

Philex Gold Incorporated

Pre-Feasibility Study Boyongan Project Island of Mindanao, Philippines

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October 13th, 2008

**Preliminary Feasibility Study
Boyongan Project
Island of Mindanao, Philippines**

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1 Executive Summary

1.1 Introduction

This Pre-Feasibility Study report ("PFS") was compiled by Independent Resource Estimations cc. ("IRES") for Philex Gold Incorporated ("PGI") on the Boyongan Porphyry Cu-Au Project ("Boyongan" or referred to as the "Project").

Since February 22, 2008, PGI has publicly disclosed that it has different points of view from Anglo American ("Anglo") on a number of assumptions; most notably regarding contingencies, owners' costs, and long term metal prices.

PGI approached IRES in May, 2008, to complete a National Instrument 43-101 ("NI43-101") compliant PFS using PGI's project development strategy and incorporating their own corporate philosophy and their own experience of costs within the Philippine context. Main points include:

- Use the mineral resource estimate which included all available data up to year end December 2007 as the mineral resource blockmodel;
- Design an open pit to optimize grade and stripping ratio;
- Incorporate PGI's corporate strategy regarding long term copper and gold prices.

This report summarizes the results of the development strategy and highlights the recommendations of work necessary to advance the project to the next stage.

1.2 Location

The Boyongan copper-gold porphyry deposit is located in the Surigao gold district at the north-eastern tip of Mindanao Island in the Philippines, approximately 750km southeast of Manila (Figure 2). Anglo American manages and operates an exploration program in the district under a Joint Venture Agreement between Anglo American Exploration Philippines Inc. ("AAEPI") and Philex Gold Inc. which is focused on the search for copper-gold mineralization in the region. The Joint Venture Company, Silangan Mindanao Mining Company Inc. ("SMMCI"), controls four mining tenements in the area covering 21,569 hectares in the provinces of Surigao del Norte and Agusan del Norte. The Boyongan deposit lies within Mineral Production Sharing Agreement ("MPSA") license No. 149-99-XIII which totals 2,880 hectares.

1.3 Geology and Mineralization

The Surigao Gold District is situated at the northern tip of the Pacific Cordillera of Eastern Mindanao. It is located within a well-defined, transtensional segment of a Pliocene-Quaternary volcano-plutonic arc related to subduction of the oceanic Philippine Sea Plate at the Philippine Trench. The NNW-trending, transcurrent Philippine Fault is related to the oblique convergence of the Philippine Sea Plate with the Eurasian continental margin, and parallels the Pacific Cordillera. District-wide structures related to mineralization consist of NE-SW and NW-SE fault sets. Boyongan is located near the intersection of these two structural trends; see Figure 4.

The Boyongan Deposit is centered on the two early-mineralization composite porphyry stocks, which appear to have been emplaced in conjunction with the mineralizing fluids. Intense quartz stockwork veining has developed within these early-mineralization stocks and the adjacent breccia wallrocks. These two cylindrical stocks, spaced approximately 150m apart, were emplaced into a diatreme breccia complex and form the West and East Zones of mineralization. The two mineralized zones are separated by a N-S trending upward-flared polyolithic breccia which may have acted as an impermeable wallrock, preventing much of the mineralizing fluids from flowing further outward and upward from the fluid upflow conduits, thus localizing the main high-grade copper-gold zones. Drilling since

2003 has further delineated an area of exotic copper mineralization to the NNW of the West Zone which appears to have been derived by leaching and dispersion of copper from the Bayugo deposit.

The alteration and ore-mineral assemblages observed at Boyongan are similar in type and style to those documented within other island arc porphyry copper deposits. The potassic alteration zone at Boyongan is manifested by secondary biotite, orthoclase, magnetite, quartz, and minor actinolite and formed largely in the centre of the system. This alteration type generally contains the most abundant mineralization. The potassic alteration grades outward to a propylitic aureole as indicated by the presence of chlorite and albite associated with epidote, illite and calcite. The potassic alteration is overprinted by an intermediate argillic alteration consisting of chlorite, illite-montmorillonite, sericite, and specular hematite and also partially overprinted by a phyllic alteration assemblage of quartz, pyrite and sericite. A remnant advanced argillic lithocap is inferred by clasts of clay, quartz, pyrophyllite, alunite, and tourmaline within Quaternary sediments in drill hole TSD-6. Magnetite occurs as clusters, fine disseminations and as replacements within volcanoclastic rocks adjacent to the deposit.

1.4 Mineral Processing and Metallurgy

Three different campaigns of metallurgical test work were completed at Anglo American's research laboratories 'Anglo Research' (AR) in Johannesburg, South Africa. The first two campaigns were carried out in 2002 and 2003 and were discussed in the report on the mineral resource estimates published in November 2003. The last campaign was carried out in 2006 and 2007 and it is largely the results of this latest test work that are used to develop a flow sheet and estimate metal recoveries and plant costs for this PFS.

For the 2006-7 metallurgical test work one "trial" bulk sample was composited from various parts of the oxide portion of each of the East and West Zones as well as one sulphide trial sample were provided to AR for developing a flow sheet that would be appropriate for the whole deposit. Once established, possible variations in metallurgical performance across the deposit were tested by running smaller "variability" samples through the proposed flow sheet.

68% of the samples submitted for metallurgical test work were from the oxide zone of the deposit. This proportion is considered to reflect the relative volumes of material in the deposit likely to be mined from an open-pit. The copper and gold contained in the ore have approximately the same value in terms of equivalent copper, suggesting that equal emphasis should be placed on recovery of both metals to realize full contained value.

The following factors influenced the flow sheet selection:

- Presence of coarse grained "free" gold which would require gravity concentration
- Presence of secondary sulphide minerals in the oxide zone (particularly in the east) necessitating sulphide flotation
- High proportion of acid soluble copper amenable to acid leaching
- Possibility of recovery of oxide copper by sulphidizing oxide flotation
- High proportion of gold not reporting to either the gravity or sulphide flotation concentrates, but recoverable by cyanidation
- To achieve this, the proposed flow sheet includes gravity separation, sulphide flotation, acid leaching and cyanidation.

1.5 Mineral Resource and Reserve Estimates

Resource estimation has been undertaken by ordinary kriging interpolated into a block model using copper, gold, arsenic and acid soluble copper zones determined by both grade distributions and geology. The mineral resources at Boyongan have been classified according to the guidelines as specified in National Instrument 43-101 and the CIM

Standards on Mineral Resources and Reserves. The Competent Person for the resource estimation is Mr. Dexter Ferreira.

Results were tabulated at a cut-off of 0.50% Cu and are presented below for Measured and Indicated Resources; separately for the oxide and sulphide zones down to the -400m elevation mark, which corresponds to the broad bottom of the low-grade Cu shell (see Table 1).

Table 1.0 Measured and Indicated Resources

Type	Category	Tonnes	Cu Grade (%)	Au Grade (g/t)	As Grade (ppm)	ASCu Grade (%)
Oxide	Measured	74,000,000	0.83	1.08	14.04	0.57
Oxide	Indicated	250,000	0.75	0.08	3.37	0.38
Sulphide	Measured	30,500,000	0.73	0.77	115.66	0.17
Sulphide	Indicated	620,000	0.82	0.08	87.31	0.17
Measured	Total	104,500,000	0.80	0.99	43.66	0.45
Indicated	Total	870,000	0.80	0.08	62.90	0.23
Total		105,400,000	0.80	0.98	43.82	0.45

Of the total resources estimated for Boyongan, it has been determined that about 67% are convertible into reserves. The Base Case adopted for the PFS includes an open pit operation which would provided a mill throughput of 5 million tonnes per annum ("Mtpa") and a metallurgical process comprising gravity concentration, flotation, acid leaching of flotation tails for oxide copper recovery and cyanide leaching of acid leach tailings for additional gold recovery; the assumptions are listed in Table 2 below.

Table 2.0 Base Case Assumptions

Parameter	Amount	Unit
Long term Cu price	2.75	\$/lb
Long term Au price	700.00	\$/oz
Selling cost – Cu	0.1279	\$/lb
Selling cost – Au	3.125	\$/oz
TC/dmt conc.	60	\$/t
RC costs– Au	5.00	\$/oz
RC costs – Cu	0.06	\$/lb
Mining method	Open pit	-
Overall pit slope	42	Degrees
Mining/processing rate	5,000,000	tpa
Capital costs	737,000,000 ¹	\$
Discount Rate	12	%
Mining cost (ore or waste)	1.20	\$/t mined
Mining Recovery	95	%
Cut-off grade – Cu	0.60	%
Cut-off grade – Au	0.80	g/t
Mining cost adjustment factor	1.141	-
Processing cost	24.88	\$/t milled

The project yields a positive net present value ("NPV") despite the initial high stripping ratio. The resultant reserves are presented in Table 3 below.

¹ Includes pre-stripping; pre-stripping costs have been capitalized.

Table 3.0 Proven and Probable Reserves

Type	Category	Tonnes	Cu Grade (%)	Au Grade (g/t)
Oxide	Proven	54,500,000	0.88%	1.43
Sulphide		11,300,000	0.82%	1.21
Oxide	Probable	490,000	0.97%	0.82
Sulphide		1,480,000	0.81%	0.41
Proven	Total	65,800,000	0.87%	1.39
Probable	Total	1,970,000	0.85%	0.52
Total		67,800,000	0.87%	1.37

1.6 Mine Plan

The deposit is proposed to be developed as an open pit. Mining of the deposit will produce a total of 67.8Mt of mill feed and 518Mt of waste over 15 year mine operating life. Additional resources remain within the deposit, especially in the North West portion of the deposit (formally referred to as the *Exotic Zone*). It is believed, although not part of this study that these resources will be accessed at some point during the life of the pit, through an adit through the pit wall and accessed via the pit ramp.

The current life of mine ("LOM") plan focuses on achieving the required mill feed production rate, while balancing grade and strip ratios. In this PFS, no stockpile and blending capacity was assumed.

Mine design for the pit was initiated with estimates of metal prices, mining recovery, mill recoveries, and refining payables and costs, freight. The model was then used with the Datamine NPVS Scheduler™ software to determine the optimal mining shell. Mine planning and scheduling was then conducted on the optimal pit shell with the use of the same software and mineral reserves within the pit shell were estimated using cut-off grades of 0.60% Cu and 0.80g/t Au.

The mining sequence was divided into 14 phases designed to maximize grade, reduce pre-stripping requirements while maintaining a constant mill feed of 5Mta. The LOM schedule is shown in Table 4 below which includes an initial 2 years of pre-stripping prior to production.

Table 4.0 LOM Mine Production

Parameter	Units	Year							
		-1	0	1	2	3	4	5	6
Ore	Mt	-	-	5.00	5.00	5.00	4.99	4.99	5.00
Waste	Mt	33.13	33.13	33.13	4.79	1.88	0.93	23.89	30.18
Strip Ratio	-	-	9.62	0.96	0.37	0.19	4.78	6.04	6.04
Cu Grade	%	-	-	1.27	0.90	0.90	1.01	0.98	0.85
Au Grade	g/t	-	-	2.53	1.17	1.02	1.61	1.70	1.37
Parameter	Unit	Year							
		7	8	9	10	11	12	13	14
Ore	Mt	4.99	5.00	4.99	5.00	4.99	4.99	5.00	2.96
Waste	Mt	27.97	6.19	105.61	4.65	78.84	121.49	11.05	1.89
Strip Ratio	-	5.59	1.24	21.13	0.93	15.78	24.30	2.21	0.68
Cu Grade	%	0.83	0.80	0.74	0.75	0.72	0.80	0.71	0.84
Au Grade	g/t	1.38	1.10	0.83	1.27	1.31	1.09	1.39	1.34

1.7 Waste Management

Of the two main sources of waste to be generated by future mining activities, the most critical will be tailings material from the process plant. Three alternative tailings disposal

options have been considered in terms of technical merit ease of permitting, and likely public acceptance. These are: land-based disposal in a constructed tailings dam, subaqueous disposal in the deep waters of the adjacent freshwater lake (Lake Mainit), and subaqueous marine disposal in deep offshore trenches. For the purposes of the PFS, it has been assumed that the tailings will be contained within a land-based facility, and this option has been costed into the project estimate. Terrestrial disposal of tailings will present several challenges that will need to be carefully managed to prevent unwanted impacts on surrounding communities and the environment. These are: sterilization of land from agricultural use, dust generation, and seepage of contaminated water from the facility in to the local surface streams and groundwater aquifer. By volume and tonnage the main waste will be waste rock from the open pit stripping. In the base-case scenario there will be 518.8Mt of waste rock generated by mining activities. Impact, risks, and mitigation of the impact and risks will be examined in detail in the next phase of the study. However due to the fact that Boyongan is a low pyrite porphyry, acid generation potential is expected to be low.

1.8 Environmental Considerations

The key issues and risks around safety, environmental and community, with current knowledge, are summarized below. The key environmental risks facing any future mining development in the Boyongan area are the possible impacts on biodiversity and water quality and supply. Although the forests on and around the Boyongan prospect have been highly modified by the removal of most of the indigenous hardwoods, and replanting with coconut trees, there are seven endangered plant species recorded in the area. Any mining development at Boyongan would need to implement a conservation plan to protect these species, and reduce the risk for loss of important biodiversity. Possible future mining will impact on the water resource in two ways. Firstly by reducing stream flow in the main drainage lines leaving the area, and also by affecting water quality in these streams. Unless mitigated, open pit or underground mining will reduce the catchments' yield by intercepting some of the direct surface runoff and also by lowering the water table in the groundwater aquifers that ultimately feed surface streams through springs. Because of the strong reliance of several villages in the Tubod municipality on dry season stream flow for their livelihoods (stock watering, irrigation of rice paddies, etc), any reduction in base flow in streams would need to be substituted or compensated, probably into the post closure phase. The risk for reduced stream flow and loss of quality is high. During the next phase of the project, detailed studies will be completed in order to identify the most appropriate mitigation study.

1.9 Economic Analysis

Operating and capital expenditures were received from PGI (see Table 5). Capital costs reflect costs more in tune with the Philippine context.

Table 5.0 Operating Cost Estimates

Parameter	Unit Cost	Unit
Mining Cost (ore & waste)	\$1.20	\$/tonne
Services	3.56	\$/tonne
Labor	3.36	
Regents	13.46	
Tailings	0.70	
Laboratory	0.07	
Maintenance	0.84	
Spares	1.40	
Contractors	0.25	
Contingency	2.33	
Total Milling costs	\$24.88	\$/tonne

Capital costs used for this project were \$657M and include a 10% contingency. Detailed capital can be found in Section 19.10 that will be used over a 14 year period. Approximately 50% of the capital will be used for equipment whereas the remainder will be used for costs associated with the mill.

The economic assessment in this report is detailed enough and does not use inferred mineral resources, and as such, this has enabled to have them to be categorized as mineral reserves. Three cash flow cases were studied. All cash flow cases used the same mineral resource estimate, LOM plan and production factors as shown in Table 1.10.

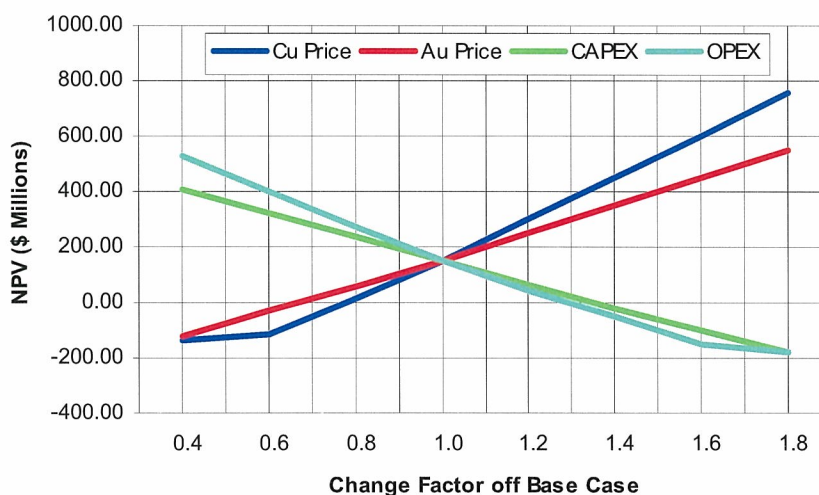
The cash flow cases were differentiated by metal prices. The base used metal prices of \$2.75/lb for Cu and \$700/oz for Au. The worst case used \$2.00/lb for Cu and \$600/oz for Au and the best case used \$4.00/lb for Cu and \$2000/oz for Au. All cases used a discount rate of 12% and identical OPEX and CAPEX costs. In addition, all scenarios used a pre-production scenario of 2 years prior to production; see Table 6.

Table 6.0 Summary of Cash Flow Results by Case

Parameter	Unit	Case		
		Base	Worst	Best
Average Cu Price	\$/lb	2.75	2.00	4.00
Average Au Price	\$/oz	700.00	600.00	2000.00
Mining Costs	\$/t	1.20	1.20	1.20
Milling Costs	\$/t	24.88	24.88	24.88
NPV at 12% discount	\$ (millions)	150.00	(93.00)	1485
Payback	Years	3.5	-	1.9

Sensitivities of net present values ("NPV") to metal price, grade and recovery, operating expenses ("OPEX") and capital expenses ("CAPEX") were investigated. As with many projects, the NPV results of the cash flow models were most sensitive, in order of magnitude, to metal price, grade and recovery, OPEX and capital expenses CAPEX. The results for Case 1 are shown in Figure 1.

Figure 1.0 Sensitivity of Project Economics



1.10 Conclusions

Industry standard mining, process design, construction methods and economic evaluation practices have been used to assess the Boyongan project. The conclusion reached is that there is adequate geological, metallurgical and cost data available to generate a pre-feasibility study.

The likelihood of improving the project economics by accessing more mineralization within the deposit is extremely high, plus, if strong metal prices continue, there is a possibility of expanding the current pit design to include more resources and, therefore, extend the mine life.

Sufficient work has been done on permitting and environmental aspects of the project to indicate that required permits to proceed with the project, to operate the mine and to deal with mine closure at the end of the mine life, have a reasonable likelihood of being granted.

Based on current knowledge and assumptions, the results of this study show that the project is economic and should be advanced to the next level of study of a bankable feasibility study.

The project is very sensitive to metal prices and requires a copper price in excess of US\$2.12/lb and a gold price of \$470/oz to be economic, assuming none of the opportunities for economic improvement are realized. The project's potential short-term start-up permits the hedging of metal prices to mitigate metal price risks.

As with almost all mining ventures, there are a large number of risks and opportunities that can affect the outcome of the Boyongan project. Most of these risks and opportunities are based on the lack of control over external drivers (metal price, exchange rates, etc.). The following section identifies the most significant potential risks and opportunities currently identified for the Boyongan project, almost all of which are common to mining projects at this stage of study. Subsequent higher-level engineering studies will need to further refine these risks and opportunities, identify new ones and define mitigation or opportunity implementation plans.

1.10.1 Summary of Major Risks

Project risk assessment has been undertaken for technical, financial and the Philippines country risk. The risk assessment found the following categories of risk that would impact the Northern part of Mindanao Island in the Philippines:

Political Risk

The study has determined that it is unlikely that the Boyongan operation would be threatened by local/national politicians given that PGPI is a Philippine-based company run by Filipinos. In addition, PGIs' parent company, Philex, ISO 14001 certified, has a proven track record as a socially-responsible, capable and profitable mining company within the Philippines.

Economic Risk

The Philippines has recently experienced strengthened economic growth of 5.3% in 2006 and 6.5% in 2007. If this trend is to continue, the study determined that there will be very little vulnerability to political upheaval and drastic economic reforms.

Operational Risk

Although mining has proven to be one of the most politically-charged industries in the Philippines, PGIs' parent company, Philex, has a proven track record of operating within the Philippines since 1958. The study concluded that operational risks pose very little threat to the operation.

Security Risk

Security risk poses very little or almost no threat to the operation. PGIs' parent company, Philex, has had a very strong positive impact on the local community at their existing operation, Padcal, and it is expected that they will establish a similar relationship with the local communities around Boyongan.

1.10.2 Opportunities

Based upon the assumptions used in this report, the results of the PFS indicate that the Boyongan project is technically and financially feasible. It also has revealed that in addition to the reserves extractable from the open pit mining activity, additional opportunity exists for the extraction of additional resources, albeit not yet reportable, located in the northwest area of the deposit via an adit accessed through the pit ramp. This would minimize initial pre-production costs given that the development would be through mineralized rock.

1.11 Recommendations

IRES recommend that PGI (or its parent company) proceed with information gathering for a detailed and bankable feasibility study for the Boyongan deposit. The following are the major recommended tasks:

- Additional metallurgical and mineral processing tests including, but not limited to, tailings performance, crushing and milling characteristics, process recovery optimization, reagent consumption, reagent type, mill feed reactivity and characterization of mineralization,
- Pursuit of environmental and permitting,
- Conduct a bankable feasibility study.

It is IRES's opinion that the data used in support of and for the completion of this pre-feasibility study are compliant with CIM definitions and that the mineral resources and reserves presented meet the requirements of National Instrument 43-101.

2 Introduction

This pre-feasibility study ("PFS") was prepared for Philex Gold Incorporated ("PGI") to assess the economic viability of mining a mineral deposit known as Boyongan. This PFS includes a 5M tonne per year ("tpy") mill and a combination of a long term price of \$2.75 per pound ("lb") for copper ("Cu") and \$700 per ounce ("oz") for gold ("Au") as its base case.

2.1 Terms of Reference

This report has been prepared in accordance with the guidelines provided in National Instrument 43-101 ("NI43-101"). The Qualified Person responsible for this report is Mr. Dexter S. Ferreira, Senior Geostatistician and Mining Engineer at IRES.

2.2 Scope of Work

The scope of work undertaken by IRES involved a pre-feasibility study for the Boyongan deposit. This PFS used the 2008 Mineral Resource estimate, completed by IRES, as the blockmodel input. Whittle runs were run along with the scheduling and updated financial outputs.

2.3 Effective Date

The effective date of the Boyongan Pre-Feasibility report is October 13th, 2008.

2.4 Units

Unless explicitly stated, all units presented in this report are based on the Système Internationale (i.e. metric tonnes, kilometers, meters, and centimeters) with the exception of industry standards such as pounds (lbs) for base metals and ounces (ozs) for gold. All currency values are United States dollars ("US\$") unless otherwise noted.

Throughout this report, many industry-standard abbreviations and acronyms can be found, most of which are defined within the text body; a further reference can be found in Section 24.

2.5 Qualifications of Consultant

Dexter S. Ferreira visited the Boyongan Project in May 2007, November 2007 and June 2008. During his visits Mr. Ferreira examined the Project's core logging and storage procedures, observed exploration drilling and locations of previous drillholes. This report has been prepared based on a technical review and preparation of Mineral Resource estimates by the author. Mr. Ferreira is a senior geostatistician with over 18 years experience in project evaluation internationally, including extensive involvement with mineral projects throughout Canada, South America and Africa. He is a member of the South African Council for Natural Scientific Professions, and qualifies as an 'Expert', 'Competent Person' and 'Qualified Person' as defined in National Instrument 43-101 and the JORC Code.

The author of this report has not, or has not previously had, any material interest in Philex or any of the mineral properties in which Philex has an interest. The relationship with Philex is solely one of professional association between client and independent consultant. This report is prepared in return for professional fees based upon agreed commercial rates and the payment of these fees is in no way contingent on the results of this report.

2.6 Basis of Report

IRES have prepared this report exclusively for Philex. The information presented, opinions and conclusions stated, and estimates made are mainly based on the following information:

- The report "Qualifying Technical Report on the North Project Area, Surigao del Norte, Mindanao Island, Republic of the Philippines", by Mr. Michael C. Corey, June 30, 2004;
- The report "Boyongan Copper/Gold Deposit, Mineral Resource Evaluation and Exploration Potential", by Anglo American plc., Mr. Duncan Campbell et al, January 2008;
- Information available at the time of the preparation of the report as provided by Philex;
- Assumptions, conditions and qualifications as set forth in the report;

IRES has not independently conducted any title or other searches but has relied upon Philex for information on the status of the claims, property title, agreements and other pertinent conditions.

3 Reliance on Other Experts

This report relies upon the work and opinion of several non-IRES experts without verification by IRES. The information provided by independent, third party experts at Philex:

Dr. Rogelio G. Laraya: Senior Adviser	Responsible for metal price assumptions, hedging strategy, concentrate marketing terms
Ms. Redempta P. Baluda: AVP Exploration	Responsible for exploration and geology
Mr. Noel C. Oliveros: Exploration Manager	Responsible for geology and database
Mr. Libby R. Ricafort: Mill Division Manager	Responsible for milling costs

IRES do not take responsibility for the accuracy or completeness of the contributions of any non-IRES expert.

4 Location and Property Description

4.1 Location

The Boyongan Project is located in the Surigao gold district at the north-eastern tip of Mindanao Island in the Philippines. It is located 20km south of Surigao City, the capital of Surigao del Norte, and approximately 750km southeast of Manila; see Figure 2. Anglo manages and operates an exploration program in the district under a joint venture ("JV") agreement between Anglo and Philex, focusing on copper-gold exploration within the region.

Figure 2.0 Location Map



4.2 Tenements

In September 1999, a JV agreement between Philex Gold Inc. ("PGI"), Philex and Anglo was signed covering the Surigao Project area. The joint venture company, Silangan Mindanao Mining Company Inc ("SMMCI") was incorporated in 2000 and has control over the 25,184 hectares of mining tenements comprising the Surigao Project area.

The listing of the tenements is shown in Table 7. The JV controls four mining tenements covering an area of 21,569ha in the provinces of Surigao del Norte and Agusan del Norte. The Boyongan deposit lies within the Mineral Production Sharing Agreement ("MPSA") license no. 149-99-XXII which totals 2,880ha; see Figure 3. The MPSA was approved and issued by the Philippines Government on December 29th, 1999 with a 25 year term which is renewable for another 25 year period.

4.3 Lease Structure

SMMCI currently controls a total of four mining tenements covering 21,569 hectares in the provinces of Surigao del Norte and Agusan del Norte in northeast Mindanao. The MPSA encompasses an area of 2,880 hectares. The tenement was assigned to SMMCI by PGI in October, 2000. The total SMMCI property holdings currently consists of five non-

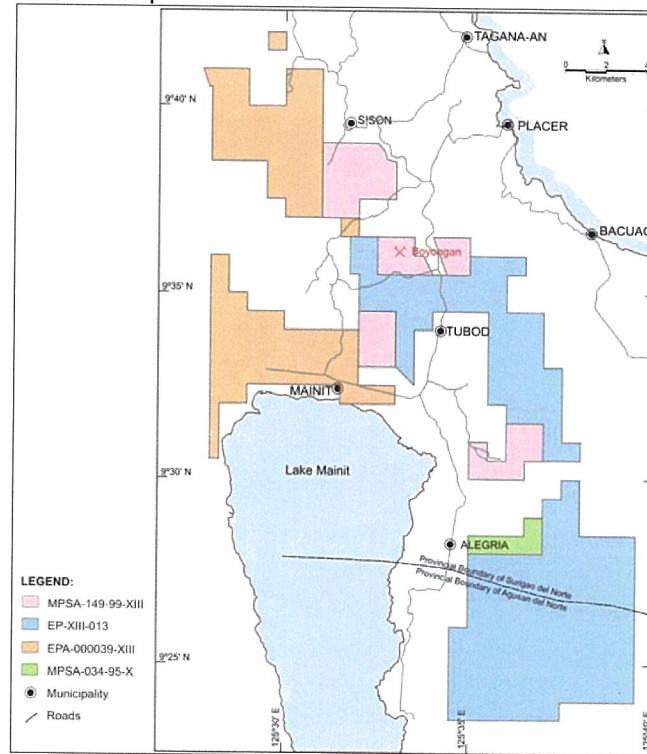
contiguous tenements in the form of two MPSAs, two Exploration Permit Applications (EXPAs) and one approved Exploration Permit (EXP).

MPSA No. 149-99-XIII was granted after certifications were obtained from the National Commission for Indigenous People that there are no indigenous tribal communities or ancestral domain claims within the MPSA. Furthermore the Department of Environment has certified that the MPSA area is clear of any critical watersheds, water reservations or forest reservations. The primary purpose of the MPSA is to enable the commercial utilization of gold, copper and other mineral deposits within the MPSA area. The MPSA allows SMMCI to undertake exploration and evaluation of the property for a maximum of six years and three years for development and construction. Extensions to the duration of the exploration and development periods may be approved based on the recommendation of the Regional Director of the Mines and Geosciences Bureau. The exploration phase of MPSA-149-99-XIII was extended beyond the normal six year term to include a discretionary two-year period in order for SMMCI to complete a Preliminary Assessment on the Boyongan Deposit. The renewal of the exploration phase of MPSA-149-99-XIII was not officially registered by the DENR and MGB until 7 November 2006 and therefore SMMCI has until 7 Nov 2008 in which to complete the Preliminary Assessment and decide whether or not to enter into the official Feasibility Stage of MPSA-149-99-XIII.

Table 7.0 Project Tenement Details

TENEMENT	LOCATION	AREA (Ha)	DATE FILED	DATE APPROVED	REGISTRATION DATE	TRANSFER DATE
MPSA-149-99-XIII (Boyongan)	Surigao del Norte	2,880	March 1995	Dec. 1999 Renewed on Oct. 8, 2004 Renewed on Nov. 7, 2006	March 2000	October 2000
EP-XIII-013	Surigao del Norte & Agusan del Norte	11,976	April 1997	March 11, 2002 June 1, 2005	March 11, 2002	September 2000
EPA-000039-XIII	Surigao del Norte	6,308	March 2000	Pending	Pending	Staked Directly
MPSA-034-95-X	Surigao o del Norte	405	1995	February 1, 1996	February 1, 1996	April 1, 2004

Figure 3.0 Project Tenement Map



4.4 Ownership

Anglo is sole funding and directing the exploration programs and has already earned a 40% interest in the Surigao District tenements by spending a total of US\$2.2m on exploration. In December 2001, PGI agreed to sell Anglo an effective 10% equity interest in SMMCI for US\$20 million. Additional payments up to US\$5 million may be made, depending on an increase in metal content of the Boyongan deposit or other discoveries, as determined by feasibility studies. The transaction closed in March 2002 and resulted in PGI and Anglo each owning 50% equity in the Boyongan project.

Should Anglo American complete a Bankable Feasibility Study on the Boyongan Property, it will obtain an additional 30% interest in the respective Properties either directly or if necessary through a joint venture with a qualified entity. Anglo will conduct the operations should the project proceed to development. Philex Gold Inc will be required to raise its pro-rata share of the project funding while Anglo American would provide 100% of the completion guarantees that may be required by third party lenders. Under these agreements, SMMCI will pay a royalty to the original MPSA-149-99-XIII claim owner of 0.25% of the gross sales of minerals and metals extracted from the mining claims less any smelter and refining charges.

On September 26, 2008 PGI's parent Philex Mining Corporation, announced it has agreed to buy Anglo's 50% share in the Joint Venture subject to completion of transaction documents.

4.5 Philippine Mining Law

Mineral resources in the Philippines are owned by the State and the exploration, development and utilization of mineral resources may only be undertaken through grants of authority or rights to undertake such activities from the government. The Philippine Mining Act of 1995 is the present governing law on mining rights in the Philippines. It is considered

to be a progressive law as it allows 100% foreign ownership of a mining project under a specific form of mineral agreement called the Financial or Technical Assistance Agreement ("FTAA"). The legality of the FTAA, however, was challenged in court by the La-Bugal Tribal Association and the Legal Rights Centre NGO groups in 1997.

A decision of the Philippine Supreme Court, *La Bugal - B'laan Tribal Association v. Ramos*, issued on 1 December 2004, upheld the constitutionality of the FTAA. In this landmark case, certain non-government organizations claimed, among others, that the FTAA allows foreign contractors to manage mining operations, in violation of the Constitutional Principle that the State owns and controls all natural resources. Rejecting this contention, the Supreme Court declared that "full control is not anathematic to day-to-day management by the contractor, provided that the State retains the power to direct overall strategy. [The State] need not micro-manage mining operations and day-to-day affairs of the enterprise." The Supreme Court also upheld the provision in the Mining Act allowing companies that are up to 100% foreign-owned to enter into an FTAA with the government. The Executive Department continues to be a staunch advocate of responsible mining and has issued an executive order enunciating the government policy to promote mining.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

The Project can be easily accessed by the Philippine National Highway. This is a single lane (6m total width) concrete and bitumen surfaced road. This good quality road has numerous bridge crossings between the town of Surigao and the Project site of 25t capacity.

The main port is in the centre of the town of Surigao, approximately 36km from the Project. The port has a single quayside and handles all products from food to industrial materials. Access to this port is through Surigao which is difficult due to the low electrical cable infrastructure and narrow roads. There is little container handling infrastructure within the port, no on quay crane facilities and no storage facilities; albeit several abandoned buildings exist in close proximity. Construction is currently under way to extend the port quay side to handle more ships.

A secondary port can be found just outside Surigao. This smaller port is mainly a passenger facility for ferry connections. It too has no infrastructure for the handling of containers and bulk materials.

There is a commercial airstrip located in close proximity to the town of Surigao. This airstrip is 32km from the Project site and carries daily schedule flights from Manila.

5.2 Climate

The prevailing climate at the Boyongan project site is classified as Type II, with no definite dry season, but with a very pronounced wet season from November to January. The average annual rainfall in the area is 3.6m, based on a 50-year rainfall history. The area is one of the "wettest" in the country with an average of 216 rain days per year. Humidity levels average 85%, with a mean annual temperature of 26.9°C. Temperatures range from a low of 21.1°C in October to a high of 35.2°C in June.

5.3 Local Resources

During exploration at Boyongan, the JV has done much to assist local communities, both in direct employment and in community development projects. Three villages adjacent to the Boyongan prospect have been the focus for the company's community engagement activities. Amongst the key initiatives, the joint venture company has provided: a community centre with a library and computers for use by the community; educational scholarships, materials for a wide range of community infrastructure projects; vegetable seeds for backyard and community vegetable gardens; start up capital for several sustainable businesses.

The JV also assists in the provision of medical and dental healthcare in the communities within which it works. A preliminary assessment of the impact of the current exploration activities on adjacent communities is reasonable but very positive. A detailed community survey is being conducted as part of the stakeholder engagement process to establish the level community acceptance of, and perceptions about, the project.

Possible impacts of future mining at Boyongan on local communities will include some loss of agricultural land, increased noise from mining activities, increased traffic on roads, a reduction of base flow in streams and a possible deterioration of some surface water streams associated with any land-based tailings disposal facility. Plans to address and mitigate these impacts will be included in the next phase of the study.

Economically, the mine is expected to have a positive impact on the community through job creation and the supply of goods and services to the mine from the local community. Other financial benefits will come directly from the social and economic development programs.

In addition, the government will receive 2% of the Net Smelter Return ("NSR") in the form of excise tax of which 60% will go directly to the National Government, and 40% will be distributed to: the Municipality of Tubod (45%), the Barangay (35%) and the Province of Surigao del Norte (20%).

In the absence of a detailed mine design, it is not known what relocation, if any, would be needed for households in the vicinity of the Boyongan area. This will need to be re-evaluated when the project design details emerge. It is clear that any mining development in the region will have positive benefits for the surrounding communities and municipalities in providing direct employment and by stimulating the economy through the multiplier effect.

5.4 Infrastructure

The closest point of electrical power supply is a National Power Company ("NPC") 132kV overhead power line. The line runs north-south and is approximately 1km from the Project site. Indications are that there is surplus capacity in the line, that the availability of supply is 100% when supplied through the NPC and that the supply is hydro based attracting a tax incentive.

5.5 Environmental Conditions and Permits

The Philippine government grants MPSAs on the condition that the subject mining activities are managed in a technically, financially, socially, culturally and environmentally responsible manner to enhance the national growth and welfare of the Philippines.

The Department of Environment and Natural Resources ("DENR") requires an Environmental Clearance Certificate ("ECC") for any mining activity at a more advanced stage than mineral exploration. The ECC is issued by the DENR based on an Environmental Impact Statement prepared by the company in accordance with the procedures under the Environmental Impact Assessment System. A completed ecological profile of the proposed mining area is required as part of these procedures.

The ECC is the basis for the Environmental Protection and Enhancement Program ("EPEP"), which every party to an MPSA must undertake within the subject area of the MPSA. The EPEP must be submitted by the company within 30 days from receipt of the ECC. To effectively implement an approved EPEP, an Annual Environmental Protection and Enhancement Program ("AEPEP") is required to be submitted 30 days prior to the beginning of every calendar year. Mine site inspections and quarterly monitoring are conducted by the DENR and a multipartite monitoring team composed of local government units, host communities, any affected Indigenous Cultural communities, non-governmental organizations, the DENR, and the company to ensure compliance with the AEPEP.

Companies are required to rehabilitate technically and biologically all areas that are excavated, mined out, covered with tailings or otherwise disturbed and to establish a mine rehabilitation fund, based on their work program.

6 History

Silangan Mindanao Mining Company Inc ("SMMCI") is a joint venture company that was formed by Anglo American Exploration (Philippines) Inc ("AAEPI") and Philex Gold Inc ("PGI") in 1999. The objectives of the joint venture are to explore, and where feasible, develop the company's interests in the Surigao Project properties.

Reconnaissance sampling of the tenements in 1998 generated several significant drainage geochemical anomalies in the Bagacay area, and follow-up prospecting located float of mineralized diorite porphyry in June 1999. Based on geological mapping it was interpreted that a gold-rich porphyry copper deposit maybe preserved, beneath Quaternary cover. Palaeo-debris flows containing mineralized diorite porphyry boulders were intersected in several drillholes in March 2000, and interpreted to have been derived from a palaeo-high in the vicinity of Boyongan, some 2-3km to the southeast.

6.1 Exploration History

Diamond drilling at Boyongan commenced in May 2000. In August of the same year the discovery hole (TSD-6) intersected oxide copper mineralization, comprising malachite and azurite veinlets and fracture coatings within potassic altered diorite porphyry. The weighted average value of the intercept from 93m downhole is 329m at a grade of 0.90% Cu and 2.07g/t Au.

By November 2003, 55,327m of diamond drilling had been completed to define the configuration of the deposit.

In early 2003, Dr Harry Parker of AMEC Engineering and Construction Services Ltd was commissioned by AAEPI to conduct an independent review of the geological information, data collection procedures and methods used in estimating mineral resources. Dr Parker's report concluded that the geological data collection was well conceived; the database exceeded the industry standard and the estimation of grades and mineral zones were consistent with the data interpretation. This report was submitted by AMEC in October 2003. In November 2003, AAEPI prepared a comprehensive Mineral Resources Evaluation and Exploration Potential which incorporated Dr Parker's report and defined the requirements for continuing the project beyond the advanced exploration and into the pre-feasibility stage.

In February 2004, PGI retained AMEC to prepare Mineral Resource and Reserve estimates; a NI43-101 compliant Technical Report was completed by the Qualified Person, Mr. M. Corey.

In 2006 Anglo American commenced a detailed study. The scope of the work was to carry out a comprehensive study of the viability of the project to a stage where the mining method or the pit configurations are established and an effective method of mineral processing is determined. This includes a financial analysis based on reasonable assumptions of technical, engineering, legal, operating, economic, social, and environmental factors and the evaluation of other relevant factors which are sufficient for a qualified person, acting reasonably, to determine if all or part of the mineral resource may be classified as a mineral reserve. The cost estimate was done at a +/-25% level of accuracy for plant engineering and the rest of the estimates were factored based on similar operations.

Anglo American also re-engaged Dr Harry Parker of AMEC Engineering and Construction Services Ltd to provide an independent review of the geological information, data collection and methods used to arrive at the estimate of mineral resources for the Boyongan deposit and to then provide further comment and views as to the appropriateness of the resource to reserve conversion process.

6.2 Current Exploration

To May 2007 118 triple-tube diamond drill holes, totaling 66,103m, have been completed within the Boyongan deposit. The drillholes were positioned on a nominal 100m x 50m grid and angled predominantly 60° to the north. This drilling has enabled the definition of a mineral resource in the upper 850m of the bedrock. Ongoing exploration in late 2007 and early 2008 followed, due to indications of deeper (1,500 to 2,000m) porphyry centers beneath the currently-defined resource at Boyongan.

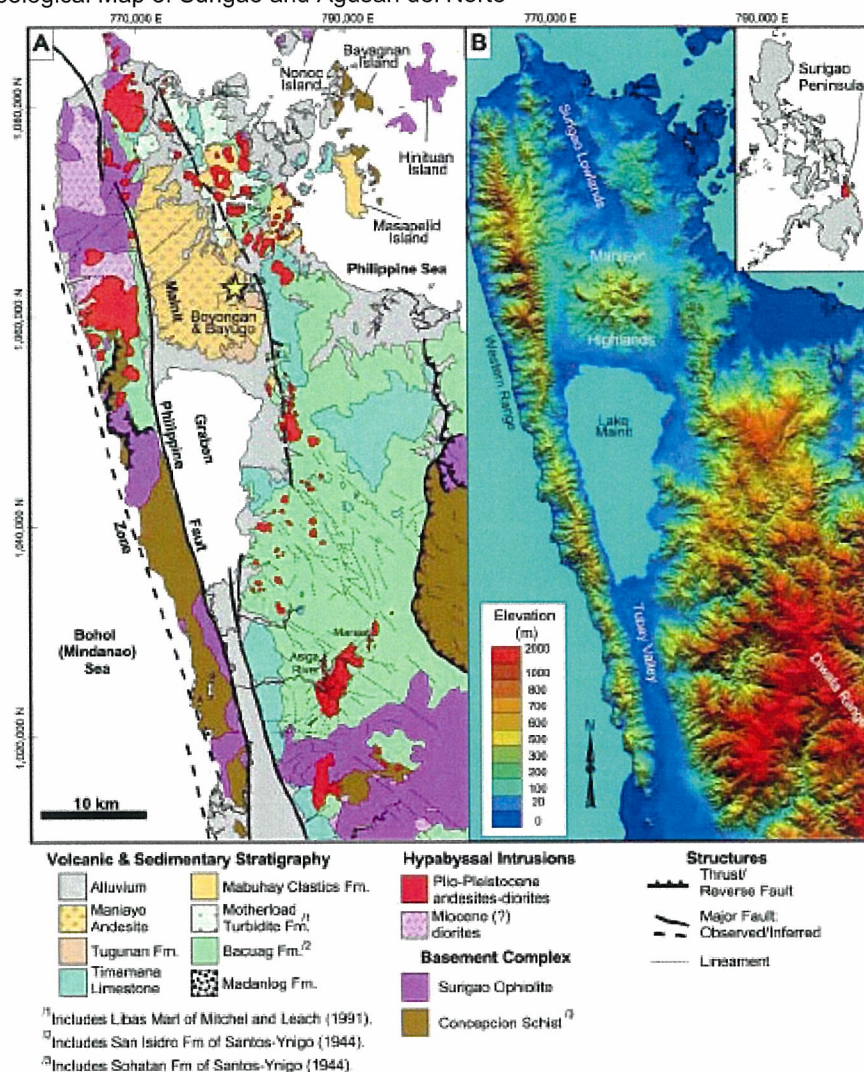
In June 2008, Philex engaged the services of IRES to re-estimate the Mineral Resources for Boyongan using a more complete dataset than was used for Anglo's resource estimate. The former resource model used data up to mid-May 2007 whilst the current IRES resource model uses data up to December 31st, 2007. This model was used in this PFS study.

7 Geological Setting

7.1 Regional Geology

The Surigao Gold District is situated at the northern tip of the Pacific Cordillera of Eastern Mindanao. It is located within a well-defined, transtensional segment of a Pliocene-Quaternary volcano-plutonic arc related to subduction of the oceanic Philippine Sea Plate at the Philippine Trench. The NNW-trending, transcurrent Philippine Fault is related to the oblique convergence of the Philippine Sea Plate with the Eurasian continental margin, and parallels the Pacific Cordillera. District-wide structures related to mineralization consist of NE-SW and NW-SE fault sets. Boyongan is located near the intersection of these two structural trends; see Figure 4.

Figure 4.0 Geological Map of Surigao and Agusan del Norte²



² Figure modified from Braxton (2007)

7.2 Local Geology

The Boyongan Deposit is centered on the two early-mineralization composite porphyry stocks, which appear to have been emplaced in conjunction with the mineralizing fluids. Intense quartz stockwork veining has developed within these early-mineralization stocks and the adjacent breccia wallrocks. These two cylindrical stocks, spaced approximately 150m apart, were emplaced into a diatreme breccia complex and form the West and East Zones of mineralization. The two mineralized zones are separated by a N-S trending upward-flared polyolithic breccia which may have acted as an impermeable wallrock, preventing much of the mineralizing fluids from flowing further outward and upward from the fluid upflow conduits, thus localizing the main high-grade copper-gold zones. Drilling since 2003 has further delineated an area of exotic copper mineralization to the NNW of the West Zone which appears to have been derived by leaching and dispersion of copper from the Bayugo deposit.

7.3 Stratigraphy

The North Project Area is located in the northernmost sector of eastern Mindanao's Pacific Cordillera. The stratigraphic succession is well preserved, from the oldest basement units through the youngest Quaternary volcanics (Fig. 5). Cretaceous to Paleogene basement rocks consisting of ophiolitic assemblages such as the Dinagat-Surigao Ophiolite and dismembered metamorphic complexes are well exposed.

Post-ophiolite sedimentary rocks consisting of Oligocene interbedded limestones and calcareous mudstones overlie pillow-basalts of the Lower Bacuag Formation (Santos-Ynigo, 1944). The Miocene section is represented by regressive deep marine to shallow-water sediments, which compose the Motherlode Formation. Shallow-water sediments that overlie the deep marine turbidites include sandstones, siltstones and mudstones including the Taganaan Marl Member. The Mabuhay Clastics, a sub-aerial to shallow-marine volcanoclastic-dominated Pliocene unit, blanket large portions of the older sequence. The Timamana limestone was deposited during marine transgression in the Upper Miocene-Pliocene. Post-Pliocene to Pleistocene uplift is represented by the volcanic-derived Tugunan Formation (Santos-Ynigo, 1944). Renewed volcanism in the Quaternary was wholly sub-aerial and resulted in the emplacement of intrusive plugs, endogenous domes and strato-volcanoes such as Mount Maniayao.

8 Deposit Type

The following section was extracted from the "Qualifying Technical Report on the North Project Area, Surigao del Norte, Mindanao Island, Republic of the Philippines", June 30, 2004, by Mr. Michael C. Corey and is included for completeness.

Diamond drilling at the Boyongan prospect has confirmed that the property is host to significant copper-gold porphyry-type mineralization. Field observations further indicate that the porphyry deposit has been overprinted by high sulphidation epithermal-vein type copper-gold mineralization. The porphyry copper-gold mineralization at Boyongan is associated with an altered multi-phased diorite stock, and the current deposit model is based on observations by Dr. R. Sillitoe, a renowned porphyry Cu expert. The Boyongan deposit is similar in many respects to other well-documented Cu-Au porphyry deposits in Southeast Asia. These include the giant Grasberg Cu-Au deposit in West Papua, Wafi deposit in Papua New Guinea and the Lepanto deposit in the Philippines.

The Boyongan system is centered on the steeply inclined magmatic hydrothermal breccia body measuring >1 km across that was emplaced mainly into a precursor diorite porphyry pluton and locally into the basaltic and the overlying non-intrusive wallrocks. Emplaced into the magmatic hydrothermal breccia body is the Boyongan Cu-Au deposit, which is centered on the two early mineral porphyry stocks that appear to have served as the primary conduits for the mineralizing fluids. Intense quartz stockwork veining has developed in these early-mineral stocks and a few meters into the immediately adjoining breccia wallrocks, which along with the surrounding zone of generally lower grade copper-gold mineralization, is associated with the pervasive K-silicate alteration.

The two early-mineral stocks were emplaced into the magmatic hydrothermal breccia body as cylindrical stocks and are about 150m apart forming the West and East Zones of mineralization. The two mineralized zones are separated by a N-S trending upward-flared polyolithic breccia. The polyolithic breccia underwent milling and upward transport of clasts, thus containing several rock types and typified by clast rounding with higher matrix to clast ratios. The geological model established to date has also developed metal and mineral zoning and its application to the definition of ore types and nearby exploration targets. Four transitional metal zones are defined at present: an outer copper zone containing only low-grade gold, a main central copper-gold zone, a very restricted and speculative central gold-only and a deep underlying low-grade core.

The polyolithic breccia may have acted as an impermeable wallrock, preventing much of the mineralizing fluids from flowing further outward and upward from the fluid upflow conduits, thus localizing the main high-grade copper-gold zone especially at the east zone. The gold may have been confined essentially in the quartz stockwork veinlets and the precipitation of copper continued further outward into the polyolithic breccia. The values around the western stock show less clear copper dominance over gold. This is due probably to the more outward development of quartz veinlets and the dominance of the more permeable fine- and medium-grained diorite breccias around the western stock.

The zoning pattern is attributed to the thermal structure at the time of early stock emplacement and associated mineralization, with temperatures decreasing both outwards and upwards. Copper and gold precipitation was minimal in the deep core to the system, presumably because temperatures were too high there to destabilize the chloride complexes of copper and gold. The upward and outward increase in Cu/Au ratios may be explained in terms of decreasing temperatures. The copper-rich halo is well developed around the eastern early porphyry stock, but rather less well around the western stock, where only parts of a few holes show a clear dominance of copper over gold.

9 Mineralization

The following section was extracted from the "Qualifying Technical Report on the North Project Area, Surigao del Norte, Mindanao Island, Republic of the Philippines", June 30, 2004, by Mr. Michael C. Corey and is included for completeness.

The alteration and ore-mineral assemblages observed at Boyongan are similar in type and style to those documented within other island arc porphyry copper deposits. The potassic alteration zone at Boyongan is manifested by secondary biotite, orthoclase, magnetite, quartz, and minor actinolite and formed largely in the centre of the system. This alteration type generally contains the most abundant mineralization. The potassic alteration grades outward to a propylitic aureole as indicated by the presence of chlorite and albite associated with epidote, illite and calcite. The potassic alteration is overprinted by an intermediate argillic alteration consisting of chlorite, illite-montmorillonite, sericite, and specular hematite and also partially overprinted by a phyllic alteration assemblage of quartz, pyrite and sericite. A remnant advanced argillic lithocap is inferred by clasts of clay, quartz, pyrophyllite, alunite, and tourmaline within Quaternary sediments in drill hole TSD-6. Magnetite occurs as clusters, fine disseminations and as replacements within volcanoclastic rocks adjacent to the deposit.

Three zones of copper-gold mineralization are indicated by diamond drilling within the Boyongan deposit. These are, a) a shallow, oxide, copper-gold zone comprised predominately of cuprite, azurite, malachite, chrysocolla, neotocite, chalcocite and native copper. The oxide zone is associated with the two early-mineral diorite porphyry intrusives, b) a mixed oxidesulphide zone consisting of occasional limonite, native copper, chalcocite, pyrite and chalcopyrite, and c) a hypogene zone at depth containing pyrite, chalcopyrite, bornite, and rare molybdenite associated with the inter-mineral diorite stock. Anglo indicates that pyrite content increases outward from the center of the mineralization. Molybdenite is minor, occurring within quartz stringers and veinlets, and fillings on some fractures, associated with pyrite and chalcopyrite. Remobilization of copper and gold appears to be minimal in the mineralized zones although some local secondary enrichment of copper and gold is apparent in the oxide and mixed zones. Boyongan drillhole cross sections display individual assays from within the East and West sections of the Boyongan deposit and show significant Cu + Au enrichment within the oxide zone.

An intense quartz stockwork zone associated with the early mineral porphyry intrusives forms a gold rich zone as intersected in TSD-14, 15, 18, and 20. A core sample taken from TSD-15 showed a maximum assay value of 34.9g/t Au and 6.0% Cu, and Anglo reports that visible gold does occur within sections of intensely mineralized drill core. Zones of low-grade mineralization consist of minor disseminated sulphides within quartz stockwork veins.

The Boyongan deposit is thought to have developed beneath a prominent ridge, which subsequently subsided rapidly into a pull-apart extensional basin within the Philippine fault system. This was then covered and preserved by Quaternary lacustrine sediments and volcanic formations. The deep oxidation of the deposit is likely in part attributable to the high permeability of the intrusion breccia host rock.

Basaltic wall-rock adjacent to the diorite porphyries is also locally mineralized. Drilling has intersected brecciated and chloritic basalt containing chalcopyrite and bornite within quartz stockwork veins. Also intersected was skarn-type mineralization manifest as magnetite, epidote, calcite, pyrite, chalcopyrite, vuggy quartz, and chlorite developed within altered basalt wallrock adjacent to the intrusive-hosted mineralization.

10 Exploration

Geological prospecting and sampling has been conducted within most of the North project tenements. A combined aeromagnetic-radiometric survey totaling 2,256 line km was flown by World Geoscience Corporation from February 12 to 27, 1999. This survey defined anomalous areas such as Suyoc, Lascogon, Danao, Mapawa, Mapaso, Bagacay, Nabago and Boyongan. Follow-up prospecting programs entailing collection of stream sediments, which are sieved to fine sizes (less than 177 micron), and soil sampling along ridges and spurs were done on all the anomalous zones defined by the geophysical survey.

11 Drilling

The deposit has been drilled at an approximate grid of 100m x 50m with the majority of the holes drilled at an angle of 60° to the north.

Drilling was carried out by United Philippines Drilling ("UPD") using triple-tube diamond coring. Core recoveries from the bedrock average >95%, and there is no conclusive evidence that there is any relationship between core recovery and grade. Downhole surveys were taken approximately every 100m down the hole. The drillhole collar coordinates, and tenement corners and boundaries were surveyed by an independent contractor. A topographic surface for the Boyongan area was generated using surveyed borehole collars and a series of compass and tape traverses, tied into the surveyed collars.

12 Sampling Method and Approach

Cores are transported by truck from the site facilities to the project offices in Surigao City. An excellent logging manual was written by Gonzales and Braxton (2003, 2007). The logging form includes a graphic log (mainly to designate rocktype and core angles for fractures) and columns for lithology, alteration, texture, mineralization and oxides. These columns are subdivided further so as to permit a good checklist for describing the rock. There is also space for written comments. The core logs have a vertical scale of 1:20, leading to 20 m being logged per sheet. Information is entered every meter. This can lead to some logs having many tens of pages.

The drill core logging procedures follow internationally accepted best practice and have been documented in detail. Sampling (half core) is carried out in 3m intervals with an additional quarter core 'Field Duplicates' being taken every one in ten samples. The core cuttings from every ten samples were composited and panned for gold and native copper.

13 Sample Preparation, Analysis and Security

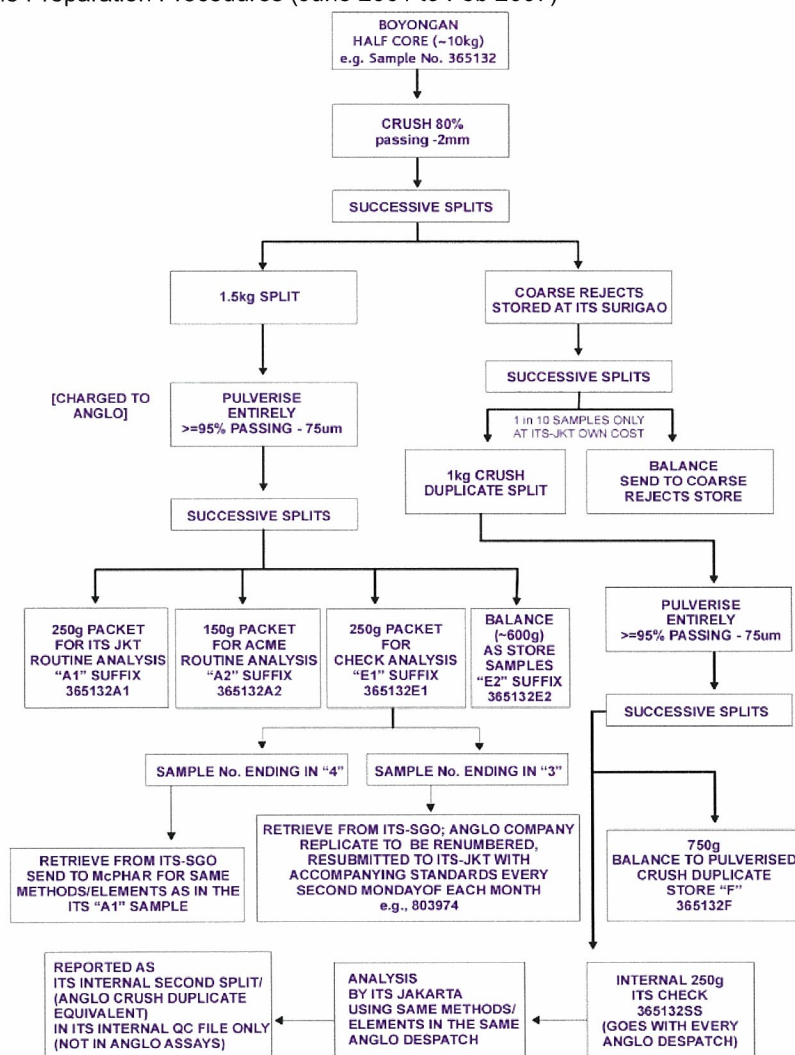
Using a "Clipper" core saw, half core sampling of the drill core is carried out at 3m intervals, or less where there is a clear change in lithology, alteration or mineralogy. Sampled core is placed into pre-numbered calico bags and dispatched for sample preparation. Half core is retained in plastic core trays and stored in the Surigao core shed.

13.1 Sample Preparation

The following section was extracted from the "Boyongan Copper/Gold Deposit, Mineral Resource Evaluation and Exploration Potential", by Anglo American plc., by Mr. Duncan Campbell et al, January 2008 and is included for completeness.

From project commencement (August 2000) until June 2001 (to hole TSD17) the entire 3 to 10 kg sample was pulverized in an LM5 and split to obtain a 250 gm sample for analysis. Following extensive test work on the reliability of splitting crushed material, a modified sample preparation protocol, described in Figure 5, was introduced.

Figure 5.0 Sample Preparation Procedures (June 2001 to Feb 2007)

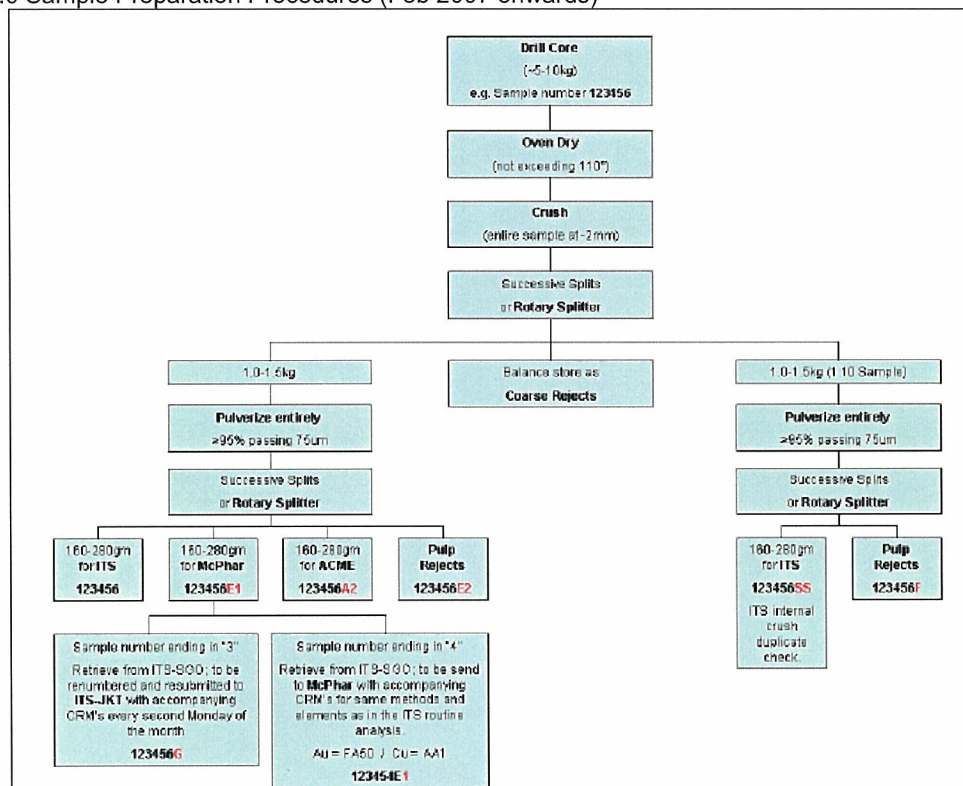


Between June 2001 and July 2003, all dried core from each sample was crushed in a single step to <2mm using a TM Engineering Rhino Jaw Crusher. Test work in July 2003 on 100 samples from commencement of ITS Surigao facility showed an average of 87% passing –2mm versus the minimum requirement of 80% passing –2mm. In late 2003, the procedure changed and the sample was first coarse crushed using a TM Engineering Rhino Jaw Crusher to -5mm, then secondary crushed using a TM Engineering Rhino Jaw Crusher to -2 mm. A test of 60 samples in August 2006 showed that 87.4% passed –2mm versus the minimum requirement of 80% passing –2mm. A lab audit in March 2007 showed that 88.1% passed –2mm versus the minimum requirement of 80% passing –2mm.

Between project commencement and February 2007, the -2mm crushed sample was split using an 18 mm wide, 24-channel Jones riffle splitter to approximately 1.5 kg. The crushed reject material was re-bagged and stored at the laboratory. Duplicates of the -2mm primary split were taken by routinely splitting another 1.5Kg sample from the reject material by ITS to monitor the efficiency of the splitting process. Since February 2007 the - 2mm sample is split using an eight cup rotary splitter and the duplicate is taken at this stage of splitting.

The 1.5kg crushed material is pulverized in a Labtech LM2 pulverizer to 95% passing 75µm. The granulometry of 1 in 10 pulps is screen checked to ensure this specification is met and results recorded by the laboratory. The crusher is flushed with barren material (unoxidized andesite) after every 10th sample. A flow diagram showing the sample preparation protocol employed since February 2007 is shown in the Figure 6 below

Figure 6.0 Sample Preparation Procedures (Feb 2007 onwards)



13.2 Sample Analysis

The primary laboratory was ITS in Jakarta who use a 50g fire assay with AAS finish for total gold and 3 acid digest with AAS for total copper. McPhar in Manila is being used as a

check laboratory. Each sample also underwent multi element analysis at ACME in Vancouver, while ACME's Santiago Laboratory carried out acid soluble copper analyses using both "hot" and "cold" (ambient temperature) methods. QC samples submitted included, 1/4 core (10%), coarse duplicates (10%), Certified Reference Materials ("CRM's") (11%) and pulp duplicates (>15%).

Five matrix-matched CRM's from Boyongan were prepared by Ore Research Laboratories in Australia and represented a spread of grade ranges and compositions. The construction of the CRM's and their performance is good and generally within 5% of the accepted value. Performance on blanks is generally excellent. Coarse crushed duplicates show acceptable variability. Pulp duplicates returned blind to ITS indicate acceptable precision and the duplicates sent to the check lab (McPhar) suggest that ITS report slightly elevated gold in the low grades compared to McPhar. Investigations into apparent bias are on going.

The QAQC program consists of the following:

- For every 100 samples there are 10 field duplicates. These represent a quarter split of the core.
- For every 100 samples there are 10 crushed duplicates (split from rolls crushed product). These are analyzed at ITS Jakarta, and are not blind to the lab.
- There are 9 pulp duplicates run at ITS (Jakarta) and 9 pulp duplicates (run at McPhar – Manila) per 100 samples.
- For every 100 samples there are 10 matrix matched certified reference materials ("CRMs"). The CRMs consist of one sulphide, one carbonate and three oxides. The CRM selected depends on the logged mineralogy.
- There is one blank (quartz prepared by ORE Research and Exploration in Australia) per batch of 25 to 100 samples.

13.3 Security

From commencement of the project until January 2002, drill core prepared at the Surigao Core Farm was dispatched in batches to ITS Manila through LBC Couriers as airfreight. A Chain of Custody ("COC") procedure was implemented to track the sample dispatch as shown in Appendix 4. However, due to increased internal Philippines security, the COC procedure was ultimately unsuccessful and ceased in July 01. As of February 2002, all batches of drill core were dispatched to ITS's purpose built sample preparation facility in Surigao, personally accompanied by an Anglo American staff member. In 2006, upon recommencement of the drilling, the COC form was expanded to include extra information.

A COC form accompanies each batch of samples to the ITS Surigao facility and they are signed off by the lab manager after checking the received samples with the sample list provided. Each sample is batched into multiple pulp packages for analysis at ITS Jakarta and Acme Analytical Laboratories (Chile) Ltda, via Acme Analytical Laboratories Ltd, Vancouver. One in ten samples is sent to a check lab, McPhar, in Manila, with a duplicate of this pulp going also going to ITS in Jakarta.

The pulp samples dispatched to the ITS laboratory in Jakarta are accompanied by the COC form. ITS Jakarta note the condition of the samples, check that all samples have been received, fill in the COC form and fax it back to the Silangan Mindanao office in Surigao. The pulp samples distributed to ACME Analytical in Vancouver are not accompanied by a COC form, but only a sample list. Nevertheless, ACME, Vancouver, sends the Silangan Mindanao office in Surigao a confirmation of receipt note which records any missing samples or discrepancies. Sample pulps sent to McPhar, Manila, are accompanied by a COC form and this is filled in as specified and faxed back to the Silangan Mindanao office in Surigao.

13.4 Geophysical Characterization

Down-hole geophysical logging of holes TSD-50, TSD-52 and TSD-53 was undertaken by Reeves Wireline Services during August 2002, as a component of Anglo Technical

Division's Geosciences Resources Group 'Proof-of-Concept' downhole wireline geophysical logging of porphyry deposits project.

The logging suite comprised three-arm calliper, natural gamma ray, magnetic susceptibility, electrical resistivity, inductive conductivity, induced polarization, sonic velocity, and three-arm dipmeter. The wireline logging could be carried out down to depths of 800m in angled boreholes drilled in highly altered and fractured rocks, with exceptions only where casing could not be extracted.

On the basis of the measured physical properties, none of the conventional surface geophysical exploration techniques such as resistivity, EM, IP, magnetics or seismics, would be suitable for exploration of Boyongan-style mineralization elsewhere in the district. Even if the deposit had no conductive cover, the oxide ore would not represent a geophysical target, although magnetics may assist in mapping certain lithological units and structures.

It was not possible with the suite of wire-line logging tools used to discriminate different grain sizes in the diorite porphyry intrusives, and therefore it cannot be utilized as a basis for the discrimination or lithological mapping of intrusive phases and breccia units. A hybrid lithological classification scheme has been proposed by subdividing lithological units on the basis of distinctively different geophysical wire-line signatures, however this results in a dramatic over simplification of the geology and the controls to mineralization.

Grade estimation from the wire-line logging results cannot be achieved with sufficient accuracy to aid in orebody modeling and only defines gross trends of grade. Structural trends have been derived from dipmeter logs and are consistent with the expected genetic models of porphyry intrusives.

The geophysical wire-line logging has contributed additional petrophysical information that could be used to revise geotechnical classification of the Boyongan deposit; however routine geochemical and geological parameters are far more reliable in ore body modeling and grade estimation. The technique may be more useful if a more representative logging coverage and hence better sampling statistics could be achieved. However, given the expense and difficulty of completing borehole work at Boyongan, no more geophysical logging can be justified.

There has been no additional geophysics undertaken at Boyongan since 2003.

14 Data Verification

The following section was extracted from the "Boyongan Copper/Gold Deposit, Mineral Resource Evaluation and Exploration Potential", by Anglo American plc., by Mr. Duncan Campbell et al, January 2008 and is included for completeness.

Anglo American have implemented the acQuire data management system (DMS) in its numerous offices and exploration sites worldwide. The acQuire DMS was set up for the Surigao Project in April 2002 by Metech Australia and is used primarily for the day-to-day management and effective storage of drillhole data. As a window to the Project's database, acQuire provides the tools to establish, access and maintain a database that is structured, efficient and validated.

Each geologist is responsible for the integrity of the information in the database relating to the holes that they have logged. The procedure leading up to the sign-off are: Drill hole data from the Geological and Geotechnical Log Sheets are encoded using both the acQuire data entry and the Excel data entry forms. Data encoded using the acQuire data entry objects are directly saved in the SQL database (DB), while those of the Excel data entry forms are imported to acQuire to be stored in the DB. Validation rules that were built into the data entry forms include parameters such as minimum, maximum values and Look-up Tables for geological codes and filter errant entries. Occurrence of validation errors are brought to the attention of the geologist and corrected in the log sheets immediately.

Once the data entry for a hole is complete, the encoder generates print-outs. The geologists compare and validate the entries in the Log Print-outs against the Original Log Sheets. Corrections, if any, are made in the Log Print-out sheets, and the corresponding edits in the DB are made through acQuire.

Another set of print-outs are generated for the edited holes and the geologists recheck the printout. If no errors are found, the geologist signs the original log sheets and the log printouts.

By June 2007 the geological records of all holes had been validated and signed-off by the geologists.

At the time that the database was "frozen" for the resource estimation, the database contained holes TSD01 to TSD114. By June 2007 the geological records of all holes had been validated twice and were prepared for signed-off by the geologists. In the acQuire database there were some 12789 geological samples of which 9,221 samples had been sent for assay of copper and gold. There were some 8,169 assays for hot and cold acid soluble copper.

14.1 Analytical Verification

The QA/QC procedures employed on the Boyongan Project follow the principles of Anglo American's standard operating procedure. In addition to the routine analysis of QAQC data, periodic ad-hoc assessments have been carried out during the project.

Table 8, below summarizes the number of samples submitted to the various laboratories for the Boyongan Project TSD01 to TSD109. The green highlighted boxes are reported on in this report.

Table 8.0 Samples Submitted by Laboratory³

Laboratory	Analysis	Status	Drill Sample (1/2 Core)	Field Dup (1/4 Core)	Company CRMs	Company Blank	Laboratory Crushed Dupe	Dupe	Intra- (ITS) & Inter- (McPhar) Laboratory
ITS, Jakarta	Total Cu and Au	Principal	16,135	1,562 (9.6%)	1,737 (10.8%)	384 (2.4%)	1,604 (9.9%)	859 (5.3%)	2,450 (15.2%)
McPhar	Total Cu and Au	Secondary	1,598 (9.9%)	13 (0.8%)	154 (0.9%)	28 (1.7%)	NII	NII	2,393 (14.8%)
ACME, Santiago	ASCu	Principal	15,093 (93.5%)	1,478 (9.8%)	1,519 (10.1%)	350 (2.3%)	NII	NII	NII
ACME, Vancouver	Multi- elements	Principal	14,305 (88.7%)	1,446 (10.1%)	1,445 (10.1%)	351 (2.4%)	NII	NII	NII
AARL	Metallurgy		400 (2.5%)	NII	NII	NII	NII	NII	NII
Lakefield Research	Metallurgy		32 (0.2%)	NII	NII	NII	NII	NII	NII

Terminology used in this report for QAQC sample types is as follows:

- **Field duplicate:** Quarter core prepped using routine sample preparation procedures and submitted as a routine sample at a frequency of 1 in 10.
- **Company CRM:** Company Certified Reference Material (CRM) and Company Blanks submitted with each batch of drill pulps
- **Company Blank:** Blank, inert material submitted by the company to assay batches. *CRM's and Blanks should be submitted at a combined frequency of at least 1 in 10.*
- **Laboratory Crushed Duplicate:** Split of the 2mm jaw crushed core, pulverized separately from the original split and submitted as a routine sample. This is an ITS internal check and submitted at a frequency of 1 in 10 (these duplicates were initially not blind to the laboratory).
- **Pulp duplicates:** A 250gm split from the pulverized sample and submitted as a routine sample at a frequency of 1 in 10.
- **Intra-laboratory duplicate:** A 250gm pulp split submitted on a periodic basis (usually monthly) to the principal laboratory (i.e. different time and number).
- **Inter-Laboratory duplicate:** A 250gm pulp split submitted on a periodic basis (usually monthly) to the check laboratory.

Certified reference materials have been included in all batches dispatched from the Boyongan project at an approximate frequency of 1 per 10 pulps. Prior to the inclusion of the AABY matrix-matched standards (Batch 4446, 09/02/02), selected Geostats and Gannet certified reference materials were used.

Evaluations of standards are completed on a batch wise basis; control plots and a paper trail of action taken and issues arising are stored on site. In this report, the overall performance of laboratories based Cu – Au standards submitted are presented along with key observations. The evaluation of performance is based on the value received versus the certified value and presented as a percent relative difference.

14.1.1 Standards Analyzed by ITS

The performances of the copper and gold standards over time were plotted. The figures provide the relative difference from the expected value of the standard up until March 2007. All standards for the one element are plotted together and the red line shows the 15 point moving average. For copper the graph indicates that:

Quality of standards submitted and the precision of analysis improved with the introduction of matrix-matched Certified Reference Materials (>200 time series), but there has been a lapse in precision since mid 2006.

³ Source: Boyongan Copper/Gold Deposit Mineral Resource Evaluation and Exploration Potential, Anglo American, Duncan Campbell et al; January 2008.

Pre matrix-matched standard performance at ITS showed a negative bias for two CRM's (BM11 and GBM396-5) but no bias for the other CRM's. Nevertheless, all of these CRM's produced highly variable results.

Pre matrix-matched standards appear less homogenous. Post matrix-matched CRM performance at ITS shows a significant improvement on the pre matrix-matched standards, but a slight positive bias (mean +1.74 %RD) has never been consistent over the last 5 years.

ITS precision on the matrix-matched CRMs appears to have been on the decline since mid-2006, as demonstrated by a wider range of values from the 1100 time series until present. This is despite there being few failures reported. That is, more CRMs are reporting either just within the 2 standard deviation limits or between the 2 and 3 standard deviation limits. A sample is only considered a failure when a CRM falls outside of the 3 standard deviation limit or there are two consecutive CRM's that fall between the 2 and 3 standard deviations.

The standard and CRM performance for gold analyses at ITS (figure below) indicate that:

- Quality of standards submitted and the precision of analysis improved with the introduction of matrix-matched Certified Reference Materials (>300 time series), but there has been a lapse in precision since early to mid 2006.
- ITS bias has fluctuated from negative in the first 12 months of the project when variable quality standards were used (mean bias -2.41%) to a slight positive bias during the next 15 months of the project to the end of 2003 (mean bias +1.73%). Since recommencement of drilling at the project, the precision on results by ITS on Au analysis has decreased significantly.
- There is an obvious transition phase from negative to positive bias results that correlates with the introduction of the matrix-matched CRMs and preparation by ITS for ISO17025 accreditation.
- The analytical precision "range" was reduced by half after the introduction of matrix matched standards and the initial phase of lab audits. Pre matrix standards precision range = 28% around certified value; post = 14% (outliers removed), showing improved homogeneity in matrix standard.
- QAQC procedural changes improved the results of the standard data (i.e. targeted lab audits, matrix matched standards).
- Mapped bias during the course of the project is reflected in Geostats half yearly round robins.

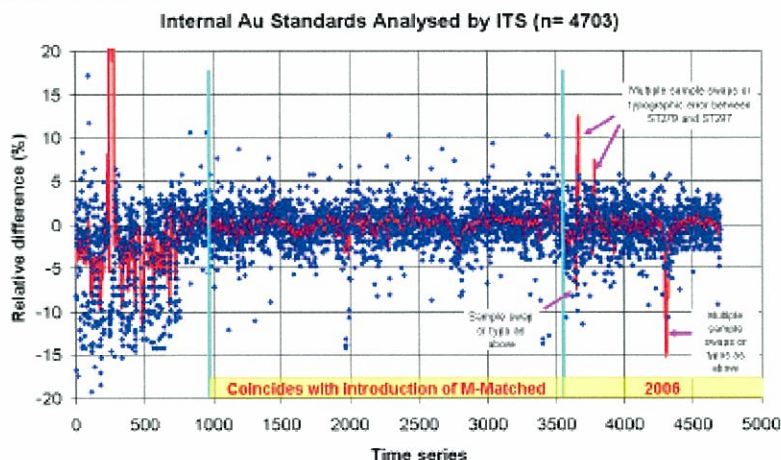
Since recommencement of drilling in late 2005, ITS has struggled to maintain the precision it provided during 2003. All CRMs are affected, some more so than others, but there have been few failures. It appears that like Cu, more CRMs are reporting either just within the 2 standard deviation limits or between the 2 and 3 standard deviation limits for each CRM. The problem is magnified by non-matrix-matched CRM, OREAS 53, which has consistently produced positively biased results since its introduction in mid 2006. This CRM was introduced to replace AABY-5. There has also been a slight increase in the use of CRM AABY-1 since recommencement of the project in 2005. This CRM has consistently reported at the lower limits of the acceptance range since its introduction, despite the fact that ITS was one of ten labs used as part of the round robin certification team and at that time provided Au grades between 7.63 and 7.79 ppm, consistent with other labs and identical to the certified value of this CRM (7.64 ppm). Overall results for Au CRM's acceptable, but they can be improved.

14.1.2 ITS Internal Standards

The performance of ITS's own internal standards for Cu and Au over time were plotted; see Figure 7. The figure provides the relative difference from the expected value of the standard up until March 2007. All standards for the one element are plotted together and the red line shows the 15 point moving average.

The plot below mimics those produced when using Anglo Americans matrix-matched CRMs. They appear to show that constant interaction with the lab, through lab audits and constructive feedback, have a larger influence on the precision of the results than the composition of the matrix-matched material alone.

Figure 7.0 ITS Internal Au Standards - Relative Difference from Calculated Value



The precision of ITS's internal standards was poor before the lab achieved ISO17025 accreditation, but the improvement does not coincide with the introduction of the matrix-matched CRMs, but probably with a period of intense interaction between the local, regional and global geochemists and the lab.

Since recommencement of drilling at Boyongan in late 2005, the precision of ITS's own standards has deteriorated. This suggests that ITS's internal QA/QC needs to be tightened and that staff retraining may be required as illustrated by frequent sample swapping between two similarly labeled internal standards.

14.1.3 ITS Blanks

The blanks submitted were a blank standard called ORE 22 (from Ore Research). Blank data from ITS is generally of good quality; however contamination of flux was picked up at one stage during the course of the project. This was corrected and the low levels of contamination would not have impacted on the results. Prior to the inclusion of matrix-matched standards, no blank material was submitted. In 2006 it was noted that several blank CRMs were reporting between 0.005 and 0.01 ppm Au.

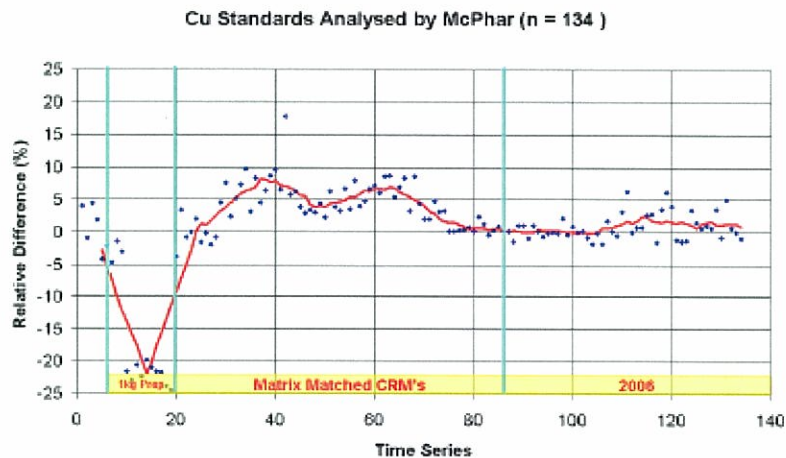
14.1.4 McPhar Standards

All standards analyzed by McPhar are plotted as relative differences in Figure 8 and Figure 9, below. The figures provide the relative difference from the expected value of the standard up until March 2007. All standards for the one element are plotted together and the red line shows the 10 point moving average. The main observations for Cu are:

The introduction of matrix matched CRMs saw a shift in the base line bias from negative and strongly negative to slightly positive.

- McPhar was producing positively biased results during 2002 and 2003 (+4.8%), but major improvements coincide with lab audits and constructive interaction between Anglo American staff and McPhar.
- In late 2006, McPhar began to produce slightly positive biases (1.5%).
- Limited blank data (n = 8) report Cu < 10ppm.

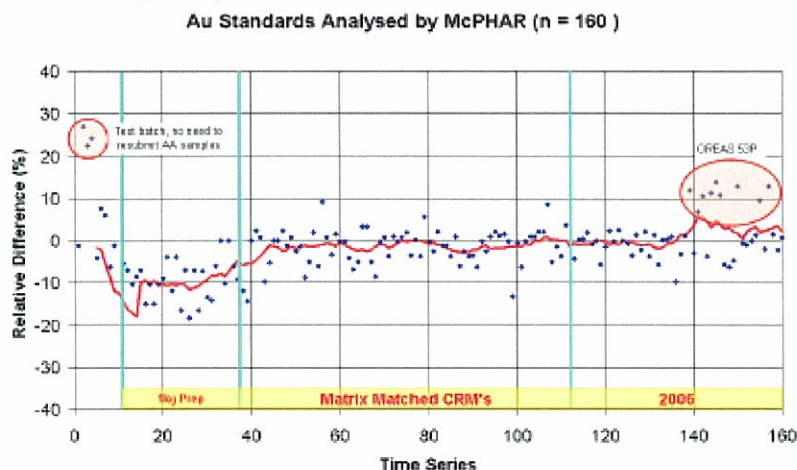
Figure 8.0 Cu Standards Analyzed by McPhar



For Au the main observations are:

- Pre matrix matched standards are less homogenous and significantly under report (bias -5.73).
- Post matrix matched standards bias = -1.3% RD
- The worst batch with poor analysis (e.g. 3651) was a test batch of samples sent to McPhar to determine their suitability to assess Au concentrations using different methods and to compare their results with ITS. The batch did not require resubmission.
- Limited blank data (n = 11) up until mid 2003 regularly reported Au concentrations up to 4 times the detection limits (0.005 ppm Au). McPhar were subsequently instructed to use new crucibles for every sample. More recent data (total n = 29) shows that there have been no Au concentrations reported above 0.01 ppm.

Figure 9.0 Au Standards Analyzed by McPhar

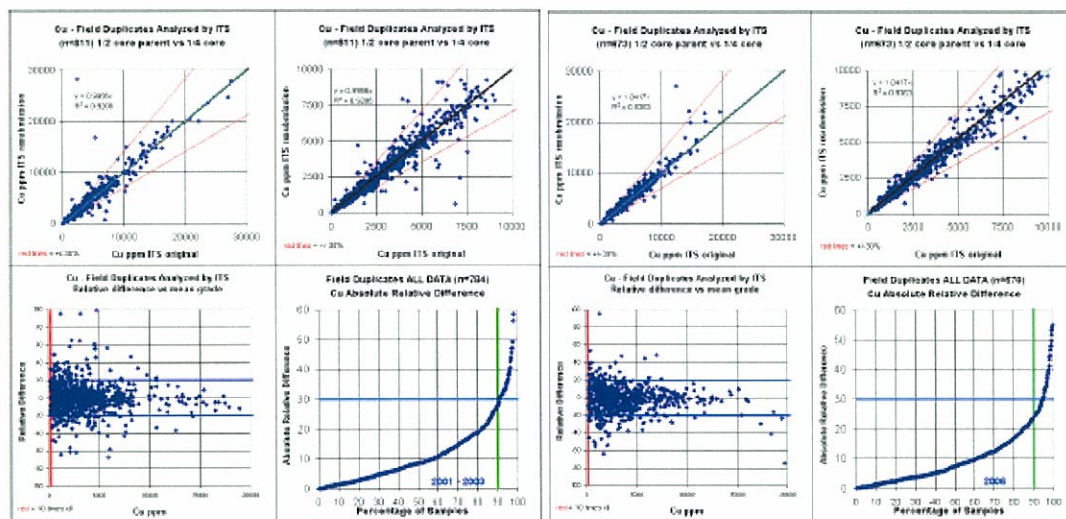
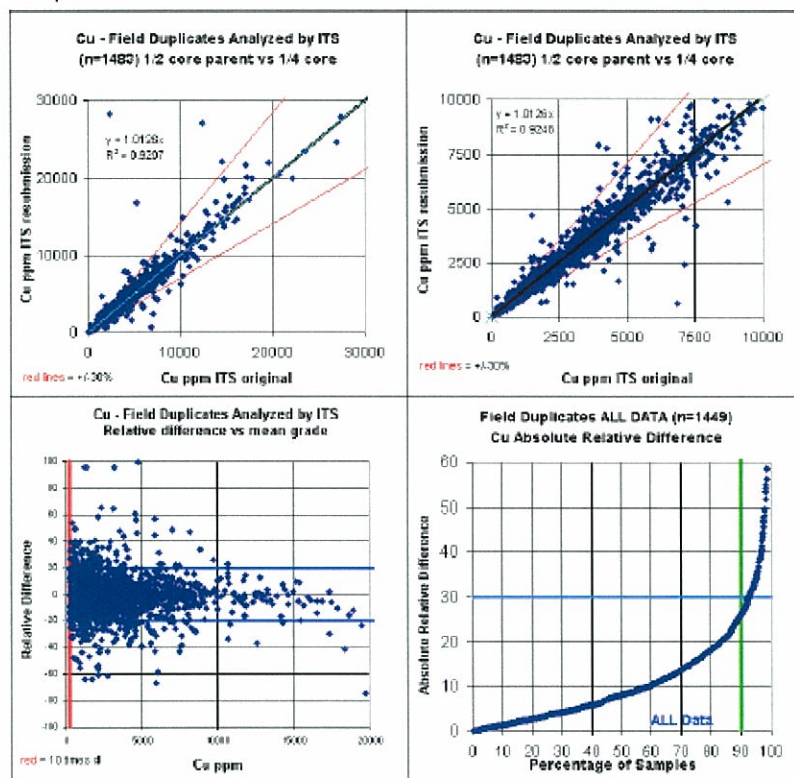


14.1.5 Field Duplicates - Assayed At ITS

Core duplicates have been routinely collected since April 2001. Quarter core is prepped using routine sample preparation procedures and submitted to the assay laboratory at a frequency of 1 in 10. The comparison of the field duplicates (1/4 core) with the original (1/2 core) is given in Figure 10, below which show scatter plots as well as cumulative absolute relative difference plots (Absolute relative difference = absolute (original - duplicate) / mean of original and duplicate x 100). These duplicates are taken from different parts of the drill core and therefore the heterogeneity between samples is expected to be greater than for coarse crush duplicates. Nevertheless, depending on the element being analyzed and the nature of the mineralization, at least 90% of the samples should reasonably report an absolute relative difference <30% (Cu) to <40% (Au), if sample collection at the core cutting stage is representative of the ore body. A complete list of data including a break down of the pre-2003 and post-2006 observations is presented in Figure 10. The 30% line has been drawn on the scatter graphs as a guide. The main observations for copper are:

- There is a strong correlation between the parent 1/2 and the 1/4 core field duplicate ($R^2 = 0.9207$) and the mean grades are virtually the same (1/2 core 3915 ppm; 1/4 core - 3996 ppm).
- 92.6% of the data falls within a 30% absolute relative difference which is reasonable.
- A grade versus percent relative difference plot demonstrates that there is no bias between the 1/2 core and 1/4 core samples.

Figure 10.0 Cu - Field Duplicates Scatter Plots and Relative Difference Plots



The comparisons for gold are given in Figure 11, below and the main observations include:

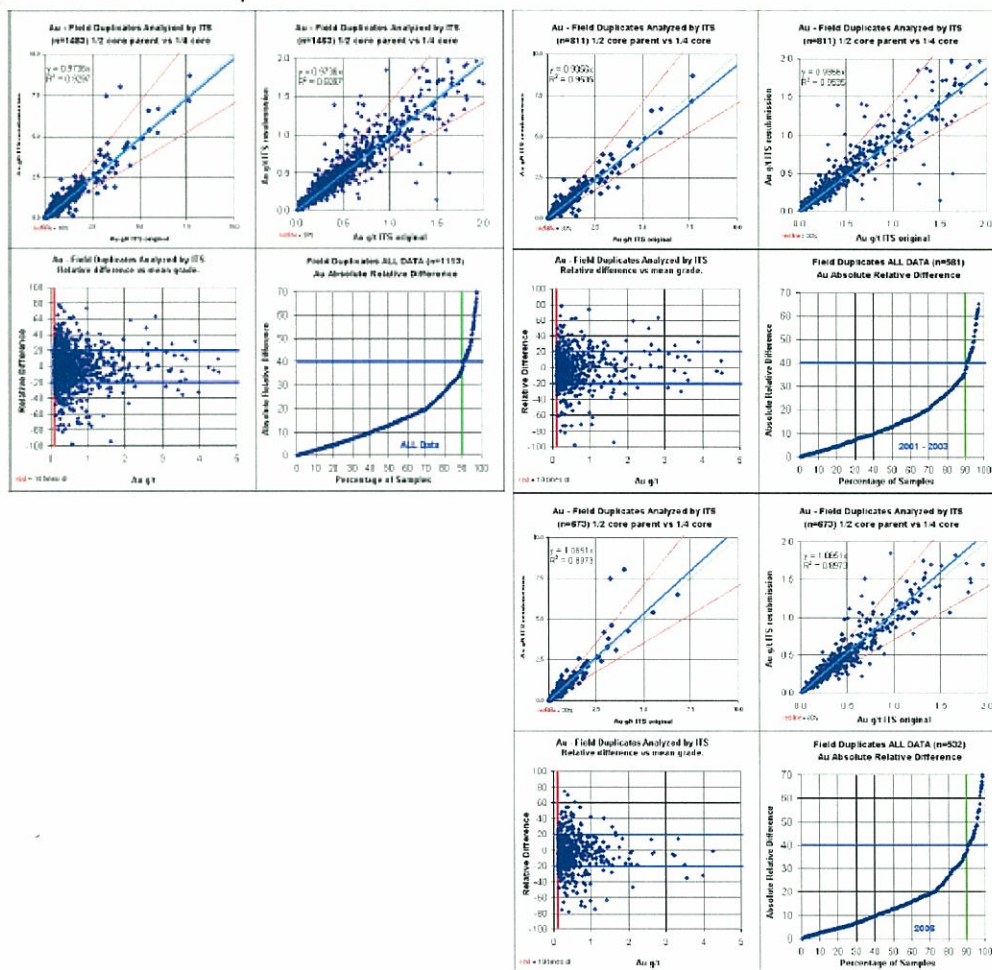
- There is a strong correlation between the parent 1/2 and the 1/4 core field duplicate ($R^2 = 0.9297$) and the mean grades are similar (1/2 core 0.676g/t; 1/4 core 0.687g/t, a 1.6% difference).
- The relative difference plot shows that 90% of the pairs have relative differences of less than 37%. This appears high, but given the nature of gold in drill core and the

fact that these samples are field duplicates from different parts of the core, it is a reasonable result.

- There is no apparent relationship between Au concentration and % relative difference, indicating that there is no bias between the samples.

The results presented here from field duplicate data demonstrates that the drill core is being carefully cut and sampled and that there does not appear to be any bias by the samplers toward collecting one half of the core over another.

Figure 11.0 Au - Field Duplicates Scatter Plots and Relative Difference Plots



14.1.6 Crushed Duplicates – Original

Crushed Duplicates are a split of the 2mm jaw crushed core, pulverized separately from the original split and submitted as a routine sample. This is an ITS internal check (carried out by and submitted from, the ITS sample preparation facility in Surigao) and submitted at a frequency of 1 in 10. Note that the crushed duplicates are not blind to the laboratory. The coarse duplicates are expected to be less heterogeneous than the field duplicates and an internationally recognized benchmark is that least 90% of the samples should report an absolute relative difference <20% if sample crushing and splitting is being done correctly and free of contamination. A review of all duplicate data at the end of 2003, and again during 2005 revealed that for Au, 90% of the coarse duplicates had an absolute relative

difference as low as 10%, whereas for Cu, 90% of the coarse duplicates had a very low absolute relative difference of only 3%. The samples did not show any bias. For comparison, 90% of the pulp duplicates analyzed at the same lab had a higher absolute relative difference of 17% for Au and 6% for Cu. It is very usual (if not irregular) to have there variability of pulps being greater than the coarse crush duplicates.

14.1.7 Crushed Duplicates – Resubmission

Given that the crushed duplicate samples were not blind to the laboratory and were performing better than the pulp duplicates, there was a suspicion that there was laboratory influence on these results. Two independent tests were conducted in 2004 and 2006 to determine if any of the samples were handled in a special manner. The first test involved sending 100 coarse rejects to ITS for reanalysis after they had been retrieved from storage and renumbered. Unfortunately, this test failed to resolve the issue because only samples containing > 0.6 ppm (600 ppb) Au were included in the batch. This meant that there was a bias against low grade samples where variation between samples and errors are typically higher. These samples did not show any bias, but 90% of these coarse duplicates had an absolute relative difference of 8.8% for Au and 2.4% for Cu, lower than the ARD values obtained from the original coarse duplicate data set.

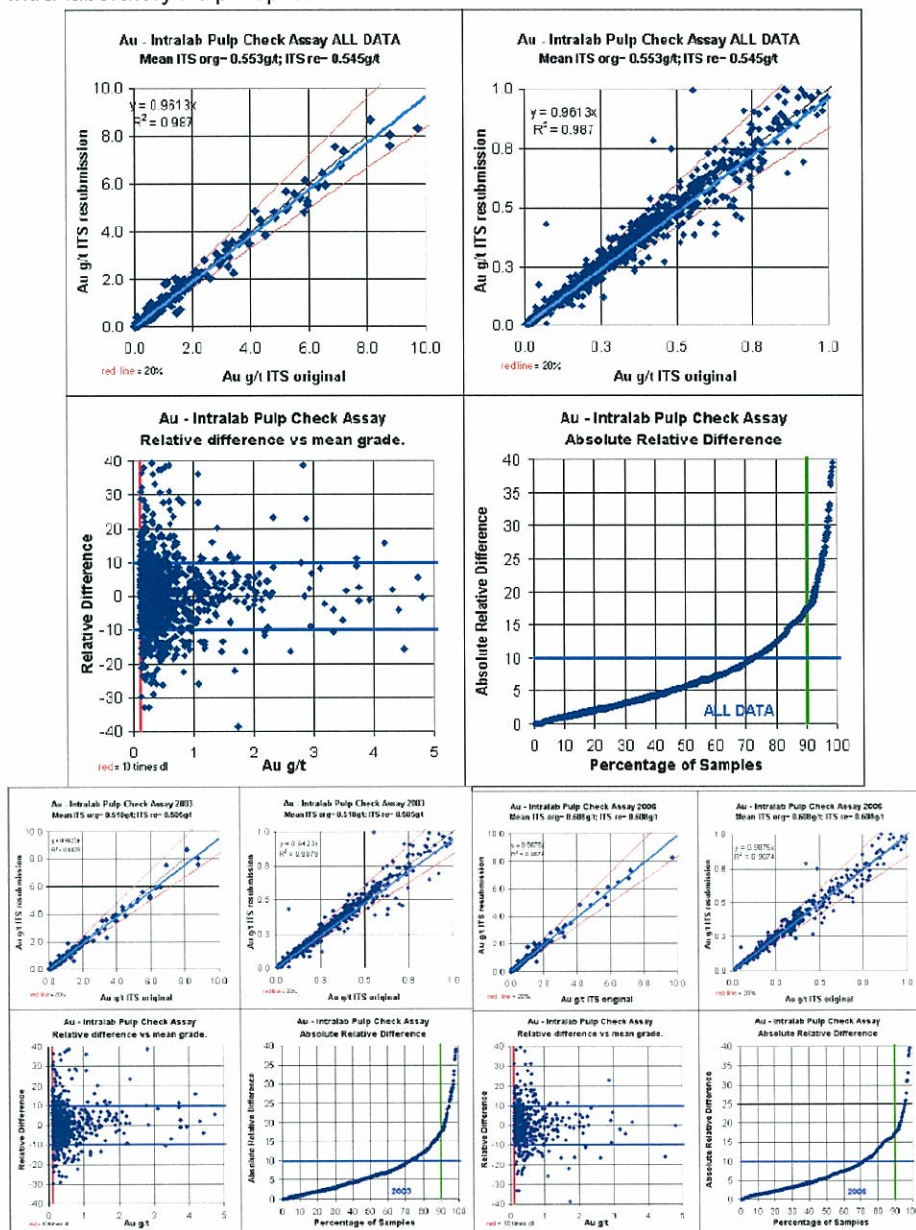
In 2006 another test was conducted after lengthy discussions with the lab regarding the issue of perfectly performing coarse duplicates. This time Anglo American resubmitted 100 coarse rejects that had a Au range that better represented the Au concentration in the deposit. This time the blind coarse duplicates had higher ARD values at the 90% of population cut off than other original coarse duplicates somewhat contradicting assurances from the lab that no extra attention is paid to these duplicates. Unlike the other two batches, these samples did show a slight low bias, and in this test, 90% of the coarse duplicates had an absolute relative difference of 29.4% for Au and 6.6% for Cu, higher than the ARD values obtained from the original coarse duplicate data set.

14.1.8 Pulp Duplicates - Intra Laboratory (ITS vs. ITS)

Ten percent of pulp splits are resubmitted on a periodic basis (usually monthly) to the principal laboratory. The data for copper shows that there is no bias and that the variability is well within expected limits (90% of the duplicates having a relative difference of less than 10%).

The comparison for gold for the ITS pulp duplicates is given in [Figure 12](#) which shows that there is strong correlation and no bias. The relative difference plot shows that 90% of the data have a relative difference of less than 20%, which is higher than the target (10%) but, as the element is gold, this is probably unavoidable.

Figure 12.0 Intra-laboratory Pulp Duplicates Au - Scatter Plots and Relative Difference Plots



14.1.9 Pulp Duplicates - Inter Laboratory (ITS Vs. McPhar)

Ten percent of the 250 gm pulp splits are submitted on a periodic basis (usually monthly) to the check laboratory. Including special check batches, a total of approximately 18 percent of the samples have been sent to the McPhar Laboratories in Manila as the check laboratory, where they are assayed for Cu and Au. The comparisons for copper show that:

- The scatter plot suggests that there is no or very little bias of ITS relative to McPhar.
- The relative difference plot shows that 90% of data has a relative difference of 9.2%, which satisfies the target of 10%.

- The comparison between the relative difference and grade indicates that at the lower grades ($<1\%$ Cu), McPhar appears to over report Cu values relative to ITS. The difference between McPhar and ITS tends to be greatest at lower Cu grade ($<0.4\%$ (4000ppm) Cu). This is probably due to carry over from the pre 2003 standards results, when ITS was closer to the standard values than McPhar. This difference is shown in the bottom two graphs.

The comparison for gold between ITS and McPhar pulp duplicates was made and the main observations can be summarized as follows:

- Though there is a strong correlation between the two data sets ($R^2 = 0.984$), McPhar is reporting slightly more gold in the higher-grade samples ($>3\text{g/t}$).
- Interestingly, the comparison between relative difference and grade shows that at lower grades ($<0.5\text{g/t}$) ITS is reporting slightly more gold than McPhar. In the past, the laboratory performance of standards indicated that ITS was the better performing lab, but recent results place both labs on equal terms.
- This bias has an impact on the relative difference plot which indicates that 90% of the data has a relative difference of 29%, which is considerably higher than the target of between 10% and 20%.

There has been some improvement in percentage relative difference between the laboratories following the procedural change in sample preparation, however, these investigations and improvements are still ongoing.

14.2 Density Measurements

Density measurements ($n=2339$) have been taken every 20m down the hole on each of the drillholes in the deposit resulting in a comprehensive coverage of density measurements. Density is measured by water immersion after wrapping in plastic. The Boyongan deposit is characterized by relatively low rock density with densities in the sulphide zone averaging slightly more than 2.3t/m^3 while in some parts of the oxide zone the rock density can be less than 2.0 t/m^3 . In the overlying Quaternary formations, the densities in the sediments and volcanics average approximately 1.95t/m^3 and 2.14 t/m^3 , respectively.

15 Adjacent Properties

The Surigao del Norte region is an area containing several historical and operating gold mines. The most significant of these perhaps is Manila Mining Corporation ("MMC"), located northeast of Boyongan, which stopped operation in 2001. MMC used to operate a 3,000tpd open-pit gold mine for heap leaching and a 3,000tpd copper flotation plant. MMC is drill testing an area northwest of Boyongan. Drilling activities however were suspended after completing hole MMC-14 which intersected a porphyry system at 340m depth under the quaternary cover. MMC is concentrating on obtaining community support to continue their drilling operation.

Other properties being evaluated nearby include;

- Greenstone Resources of Australia (Red 5) has embarked on an intensive level of exploration activity with targets at the historic Siana gold mine located 8km SE of Boyongan and other prospects in the vicinity of Siana. Greenstone's diamond drilling campaign aimed to upgrade and increase the resource and recover mineralization for metallurgical test work while their RC drilling is testing three IP prospects in the vicinity of Siana and the recent Crackerjack discovery. The company aimed to carry out a full feasibility study at the completion of drilling.
- Greenstone has initialized drilling at the Madya Cu-Au porphyry prospect on the southern portion of the Siana gold project to the SE of Boyongan.
- A new gold-base metal discovery by Indophil at the Manat project is currently the subject of mine-scoping studies.

The high level of exploration activity on tenements surrounding Boyongan is testament to the mineral potential of the Surigao Del Norte region.

16 Mineral Processing and Metallurgical Testing

16.1 Metallurgical Testing

Anglo Research (AR) was approached by Anglo Base Metals to conduct metallurgical test work on drill cores of the Boyongan ore body. The purpose of the proposed test work at AR was to derive two conceptual flow sheets: one flowsheet for the treatment of the oxide constituent of the Boyongan ore body, and another flowsheet for the sulphide portion of the ore body.

Anglo Base Metals chose the Boyongan samples on which the flowsheet development work had to be conducted (these samples were referred to as “trial” samples), as well as the variability samples that were tested after the flowsheet development work had been completed (these samples were referred to as “variability” samples). During the test work program at AR, Anglo Base Metals subsequently requested that the test work on the sulphide portion of the ore body be terminated. Thus, a conceptual flowsheet was developed to treat the oxide portion of the ore body only - with the focus on maximizing copper and gold recovery.

Previous test work conducted at Anglo Research, as well as the test work program received from Anglo Base Metals in the project charter, was used as the starting point in the development of the oxide flowsheet. This report covers all metallurgical test work conducted during 2007 to derive the conceptual oxide flowsheet for the Boyongan material.

The metallurgical test work was divided into the following phases:

- Phase I:
Flowsheet development using oxide trial samples: Oxide samples taken from the eastern and western zones of the Boyongan deposit were blended and used for the metallurgical test work to derive the oxide flowsheet. The upgrading of the oxide ore through flotation proved unsuccessful, with 53% of the copper reporting to the flotation tailings for the eastern oxide material, while 67% of the copper reported to the flotation tailings for the western oxide material when performing a sulphide and oxide float. Thus, the flowsheet selected centered on a whole ore leach for the oxide ore.
- Phase II:
Flowsheet validation: A separate blend of the eastern oxide trial samples, as well as a separate blend of the western oxide trial samples were prepared and processed through the derived flowsheet.
- Phase III:
Variability test work: Individual drill core samples taken from and around the Boyongan ore body were treated individually through the derived flowsheet to assess the impact of variability on the derived oxide flowsheet.

16.1.1 Mineralogy

The residence of copper is well-understood at Boyongan, based primarily on core logging, and petrographic studies. Table 9 provides the breakdown of copper phases at Boyongan as logged.

Table 9.0 Summary of Copper Phases Observed in Core Samples

Logged mineralogy for individual metallurgical samples												
Samp No	Mineral zone	Logged mineralogy (volume percent): Average values composited from interval logged										
		py	cpy	bn	cc	mal	az	chry	cup	CuN	neo	P lim
East Zone Trial Samples												
East Zone composite	Mixed	0.1	0.1	0.1	0.2	0.2	0.1	0.1	0.2			
West Zone Trial Samples												
West Zone composite	Oxide					0.2	0.2		0.4	0.1	0.1	0.1
Composited 'Oxide' Trial sample (E and W zone trial samples)												
	Mixed	0.1		0.1	0.2	0.1	0.1	0.3	0.1	0.1	0.1	0.1
Sulfide Trial Sample												
MET3-TES	Sulfide	0.2	1.2	0.6								
East Zone Variability Samples												
MET3-VE1	Oxide					0.7	0.4					
MET3-VE2	Mixed	0.2	1.0	0.7	0.1				0.1	0.1		
MET3-VE3	Oxide							0.1	0.8			
MET3-VE4	Mixed	2.3	1.6	1.1	0.8							
MET3-VE5	Mixed	0.2	0.3	0.1	0.2				0.3	0.2		
MET3-VE6	Oxide					0.7	0.1	0.4				
MET3-VE7	Oxide								0.1			
MET3-VE8	Sulfide	0.9	1.0	0.2								
MET3-VE9	Sulfide	1.5	0.4	0.2								
West Zone Variability Samples												
MET3-VW1	Oxide				0.1	0.2	0.1					0.1
MET3-VW2	Oxide				0.1	0.1	0.2		0.1			0.2
MET3-VW3	Oxide				0.5	0.6			1.0			0.3
MET3-VW4	Oxide								0.4			0.1
MET3-VW5	Oxide						0.5	0.2	0.3		0.3	
MET3-VW6	Oxide							0.2	1.5			0.1
MET3-VW7	Oxide				0.2	0.2						0.4
MET3-VW8	Oxide				0.1	0.6	1.0					
MET3-VW9	Oxide				0.2	0.7			0.2			0.3
MET3-VW10	Oxide						0.1		1.4	0.1		0.6
MET3-VW11	Oxide				0.1	0.1			0.5	0.1		0.2
Composite values for individual metallurgical samples derived by weighting by estimated sample mass (Incorporating core diameter and estimated density), and excluding check assay intervals.												
Abbreviations: az = azurite, bn = bornite, cc = chalcocite, chry = chrysocolla, cpy = chalcocypite, CuN = native copper, cup = cuprite, mal = malachite, P lim = pitch limonite, py = pyrite.												

As an independent check, the copper mineral deportment was determined on a sub-sample from the East / West composite and sulphide composite flow sheet development samples. These determinations were made using a QEMscan, and the results are summarized in Table 10.

Table 10.0 Mineralogical Deportment of Copper in Oxide and Sulphide

Mineral	Oxides		Sulphides	
	Grade (%)	Dist (%)	Grade (%)	Dist (%)
Chalcopyrite (CuFeS ₂)	0.11	15	0.24	27
Chalcocite (Cu ₂ S)	0.19	26	0.11	13
Covellite (CuS)	0.01	1	0.01	1
Bornite (Cu ₅ FeS ₄)	0.20	26	0.41	48
Total Sulphide Minerals	0.51	68	0.78	90
Copper oxides/hydroxides	0.14	19	0.07	8

Mineral	Oxides		Sulphides	
	Grade (%)	Dist (%)	Grade (%)	Dist (%)
Pseudomalachite	0.01	2	-	-
Native Copper (Cu)	0.03	4	-	-
Cuprite (Cu ₂ O)	0.04	6	0.01	1
Copper Biotite	0.01	1	0.01	1
Total Oxide Minerals	0.23	32	0.08	10
Total Copper Minerals	0.74	100	0.87	100

The proportion (68%) of sulphide minerals in the oxide sample is higher than predicted by the head grade assays. According to the head grade assay the proportion of sulphide mineral should have been 30 to 40%. One possible explanation is that the QEMScan instrument can easily identify the sulphide minerals, but is limited in its ability to identify oxide copper phases, and therefore the results could be biased towards the sulphide phases. The results, however, do confirm the presence of copper sulphide phases in the oxide sample.

As expected 90% of the copper in the sulphide sample occurs in copper sulphide phases.

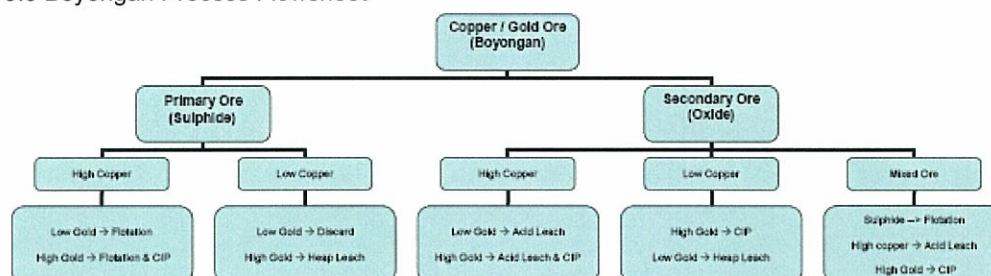
16.1.2 Test Work Methodology

All the metallurgical and mineralogy test work was contracted to Anglo Research (AR) in Johannesburg. Details of the test work program and methodologies are discussed in Appendix 3 – Section 2.3 to 2.13.

16.1.2.1 Flowsheet Selection

The Boyongan ore has numerous metallurgical challenges and makes for complex flow sheet selection. Several different flow sheet options exist depending on the factors such as degree of oxidation and relative proportions of copper and gold. The more typical flow sheet options for a copper / gold ore are illustrated Figure 13.

Figure 13.0 Boyongan Process Flowsheet



The majority of the ore at Boyongan is oxidized, with a large proportion categorized as mixed ore, and contains similar grades of copper and gold (expressed as equivalent copper). The implication of this is that both the copper and gold have to be recovered to realize the contained value. In addition the east oxide zone contains appreciable amounts of secondary sulphide minerals. For these reasons any proposed flow sheet for Boyongan will have to include sulphide floatation, acid leaching and cyanidation.

Factors that were considered when selecting the flowsheet were:

- Presence of course grained “free” gold which would require gravity concentration,
- Presence of secondary sulphide minerals (specifically in the east oxide zone) necessitating sulphide floatation,
- High proportion of acid soluble copper amenable to acid leaching,

- Possibility of recovery of oxide copper by sulphidizing oxide flotation,
- High proportion of gold not reporting to either the gravity of sulphide flotation concentrates, but recoverable by cyanidation.

The metallurgical test work program was designed to evaluate each of the above unit operations and determine to optimum flow sheet configuration. The test work program would also provide the parameters necessary to design and cost a full scale plant. The option of a twin stream process was considered whereby the oxides and sulphides would be processed separately. Plants that have twin stream processes typically mine ore from two different sources, either underground or open pit. This was not practical due to the small sulphide resource and that this ore would only be mined towards the end of the project.

Several different copper plants in Zambia were visited prior to beginning the test work. These were:

- Kanshanshi in Solwezi
- Bwana Mkubwa in Ndola

In addition to these plant visits, the flow sheets from several other copper / gold operations were studied; these were:

- Grasberg in Indonesia
- Red Dome in Australia
- Marté in Chile
- OK Tedi in Papua New Guinea

16.1.2.2 Ore Characterization

The ore characterization tests included:

- Crusher work index (CWI)
- Ball mill Bond Work Index (BWI)
- Drop weight and abrasion tests
- Gold diagnostic leach

16.1.3 Crusher Work Index

15 rocks were hand picked from each sample type for the Bond Crusher Work Index ("CWI") tests; Table 11 below summarizes the results.

Table 11.0 Crusher Work Index Results

Sample Type	CWI (kWh/t)	Specific Gravity (g/cc)
East / West Composite	5.4 ± 1.2	2.22
East Variability	5.8 ± 1.5	2.39
West Variability	4.3 ± 0.8	2.32
Sulphide Composite	8.1 ± 0.7	2.68
Sulphide Variability	11.2 ± 1.2	2.51

Generally the samples would all be characterized as relatively soft, although there are slight differences between the different samples. As expected the sulphide samples are harder than the oxides. It is interesting to note that on average the sulphide variability samples are harder than the trial samples. This indicates some degree of hardness variability within the sulphide ore body, although this variability will not affect the performance of the crusher. The oxide samples showed less variability.

16.1.4 Bond Work Index

The BWI of the individual East / West oxide samples were determined as shown in Table 12.

Table 12.0 Bond Work Index Results

Sample Type	BWI (kWh/t)
East Oxide	5.68 ± 2.16
West Oxide	7.43 ± 1.78

Again the samples can be characterized as relatively soft and no problems are anticipated in the milling circuit. The results, however, do show that there is some degree of variability both within and between the two oxide zones. The range of variability will not present a problem in the design of the milling circuit as the power consumption will be calculated on the 95th percentile.

Accumulation of critical sized material within the mill is an area of concern when designing comminution circuits. A potential accumulation is normally indicated when the Crusher Work Index (CWI) exceeds the Bond Work Index (BWI); this is not the case for Boyongan and hence no critical size build-up is anticipated.

16.1.5 Drop-Weight and Abrasion Results

Samples from the oxide and sulphide flow sheet development samples were subjected to drop-weight and abrasion tests. The results are shown in Table 13.

Table 13.0 Drop Weight and Abrasion Test Results

Sample	DWi	A	B	A*b	SG	T _a
East Oxide	1.65	72.09	2.22	160	2.51	2.36
West Oxide	1.87	84.13	2.12	178	2.51	1.56
Sulphide	4.69	63.88	0.83	53	2.55	0.79

The A*b parameter measures an ores impact hardness while T_a measures the abrasion resistance of the ore. Ores can typically be classified from very hard to very soft on the basis of their A*b and T_a parameters as illustrated in Table 14.

Table 14.0 Impact Hardness Summary Results

Ore Type	A*b	T _a
Very Hard	< 30	< 0.24
Hard	30 – 38	0.24 – 0.35
Moderately Hard	38 – 43	0.35 – 0.41
Medium	43 – 56	0.41 – 0.54
Moderately Soft	56 – 67	0.54 – 0.65
Soft	67 – 127	0.65 – 1.38
Very Soft	> 127	> 1.38

The results indicate that the oxide sample is very soft with low abrasion characteristics. The sulphide sample, however, has a medium hardness hard and exhibits a soft abrasion coefficient.

16.1.6 Gold Diagnostic Leach Results

A standard AR gold diagnostic leach was performed on the East / West oxide composite. The gold association is shown in Table 15 below.

Table 15.0 Diagnostic Gold Leach Results

Stage	Gold Association	Proportion
Direct NaCN	Cyanide leachable gold	97.6
H ₂ SO ₄ /NaCN	Labile sulphides	0.9
HNO ₃ /NaCN	Refractory sulphides	1.3
Residue	Locked in silicates	0.2

The vast majority (98%) of the gold is liberated and amenable to direct cyanidation. As expected the cyanide consumption was exceedingly high (10 kg/t) due to the presence of cyanide soluble copper mineral phases. This phenomenon is discussed in more detail in the cyanidation test work discussion.

16.1.7 Oxide Trial Samples

The initial open pit design suggested that both the East and West oxides zones would be mined simultaneously, and hence it was decided to conduct the flow sheet development test work on a blend of the East and West oxide zones. Once the flow sheet had been defined and optimized its applicability was tested to discrete composite samples from the East and West.

16.1.7.1 East / West Composite

16.1.7.1.1 Gravity Test Work

Prior to the gravity recovery tests, the crushed ore was milled to 80% passing 150 microns to simulate a typical cyclone underflow granulometry. The results from three initial scouting tests indicated significant recovery variability due to the presence of course gold. The only way to mitigate against the effect of coarse gold was to:

- Process large amounts of sample through the Knelson concentrator. For each test twenty 1 kg samples were treated. The individual concentrate samples from each of the twenty tests were hand panned to reduce the mass pull to 1%. The individual cleaned concentrates were combined together and hand panned again to produce a re-cleaner concentrate with a mass pull of less than 0.1% of the total feed,
- Total analysis of all high grade (>20 g/t) gold products by analyzing the entire sample and not a sub-sample as is conventional practice.

Both interventions resulted in increased cost and duration for the gravity test work, but provided a reliable measure of the gravity circuit recovery and gold head grade. The results from the gravity recovery tests are shown in Table 16.

Table 16.0 Gravity Recoveries - Summary

Parameter	Gold		Copper	
	Grade (g/t)	Recovery (%)	Grade (g/t)	Recovery (%)
Calculated head	1.52	100	0.77	100
Gravity concentrate	27.70	15	14.10	15
Gravity tails	1.33	85	0.64	85

The results show that 15% of both the gold and copper was recovery to the gravity concentrate. The gold and copper grades were 28g/t and 14% respectively. There are several processing options for the gravity concentrate:

- Intensive cyanidation
- Smelting of the concentrate
- Blending with the sulphide flotation concentrate

Of the three options above, the only viable option for Boyongan is the latter as this is the only scenario whereby both the gold and copper are recovered. This is important because of the high (15%) copper deportment to the gravity concentrate.

16.1.7.1.2 Flotation Test Work (Sulphide Flotation)

The Knelson tails from the twenty individual tests were combined together and milled to 80% passing 75 microns. The flotation recipe was based on the results of the test work performed on the same material at AR in 2001. A specific gold collector (AP-25) was included in the reagent regime to maximize the gold recovery. The results are summarized in Table 17.

Table 17.0 Flotation Recoveries - Summary

Parameter	Gold		Copper	
	Grade (g/t)	Recovery (%)	Grade (g/t)	Recovery (%)
Calculated head	1.52	100	0.77	100
Gravity concentrate	27.70	15	14.10	15
Flotation concentrate	28.90	26	7.02	12
Combined concentrate	28.50	41	9.71	28
Flotation tails	0.92	58	0.58	73

The combined gravity / flotation circuit will recovery 41% of the gold and 28% of the copper. The copper recovery, expressed as a percentage of the circuit feed, is expected to be low because a large proportion of the copper in the feed is acid soluble copper. To evaluate the efficiency of the flotation circuit it is necessary to express the recovery of both acid soluble copper and non-acid soluble copper. The calculated recoveries are:

- Total copper → 28%
- Acid soluble copper → 6%
- Non-acid soluble copper → 72%

As expected the acid soluble copper recovery is low, whilst the non-acid soluble copper recovery is high. Samples of the rougher concentrate were collected for mineralogical evaluation to investigate the copper phases in the concentrate and determine the gold association to better understand the mechanism of gold recovery to the concentrate. These results are discussed later in the report.

16.1.7.1.3 Flotation Test Work (Oxide Flotation)

The possibility of using a sulphidizing reagent (NaHS) to float the copper oxide minerals was considered as an alternative to sulphuric acid leaching. A sample of the gravity recovery tails was first floated to recovery the sulphide minerals, after which NaHS was added and the oxide minerals floated. The recipe used was based on the 2001 test work at AR. The results are summarized below in Table 18.

Table 18.0 Oxide Flotation Recoveries

Parameter	Gold		Copper	
	Grade (g/t)	Recovery (%)	Grade (g/t)	Recovery (%)
Calculated head	1.52	100	0.77	100
Gravity concentrate	27.70	15	14.10	15
Sulphide concentrate	28.90	26	7.02	12
Oxide concentrate	5.74	19	2.95	4
Combined concentrate	12.6	60	4.99	31
Flotation tails	0.62	40	0.48	69

The oxide flotation stage was shown to increase gold recovery by an additional 19% and increase copper recovery by 4%. Based on these results oxide flotation is not a substitute for sulphuric acid leaching, but it could be a possible mechanism for increasing gold recovery if the cyanidation circuit proves to be uneconomical. A concern with the oxide flotation circuit is the high mass pull (5.3%), low rougher grades and dilution effect on the combined concentrate.

16.1.7.2 Acid Leach

The sulphide flotation tails was filtered, dried and split into 500g sub-samples for the sulphuric acid leach tests. The optimization of the sulphuric acid leach conditions focused on the effect of temperature and residual free acid. All tests were performed for 6 hours at a solids concentration of 35% (m/m). Four temperatures (25, 40, 60 and 85°C) and four terminal free acid concentrations (1, 5, 10 & 15g/l) were evaluated. The results are summarized in Table 19.

Table 19.0 Free Acid Consumption - Summary

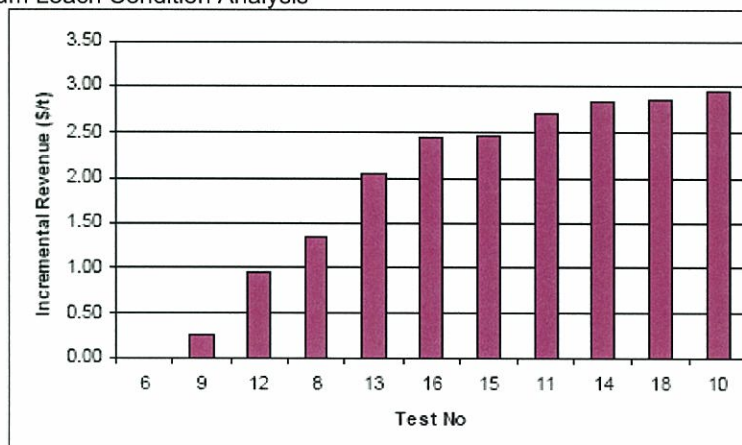
Test	Free Acid (g/l)	Temp. (°C)	Cu Extraction (%)		Acid Consum. (kg/t)	Iron Dissol. (ppm)
			TCu	A/S Cu		
6	1	25	51	67	12.20	439
8	12	25	62	83	28.60	756
13	4	40	79	90	34.00	1045
10	5	40	78	93	26.20	1115
14	5	40	78	96	35.50	1154
18	9	40	79	95	32.60	1289
15	11	40	77	96	41.70	1399
16	11	40	79	96	42.20	1554
11	11	60	81	98	42.60	1965
12	12	80	86	97	69.40	3278
9	13	80	85	98	83.40	3444

The objectives of the optimization test work were to:

- Maximize acid soluble copper extraction
- Minimize acid consumption
- Minimize iron dissolution for subsequent down stream processing

A mathematical model was used to determine the optimum leach conditions. The mathematical model used Test 6 as a basis and evaluated the net effect of increased copper revenue and increased acid consumption cost. The output of the model is illustrated below in Figure 14.

Figure 14.0 Optimum Leach Condition Analysis



The graph clearly illustrates that the optimized leach conditions are:

- Temperature → 40°C
- Terminal free acid concentration → 5 g/l

At these conditions an acid soluble copper extraction of 96% and iron dissolution of approximately 1,100ppm can be expected at a corresponding acid consumption of 29kg/t. Typical residue values will be 0.15% total copper and 0.02% acid soluble copper. No gold was leached in the process.

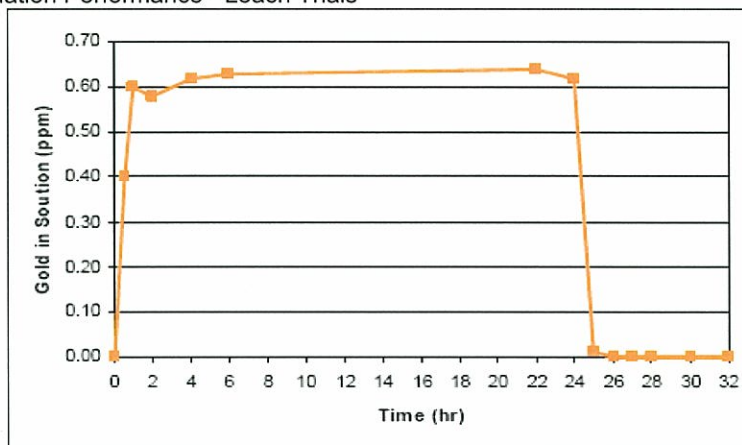
16.1.7.2.1 Gold Recovery by Cyanidation

The cyanidation tests were performed on the residue from the acid leach tests performed under the optimized conditions of 40°C and 5 g/l terminal acid. The leach conditions for the cyanidation tests were:

- Temperature → ambient (25°C)
- Solids concentration → 50% (m/m)
- Residence time → 24 hrs
- pH → 11
- Dissolved oxygen → air sparging
- Residual CN- tenor → 300ppm

After the 24 hour leach period activated carbon was added to the leach reactor for a further 8 hours to adsorb the gold. Previous experience with gold cyanidation in the presence of cyanide soluble copper showed that high pH values (>11) are necessary to restrict the amount of copper that adsorbs onto the carbon.

Figure 15.0 Cyanidation Performance - Leach Trials



The following points are noted from the kinetic profile given in Figure 15:

- Total dissolution of the gold occurs after 8 hours,
- The adsorption of gold onto the activated carbon is complete after 1 hour.

The gold assay on the cyanidation residue was below the detection limit of the fire assay technique (0.05 g/t) indicating total dissolution of the gold in the feed sample. The cyanide consumption was measured at 1.09 kg/t. This is due to the presence of cyanide soluble copper species in the feed to the gold circuit. Table 20 indicates the solubility of the different copper mineral in sodium cyanide at room temperature.

Table 20.0 Cyanide Solubility of Selected Copper Minerals

Mineral	Solubility in NaCN at 23°C
Chalcopyrite (CuFeS_2)	5.60
Chalcocite (Cu_2S)	90.20
Bornite (Cu_5FeS_4)	70.00
Azurite ($2\text{CuCO}_3 \cdot \text{Cu}(\text{OH})_2$)	94.50
Malachite ($\text{CuCO}_3 \cdot \text{Cu}(\text{OH})_2$)	90.20
Native Copper (Cu)	90.00
Cuprite (Cu_2O)	85.50

Both sulphide and oxide copper mineral are readily soluble in NaCN under typical gold leaching conditions and each mole of dissolved copper will complex with three moles of cyanide. By way of illustration, 0.02% soluble copper will consume 500g/t of sodium cyanide.

16.2 Mineral Processing

Three different campaigns of metallurgical test work were completed at Anglo American's research laboratories 'Anglo Research' (AR) in Johannesburg, South Africa. The first two campaigns were carried out in 2002 and 2003 and were discussed in the report on the mineral resource estimates published in November 2003. The last campaign was carried out in 2006 and 2007 and it is largely the results of this latest test work that are used to develop a flow sheet and estimate metal recoveries and plant costs for the PFS.

For the 2006-7 metallurgical test work one "trial" bulk sample was composited from various parts of the oxide portion of each of the East and West Zones as well as one sulphide trial

sample were provided to AR for developing a flow sheet that would be appropriate for the whole deposit. Once established, possible variations in metallurgical performance across the deposit were tested by running smaller "variability" samples through the proposed flowsheet.

Sixty eight percent of the samples submitted for metallurgical test work were from the oxide zone of the deposit. This proportion is considered to reflect the relative volumes of material in the deposit likely to be mined from an open-pit. The copper and gold contained in the ore have approximately the same value in terms of equivalent copper, suggesting that equal emphasis should be placed on recovery of both metals to realize full contained value.

The following factors influenced the flow sheet selection:

- Presence of coarse grained "free" gold which would require gravity concentration,
- Presence of secondary sulphide minerals in the oxide zone (particularly in the east) necessitating sulphide flotation,
- High proportion of acid soluble copper amenable to acid leaching,
- Possibility of recovery of oxide copper by sulphidizing oxide flotation,
- High proportion of gold not reporting to either the gravity of sulphide flotation concentrates, but recoverable by cyanidation,

To achieve this, the proposed flow sheet includes gravity separation, sulphide flotation, acid leaching and cyanidation.

16.2.1 Process Design Criteria

The flow sheet was designed based on the following parameters:

- Throughput of 5 million tonnes per annum
- 7 days per week operation
- 345 days per year operation
- 12 hour shifts
- Plant availability 93%

The 5 million tonnes per annum throughput was considered to be reasonably optimum in trying to balance life of mine, CAPEX and optimizing cash flow the best NPV.

16.2.2 Final Optimized Flowsheet

The optimized oxide flow sheet developed by Anglo consists of the following unit operations:

- A comminution plant consisting of ore receipt, crushing and milling,
- Gravity gold separation,
- Sulphide/gold flotation to recover sulphide copper and gold,
- Non-acid dewatering consisting of thickening and filtration ahead of the acid leach,
- Sulphuric acid leaching of the acid soluble copper in the flotation residue,
- Acid dewatering consisting of thickening and filtration (including washing) for the recovery of copper bearing solutions (thickener overflow (PLS), primary filtrate (recycled to acid leach) and secondary filtrate (copper recovery from the filtrate),
- A copper recovery circuit for secondary acid filter filtrate,
- Iron removal ahead of solvent extraction using limestone,
- Neutralization of acid tailings prior to gold extraction/tailings disposal,
- Cyanidation of gold bearing acid leach tailings,
- Resin in pulp for the recovery of copper from the CIP feed stream,
- CIP plant for gold recovery,
- Gold elution and electro-winning,
- Tailings thickening with the inclusion of cyanide destruction,

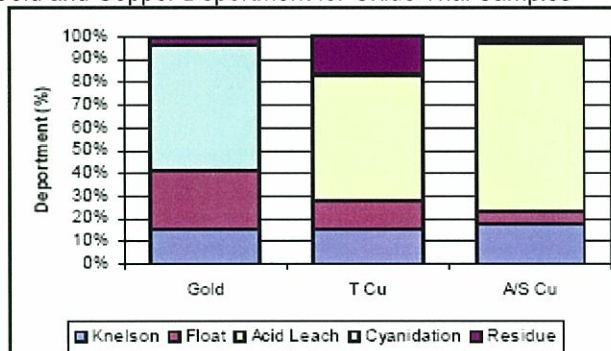
- Reagent make-up for reagents required in process,
- Water handling circuits.

The optimized process flow diagram is shown in Figure 19.

16.2.2.1 Overall Gold and Copper Department

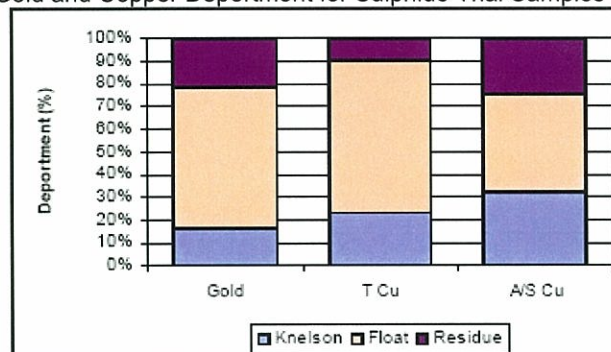
The department of gold and copper in the circuit based on laboratory test results, without taking into account process inefficiencies, is shown graphically in Figure 16.

Figure 16.0 Overall Gold and Copper Department for Oxide Trial Samples



The total gold extraction in the oxide trial samples is 96% with 41% reporting to the combined gravity / flotation concentrate and 55% extracted in the cyanidation circuit. The copper recovery is 83% with 28% reporting to the combined gravity / flotation concentrate and 55% extracted during the sulphuric acid leach. The total recovery of acid soluble copper in the circuit is 97%. The final residue (cyanidation tails) contains 0.13% copper of which only 7% is acid soluble.

Figure 17.0 Overall Gold and Copper Department for Sulphide Trial Samples



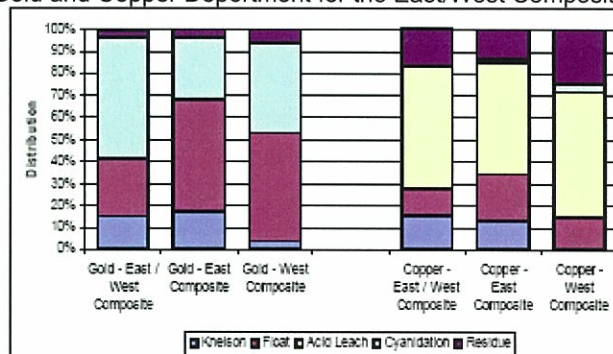
The results for the sulphide trail samples indicate that the combined concentrate (gravity and flotation) will recover 78% of the gold and 91% of the total copper (Figure 17). The rougher concentrate grade was 11.5% at a mass pull of 5.3%. No cleaner tests were performed on the rougher concentrate, but instead a flotation simulation package was used to determine the number of cleaning stages that would be required to achieve the required rougher concentrate grade of 24%.

16.2.2.2 Confirmation of Flowsheet

The optimized flow sheet conditions, as determined for the East / West composite, were applied individually to the East and West composites to measure the applicability of the conditions to the two areas of the deposit and to highlight possible process issues that

could arise if the areas were treated separately. Overall gold and copper deportment for the composite samples are shown in Figure 18.

Figure 18.0 Overall Gold and Copper Deportment for the East/West Composites



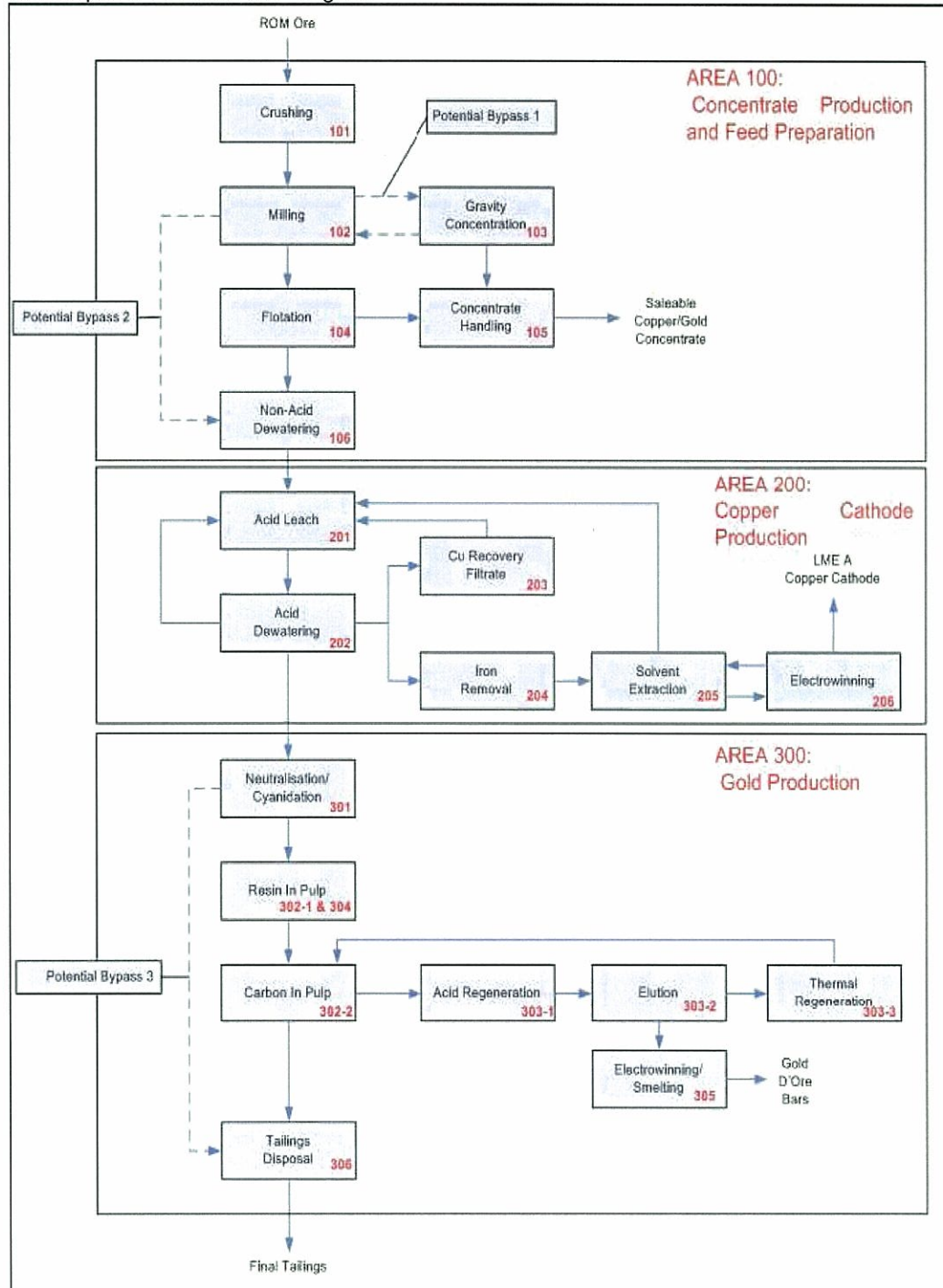
The following can be noted from the graph:

- The overall gold recovery is in excess of 95% for all three samples (East/West composite and the separate East and West composites). The proportion of gold recovered in the gravity and flotation operations is variable, and ranged from 40 to 70%. There are indications that some free gold is recovered in the sulphide float,
- The cyanidation circuit recovers the balance of the gold not recovered by the gravity / flotation circuit, albeit at high cyanide consumptions,
- The copper recovery in the gravity / flotation circuits was also variable and ranged from 15 to 35%. The performance of these circuits was largely determined by the proportion of non-acid soluble copper in the feed,
- The sulphuric acid leach efficiencies were generally high (>95%), with the exception of the West composite where the higher proportion of native copper and cuprite had a negative effect on leach efficiency. The bulk of the copper (50% to 57%) was recovered in this circuit,
- Very small amounts of copper (0.02%) were leached in the cyanide circuit and negatively affected the cyanide consumption. Due to the relative concentrations between gold and copper ($Au \ll Cu$) the high cyanide consumption will always be a reality,
- The bulk of the copper in the final residue (>90%) is non-acid soluble copper,
- Generally the optimum circuit configuration and conditions East / West composite are applicable to the East and West composites.

A number of alternative processing options were considered but were not taken further as they are not considered viable at present. The options include chloride, bromide and ammonia leaching.

The total operating cost for the base case is high due to the high cyanide consumption and cost of reagents. The cost for the base case and various processing scenarios is provided under the financial modeling section below

Figure 19.0 Simplified Process Flow Diagram



17 Mineral Resource and Mineral Reserve Estimates

17.1 Mineral Resource Estimate

17.1.1 Introduction

In early 2003, Anglo American plc contacted AMEC Americas Limited (AMEC) (formerly AMEC Engineering & Construction Services Ltd.) and requested a review of geological data and resource model for the Boyongan Deposit. The review was made by Dr. Harry Parker during 28 May to 2 June 2003. Project staff from site met Dr. Parker in Anglo American's offices located near Perth, Western Australia. Anglo American's team included Duncan Campbell of the UK; Patrick Waters, Exploration Manager for the Philippines; Max Estaris and Manny Corpuz, project geologists; Geoff Gushee, resource modeler from South Africa; and Dr. Nigel Brand, Principal Geochemist. From September 28 to 30 2003, Dr. Parker and Mr. Campbell met in Anglo American's London, UK offices to finalize and slightly update AMEC's report for work done by Anglo American during June to September 2003.

With the continued exploration drilling, an updated Mineral Resource was completed by Anglo in late 2007. This resource includes all the data up to mid-May 2007.

As Anglo shifted its interest to potential mineralization beyond the Boyongan Project, Philex engaged the services of IRES to provide a new Mineral Resource using all the data available for the Boyongan deposit up to and including December 31st, 2007.

The following sections will outline the methodologies used by the most recent one completed by IRES⁴.

17.2 Zonation

The main objective of domaining (zonation) is to contain the influence of sampling data to areas which they are expected to represent and to limit the undue projection of low grades into higher grade areas and vice versa. Where there is a distinct change in grade between zones, and the boundary is reasonably well defined (or will be at the time of mining), then estimation within the zone can be restricted to samples within the zone (known as "hard" boundaries). Where the boundary is more gradational, using samples from both sides of the boundary can be considered ("soft" boundaries).

Aside from the domains (zones) described, other zones were defined including: the North West Hypogene Zone, the East Deep Zone, and the West Deep Zone. The recent Mineral Resource study did not consider these zones but focuses on the main Boyongan orebody itself. Thus the focus of this PFS Report relates directly to the main orebody only.

A surface was constructed in 2004 representing the interface between the mineralized porphyry and the sediments that overly the deposit. Each drill hole was tagged with a point for the sediments/porphyry transition, and a surface was built using these points.

Another surface representing the oxide/sulphide boundary at the last visible sign of oxide in the rocks was constructed in 2004. Anglo found no need to create a mixed zone, as the transition from oxide to sulphide material is characterized as sharp. Each drill hole was tagged with a point for the oxide/sulphide transition, and a surface was built using these points.

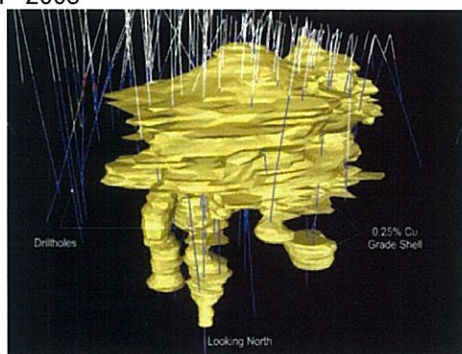
17.2.1 Zonation - 2008

Using the complete database (up to and including December 31st, 2007), the grade shells were re-visited by IRES. Beginning with the low grade (or outer) shells, the 0.25% Cu and 0.25g/t Au were investigated using IK. With the additional drillhole data, the low grade Cu

⁴ A complete report on the 2008 methodology employed in the estimation of the mineral resources at Boyongan can be found in Appendix I.

shell was reconstructed (Figure 20); whilst the 2007 low-grade Au shell from Anglo was maintained.

Figure 20.0 Low Grade Cu Shell - 2008



A statistical analysis completed in 2008 demonstrated that a high density of data currently exists within the high-grade areas at higher elevations – compared to previous studies. Subsequent statistics (see Section 17.2.2) reveal that the construction of constraining high-grade Cu and Au domains was not necessary since there is sufficient high and low grade data; thus, high grade zones were not utilized in the 2008 estimation.

17.2.2 Sample Spacing

The spacing between samples in three-dimensional space (i.e. Euclidean) was examined. Overall, the distances are typically 45m (between different drillholes). Table 21 shows the spacing between samples in three-dimensional space for composited surface drillholes.

Table 21.0 Euclidean Spacing

Data	Zone	Average Meters	Maximum Meters	Lower Quart. Meters	Median Meters	Upper Quart. Meters
Samples	All Data	75.72	1081.46	35.56	52.31	78.47
2.5m Comps	0.25%Cu Oxide	40.64	117.46	26.34	41.04	53.15
2.5m Comps	0.25%Cu Sulphide	55.04	188.66	34.51	45.66	65.54

The statistics reveal that the low-grade Cu and Au shells have a relatively high data density. Just below the paleo surface, within the oxide zone, the samples have a median spacing of 41m whilst below, in the sulphide region, the samples have a median spacing of 46m. For typical porphyry Cu deposits, the sample density is quite tight.

17.3 Composites

For the 2008 study, statistics were compiled on the sample lengths occurring within the low grade Cu and Au shells. They reveal that the mean sample length is about 2.46m long and thus IRES determined that a composite length of 2.5m was an appropriate length for compositing.

In this recent Mineral Resource model, samples were composited at 2.5m within the low grade Cu and Au shells only. This constraint pre-empted the potential mixing of zones around the constraining wireframes experienced by Anglo.

17.3.1.1 Validation of Compositing

The statistics of the composites were then compared to the statistics of the uncomposited datasets for drillhole samples. This was done in order to check whether data has been gained or lost. From the tables above, the amount of drillhole data has decreased by approximately 6.0% within the Cu shell and similarly for the Au shell.

The means of both the uncomposited datasets and the composited datasets were also compared in order to ensure that they are not significantly different. A comparison of the Cu, Au, As, and SG (Specific Gravity) means on a per shell basis shows that there are no significant changes in the means due to compositing; differences are less than 3%.

17.4 Grade Outlier Analysis

Cutting statistics were performed with the help of cumulative log probability plots, indicator correlation for lag 1 plots, coefficient of variation plots and finally percent metal contained plots. It should be noted that these are merely guidelines and that ultimately, the cutting limit chosen is a grade limit suggested by these plots.

The indicator correlation for lag 1 plots show the correlation between samples for the first lag set. Plotting this indicator against increasing minimum thresholds for Cu, Au, As, and SG values leads to a line tending closer towards zero. In other words, at ever increasing thresholds of Cu, Au, As, and SG values, there are fewer and fewer samples of similar grade. At this point, it indicates a lack of correlation between samples within the first lag set, and suggests an ideal cutting limit for assay values.

The coefficient of variation plots shows the change in this coefficient with increasing Cu, Au, As, and SG values. A rapid change in this coefficient indicates a rapid change in the standard deviation and/or a change in the mean. This suggests an ideal cutting limit for Cu, Au, As, and SG values.

Kinks, plateaus and/or changes in the cumulative log probability plots also suggest changes in populations (perhaps subpopulations) and serve as a good indicator of cutting limits for Cu, Au, As, and SG values. A slightly different plot is the percent of contained metal in samples versus increasing trimming levels for metal grades. This plot enables one to check how much metal is being lost to cutting at a certain Cu, Au, As, and SG values thresholds. The limits determined from these plots yielded a limit of 5.0% for Cu and 10.0g/t for Au; see Table 22.

Table 22.0 Cutting Limits per Metal

Zone	Cu	Au
0.25%Cu Shell	5%	-
0.25g/tAu Shell	-	10g/t

Table 23 lists the impact of the cutting thresholds suggested by all methodologies described. The reasons for choosing one methodology over another is based on how well defined that limit is depicted on the appropriate plot(s).

Table 23.0 Cutting Limits per Metal

Metal	Limit	# of Original Data	# of Comps cut	% of Data
Cu (%)	5	10851	16	0.15%
Au (g/t)	10	9592	61	0.64%

The overall effect of imposing cutting limits is less than 1% - an insignificant amount.

17.5 Variography

The models were estimated using data only contained within the defined limits. Pairwise relative variograms were used in this study; therefore no data transformation was necessary. There is some argument that this variogram is not a valid covariance model and should not be used for estimation, although in many practical applications this structural measure has been used as a variogram. Some consider that mixing all data together maximizes the variability of the data set and subsequently reduces the chances of deriving an interpretable variogram structure. This practice necessitates the use of non-linear transforms such as the pairwise relative variogram, in order to derive a measure of the

spatial variability of grades. Logically, segregating the deposit into stratigraphically distinct entities should reduce the variance within the data subsets allowing better and more appropriate local variograms to be developed for each of the entities modeled. This process is aimed at further reducing the risks inherent in the estimation process. The reader is referred to additional work done on this subject refuting this argument in support of pairwise relative variograms (*Robust Measures of Spatial Continuity*, Mssrs. M. Strivastava and H.M. Parker, *Geostatistics Volume I*, pp. 295 – 308), which concludes that heteroscedastic^Ω datasets in which samples have been preferentially located in areas with high values, cannot be characterized adequately with traditional sample variograms; relative variograms will produce more interpretable and reliable results.

In this study, variography was conducted on the dataset beginning at 0° and rotating clockwise in 20° increments. The first run was done with 0° dip and 0° plunge using a horizontal and vertical tolerance of ±12.5°. The lag size was varied from 50m to a maximum of 80 lags in any one direction was calculated.

An additional series of runs were done with a wider tolerance set at ±22.5°. Nugget contribution was taken from downhole variograms.

Directional variography revealed no obvious anisotropism in any direction. Omni directional variograms were then generated.

17.6 Interpolation

Ordinary kriging was selected as the final estimation method of interpolating Cu, Au, As, CUCXPCT, CUHXPCT, CUCXRAT and SG values into a three-dimensional blockmodel. The block size chosen was identical to that discretizing the geological model for both models, 20 x 20 x 10 meters (Northing x Easting x Elevation). Within the project area there were 130 blocks in the X direction, 120 blocks in the Y direction and 136 blocks in the Z direction, for a total of 2,121,600 blocks. The project area consists of an area from: 1061200N and 1063600N, 778200E to 780800E, and -800E to 550E.

A minimum of 3 and a maximum of 20 composites were utilized for an estimate. Rockcodes tagged in the estimation model came from the final geology model, and not from the tagging of the composite drillhole file itself. Ordinary kriging was performed with a discretization of 2 x 2 x 1 (X,Y,Z). The search radii used approximately equaled the variogram ranges in the plane of the deposit. Boundary conditions were set to soft across the oxide/sulphide boundary for Cu, Au, As and SG; but hard for CUCXPCT, CUHXPCT and CUCXRAT. Only composites within the 0.25% Cu shell could estimate blocks within this shell; with the exception of Au which used a different grade shell (0.25g/t Au Shell) to interpolate within it only. The search strategies utilized in the ordinary kriging runs are listed in Table 24 and Table 25; the radii shown relate to ellipse dimensions used.

Table 24.0 Initial Estimation Search Strategies

Metal	Principal Direction		Minor Direction		Vertical Direction	
	Radius Meters	Azimuth/Dip Degrees	Radius Meters	Azimuth/Dip Degrees	Radius Meters	Azimuth/Dip Degrees
Cu	75.0	0°/0°	75.0	90°/0°	75.0	90°/90°
Au	75.0	0°/0°	75.0	90°/0°	75.0	90°/90°
As	75.0	0°/0°	75.0	90°/0°	75.0	90°/90°
CUCXPCT	75.0	0°/0°	75.0	90°/0°	75.0	90°/90°
CUHXPCT	75.0	0°/0°	75.0	90°/0°	75.0	90°/90°
CUCXRAT	75.0	0°/0°	75.0	90°/0°	75.0	90°/90°
SG	75.0	0°/0°	75.0	90°/0°	75.0	90°/90°

^Ω **Heteroscedastic** is an adjective describing a data sample or data-generating process in which the errors are drawn from different distributions for different values of the independent variables. Most commonly heteroscedasticity takes the form of changes in variance with the magnitude of X. That is, in $y = Xb + e$ that the e's vary in magnitude with the X's. If the errors are drawn from different distributions, or if higher moments of the error distributions vary systematically, these are also forms of heteroscedasticity

Table 25.0 Secondary Estimation Search Strategies

Unit	Principal Direction		Minor Direction		Vertical Direction	
	Radius Meters	Azimuth/Dip Degrees	Radius Meters	Azimuth/Dip Degrees	Radius Meters	Azimuth/Dip Degrees
Cu	220.0	0°/0°	220.0	90°/0°	220.0	90°/90°
Au	250.0	0°/0°	250.0	90°/0°	250.0	90°/90°
As	250.0	0°/0°	250.0	90°/0°	250.0	90°/90°
CUCXPCT	250.0	0°/0°	250.0	90°/0°	250.0	90°/90°
CUHXPCT	250.0	0°/0°	250.0	90°/0°	250.0	90°/90°
CUCXRAT	180.0	0°/0°	180.0	90°/0°	180.0	90°/90°
SG	200.0	0°/0°	200.0	90°/0°	200.0	90°/90°

Tests were previously performed to investigate the effects of certain interpolation parameters on the variability of the estimates. Firstly, the maximum number of samples utilized for an estimate was examined. In this estimation model, the maximum is set to 20 composites, with an average of 18 composites used. A number of ordinary kriging runs with various maximum sample values were done, and the average variance of each run was compared to the maximum number of samples utilized. As the maximum number of samples is increased, the change in the variance decreased. The maximum number of samples is then selected from the area where a change in slope (becomes flatter) occurs, which is in this case, anything more than 20 samples. At this point, the addition of more samples does not significantly change the variance at all. Thus, a maximum of 20 samples was chosen in order to generate an estimate.

A final series of estimation runs were done, with an isotropic search ellipsoid size of 75m. This was done for all composites outside the low grade shells (0.25%Cu and 0.25g/t Au). Once complete, the estimates were coded according to estimation schemes applied for resource categorization (see Table 26).

Table 26.0 Project Rock Coding

Zone	Rock Code		
	Measured	Indicated	Inferred
Oxide	8	4	2
Sulphide	10	5	3

17.7 Validation

The cross validation tests performed here were done on the model. Numerous cross validation tests were performed on this block estimation study; one of them being naïve cross-validation. This technique consists of removing one sample and using the parameters to estimate it, and then comparing it to the original sample.

This was done systematically for all samples with a final correlation, comparing estimates to actual values, being reported. This allows for the testing of the parameters utilized in the estimation process, see Table 27 lists the results.

In the practioner's experience, the Cu, Au, As, CUCXPCT, CUHXPCT, CUCXRAT and SG values indicate an adequate amount of local variability and smoothing.

Table 27.0 Naïve Cross Validation Results

Metal	Number Of Data	Correlation Coefficient
Cu	10564	0.92
Au	7865	0.91
As	10436	0.88
CUCXPCT	10116	0.94
CUHXPCT	10124	0.94
CUCXRAT	10116	0.96
SG	1263	0.75

Another test was to take all the cut composites that actually intersect an estimated block, calculate a grade weighted by length (of intersection), and then compare these grades to the estimated block grades. This test is done in order to verify the amount of variability and/or smoothing of the estimate.

Obviously, samples occurring within a certain estimated block should imply a grade very similar to the estimated grade itself. This test was done for all the estimated blocks, and the result is presented in Table 28 below. As a guide, a correlation of 0.8 represents an adequate level of smoothing within the estimate.

Table 28.0 Simple Validation Results

Metal	Number Of Data	Correlation Coefficient
Cu	2816	0.95
Au	2482	0.94
As	2800	0.94
CUCXPCT	2725	0.96
CUHXPCT	2725	0.97
CUCXRAT	2725	0.98
SG	1210	0.96

A comparison between the average Cu, Au, As, CUCXPCT, CUHXPCT, CUCXRAT and SG values of the cut composites and the average estimated grades of the entire estimate was done. The values are shown below in Table 29 below. The comparison was done with polygonally-declustered drillhole cut composites (declustered due to their spatiality), in order to determine how well global block grades compare with declustered input data by domain.

Table 29.0 Actual and Estimated Means

Zone	Metal	Cut Composites	Estimates
Oxide	Cu (%)	0.58	0.57
Sulphide		0.45	0.47
Oxide	Au (g/t)	0.99	0.74
Sulphide		0.61	0.53
Oxide	As (ppm)	10.46	9.64
Sulphide		58.84	54.66
Oxide	CUCXPCT%	0.39	0.38
Sulphide		0.09	0.09
Oxide	CUHXPCT%	0.42	0.42
Sulphide		0.10	0.10
Oxide	CUCXRAT%	0.63	0.65
Sulphide		0.19	0.18
Oxide	SG	2.12	2.14
Sulphide		2.34	2.34

The results indicated that the resource is slightly conservative for all elements estimated with the exception of Cu grade estimates being slightly higher than the declustered composites within the sulphide zone. Therefore, the generation of higher block estimates is not in anomalous proportions, thus indicating the preclusion of overestimation.

17.7.1.1 Residuals

The amount of residual bias was also studied by determining the difference between the actual grade and the estimated grade; a test done via naïve cross-validation. These differences are then plotted on a frequency distribution plot and the mean established. In a perfectly unbiased estimate, the mean should be zero. Table 30 shows the amount of bias for Cu, Au, As, CUCXPCT, CUHXPCT, CUCXRAT and SG estimates.

Table 30.0 Residual Analysis: Entire Project

Metal	Number Of Data	Correlation Coefficient
Cu	10564	+0.0002
Au	7865	+0.0008
As	10436	+0.1085
CUCXPCT	10116	+0.0000
CUHXPCT	10124	+0.0000
CUCXRAT	10116	+0.0001
SG	1263	+0.0002

As can be seen from Table 30, there is very little in the way of bias. The estimates have very slight negative means overall.

17.7.1.2 Swath Plots

The trends indicate that the estimated blockmodel closely follows the trends as present in the composites – in all three directions; Northing, Easting and Elevation. Visual checks of the estimation methodologies revealed that ordinary kriging showed excellent grade variability.

17.8 Mineral Resource Classification

The resources estimated within the Boyongan area are classified as measured, indicated and inferred. The criteria utilized were as follows, block estimates estimated within the first variogram range of the double structure spherical models, at a 0.5% Cu cut-off, were classified as being within the measured category. This range of 75m approximately represents two thirds of the total variance. Resources estimated via the second structure of the double spherical model, at a 0.5% Cu cut-off, were classified as indicated. Resources estimated outside the Cu and Au grade shells (0.25%Cu and 0.25g/tAu respectively) were classified as inferred. No CuEq calculations were utilized within this recent study as it is discouraged by the National Instrument 43-101.

17.9 Mineral Resource Totals

Results were tabulated at a cut-off of 0.50% Cu and are presented below for Measured and Indicated Resources separately for the oxide and sulphide zones down to the -400m elevation mark; which corresponds to the broad bottom of the low-grade Cu shell (see Table 31). The Inferred resources have been tabulated separately⁵ and are shown in Table 32.

Table 31.0 Measured and Indicated Mineral Resources

Type	Category	Tonnes	Cu%	Au (g/t)	As (ppm)	ASCu (%)
Oxide	Measured	74,000,000	0.83	1.08	14.04	0.57
Oxide	Indicated	250,000	0.75	0.08	3.37	0.38
Sulphide	Measured	30,500,000	0.73	0.77	115.66	0.17
Sulphide	Indicated	620,000	0.82	0.08	87.31	0.17
Measured	Total	104,500,000	0.80	0.99	43.66	0.45
Indicated	Total	870,000	0.80	0.08	62.90	0.23
Total		105,400,000	0.80	0.98	43.82	0.45

Table 32.0 Inferred Mineral Resources

Type	Category	Tonnes	Cu%	Au (g/t)	As (ppm)	ASCu (%)
Oxide	Inferred	2,900,000	0.87	0.40	17.27	0.41
Sulphide	Inferred	16,900,000	0.77	0.75	12.27	0.12
Total		19,800,000	0.79	0.70	13.02	0.16

⁵ As per Item 2.2 (c) of NI43-101 Standards of Disclosure guidelines.

17.10 Economic Pit Limit

The ultimate economic pit limit was based on a Whittle pit optimization evaluation of the resources in blockmodel. This evaluation included geotechnical parameters, mining/milling costs and PGI's long-term selling price assumptions. The economic pit limit included measured and indicated mineral resources material only – no inferred mineral resources were included in this study⁶.

17.11 Optimization Parameters and Results

A mining cost of \$1.20/tonne has been used for both handling of ore as well as waste. This cost has been benchmarked against the Anglo Base Metals Chilean operations using Owner Operated Fleets.

The mining cost adjustment factor ("MCAF") has been calculated to be 1.141, which equates to \$0.0137/tonne per 10m bench.

The processing cost for the base case has been calculated to be \$24.88/tonne treated, which assumes a gravity plant, a flotation plant, an acid leach plant, and a neutralization/cyanidation plant.

The entire blockmodel, which was updated in mid-2008, was loaded into the Whittle optimizer database. It basically contains two rocktypes, namely ore and waste. No differentiation was made between the copper oxide and the copper sulphide. This is due to the fact that the mill costs are the same for both and the Cu recoveries are very similar; 0.81% versus 0.83% for oxide and sulphide respectively. A cut-off grade constraint was imposed on the imported resource blockmodel; 0.60% for Cu and 0.80g/t for Au.

The geotechnical parameters as well as mining, milling, and selling costs are summarized in Table 33. The surveyed projected topography as of 2007 was used as the starting surface for the pit optimization. A series of Whittle pit shells were generated based on varying revenue factors. The results were analyzed with pit shells chosen as the basis for further design work and preliminary phase designs.

Table 33.0 Whittle Input Parameters

Parameter	Amount	Units
Mining		
Mining Cost	1.20	\$/Tonne
Mining Cost Adjustment Factor	1.141	-
Overall Pit Slope Angle	42	Degrees
Mining Rate	5,000,000	Tonnes/yr
Mining Dilution	5	%
Mining Recovery	95	%
Processing		
Flotation/Leach/Cyan.	24.88	\$/Tonne
Overall Cu Recovery	80	%
Overall Au Recovery	90	%
Selling		
Selling Expense for Cu	0.1279	\$/lb
Selling Expense for Au	3.125	\$/oz
RC costs for Cu	0.06	\$/lb
RC costs for Au	5.00	\$/oz
Long Term Cu Price ⁷	2.75	\$/lb
Long Term Au Price	700.00	\$/oz

⁶ As per item 2.3 (b) of NI43-101 Standards of Disclosure guidelines.

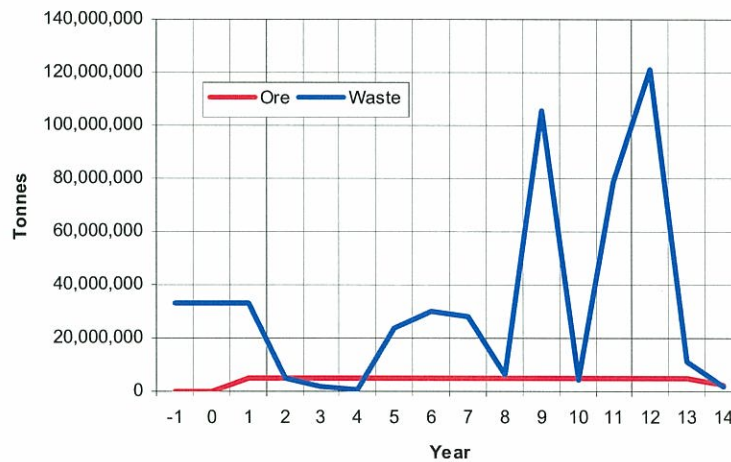
⁷ Base case long term prices; the approximate average of the last three years.

17.11.1 Whittle Results

With a production limit of 5M tonnes of ore feed to the mill output, the Whittle run returns a 14 year full production schedule. The pre-stripping required for this approximately 100M tonnes of material before 5M tonnes of ore are exposed.

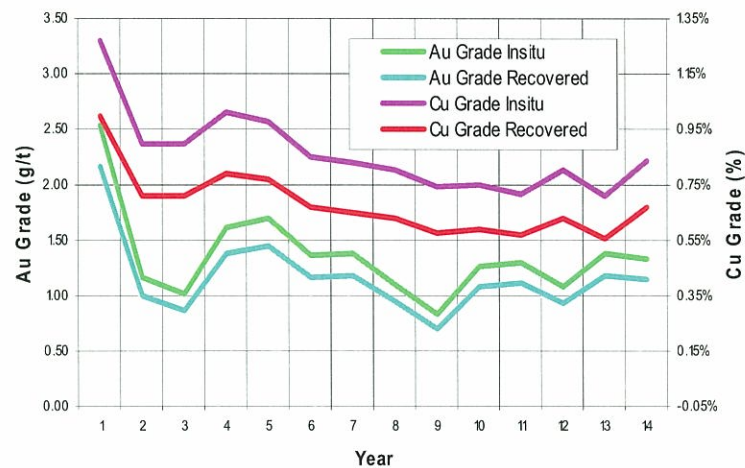
The total mineable reserve is 67M tonnes with an overall stripping ratio of 7.7 tonnes of waste for every tonne of ore. Due to the thickness of the overburden (even at the shallowest part; i.e. 50m), Whittle scheduled 100M tonnes of material for removal in the first year. Since it is not practical to strip such an amount in one year, a pre-strip schedule has been defined whereby 33M tonnes of waste is to be removed the two years, whilst another 33M tonnes is stripped in year 3 exposing 5M tonnes of ore; see Figure 21.

Figure 21.0 Production Schedule - Base Case



The production schedule shows the pre-stripping for 2 years with no ore production. It also shows two very large push backs for years 9 and 12. The mined grades of both Cu and Au are shown in Figure 22 below. Note that there is no ore production in year 1 thus grades are both zero. Also note the Cu grade gradually drops as depth is increased.

Figure 22.0 Cu and Au Grades Mined



The graph shows a decline in the oxide Cu grade as the oxide is depleted and a relatively constant Au grade. The grades shown are insitu grades with no dilution applied since Cu and Au mineralization completely surrounds the mineralized blocks used in the Whittle runs.

17.12 Mineral Reserve Estimate

With all the parameters applied to the Whittle runs, and a detailed financial analyses applied, the following mineral reserves are obtained; see Table 34.

Table 34.0 Proven and Probable Reserves

Type	Category	Tonnes	Cu%	Au (g/t)
Oxide	Proven	54,500,000	0.88%	1.43
Sulphide		11,300,000	0.82%	1.21
Oxide	Probable	490,000	0.97%	0.82
Sulphide		1,480,000	0.81%	0.41
Proven	Total	65,800,000	0.87%	1.39
Probable	Total	1,970,000	0.85%	0.52
Total		67,800,000	0.87%	1.37

The reserves have been categorized as per the CIM Standards on Mineral Resource and Reserves and are thus compliant to National Instrument 43-101.

18 Other Relevant Data and Information

IRES are unaware of any other data and/or information that would be relevant to this report and is not contained in one of the existing sections of this report. The Competent Person is satisfied that all available data relevant to the Boyongan deposit discussed in this report has been reviewed.

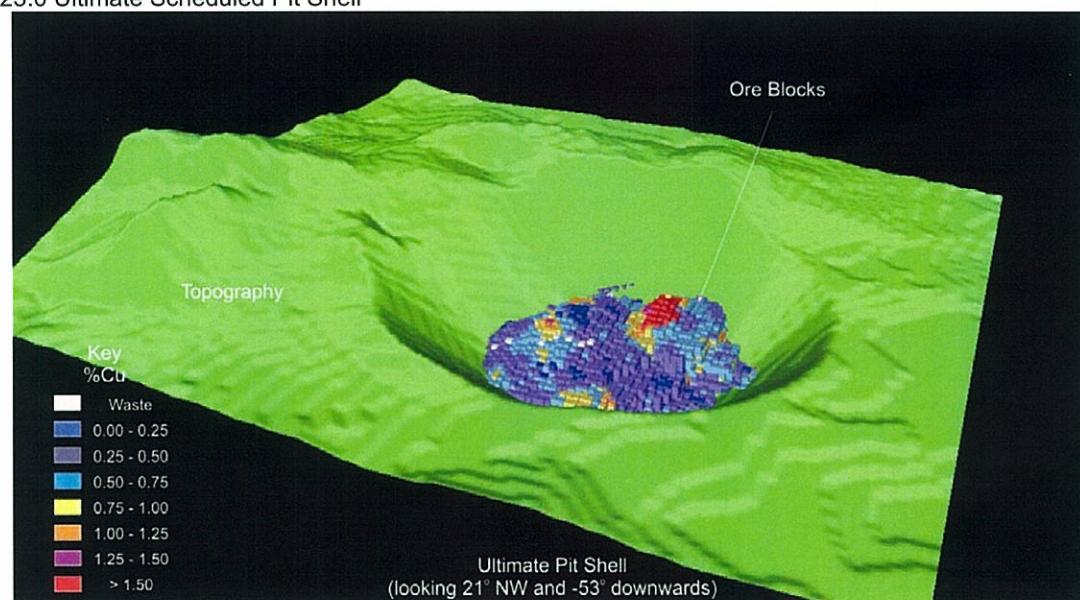
19 Additional Requirements for Technical Reports on Development Properties and Production Properties

19.1 Mining Operations

Mine planning for the Boyongan deposit was conducted using a combination of Datamine Studio 3™ and NPV Scheduler 4™ software. The 3D blockmodel was produced by Mr. D.S.Ferreira of IRES using GSLIB and imported into Datamine Studio 3™. The phase design and production scheduling was undertaken with the use of NPV Schedule 4™ software.

A preliminary pit design, along with associated pit phasing was then based on the Whittle shell analysis described in this report. Preliminary waste dumps were then designed to account for the material produced in each mining phase. Figure 23 shows the scheduled pit for year 14 using the parameters outlined above in 3D.

Figure 23.0 Ultimate Scheduled Pit Shell



19.1.1 Mine Operation

The open pit mining activities for the Boyongan pit are assumed to be undertaken by the owner for the basis of this preliminary economic assessment. The unit rate used in the Whittle optimization was \$1.20/tonne for drilling, blasting, loading, hauling and dumping and was based on current experiences of Anglo.

19.1.2 Equipment

Pre-production stripping would require 33Mt to be moved for the first year, followed by another 33Mt to be removed in the second year; this will allow another 33Mt to be mined exposing 5Mt of ore in year 3.

The maximum haul distance outside the pit is likely to be 1500m. Average haul distance within pit, based on a 10% grade with good hauling roads (3% rolling resistance) would be 1500m. Maximum distance within the pit at 300m depth would be approximately 3000m.

Whether Caterpillar, Komatsu, Terex, O&K, Liebherr or Hitachi are selected as an equipment supplier, the equipment detailed below is indicative of the capital investment required.

Hauling unit selected would be the Cat 789C Off Highway Truck (OHT) (or similar) rated at 180 metric tonnes targeted payload capacity. The loading tool used to match the 180 metric tonne Off Highway Truck, would be a 24m³ bucket capable of loading 40 tonnes per pass.

19.1.2.1 Stripping

Based upon an average 'flat averaged haul' distance of 1500m and a loose density of 2100kg/m³ with averaged dump times, material well blasted and good fragmentation, haul roads well maintained, maximum payload capacity, the Cat789C would achieve 946tph, per truck.

Based upon a 3 x 8-hour shift, 7 days a week, 350 days per year at a 85% utilization, a fleet of 16 x Cat 789C trucks and 4 x Liebherr 996 Shovels would be required to achieve the 33Mt/annum stripping schedule. Due to the high stripping necessary on the onset of the project, it would take 2 years of pre-production stripping before ore production can begin.

19.1.2.2 Pit Depth of 150 Meters

Based on a 'flat averaged haul' distance of 1500m and In-Pit haul at 10% Grade of 2000m and a loose density of 2100kg per m³ with averaged load time and averaged dump times, material well blasted and fragmented, haul roads well maintained, Boyongan Pre-Feasibility maximum payload capacity – the Cat 789C would achieve 333 tonnes per hour , per truck. Based upon a 3 x 8-hour shift, 7 days a week, 350 days per year at a 85% utilization, a fleet of 14 x Cat 789C trucks and 2 x Liebherr 996 Shovels would be required to achieve the required mining and stripping rates.

19.1.2.3 Pit Depth of 300 Meters

Based on a 'flat averaged haul' distance of 1500m and In-Pit haul at 10% Grade of 3500m and a loose density of 2100kg per m³ with averaged load time and averaged dump times, material well blasted and fragmented, haul roads well maintained, maximum payload capacity – the Cat 789C would achieve 254 tonnes per hour , per truck.

Based upon a 3 x 8-hour shift, 7 days a week, 350 days per year at a 85% utilization, a fleet of 18 x Cat 789C trucks and 2 x Liebherr 996 Shovels would be required.

19.1.2.4 Ancillary Equipment

A list of ancillary equipment is shown Table 35 below.

Table 35.0 List of Ancillary Equipment

Amount	Equipment
Support Equipment	
2	CAT 16M road graders
2	CAT D10T TTT bulldozers
1	CAT D11T for heavy dozing
1	CAT 777F water tanker
1	CAT 834H wheel dozer
1	CAT 966 with 4m ³ bucket
1	CAT 994 wheel loader
Drill Rigs	
2	Ingersoll Rand drills
1	Atlas Copco drill
Extra Equipment	

Amount	Equipment
1	55t crane
2	30t cranes
2	7t mobi-lift carry crane
8	7t forklifts
20	4x4 Landcruisers
8	55/60 person buses
6	11/16 seat mini bus
1	Ambulance
1	Panel van
1	Fire engine
2	3t flat bed truck
2	5t flat bed trucks

19.2 Infrastructure and Site Layout

Currently, a number of key infrastructure components exist in and around the Boyongan Open Pit. They are:

- City of Surigao, in the Philippine province of Mindanao
- National road linking Surigao with the Southern part of Mindanao
- Commercial Airport
- Main and secondary ports
- Proposed location of the plant
- Lake Mahukdam (will be used as a temporary supply of water during the construction phase of the project)
- Lake Mainit (will be used for the permanent supply of water to the operation)

19.2.1 Access Roads

There are four main areas covered by the road system:

- National road; linking the town of Surigao with the plant. Also linking the plant to the southern road system of the Mindanao island including cities like Davao, General Santos etc,
- Access road; linking the existing national road to the plant site,
- Plant roads; roads within the plant area.
- Mine roads; access road between the plant and mine and roads within the mine service area. Note the haul road in the pit is part of the mining function.

19.2.1.1 National Road

This is a single lane (6 meter total width) concrete and bitumen surfaced road. The quality of the road surface is good; however there are numerous bridge crossings between the town of Surigao and the plant which are problematic in that their rated capacity is only 25 tonnes. While plans can be made for adhoc construction loads in excess of 25t the long term operational loads will require that the bridges be strengthened to accommodate more substantial loads.

An allowance has been made in the infrastructure capital for enhancing the bridge structures. Also adding to the transportation risks are the narrow town roads and the low level electrical cabling between buildings.

19.2.1.2 National Road to Plant Site

The approximately 2 km access road will extend from the Surigao national road. The current access to the Geological camp (close to the proposed plant location) is a gravel road which would not be satisfactory should the project proceed to implementation. Should the project advance to implementation a new road linking the national road to the plant will be provided.

The road will be 8 m wide, with 19 mm bitumen double seal surfacing and a 2% camber for drainage purposes. The access road will contain mountable kerbs and a 3 m wide gravel shoulder along its length, and will be fenced off within an 85 m wide road reserve.

The road layer works have been designed for a standard 20 year life span with medium to heavy traffic loading and consist of a 150 mm layer crusher run base course, two 125 mm gravel sub-base layers and a 150 mm sub-grade layer of in-situ preparation. This does not mean that the road will need to be reconstructed at the end of the 20 year period, it will however require resurfacing.

19.2.1.3 Plant Roads

19.2.1.3.1 Surfaced Roads

All the Plant surfaced roads will have a 2% crossfall to suit terrace slopes and storm water drains. The 8m surfaced plant roads will extend from the Plant access road, described above.

The same surfaced road construction as described above for the plant access road will be used. This will also apply to parking areas and stoppage/access areas around buildings.

Drain crossings will be accommodated by the use of circular concrete pipe culverts. Electrical cable/road crossings will be constructed in buried uPVC sleeving and/or sub-surface box culverts.

19.2.1.3.2 Gravel Roads

Gravel roads within the Plant area will be to access the tailings dam, mining area and other utilities like the electrical substation and water supply infrastructure.

The gravel access roads will be 4 m wide, unsurfaced, with no kerbing. The design of the gravel roads is based on a light traffic loading and construction will consist of a 150mm wearing course on compacted in-situ material after removal of 100mm topsoil and surface vegetation. Where fill layers are required, these will be constructed in 150mm gravel layers.

19.2.1.4 Mine Roads

The access road for the Mining area will be an 8m wide, unsurfaced haul road extending from the Plant area. The haul road construction will be of a 150mm wearing gravel course, a 150mm gravel sub-base and an in-situ surface preparation layer. Where fill layers are required, these will be constructed in 150 mm gravel layers.

19.2.2 Airstrip

No allowance has been made in the infrastructure capital for the construction of an airstrip at the plant. The project will make use of the existing commercial airstrip located in close proximity to the town of Surigao. This airstrip is 32km from recommended location of the plant and carries daily schedule flights from Manila. Gold bullion would be dispatched through the airport.

19.2.3 Port Facilities

19.2.3.1 Main Port

The main port is in the centre of the town of Surigao some 36km from the proposed plant location. The port is a single quayside handling all products from food to industrial materials. The only access to this port is through the town of Surigao which is difficult due to the low electrical cable infrastructure and narrow roads. There is little container handling infrastructure within the port (no on quay crane facilities). Construction is under way to extend the port quay side to handling more ships. Within the port are there are no storage

facilities which will make the dispatch of concentrate a problem, however in close proximity to the port there are abandoned buildings which will be converted to concentrate storage facilities. Naturally this would entail double handling; see Figure 24.

It is envisaged that the main port would be used for the dispatch of products (both concentrate and copper cathode) and also the input handling of reagents, spares and light equipment.

Of particular attention is the handling of sulphuric acid (15tpm), which will require special consideration in the port and also special equipment. The following is a list of the key components incorporated into the infrastructure capital:

- Acid storage at the port or close proximity to the port (to handle 15,000ton sulphur shipment). This will mean two 18m x 18m enclosed mild steel acid tanks.
- Acid tanker trucks for the transport of the acid from the port storage to the mine. This will require a fleet of 2 trucks on the basis of a 3 hour turnaround between the mine and the port. This equates to approximately 15 trips per day. Storage on the plant has been provided and is included in the plant CAPEX estimate.

Although the supply of sulphuric acid at these quantities is extremely feasible the infrastructure within the main Surigao port may not be capable of sustaining acid storage tanks. Equally the continual transporting of acid through the city of Surigao would attract public concern and would be particularly risky from ensuring a safe passage through the city. If the project proceeds to the detailed feasibility stage, an acid plant should be incorporated into the plant. This would then require the bulk or bag importing of elemental sulphur. Elemental sulphur is easily handled in the bulk form and can easily be stored in a warehouse in close proximity to the plant. Coupled to this the transport of bulk sulphur is safe and would equate to only 5 truck trips per day.

Figure 24.0 Main Port at Surigao City



The export of copper cathode (7500tpa) through the main port of Surigao would not involve warehousing facilities within the port as the quantities per shipment can easily be handling from the plant storage and trucking.

Provision has been made in the infrastructure capital estimate to dispatch the concentrate (60ttpa) through the main port in Surigao. This involves a warehouse in close proximity to the port capable of handling about 5000t equivalent to one month's production. An

alternative scenario would be to dispatch concentrate near the town of Placer by barge (shallow draft) which will be loaded on bulk vessels in deep water. This concept would allow storage of concentrate on the mine and transport to barge as required.

19.2.3.2 Secondary Port

This port falls outside the town of Surigao and is mainly a passenger facility for ferry connections. This port has no infrastructure for the handling of containers and bulk materials. It would however be very useful for the handling of abnormal construction equipment like the mills, crushers, mining equipment etc. Should this prove viable then mobile cranes would be required during the offloading of such equipment. This scenario would require that the project buy mobile crane equipment early in the project. After the project these cranes would become the operational cranes for mine and plant maintenance.

19.2.4 Camp and Office

19.2.4.1 Temporary Facilities

During the construction period, the following facilities will be provided:

- Residential accommodation
- Potable water supply
- Sewage treatment
- Telecommunications
- Electrical supply
- Residential accommodation.

A contractor's residential village will be constructed in close proximity to the plant. The village will cater for all contractor employees, with the exception of the contractor's management staff. The village will be provided with potable water, waterborne sewage, electricity, telephones and a gravel access road. The village will be contained within a security fence which will be controlled by a full time security team.

19.2.4.2 Store, Offices and Other Buildings

The design philosophy for these buildings will be based on that used on recent Anglo Base projects. The emphasis will be on low cost, low maintenance, user friendly buildings which have a concern for the environment. This results in the specification of such products as:

- Colomet coated steel sheeting in colors sensitive to the local environment.
- Face brick external walls for low maintenance
- Deep eaves and solar glazing to reduce air conditioning running costs.
- Floor finishes appropriate to the end use.
- The specification of standard 'off the shelf' fittings to reduce costs.

Office type air conditioners will be specified, which give flexibility during out-of-hours working, and during servicing periods. Any specialist ventilation within the plant buildings will be designed and specified by consultants to be in accordance with current regulations.

General office furniture, furnishings and fittings will be provided which will meet standards currently appropriate in terms of cost and quality. Specialist equipment, loose equipment or computer installations will be installed where advised.

All fire Protection systems (excluding hose reels within the buildings) and any Security systems required will be provided by outside consultants. Lightning protection will be provided to all buildings. The main building requirements would be:

- Plant buildings (including labs, control rooms and offices)
- Administration buildings

- Mining EMV workshop (this facility would also contain the mining offices)
- Stores and plant workshops

The light vehicle repair/fuel Issue building is designed to be operated by an outsourced service provider on a private and commercial basis and is therefore located near to the main road.

19.2.5 Power

The engineering and estimated cost of the electrical power reticulation scheme and control system for the new metallurgical plant, mining pit and administration buildings has been based on the following criteria:

- Optimum utilization of the existing infrastructure and equipment
- Maximize the cost-effective use of power.
- Identify the most cost-effective tariff structure.
- Minimize personnel and operating cost.
- Provide redundant back-up supply routes for critical supplies.
- Provide standby power for emergency personnel evacuation.
- Maximize mine production by ensuring a reliable power supply system to all equipment.

All electrical equipment will be standard electrical equipment for non-flery mines, non-hazardous and non-flammable environment, except in specifically designated areas.

19.2.5.1 Bulk Power Supply

The closest point of electrical power supply is a National Power Company (NPC) 132 kV overhead power line (see Figure 6-2 below). The line runs north south and is approximately 1km for the proposed plant location. Indications are that there is surplus capacity in the line, that the availability of supply is 100% when supplied through the NPC and that the supply is hydro based attracting a tax incentive.

Overhead lines will connect the 132kV consumer substation to the plant.

Medium voltage shall be 11kV. Surface MV reticulation will be done by cables which may be buried or run on vertically mounted cable racks. MV cables will be rated for their nominal current carrying capacity and short circuit current of 19 kA for 1 sec.

The 11kV reticulation system will consist of substations and mini subs units. Outside of the plant boundaries the 11kV power will be reticulated via two overhead power lines, for the tailings dam supply and mining pit supply. The mining pit will be reticulated with electrical power at 6.6kV.

Low voltage for pant and operational infrastructure shall be 525V and 400V shall be used in buildings and lighting. Primary feeders to MCCs and distribution boards will be PVC 4c cables or in case of the transformer to MCC feeder a 3c PVC cable with a separate earth conductor; all cables used on surface will be fully flame retardant.

19.2.5.1.1 Substations

The substations will be constructed as brick buildings with adequate ventilation, pressurization, lighting and a fire detection system. Internal floors will be raised 750 to 1000 mm above the surrounding outside ground levels. Transformer bays will be walled on three sides with a mesh gate on the access side. The transformer bays will not be roofed. The transformers will sit on raised concrete plinths surrounded by drained, crushed stone filled, soak pits. Each substation will be equipped with:

- Medium voltage vacuum or SF6 switchgear.

- Pressurization/ Air filtration equipment.
- Lighting and small power supplies
- Lighting and small power emergency system
- Power transformers, these will be oil filled outdoor units as will any mini subs.
- Lighting transformers
- Motor control centers
- 110 V DC system, battery, and charging unit
- PLC and Control/Protection/Measurement, Marshalling Panels
- Earthing system
- Fire detection equipment and portable fire extinguishers
- Telephone communications

Power for lighting and small power will generally be obtained from 6.6kV/400 V mini substations. Lighting, small power, air conditioning/ventilation system will be supplied at 400 V and welding plugs, crane motors and test equipment will be supplied at 525V.

Lighting wherever possible will be mounted on building structures whether in the plant area or at the mine. Where no structures are available then luminaries will be placed on low height mast structures. The luminaries will generally either be 400 or 1 000 W high pressure sodium lights which will give a yellow light this being advantageous for not attracting insects.

19.2.5.1.2 Emergency Power Supply

The plant area emergency lighting for the various plant buildings will be powered by standby generator sets. This lighting will also be supplemented with self maintained emergency light fittings positioned in critical areas which require continuous lighting for safety reasons.

There will only be a few motors requiring emergency power (thickener mechanisms, agitators etc) and thus it would not be practical to have dedicated diesel generators. Instead the emergency power will be obtained from the lighting power generators and be stepped up to 525 V where necessary.

The final earthing design will be dependent on results from the earth resistivity survey, the aim being to obtain one ohm resistance at the earth mat. All transformer areas, main substations, mini substations, structures and buildings will be earthed via earthing mats or wells under structure foundation level.

19.2.6 Dewatering

Mine dewatering requirements were done using preliminary assumption of the phreatic level. More detailed hydro geological work would still need to be done in the next phases of the project to support these assumptions.

The highest daily rainfall for the region is at 30mm per day November to February.

The Pit design, layout and mining method will dictate de-watering discipline and reticulation. Indicative capital cost requirements for the de-watering scheme is detailed below:

Opting for earthen dams as apposed to concrete lined dams lends to the dynamics of a fluid and changing pit, coupled with the quicker time within which to relocate / construct and commission Relay Pump Stations

Handling of All Water:

- Pit Area = 142Ha
- The 100m radius around pit equates to 45.7Ha

20% of rainfall up to 100m outside of the pit will penetrate the ground, and of this 50% (conservative) will end up as water generated from the pit. Considerations were also given to the impact of the remaining 80% surface run-off water and evapotranspiration.

- Pit rainfall equates to - $142.0\text{Ha} \times 10,000 \times 3.5\text{m} = 4970.0 \times 10^3 \text{ m}^3$ per year.
- 100m radius rainfall = 159.950 m^3
- This equates to 1781 m^3 liters per hour over a 4 month rainy season, 24 hours per day

10 Self Priming Diesel Powered pumps (8 required) powered by 75Kw BF4M 1012 Deutz engines would relay pump the dirty water into earthen dams @ 250m^3 per hour at head of 50m. This would require a total of 160 pumps (including spare) and 16 sets of earthen dams when having to pump to a vertical height of 300m to a lined and inclined Pump Station c/w grit traps outside of the pit which would then pump to the Process Plant or Water Treatment Plant and settling dams.

Doubling up on forecast requirement allows for flexibility within the pit. A combination of submersible pumps and self priming centrifugal pumps will be used to pump to a Relay Pump Station located every 50m vertical height apart. Smaller self-priming and submersible pumps will be used to pump to the Earthen Relay Pump Station equipped with 10 x Self-Priming diesel powered pumps.

19.2.7 Waste Management

Of the two main sources of waste to be generated by future mining activities, the most critical will be tailings material from the process plant. Three alternative tailings disposal options have been considered in terms of technical merit ease of permitting, and likely public acceptance. These are: land-based disposal in a constructed tailings dam, subaqueous disposal in the deep waters of the adjacent freshwater lake (Lake Mainit), and subaqueous marine disposal in deep offshore trenches.

19.2.8 Waste Rock Designs

One of the key factors to be considered in the mine design is the disposal of tailings. ERM have presented a draft note evaluating three options for tailings disposal, namely, terrestrial disposal in a tailings facility, subaqueous disposal in Lake Mainit, which is a freshwater lake, and subaqueous disposal in the ocean, in deep near shore waters. All of these options have pros and cons which would have to be assessed in detail as the project moves into the Feasibility Phase. From the environmental perspective the impact of a land-based facility would have to be designed to withstand seismic events, and would need to consider the potential contamination of surface streams and ground water aquifers from polluted seepage. Treatment of excess polluted water would be a key consideration in an area where rainfall excess evaporation. The impacts of windblown dust would also need to be considered in a land based tailings facility. Subaqueous disposal would need to carefully consider the impacts along pipelines and the potential impact of spills, and then to focus on the possible ecological and community impacts of disposing of tailings into Lake Mainit or into deep coastal waters.

For the purposes of the PFS, it has been assumed that the tailings will be contained within a land-based facility, and this option has been costed into the project estimate. Terrestrial disposal of tailings will present several challenges that will need to be carefully managed to prevent unwanted impacts on surrounding communities and the environment. These are: sterilization of land from agricultural use, dust generation, and seepage of contaminated water from the facility in to the local surface streams and groundwater aquifer.

By volume and tonnage the main waste will be waste rock from the open pit stripping. In the base-case scenario there will be 518.9Mt of waste rock generated by mining activities. Impact, risks, and mitigation of the impact and risks will be examined in detail in the next

phase of the study. However due to the fact that Boyongan is a low pyrite porphyry, acid generation potential is expected to be low.

19.3 Production Schedule

The current life of mine ("LOM") plan focuses on achieving the required mill feed production rate, while balancing grade and strip ratios. In this PFS, no stockpile and blending capacities were assumed.

Mine design for the pit was initiated with estimates of metal prices, mining recovery, mill recoveries, and refining payables and costs, freight. The model was then used with the Datamine NPVS Scheduler™ software to determine the optimal mining shell. Mine planning and scheduling was then conducted on the optimal pit shell with the use of the same software and mineral reserves within the pit shell were estimated using cut-off grades of 0.60% Cu and 0.80g/t Au.

The mining sequence was divided into 14 phases designed to maximize grade, reduce pre-stripping requirements while maintaining a constant mill feed of 5Mta. The LOM schedule is shown in Table 36 below which includes an initial 2 years of pre-stripping prior to production, and shows mill feed grades for both Cu and Au.

Table 36.0 LOM Mine Production

Parameter	Units	Year							
		-1	0	1	2	3	4	5	6
Ore	Mt	-	-	5.00	5.00	5.00	4.99	4.99	5.00
Waste	Mt	33.13	33.13	33.13	4.79	1.88	0.93	23.89	30.18
Strip Ratio	-	-	9.62	0.96	0.37	0.19	4.78	6.04	6.04
Cu Grade	%	-	-	1.27	0.90	0.90	1.01	0.98	0.85
Au Grade	g/t	-	-	2.53	1.17	1.02	1.61	1.70	1.37
Parameter	Unit	Year							
		7	8	9	10	11	12	13	14
Ore	Mt	4.99	5.00	4.99	5.00	4.99	4.99	5.00	2.96
Waste	Mt	27.97	6.19	105.61	4.65	78.84	121.49	11.05	1.89
Strip Ratio	-	5.59	1.24	21.13	0.93	15.78	24.30	2.21	0.68
Cu Grade	%	0.83	0.80	0.74	0.75	0.72	0.80	0.71	0.84
Au Grade	g/t	1.38	1.10	0.83	1.27	1.31	1.09	1.39	1.34

19.4 Recoverability

Detailed metallurgical test work has been conducted on the Boyongan deposit resource. Based on results from the metallurgical tests, the Cu recovery was assumed to be 80% whilst Au was assumed to be 90%.

19.5 Markets

The base case of the Boyongan operation generates the following products over the life of mine:

- 215,369 tonnes of Copper concentrate and 1,091,000 ounces of gold,
- 253,844 tons of Copper cathode
- 49,353 tonnes of Gold doré

A preliminary market assessment was carried out as summarized below.

19.5.1 Cathode Market

The cathodes market is a global market of approximately 17.5 million tonnes with a large number of consumers.

Cathodes are produced by two different processes; the electrolytic one which is produced from concentrated to blister and then to cathodes and the SX/EW one which is produced directly from the mining oxide material using sulphuric acid. The second one could be produced at a much lower cost.

The consumers market is segmented as follows:

- Rod Mills to produce Magnet wires,
- Tube Mills to produce special pipes for air conditioners and refrigerators for example,
- Rod Mills to produce building and power cables,
- Tube Mills to produce water pipes and other type of tubes,
- Brass Mills to produce alloys.

The Rod Mills represents about 55% of total consumption, Brass Mills 15%, tubing 10% and 20% other various products.

Grade A or Premium Grade cathodes could be registered at the LME, Shanghai or Comex, if they comply with the chemical specifications required. Other quality cathodes could be sold as Non-LME Registered or Off Grade cathodes depending on their physical and chemical characteristics.

Grade A registered cathodes could be delivered to the Exchange's warehouse at a market price. This makes the cathodes market very liquid.

Producers charge a premium over the price to Consumers to deliver a Grade A quality cathodes. The premium levels set by Codelco, the biggest producer, are widely used as a benchmark premium by producers, consumers and traders in the copper industry when negotiating their annual contracts. Supply and demand regulates the annual and spot premiums.

In the last three years, higher consumption has lowered the global market stocks from about 1.2 million tons in 2004 down to 0.3 million tons in 2007. China, the actual biggest consumer of copper, has been an important factor to reduce global stocks but also lack investments in new copper projects by producers due to a long period of low copper prices has limited supply. According to analysts the World refined copper market will remain in balance for the next two years and from 2010 to 2012 supply will exceed demand due to new projects coming on stream. Annual benchmark premiums in 2007 were settled at \$125/mt basis CIF Rotterdam for European customers and about \$115/mt for Asian countries. The Spot market has been lower than the annual terms due to lower consumption rate in Europe and US and high copper prices.

19.5.2 Concentrate Market

The concentrate market represents about 13.0 million tons of concentrates or 4.0 millions tons of copper contained in concentrates. Copper concentrates are the main raw material for smelters to produce copper metal. Sellers of concentrates pay a Treatment and Refining Fee (TC/RC) to smelters for processing it into copper metal. Copper concentrates are mainly produced in South America, Africa and Australia. Chile is the biggest copper concentrate producing country, representing about 1/3 of world supply. Smelting capacity is mainly located in industrialized countries in Europe, Japan and US. In the past four or five years, China has increased smelting capacity and has turned out to be the biggest consumer of copper concentrates in the world, demanding about 4.0 million of concentrates per year.

The concentrate market today is very tight and there may be a deficit of approximately 0.5 millions tons of copper in concentrates. Annual negotiations for 2007's procurement of concentrates were mostly settled at TCs ranging \$50 to 60/mt and RCs of 5 to 6 ¢/lb. Actual

spot market rates are even tighter; TC/RC levels were down to a range of \$20 to 30/mt and 2 to 3 ¢/lb.

According to specialists the concentrate market will remain tight for the next few years until new mining projects come on stream in 2010 or until some smelter shuts down due to higher operating costs and low TC/RC fees.

19.5.3 Logistics

A custom built shed at the port would benefit the supply chain. It is better to have extra storage capacity at port as opposed to the mine in order to optimize spot shipping opportunities. Depending on the format of shipments, consideration should be given to loading bulk and containers. It is advisable to have a shed as close as possible to the loading berths to improve load rates and reduce the number of trailers required.

Shipping rates have nearly doubled over the last 18 months but indications are that this could ease by 2009 when new buildings are expected to come into the market. Reducing possible congestion at port and improved loading rates are key to minimizing freight costs.

19.6 Contracts

PGL is taking the view that gold prices will stay above \$700/oz in the medium to long term on account of the US dollar's weakness, fears of inflation, geopolitical tension, etc. It also believes that the copper prices will not sink to the levels (near \$1.50 per lb) of five years ago because unit production costs, in most mines, have generally increased due to various reasons such as declining ore grades, rising prices of electricity and fuel, availability of industrial water, etc.

PGL used \$2.75/lb of copper and \$700/oz of gold as base case assumptions. These prices are close to the 3 year average market prices in the London Bullion Markets Association and the London Metal Exchange ("LME"). The 3 year average (Sept 23, 2005 – Sept 22, 2008) of the PM Fix was \$691/oz of gold. The average for the same period of the official morning cash settlement price at the LME was \$3.12/lb for copper

In order to protect the Project's viability and not leave it exposed to the vagaries of the markets, PGL can and will take measures to protect the Project's revenues, particularly in the first few years of production when there will be Project loan obligations to be paid. As a matter of general policy, the Company will not, however, enter into contracts that will commit more than 50% of metal production at pre-set price levels.

On Sept 22, 2008 the market conditions were such that it was commercially possible for the Company to enter into any or a combination of the hedge instruments described below at (or close to) the price levels and costs indicated. For reference, on that date, the LME PM Fix was \$889/oz and the cash settlement price for copper at the LME was \$3.25/lb.

The Company considered the following hedge instruments and the respective market terms are indicated in the discussions below:

1. A forward contract: where the Company commits to sell, and the hedging counter party commits to buy, a specified quantity of gold every month at the price of \$1045/oz for the period 2012 to 2016. The market information on forward price levels was provided by hedging counter parties of Philex who submitted "indications" but without commitment. The indications were then compared with information from a market data, computation and consulting service company (Kiodex/Sungard).

As an example, the Company can use a forward contract to hedge in the following manner: The Company can hedge 1/3 of anticipated annual production at approximately the above indicated price of 1045/oz for the 5-year period. The market price in 2012 to 2016 will have

to fall below \$400 per oz for the realized price (or the combined revenue from the hedged and unhedged gold) to average below \$615/oz. Under these conditions, the hedge-unhedged ratio of 1:2 yields a robust price for the Project's gold production and yet retains 67% of the upside potential.

2. The Company also considered Put options for price protection. A Put Option gives the option holder the right but not the obligation to sell an asset at a specified price (the strike price) up to or at a specific date in the future. For this right, the buyer of the option pays the grantor of the option an amount of money called the option premium. The premiums are a function of the current price of the asset, the volatility of the asset prices (or the "returns" on said assets), the strike price, the risk-free interest rate, and the time to expiry of the option.

On the same date 22 September⁸, the Company could have purchased Put options for the 5 year period 2012 to 2016 to protect the price of gold at \$600/oz at a cost of \$75/oz (the option premium). Buying Puts to protect all or a portion of the Project's revenues is expensive because these would necessarily be long-dated (i.e., 2012 onwards) and especially today because of the recent volatility in gold prices. Therefore, at this time, hedging by buying Put options is deemed impractical.

Table 37.0 Indications for Premiums for Put Options⁸ expiring 2012 to 2016

Parameter	In US\$/oz			
Put Strike Prices	500	600	700	800
Premium	50	75	110	150

3. Collar: A gold producer can enter into a structure which involves the purchase of a Put option with a strike price that is called the "floor", and the sale of a Call option to generate a premium to offset the cost of the Put option. The buyer of the Call option then has the right to buy gold at the strike price of the Call, thereby setting a "ceiling" on the revenue of the gold producer. Where the cost of the Put is equal to the revenue from the Call, the structure is called a zero-cost collar.

Under the conditions existing on September 28, 2008 the Company could have entered into a collar where the protected floor price on the hedged volume is \$600/oz for 2012 through 2016, and the ceiling is around \$2000/oz. Various combinations or pairs of floors and ceilings available as of that day in September 2008 are shown in Table 38.

Table 38.0 Indications for Zero Cost Collars for Gold expiring 2012 to 2016

Parameter	In US\$/oz			
Ceilings (Call Strikes)	2424	2025	1650	1375
Floors (Put Strikes)	500	600	700	800

The Company believes that with a modest level of hedging, the price of \$600/oz or even \$700/oz can be used as a "Worst" case scenario for gold for purposes of this pre-feasibility study.

The floor for copper that we would choose for 2012 to 2013, if deciding on September 22, 2008 would be \$2.00/lb. The Company sought indications for copper ceilings corresponding to \$2.00/lb floor, and received a wide range of numbers of up to \$4.75. Unfortunately, there are no reliable market data for forward copper prices more than 65 months ahead. The Company has chosen to use \$2.00/lb for copper as "Worst" case and \$4.00/lb as "Best" case for the first two years of operations. Hedging for the succeeding years will be best done at a later date because of the heavy effect of the backwardation in copper prices. Backwardation is also the reason why the Company will not do a forward sale for copper for more than two years into the future. For reference, prices in the last three years ending

⁸ On September 22, 2008; spot price was \$889/oz.

September 22, 2008 averaged 3.12/lb. The last time copper was below \$2.00/lb was in November 2005.

Table 39.0 Indicative Zero Cost Collars for Copper⁹ expiring 2012 to 2013

Parameter	US\$/lb	
Ceilings (Call Strikes)	4.00	3.50
Floors (Put Strikes)	2.00	2.50

It is noteworthy that all of the above structures are easy to understand and “hedge accounting friendly”. This means that any mark-to-market losses (or gains) will not impact the P/L, except for the portion that is settled during the accounting period covered by the financial statement.

19.7 Environmental Considerations

Current exploration activities at Boyongan are conducted so as to minimize environmental impacts. Significant environmental aspects associated with project activities are managed through the ISO 14001 environmental management system, for which certification was obtained in 2005. The footprint of the Boyongan camp site and drilling sites is kept small, and drill sites are rehabilitated as rigs move away. Water used in the drilling process is carefully managed to reduce the impact on adjacent streams. Water quality is monitored downstream of all drilling rigs. Hydrocarbons (diesel & oils) used by the drill rigs are managed responsibly, and are stored under cover and in bunded areas, with spill kits available if needed. All drilling equipment and diesel is transported through narrow paths in the forest using carabao (water buffalo) owned by the members of the local communities.

The key environmental risks facing any future mining development in the Boyongan area are the possible impacts on biodiversity and water quality and supply. Although the forests on and around the Boyongan prospect have been highly modified by the removal of most of the indigenous hardwoods, and replanting with coconut trees, there are seven endangered plant species recorded in the area. Any mining development at Boyongan would need to implement a conservation plan for these species. The risk for loss of biodiversity is medium.

Possible future mining will impact on the water resource in two ways. Firstly by reducing stream flow in the main drainage lines leaving the area, and also by affecting water quality in these streams. Open pit and underground mining will reduce catchment yield by intercepting some of the direct surface runoff and also by lowering the water table in the groundwater aquifers that ultimately feed surface streams through springs. Because of the strong reliance of several villages in the Tubod municipality on dry season stream flow for their livelihoods (stock watering, irrigation of rice paddies, etc), any reduction in base flow in streams would need to be substituted or compensated, probably into the post closure phase. The risk for reduced stream flow and loss of quality is high.

19.7.1 Waste Management

Boyongan will produce a 200mt of tailings during the life of mine. The tailings will be mainly in the “silt” size range (particles between 0.002mm and 0.06mm in diameter) and will therefore have a low permeability after deposition and will consolidate slowly. There are three possible alternatives for the disposal of Boyongan tailings namely:

- Land-based tailings containment in a valley (either in dry or wet form)
- Tailings placement in Lake Mainit; and
- Deep sea submarine tailings placement.

For the purpose of this study it has been decided to pursue the conventional land based wet disposal alternative, even though in the cases of tailings discharged to the lake or the sea,

⁹ On September 22, 2008; spot price was \$3.25/lb.

the tailings liquor as discharged will either directly meet Philippine water quality discharge standards or will meet such standards at the end of a defined *mixing zone* in which the effluent mixes with ambient water. The reason for this decision is that it would be environmentally and socially irresponsible for Anglo to peruse tailings disposal into the sea or inland lake. By way of example Lake Mainit is the source of water and income (fishing and tourism) for many hamlets around the lake.

Environmentally, there are two main issues involved with conventional on-land disposal in this situation:

- Land use concerns, as suitable land in this area would be relatively flat which could be used for agriculture, livestock grazing and human habitation,
- Risk of damage or collapse during earthquakes and exceptionally high precipitation. This risk would extend from the operational mode to the post closure maintaining of the tailings facility. It should be noted that collapse of a tailings dam at Placer has created a negative perception and should the project extend to implementation a lot of detail will be necessary to overcome the negative perceptions in the local communities.

The land base tailings design for Boyongan would involve the following:

- The consolidation of mine waste with clay to form a dam structure suitable for the deposit of tailings,
- Tailings disposal system from the plant,
- Overflow management for storm water control. This would involve the control of excess water on the dam by a bypass decanter arrangement. It would also involve the possible treatment of this of overflow water should it not meet the Philippine water quality discharge standards.

Should the project advance through to feasibility the alternatives to conventional wet base land tailings; either Dry Stacking or Co-disposal should be explored. Unlike conventional TSFs which have a flat to slightly concave surface, storages formed by dry stacking generally have a convex surface, facilitating drainage. For equivalent tonnage of tailings solids, the footprint of a dry stacked storage may be only 40% to 60% that of a conventional TSF. More importantly, the filter cake can be selectively placed higher in the landscape so as to minimize burial of agriculturally valuable, valley floor land. The main issue of concern for dry stacking is the risk of erosion, particularly during or immediately following placement operations, and until such time as rehabilitation has established sufficient ground cover to provide effective erosion control.

19.7.1.1 Sewage Disposal

The sewage system will be a waterborne system with a central treatment works. Although the system will be sized for the construction work force the plant will ultimately be used for the operational workforce.

All sewage reticulation will be via buried pipes with maintenance manholes at regular intervals and all junctions.

19.7.1.1.1 Solid waste

The waste generated from this site will generally be classified as non hazardous waste with only a very small portion of the waste being hazardous.

19.7.1.1.2 Hazardous Waste

The hazardous waste will be removed from site and transported to a licensed hazardous waste dump; these would include:

- Medical waste
- Fluorescent tubes
- Fuel/chemical contaminated soil.
- Laboratory effluent
- Cadmium sulphate

The other hazardous waste generated:

- Used oil
- Used fuel/oil filters
- Spent fuel, oil, paint, chemical containers

These will be removed from site by the supplier or by the contractor using the commodities.

19.7.1.1.3 Non-Hazardous Waste

The non-hazardous waste will be disposed of in a land fill site nearby the sewage plant. The positioning of the solid waste dump will take into consideration the following; ground water, prevailing wind direction, access roads, visual impact and proximity to inhabited areas.

19.8 Environmental Aspects

The Boyongan Deposit and Bayugo Prospect are located in the central highlands of the Surigao peninsula. The topography is characterized by moderate to rolling terrain. The dominant landforms in the area are: Mt. Maniayao, an andesitic dome that peaks at 644 masl and Mt. Silop with a summit elevation of 664 masl. Mt. Maniayao is situated south of Boyongan Project site while Mt. Silop is 3km to the northwest. The topography of the project area, its proximity to major geological structures (e.g. the Philippine Fault and Mindanao Trench) and the inherent seismicity of the region combine to signify the importance of hazard mapping and geotechnical studies of the area. Existing topography and seismicity will impose challenges to the design of on-land tailings storage facilities.

19.8.1 Site Drainage and Surface Water Quality

Effective water management will be central to the successful environmental management of the mine. Since August 2000, Anglo has commissioned Coffey Philippines Inc. to conduct surface water quality survey and in some instances, the surveys included limited terrestrial cover assessment. A total of five surveys were undertaken (i.e. in August 2000, June 2001, September 2002, July 2003 and November 2005) which constitute the present surface water quality database.

The Surface Water Quality Study conducted by Coffey in 2006 which covered an area of 8.75km² in Boyongan Project area and 14 km² in Bagacay Prospect area reportedly identified eight (8) major river systems and eight (8) creeks which presumably lie within the impact area of the project. The major rivers are Bayugo, Cagaasan, Paragayo I, Paragayo II, Timamana, Boyongan, Macalaya, and Motorpool while the creeks are Piakle, Kagang, Batac, Kinapuyan, Pista, Bagacay, Upper Libas, and Alang-alang creeks. These water bodies plus a spring reservoir at Lower Patag were covered by 45 sampling points in the latest water survey (November 2005) – 31 sampling sites in the Boyongan Prospect area and 14 sampling locations in the Bagacay (Bayugo) Prospect area. Mt. Maniayao is practically the catchment of all the river and creeks in the area.

Water uses as observed were for irrigation, domestic and livestock use. Drinking water of the communities is sourced from spring. Estimate of river discharge during the field investigation were done using the velocity-area method.

Among the salient findings of the latest study were as follows: (1) Thirteen (13) samples yielded lower pH values than the standard, which included the source of drinking water

(Lower Patag Spring reservoir). Slightly acidic water was found in the Bayugo, Cagaasan, Boyongan and Macalaya rivers, and in Plakle, Kagang and Bagacay creeks; and (2) Based on the parameters used to evaluate water quality, the water samples were found to be within the DENR Class C Standard, except for coliform content which makes the water unsuitable for drinking. Some of the samples were tested to have exceeded the drinking water standards in terms of arsenic and fluoride contents.

It is advisable to adjust the existing water monitoring program. For one, the current monitoring efforts are preliminary focused around establishing water quality data. Based on past monitoring results, the number and frequency of water quality sampling can be greatly reduced. Second, there is the need to commence data collection on water quantity at selected streams leaving the project area. Flow measurements are best integrated into community development programs by delegating maintenance of installed equipment to trained community members.

19.8.2 Surface and Groundwater and Use

Water is a significant aspect of the environment, and an important resource in the Tubod municipality. Although the area has a high rainfall, future mining activities are likely to affect the water resource, especially during the drier months. The key impacts would relate to reduction in water flow from the project site, both in the groundwater aquifer and in the surface flow in the streams flowing away from the site. Mining and mining-related activities in the area would also likely impact surface and groundwater quality, with a typical increase in suspended solids and salt content. Currently water quality is monitored in drainage lines immediately downstream of each drill rig. The suite of water quality parameters monitored is probably inadequate to verify water analyses through an ion balance (current parameters measured are pH, TSS, BOD, COD, TDS, OSG & Sulphate). However, the early baseline work completed by Coffey, did include all major cations and anions.

Besides the obvious impacts of flow reduction on in-stream biota, the main impacts would be felt by the communities who rely on the water, especially the base flow during the drier months, for drinking water, irrigation water for rice paddies and subsistence gardening, and watering of livestock.

An alternative water supply would probably need to be provided for downstream communities when mining commences. Lake Mainit is the only large body of fresh water in the area that could provide a reliable supply of water. This lake is eutrophic, so any water taken from it would need some pre-treatment at least before this could be supplied to affected communities, or before it was used to replenish streams to maintain the base flows during low flow periods.

The highly porous weathered zone at the Boyongan site carries groundwater that feeds the Song Koy Spring. This spring supplies freshwater to approximately 14,000 people in nine villages in the municipality of Tubod. The water is piped directly from the spring to the villages, many of which are several kilometers away.

The nearby Lake Mahukdam, which lies in the depression in the hillside, has been considered as a possible process water return dam. Closer assessment of this option revealed that this would have a significant impact on the surrounding community, which comprises approximately 53 households, of which 24 are wholly dependent on the water body as a fishery. Lake Mahukdam is a perennial lake that is stocked annually by the local municipality with fingerlings of several species of edible fish. Fish harvesting is accomplished primarily using gill nets.

Surface water monitoring commenced with the early baseline studies done by Coffey Philippines in 2000, and has been continued by the ED SHEC team. Nine surface water monitoring stations are sampled on a regular basis, but no regular borehole sampling is done. Surface water samples are passed to a third party laboratory for analysis.

It is recommended that surface water sampling be done on a more regular basis at appropriate sites on all major streams leaving the Boyongan prospect, and that a wider suite of chemical parameters be measured, along with stream flow (flow measurements are needed to calculate salt loads). It is also recommended that regular quarterly monitoring be done in selected boreholes, to determine water table depth and groundwater chemistry

19.8.3 Land Cover and Land Use

In conjunction with the surface water quality surveys in 2000 and in 2002, Coffey conducted vegetation survey. In terms of vegetation cover, the study revealed the presence of the following seven endangered and one banned species in the project site:

Endangered Species

- Kalantas (*Toona Calantas*),
- Kalingag (*Cinnamomum mercadoi*),
- Tindalo (*Azelia rhomboidea*),
- Pakong Buaya or Giant Fern (*Cyathea contaminans*),
- Pakpak Lawin (*Asplenium musaefolium*),
- Pakpak Lawin Lalaki (*A. nidus*), and
- Orchid (*Dendrobium*).

Banned Species – Prickly Narra

Mt. Maniayao and the major river systems in the project area are considered as the haven for these endangered species.

The study identified three vegetation types: (1) Mixed secondary forestland and brushland, (2) Coconut grove, and (3) Open land or grassland, cultivated and settlement area.

Except for small patches of rice fields and limited settlements, the whole project area is covered by coconut plantation. Being a disturbed area, the site may prove to be uncritical for wildlife species.

19.8.3.1 Mineral Production Sharing Agreement (MPSA)

An MPSA is an agreement under which the government grants the contractor the exclusive right to conduct mining operations within, but not title over, the contract area during a defined period in consideration of a share in the production that takes the form of an excise tax on mineral products. The MPSA is valid for a term as may be mutually agreed upon between the applicant and the government not exceeding 25 years, subject to renewal for another period of 25 years.

19.8.3.2 Financial and Technical Assistance Agreement (FTAA)

Pursuant to the Constitution, the Mining Act provides that the State may enter into an FTAA with qualified persons for the large-scale exploration, development and utilization of mineral resources. Under the Mining Act, qualified persons for an FTAA include 100% foreign-owned corporations. The contractor must make a minimum investment for development, construction and utilization of US\$50,000,000, or its Philippine Peso equivalent in the case of a Filipino contractor. An FTAA may be entered into for the exploration, development and utilization of gold, copper, nickel, chromite, lead, zinc and other minerals. An FTAA may not be granted with respect to cement raw materials, marble, granite, sand and gravel and construction aggregates.

19.8.3.3 Exploration Permit Application (EXPA)

An EXPA is an agreement under which the government grants the contractor the exclusive right to conduct exploration within the contract area during a defined period. The term of an

Exploration Permit is two years from the date of the issuance, renewable for like periods but not to exceed a total term of six years. Unlike the MPSA, an EXPA is limited to exploration.

19.8.3.4 Legal and Environmental

The Philippine government grants MPSAs on the condition that the subject mining activities are managed in a technically, financially, socially, culturally and environmentally responsible manner to enhance the national growth and welfare of the Philippines.

The Department of Environment and Natural Resources (DENR) requires an Environmental Clearance Certificate (ECC) for any mining activity at a more advanced stage than mineral exploration. The ECC is issued by the DENR based on an Environmental Impact Statement prepared by the company in accordance with the procedures under the Environmental Impact Assessment System. A completed ecological profile of the proposed mining area is required as part of these procedures.

The ECC is the basis for the Environmental Protection and Enhancement Program (EPEP), which every party to an MPSA must undertake within the subject area of the MPSA. The EPEP must be submitted by the company within 30 days from receipt of the ECC. To effectively implement an approved EPEP, an Annual Environmental Protection and Enhancement Program (AEPEP) is required to be submitted 30 days prior to the beginning of every calendar year. Mine site inspections and quarterly monitoring are conducted by the DENR and a multipartite monitoring team composed of local government units, host communities, any affected Indigenous Cultural communities, non-governmental organizations, the DENR, and the company to ensure compliance with the AEPEP.

Companies are required to rehabilitate technically and biologically all areas that are excavated, mined out, covered with tailings or otherwise disturbed and to establish a mine rehabilitation fund, based on their work program.

19.8.4 Environmental and Socio-Economic Baseline Studies

Since August 2000, Anglo American has, through its joint venture company Silangan Mindanao Mining Company (SMMC), used consultants to do the environmental and community baseline work for the Boyongan project. The early studies were done by Coffey Philippines, and included both terrestrial assessments (vegetation surveys) and hydrological studies (mainly water quality).

In December 2006, environmental consultants PT ERM Philippines ("ERM") were commissioned to undertake additional baseline work and to do a preliminary environmental impact assessment, covering the bio-physical environment and communities, to support a Pre-feasibility study to be completed in the latter part of 2007. ERM reviewed previous work done by Coffey, and conducted a site visit in February 2007. They produced a Site Visit Report in March 2007 which describes the project location, the environmental setting, and the baseline data known from previous studies. The report outlined the prevailing climate, land forms, site drainage and water quality, as well as ground water occurrence and use, and land cover and land use. An evaluation of environmental management practices on site was also covered. On community issues, the company's current corporate social responsibility activities were also described. One of the main objectives of the report was also to identify gaps in the existing environmental and community baseline studies that would need to be covered for the PFS. The additional work required was presented in a proposal submitted in April 2007.

At the time of writing ERM are compiling the environmental and social baseline reports. These reports will contain the level of detail required for the Pre-Feasibility Study. ERM have already completed most of the specialist studies that provide the basis for compiling the Environmental Baseline Report. In addition to the baseline studies completed by Coffey, the following additional studies have been completed to date and will be compiled to for the ERM Environmental Baseline Study:

- Lake Mainit Legislative Study and Preliminary Local Land Use Assessment,
- Rapid Aquatic Biological Assessment of Boyongan Copper/Gold Project Site, Surigao, Philippines,
- Rapid Biological Assessment of Tailings Disposal Options for Boyongan Copper/Gold Project Surigao, Philippines,
- Preliminary Assessment of the Fisheries of the Lake Mainit Watershed,
- Preliminary Review of tailings disposal Options for Boyongan Copper/Gold Project Surigao, Philippines.

The two key ERM consultants working on the project are Ramon Santos (Project Coordinator, with training in mining engineering and geology) and Anacleto Suelto (a geologist with experience in hydrology and hydrogeology). Both are Filipinos, who work out of the ERM offices in Jakarta, Indonesia. They have a good understanding of the environmental law in the Philippines and are familiar with structures and processes within the DENR (Dept. of Environment and Natural Resources) and the Bureau on Mines. As a consultancy, ERM Indonesia has a proven track record, having worked on a wide variety of mining project in both Indonesia and in the Philippines.

19.8.5 Consultation Program

Community development projects are handled by Anglo's joint venture partner, Philex, but Anglo retains an oversight role in that the full budget for these projects is provided by Anglo

Four community development coordinators have been appointed to run the community development program for the JV partnership, and they work out of a dedicated community centre in Timamana village. Community development projects have been rolled out in three barangays (villages), viz., Timamana, San Isidro and Capayahan.

Two community development coordinators hosted the Anglo team during the visits into the communities and to look at community development initiatives and to understand the impact of any future developments on the surrounding villages. These coordinators were John Darwin Eludo and Loida Apatan.

The projects supported by the JV are varied and include education projects, community infrastructure projects, sustainable livelihoods project, healthcare and so forth. The annual CSR spend is between 1.5 and 10 million pesos.

For all community projects, the company requires communities to participate in what is known as "counter parting". This simply means that communities have to make an independent and substantial (usually labor) contribution to company funded projects

The community centre in Timamana is open daily, and has a library and computers for use by the community. Students are assisted in developing reading and computer skills.

Several community development projects were seen during the visit. The following represent a few of the initiatives seen:

- A wholesale rice business: the team visited the house of the president of the business who explained how seed capital had been provided by SMMC to purchase rice in bulk at wholesale prices, which is repacked by woman partners in the company who sell this on to the community at a profit. This successful project supports several families.
- Repair of an irrigation weir: at the time of the visit a community work team was busy reconstructing an irrigation weir that had been destroyed by floodwaters. The company had provided cement and reinforcing steel for this project, and the community was counter parting by providing labor to recover suitable sand from the

river, to mix this with the cement, and to lay the reinforcing rods and cast the concrete sections to rebuild broken parts of the weir. This weir will be used to direct water to extensive rice paddies situated along the floodplain of the river. These paddies have not been productive since the weir was destroyed some years before.

- Provision of seed and fertilizer: two gardening project were seen during the visit, one in the village of Capayahan, where a variety of vegetable crops had been planted behind the school for use by the community. The second project was seen adjacent to Lake Mahukdam, where egg plants, cassava and chillies had been planted over extensive areas of cleared forest, interspersed with areas occupied by coconut trees, banana and pawpaw trees.
- Repairs to school clinic: capital provided by the JV was also being used to upgrade the school clinic at Capayahan.
- Medical and dental services: on a regular basis the community development team arrange for medical and dental services to be provided for the community. Medical and dental practitioners are sent into the communities with supplies of medicines and equipment to provide this service.

19.9 Taxation

The mineral projects in the Philippines are subject to excise of 2% on the net smelter revenues (gross revenues less marketing and treatment charges; a tax of approximately 30% on the net income of the corporate entity; a value-added tax of 12% will be levied on purchases and contracts during the pre-production stage.

19.10 Capital Costs

The mining capital estimate was prepared by PGI and reflects the Philippine context.

The total project implementation capital including contingencies is \$657 million as at September 2008. This CAPEX figure is for the base case and includes the following:

- Contingency applied at 10%,
- Full mine CAPEX which includes EMV workshops, mining vehicles, service vehicles etc,
- Plant capital including milling, flotation, acid leaching and cyanidation,
- Infrastructure (includes tailings disposal, water management, electrical distribution, roads, port handling, vehicles, communication etc). The costs for these components have been factorized off the plant capital numbers,
- No escalation component has been allowed for in the capital estimate.

The detailed capital breakdown of the base case estimate is reflected in Table 40 below.

Table 40.0 Detailed Capital Cost Estimate

Area	Description	US\$M
Mining	Mining equipment	\$129.87
	Explosive storage and mixing	\$4.89
	Fuels and lubrication infrastructure	\$5.71
	Pit dewatering	\$11.24
	EMV workshop	\$12.26
	Subtotal	\$163.98
Plant	ROM Crushing & Stockpile	\$15.08
	Milling	\$39.23
	Gravity Concentration	\$3.19
	Sulphide Ore Flotation	\$6.87

Area	Description	US\$M
	Cu/Au Combined Concentrate Dewatering Plant	\$6.61
	Non-Acid Dewatering thickener & Belt filters	\$21.64
	Cu Acid Leach	\$3.14
	Acid Thickening and Filtration	\$37.04
	Cu Recovery from filtrate	\$2.18
	Iron Precipitation	\$1.68
	Cu Solvent Extraction	\$9.14
	Cu Electrowinning	\$31.25
	Neutralization and Cyanide Leach	\$11.30
	Resin and Carbon in Pulp	\$11.76
	Loaded Carbon Elution and Regeneration	\$3.26
	Resin elutriation & resin regen.	\$3.26
	Gold Room	\$1.12
	H2SO4, HCl, Caustic & CN	\$1.93
	Limestone	\$8.32
	Water Treatment & Reticulation; Compressed Air	\$14.51
	Site Preparation & Infrastructure	\$7.61
	First fill of Reagents and Raw Material	\$16.67
	Subtotal	\$256.79
Infrastructure	Electrical infrastructure	\$15.76
	Tailings disposal	\$12.20
	Tailings water treatment	\$5.29
	Spares	\$6.61
	Roads (including bridge enhancement)	\$1.10
	Acid storage tanks at Main Surigao Port	\$2.42
	Concentrator warehouse	\$2.12
	Infrastructure site preparation	\$1.07
	Communication systems (IT)	\$5.77
	Stores offices and amenity buildings	\$7.28
	Security	\$0.07
	Recreational facilities	\$0.75
	General service vehicles	\$1.65
	Temporary facilities for construction	\$2.64
	Fire protection	\$1.12
	Water supply and treatment	\$10.58
	Solid waste disposal	\$0.28
	Sewerage treatment	\$0.31
	Storm water drainage	\$1.66
	Subtotal	\$78.67
	Total	\$499.45
	Engineering+ Owner's costs	\$61.30
	Project Overheads (Insurances etc.)	\$36.95
	Subtotal	\$597.70
	Contingency	\$59.77
	Grand Total	\$657.47

All financial runs assumed the same capital as outlined in the table above. The grand total above does not include the pre-stripping costs of \$80 million.

19.11 Operating Costs

The operating costs are described in more detail in the specific sections, but in summary, the numbers shown in Table 41 have been used in the financial evaluation. All OPEX costs were received from PGI.

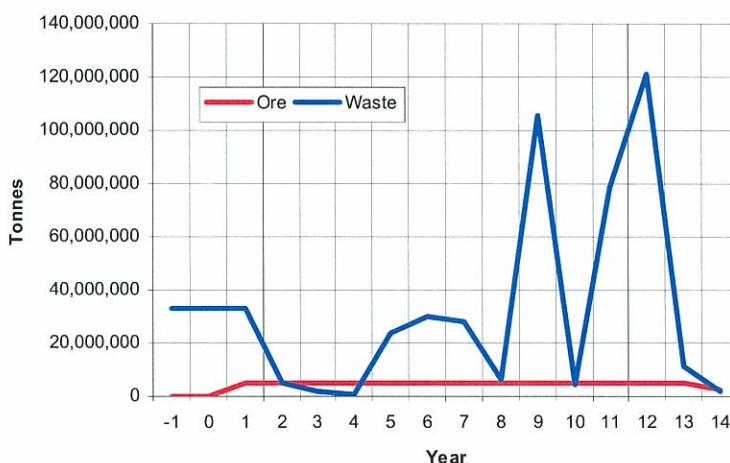
Table 41.0 Operating Cost Estimates

Parameter	Unit Cost	Unit
Mining Cost (ore & waste)	\$1.20	\$/tonne
Services	3.56	\$/tonne
Labor	3.36	
Regents	13.46	
Tailings	0.70	
Laboratory	0.07	
Maintenance	0.84	
Spares	1.40	
Contractors	0.25	
Contingency	2.33	
Total Milling costs	\$24.88	\$/tonne

19.12 Economic Analysis

The Whittle inputs for the base case have been summarized in Table 33. With a production limit of 5Mt of ore feed to the plant output from Whittle, a 14 year production schedule is achieved. This is exclusive to the 2 first years of pre-production stripping. The northwest portion of the orebody (formerly referred to as the *exotic zone*) is not reached by the ultimate pit shell as the stripping ratio for this area is too high to be economic.

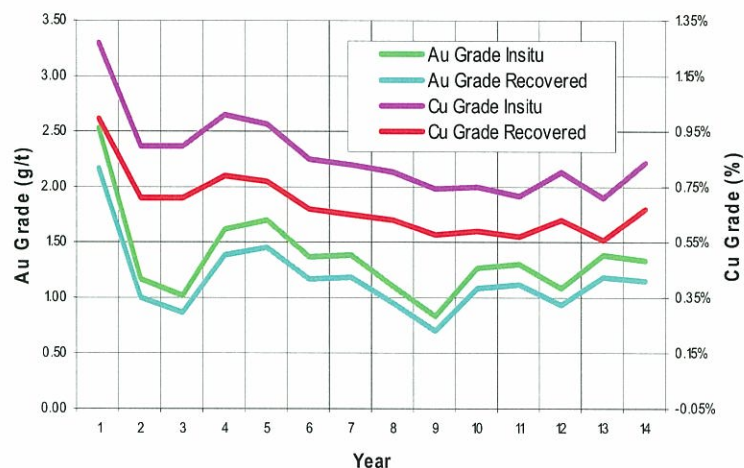
Figure 25.0 Production Schedule - Base Case



The total mineable reserve is 67.8Mt with 518Mt of waste; therefore the overall stripping ratio is 7.7 tonnes of waste for every tonne of ore. The thickness of the overburden required removal prior to accessing ore caused the Whittle run to schedule about 100Mt of waste removal for the first year of production. Since it is not practical to strip such an amount in one year, a pre-strip schedule has been defined in which 33.1Mt of waste is to be removed annually commencing 2 years in advance of mining ore.

The mined grades are shown in Figure 26 for both insitu and recovered ore. Note that during the 14 year production life of the pit, oxide copper grades decrease whilst Au grades remain fairly consistent.

Figure 26.0 Grade Schedule

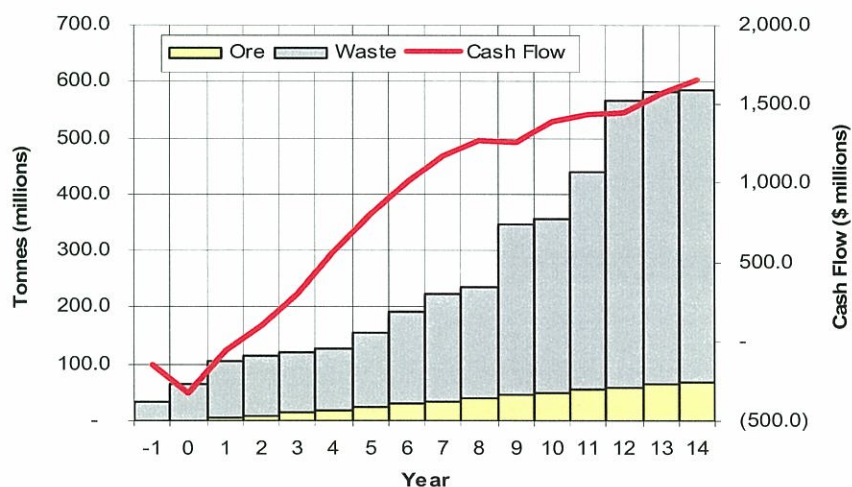


The insitu grades refer to mineralized blocks in the ground which are mined. This material is then feed to the mill (i.e. millfeed). The recovered grades refer to material processed by the mill; mill output.

19.12.1 Financial Evaluation

The project base case scenario returns a positive NPV of about \$150M at a 12% discount rate and an internal rate of return of 18% (prior to financing costs) using long term prices of \$2.75/lb for Cu and \$700/oz for Au. Extensive work was done in order to determine the sensitivity of the project to operating costs and metal prices. The cumulative cash flow (prior to financing) along with the cumulative ore and waste tonnes, over time, are shown in Figure 27 below.

Figure 27.0 Cumulative Tonnes and Cash Flow Results



19.12.2 Sensitivity Analysis

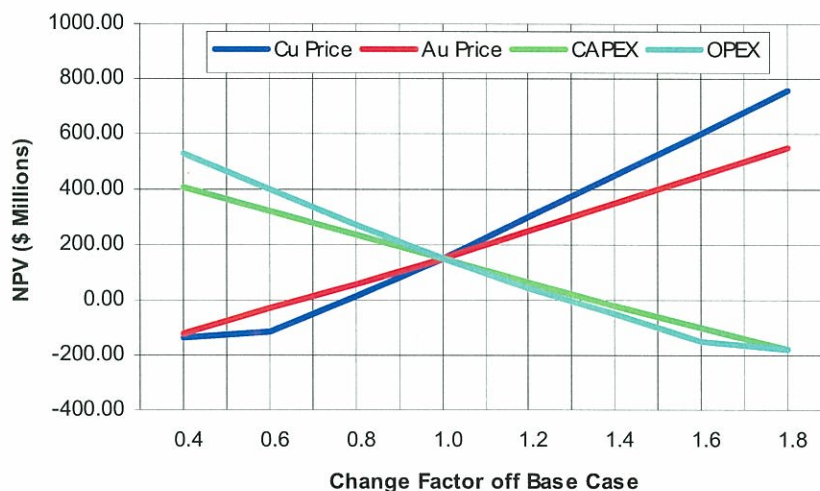
Further sensitivity analyses were conducted at varying metal prices, discount rates and recoveries. The inputs for these exercises are shown in Table 42 below.

Table 42.0 Effect of Sensitivity Factor

Parameter	Factor (multiplier)							
	0.4	0.6	0.8	1	1.2	1.4	1.6	1.8
Cu Price	1.10	1.65	2.20	2.75	3.30	3.85	4.40	4.95
Au Price	280.0	420.0	560.0	700.0	840.0	980.0	1,120.0	1,260.0

The NPV versus the various factored prices and costs were plotted using a constant discount rate of 12%; see Figure 28 below.

Figure 28.0 Sensitivities vs. NPV₁₂

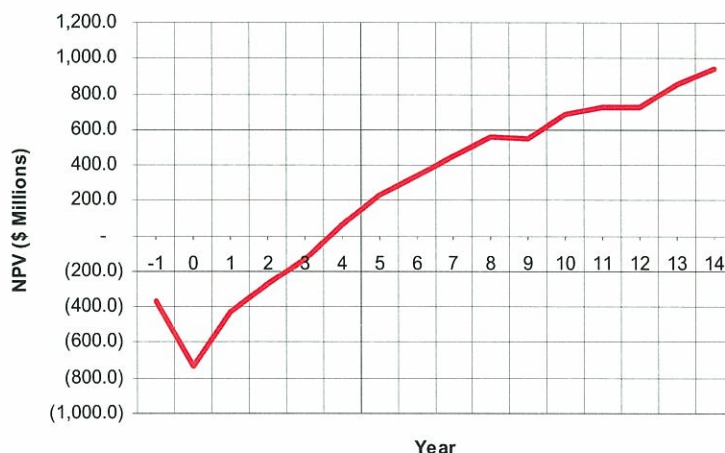


The sensitivity analyses reveal that the project is more sensitive to the price of Cu followed by the price of Au, then OPEX and lastly CAPEX. The break-even price for project viability in terms of Cu price is \$2.12/lb, and for Au it is \$470/oz.

19.12.3 Payback

The payback period for the base case is shown in Figure 29 below. The figure displays the cumulative cash flow (undiscounted) versus time. It will take 3.5 years to payback the capital (not taking into account financing options).

Figure 29.0 Cumulative Cash flow



19.12.4 Mine Life

The mine life of the Boyongan project, based on the assumptions made in this study, is 14 years, with an additional 2 years of pre-stripping; for a total of 16 years. There are a number of potential factors that could extend the mine life that have not been included in this report.

The pit will still contain mineralization after the current proposed pit is mined out. The PFS Whittle optimization analysis and subsequent mine design were conducted using a \$2.75/lb copper price and a \$700/oz gold price. If a higher copper price is used, then, all things being equal, the optimal Whittle shell would expand and the total mill feed and mine life would increase; likewise for gold. However, using the current cost/pricing structure for the base case, a relatively high grade (i.e. copper) extension of the main mineralized body exists in the northwestern side of the main body. Once open pit mining is in full operation, geotechnical parameters allowing, it should be possible to access this area via an adit via the pit ramp. This would benefit the operation since all development would be completely in mineralized rock, thus minimizing development costs.

There is the potential for further exploration discoveries and/or expansions of current known resources along known mineralized trends.

20 Risks and Opportunities

20.1 Introduction

As part of the ongoing PFS, Anglo American commissioned Control Risks Group ("CRG") to prepare a risk assessment in order to gain some insight into the possible changing political landscape in the Philippines over the next five years. Particular issues of interest include changes in the Philippines' system of government; adverse policy changes in respect of mining investment; the influence of the political elite in Caraga region; insurgent activity by the New People's Army ("NPA"); the activities of anti-mining non-governmental organizations ("NGOs") and church groups and labor related issues. The CRG report examines the security risks to Silangan Mindanao Mining Company Inc.'s ("SMMCI") planned operations, including analysis of the future business environment in the Philippines in terms of overall political risks and project-specific operational and security risks.

20.2 Political Risk

The Philippines has a long history of political change through mass demonstrations, with two presidents deposed in "people power" uprisings. Although opposition groups continue to conspire against the current administration and stage opportunistic "coups", the lack of popular support and a viable alternative leader make it unlikely that the President Arroyo will be deposed during a coup d'état prior to the end of her term in mid-2010.

Within the Surigao Project area, Congressmen Guillermo Romarate and Francisco Matugas, Provincial Governor Ace Barbers and Mayor Cristina Romarate (daughter of Cong. Romarate) of Tubod Municipality are openly supportive of mining investments. While there exists some opposition in nearby towns, it is highly unlikely that the SMMCI's operation would be threatened within the medium term by local politicians, however they are vulnerable to public pressure and sudden changes in their stated position on mining is possible.

20.3 Economic Risk

Although the Philippines' economic growth has strengthened in recent years to 5.3% in 2006 and ~6.5% in 2007, the economy has rarely experienced high rates of economic growth for sustained periods to deliver major developmental advances and unbridled population growth will negate many of the gains. The proportion of urban and rural poor is predicted to increase despite the government's limited poverty alleviation programs. This will result in the country's continued vulnerability to political upheaval and the capacity of politically entrenched business elites to block far-reaching economic reforms.

20.4 Operational Risk

Historically, mining has proved to be one of the most politically controversial and divisive industries in the Philippines. The country's extensive but under-exploited mineral inventory has attracted relatively little foreign investment from international mining companies with the projects of Xstrata-Indophil at Tampakan and SMCCI at Boyongan being the only major projects in the pipeline at present. This is largely due to persistent legal challenges to the Mining Law and to specific projects, opposition from NGOs and various security risks.

Successive administrations have made significant changes to the investment regime and even revoked contractual agreements entered into by their predecessors. Mining sector legislation remains a divisive domestic political issue and subject to "National Patrimony" considerations. So-called "creeping expropriation" wherein a series of discriminatory legal judgments or changes to prevailing legislation make a venture commercially unviable for a foreign company is possible. The Supreme Court's revocation of many of the provisions of the 1995 Mining Act in its initial 2004 decision is a case in point.

Despite frequent government pledges to eliminate corruption and occasional legislative actions, corruption will remain prevalent, both within government and business circles.

Initiatives such as the Extractive Industries Transparency Initiatives (EITI) have gained little support in the Philippines. The withholding of mining permits, environmental compliance certificates and operating permits by a variety of government departments could certainly slow down the permitting of a mining project.

The Philippines has one of Asia's most active "civil societies", with myriads of NGOs. Although moderate, apolitical NGOs can make a valuable contribution in stake-holder engagement programs, more radical groups will consistently oppose any form of mining activity on ideological grounds. Equally active in anti-mining protests are the pervasive church groups, left wing groups and some indigenous people's organizations.

There are thousands of trade unions in the Philippines, some of them very militant. However, generally if companies manage their labor relations effectively, problems do not often occur and most foreign investors enjoy relatively smooth industrial relations. The most common labor related issue is redundancy which is covered by the cumbersome Labor Code. SMMCI enjoys substantial public support within its project area as the company is perceived to be interested in community affairs and public welfare.

Reputation risks could result from allegations of environmental degradation and human rights abuses even if subsequently proven to be baseless. As SMMCI's operations are not situated in a remote area, the necessity for military assistance and the possibility of associated human rights issues is not a significant issue at present.

20.5 Security Risk

Crime and kidnapping present significant risks, although typically to expatriates; most of these problems are concentrated in Metro Manila. Armed robbery, both at home and on the street and extortion are commonplace and increasing. In general Surigao del Norte reports low levels of criminal activity and should have little impact on Anglo's personnel assuming they exercise commonsense and discretion.

Ceasefire lapses and government offences against various Islamic based rebel groups have a tendency to provoke reprisal attacks in major cities.

The Communist Party of the Philippines (CPP) and the New People's Army (NPA) have been particularly vocal in recent months in their denunciation of foreign owned mining firms and have vowed on several occasions to increase their attacks on mining operations. In the past year the NPA has attacked and partially destroyed the operations of Filminera at Masbate, Pelican Resources at Mabuhay, Oceana Gold at Didipio, Eldore Gold at Nalesbitan and most recently Xstrata at Tampakan. SMMCI's operations in Surigao also pose a credible target for the NPA. Within Surigao del Norte, the NPA has engaged in opportunistic ambushes and targeted assassinations against government or private targets identified as "counter-revolutionary" or abusive to the local community.

The NPA are reported to have 40-50 armed cadres within the Municipalities of Placer and Sison and a transient presence in the strongly anti-mining Barangay Anislagan. It therefore does represent a possible threat to Anglo's exploration office in Barangay Luna, Surigao City and also to the northwestern portion of the Boyongan Project area. However the possibility of such an attack is mitigated by the perceptively strong positive impact of SMMCI on the community. The local community has developed a keen interest in the presence of SMMCI and any targeting of the company's operations could provoke a backlash from the local community.

20.6 Mineral Resource Estimate

High grade zones might not be continuous enough to allow for increased resource grades in the early years of production. There might be a variation between the predicted and actual deposit grades, potentially leading to higher mining dilution.

20.7 Mineral Reserve Estimate

There might be a variation in the metallurgical characteristics of the ore mined. This will invariably lead to changes in the mill circuit design and impact on the recoveries of both Cu and Au.

In addition, metal price fluctuations are difficult to predict and can easily lead to erroneous predictions and therefore affect project viability.

20.8 Mining Operations

Ground water inflow may occur in greater quantities than predicted and therefore adversely hamper mining operations. Actual pit slope angles may vary from the geotechnically-determined results. This can lead to flatter slopes thus increasing strip ratios and decrease in-pit reserves.

21 Interpretation and Conclusions

21.1 Interpretation

IRES have worked diligently since acquiring the project to ensure that all of the data, whether current or former, meets current Canadian National Instrument 43-101 standards. The new data have been generated by continued exploration drilling, completion of bench-scale metallurgical test work, and comprehensive geotechnical, hydrological, and environmental studies. All of these various studies and data have advanced the project to a point that supports the next level of development and expenditure of funds through the development of a detailed feasibility study.

21.2 Conclusions

After a thorough review of all the data available for this pre-feasibility study, and the results of this study's Whittle and financial runs, the author of this report summarizes the following:

- This study was themed on an experienced Filipino mining company operating within the Philippines,
- Costs for salaries and construction in this study are in tune within the context of a Filipino company's cost experiences,
- Long-term metal prices, as postulated in this PFS, were based on PGI's and its parent company, Philex's long term corporate strategy;
 - Sales forecasting, price hedging and other strategic financial scenarios are typically company dependent. This study has incorporated PGI's and Philex's views.
- The mineral resource model used in the determination of mineral reserves and project economics was the one completed by IRES in 2008 using all available drillhole data up to, and including, December 31st, 2007.

Based upon the assumptions used in this report, the results of the PFS indicate that the Boyongan project is technically and financially feasible. It also has revealed that in addition to the reserves extractable from the open pit mining activity, additional opportunity exists for the extraction of additional resources located in the northwest area of the deposit via an adit accessed through the pit ramp. This would minimize initial pre-production costs given that the development would be through mineralized rock.

22 Recommendations

IRES recommend that PGI (or its parent company) proceed with information gathering for a detailed and bankable feasibility study for the Boyongan deposit. The following are the major recommended tasks:

- Additional metallurgical and mineral processing tests including, but not limited to, tailings performance, crushing and milling characteristics, process recovery optimization, reagent consumption, reagent type, mill feed reactivity and characterization of mineralization,
- Pursuit of environmental and permitting,
- Conduct a bankable feasibility study.

It is IRES's opinion that the data used in support of and for the completion of this pre-feasibility study are compliant with CIM definitions and that the mineral resources and reserves presented meet the requirements of National Instrument 43-101.

23 References

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24 Standard Acronyms and Abbreviations

Distance		Others	
µm	micron (micrometer)	°C	degree Celsius
mm	millimeter	°F	degree Fahrenheit
cm	centimeter	Btu	British Thermal Unit
m	meter	cfm	cubic feet per minute
km	kilometer	elev	elevation above sea level
"	inch	amsl	above mean sea level
in	inch	hp	horsepower
'	foot	hr	hour
ft	foot	kW	kilowatt
Area		kWh	kilowatt hour
m ²	square meter	M	Million
km ²	square kilometer	mph	miles per hour
ac	acre	ppb	parts per billion
Ha	hectare	ppm	parts per million
Volume		s	second
l	liter	s.g.	specific gravity
m ³	cubic meter	usgpm	US gallon per minute
ft ³	cubic foot	V	volt
usg	US gallon	W	watt
lcm	loose cubic meter	Ω	ohm
Mlcm	Million lcm	A	ampere
bcm	bank cubic meter	tph	tonnes per hour
Mbcm	Million bcm	tpd	tonnes per day
Mass		ø	diameter
kg	kilogram	Cu eq.	Copper equivalent
g	gram	Acronyms	
t	metric tonne	ABA	Acid- base accounting
Kt	Thousands of tonnes	AP	Acid potential
lb	pound	CAPEX	Capital costs
Mt	Millions of tonnes	CIM	Canadian Institute of Mining
oz	troy ounce	CV	Coefficient of Variation
wmt	wet metric tonne	DDH	Diamond drill hole
dmt	dry metric tonne	HDPE	High-density polyethylene
Pressure		IRES	Independent Resource Estimations
psi	pounds per square inch	IRR	Internal Rate of Return
Pa	Pascal	LOM	Life of mine
kPa	kilopascal	ML/ARD	Metal leaching/ acid rock drainage
MPa	megapascal	N, S, E, W	North, South, East, West
Elements and Compounds		NI 43-101	National Instrument 43-101
Au	gold	non-PAG	Non-potentially acid generating
Cu	copper	NP	Neutralization potential
As	arsenic	NPTIC	Carbonate neutralization potential
Conversion Factors		NSR	Net Smelter Return
1 tonne	2204.6 lb	OPEX	Operating costs
1 oz	31.1 g	PAG	Potentially acid generating
		PFS	Preliminary Feasibility Study
		PGI	Philex Gold Incorporated
		% or pct	percent
		QA/QC	Quality Assurance/Quality Control
		X, Y, Z	Cartesian Coordinates, or Easting, Northing & Elevation

25 Date and Signature Page

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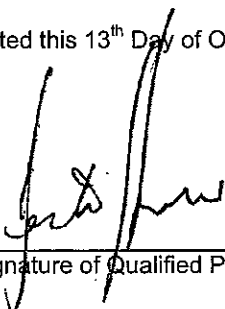
Certificate of Author

I, Dexter S. Ferreira, do hereby certify that:

- I am currently employed by Independent Resource Estimations cc. at:
11 @ 13 Jupiter Street,
Westonaria, 1780
Gauteng, South Africa
- I have graduated with a Bachelors of Science (BSc.) degree in geology from McGill University in Montreal, Quebec, Canada in 1983. In addition, I graduated with a Bachelors of Engineering (BEng.) degree in mining from McGill University in Montreal, Quebec, Canada in 1989.
- I am a Member of the South African Council for Natural Scientific Professions (400048/03) and in good standing.
- I have worked as a geostatistician for a total of almost 19 years since my graduation from university, as an employee of various major mining companies; the last one being Placer Dome Inc. as a geostatistician. I have been a consultant for over 3 years.
- I have read the definition of a "qualified person" as set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
- I am responsible for the preparation of the pre-feasibility report entitled "Pre-Feasibility Study, Boyongan Project." I have visited the property and the field office in Surigao, Mindanao, Philippines.
- I have completed an independent review of the data collection, preparation and analysis of the data presented herein. I have personally completed the Pre-Feasibility study for Boyongan.

- I am not aware of any material fact or material change with respect to the subject matter of the Pre-Feasibility of the Boyongan Porphyry deposit that is not reflected in the pre-feasibility report, the omission to disclose which makes the pre-feasibility report misleading.
- I do not hold, nor do I expect to receive, any securities or any other interests in any corporate entity, private or public, with interests in the property that are the subject of this report or in the property itself, nor do I have any business relationship with any such entity apart from a professional consulting relationship with the Issuer, nor to the best of my knowledge do I have any interest in any securities of any corporate entity mentioned in this report.
- I have read NI43-101 and Form 43-101F, and the Pre-Feasibility Report has been prepared in compliance with that instrument form.
- I consent to the filing of the Pre-Feasibility Report with any stock exchanges or other regulatory authority and any publication by them, including electronic publication in the public company files on the websites accessible by the public, of the Pre-Feasibility Report.

Dated this 13th Day of October, 2008



Signature of Qualified Person

Dexter Stewart Ferreira

Print Name of Qualified Person

Appendix I Mineral Resource Estimation



**BOYONGAN
RESOURCE MODEL**

**A REPORT ON THE METHODOLOGY EMPLOYED IN THE
ESTIMATION OF THE MINERAL RESOURCE MODEL
WITHIN THE BOYONGAN PROJECT AREA**

Compiled for Philex Gold Incorporated

August 2008

Dexter S. Ferreira

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Executive Summary

Philex Gold Inc. ("Philex"), commissioned Independent Resource Estimations ("IRES") to estimate the mineral resources for the Boyongan Cu/Au deposit, in Mindanao, Philippines. The resources were estimated using all drillhole information up to December 31st, 2007.

Detailed statistics were done and omni directional variograms generated given that no anisotropism could be detected; typical of porphyry copper deposits. Two limiting grade shell wireframes were utilized; one for 0.25%Cu and another for 0.25g/t Au. Samples were composited into regular 2.5m composites. Grade capping was applied on Cu at 5% and on Au at 10g/t.

A block size of 20m x 20m x 10m (XYZ) was used and ordinary kriging was the method of interpolation to estimate grades. A minimum of 3 samples and a maximum of 20 composites were used to generate an estimate. An initial isotropic search ellipsoid of 75m was used as the first search strategy; a range defining approximately two-thirds of the total variance.

A secondary isotropic search of larger dimensions was used as a second run, whilst estimate outside the grade shells were estimated using a restrictive isotropic search of 75m.

Cross validation tests demonstrate the final ordinary kriged model is robust and adequately mimics the trends observed in the original dataset. The data density within the Cu shell is sufficiently great enough (40m), to have a high confidence in the estimated mineral resources. The resources were only tabulated down to the -400m elevation.

At a Cu cut-off grade of 0.5%, resource calculations yield a Measured Mineral Resource of 104.5Mt at an insitu grade of 0.80% Cu, and 0.99g/t for Au; an Indicated Mineral Resource of 0.87Mt at an insitu grade of 0.80% Cu and 0.08g/t Au, for a total of Measured and Indicated Mineral Resources of 105.4Mt at an insitu grade of 0.80% Cu and 0.98g/t Au.

These Mineral Resource figures are reported in accordance to NI43-101 guidelines and conform to the CIM resource classifications.

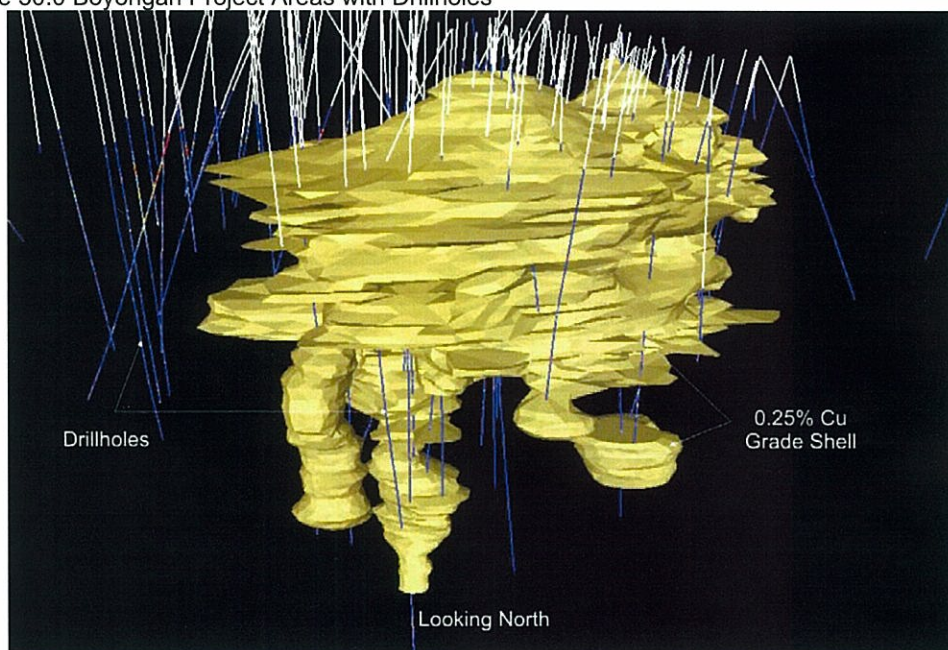
26 Objective

This document will present an overview of the methodology used and results obtained from the creation of the June, 2008 mineral resource model for the Boyongan copper/gold deposit in Mindanao, Philippines.

27 Location

The project area consists of the Boyongan deposit as outlined by a Cu grade shell (Figure 30) wireframe representing a 0.25% Cu grade threshold body. This project area is completely encompassed within the following limits: 1061200N and 1063600N, 778200E to 780800E, and -800EI to 550EI.

Figure 30.0 Boyongan Project Areas with Drillholes



28 Data Collection and Validation

The dataset available for the Boyongan project consists of diamond drillholes. Data verification was done by checking whether the FROMs and TOs were consistent for each drillhole sample. The data received was Cu (%), Au (g/t), As (ppm), specific gravity, cold acid soluble copper (CUCXPCT), hot acid soluble copper (CUHXPCT) and soluble copper (CUCXRAT). In addition, zero grades were checked for (none in database), and other obviously erroneous data such as negative grades. Drillhole numbering was checked within Datamine™ in order to ensure no duplication of collar identifiers (BHID).

The location of each surface drillhole was checked and verified by site staff as well as the lithological and assay tagging within those drillholes. Survey points denoting the locations of the holes were received and plotted in three dimensions using Datamine™ alongside images obtained from the mine site in order to assess whether or not the drillhole *collars* were in the correct place. No issues were discovered with data location.

The data used in this study involved all the drillhole samples available to the end of December 2007. In addition, the Anglo report detailing the estimation of the Boyongan (using drillhole data up to May 2007) was also received along with their blockmodel and grade shells.

28.1 Sampling

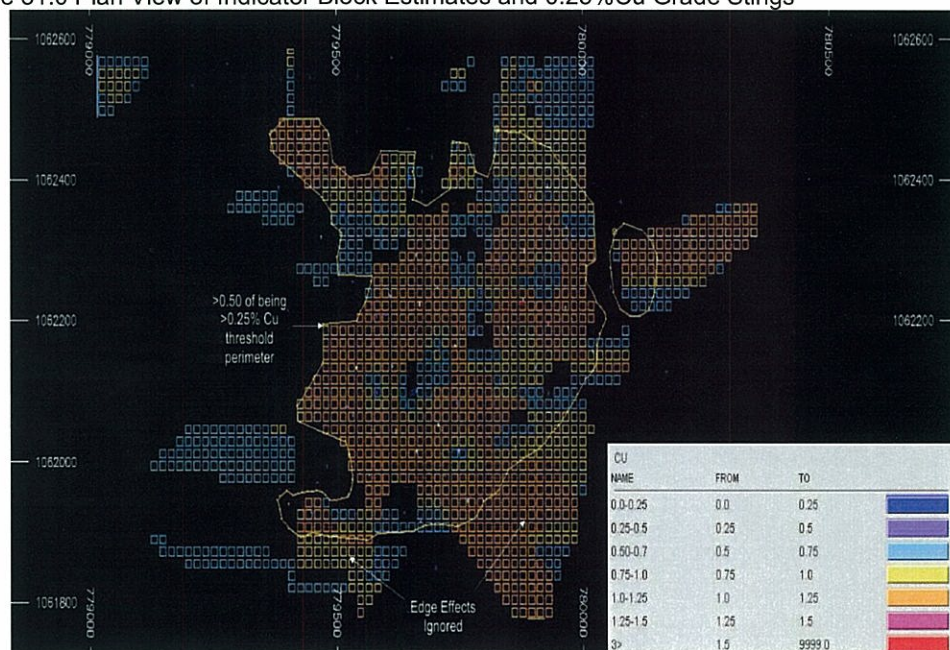
Details regarding sampling procedures and QA/QC protocols will not be covered herein given that they are thoroughly detailed in the Anglo American report (*Mssrs D.Campbell et al, Boyongan Copper/Gold Deposit Mineral Resource Evaluation and Exploration Potential, January 2008*).

29 Geology Model – Interpretation and Definition

The previous wireframes (solids) were investigated for this study. Given that there are an additional 19 drillholes in addition to the previous 142 drillholes, it was decided that there was merit in re-constructing a new 0.25%Cu grade shell. This Cu envelope was constructed in 10m plan sections and then wireframed together using Datamine™.

In order to begin this 0.25%Cu perimeter, indicator interpolation was done using the 0.25% value as a cut-off. Any sample having a value of 0.25% or greater was assigned a value of one whereas any sample having lower values were set to zero. Inverse distance squared was then used to generate the probability values. A threshold of >0.5 was used as a guide in generating the grade shell perimeter strings (Figure 31).

Figure 31.0 Plan View of Indicator Block Estimates and 0.25%Cu Grade Stings



It was noted that the previous resource work utilized an inner higher-grade copper shell at a 0.60%Cu threshold. Although the drilling has shown that copper mineralization is more variable due to the greater mobility of Cu during oxidation, for this study, it was felt that the drilling density was sufficiently high enough to adequately constrain the spread of these higher grades (see [Sample Spacing](#)); the median Euclidean spacing between composites within the 0.25%Cu shell was calculated to be 43m.

This outer Cu shell wireframe was then discretized into blocks 20m x 20m x 10m (XYZ) in size and coded for each lithological unit using the TRIFIL process within Datamine™. No subcelling was done. The project areas and related rockcodes are shown in [Table 43](#) below.

Table 43.0 Project Rock Coding

Unit	Zone	Rock Code
Host	Air	15
Oxide	0.25%Cu Shell	4
Sulphide	0.25%Cu Shell	5
Oxide	0.25g/t Au Shell	6
Sulphide	0.25g/t Au Shell	7
Oxide	Beyond Shells	2
Sulphide	Beyond Shells	3

The current outer gold grade shell of 0.25g/t was reviewed in detail and was subsequently utilized in this resource study. For this study, the current higher-grade Au shells of 1g/t were also abandoned due to sufficient data density contained within the lower-grade shell.

Once the geology model was discretized into blocks, it was reviewed on a section-by-section. The wireframes were studied in Datamine in order to verify this lack of filling and errors in the triangulation were present. The wireframe volumes (as completed in Datamine™) were then compared to the total block volume; the results are shown in Table 44 below.

Table 44.0 Volume Comparisons - Wireframes vs. Blockmodel

Zone	Wireframe (m3)	Blocks (m3)	Difference (%)
0.25% Cu Shell	140,128,308	139,908,000	-0.16%
0.25g/t Au Shell	114,046,007	113,256,000	-0.69%

As is evident in the table above, the volumetric differences between the modeled wireframes in Datamine™ and the block-discretized wireframes is negligible thus precluding the need for subcelling.

30 Statistical Analysis - Naïve Statistics

A complete set of naïve statistics was performed on the drillhole database that was contained within the following limits: 1061200N and 1063600N, 778200E to 780800E, and -800EI to 550EI. The statistics looked into the characteristics of Cu, Au, As, CUCXPCT, CUHXPCT, CUCXRAT and SG values as original samples, sample composites, as well as composites coded per rocktype. This latter dataset utilizes coding obtained from the geological model and not from the original drillhole tagging itself. The original, uncomposited and uncut dataset, presented in Table 45 show the naïve statistics for samples contained within the modeled 0.25%Cu shell wireframes for drillholes.

Table 45.0 Naïve Statistics on Uncut Samples – Inside 0.25%Cu Shell

STATISTIC	Units ¹⁰						
	%	g/t	ppm	%	%	%	-
	Cu	Au	As	CUCXPCT	CUHXPCT	CUCXRAT	SG
Number of Data	11701	11701	11523	11043	11059	11043	1249
Mean	0.59	0.84	27.78	0.52	0.37	0.52	2.19
Standard Deviation	0.55	1.84	183.02	0.31	0.46	0.31	0.24
Coeff. Of Variation	0.393	2.20	6.59	0.59	1.26	0.59	0.11
Maximum	7.91	109.00	7769.30	6.07	7.21	6.07	3.29
Upper Quartile	0.69	0.83	9.70	0.78	0.46	0.78	2.35
Median	0.45	0.41	3.00	0.60	0.25	0.60	2.21
Lower Quartile	0.31	0.20	1.00	0.20	0.08	0.20	2.08
Minimum	0.01	0.01	0.05	0.01	0.01	0.01	1.21
Number of Holes	88	88	88	87	87	87	87

¹⁰ Units apply only to the appropriate statistic.

Table 46.0 Naïve Statistics on Uncut Samples – Inside 0.25g/t Au Shell

STATISTIC	Units						
	%	g/t	ppm	%	%	%	-
	Cu	Au	As	CUCXPCT	CUHXPCT	CUCXRAT	SG
Number of Data	10367	10367	10196	9843	9872	9843	1151
Mean (%)	0.54	0.96	31.58	0.49	0.32	0.49	2.24
Standard Deviation	0.55	1.90	194.49	0.32	0.46	0.32	0.21
Coeff. Of Variation	1.02	1.97	6.16	0.66	1.44	0.66	0.09
Maximum (%)	7.91	109.00	7769.30	6.07	7.21	6.07	3.29
Upper Quartile (%)	0.62	0.95	10.90	0.78	0.40	0.78	2.39
Median (%)	0.40	0.51	2.80	0.56	0.18	0.56	2.26
Lower Quartile (%)	0.26	0.31	0.90	0.15	0.06	0.15	2.12
Minimum (%)	0.01	0.02	0.05	0.01	0.01	0.01	1.21
Number of Holes	77	77	77	77	77	77	77

When the data was imported into GSLIB, missing samples were treated as -1.0 (the notation in GSLIB). There were no zero values in the database.

31 Compositing

Statistics were then compiled on the sample lengths of drillhole data. These naïve statistics are shown in Table 47 below.

Table 47.0 Naïve Statistics on Sample Lengths

STATISTIC	Inside 0.25%Cu Shell	Inside 0.25g/t Au Shell	All Data
Number of Data	11788	10417	34699
Mean (m)	2.33	2.32	2.46
Standard Deviation	1.25	1.28	1.28
Coeff. Of Variation	0.54	0.55	0.52
Maximum (m)	74.00	74.00	81.00
Upper Quartile (m)	3.00	3.00	3.00
Median (m)	3.00	3.00	3.00
Lower Quartile (m)	1.70	1.70	2.20
Minimum (m)	0.01	0.01	0.01

Sample length statistics for the drillholes reveal mean values around 2.3m whilst the range is from 0.01 to 81.0m. A detailed look at the drillholes having sample lengths greater than 5.0m was done using Datamine™ in order to determine whether their occurrence was significant. It was revealed that only 2 samples have lengths greater than 5m and they are both not assayed. In order to preserve inherit grade variability, especially in terms of Cu and Au, it was then decided that 2.5m would be an ideal composite length for drillhole data.

The samples were thus composited at 2.5m lengths beginning at the beginning of the drillhole as constrained by each grade shell wireframe. Due to the lack of sufficient density measurements on a per sample basis, density was not utilized for weighting. The results are shown in Table 48 and Table 49 below for composites (composited within the wireframes). The composite statistics were done in order to assess whether the compositing has maintained the distribution characteristics of the original samples.

Table 48.0 Composite Statistics on Uncut Composites – Inside 0.25%Cu Shell¹¹

STATISTIC	Units						
	%	g/t	ppm	%	%	%	-
	Cu	Au	As	CUCXPCT	CUHXPCT	CUCXRAT	SG
Number of Data	10851	10851	10715	10349	10357	10349	1284
Mean (%)	0.59	0.82	26.77	0.34	0.36	0.52	2.19
Standard Deviation	0.52	1.52	153.20	0.43	0.45	0.30	0.24
Coeff. Of Variation	0.89	1.85	5.72	1.30	1.23	0.58	0.11
Maximum (%)	7.88	35.04	7769.30	7.50	7.21	5.83	3.29
Upper Quartile (%)	0.67	0.82	10.10	0.41	0.46	0.77	2.35
Median (%)	0.45	0.43	3.04	0.22	0.25	0.60	2.21
Lower Quartile (%)	0.31	0.21	1.10	0.07	0.08	0.19	2.07
Minimum (%)	0.01	0.01	0.05	0.01	0.01	0.02	1.21
Number of Holes	88	88	88	87	87	87	87

Table 49.0 Composite Statistics on Uncut Composites – Inside 0.25g/t Au Shell²

STATISTIC	Units						
	%	g/t	ppm	%	%	%	-
	Cu	Au	As	CUCXPCT	CUHXPCT	CUCXRAT	SG
Number of Data	9592	9592	9460	9220	9240	9220	1151
Mean (%)	0.54	0.95	30.60	0.30	0.32	0.49	2.24
Standard Deviation	0.53	1.57	162.58	0.44	0.44	0.32	0.21
Coeff. Of Variation	0.98	1.66	5.31	1.47	1.40	0.65	0.09
Maximum (%)	7.88	35.04	7579.56	7.50	7.21	4.69	3.29
Upper Quartile (%)	0.61	0.94	11.54	0.38	0.40	0.77	2.39
Median (%)	0.40	0.52	2.90	0.17	0.18	0.56	2.26
Lower Quartile (%)	0.27	0.33	1.00	0.05	0.06	0.15	2.12
Minimum (%)	0.01	0.03	0.05	0.01	0.01	0.02	1.21
Number of Holes	77	77	77	77	77	77	77

Note that in both datasets, the coefficient of variation (standard deviation / mean) is extremely low indicating a very low degree of variability.

31.1 Validation of Compositing

The statistics of the composites were then compared to the statistics of the uncomposited datasets for drillhole samples. This was done in order to check whether data has been gained or lost. From the tables above, the amount of drillhole data has decreased by approximately 6.0% within the Cu shell and similarly for the Au shell.

The means of both the uncomposited datasets and the composited datasets were also compared in order to ensure that they are not significantly different (see Appendix II for boxplots). A comparison of the Cu, Au, As, CUCXPCT, CUHXPCT, CUCXRAT and SG means on a per shell basis shows that there are no significant changes in the means due to compositing; differences are less than 3%.

31.2 Sample Flagging

The composites contained within the wireframes were then compared to the composites contained within the block-discretized wireframes (see Geology Model) in order to verify whether they were similar. The results from the block-discretized model are shown in Table 50 and Table 51; boxplots are shown in Appendix II. Ideally the composites should be a multiple or a sub-multiple of the estimation block height, and in this study the block height is 10m, equal to four times ("4x") the composite length. Recall that the block height was deliberately set to a small dimension in order to allow for three dimensional estimation across faults (no need for *unfaulting*) and to retain the gradational grade changes as one progresses from one defined unit to another in the final estimation; minimize smoothing perpendicular through the entire package.

¹¹ Samples have been composited within each stratigraphic unit.

Table 50.0 Composite Statistics on Uncut Samples – Inside 0.25%Cu Shell¹²

STATISTIC	Units						
	%	g/t	ppm	%	%	%	-
	Cu	Au	As	CUCXPCT	CUHXPCT	CUCXRAT	SG
Number of Data	10564	10564	10436	10116	10124	10116	1246
Mean (%)	0.59	0.82	26.68	0.34	0.37	0.52	2.19
Standard Deviation	0.51	1.43	154.24	0.43	0.44	0.30	0.24
Coeff. Of Variation	0.87	1.75	5.78	1.27	1.20	0.58	0.11
Maximum (%)	7.88	35.05	7769.30	7.50	7.21	5.83	3.29
Upper Quartile (%)	0.68	0.83	9.77	0.42	0.47	0.78	2.34
Median (%)	0.45	0.43	3.00	0.23	0.25	0.60	2.21
Lower Quartile (%)	0.32	0.21	1.10	0.08	0.09	0.20	2.07
Minimum (%)	0.01	0.01	0.05	0.01	0.01	0.02	1.21
Number of Holes	88	88	88	87	87	87	87

 Table 51.0 Composite Statistics on Uncut Samples – Inside 0.25g/t Au Shell²

STATISTIC	Units						
	%	g/t	ppm	%	%	%	-
	Cu	Au	As	CUCXPCT	CUHXPCT	CUCXRAT	SG
Number of Data	7865	7865	7743	7543	7548	7543	941
Mean (%)	0.59	1.03	31.13	0.35	0.36	0.53	2.22
Standard Deviation	0.54	1.57	175.22	0.46	0.46	0.32	0.21
Coeff. Of Variation	0.90	1.53	5.63	1.32	1.26	0.59	0.09
Maximum (%)	7.88	35.05	7769.30	7.50	7.21	5.83	3.29
Upper Quartile (%)	0.66	1.05	10.33	0.43	0.46	0.79	2.36
Median (%)	0.46	0.56	2.96	0.23	0.25	0.64	2.24
Lower Quartile (%)	0.32	0.35	1.04	0.07	0.08	0.17	2.10
Minimum (%)	0.01	0.02	0.05	0.01	0.01	0.02	1.21
Number of Holes	77	77	77	77	77	77	77

The data above is also shown in Appendix II as boxplots. The plots and the statistics clearly reveal that the sample tagging, when taken from the block-discretized geology model, has not altered the distribution characteristics of the wireframed-tagged data.

32 Sample Spacing

The Euclidean spacing between samples was examined. Overall, the distances are typically 45m. Table 21 shows the spacing between samples in three-dimensional space for all rocktypes for composited surface drillholes.

Table 52.0 Euclidean Spacing

Data	Zone	Average Meters	Maximum Meters	Lower Quart. Meters	Median Meters	Upper Quart. Meters
Samples	All Data	75.72	1081.46	35.56	52.31	78.47
2.5m Comps	0.25%Cu Oxide	40.64	117.46	26.34	41.04	53.15
2.5m Comps	0.25%Cu Sulphide	55.04	188.66	34.51	45.66	65.54

A block size of 20m x 20m x 10m was chosen to discretize the blockmodel at Boyongan. This dimension is based on not having any unsupported blocks in between 2 supported blocks (blocks pierced by drillholes). Therefore, since the median distance between drillhole samples is approximately 45m, which would imply no blocks in between drillhole pierce points. In the ideal case where there is a drillhole at either end, this would imply 3 unsupported blocks in between samples. In this study, a consistent block size was utilized throughout.

¹² Samples have been composited within each stratigraphic unit.

33 Cutting Limits

Cutting statistics were performed with the help of cumulative log probability plots, indicator correlation for lag 1 plots, coefficient of variation plots and finally percent metal contained plots. It should be noted that these are merely guidelines and that ultimately, the cutting limit chosen is a grade limit suggested by these plots.

The indicator correlation for lag 1 plots show the correlation between samples for the first lag set. Plotting this indicator against increasing minimum thresholds for Cu, Au, As, CUCXPCT, CUHXPCT, CUCXRAT and SG values leads to a line tending closer towards zero. In other words, at ever increasing thresholds of Cu, Au, As, CUCXPCT, CUHXPCT, CUCXRAT and SG values, there are fewer and fewer samples of similar grade. At this point, it indicates a lack of correlation between samples within the first lag set, and suggests an ideal cutting limit for assay values.

The coefficient of variation plots shows the change in this coefficient with increasing Cu, Au, As, CUCXPCT, CUHXPCT, CUCXRAT and SG values. A rapid change in this coefficient indicates a rapid change in the standard deviation and/or a change in the mean. This suggests an ideal cutting limit for Cu, Au, As, CUCXPCT, CUHXPCT, CUCXRAT and SG values.

Kinks, plateaus and/or changes in the cumulative log probability plots also suggest changes in populations (perhaps subpopulations) and serve as a good indicator of cutting limits for Cu, Au, As, CUCXPCT, CUHXPCT, CUCXRAT and SG values. A slightly different plot is the percent of contained metal in samples versus increasing trimming levels for metal grades. This plot enables one to check how much metal is being lost to cutting at a certain Cu, Au, As, CUCXPCT, CUHXPCT, CUCXRAT and SG values thresholds.

Table 53.0 Cutting Limits per Metal

Zone	Cu	Au
0.25%Cu Shell	5%	-
0.25g/tAu Shell	-	10g/t

Table 23 lists the impact of the cutting thresholds suggested by all methodologies described. The reasons for choosing one methodology over another is based on how well defined that limit is depicted on the appropriate plot(s).

Table 54.0 Cutting Limits per Metal

Metal	Limit	# of Original Data	# of Comps cut	% of Data
Cu (%)	5	10851	16	0.15%
Au (g/t)	10	9592	61	0.64%

The overall effect of imposing cutting limits is less than 1% - an insignificant amount.

Additional statistics were performed on the cut composited drillhole datasets and are presented in Table 55 and Table 56 below. Data shown below is also presented in Appendix II as boxplots.

Table 55.0 Composite Statistics on Cut Composites – Cu (%)

STATISTIC	Soil	Oxide	Sulphide	Cu Oxide	Cu Sulphide
Number of Data	31	107	92	7261	3303
Mean	1.69	0.31	0.32	0.64	0.47
Standard Deviation	1.74	0.20	0.21	0.57	0.29
Coeff. Of Variation	1.03	0.65	0.67	0.88	0.62
Maximum	5.00	1.01	1.54	5.00	3.13
Upper Quartile	2.15	0.34	0.37	0.73	0.57
Median	1.05	0.24	0.27	0.48	0.40
Lower Quartile	0.42	0.18	0.20	0.33	0.29
Minimum	0.17	0.03	0.02	0.02	0.01
Number of Holes	15	35	31	81	56

Table 56.0 Composite Statistics on Cut Composites – Au (g/t)

STATISTIC	Soil	Oxide	Sulphide	Cu Oxide	Cu Sulphide
Number of Data	27	162	163	5516	3699
Mean	4.01	0.68	0.57	1.13	0.61
Standard Deviation	3.99	1.13	0.38	1.56	0.58
Coeff. Of Variation	1.00	1.67	0.66	1.38	0.94
Maximum	10.00	9.40	2.47	10.00	10.00
Upper Quartile	9.52	0.57	0.69	1.19	0.71
Median	2.06	0.40	0.45	0.60	0.45
Lower Quartile	0.88	0.26	0.32	0.35	0.31
Minimum	0.21	0.05	0.12	0.03	0.03
Number of Holes	11	36	26	62	52

34 Methodology

34.1 Variography

The models were estimated using data only contained within the defined limits. Pairwise relative variograms were used in this study; therefore no data transformation was necessary. There is some argument that this variogram is not a valid covariance model and should not be used for estimation, although in many practical applications this structural measure has been used as a variogram. Some consider that mixing all data together maximizes the variability of the data set and subsequently reduces the chances of deriving an interpretable variogram structure. This practice necessitates the use of non-linear transforms such as the pairwise relative variogram, in order to derive a measure of the spatial variability of grades. Logically, segregating the deposit into stratigraphically distinct entities should reduce the variance within the data subsets allowing better and more appropriate local variograms to be developed for each of the entities modeled. This process is aimed at further reducing the risks inherent in the estimation process. The reader is referred to additional work done on this subject refuting this argument in support of pairwise relative variograms (*Robust Measures of Spatial Continuity*, Mssrs. M.Strivastava and H.M.Parker, *Geostatistics Volume I*, pp. 295 – 308), which concludes that heteroscedastic^Ω datasets in which samples have been preferentially located in areas with high values, cannot be characterized adequately with traditional sample variograms; relative variograms will produce more interpretable and reliable results.

In this study, variography was conducted on the dataset beginning at 0° and rotating clockwise in 20° increments. The first run was done with 0° dip and 0° plunge using a horizontal and vertical tolerance of ±12.5°. The lag size was varied from 50m to a maximum of 80 lags in any one direction was calculated.

An additional series of runs were done with a wider tolerance set at ±22.5°. Nugget contribution was taken from downhole variograms (see Appendix IV).

Directional variography revealed no obvious anisotropism in any direction. Omni directional variograms were then generated (see Appendix V).

34.2 Interpolation

Ordinary kriging was selected as the final estimation method of interpolating Cu, Au, As, CUCXPCT, CUHXPCT, CUCXRAT and SG values into a three-dimensional blockmodel. The block size chosen was identical to that discretizing the geological model for both models, 20 x 20 x 10 meters (Northing x Easting x Elevation). Within the project area there were 130 blocks in the X direction, 120 blocks in the Y direction and 136 blocks in the Z direction, for a total of 2,121,600 blocks. The project area consists of an area from: 1061200N and 1063600N, 778200E to 780800E, and -800E to 550E.

^Ω **Heteroscedastic** is an adjective describing a data sample or data-generating process in which the errors are drawn from different distributions for different values of the independent variables. Most commonly heteroscedasticity takes the form of changes in variance with the magnitude of X. That is, in $y = Xb + e$ that the e's vary in magnitude with the X's. If the errors are drawn from different distributions, or if higher moments of the error distributions vary systematically, these are also forms of heteroscedasticity

A minimum of 3 and a maximum of 20 composites were utilized for an estimate. Rockcodes tagged in the estimation model came from the final geology model, and not from the tagging of the composite drillhole file itself. Ordinary kriging was performed with a discretization of 2 x 2 x 1 (X,Y,Z). The search radii used approximately equaled the variogram ranges in the plane of the deposit. Boundary conditions were set to soft across the oxide/sulphide boundary for Cu, Au, As and SG; but hard for CUCXPCT, CUHXPCT and CUCXRAT. Only composites within the 0.25%Cu shell could estimate blocks within this shell; with the exception of Au which used a different grade shell (0.25g/t Au Shell) to interpolate within it only. The search strategies utilized in the ordinary kriging runs are listed in Table 24 and Table 25; the radii shown relate to ellipse dimensions used.

Table 57.0 Initial Estimation Search Strategies

Metal	Principal Direction		Minor Direction		Vertical Direction	
	Radius Meters	Azimuth/Dip Degrees	Radius Meters	Azimuth/Dip Degrees	Radius Meters	Azimuth/Dip Degrees
Cu	75.0	0°/0°	75.0	90°/0°	75.0	90°/90°
Au	75.0	0°/0°	75.0	90°/0°	75.0	90°/90°
As	75.0	0°/0°	75.0	90°/0°	75.0	90°/90°
CUCXPCT	75.0	0°/0°	75.0	90°/0°	75.0	90°/90°
CUHXPCT	75.0	0°/0°	75.0	90°/0°	75.0	90°/90°
CUCXRAT	75.0	0°/0°	75.0	90°/0°	75.0	90°/90°
SG	75.0	0°/0°	75.0	90°/0°	75.0	90°/90°

Table 58.0 Secondary Estimation Search Strategies

Unit	Principal Direction		Minor Direction		Vertical Direction	
	Radius Meters	Azimuth/Dip Degrees	Radius Meters	Azimuth/Dip Degrees	Radius Meters	Azimuth/Dip Degrees
Cu	220.0	0°/0°	220.0	90°/0°	220.0	90°/90°
Au	250.0	0°/0°	250.0	90°/0°	250.0	90°/90°
As	250.0	0°/0°	250.0	90°/0°	250.0	90°/90°
CUCXPCT	250.0	0°/0°	250.0	90°/0°	250.0	90°/90°
CUHXPCT	250.0	0°/0°	250.0	90°/0°	250.0	90°/90°
CUCXRAT	180.0	0°/0°	180.0	90°/0°	180.0	90°/90°
SG	200.0	0°/0°	200.0	90°/0°	200.0	90°/90°

Tests were previously performed to investigate the effects of certain interpolation parameters on the variability of the estimates. Firstly, the maximum number of samples utilized for an estimate was examined. In this estimation model, the maximum is set to 20 composites, with an average of 18 composites used. A number of ordinary kriging runs with various maximum sample values were done, and the average variance of each run was compared to the maximum number of samples utilized. As the maximum number of samples is increased, the change in the variance decreased. The maximum number of samples is then selected from the area where a change in slope (becomes flatter) occurs, which is in this case, anything more than 20 samples. At this point, the addition of more samples does not significantly change the variance at all. Thus, a maximum of 20 samples was chosen in order to generate an estimate.

A final series of estimation runs were done, with an isotropic search ellipsoid size of 75m. This was done for all composite outside the low grade shells (0.25%Cu and 0.25g/t Au). Once complete, the estimates were coded according to estimation schemes applied for resource categorization (see Table 26).

Table 59.0 Project Rock Coding

Zone	Rock Code		
	Measured	Indicated	Inferred
Oxide	8	4	2
Sulphide	10	5	3

Once complete, the model was exported from GSLIB into ASCII format. This file contains a header describing the fields. The block centroid location was output, along with the Cu, Au, As, CUCXPCT, CUHXPCT, CUCXRAT and SG values. The ASCII file was then imported into Datamine™ and re-

indexed using the IJGEN process and the project limits file (as defined by the project limits; see Location).

35 Validation

35.1 Cross Validation

The cross validation tests performed here were done on the model. Numerous cross validation tests were performed on this block estimation study; one of them being naïve cross-validation. This technique consists of removing one sample and using the parameters to estimate it, and then comparing it to the original sample.

This was done systematically for all samples with a final correlation, comparing estimates to actual values, being reported. This allows for the testing of the parameters utilized in the estimation process, see Table 27 lists the results.

Through practioner's experience, the Cu, Au, As, CUCXPCT, CUHXPCT, CUCXRAT and SG values indicate an adequate amount of local variability and smoothing.

Table 60.0 Naïve Cross Validation Results

Metal	Number Of Data	Correlation Coefficient
Cu	10564	0.915
Au	7865	0.912
As	10436	0.882
CUCXPCT	10116	0.936
CUHXPCT	10124	0.938
CUCXRAT	10116	0.955
SG	1263	0.747

Another test was to take all the cut composites that actually intersect an estimated block, calculate a grade weighted by length (of intersection), and then compare these grades to the estimated block grades. This test is one which done in order to verify the amount of variability and/or smoothing of the estimate.

Obviously, samples occurring within a certain estimated block should imply a grade very similar to the estimated grade itself. This test was done for all the estimated blocks, and the result is presented in Table 28 below. As a guide, a correlation of 0.8 represents an adequate level of smoothing within the estimate.

Table 61.0 Simple Validation Results

Metal	Number Of Data	Correlation Coefficient
Cu	2816	0.950
Au	2482	0.942
As	2800	0.935
CUCXPCT	2725	0.962
CUHXPCT	2725	0.965
CUCXRAT	2725	0.982
SG	1210	0.955

A comparison between the average Cu, Au, As, CUCXPCT, CUHXPCT, CUCXRAT and SG values of the cut composites and the average estimated grades of the entire estimate was done. The values are shown below in Table 29 below. The comparison was done with polygonally-declustered drillhole cut composites (declustered due to their spatiality), in order to determine how well global block grades compare with declustered input data by domain.

Table 62.0 Actual and Estimated Means

Zone	Metal	Cut Composites	Estimates
Oxide	Cu (%)	0.582	0.568
Sulphide		0.446	0.472
Oxide	Au (g/t)	0.993	0.740
Sulphide		0.609	0.533
Oxide	As (ppm)	10.458	9.635
Sulphide		58.839	54.661
Oxide	CUCXPCT%	0.385	0.384
Sulphide		0.092	0.089
Oxide	CUHXPCT%	0.417	0.416
Sulphide		0.101	0.098
Oxide	CUCXRAT%	0.631	0.654
Sulphide		0.191	0.178
Oxide	SG	2.124	2.138
Sulphide		2.338	2.336

The results indicated that the resource is slightly conservative for all elements estimated with the exception of Cu grade estimates being slightly higher than the declustered composites within the sulphide zone. Therefore, the generation of higher block estimates is not in anomalous proportions, thus indicating the preclusion of overestimation.

35.2 Residuals

The amount of residual bias was also studied by determining the difference between the actual grade and the estimated grade; a test done via naïve cross-validation. These differences are then plotted on a frequency distribution plot and the mean established. In a perfectly unbiased estimate, the mean should be zero. Table 30 shows the amount of bias for Cu, Au, As, CUCXPCT, CUHXPCT, CUCXRAT and SG estimates.

Table 63.0 Residual Analysis: Entire Project

Metal	Number Of Data	Correlation Coefficient
Cu	10564	+0.0002
Au	7865	+0.0008
As	10436	+0.1085
CUCXPCT	10116	+0.0000
CUHXPCT	10124	+0.0000
CUCXRAT	10116	+0.0001
SG	1263	+0.0002

As can be seen from Table 30, there is very little in the way of bias. The estimates have very slight negative means overall.

35.3 Trend Analysis

Once the blockmodel was complete, a study referred to as trend (or "swath") analysis was undertaken. This work involves the examination of the composites used in the estimation process and comparing them to the final blockmodel. Various trends are checked, Northings versus Cu, Au, As, CUCXPCT, CUHXPCT, CUCXRAT and SG values, Eastings versus Cu, Au, As, CUCXPCT, CUHXPCT, CUCXRAT and SG values and Elevation versus Cu, Au, As, CUCXPCT, CUHXPCT, CUCXRAT and SG values. The idea here is that whatever trends are present within the dataset (2.5m cut composites) must also be reflected in the final blockmodel. The results are presented in Figure 32 to Figure 38 below.

Figure 32.0 Trend Analysis in Cu Shell – Cu (%)

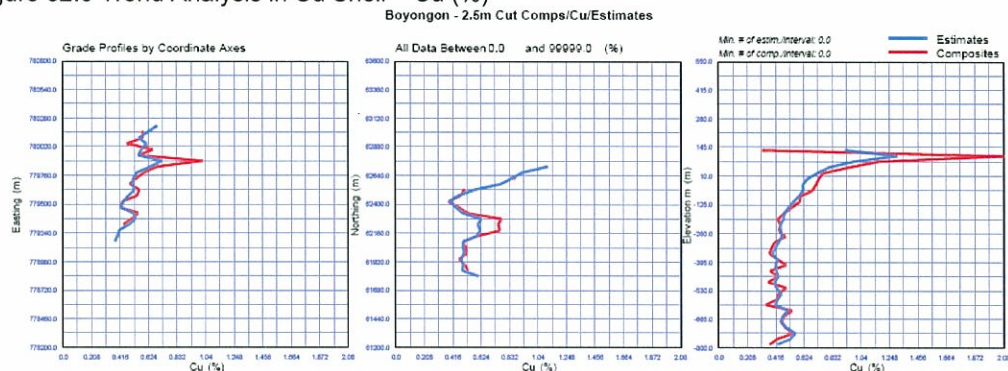


Figure 33.0 Trend Analysis in Au Shell – Au (g/t)

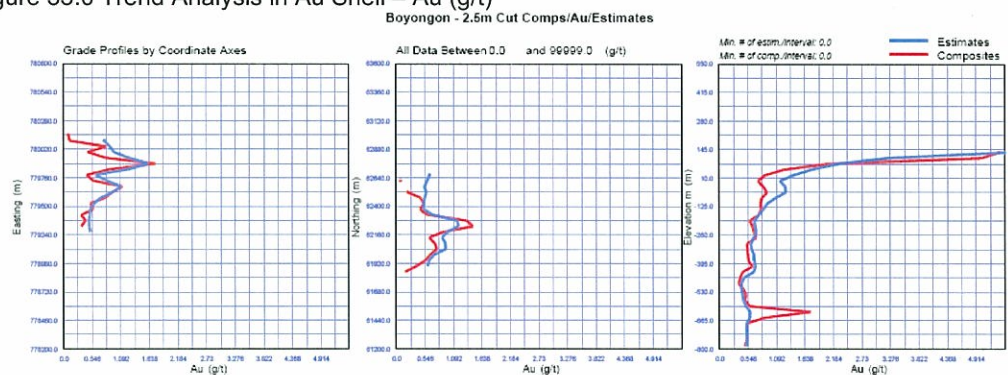


Figure 34.0 Trend Analysis in Cu Shell – As (ppm)

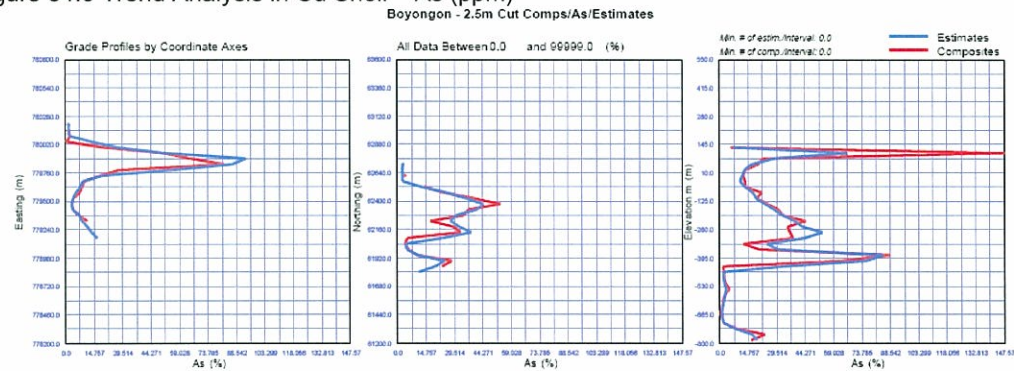


Figure 35.0 Trend Analysis in Cu Shell - CUCXPCT (%)

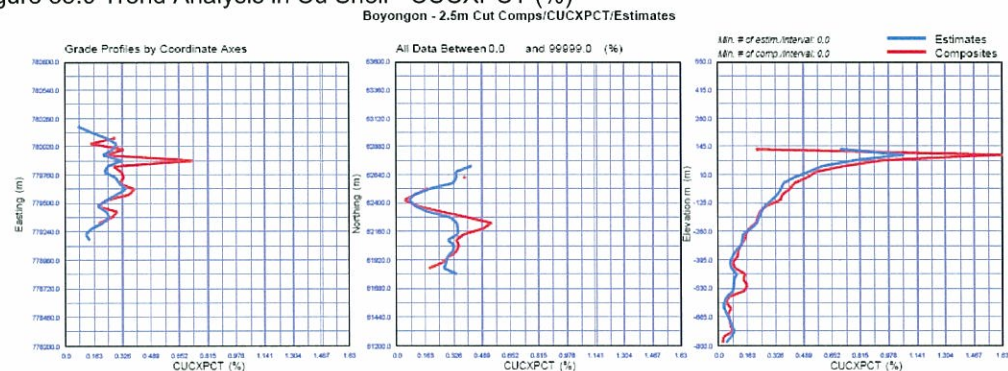


Figure 36.0 Trend Analysis in Cu Shell – CUHPCT (%)

Boyongon - 2.5m Cut Comps/CUHPCT/Estimates

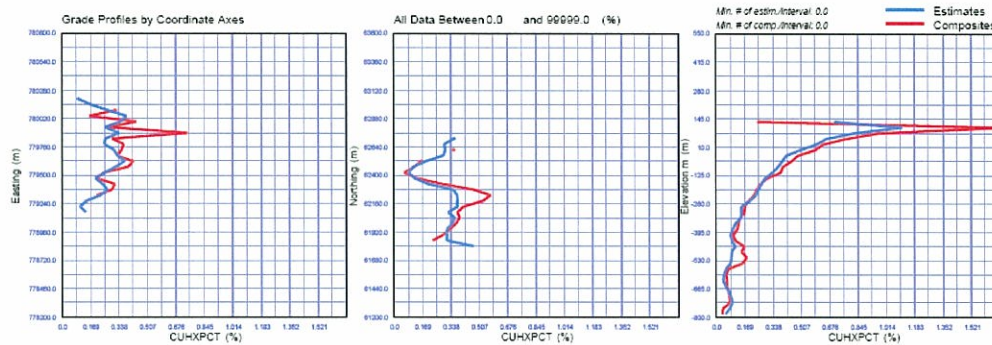


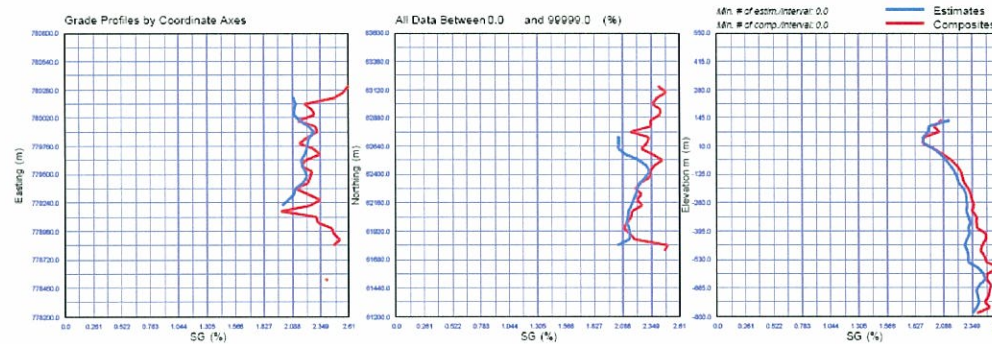
Figure 37.0 Trend Analysis in Cu Shell - CUCXRAT (%)

Boyongon - 2.5m Cut Comps/CUCXRAT/Estimates



Figure 38.0 Trend Analysis in Cu Shell - SG

Boyongon - 2.5m Cut Comps/SG/Estimates



The trends indicate that the estimated blockmodel closely follows the trends as present in the composites – in all three directions; Northing, Easting and Elevation. Visual checks of the estimation methodologies revealed that ordinary kriging showed excellent grade variability.

36 Specific Gravity

The estimated specific gravity, as estimated on a per block basis, was applied to each block in order to convert volume into tonnes.

37 Classification

The resources estimated within the Boyongan area are classified as measured, indicated and inferred. The criteria utilized were as follows: block estimates estimated within the first variogram

range of the double structure spherical models were classified as being within the measured category. This range of 75m approximately represents two thirds of the total variance. Resources estimated via the second structure of the double spherical model were classified as indicated.

Resources estimated outside the Cu and Au grade shells (0.25%Cu and 0.25g/tAu respectively) were classified as inferred.

38 Results

Results were tabulated at various Cu% thresholds and are presented below for Measured Resources separately for the oxide and sulphide zones; see Table 64 and Table 65 down to the -400m elevation.

Table 64.0 Measured Resources - Oxide

Cut-off Cu (%)	Tonnes	Cu%	Au (g/t)	As (ppm)	ASCu (%)
0.00	169,408,000	0.569	0.743	9.677	0.384
0.10	169,320,000	0.569	0.743	9.679	0.385
0.20	168,553,000	0.571	0.742	9.681	0.386
0.30	150,578,000	0.608	0.787	9.978	0.413
0.40	111,601,000	0.698	0.909	11.509	0.480
0.50	74,053,000	0.826	1.075	14.038	0.570
0.60	53,094,000	0.936	1.208	16.029	0.648
0.70	36,662,000	1.065	1.354	18.910	0.734
0.80	25,825,000	1.199	1.497	22.161	0.825
0.90	18,703,000	1.334	1.642	24.850	0.906
1.00	13,668,000	1.477	1.832	27.140	0.990

Table 65.0 Measured Resources - Sulphide

Cut-off Cu (%)	Tonnes	Cu%	Au (g/t)	As (ppm)	ASCu (%)
0.00	95,393,000	0.482	0.535	65.278	0.098
0.10	95,393,000	0.482	0.535	65.278	0.098
0.20	94,747,000	0.484	0.536	65.570	0.099
0.30	82,634,000	0.515	0.567	69.391	0.105
0.40	52,039,000	0.613	0.666	84.598	0.131
0.50	30,471,000	0.732	0.766	115.664	0.167
0.60	19,445,000	0.839	0.843	135.425	0.199
0.70	12,761,000	0.938	0.918	150.720	0.228
0.80	8,190,000	1.046	0.893	175.236	0.248
0.90	5,469,000	1.144	0.925	204.522	0.253
1.00	3,737,000	1.233	1.061	254.342	0.223

The total Measured Mineral Resources are shown in Table 66 below.

Table 66.0 Measured Resources - Total

Cut-off Cu (%)	Tonnes	Cu%	Au (g/t)	As (ppm)	ASCu (%)
0.00	264,801,000	0.538	0.668	29.707	0.281
0.10	264,713,000	0.538	0.668	29.715	0.282
0.20	263,300,000	0.540	0.668	29.792	0.283
0.30	233,212,000	0.575	0.709	31.030	0.304
0.40	163,640,000	0.671	0.832	34.752	0.369
0.50	104,524,000	0.799	0.985	43.664	0.453
0.60	72,539,000	0.910	1.110	48.035	0.528
0.70	49,423,000	1.032	1.241	52.943	0.603
0.80	34,015,000	1.162	1.352	59.018	0.686
0.90	24,172,000	1.291	1.480	65.501	0.758
1.00	17,405,000	1.425	1.666	75.922	0.825

Results were tabulated at various Cu% thresholds and are presented below for Indicated Resources separately for the oxide and sulphide zones; see Table 67 and Table 68.

Table 67.0 Indicated Resources - Oxide

Cut-off Cu (%)	Tonnes	Cu%	Au (g/t)	As (ppm)	ASCu (%)
0.00	3,631,000	0.359	0.044	4.118	0.227
0.10	3,631,000	0.359	0.044	4.118	0.227
0.20	3,574,000	0.362	0.044	3.790	0.230
0.30	2,271,000	0.410	0.050	3.415	0.266
0.40	790,000	0.541	0.059	3.472	0.335
0.50	253,000	0.746	0.081	3.372	0.379
0.60	143,000	0.916	0.009	3.551	0.395
0.70	99,000	1.036	0.013	3.205	0.389
0.80	75,000	1.130	0.009	2.658	0.446
0.90	75,000	1.130	0.009	2.658	0.446
1.00	58,000	1.186	0.012	2.722	0.481

Table 68.0 Indicated Resources - Sulphide

Cut-off Cu (%)	Tonnes	Cu%	Au (g/t)	As (ppm)	ASCu (%)
0.00	3,470,000	0.446	0.025	22.457	0.099
0.10	3,470,000	0.446	0.025	22.457	0.099
0.20	3,452,000	0.448	0.025	22.506	0.099
0.30	2,902,000	0.481	0.028	25.445	0.101
0.40	1,664,000	0.582	0.038	39.729	0.122
0.50	617,000	0.816	0.076	87.313	0.172
0.60	461,000	0.912	0.089	107.737	0.196
0.70	342,000	1.007	0.112	120.776	0.225
0.80	297,000	1.045	0.091	121.324	0.231
0.90	262,000	1.069	0.094	123.050	0.225
1.00	226,000	1.087	0.100	107.245	0.244

The total Indicated Mineral Resources are shown in Table 69 below.

Table 69.0 Indicated Resources - Total

Cut-off Cu (%)	Tonnes	Cu%	Au (g/t)	As (ppm)	ASCu (%)
0.00	7,101,000	0.402	0.035	13.080	0.164
0.10	7,101,000	0.402	0.035	13.080	0.164
0.20	7,026,000	0.404	0.035	12.986	0.166
0.30	5,173,000	0.450	0.038	15.774	0.173
0.40	2,454,000	0.569	0.045	28.057	0.191
0.50	870,000	0.796	0.077	62.903	0.232
0.60	604,000	0.913	0.070	83.070	0.243
0.70	441,000	1.014	0.090	94.383	0.262
0.80	372,000	1.062	0.074	97.399	0.274
0.90	337,000	1.083	0.075	96.257	0.274
1.00	284,000	1.107	0.082	85.899	0.292

Total Measured and Indicated resources, for the Boyongan deposit, are shown in Table 70 and Table 71 for the oxide and sulphide zones respectively.

Table 70.0 Measured and Indicated Resources - Oxide

Cut-off Cu (%)	Tonnes	Cu%	Au (g/t)	As (ppm)	ASCu (%)
0.00	173,039,000	0.565	0.728	9.560	0.381
0.10	172,951,000	0.565	0.728	9.562	0.382
0.20	172,127,000	0.567	0.728	9.559	0.383
0.30	152,849,000	0.605	0.776	9.880	0.411
0.40	112,391,000	0.697	0.903	11.453	0.479
0.50	74,306,000	0.826	1.072	14.002	0.569
0.60	53,237,000	0.936	1.205	15.995	0.647
0.70	36,761,000	1.065	1.350	18.868	0.733
0.80	25,900,000	1.199	1.493	22.105	0.824
0.90	18,778,000	1.333	1.635	24.761	0.904
1.00	13,726,000	1.476	1.824	27.037	0.988

Table 71.0 Measured and Indicated Resources - Sulphide

Cut-off Cu (%)	Tonnes	Cu%	Au (g/t)	As (ppm)	ASCu (%)
0.00	98,863,000	0.481	0.517	63.775	0.098
0.10	98,863,000	0.481	0.517	63.775	0.098
0.20	98,199,000	0.483	0.518	64.056	0.099
0.30	85,536,000	0.514	0.549	67.900	0.105
0.40	53,703,000	0.612	0.647	83.208	0.131
0.50	31,088,000	0.734	0.752	115.101	0.167
0.60	19,906,000	0.841	0.826	134.784	0.199
0.70	13,103,000	0.940	0.897	149.938	0.228
0.80	8,487,000	1.046	0.865	173.349	0.247
0.90	5,731,000	1.141	0.887	200.797	0.252
1.00	3,963,000	1.225	1.006	245.953	0.224

The total Indicated Mineral Resources are shown in Table 69 below.

Table 72.0 Measured and Indicated Resources - Total

Cut-off Cu (%)	Tonnes	Cu%	Au (g/t)	As (ppm)	ASCu (%)
0.00	271,902,000	0.534	0.652	29.273	0.278
0.10	271,814,000	0.534	0.651	29.280	0.279
0.20	270,326,000	0.536	0.651	29.356	0.280
0.30	238,385,000	0.572	0.694	30.699	0.301
0.40	166,094,000	0.669	0.820	34.653	0.366
0.50	105,394,000	0.799	0.977	43.823	0.451
0.60	73,143,000	0.910	1.102	48.324	0.525
0.70	49,864,000	1.032	1.231	53.310	0.600
0.80	34,387,000	1.161	1.338	59.433	0.682
0.90	24,509,000	1.288	1.460	65.924	0.752
1.00	17,689,000	1.420	1.641	76.082	0.817

Total Inferred Mineral Resources are shown in Table 73 and Table 74 for oxide and sulphide zones respectively.

Table 73.0 Inferred Resources - Oxide

Cut-off Cu (%)	Tonnes	Cu%	Au (g/t)	As (ppm)	ASCu (%)
0.00	84,140,000	0.201	0.214	9.871	0.102
0.10	68,362,000	0.233	0.242	7.069	0.118
0.20	34,285,000	0.308	0.285	7.023	0.152
0.30	8,293,000	0.545	0.305	10.832	0.243
0.40	4,457,000	0.725	0.350	13.738	0.327
0.50	2,940,000	0.868	0.400	17.274	0.410
0.60	2,116,000	0.991	0.442	20.973	0.489
0.70	1,652,000	1.092	0.486	24.176	0.550
0.80	1,334,000	1.174	0.529	27.357	0.604
0.90	1,038,000	1.267	0.560	30.254	0.668
1.00	825,000	1.352	0.570	31.054	0.712

Table 74.0 Inferred Resources - Sulphide

Cut-off Cu (%)	Tonnes	Cu%	Au (g/t)	As (ppm)	ASCu (%)
0.00	310,114,000	0.221	0.243	16.378	0.030
0.10	249,294,000	0.260	0.287	17.605	0.034
0.20	145,414,000	0.338	0.369	19.456	0.042
0.30	54,663,000	0.497	0.538	11.634	0.061
0.40	29,541,000	0.631	0.666	11.018	0.086
0.50	16,888,000	0.771	0.749	12.274	0.120
0.60	10,676,000	0.903	0.818	13.881	0.161
0.70	7,673,000	1.002	0.817	16.226	0.202
0.80	5,013,000	1.137	0.744	21.923	0.276
0.90	3,693,000	1.244	0.748	27.202	0.337
1.00	2,690,000	1.355	0.742	34.542	0.415

The total Inferred Mineral Resources are shown in Table 75 below.

Table 75.0 Inferred Resources - Total

Cut-off Cu (%)	Tonnes	Cu%	Au (g/t)	As (ppm)	ASCu (%)
0.00	394,254,000	0.217	0.237	14.989	0.045
0.10	317,656,000	0.254	0.277	15.338	0.052
0.20	179,699,000	0.332	0.353	17.084	0.063
0.30	62,956,000	0.503	0.507	11.528	0.085
0.40	33,998,000	0.643	0.625	11.375	0.118
0.50	19,828,000	0.785	0.697	13.015	0.163
0.60	12,792,000	0.918	0.756	15.054	0.215
0.70	9,325,000	1.018	0.758	17.634	0.264
0.80	6,347,000	1.145	0.699	23.065	0.345
0.90	4,731,000	1.249	0.707	27.872	0.410
1.00	3,515,000	1.354	0.702	33.723	0.485

The grand total of all resources, Measured, Indicated and Inferred, are shown in Table 76.

Table 76.0 Measured, Indicated and Inferred Resources - Total

Cut-off Cu (%)	Tonnes	Cu%	Au (g/t)	As (ppm)	ASCu (%)
0.00	666,156,000	0.346	0.406	20.819	0.140
0.10	589,470,000	0.383	0.449	21.766	0.156
0.20	450,025,000	0.455	0.532	24.455	0.193
0.30	301,341,000	0.558	0.655	26.694	0.256
0.40	200,092,000	0.665	0.787	30.698	0.324
0.50	125,222,000	0.796	0.933	38.945	0.405
0.60	85,935,000	0.911	1.050	43.372	0.479
0.70	59,189,000	1.030	1.157	47.690	0.547
0.80	40,734,000	1.159	1.238	53.766	0.629
0.90	29,240,000	1.282	1.339	59.767	0.697
1.00	21,204,000	1.409	1.485	69.061	0.762

39 Conclusion

The Boyongan global resource model is now complete. Statistics demonstrate the final estimated ordinary kriged model is fairly robust and adequately mimics the trends observed in the original dataset.

Given the quality of the geological model definition and available assay and survey information, I believe that this model is representative of the distribution and grades within the Boyongan deposit as presented.

The methodology applied was ordinary kriging and the resultant blockmodel has proven to be very robust for all metals estimated. Cross validation values indicate that there is an adequate level of confidence in the final blockmodel. The resources were only tabulated down to the -400m elevation.

At a Cu cut-off grade of 0.5%, resource calculations yield a Measured Mineral Resource of 104.5Mt at an insitu grade of 0.80% Cu, and 0.99g/t for Au; an Indicated Mineral Resource of 0.87Mt at an insitu grade of 0.80% Cu and 0.08g/t Au, for a total of Measured and Indicated Mineral Resources of 105.4Mt at an insitu grade of 0.80% Cu and 0.98g/t Au.

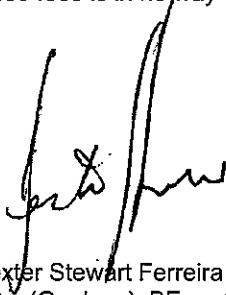
These Mineral Resource figures are reported in accordance to NI43-101 guidelines and conform to the CIM resource classifications.

This study attempted to generate a blockmodel with metal values that are representative of the insitu resources of the Boyongan deposit. Given the amount and quality of data available for resource estimation, and the data received, it is believed that the resultant model is representative of the insitu resources.

40 **Qualifications, Experience and Independence**

The author and practioner of this resource estimation report, Mr. Dexter Ferreira, is a senior geostatistician with over 18 years experience in project evaluation internationally, including extensive involvement with mineral projects throughout South America and Africa. He is a member of the South African Council for Natural Scientific Professions, and qualifies as an 'Expert', 'Competent Person' and 'Qualified Person' as defined in National Instrument 43-101 and the JORC Code.

The author of this report has not or has not previously had any material interest in Philex Gold Inc. ("Philex") or the mineral properties in which Philex has an interest. The relationship with Philex is solely one of professional association between client and independent consultant. This report is prepared in return for professional fees based upon agreed commercial rates and the payment of these fees is in no way contingent on the results of this report.



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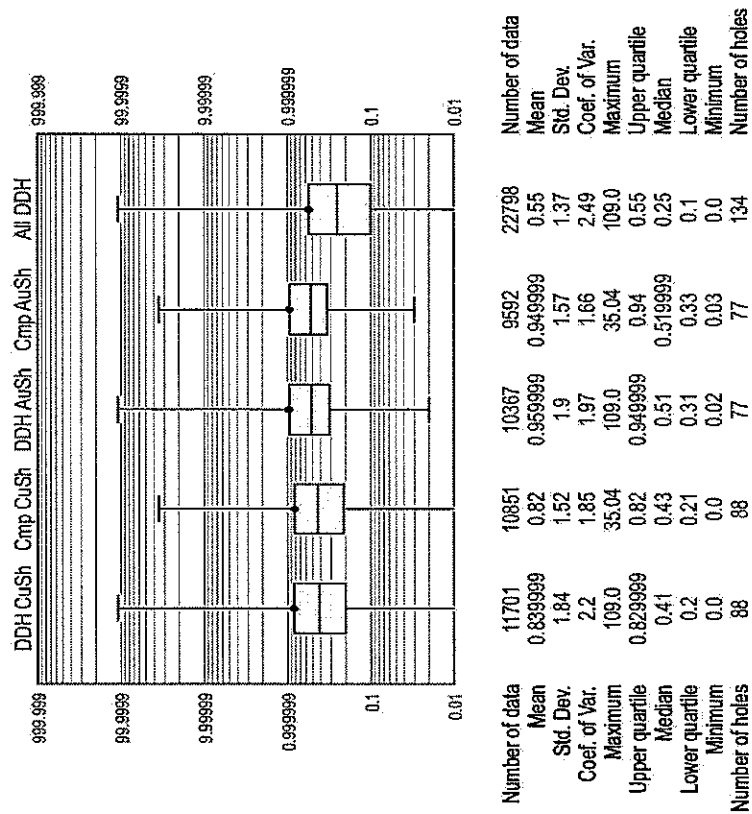
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Appendix II Boxplots

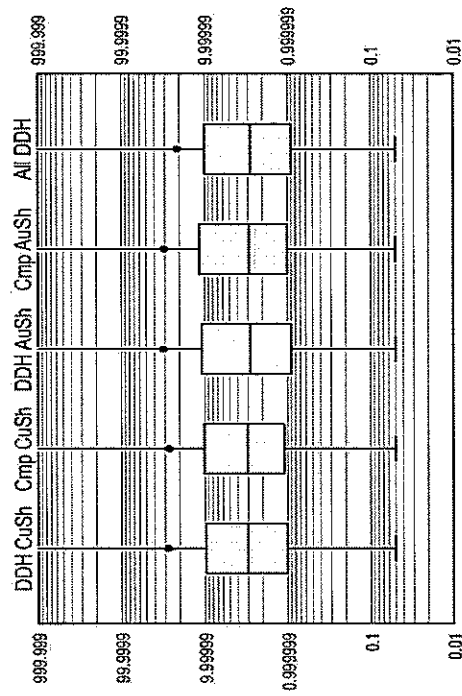
Cu % - Samples/2.5m Comps/Uncut



Au g/t - Samples/2.5m Comps/Uncut

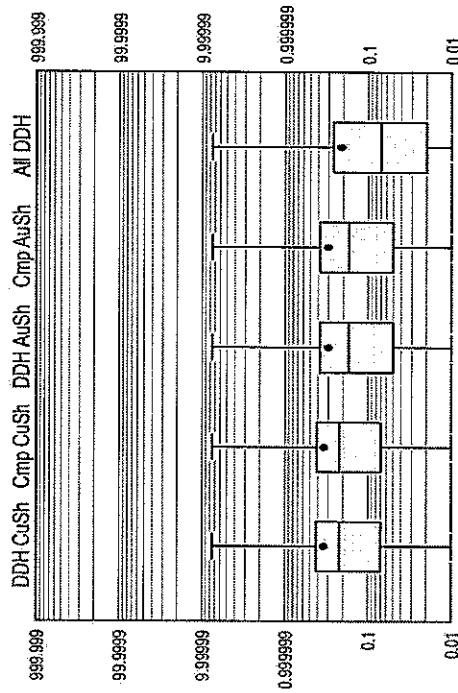


As ppm - Samples/2.5m Comps/Uncut



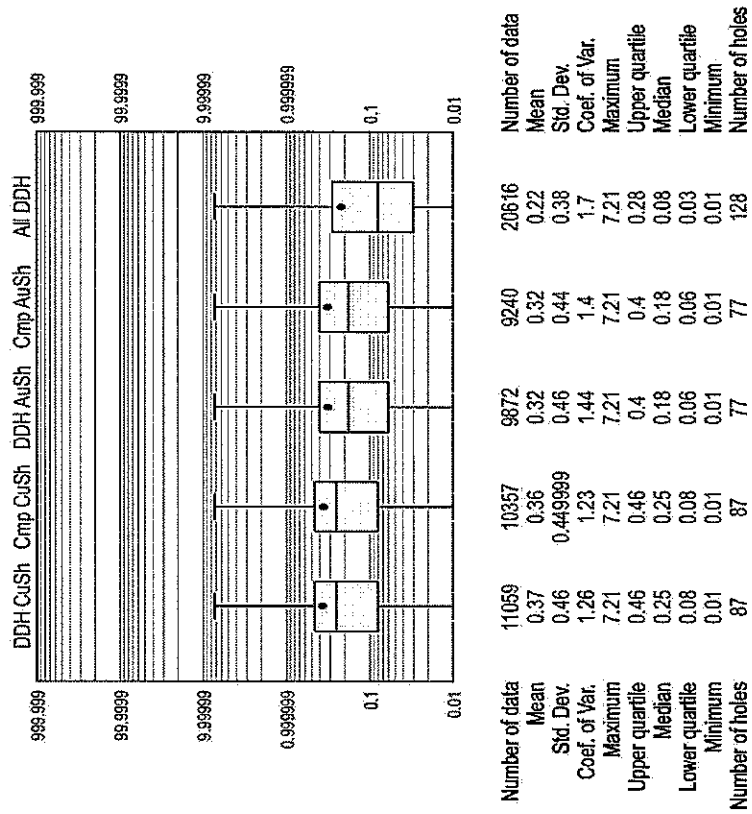
Number of data	11523	10715	10196	9460	22254	Number of data
Mean	27.78	26.77	31.58	30.6	21.13	Mean
Std. Dev.	183.02	153.2	194.49	162.58	134.5	Std. Dev.
Coef. of Var.	6.59	5.71999	6.15999	5.31	6.36999	Coef. of Var.
Maximum	7769.29	7769.29	7769.29	7579.56	7769.29	Maximum
Upper quartile	9.69999	10.1	10.9	11.54	10.0	Upper quartile
Median	3.0	3.04	2.8	2.9	2.8	Median
Lower quartile	1.0	1.1	0.899999	1.0	0.899999	Lower quartile
Minimum	0.05	0.05	0.05	0.05	0.05	Minimum
Number of holes	88	88	77	77	133	Number of holes

CUCXPCT % - Samples/2.5m Comps/Uncut

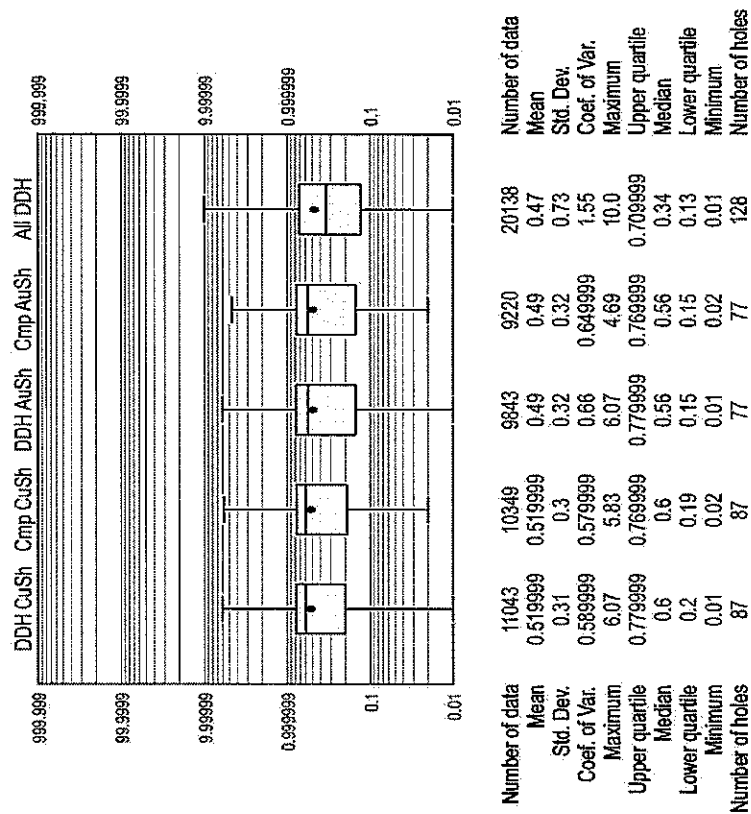


Number of data					Number of data				
Mean	0.34	11043	10349	9843	9220	20138	Mean	0.21	20138
Std. Dev.	0.44	0.44	0.43	0.3	0.3	0.21	Std. Dev.	0.37	20138
Coef. of Var.	1.34	1.3	1.3	1.51	1.47	1.76	Coef. of Var.	1.76	20138
Maximum	7.5	7.5	7.5	7.5	7.5	7.5	Maximum	7.5	20138
Upper quartile	0.41	0.41	0.41	0.38	0.38	0.26	Upper quartile	0.26	20138
Median	0.22	0.22	0.22	0.17	0.17	0.07	Median	0.07	20138
Lower quartile	0.07	0.07	0.07	0.05	0.05	0.02	Lower quartile	0.02	20138
Minimum	0.01	0.01	0.01	0.01	0.01	0.01	Minimum	0.01	20138
Number of holes	87	87	87	77	77	128	Number of holes	128	20138

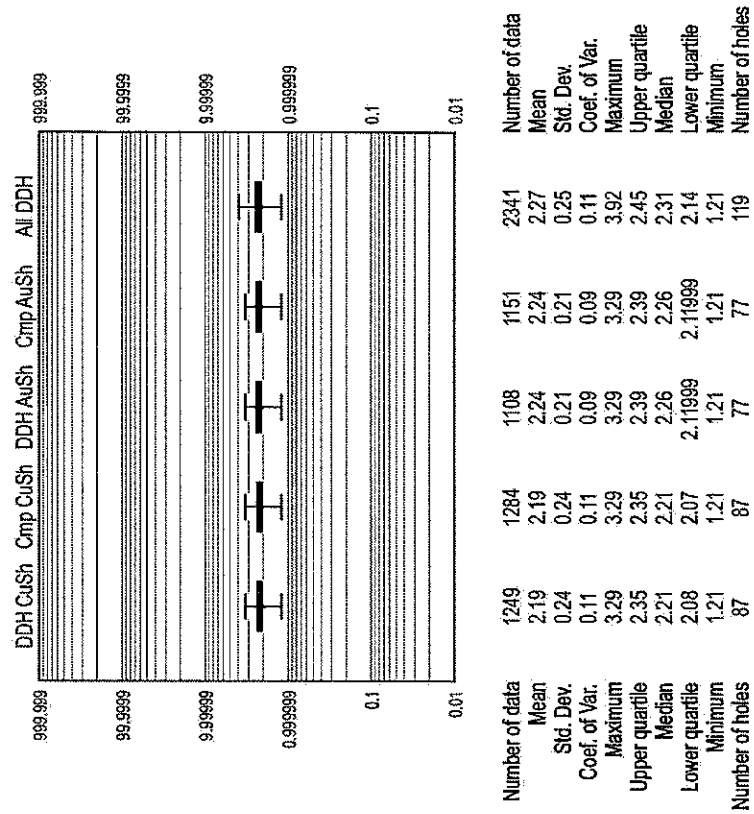
CUHXPCT % - Samples/2.5m Comps/Uncut



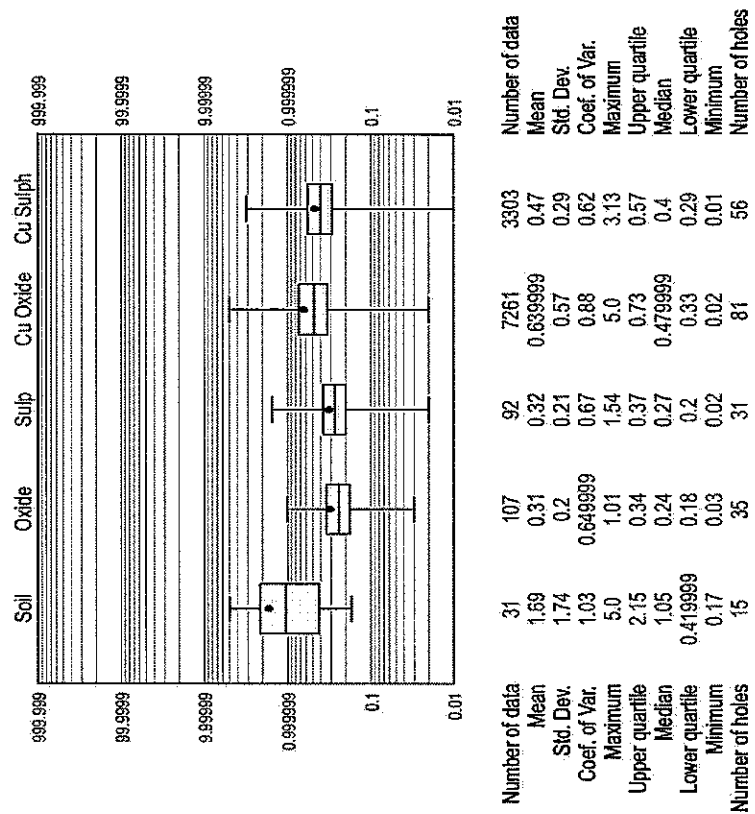
CUCXRAT % - Samples/2.5m Comps/Uncut



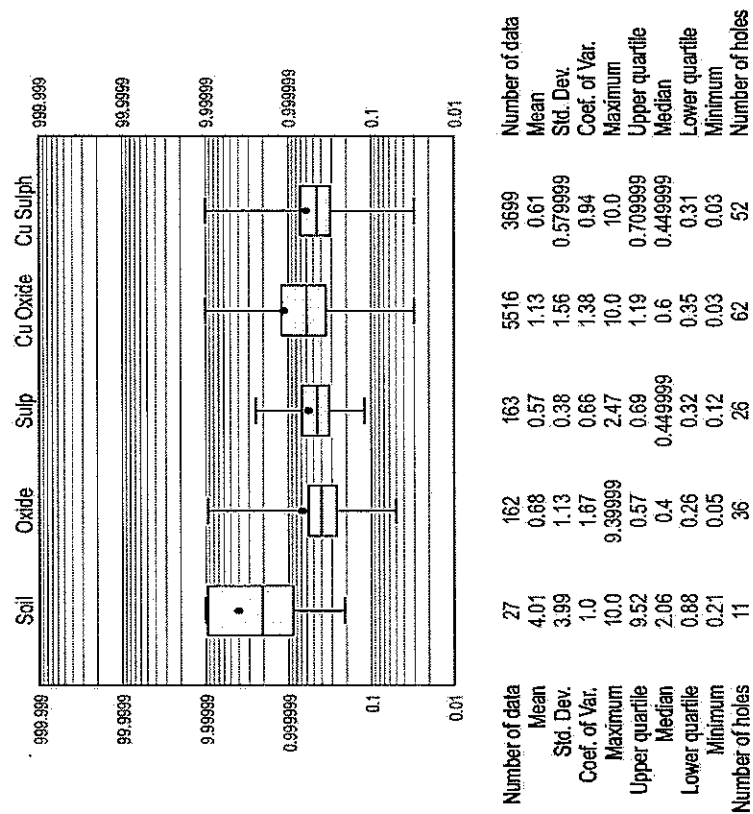
SG - Samples/2.5m Comps/Uncut



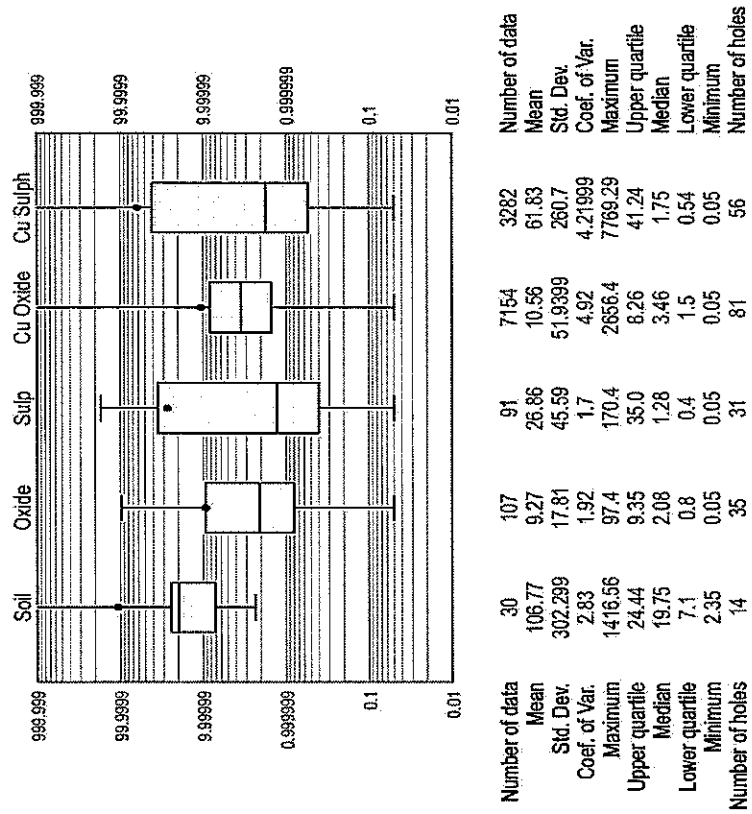
Cu % - 2.5m Comps/Cut/Litho



Au g/t - 2.5m Comps/Cut/Litho

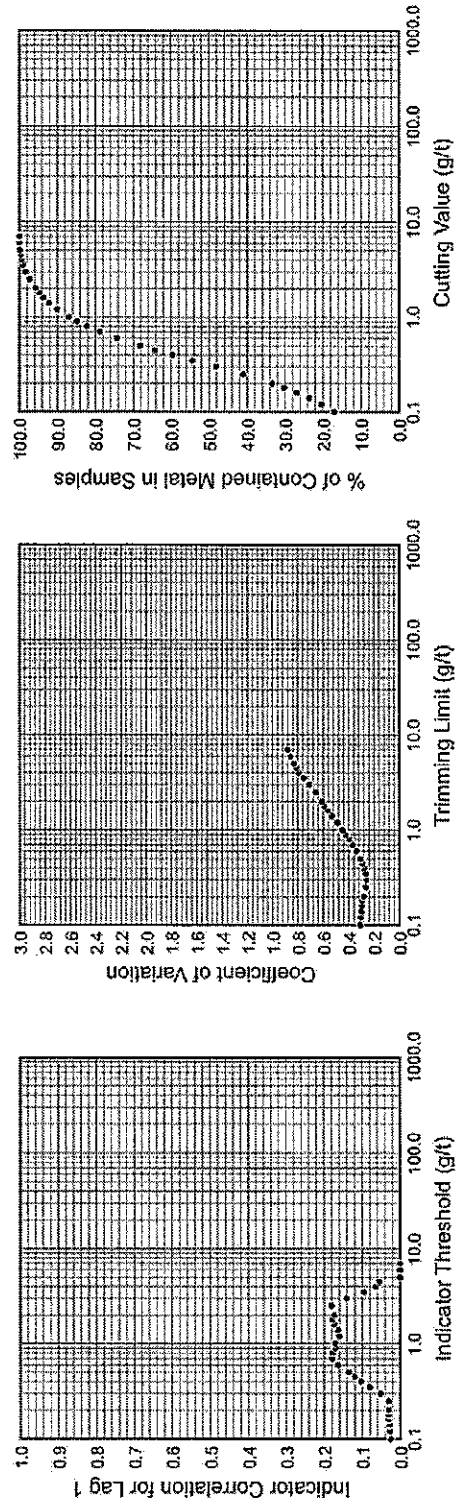


As ppm - 2.5m Comps/Cut/Litho

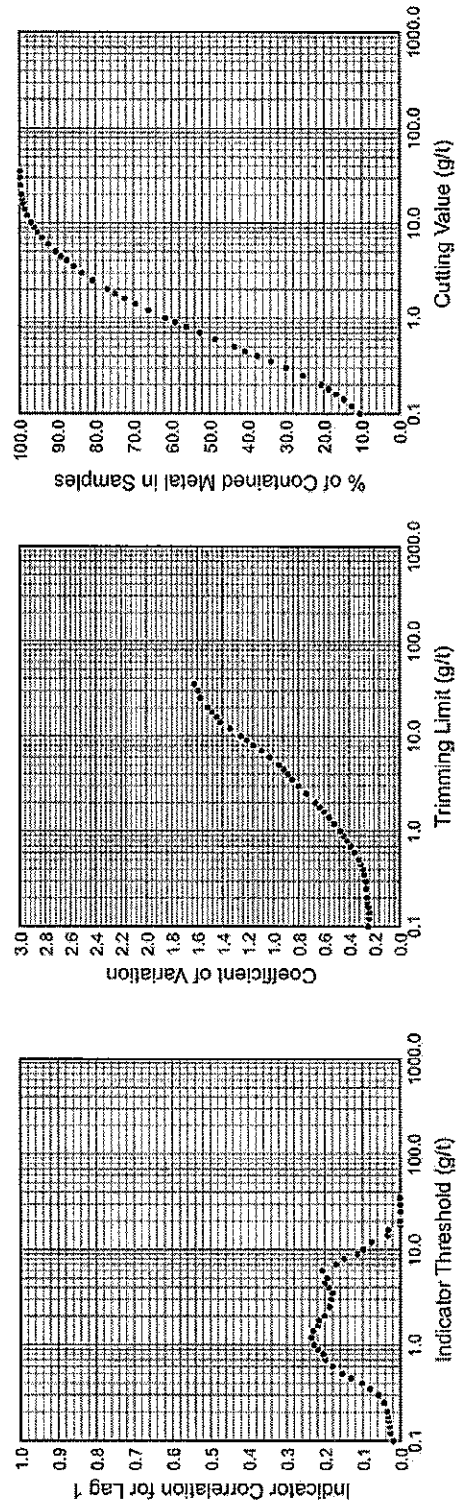


Appendix III Cutting Statistics

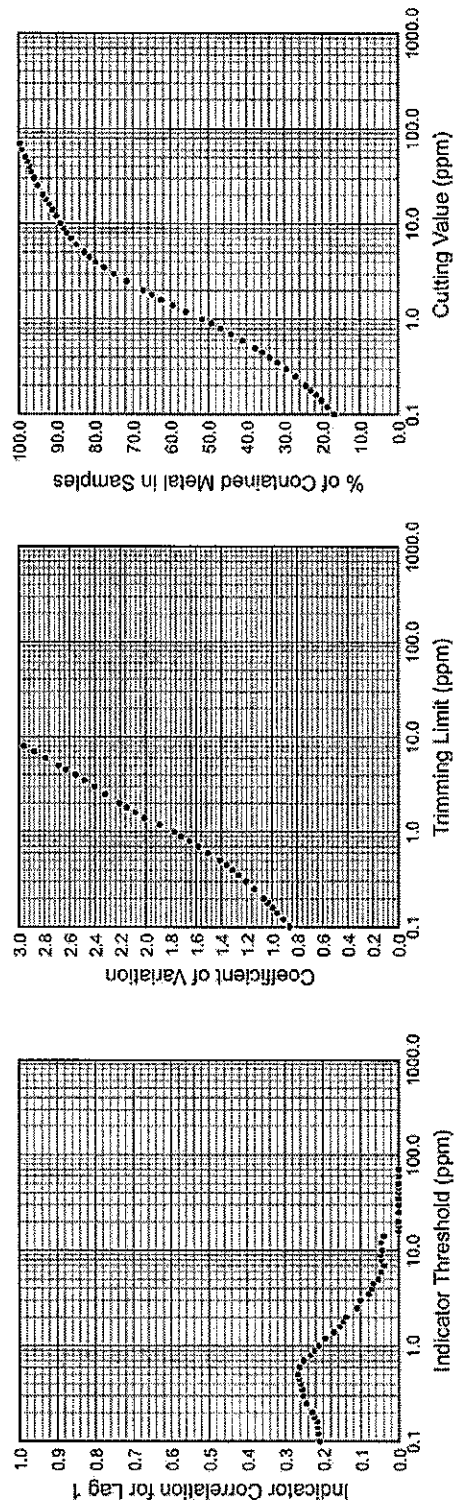
Cutting Statistics Boyongon June 2008 - Cu% - Cu Shell



Cutting Statistics
Boyongon June 2008 - Au g/t - Au Shell



Cutting Statistics Boyongon June 2008 - As (ppm/100) - Cu Shell

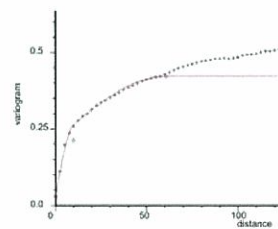


Appendix IV Down The Hole Variograms

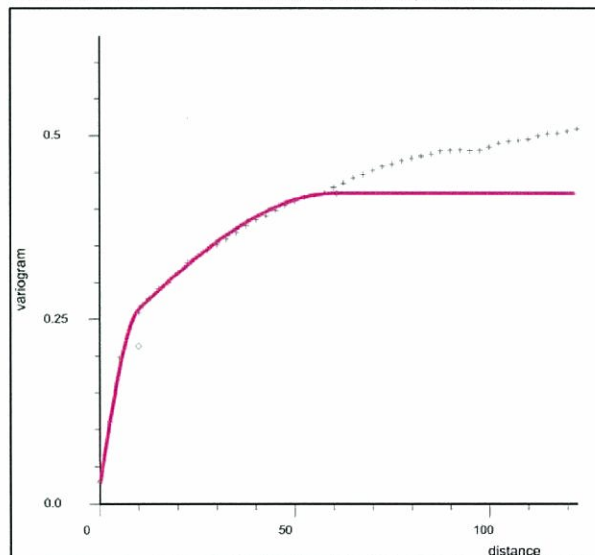
PROJECT: Boyongon June 2008
VARIOGRAM 1: DTH RLP CU 2.5m Comps/Cut/Cu/Cu Shell
LAGS: 50 of 2.5

Direction Number 1

Azimuth = 0.0 +/- 90.0
Plunge = 0.0 +/- 90.0



DIRECTION Number 1 DTH RLP CU 2.5m Comps/Cut/Cu/Cu Shell



Variogram Model:

Nugget effect = 0.0290

Directions:

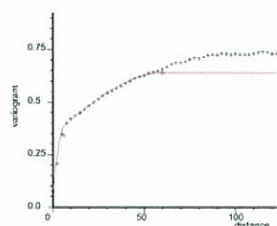
1 SPH: c1 = 0.184 a1 = 10.0

1 SPH: c2 = 0.209 a2 = 60.7

PROJECT: Boyongon June 2008
VARIOGRAM 1: DTH RLP AU 2.5m Comps/Cut/Au/Au Shell
LAGS: 50 of 2.5

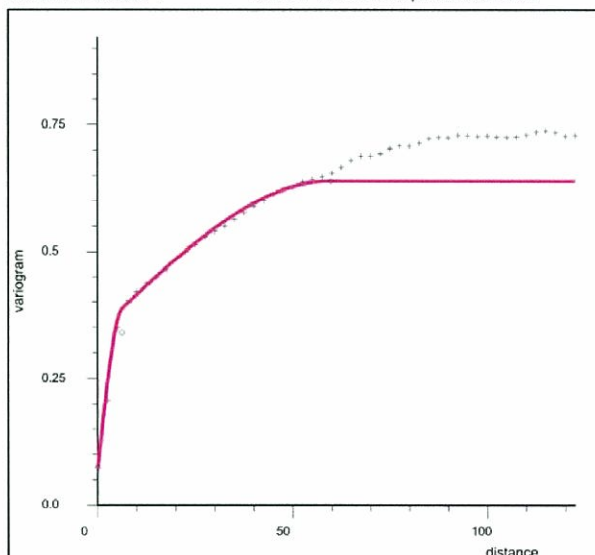
Direction Number 1

Azimuth = 0.0 +/- 90.0
Plunge = 0.0 +/- 90.0



DIRECTION Number 1

DTH RLP AU 2.5m Comps/Cut/Au/Au Shell



Variogram Model:

Nugget effect = 0.0735

Directions:

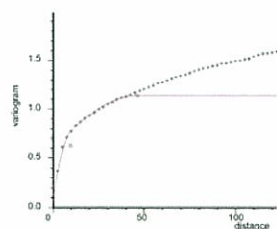
1 SPH: c1 = 0.267 a1 = 6.26

1 SPH: c2 = 0.299 a2 = 59.9

PROJECT: Boyongon June 2008
VARIOGRAM 1: DTH RLP AS 2.5m Comps/Cut/As/Cu Shell
LAGS: 50 of 2.5

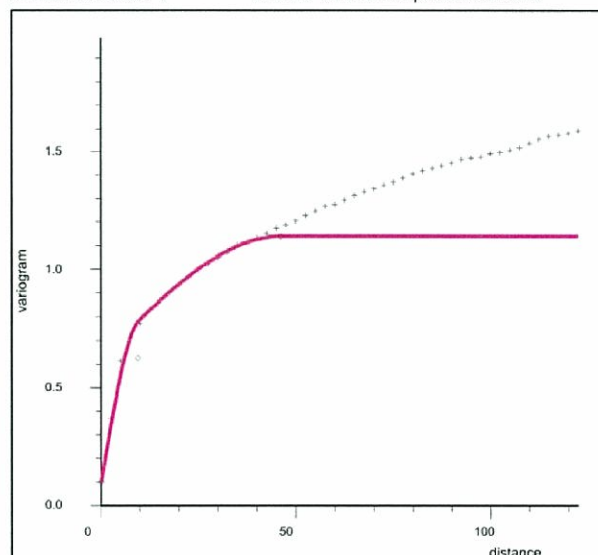
Direction Number 1

Azimuth = 0.0 +/- 90.0
Plunge = 0.0 +/- 90.0



DIRECTION Number 1

DTH RLP AS 2.5m Comps/Cut/As/Cu Shell



Variogram Model:

Nugget effect = 0.0991

Directions:

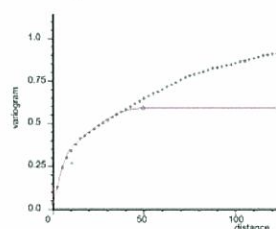
1 SPH: c1 = 0.525 a1 = 9.59

1 SPH: c2 = 0.516 a2 = 46.2

PROJECT: Boyongon June 2008
 VARIOGRAM 1: DTH RLP CUCXPCT 2.5m Comps/Cut/Cucxpct/Cu Shell
 LAGS: 50 of 2.5

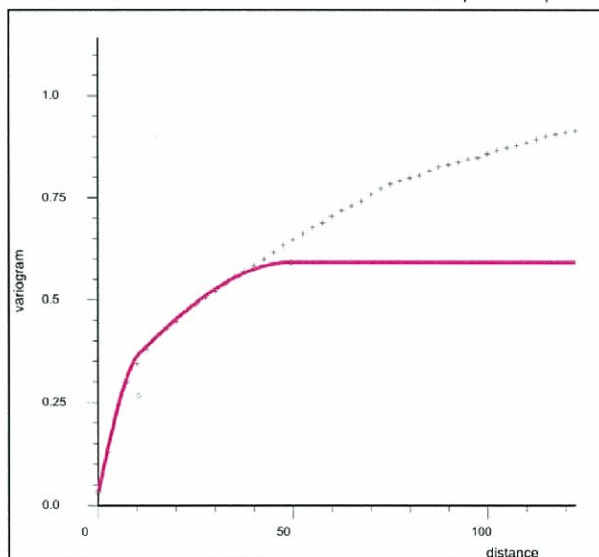
Direction Number 1

Azimuth = 0.0 +/- 90.0
 Plunge = 0.0 +/- 90.0



DIRECTION Number 1 DTH RLP CUCXPCT 2.5m Comps/Cut/Cucxpct/Cu Shell

Variogram Model:



Nugget effect = 0.0326

Directions:

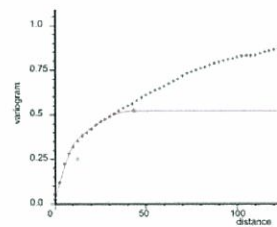
1 SPH: c1 = 0.234 a1 = 10.4

1 SPH: c2 = 0.326 a2 = 49.5

PROJECT: Boyongon June 2008
VARIOGRAM 1: DTH RLP CUHXPCT 2.5m Comps/Cut/Cuhxpct/Cu Shell
LAGS: 50 of 2.5

Direction Number 1

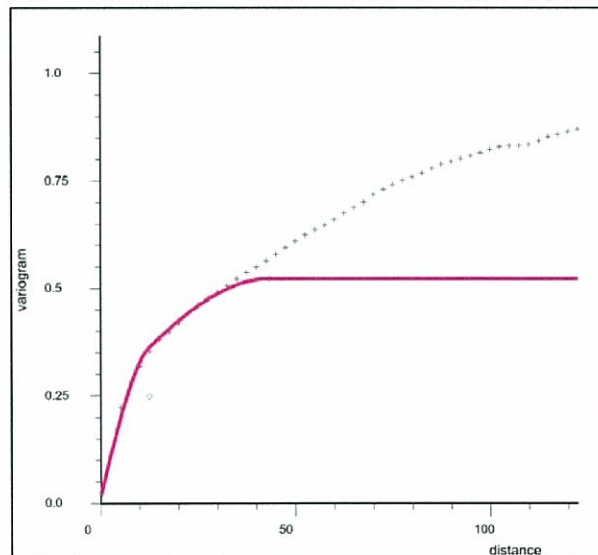
Azimuth = 0.0 +/- 90.0
Plunge = 0.0 +/- 90.0



DIRECTION Number 1

DTH RLP CUHXPCT 2.5m Comps/Cut/Cuhxpct/Cu Shell

Variogram Model:



Nugget effect = 0.0218

Directions:

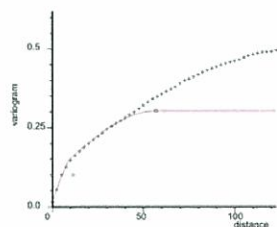
1 SPH: c1 = 0.227 a1 = 12.5

1 SPH: c2 = 0.273 a2 = 43.3

PROJECT: Boyongon June 2008
 VARIOGRAM 1: DTH RLP CUXCRAT 2.5m Comps/Cut/Cucxrat/Cu Shell
 LAGS: 50 of 2.5

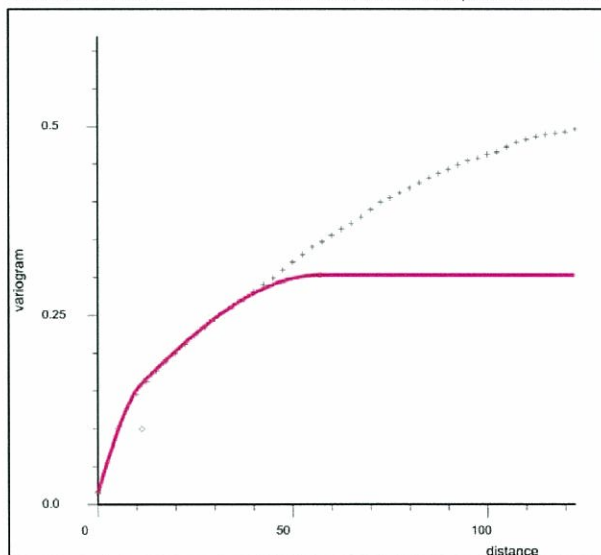
Direction Number 1

Azimuth = 0.0 +/- 90.0
 Plunge = 0.0 +/- 90.0



DIRECTION Number 1 DTH RLP CUXCRAT 2.5m Comps/Cut/Cucxrat/Cu Shell

Variogram Model:



Nugget effect = 0.0151

Directions:

1 SPH: c1 = 0.0845 a1 = 11.3

1 SPH: c2 = 0.203 a2 = 57.0

PROJECT: Boyongon June 2008

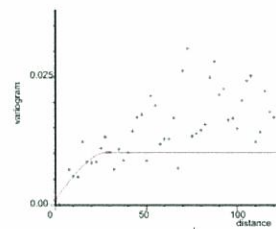
VARIOGRAM 2: DTH RLP SG

LAGS: 50 of 2.5

Direction Number 1

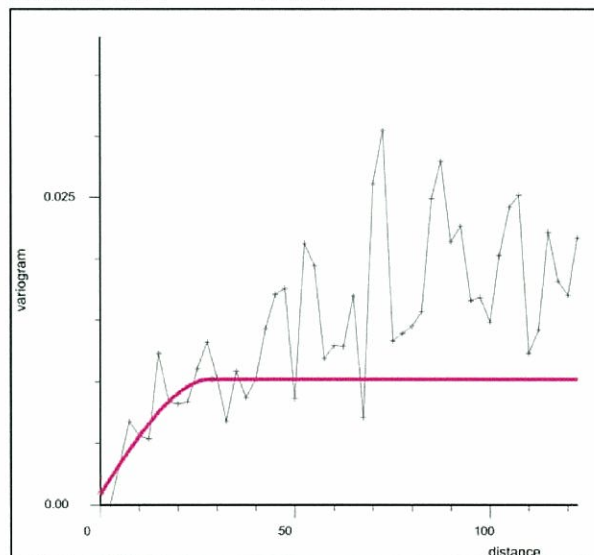
Azimuth = 0.0 +/- 90.0

Plunge = 0.0 +/- 90.0



DIRECTION Number 1

DTH RLP SG



Variogram Model:

Nugget effect = 0.0009

Directions:

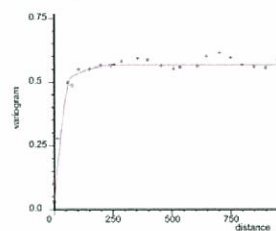
1 SPH: c1 = 0.0093 a1 = 28.7

Appendix V Omni Directional Variograms

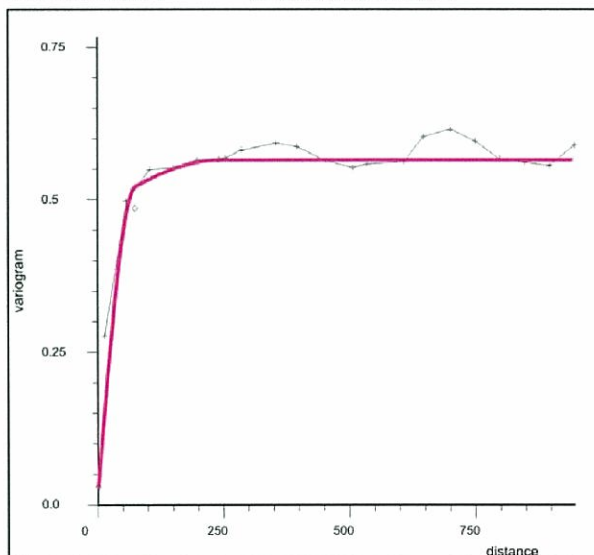
PROJECT: Boyongon June 2008
 VARIOGRAM 1: RLP CU 2.5m/Cut/CU/Cu Shell
 LAGS: 20 of 50.0

Direction Number 1

Azimuth = 0.0 +/- 90.0
 Plunge = 0.0 +/- 90.0



DIRECTION Number 1 RLP CU 2.5m/Cut/CU/Cu Shell



Variogram Model:

Nugget effect = 0.0290

Directions:

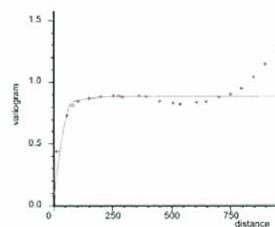
1 SPH: c1 = 0.457 a1 = 74.1

1 SPH: c2 = 0.0785 a2 = 241.

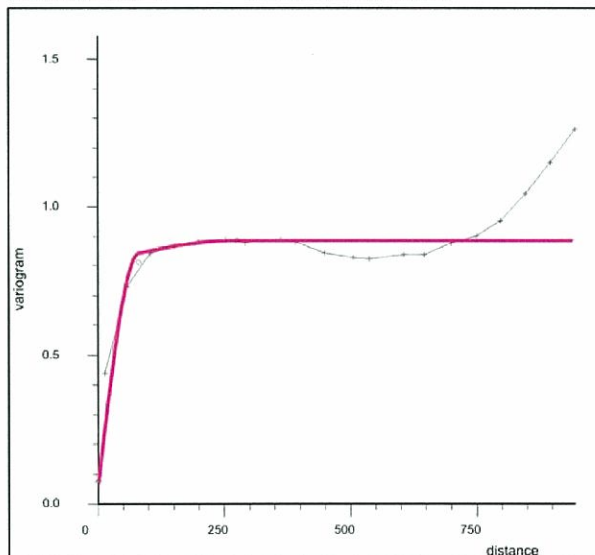
PROJECT: Boyongon June 2008
VARIOGRAM 1: RLP AU 2.5m/Cut/AU/Au Shell
LAGS: 20 of 50.0

Direction Number 1

Azimuth = 0.0 +/- 90.0
Plunge = 0.0 +/- 90.0



DIRECTION Number 1 RLP AU 2.5m/Cut/AU/Au Shell



Variogram Model:

Nugget effect = 0.0735

Directions:

1 SPH: c1 = 0.738 a1 = 80.4

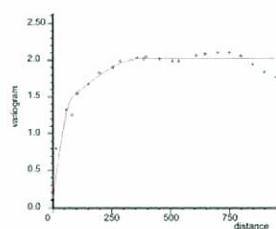
1 SPH: c2 = 0.0740 a2 = 276.

PROJECT: Boyongon June 2008
VARIOGRAM 1: RLP AS 2.5m/Cut/As/Cu Shell

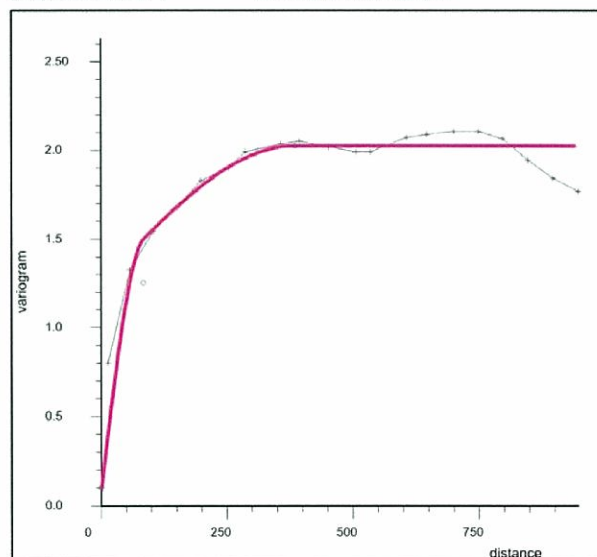
LAGS: 20 of 50.0

Direction Number 1

Azimuth = 0.0 +/- 90.0
Plunge = 0.0 +/- 90.0



DIRECTION Number 1 RLP AS 2.5m/Cut/As/Cu Shell



Variogram Model:

Nugget effect = 0.0991

Directions:

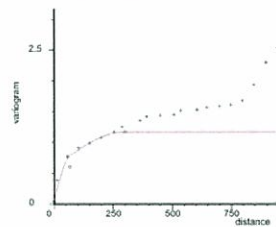
1 SPH: c1 = 1.153 a1 = 83.7

1 SPH: c2 = 0.773 a2 = 386.

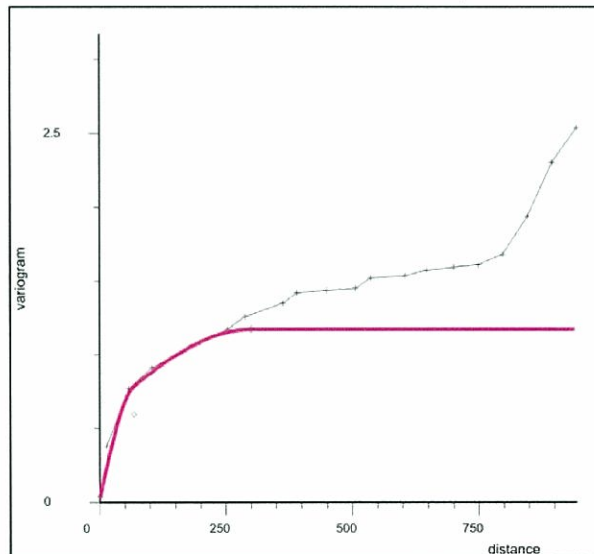
PROJECT: Boyongon June 2008
VARIOGRAM 1: RLP CUCXPCT 2.5m/Cut/Cucxpct/Cu Shell
LAGS: 20 of 50.0

Direction Number 1

Azimuth = 0.0 +/- 90.0
Plunge = 0.0 +/- 90.0



DIRECTION Number 1 RLP CUCXPCT 2.5m/Cut/Cucxpct/Cu Shell



Variogram Model:

Nugget effect = 0.0326

Directions:

1 SPH: c1 = 0.564 a1 = 67.5

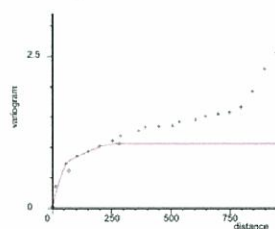
1 SPH: c2 = 0.577 a2 = 300.

PROJECT: Boyongon June 2008
VARIOGRAM 1: RLP CUHXPCT 2.5m/Cut/Cuhxpct/Cu Shell

LAGS: 20 of 50.0

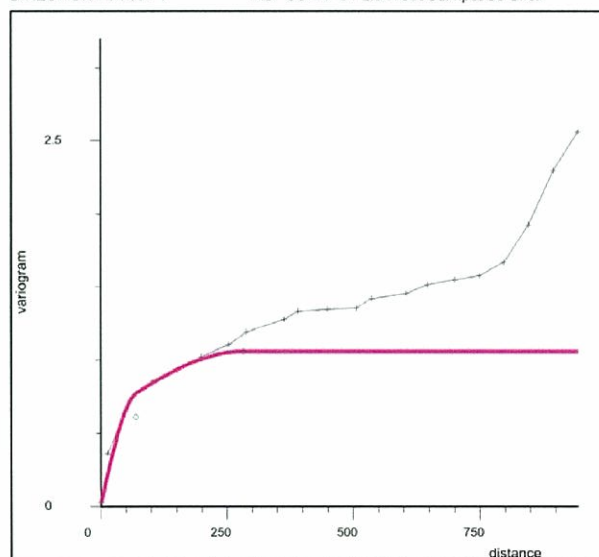
Direction Number 1

Azimuth = 0.0 +/- 90.0
Plunge = 0.0 +/- 90.0



DIRECTION Number 1

RLP CUHXPCT 2.5m/Cut/Cuhxpct/Cu Shell



Variogram Model:

Nugget effect = 0.0218

Directions:

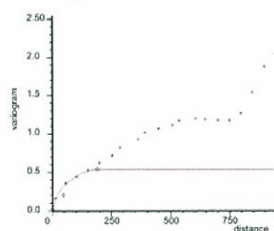
1 SPH: c1 = 0.589 a1 = 69.1

1 SPH: c2 = 0.449 a2 = 282.

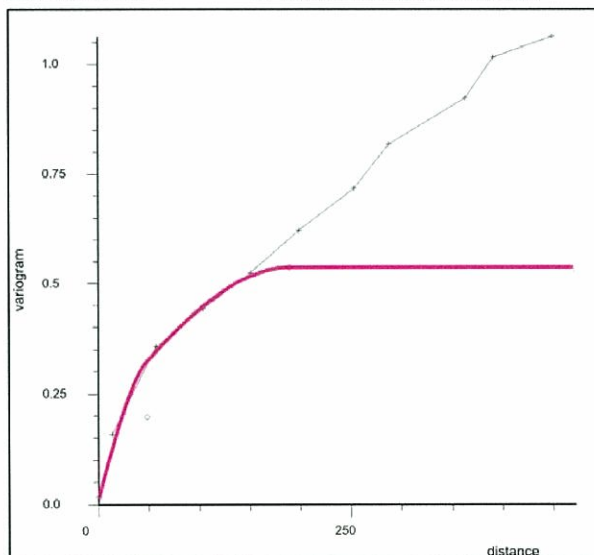
PROJECT: Boyongon June 2008
VARIOGRAM 1: RLP CUXCRAT 2.5m/Cut/Cucxrat/Cu Shell
LAGS: 20 of 50.0

Direction Number 1

Azimuth = 0.0 +/- 90.0
Plunge = 0.0 +/- 90.0



DIRECTION Number 1 RLP CUXCRAT 2.5m/Cut/Cucxrat/Cu Shell



Variogram Model:

Nugget effect = 0.0151

Directions:

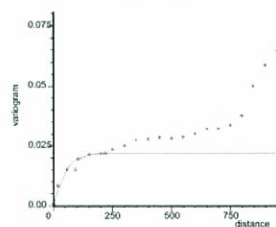
1 SPH: c1 = 0.183 a1 = 48.3

1 SPH: c2 = 0.338 a2 = 189.

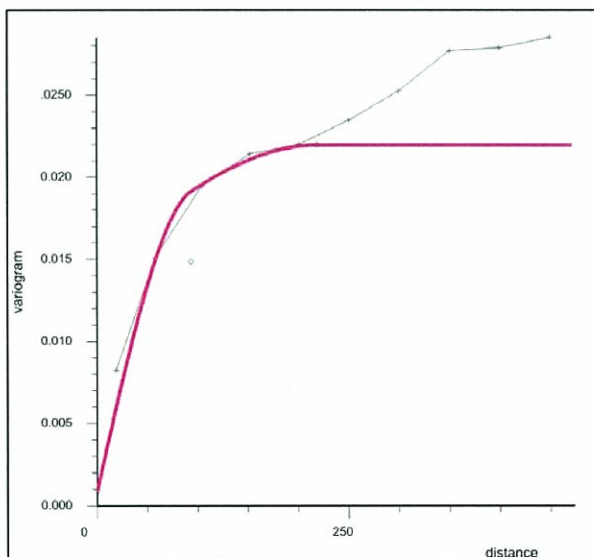
PROJECT: Boyongon June 2008
VARIOGRAM 1: RLP SG 2.5m/Cut/SG/Cu Shell
LAGS: 20 of 50.0

Direction Number 1

Azimuth = 0.0 +/- 90.0
Plunge = 0.0 +/- 90.0



DIRECTION Number 1 RLP SG 2.5m/Cut/SG/Cu Shell



Variogram Model:

Nugget effect = 0.0009

Directions:

1 SPH: c1 = 0.0140 a1 = 93.4

1 SPH: c2 = 0.0071 a2 = 219.

