is reached when the grade falls below 60% $CaCO_3$. In periods when this occurs, just enough imported limestone at 90% $CaCO_3$ will be purchased to keep the feed grade at no less then 60% $CaCO_3$ and the limestone feed rate at no more than 1 Mt/a.

High-grade limestone can be purchased from commercial quarry operations in mainland Mexico and barged across Sea of Cortez from Guaymas.

17.1.3 OTHER SURFACE MINING TASKS

In addition to quarrying limestone and mining copper ore, the open cut mining fleet will be used other functions requiring bulk excavation or haulage capability. These tasks will be achieved by either using spare time in the fleet work roster or by providing additional equipment, which can be used interchangeably with the quarry or mining operations.

Rock Fill for Tailings Dam Wall Construction

During the pre-production year the Stage 1 of the tailings dam wall will require 1.3 million cubic metres of volcanic basement rock to be quarried from within the tailings dam impoundment area to form the bulk of the initial wall structure. If a contractor is brought in to excavate this rock, the cost will include mobilization, accommodation for a significant workforce, amortization of the contractor's equipment and the contractor's margin. In order to avoid these additional costs part of the limestone quarry fleet will be mobilized in Month 4 of the pre-production year. One 120 tonne excavator, two 55 tonne trucks, a blast hole drill and several support units including a bulldozer, a grader and a water truck will work 3 x 8 hour shifts per day, seven days per week for seven months delivering rock to the dam wall site. At the conclusion of this work, the fleet will move to the limestone quarry to build stocks ready for commissioning of the process plant.

Ore Haulage from Underground Portal Sites

Most of the underground mine entry points will be serviced by a conveyor to take crushed ore to the process plant. Some of the portal sites, however, are in areas where the topography makes conveyor access impractical. Ore from these sites will be re-handled off stockpile by a 6.5 m^3 wheeled loader into 55 tonne rigid body dump trucks for haulage to the nearest conveyor loading point. The truck and loader size selection is to be consistent with the rest of the open cut mining fleet and is large enough for the ore from two portal sites to be handled on a "batch" basis so that it is not necessary to have a loading and haulage fleet dedicated to each portal site.

Underground ore haulage is only required for 20 months during the first 12 years of the project life but from Year 13 to 21 it is required almost constantly.

General Earthworks

The open cut mining fleet will carry out other support roles for the operation including road surfacing with gypsum won from outcrops in Boleo Arroyo, road watering for dust suppression and general site earthworks as required.

17.1.4 Surface Mining Costs

Capital Costs

Surface capital mining costs are included as part of the owner's cost in years 2007 to 2009, inclusive. Beginning in year 2010, there are capital needs to

- extend haulroads, utilities and beltlines

- continue with the tailings dam lifts
- open additional underground mine portal sites.

OPERATING COSTS

Average surface mining costs, including equipment leases, for each activity are:

- Mining volcanic rock for Stage 1 of the tailings dam wall:

  1.3 Mbcm at US$4.74/bcm
- Limestone quarry operations:

  average mining cost per BCM waste and limestone US$4.37

Table 55: Average Cost per Tonne for Limestone

Life of Project	Limestone Tonnes	$CaCO_3$ Tonnes	US$/t Limestone	US$/t $CaCO_3$
Quarried Limestone	18,714,068	10,321,121	6.88	12.47
Purchased Limestone	5,206,905	4,686,215	24.00	26.67
Total	23,920,973	15,007,336	10.61	16.90

- Loading and hauling underground ore to conveyor loading points:

  7.8 Mt (wet) at US$0.65/t
- Open cut mining operations:

Table 56: Average LOM Costs

Life of Project	US$/$m^3$ Ore and Waste	US$/wt Ore and Waste	US$/wt Ore	US$/dt Ore
Average Cost	2.23	1.18	5.28	7.08

17.1.5 UNDERGROUND MINING & MINE DESIGN

Underground mining operations at Boleo will consist of mechanised 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. Mine planning by Agapito has been based on the geologic model provided by Hellman & Schofield Pty Ltd.

Underground mining will take place in Mantos 1, 2, 3 and 4 where (1) there are reasonably sized, contiguous mineralized zones classified as either measured or indicated resources 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 69% of the underground mining is in Manto 3, 18% in Manto 2, 11% in Manto 1 and 2% in Manto 4.

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 higher than 2.0% 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 underground mine plan were modelled by Agapito using SurvCADD™ software, a mining design program popular with seam type of underground mining.

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 Mdt/a, coinciding with the start of the second year of plant operation. Underground production is ramped up to 3.1 Mt in year 4. During year 7, a sixth underground mining unit is added to compensate for declining ore grade and production fluctuations brought about by the scheduling of advance and retreat mining.

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

Because the Boleo mantos display similar depositional and geotechnical similarities with coal seam deposits (bedded and relatively soft ground), the recommended underground mining method is similar to coal, trona, potash, and salt seam room-and-pillar mining with pillar removal as successfully used 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. Most mine portals can be established above the typical food levels but 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 production scheduling and mine conditions allow, development work for an entire district will be completed prior to initiating pillar removal. Multiple, independent blocks of ore, called “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 typical mining district is shown in Figure 84. A preliminary layout for proposed Mining Area 3-10 is shown in Figure 85.

Figure 84: Conceptual Mine Layout for a Typical District

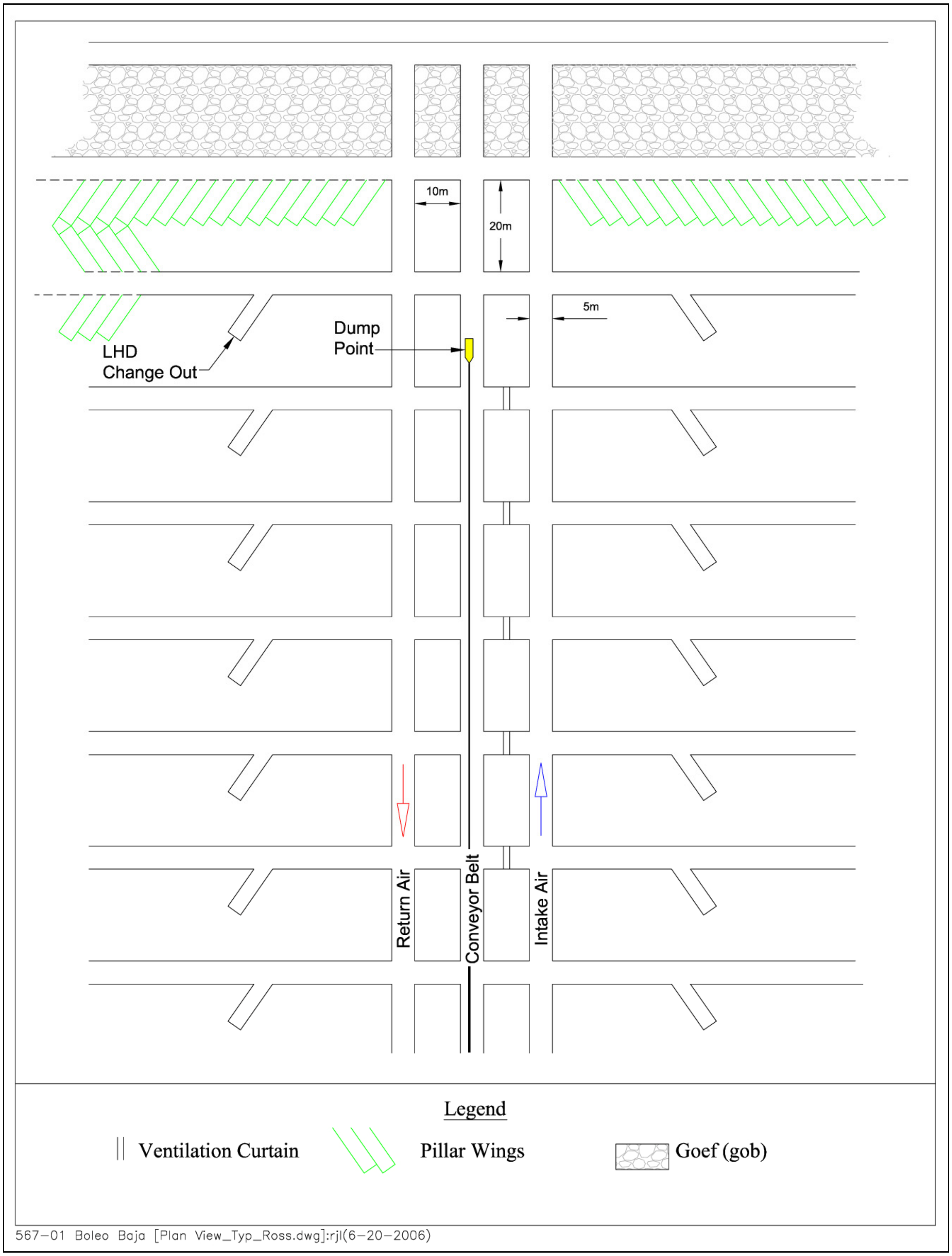
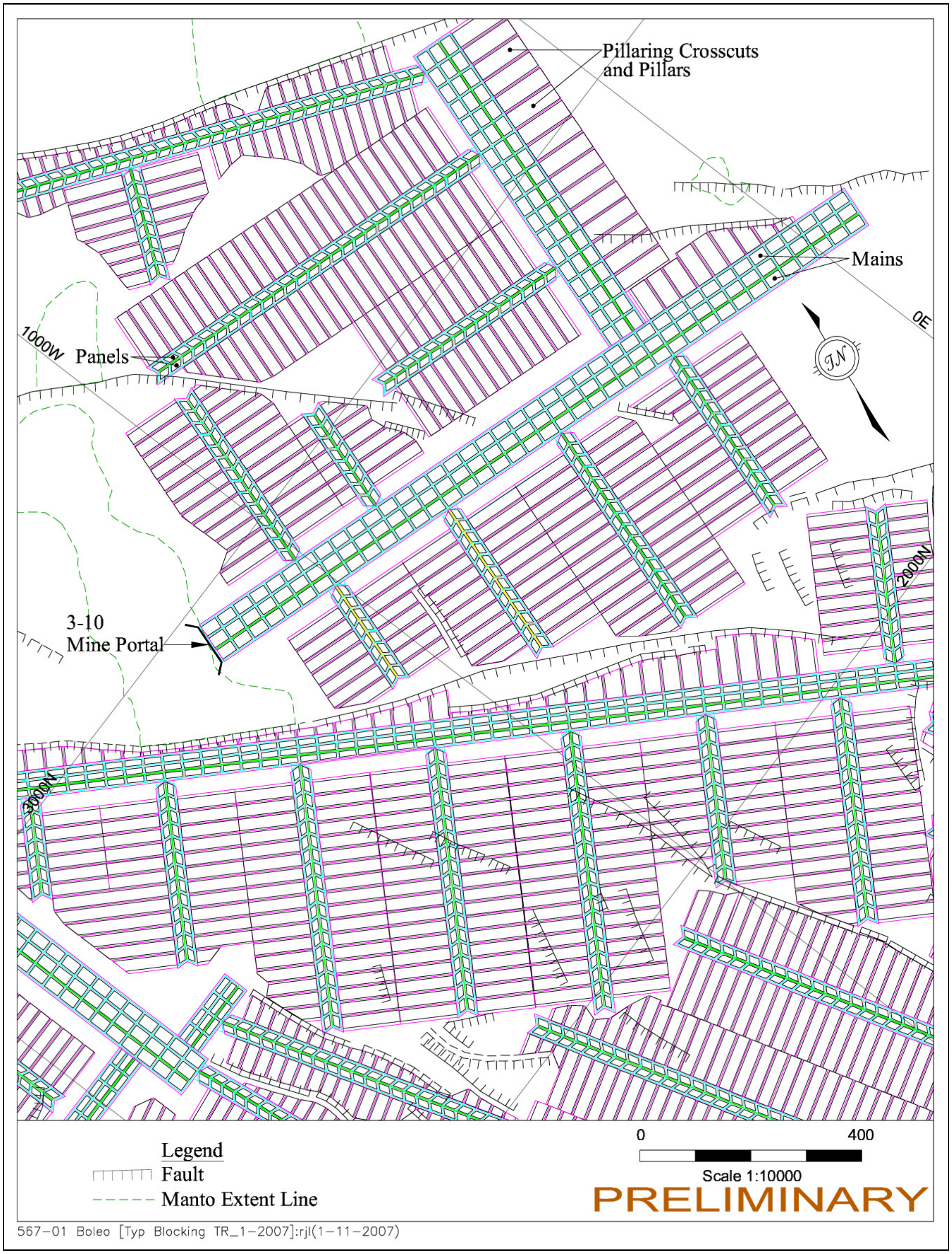


Figure 85: Mining Area 3-10 Preliminary Layout



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.

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 Boleo can adapt to variable mining heights between 1.8 m 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.

Geotechnical Mine Design

All underground pillar and excavated opening dimensions used for mine layout and design are based on (1) observations made and documented[4] during test mining at Boleo's Texcoco mine during the last quarter of 2005 and 2006, and (2) a geotechnical study conducted by AAI.[5]

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.

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

WORKER SAFETY

Underground mining inherently contains hazardous working conditions. The designed mining systems, including fire prevention and fire fighting, emergency transportation, ventilation, power, communications, first aid and production equipment specifications have incorporated not only applicable Mexican mining laws, but also North American, specifically, US Mine Safety and Health Administration (MSHA) standards as promulgated in Chapter 30, Part 57 where applicable.

Worker health standards for hazards such as noise and dust were incorporated into the basic mining plans where applicable.

Worker safety training has been incorporated into schedules and plans as both newly hired experienced and newly hired inexperienced programs. Annual refresher training has been allowed for in the basic mine production schedules.

MINE LAYOUTS

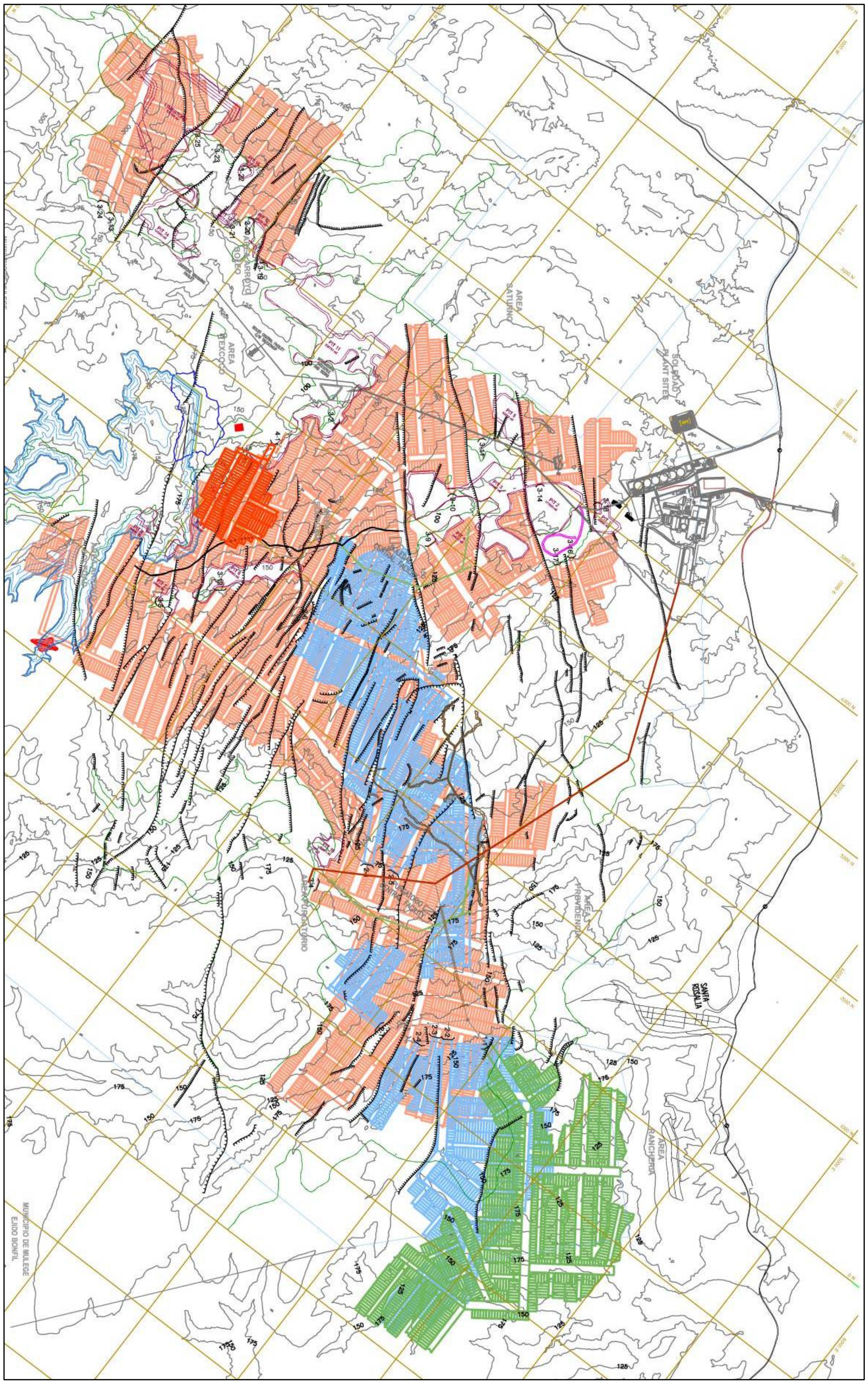
Figure 86 shows the overall mine working layouts. Because there are mine workings scheduled for production above (or below) another target manto, there is a time delay of at least two years to allow subsidence and settling to occur where necessary.

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

Figure 86: Overall Mine Work Layout



Manto 1 – Green

Manto 2 – Blue

Manto 3 – Red

Manto 4 – Orange

Capital Costs

Capital costs for underground equipment are included in owner's costs for years 2007 to 2009. Underground mining equipment, portal conveyors, mine fans, chillers, mine electrical equipment, and speciality surface equipment capital and leasing costs total US$151.06 million for the life of the project. Leasing costs exclude leasing financing and administrative costs. Equipment leases total US$119.85 million direct equipment cost, with 15% (US$17.98 million) being required as equity. Capital costs total $31.21 million excluding the equity portion of the lease requirement.

Operating leasing terms have been developed from active proposals received from third party leasing organizations. At least one potential equipment supplier to the project has stepped forward with a proposal to lease all their equipment.

Initial project underground capital (Years 0–2) totals US$7.21 million. Initial project underground leasing costs totals US$48.35 million. Combined initial project capital and lease costs are US$55.56 million.

Operating Costs

As stated above, operating costs for the projected mine plan were developed from experience and data from operating mines in similar mining conditions. Operating lease costs are not included in Table 57.

Table 57: Projected Underground Mine Costs

Year		Direct Mine Labour ($)	Roof Support Materials ($)	Other Supplies ($)	Mine Extension Materials ($)	Maintenance & Repair Materials ($)	Total* ($/dt)
0	2008	5.50	5.69	0.75	17.04	2.06	31.04
1	2009	2.06	1.88	0.75	3.10	2.06	9.85
2	2010	1.68	1.88	0.75	0.54	2.06	6.90
3	2011	1.35	1.88	0.75	0.51	2.06	6.55
4	2012	1.46	1.88	0.75	0.65	2.06	6.80
5	2013	1.12	1.88	0.75	0.56	2.06	6.37
6	2014	1.03	1.88	0.75	0.54	2.06	6.26
7	2015	1.00	1.88	0.75	0.16	2.06	5.84
8	2016	1.15	1.88	0.75	0.18	2.06	6.01
9	2017	0.96	1.88	0.75	0.16	2.06	5.81
10	2018	0.98	1.88	0.75	0.16	2.06	5.83
11	2019	0.94	1.88	0.75	0.15	2.06	5.78
12	2020	0.96	1.88	0.75	0.36	2.06	6.01
13	2021	1.12	1.88	0.75	0.16	2.06	5.96
14	2022	1.15	1.88	0.75	0.17	2.06	6.00
15	2023	1.00	1.88	0.75	0.14	2.06	5.83
16	2024	0.99	1.88	0.75	0.16	2.06	5.84
17	2025	1.20	1.88	0.75	0.19	2.06	6.07
18	2026	1.05	1.88	0.75	0.15	2.06	5.88

Year		Direct Mine Labour ($)	Roof Support Materials ($)	Other Supplies ($)	Mine Extension Materials ($)	Maintenance & Repair Materials ($)	Total* ($/dt)
19	2027	1.06	1.88	0.75	0.17	2.06	5.92
20	2028	1.07	1.88	0.75	0.16	2.06	5.92
21	2029	0.98	1.88	0.75	0.17	2.06	5.84
22	2030	1.09	1.88	0.75	0.22	2.06	5.99
23	2031	1.33	1.88	0.75	0.00	2.06	6.01
24	2032	1.50	1.88	0.75	0.00	2.06	6.18
25	2033	1.49	1.88	0.75	0.00	2.06	6.18

Note: *Excludes administrative and equipment leasing costs.

17.1.7 SURFACE INFRASTRUCTURE DEVELOPMENT

The property will be developed to support the access, utilities and movement of ROM materials from the various underground mining portals, surface quarry site and tailings dam construction site using design plans developed by Wardrop Engineering.

CENTRAL FACILITIES

The mine plan will be supported by the building of three (3) central facilities (CF) strategically located in Boleo, Purgatorio, and Soledad arroyos. These facilities will each contain changing rooms, small offices, warehouse, shop(s) and other mine related facilities. Two facilities are scheduled to be built and in use by the end of project year 1. Constructing the third will be delayed until the second ten year period.

Transporting mine labour is planned as a two-step process. First, all mine labour would assembled at the plant prior to the start of shift for transport to the active central facility changing room. After being transported to their assigned CF, the miners would change clothing and obtain their mine safety and work equipment. The start of shift will commence at the boarding of mine transports from the central facility for the ride to specific work assignments in active mining districts.

Management, engineering, surveying, and other support activities are expected to report to their assigned central facility. Offices and storage have been allowed for.

POWER

Mine power will be supplied from the co-generation plant located near the processing plant using overhead high voltage lines to the central facilities where it will then be split to the active mining operations. The power will be stepped down to supply the needed voltages at the portal areas.

WATER

Water will be pumped to the central facilities and stored in larger tanks to provide for changes in consumption. Fire water for each underground mine will be drawn from the central facility tanks to portal tanks of sufficient size to meet fire-fighting requirements based on mine size.

17.2 ENVIRONMENTAL

The project is located within the Vizcaíno Biosphere Reserve, which is centered on the Desierto de Vizcaino on the west central coast of the Baja California Peninsula. Originally, the area included 1.5 Mha dedicated to the protection of the Western Coast as a refuge zone for the grey whale and the distribution zone of the Mexican antelope (Berrendo). In 1986, the area was extended south to protect the ancient paintings at the Sierra de San Francisco; the historic buildings, dating from the late 1800s, in the town of Santa Rosalía which are associated with the early mining of the Boleo district and the coast of the Sea of Cortes. Currently, the El Vizcaino Biosphere Reserve covers a total surface area of 2,546,790 ha as stipulated in the official decree dated November 30, 1988.

Although the area of the project is recognized as being significantly affected by historical mining activity, the fact that it is located within the boundaries of the biggest natural protected area in Mexico places considerable attention on its evaluation. Final approval is, jointly, decided by two independent branches within the Secretariat for the Environment and Natural Resources (SEMARNAT): the National Commission for Natural Protected Areas (CONANP) and the General Direction for Environmental Impact and Risk (DGIRA). Mention must be made of the fact that the surface property of MMB, where the project will be developed, has no significant archaeological value as determined by the National Institute for Anthropology and History (INAH). Additionally, all the buildings which are considered a cultural heritage are outside of the Boleo project and study area boundary and will not be affected by project development.

During 2006, MMB successfully completed a full Environmental Impact Assessment that covers the construction, operation, and closure phases of the Boleo 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 fieldwork to characterize fully 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 Boleo 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 permits that are more specific. 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.

Table 58 is a summary of environmental activity showing the various project phases and the required mitigation or compensation needed to address such activity.

As with the exploration permits, MMB has setup a permit management and compliance plan. The main actions taken to date are:

- A covenant agreement with CONANP was set up and signed on January of 2007. Through this mechanism, the agreed compensation funds will be made available to the El Vizcaino Biosphere Reserve as established in this agreement. As mandated by this legal instrument, a technical subcommittee that will oversee the use of these funds was set up in March 2007. MMB holds a seat at this subcommittee with full voting powers.
- An application for the authorization to mine in a natural protected area was submitted to CONANP.
- MMB has requested for an extension of the terms and conditions that had early deadlines (one to three months after permit issuance). The extension has been granted until August of 2007.
- MMB has initiated with the integration of the technical report to request a change in land use as mandated by the General Law of Sustainable Forestry Development. Additionally, it has engaged in ongoing negotiations with the Autonomous University of Baja California Sur to act as its lead educational institution for the integration, follow up and supervision of the main mitigation measures contained in the Environmental Impact Manifest and in the permit.

After the construction engineering of the project is finished, the final design will be compared with that submitted in the EIM and approved in the permit. As allowed by the Regulations of the LGEEPA on Environmental Impact, a modification of the original permit will be applied for if needed. Significant and environmentally adverse modifications to the original design could require a full impact assessment and the issuance of a supplementary permit.

Table 58: Environmental Activities

Phase of the Project	Activity	Environmental Parameter Affected	Mitigation Measure	Compensation Measure
Construction	Construction of facilities (camp, concrete plant, contention dyke, etc.)	Vegetation abundance and diversity as well as affectation to endemic and protected species	Vegetation Rescue and Transplant Program	Creation of a Trust Fund for CONAMP aimed at supporting conservation activities within the Biosphere Reserve of El Vizcaíno
	Construction and rehabilitation of access roads	Disturbance of passages, creating barriers for natural movement of fauna	Fauna Protection Plan Road design considering communicating passages (for fauna)	
Operation	Open Cuts	Changes in terrain slope	Protection against Erosion Program (program for soil recovery)	
		Changes in runoff patterns		
		Changes in soil permeability by the displacement of soil layer		
		Soil removal (change of type)		
		Abundance: removal of vegetation layer (abundance)	Vegetation Rescue and Transplant Program	
		Diversity: selective removal of vegetation species		
		Removal of protected and endemic vegetation species		
		Abundance: Fauna's driving away and/or elimination	Fauna Protection Plan	
		Interruption of fauna transit		
		Some species of protected or endangered species might be eliminated or driven away		
Operation	Tailings Dam	Changes in runoff patterns	Protection against Erosion Program (program for soil recovery)	Creation of a Trust Fund for CONAMP aimed at supporting conservation activities within the Biosphere Reserve of El Vizcaíno
		Soil removal (changes in permeability, composition and its characteristics)		
		Abundance: the flooding of the tailings dam will prevent vegetation from re-colonizing the area thus decreasing abundance	Vegetation Rescue and Transplant Program	
		Flooding of the tailings dam will cause some habitats to disappear hence decreasing vegetation and disappearance of endemic and protected flora		
		Abundance: Fauna's driving away and/or elimination	Plan de protección de fauna	
		Interruption of fauna transit		
		Some species of protected or endangered species might be eliminated or driven away		

17.3 COMMUNITY DEVELOPMENT

In order to comply with IFC standards and ensure the acceptance of the project from the population of Santa Rosalía, a Public Consultation and Disclosure Plan was integrated (PCDP). This plan included the following implementation stages:

- Speaker Training: that included a strength – weakness – threat – opportunity analysis
- Stakeholder Identification: derived from a fieldwork conducted at the State and Municipal levels
- Design of Communications Strategy
- Execution of Public Presentations of the project.

A detailed description of this process and its results is contained in the report of the PCDP. Salient points are summarized in the following paragraphs:

As a result of the field investigation to map out the stakeholders of the project, a communication strategy was set up for the public presentations of the project. Its main characteristics are:

- **Objectives:** to improve the community's knowledge of the project; to assess its acceptance and to address any concerns or opposition that may arise to its development.
- **Scope:** given the size and area of influence of the project, the consultation is to be directed to all persons or organizations, with a current and active presence in Santa Rosalía; the Municipality of Mulegé or the State of Baja California Sur, which show an interest to participate in the process.
- **Communication Contents:** background of the project and its sponsors; company vision; project description; legal compliance of the project; description of the environment and socio-economic factors in its area of influence; expected impacts of the project; prevention, mitigation, control and compensation measures; social responsibility and community wellbeing.
- **Communication Approach:** verbal communication aided by visual and written material designed for the needs of each selected audience.
- **Timing:** in the interval between the initial authorization of the project and the conclusion of the development of its construction engineering to include pertinent changes that could arise from the presentation process.
- **Speakers:** management team and support staff from the Mexican company that is promoting the project (MMB).

17.3.1 *Public Presentation Events*

The calendar of events is enlisted below:

- Participation of MMB with El Boleo Project during the 13th Annual Science and Technology Week celebrated at Santa Rosalía from October 23 to October 27, 2006.
- Presentation of El Boleo Project at the Hotel Fiesta Inn in La Paz by invitation and for selected stakeholders on February 1, 2007. The invitation included a CD that contained all files relative to the environmental permitting process of the project. A balanced mix of attendants, which included government officials; academia; entrepreneurs and civil society organizations (NGO's), was sought and achieved.
- Meetings with focus groups from Santa Rosalía from March 7 to March 8, 2007.
- Public Presentation of the project at Santa Rosalía on April 19, 2007.

There is strong support from the community who perceive it as an opportunity to promote economic growth for the region. No opposition to the project was detected during the celebration of these events.

17.3.2 *Grievance Mechanism*

MMB has pledged to establish a formal office to manage the relationship with the community once the availability of financing for the project is confirmed. Its initial project is to coordinate the integration of the Land Ordinance Plan that has the objective to identify the most pressing present and future needs of the community. This identification will enable all involved to set up activities to resolve these needs under a shared responsibility scheme. The liaison officer will have the task to establish a grievance mechanism to answer the concerns of all valid stakeholders.

18 ECONOMIC ASSESSMENT

18.1.1 SUMMARY

This economic assessment of El Boleo project is based upon:

- The mineral resource estimate for copper, cobalt and zinc prepared by Hellman and Schofield discussed in Section 16
- The open cut mine design developed by Australian Mine Development and Design, as discussed in Section 17.1
- The underground mine design developed by Agapito Associates, Inc., as discussed in Section 17.1.5.
- The process flowsheet developed by Bateman Engineering Canada Corp. 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 15.

US$ and metric measures are used throughout this section.

The current base-case is for annual mine production to deliver up to 3.1 Mdt/a to the process facility; with maximum annual metal production of 60,000 tonnes of copper, 3,100 tonnes of cobalt and 36,000 tonnes of zinc sulphate monohydrate.

Capital cost of the construction of the mine and process plant complex is currently estimated at US$568 million (including Owner’s Costs and Contingencies). Total operating costs are estimated at US$30.53/dt of ore treated, averaged over the 25 year project life being modelled. During the initial 25 years project life a total of 74.4 Mt of ore will be processed. The Total Measured and Indicated Resource is 253.2 dry metric tonnes, potentially leaving a large amount of ore to be processed after the initial 25 years.

Mine scheduling was based upon the scheduling of measured and indicated resource blocks using both copper and copper equivalent grades and applying process recovery factors. A copper equivalent grade was determined using the following base-case metal prices:

- Copper – US$1.50/lb
- Cobalt – US$15.00/lb
- Zinc – US$1.20/lb.

Process recoveries used in the economic analysis are as follows:

- Copper recovery – 91.2%
- Cobalt recovery – 78.2%
- Zinc recovery – 65.6%.

Base Case financial modelling of the project was done using conservative long-term metal prices. The assumed Base Case long-term copper price is $1.50, starting in 2013, with step down pricing between current levels and long term prices according to the LME 5 year forward price curve published as of May 11, 2007. Those prices are: $3.00 in 2009, $2.65 in 2010, $2.35 in 2011, and $2.00 in 2012.

Expressions of interest have been received from potential metal off-take 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 $15.00/lb flat over the life of the project. The Base Case price for zinc sulphate is $1200/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 mine schedule, indicates that the project is financially attractive at base-case metal prices. Financial modelling, using the base case prices and 25 years for the project life, shows that the project could generate a net after-tax Internal Rate of Return (IRR) of 24.7% with a discounted present value, at an 8% discount rate, of US$700.3 million.

Using a 6% discount rate generates an NPV, after tax, of $US924.0 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 59:

- 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 $2.25/lb for copper and $16.00/lb for cobalt.
- The current prices which, as of the end of January 2007 are $3.50/lb of copper and $30.00/lb of cobalt and $1,500/t 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.

The project is sensitive to four 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 59: 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 5
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$568 million
24 year average Operating Cost, excluding start-up year	US$30.53/t of ore
Long term metal prices	Copper – US$1.50/lb Cobalt – US$15.00/lb ZSM – US$1200/t
(After tax) Internal rate of return (IRR)	24.7 %

Table 60: Sensitivity to Key Variables

	After Tax IRR				After Tax NPV at 8% ($ millions)		
Variable	-10%	Base Case	+10%		-10%	Base Case	+10%
Copper Price	21.6%	24.7%	27.5%		$571	$700	$822
Cobalt Price	24.1%	24.7%	25.3%		$663	$700	$738
Capital Cost	27.3%	24.7%	22.3%		$744	$700	$652
Operating Cost	26.0%	24.7%	23.4%		$765	$700	$635

18.1.2 Product Marketing

The processing facility at El Boleo 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 micronutrient 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 off-take candidates that support the assumed pricing structure.

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 off-take 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

A number of specialist organizations have contributed to the development of the capital cost of implementing the Boleo Project. These organizations are listed in Table 61, which shows a summary of 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 61: 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 Geotechnical	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

Table 62 provides a summary breakdown of the estimated total project capital costs. The base date of the cost estimate is July 31, 2006. There is no allowance for escalation. All costs are listed in US$. The numbers are rounded to reflect that they are approximations.

Table 62: Capital Cost Estimate Summary

Project Area	Estimate Capital Cost US$
Overall Site	39,142,000
Mining	59,678,000
Process Plant	160,322,000
Services & Infrastructure	94,285,000

Project Area	Estimate Capital Cost US$
Buildings	14,956,000
Construction Indirects & Freight	39,063,000
Sub-total – Direct Field Costs	407,446,000
EPCM	45,805,000
Owners Costs	36,773,000
Contingency	62,349,000
Mine Pre-Development	16,019,000
Total – Project Capital Cost	**568,392,000**

The overall capital cost is estimated at US$568 million.

This number includes the capital cost components of all infrastructures, 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, other owners' costs and various community initiatives.

The expected range, estimate accuracy, and contingency of the estimate were assessed through a formal process of estimate review and risk assessment during the study, culminating in a statistical analysis of the cost and quantity data using @RISK software to arrive at the final figures. This contingency – also known as the estimating accuracy allowance – covers quantity and costing variability only. Risk contingency is not included.

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
- escalation from the cost estimate base date of July 31, 2006.

Provision for the above costs – with the exception of escalation – has been made in Baja Mining Corp's economic model. 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 on the basis of a 25-year mine 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 25 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 25 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 Engineering Canada Corp. and Baja Mining Corp. from the following sources:

- Extensive bench scale metallurgical and pilot plant testwork 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 Boleo 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 Boleo 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.

Operating costs will vary over the life-of-mine as a function of tonnage treated and ROM ore grade. Operating costs are generally presented by cost element and cost information is developed in United States dollars with a base date of 3Q 2006. There is no allowance for escalation or contingency in the numbers presented.

Table 63 shows the operating costs for a typical year of operation.

Table 63: Operating Cost Summary Estimate

Operating Cost Category	Annual Operating Cost (US$ '000s) *
Mining Costs	
Labour	4,1588
Fuel	1,040
Operating Leases	4,616
Consumables	19,916
Limestone credit	(3,494)

Operating Cost Category	Annual Operating Cost (US$ '000s) *
Total Mining	**26,236**
Process Plant	
Labour	6,936
Sulphur	19,726
Soda Ash	6,5340
Flocculant	3,968
Limestone	9,334
LIX 84	663
LIX 63	2,180
Other Supplies	4,181
Maintenance Supplies	4,898
Fleet Leases	0
Lab Operation	268
Diesel (Power)	3,841
Total Process Plant	**62,526**
Insurance	1,600
Total Operating Cost	**90,362**
Total Operating Cost per ROM Tonne Treated	**28.96**
Total Freight, Selling & Distribution Costs	**2,701**
Total Cash Cost Before Taxes	**93,063**
Cash Cost per Tonne of Ore	**29.83**

Note: * Year 5 of the LOM, 3.1 Mdt/a plant feed.

18.1.5 SUMMARY

The financial model utilizes the feasibility study mine production schedule, showing a 25 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 Canada Corp., and projected metal prices.

The Economic Analysis of the Boleo project has been conducted by Baja Mining Corp. on an all equity basis (with no financial leveraging) using the Discounted Cash Flows, after taxes, for the construction period and first 25 years of the project life. The project is not limited to 25 years by the orebody, and it can be expected that the project will continue beyond that length of time, adding to the 25-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.

On an all-equity basis the Feasibility Study indicates that the Boleo project has a payback period for invested capital of 42 months from 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$924.0 million, or $700.3 million at 8%, and the internal rate of return on investment is 24.7% 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 of slightly less than zero cents per pound of LME grade copper production averaged over the 25 years of operation. Using higher by-product prices, but still below current levels, will drop the net cash cost of production of copper significantly below zero. The addition of a Manganese product could also drop the net cash cost of copper to significantly less than zero.

18.2 BOLEO PROJECT ECONOMIC ANALYSIS ASSUMPTIONS AND DISCUSSION

18.2.1 CAPITAL COST & EXPENDITURE TIMING

The total project capital costs for the construction of the mine and process plant are summarised in Section 18.1.3 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 than 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 Rosalía 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 Boleo Project development timing is shown in Table 64. The timing of these activities forms the basis of timing of the cash flows in the economic model.

Table 64: Summary of Boleo 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	55%
1	Start-up of copper production Construction of DSX circuit, cobalt EW, ZSM granulation	35%
2	Start-up of cobalt and zinc production	

18.2.2 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 1% 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, 6 and 11.

18.2.3 WORKING CAPITAL

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

- the net balance of accounts payable and receivable
- inventories, including first fills of reagents – US$8 million in year 1, US$15 million in year 2 and warehouse inventory of US$1 million in year 1
- capital spares – US$6.25 million in year 1, equivalent to 1.1% of total capital expenditure
- 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 off-take parties.

18.2.4 REVENUE

- Net Sales Revenue from the sale of copper is calculated based on 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 $1.50/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 $15.00/lb is assumed for cobalt, relative to the current level of $30, 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 $1,200/t of ZSM is assumed, before the deductions for freight and commissions.

18.2.5 DEPRECIATION & TAXATION

- Mexican tax laws provide for two 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.

18.2.6 MANGANESE OPPORTUNITY CASE

- The Boleo orebody contains a substantial amount of manganese 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.
- The fertilizer and animal feed market could potentially absorb approximately one third of the anticipated production of manganese from Boleo.
- Another potential end product for the conversion of manganese carbonate is electrolytic manganese dioxide. The current estimated price of manganese carbonate is $1,000/t ($2,000/t of contained Mn) and that of manganese sulphate is $550/t ($1,212/t of contained Mn).
- The largest market for manganese would be in the form of metal, which can easily be produced by introducing the manganese carbonate into an electrolytic manganese metal production facility. These types of operations consume large amounts of electric power and are generally located close to a source of low cost power. With 48% Mn content the carbonate product from Boleo is the highest concentration intermediate product that can be produced and it is easily transportable to a refinery. A third party refiner would charge a conversion cost reflecting the cost of power and their capital investment. Electrolytic manganese metal is selling for $5,000/t at the current (late May 2007) time.
- An opportunity case is included in the scenarios below, which assumes a selling price of $650/t for manganese carbonate, or $1,350/t of Mn contained.

18.3 ECONOMIC ASSESSMENT OUTPUTS

18.3.1 NPV & IRR DETERMINATIONS

Tables 65 to 68 contain NPV and IRR outputs from detailed calculations of the project economics at various metal prices.

Table 65: Economic Assessment – Base Case Summary

		Pre-Start-up	Yr 1	Yr 2 - 5	Yr 5-10	Yr 10-15	Yr 15-20	Yr 21-25	Yr 1 - 25
Capital (total)		($369,455)	($198,937)	Average	Average	Average	Average	Average	Average
Ore grades:	Cu	%	2.01	2.15	1.80	1.60	1.05	0.93	1.52
	Co	%	0.071	0.071	0.067	0.074	0.091	0.084	0.076
	Zn	%	0.60	0.36	0.46	0.59	0.61	0.77	0.57
	Mn	%	2.93	2.02	2.50	2.46	3.30	4.21	2.93
Ore treated		t/a	650	2,828	3,120	3,120	3,120	3,120	3,071
Production:	Cu	t/a	12,024	55,755	51,389	45,486	29,909	26,391	41,204
	Co		0	1,535	1,644	1,796	2,222	2,061	1,865
	ZSM		0	17,917	26,702	34,361	35,692	45,131	32,546
Revenue:	Cu	$000/a	$80,054	$259,541	$172,197	$152,416	$100,222	$88,433	$150,188
	Co		$0	$50,771	$54,379	$59,399	$73,491	$68,160	$61,676
	ZSM		$0	$21,501	$32,042	$41,233	$42,830	$54,157	$39,055
	Total		$80,054	$331,813	$258,617	$253,049	$216,544	$210,750	$250,919
Op. Costs:	Mining	$000/a	$28,283	$27,899	$23,932	$25,418	$24,771	$15,920	$23,408
	Process		$14,989	$58,824	$63,998	$65,103	$61,669	$71,221	$64,386
	G & A		$989	$2,346	$1,877	$1,876	$1,880	$1,849	$1,871
	Sales, Dist'n		$104	$2,549	$3,457	$4,306	$4,415	$4,778	$4,097
	Total		$44,365	$91,618	$93,264	$96,702	$92,736	$93,768	$93,762
			Yr 1	Yr 2 - 5	Yr 5-10	Yr 11-15	Yr 15-20	Yr 21-25	24 Yr Avg.
Unit Op Costs:	Mining	$/t ore	$43.51	$9.87	$7.67	$8.15	$7.94	$5.10	$7.62
(wtd averages)	Process		$23.06	$20.80	$20.51	$20.87	$19.77	$22.83	$20.96
	G & A		$1.52	$0.66	$0.60	$0.60	$0.60	$0.59	$0.61
	Sales, Dist'n		$0.16	$0.90	$1.11	$1.38	$1.42	$1.75	$1.33
	Total		$68.25	$32.24	$29.89	$30.99	$29.72	$30.27	$30.53
Before Tax Cash Flow		$000/a	($190,173)	$228,900	$161,297	$149,124	$118,584	$116,730	$151,845
After Tax Cash Flow		$000/a	($190,173)	$196,925	$116,007	$107,086	$85,660	$89,303	$115,749
Earnings		$000/a	($472,251)	$202,159	$116,459	$108,096	$84,661	$70,528	$112,806
Cash Cost/lb Cu:									
	Gross	$/lb CuEq.	$1.98	$0.56	$0.55	$0.58	$0.65	$0.69	$0.61
	Net of by-product credits		$1.98	$0.18	$0.06	($0.04)	($0.36)	($0.48)	($0.07)

Note: Long Term Prices: Copper $1.50/lb, Cobalt $15.00/lb, Zinc Sulphate $1,200/t

After Tax 25 year IRR:		24.69%
NPV at	0%	$2,218,355
	6%	$924,017
	8%	$700,286

Table 66: Boleo Preliminary Economic Assessment – 5 Year Price Case Summary

		Pre-Start-up	Yr 1	Yr 2 - 5	Yr 5 - 10	Yr 10-15	Yr 15 - 20	Yr 21-25	Yr 1 - 25
Capital (total)		($369,455)	($198,937)	Average	Average	Average	Average	Average	Average
Ore grades:	Cu	%	2.01	2.15	1.80	1.60	1.05	0.93	1.52
	Co	%	0.071	0.071	0.067	0.074	0.091	0.084	0.076
	Zn	%	0.60	0.36	0.46	0.59	0.61	0.77	0.57
	Mn	%	2.93	2.02	2.50	2.46	3.30	4.21	2.93
Ore treated		t/a	650	2,828	3,120	3,120	3,120	3,120	3,071
Production:	Cu	t/a	12,024	55,755	51,389	45,486	29,909	26,391	41,204
	Co		0	1,535	1,644	1,796	2,222	2,061	1,865
	ZSM		0	17,917	26,702	34,361	35,692	45,131	32,546
Revenue:	Cu	$000/a	$60,173	$279,009	$257,162	$227,622	$149,673		$132,068
	Co		$0	$54,156	$58,004	$63,359	$78,391		$72,704
	ZSM		$0	$21,501	$32,042	$41,233	$42,830		$54,157
	Total		$60,173	$354,666	$347,208	$332,214	$270,894		$258,928
Op. Costs:	Mining	$000/a	$28,283	$27,899	$23,932	$25,418	$24,771	$15,920	$23,408
	Process		$14,989	$58,824	$63,998	$65,103	$61,669	$71,221	$64,386
	G&A		$989	$2,346	$1,877	$1,876	$1,880	$1,849	$1,871
	Sales, Dist'n		$84	$2,569	$3,542	$4,381	$4,465	4,819	$4,153
	Total		$45,476	$91,637	$93,349	$96,778	$92,786	$93,809	$93,818
			Yr 1	Yr 2 - 5	Yr 5-10	Yr 11-15	Yr 15 - 20	Yr 21-25	24 Yr Avg.
Unit Op Costs	Mining	$/t ore	$43.51	$9.87	$7.67	$8.15	$7.94	$5.10	$7.62
Weighted Averages	Process		$23.57	$20.80	$20.51	$20.87	$19.77	$22.83	$20.96
	G&A		$2.75	$0.66	$0.60	$0.60	$0.60	$0.59	$0.61
	Sales, Dist'n		$0.13	$0.91	$1.14	$1.40	$1.43	$1.76	$1.35
	Total		$69.96	$32.24	$29.92	$31.02	$29.74	$30.28	$30.55
Before Tax Cash Flow		$000/a	($211,260)	$251,757	$249,803	$228,214	$172,885	$164,865	$211,911
After Tax Cash Flow		$000/a	($211,260)	$211,937	$179,731	$164,031	$124,757	$123,960	$158,756
Earnings		$000/a	($493,244)	$217,148	$180,184	$165,041	$123,758	$105,185	$155,809
Cash Cost/lb Cu:	Gross	$/lb CuEq.	$2.02	$0.61	$0.61	$0.66	$0.79	$0.84	$0.71
	Net of by-product credits		$2.02	$0.16	$0.03	($0.08)	($0.43)	($0.56)	($0.11)

Note: Long Term Metal Prices: Cu: $2.25, Co: $16.00, $ZnSO_4$: $1200

After Tax 25 year IRR:		27.65%
NPV at	0%	$3,229,435
	6%	$1,366,522
	8%	$1,045,515

Table 67: Boleo Preliminary Economic Assessment – Current Price Case Summary

		Pre-Start-up	Yr 1	Yr 2 - 5	Yr 5 - 10	Yr 10 - 15	Yr 15 - 20	Yr 21-25	Yr 1 - 25
Capital (total)		($369,455)	($198,937)	Average	Average	Average	Average	Average	Average
Ore grades:	Cu	%	2.01	2.15	1.80	1.60	1.05	0.93	1.52
	Co	%	0.071	0.071	0.067	0.074	0.091	0.084	0.076
	Zn	%	0.60	0.36	0.46	0.59	0.61	0.77	0.57
	Mn	%	2.93	2.02	2.50	2.46	3.30	4.21	2.93
Ore treated		t/a	650	2,828	3,120	3,120	3,120	3,120	3,071
Production:	Cu	t/a	12,024	55,755	51,389	45,486	29,909	26,391	41,204
	Co		0	1,535	1,644	1,796	2,222	2,061	1,865
	ZSM		0	17,917	26,702	34,361	35,692	45,131	32,546
Revenue:	Cu	$000/a	$93,308	$432,649	$398,771	$352,964	$232,093	$204,792	$319,737
	Co		$0	$101,542	$108,757	$118,799	$146,983	$136,320	$123,352
	ZSM		$0	$26,876	$40,053	$51,541	$53,538	$67,696	$48,818
	Total		$93,308	$561,067	$547,581	$523,304	$432,613	$408,808	$491,908
Op. Costs:	Mining	$000/a	$28,283	$27,899	$23,932	$25,418	$24,771	$15,920	$23,408
	Process		$14,989	$58,824	$63,998	$65,103	$61,669	$71,221	$64,386
	G & A		$989	$2,346	$1,877	$1,876	$1,880	$1,849	$1,871
	Sales, Dist'n		$117	$2,722	$3,683	$4,506	$4,547	$4,886	$4,266
	Total		$45,509	$91,791	$93,491	$96,903	$92,868	$93,876	$93,931
			Yr 1	Yr 2 - 5	Yr 5-10	Yr 11-15	Yr 15-20	Yr 21-25	24 Yr Avg.
Unit Op Costs:	Mining	$/t ore	$43.51	$9.87	$7.67	$8.15	$7.94	$5.10	$7.62
(weighted averages)	Process		$23.57	$20.80	$20.51	$20.87	$19.77	$22.83	$20.96
	G & A		$2.75	$0.66	$0.60	$0.60	$0.60	$0.59	$0.61
	Sales, Dist'n		$0.18	$0.96	$1.18	$1.44	$1.46	$1.78	$1.39
	Total		$70.01	$32.30	$29.96	$31.06	$29.77	$30.31	$30.58
Before Tax Cash Flow		$000/a	($178,158)	$458,004	$450,034	$419,179	$334,522	$314,672	$392,669
After Tax Cash Flow		$000/a	($178,158)	$361,039	$323,898	$301,526	$241,136	$231,821	$289,002
Earnings		$000/a	($460,142)	$366,249	$324,350	$302,536	$240,136	$213,046	$286,056
Cash Cost/lb Cu:	Gross	$/lb CuEq.	$2.02	$0.60	$0.60	$0.65	$0.77	$0.83	$0.69
	Net of by-product credits		$2.02	($0.27)	($0.49)	($0.73)	($1.63)	($1.88)	($0.85)

Note: Current Metal Prices Cu: $3.50, Co: $30.00, $ZnSO_4$: $1,500

After Tax 25 year IRR:		45.95%
NPV at	0%	$6,388,444
	6%	$2,912,886
	8%	$2,313,665

Table 68: Boleo PEA – Opportunity: Base Case with Manganese Production

		Pre-Start-up	Yr 1	Yr 2 - 5	Yr 5 - 10	Yr 10 -15	Yr 15 - 20	Yr 21-25	Yr 1 - 25
Capital (total):		($383,500)	($206,500)	Average	Average	Average	Average	Average	Average
Ore grades:	Cu	%	2.01	2.15	1.80	1.60	1.05	0.93	1.52
	Co	%	0.071	0.071	0.067	0.074	0.091	0.084	0.076
	Zn	%	0.60	0.36	0.46	0.59	0.61	0.77	0.57
	Mn	%	2.93	2.02	2.50	2.46	3.30	4.21	2.93
Ore treated		t/y	650	2,828	3,120	3,120	3,120	3,120	3,071
Production:	Cu	t/a	12,024	55,755	51,389	45,486	29,909	26,391	41,204
	Co		0	1,535	1,644	1,796	2,222	2,061	1,865
	ZSM		0	17,917	26,702	34,361	35,692	45,131	32,546
Revenue:	Cu	$000/a	$80,054	$259,541	$172,197	$152,416	$100,222	$88,433	$150,188
	Co		$0	$50,771	$54,379	$59,399	$73,491	$68,160	$61,676
	ZSM		$0	$21,501	$32,042	$41,233	$42,830	$54,157	$39,055
	Total		$80,054	$374,501	$345,951	$339,139	$331,791	$357,887	$348,827
Op. Costs:	Mining	$000/a	$28,283	$27,899	$23,932	$25,418	$24,771	$15,920	$23,408
	Process		$15,320	$70,545	$87,979	$88,742	$93,314	$111,623	$91,270
	G&A		$1,789	$2,346	$1,877	$1,876	$1,880	$1,849	$1,871
	Sales, Dist'n		$199	$104	$5,833	$10,175	$10,928	$13,281	$15,023
	Total		$45,496	$106,623	$123,962	$126,964	$133,247	$144,414	$128,177
			Yr 1	Yr 2 - 5	Yr 5 - 10	Yr 11-15	Yr 15-20	Yr 21-25	24 Yr Avg.
Unit Op Costs:	Mining	$/t ore	$43.51	$9.87	$7.67	$8.15	$7.94	$5.10	$7.62
(wtd averages)	Process		$23.57	$24.95	$28.20	$28.44	$29.91	$35.78	$29.72
	G & A		$2.75	$0.66	$0.60	$0.60	$0.60	$0.59	$0.61
	Sales, Dist'n		$0.16	$2.06	$3.26	$3.50	$4.26	$5.37	$3.79
	Total		$69.99	$37.54	$39.73	$40.69	$42.71	$46.85	$41.73
Before Tax Cash Flow		$000/a	($199,199)	$256,112	$217,665	$204,851	$192,892	$212,908	$215,251
After tax Cash Flow		$000/a	($199,199)	$217,817	$156,544	$147,208	$139,077	$158,953	$161,674
Earnings		$000/a	($492,389)	$223,446	$157,169	$148,225	$138,382	$138,741	$158,599
Cash Cost/lb Cu:	Gross	$/lb CuEq.	$2.02	$0.64	$0.73	$0.76	$0.94	$1.07	$0.84
	Net of by-product credits		$2.02	($0.04)	($0.44)	($0.60)	($1.49)	($2.12)	($0.77)

Note: Long Term Metal Prices: Cu: $1.50, Co: $15.00, $MnCO_3$: $650, $ZnSO_4$: $1200

After Tax 25 year IRR:		27.53%
NPV at	0%	$3,297,474
	6%	$1,345,286
	8%	$1,020,121

18.3.2 *Annual Cash Flow Determinations*

Tables 69 to 72 contain cash flow outputs from the economic models on a year by year basis.

Table 69: 25 Year Detailed Cash Flow – Base Case

		2007	2008	2009	2010	2011	2012	2013	2014
Throughput:	dmt/a			650	2,340	2,860	2,990	3,120	3,120
Grades:	%Cu			2.01	2.20	2.13	2.16	2.09	1.73
	%Co			0.07	0.07	0.07	0.07	0.07	0.07
	%Zn			0.60	0.45	0.31	0.28	0.41	0.34
Recoveries:	Cu			92.0%	92.0%	92.0%	92.0%	92.0%	92.0%
	Co			0.0%	60.0%	80.0%	80.0%	80.0%	80.0%
	Zn			0.0%	50.0%	65.6%	65.6%	65.6%	65.6%
Production:	t/a Cu			12,024	47,395	56,143	59,429	60,052	49,633
	t/a Co			0	984	1,624	1,723	1,810	1,688
	t/a ZSM			0	15,209	16,667	15,934	23,860	20,056
Prices:	Cu			$3.00	$2.65	$2.35	$2.00	$1.50	$1.50
	Co			$15.00	$15.00	$15.00	$15.00	$15.00	$15.00
	ZSM			$1,200	$1,200	$1,200	$1,200	$1,200	$1,200
Revenue:	Cu			$80,054	$278,966	$293,329	$264,645	$201,225	$166,312
($000's)	Co			$0	$32,528	$53,710	$56,979	$59,867	$55,817
	Zn			$0	$18,251	$20,000	$19,121	$28,632	$24,067
	Total			**$80,054**	**$329,745**	**$367,040**	**$340,744**	**$289,724**	**$246,196**
Total Op Cost $000's				**$52,365**	**$93,215**	**$92,686**	**$94,856**	**$98,829**	**$93,340**
Initial Capital		$56,839	$312,616	$198,937					
Sustaining Capital				$0	$1,460	$2,558	$4,995	$3,781	$8,798
Working Capital at YE				$9,925	$12,565	$13,749	$13,938	$14,198	$13,783
Working Capital Change				$9,925	$2,640	$1,184	$189	$260	-$416
Income Taxes				$0	$0	$7,738	$67,632	$52,529	$40,657
Net Cash Flow		($56,839)	($312,616)	($190,173)	$217,429	$262,874	$173,073	$134,324	$103,817
		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	1.65	2.10	1.76	1.77	1.40	1.48	1.71	1.86
	%Co	0.06	0.07	0.07	0.07	0.07	0.08	0.08	0.08
	%Zn	0.34	0.52	0.54	0.54	0.49	0.55	0.69	0.57
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	47,031	59,617	50,161	50,503	39,702	42,225	48,622	52,913
	t/a Co	1,455	1,615	1,690	1,775	1,656	1,865	1,889	1,847
	t/a ZSM	19,788	30,370	31,784	31,510	28,379	32,180	40,334	33,117
Prices:	Cu	$1.50	$1.50	$1.50	$1.50	$1.50	$1.50	$1.50	$1.50
	Co	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00
	ZSM	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200
Revenue:	Cu	$157,594	$199,768	$168,082	$169,227	$133,036	$141,488	$162,923	$177,304
($000's)	Co	$48,111	$53,409	$55,873	$58,684	$54,765	$61,666	$62,472	$61,077
	Zn	$23,746	$36,444	$38,141	$37,813	$34,055	$38,616	$48,401	$39,740
	Total	**$229,451**	**$289,621**	**$262,096**	**$265,723**	**$221,856**	**$241,770**	**$273,796**	**$278,121**
Total Op Cost $000's		**$90,648**	**$93,864**	**$94,025**	**$94,442**	**$91,157**	**$93,164**	**$101,454**	**$97,720**
Sustaining Capital		$2,991	$2,992	$2,992	$2,942	$6,524	$14,489	$6,901	$3,879
Working Capital at YE		$13,563	$13,725	$13,729	$13,766	$13,527	$13,657	$14,266	$14,022
Working Capital Change		-$219	$162	$4	$36	-$238	$129	$610	-$245
Income Taxes		$38,136	$54,083	$46,331	$47,242	$35,006	$38,080	$46,574	$49,567
Net Cash Flow		$97,895	$138,520	$118,744	$121,060	$89,407	$95,907	$118,256	$127,199

Table continues…

			2023	2024	2025	2026	2027	2028
Throughput:	dmt/a		3,120	3,120	3,120	3,120	3,120	3,120
Grades:	%Cu		1.55	1.58	1.14	0.89	0.93	0.71
	%Co		0.07	0.08	0.11	0.09	0.08	0.09
	%Zn		0.65	0.45	0.43	0.70	0.64	0.82
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		43,968	45,076	32,449	25,291	26,436	20,295
	t/a Co		1,724	2,050	2,604	2,311	2,014	2,133
	t/a ZSM		37,794	26,239	25,378	41,223	37,409	48,210
Prices:	Cu		$1.50	$1.50	$1.50	$1.50	$1.50	$1.50
	Co		$15.00	$15.00	$15.00	$15.00	$15.00	$15.00
	ZSM		$1,200	$1,200	$1,200	$1,200	$1,200	$1,200
Revenue:	Cu		$147,331	$151,043	$108,733	$84,747	$88,582	$68,004
($000's)	Co		$57,016	$67,788	$86,110	$76,420	$66,605	$70,534
	Zn		$45,353	$31,486	$30,454	$49,468	$44,891	$57,852
	Total		**$249,700**	**$250,318**	**$225,296**	**$210,635**	**$200,078**	**$196,391**
Total Op Cost $000's			**$100,016**	**$93,308**	**$88,070**	**$95,245**	**$90,141**	**$96,916**
Sustaining Capital			$3,911	$6,365	$4,161	$5,966	$5,626	$4,347
Working Capital at YE			$14,173	$13,720	$13,293	$13,747	$13,360	$7,573
Working Capital Change			$152	-$453	-$428	$454	-$387	-$5,787
Income Taxes			$40,959	$42,412	$37,410	$30,856	$29,412	$24,528
Net Cash Flow			$104,663	$108,685	$96,083	$78,114	$75,285	$70,134

			2029	2030	2031	2032	2033
Throughput:	dmt/a		3,120	3,120	3,120	3,120	3,120
Grades:	%Cu		0.64	0.77	1.14	1.09	0.99
	%Co		0.07	0.07	0.07	0.09	0.12
	%Zn		0.75	0.99	0.71	0.96	0.44
Recoveries:	Cu		91.2%	91.2%	91.2%	91.2%	91.2%
	Co		78.2%	78.2%	78.2%	78.2%	78.2%
	Zn		65.6%	65.6%	65.6%	65.6%	65.6%
Production:	t/a Cu		18,330	21,804	32,577	31,080	28,165
	t/a Co		1,801	1,642	1,615	2,260	2,988
	t/a ZSM		43,774	58,111	41,585	56,261	25,922
Prices:	Cu		$1.50	$1.50	$1.50	$1.50	$1.50
	Co		$15.00	$15.00	$15.00	$15.00	$15.00
	ZSM		$1,200	$1,200	$1,200	$1,200	$1,200
Revenue:	Cu		$61,422	$73,063	$109,160	$104,144	$94,376
($000's)	Co		$59,545	$54,313	$53,396	$74,737	$98,809
	Zn		$52,529	$69,733	$49,902	$67,513	$31,106
	Total		**$173,497**	**$197,109**	**$212,458**	**$246,394**	**$224,291**
Total Op Cost $000's			$96,708	$98,905	$89,766	$100,402	$86,403
Sustaining Capital			$4,212	$2,842	$2,842	$2,842	$0
Working Capital at YE			$7,602	$7,651	$7,041	$7,784	$0
Working Capital Change			$29	$49	-$610	$743	-$7,784
Income Taxes			$20,284	$26,318	$33,558	$40,082	$16,896
Net Cash Flow			$52,264	$68,995	$86,901	$102,325	$136,028

Table 70: 25 Year Detailed Cash Flow – 5 Year Prices Case – (3 year trailing + 2 year leading)

		2007	2008	2009	2010	2011	2012
Throughput:	t/a			650	2,340	2,860	2,990
Grades:	%Cu			2.01	2.20	2.13	2.16
	%Co			0.07	0.07	0.07	0.07
	%Zn			0.60	0.45	0.31	0.28
Recoveries:	Cu			92.0%	92.0%	92.0%	92.0%
	Co			0.0%	60.0%	80.0%	80.0%
	Zn			0.0%	50.0%	65.6%	65.6%
Production:	t/a Cu			12,024	47,395	56,143	59,429
	t/a Co			0	984	1,624	1,723
	t/a ZSM			0	15,209	16,667	15,934
Prices:	Cu			$2.25	$2.25	$2.25	$2.25
	Co			$16.00	$16.00	$16.00	$16.00
	ZSM			$1,200	$1,200	$1,200	$1,200
Revenue:	Cu			$60,173	$237,173	$280,953	$297,398
($000's)	Co			$0	$34,697	$57,291	$60,778
	Zn			$0	$18,251	$20,000	$19,121
	Total			**$60,173**	**$290,121**	**$358,244**	**$377,296**
Total Op Cost $000's				**$53,476**	**$93,174**	**$92,674**	**$94,889**
Initial Capital		$56,839	$312,616	$198,937			
Sustaining Capital				$0	$1,460	$2,558	$4,995
Working Capital at YE				$10,019	$12,565	$13,749	$13,938
Working Capital Change				$10,019	$2,546	$1,184	$189
Income Taxes				$0	$0	$0	$77,857
Net Cash Flow		($56,839)	($312,616)	($211,260)	$177,941	$261,828	$199,366
		2015	**2016**	**2017**	**2018**	**2019**	**2020**
Throughput:	dmt/a	3,120	3,120	3,120	3,120	3,120	3,120
Grades:	%Cu	1.65	2.10	1.76	1.77	1.40	1.48
	%Co	0.06	0.07	0.07	0.07	0.07	0.08
	%Zn	0.34	0.52	0.54	0.54	0.49	0.55
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	47,031	59,617	50,161	50,503	39,702	42,225
	t/a Co	1,455	1,615	1,690	1,775	1,656	1,865
	t/a ZSM	19,788	30,370	31,784	31,510	28,379	32,180
Prices:	Cu	$2.25	$2.25	$2.25	$2.25	$2.25	$2.25
	Co	$16.00	$16.00	$16.00	$16.00	$16.00	$16.00
	ZSM	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200
Revenue:	Cu	$235,355	$298,337	$251,017	$252,727	$198,678	$211,301
($000's)	Co	$51,318	$56,969	$59,598	$62,596	$58,416	$65,777
	Zn	$23,746	$36,444	$38,141	$37,813	$34,055	$38,616
	Total	**$310,419**	**$391,751**	**$348,756**	**$353,135**	**$291,149**	**$315,694**
Total Op Cost $000's		$90,725	$93,963	$94,108	$94,526	$91,223	$93,234
Sustaining Capital		$2,991	$2,992	$2,992	$2,942	$6,524	$14,489
Working Capital at YE		$13,563	$13,725	$13,729	$13,766	$13,527	$13,657
Working Capital Change		-$219	$162	$4	$36	-$238	$129
Income Taxes		$60,786	$82,652	$70,573	$71,694	$54,390	$58,759
Net Cash Flow		$156,136	$211,983	$181,080	$183,937	$139,251	$149,082

Table continues…

		2023	2024	2025	2026	2027	2028
Throughput:	dmt/a	3,120	3,120	3,120	3,120	3,120	3,120
Grades:	%Cu	1.55	1.58	1.14	0.89	0.93	0.71
	%Co	0.07	0.08	0.11	0.09	0.08	0.09
	%Zn	0.65	0.45	0.43	0.70	0.64	0.82
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	43,968	45,076	32,449	25,291	26,436	20,295
	t/a Co	1,724	2,050	2,604	2,311	2,014	2,133
	t/a ZSM	37,794	26,239	25,378	41,223	37,409	48,210
Prices:	Cu	$2.25	$2.25	$2.25	$2.25	$2.25	$2.25
	Co	$16.00	$16.00	$16.00	$16.00	$16.00	$16.00
	ZSM	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200
Revenue:	Cu	$220,027	$225,572	$162,384	$126,563	$132,290	$101,559
($000's)	Co	$60,817	$72,307	$91,851	$81,514	$71,045	$75,237
	Zn	$45,353	$31,486	$30,454	$49,468	$44,891	$57,852
	Total	**$326,198**	**$329,365**	**$284,688**	**$257,546**	**$248,226**	**$234,648**
Total Op Cost $000's		$100,088	$93,383	$88,124	$95,287	$90,185	$96,950
Sustaining Capital		$3,911	$6,365	$4,161	$5,966	$5,626	$4,347
Working Capital at YE		$14,173	$13,720	$13,293	$13,747	$13,360	$7,573
Working Capital Change		$152	-$453	-$428	$454	-$387	-$5,787
Income Taxes		$62,358	$64,524	$54,024	$43,979	$42,881	$35,231
Net Cash Flow		$159,688	$165,545	$138,806	$111,860	$109,921	$97,655

		2029	2030	2031	2032	2033
Throughput:	dmt/a	3,120	3,120	3,120	3,120	3,120
Grades:	%Cu	0.64	0.77	1.14	1.09	0.99
	%Co	0.07	0.07	0.07	0.09	0.12
	%Zn	0.75	0.99	0.71	0.96	0.44
Recoveries:	Cu	91.2%	91.2%	91.2%	91.2%	91.2%
	Co	78.2%	78.2%	78.2%	78.2%	78.2%
	Zn	65.6%	65.6%	65.6%	65.6%	65.6%
Production:	t/a Cu	18,330	21,804	32,577	31,080	28,165
	t/a Co	1,801	1,642	1,615	2,260	2,988
	t/a ZSM	43,774	58,111	41,585	56,261	25,922
Prices:	Cu	$2.25	$2.25	$2.25	$2.25	$2.25
	Co	$16.00	$16.00	$16.00	$16.00	$16.00
	ZSM	$1,200	$1,200	$1,200	$1,200	$1,200
Revenue:	Cu	$91,729	$109,114	$163,022	$155,531	$140,943
($000's)	Co	$63,515	$57,934	$56,956	$79,719	$105,396
	Zn	$52,529	$69,733	$49,902	$67,513	$31,106
	Total	**$207,773**	**$236,780**	**$269,879**	**$302,763**	**$277,445**
Total Op Cost $000's		**$96,738**	**$98,941**	**$89,820**	**$100,453**	**$86,450**
Sustaining Capital		$4,212	$2,842	$2,842	$2,842	$0
Working Capital at YE		$7,602	$7,651	$7,041	$7,784	$0
Working Capital Change		$29	$49	-$610	$743	-$7,784
Income Taxes		$29,873	$37,416	$49,621	$55,851	$31,766
Net Cash Flow		$76,922	$97,532	$128,206	$142,874	$174,265

Table 71: 25 Year Detailed Cash Flow – Current Prices Case

		2007	2008	2009	2010	2011	2012	2013
Throughput:	dmt/a			650	2,340	2,860	2,990	3,120
Grades:	%Cu			2.01	2.20	2.13	2.16	2.09
	%Co			0.07	0.07	0.07	0.07	0.07
	%Zn			0.60	0.45	0.31	0.28	0.41
Recoveries:	Cu			92.0%	92.0%	92.0%	92.0%	92.0%
	Co			0.0%	60.0%	80.0%	80.0%	80.0%
	Zn			0.0%	50.0%	65.6%	65.6%	65.6%
Production:	t/a Cu			12,024	47,395	56,143	59,429	60,052
	t/a Co			0	984	1,624	1,723	1,810
	t/a ZSM			0	15,209	16,667	15,934	23,860
Prices:	Cu			$3.50	$3.50	$3.50	$3.50	$3.50
	Co			$30.00	$30.00	$30.00	$30.00	$30.00
	ZSM			$1,500	$1,500	$1,500	$1,500	$1,500
Revenue:	Cu			$93,308	$367,775	$435,662	$461,163	$465,995
($000's)	Co			$0	$65,056	$107,420	$113,958	$119,734
	Zn			$0	$22,814	$25,000	$23,901	$35,789
	Total			**$93,308**	**$455,645**	**$568,083**	**$599,022**	**$621,519**
Total Op Cost $000's				**$53,509**	**$93,304**	**$92,828**	**$95,053**	**$99,094**
Initial Capital		$56,839	$312,616	$198,937				
Sustaining Capital				$0	$1,460	$2,558	$4,995	$3,781
Working Capital at YE				$10,019	$12,565	$13,749	$13,938	$14,198
Working Capital Change				$10,019	$2,546	$1,184	$189	$260
Income Taxes				$0	$0	$102,608	$139,895	$145,358
Net Cash Flow		($56,839)	($312,616)	($178,158)	$343,335	$368,904	$358,891	$373,025

		2015	2016	2017	2018	2019	2020	2021
Throughput:	dmt/a	3,120	3,120	3,120	3,120	3,120	3,120	3,120
Grades:	%Cu	1.65	2.10	1.76	1.77	1.40	1.48	1.71
	%Co	0.06	0.07	0.07	0.07	0.07	0.08	0.08
	%Zn	0.34	0.52	0.54	0.54	0.49	0.55	0.69
Recoveries:	Cu	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%
	Zn	65.6%	65.6%	65.6%	65.6%	65.6%	65.6%	65.6%
Production:	t/a Cu	47,031	59,617	50,161	50,503	39,702	42,225	48,622
	t/a Co	1,455	1,615	1,690	1,775	1,656	1,865	1,889
	t/a ZSM	19,788	30,370	31,784	31,510	28,379	32,180	40,334
Prices:	Cu	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50
	Co	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00
	ZSM	$1,500	$1,500	$1,500	$1,500	$1,500	$1,500	$1,500
Revenue:	Cu	$364,955	$462,620	$389,242	$391,893	$308,082	$327,657	$377,296
($000's)	Co	$96,222	$106,818	$111,747	$117,367	$109,530	$123,332	$124,944
	Zn	$29,682	$45,556	$47,676	$47,266	$42,569	$48,270	$60,501
	Total	**$490,859**	**$614,993**	**$548,665**	**$556,526**	**$460,181**	**$499,258**	**$562,741**
Total Op Cost $000's		**$90,855**	**$94,127**	**$94,246**	**$94,665**	**$91,332**	**$93,351**	**$101,669**
Sustaining Capital		$2,991	$2,992	$2,992	$2,942	$6,524	$14,489	$6,901
Working Capital at YE		$13,563	$13,725	$13,729	$13,766	$13,527	$13,657	$14,266
Working Capital Change		-$219	$162	$4	$36	-$238	$129	$610
Income Taxes		$111,273	$145,114	$126,509	$128,605	$101,689	$110,125	$127,419
Net Cash Flow		$285,960	$372,598	$324,915	$330,278	$260,875	$281,164	$326,142

Table continues…

			2023	2024	2025	2026	2027	2028
Throughput:		t/a	3,120	3,120	3,120	3,120	3,120	3,120
Grades:	%Cu		1.55	1.58	1.14	0.89	0.93	0.71
	%Co		0.07	0.08	0.11	0.09	0.08	0.09
	%Zn		0.65	0.45	0.43	0.70	0.64	0.82
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		43,968	45,076	32,449	25,291	26,436	20,295
	t/a Co		1,724	2,050	2,604	2,311	2,014	2,133
	t/a ZSM		37,794	26,239	25,378	41,223	37,409	48,210
Prices:	Cu		$3.50	$3.50	$3.50	$3.50	$3.50	$3.50
	Co		$30.00	$30.00	$30.00	$30.00	$30.00	$30.00
	ZSM		$1,500	$1,500	$1,500	$1,500	$1,500	$1,500
Revenue:	Cu		$341,187	$349,785	$251,802	$196,257	$205,136	$157,483
($000's)	Co		$114,033	$135,575	$172,220	$152,839	$133,209	$141,069
	Zn		$56,692	$39,358	$38,067	$61,835	$56,114	$72,316
	Total		**$511,911**	**$524,718**	**$462,089**	**$410,931**	**$394,460**	**$370,868**
Total Op Cost $000's			$100,209	$93,507	$88,213	$95,356	$90,258	$97,006
Initial Capital								
Sustaining Capital			$3,911	$6,365	$4,161	$5,966	$5,626	$4,347
Working Capital at YE			$14,173	$13,720	$13,293	$13,747	$13,360	$7,573
Working Capital Change			$152	-$453	-$428	$454	-$387	-$5,787
Income Taxes			$114,324	$119,188	$103,672	$86,908	$83,806	$73,357
Net Cash Flow			$293,315	$306,110	$266,471	$222,247	$215,157	$195,693

			2029	2030	2031	2032	2033
Throughput:		t/a	3,120	3,120	3,120	3,120	3,120
Grades:	%Cu		0.64	0.77	1.14	1.09	0.99
	%Co		0.07	0.07	0.07	0.09	0.12
	%Zn		0.75	0.99	0.71	0.96	0.44
Recoveries:	Cu		91.2%	91.2%	91.2%	91.2%	91.2%
	Co		78.2%	78.2%	78.2%	78.2%	78.2%
	Zn		65.6%	65.6%	65.6%	65.6%	65.6%
Production:	t/a Cu		18,330	21,804	32,577	31,080	28,165
	t/a Co		1,801	1,642	1,615	2,260	2,988
	t/a ZSM		43,774	58,111	41,585	56,261	25,922
Prices:	Cu		$3.50	$3.50	$3.50	$3.50	$3.50
	Co		$30.00	$30.00	$30.00	$30.00	$30.00
	ZSM		$1,500	$1,500	$1,500	$1,500	$1,500
Revenue:	Cu		$142,241	$169,198	$252,792	$241,176	$218,554
($000's)	Co		$119,091	$108,626	$106,792	$149,473	$197,618
	Zn		$65,661	$87,166	$62,377	$84,392	$38,883
	Total		**$326,993**	**$364,990**	**$421,961**	**$475,041**	**$455,055**
Total Op Cost $000's			**$96,789**	**$99,002**	**$89,910**	**$100,539**	**$86,528**
Sustaining Capital			$4,212	$2,842	$2,842	$2,842	$0
Working Capital at YE			$7,602	$7,651	$7,041	$7,784	$0
Working Capital Change			$29	$49	-$610	$743	-$7,784
Income Taxes			$63,240	$73,298	$92,178	$104,065	$81,475
Net Cash Flow			$162,724	$189,800	$237,640	$266,852	$302,088

Table 72: 25 Year Detailed Cash Flow – Base Case with Manganese Carbonate Production

		2007	2008	2009	2010	2011	2012	2013	2014
Throughput:	dmt/a			650	2,340	2,860	2,990	3,120	3,120
Grades:	%Cu			2.01	2.20	2.13	2.16	2.09	1.73
	%Co			0.07	0.07	0.07	0.07	0.07	0.07
	%Zn			0.60	0.45	0.31	0.28	0.41	0.34
	%Mn			2.93	2.46	2.03	1.53	2.04	1.89
Recoveries:	Cu			92.0%	92.0%	92.0%	92.0%	92.0%	92.0%
	Co			0.0%	60.0%	80.0%	80.0%	80.0%	80.0%
	Zn			0.0%	50.0%	65.6%	65.6%	65.6%	65.6%
	Mn			0.0%	0.0%	60.0%	81.0%	81.0%	81.0%
Production:	t/a Cu			12,024	47,395	56,143	59,429	60,052	49,633
	t/a Co			0	984	1,624	1,723	1,810	1,688
	t/a ZSM			0	15,209	16,667	15,934	23,860	20,056
	t/a Mn			0	0	34,858	37,149	51,459	47,770
Prices:	Cu			$3.00	$2.65	$2.35	$2.00	$1.50	$1.50
	Co			$15.00	$15.00	$15.00	$15.00	$15.00	$15.00
	ZSM			$1,200	$1,200	$1,200	$1,200	$1,200	$1,200
	MnCO3			$650	$650	$650	$650	$650	$650
Revenue:	Cu			$80,054	$278,966	$293,329	$264,645	$201,225	$166,312
($000's)	Co			$0	$32,528	$53,710	$56,979	$59,867	$55,817
	Zn			$0	$18,251	$20,000	$19,121	$28,632	$24,067
	Mn			$0	$0	$48,208	$51,376	$71,166	$66,064
	Total			$80,054	$329,745	$415,247	$392,121	$360,890	$312,261
Total Op Cost $000's				$53,496	$93,215	$109,631	$112,915	$123,845	$116,563
Initial Capital		$59,000	$324,500	$206,500					
Sustaining Capital				$0	$1,515	$2,655	$5,092	$3,879	$8,906
Working Capital at YE				$10,257	$12,803	$15,090	$15,351	$16,064	$15,532
Working Capital Change				$10,257	$2,546	$2,287	$261	$713	-$532
Income Taxes				$0	$0	$10,816	$76,937	$65,428	$52,626
Net Cash Flow		($59,000)	($324,500)	($199,199)	$217,469	$289,858	$196,915	$167,026	$134,699
		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	1.65	2.10	1.76	1.77	1.40	1.48	1.71	1.86
	%Co	0.06	0.07	0.07	0.07	0.07	0.08	0.08	0.08
	%Zn	0.34	0.52	0.54	0.54	0.49	0.55	0.69	0.57
	%Mn	1.81	2.82	2.94	3.03	1.95	2.03	2.71	2.63
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%
	Mn	81.0%	81.0%	81.0%	81.0%	81.0%	81.0%	81.0%	81.0%
Production:	t/a Cu	47,031	59,617	50,161	50,503	39,702	42,225	48,622	52,913
	t/a Co	1,455	1,615	1,690	1,775	1,656	1,865	1,889	1,847
	t/a ZSM	19,788	30,370	31,784	31,510	28,379	32,180	40,334	33,117
	t/a Mn	45,789	71,318	74,280	76,586	49,172	51,360	68,606	66,571
Prices:	Cu	$1.50	$1.50	$1.50	$1.50	$1.50	$1.50	$1.50	$1.50
	Co	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00
	ZSM	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200
	$MnCO_3$	$650	$650	$650	$650	$650	$650	$650	$650
Revenue:	Cu	$157,594	$199,768	$168,082	$169,227	$133,036	$141,488	$162,923	$177,304
($000's)	Co	$48,111	$53,409	$55,873	$58,684	$54,765	$61,666	$62,472	$61,077
	Zn	$23,746	$36,444	$38,141	$37,813	$34,055	$38,616	$48,401	$39,740
	Mn	$63,325	$98,631	$102,728	$105,917	$68,004	$71,030	$94,880	$92,067
	Total	**$292,776**	**$388,252**	**$364,824**	**$371,639**	**$289,860**	**$312,800**	**$368,676**	**$370,188**
Total Op Cost $000's		**$112,907**	**$128,534**	**$130,135**	**$131,673**	**$115,061**	**$118,132**	**$134,806**	**$130,082**

Table continues…

Sustaining Capital	$3,099	$3,100	$3,100	$3,050	$6,632	$14,597	$7,009	$3,987
Working Capital at YE	$15,250	$16,220	$16,317	$16,427	$15,321	$15,520	$16,675	$16,366
Working Capital Change	-$282	$970	$98	$109	-$1,106	$199	$1,155	-$309
Income Taxes	$49,608	$71,966	$64,958	$66,448	$47,328	$50,951	$63,776	$66,258
Net Cash Flow	$127,444	$183,683	$166,534	$170,360	$121,945	$128,921	$161,930	$170,169
			2023	2024	2025	2026	2027	2028
Throughput:	dmt/a		3,120	3,120	3,120	3,120	3,120	3,120
Grades:	%Cu		1.55	1.58	1.14	0.89	0.93	0.71
	%Co		0.07	0.08	0.11	0.09	0.08	0.09
	%Zn		0.65	0.45	0.43	0.70	0.64	0.82
	%Mn		2.99	2.28	2.45	2.95	3.81	5.00
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%
	Mn		81.0%	81.0%	81.0%	81.0%	81.0%	81.0%
Production:	t/a Cu		43,968	45,076	32,449	25,291	26,436	20,295
	t/a Co		1,724	2,050	2,604	2,311	2,014	2,133
	t/a ZSM		37,794	26,239	25,378	41,223	37,409	48,210
	t/a Mn		75,539	57,734	61,793	74,629	96,222	126,285
Prices:	Cu		$1.50	$1.50	$1.50	$1.50	$1.50	$1.50
	Co		$15.00	$15.00	$15.00	$15.00	$15.00	$15.00
	ZSM		$1,200	$1,200	$1,200	$1,200	$1,200	$1,200
	MnCO3		$650	$650	$650	$650	$650	$650
Revenue:	Cu		$147,331	$151,043	$108,733	$84,747	$88,582	$68,004
($000's)	Co		$57,016	$67,788	$86,110	$76,420	$66,605	$70,534
	Zn		$45,353	$31,486	$30,454	$49,468	$44,891	$57,852
	Mn		$104,468	$79,846	$85,459	$103,210	$133,074	$174,650
	Total		$354,169	$330,163	$310,755	$313,845	$333,151	$371,041
Total Op Cost $000's			$136,737	$121,375	$118,110	$131,524	$136,917	$158,307
Initial Capital								
Sustaining Capital			$4,019	$6,473	$4,269	$6,074	$5,734	$4,455
Working Capital at YE			$16,802	$15,785	$15,486	$16,346	$16,643	$11,569
Working Capital Change			$436	-$1,016	-$299	$860	$297	-$5,074
Income Taxes			$59,902	$56,884	$52,901	$49,570	$53,549	$56,173
Net Cash Flow			$153,075	$146,448	$135,775	$125,817	$136,654	$150,689
			2029	2030	2031	2032	2033	
Throughput:	dmt/a		3,120	3,120	3,120	3,120	3,120	
Grades:	%Cu		0.64	0.77	1.14	1.09	0.99	
	%Co		0.07	0.07	0.07	0.09	0.12	
	%Zn		0.75	0.99	0.71	0.96	0.44	
	%Mn		4.85	5.71	3.60	3.79	3.09	
Recoveries:	Cu		91.2%	91.2%	91.2%	91.2%	91.2%	
	Co		78.2%	78.2%	78.2%	78.2%	78.2%	
	Zn		65.6%	65.6%	65.6%	65.6%	65.6%	
	Mn		81.0%	81.0%	81.0%	81.0%	81.0%	
Production:	t/a Cu		18,330	21,804	32,577	31,080	28,165	
	t/a Co		1,801	1,642	1,615	2,260	2,988	
	t/a ZSM		43,774	58,111	41,585	56,261	25,922	
	t/a Mn		122,570	144,337	91,045	95,820	78,187	
Prices:	Cu		$1.50	$1.50	$1.50	$1.50	$1.50	
	Co		$15.00	$15.00	$15.00	$15.00	$15.00	
	ZSM		$1,200	$1,200	$1,200	$1,200	$1,200	
	MnCO3		$650	$650	$650	$650	$650	
Revenue:	Cu		$61,422	$73,063	$109,160	$104,144	$94,376	
($000's)	Co		$59,545	$54,313	$53,396	$74,737	$98,809	
	Zn		$52,529	$69,733	$49,902	$67,513	$31,106	
	Mn		$169,512	$199,615	$125,913	$132,517	$108,131	
	Total		**$343,008**	**$396,724**	**$338,371**	**$378,911**	**$332,421**	
Total Op Cost $000's			**$156,292**	**$169,072**	**$134,026**	**$146,983**	**$124,412**	

Table continues…

Sustaining Capital			$4,320	$2,950	$2,950	$2,950	$0
Working Capital at YE			$11,480	$12,218	$9,922	$10,816	$0
Working Capital Change			-$89	$738	-$2,296	$894	-$10,816
Income Taxes			$51,033	$62,533	$56,391	$64,114	$35,704
Net Cash Flow			$131,452	$161,431	$147,300	$163,970	$190,611

19 CONCLUSIONS

19.1 GEOLOGY & 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 & PROCESS DESIGN

- After conducting extensive testwork of both a bench scale and a pilot scale nature over the past 3 years, the testwork outputs have been studied, assimilated and incorporated into the Metallurgical Plant process design.
- Boleo ore has been successfully processed using the proposed metallurgical treatment route in two continuous pilot plant programs to leach, separate and recover pay metals in final commercial form.
- Design criteria have been confirmed for the purposes of the metallurgical plant design.
- Data for the purposes of ultimately formulating process guarantees for the Boleo 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 plan has scheduled 25 years of underground and surface reserves.
- 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 LOM 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. This work is underway.

19.5 ECONOMIC ASSESSMENT

- Financial modelling based on the DFS 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 25 year mine life shows that the project could generate a cumulative after tax cash flow of US$2.218 billion with a discounted present value of US$924 million at a 6% discount rate, or US$700 million at an 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 & MINERAL RESOURCE MODELLING

At this stage of the project development the following recommendations are made with respect to Geology and Mineral Resource Modelling:

- Resource definition for Boleo limestone should be upgraded to measured and indicated status. This will require further drilling, sampling, assaying and modelling.
- Resource definition for Manto 4 should be upgraded to measured and indicated status. This will require further drilling, sampling, assaying and modelling.

20.2 METALLURGY & PROCESS DESIGN

At this stage of the project development the following recommendations are made with respect to Metallurgy and Process Design:

- A dynamic load study is recommended in which the interactions between the co-generation plant, the acid plant and the processing and operations are studied.
- Materials of construction should be formally reviewed before tenders are placed in the marketplace.

20.3 MINING

At this stage of the project development the following recommendations are made with respect to Mining:

- Geotechnical investigation needs to be conducted in the first mines in Manto 1, 2 and 4.

20.4 ECONOMIC ASSESSMENT

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

21 REFERENCES

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Sawlan, M.G. and Smith, J.G., 1984. Petrologic Characteristics, Age and Tectonic Setting of Neogene Volcanic Rocks in Northern Baja California Sur, Mexico, In Frizzel, V.A., ed., Geology of the Baja California Peninsula: Society of Economic Paleontologists and Mineralogists, Pacific Section (Los Angeles), San Diego, California, April 18-21, 1984, Symposium Proceedings, p. 237-251.

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

Hellman &Schofield Pt Ltd (2005), Resource Estimate Study, The El Boleo Copper - Cobalt - Zinc Deposit, Baja California, Mexico, April 2005.

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 Boleo Feasibility Study Summary Report dated July, 2007 (the “Technical Report”) 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 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 1.4 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. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
12. 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 July, 2007.

Donald J. E. Hunter

I, Eric William Norton, do hereby certify that:

1. I am an employee of:
   Baja Mining Corporation
   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 Boleo Feasibility Study Summary Report, (the “Technical Report”), dated July 2007, relating to the economic evaluation of the project. I have visited the Property eight times since April 2006.
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 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 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 11 July, 2007

____
Signature of Qualified Person

Eric William Norton – B.A.Sc (Hons)
Name of Qualified Person

I, Scott G. Britton, do hereby certify that:

1. I am an employee of:
   Baja Mining Corporation
   2350 – 1177 West Hastings Street,
   Vancouver, B.C. Canada V6E 2K3
2. I graduated with a B. S. degree in Engineering (Mining) from Virginia Polytechnic and State University, Blacksburg, Virginia, USA in 1977.
3. I am a licensed Professional Engineer (#6644) in the State of Wyoming, USA.
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 Boleo Feasibility Study Summary Report, (the "Technical Report"), dated July 2007, relating to the mining operation section of the report. I have visited the Property eight times since June, 2006.
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 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 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 11 July, 2007

Scott G Britton

Signature of Qualified Person

Scott G. Britton – B.S. Mining Engineering
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 "Feasibility Study Summary Report", dated July 11, 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 "Feasibility Study Summary Report", dated July 11, 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 Feasibility that is not reflected in the Feasibility, the omission to disclose which makes the Feasibility 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.

PROFESSIONAL
PROVINCE OF
G. B. BOSWORTH
BRITISH
COLUMBIA
ENGINEER

Signature of Qualified Person

07-07-11

Grant Bryan Bosworth, P.Eng.

Print name of Qualified Person

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 of 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 Boleo Feasibility Study Summary Report, (the “Technical Report”) and dated July 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.
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 the geological database, QA/QC and 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 : 11 July, 2007

Signature of Qualified Person

William J. A. Yeo, MAusIMM PhD
Name of Qualified Person

I, Michael Richard Holmes, MSAIMM, do hereby certify that:

1. I am an employee of:
   Bateman Engineering Pty Ltd
   40 McDougal St,
   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 Boleo Feasibility Study Summary Report, (the "Technical Report") and dated July 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 37 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: 11 July 2007

MR Holmes

Signature of Qualified Person

Michael Richard Holmes BSc (Eng), BComm, MBA, MSAIMM, Pr Eng
Name of Qualified Person

I, Tim 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 a mining 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 Boleo Feasibility Study Summary Report, (the "Technical Report"), dated July 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 July, 2007

Timothy A. Ross

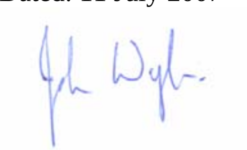
Signature of Qualified Person

Tim Ross
Name of Qualified Person

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 Practising 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 Boleo Feasibility Study Summary Report, (the "Technical Report") and dated July 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: 11 July 2007



Signature of Qualified Person

John Wyche MAusIMM, MMICA, CPMin
Name of Qualified Person