Technical Report

for the

Maricunga Gold Mine

(Located in the Maricunga District of Region III, Chile)

Prepared for Compañia Minera Maricunga

and

Kinross Gold Corporation

Prepared by:

M. Bélanger, P. Geo.

Director of Technical Services

Kinross Gold Corporation

Effective Date: December 31, 2007

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Juan Carlos Buguena	Planning Superintendent, CMM
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Jorge Rojas	Reserves Technician, CMM
Barry Gillies	Director of Exploration, Kinross
Mark Sedore	Director, Corporate Reserves, Kinross
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Kevan Ford	Manager, Metallurgy, Kinross
Sergio Neira Martinez	Exploration Geologist, CMM
Ivo Ahumada Cortes	Exploration Geologist, CMM

Certificate of Author

I, Maryse Bélanger, P. Geo., do hereby certify that:

- I am Director of Technical Services, Kinross Americas for Kinross Gold Corporation, Av. Afonso Pena, 4001 - 8º andar Ala A2 - Ed. Presidente Tancredo Neves - Bairro Serra - Belo Horizonte, MG, Brazil, CEP - 30130-
- I graduated with a Bachelor of Science degree (BSc) in Earth Sciences from the Université du Québec à Chicoutimi in 1985. I studied Geostatistics at the Centre de Géostatistique in Fontainebleau, France in 1986.
- I am a member of the Association of Professional Geoscientists of Ontario (Registration Number # 0125).
- I have worked as a geologist for a total of 23 years since my graduation from University. I have been involved in gold exploration and mining in Canada, United States of America, Russia, Niger, Burkina Faso, Ivory Coast, Ethiopia, Gabon and Chile.
- I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43 -101.
- I have read NI 43-101 and certify that the Technical Report has been prepared in compliance with NI-43-101 and Form 43-101F1.
- I am responsible for supervising the writing of the technical report titled “Technical Report on the Maricunga Gold Mine” dated December 31, 2007.
- I visited the property every quarter in 2004 and 2005 and spent more than 75 days at site during the course of active exploration in 2006. In 2007 I visited the mine on a regular basis.
- I have not had prior involvement with the property that is the subject of the technical report.
- As of the date of this certificate, to the best of my knowledge and belief, the technical report contains all the scientific and technical information that is required to omission to make the technical report not misleading.
- I am not independent of the issuer. Per section 5.3.2 of National Instrument 43-101 an independent qualified person was not required to write the technical report on the Maricunga Mine.

Dated this 31st day of March, 2008 at Brasilia, Brazil.

“Signed and Sealed”

Maryse Bélanger, P. Geo.

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1.0 SUMMARY

This Technical Report for the Maricunga Mine has been prepared to support the 2007 Kinross Gold Corporation (Kinross) Annual Mineral Resources and Reserves Statement and has been prepared under the direct supervision of the qualified person, Maryse Bélanger, P. Geo., Director Technical Services, Kinross Americas with contributions from CMM technical staff.

The Maricunga Gold Mine is located in the Maricunga Gold Belt in Region III of northern Chile. The property is located 120 km due east of the city of Copiapó at elevations between 4,200 and 4,500 meters above mean sea level. The mine operated from 1996 to 2001, producing more than 920,000 ounces of gold from 46.0 million tonnes of ore. The mine was placed on care and maintenance in 2001, due to a downturn in gold prices. The Maricunga Mine complex comprises two open-pits, Verde East and Verde West and a near-development project called Pancho.

The Maricunga heap leach mine is owned and operated by Compañía Minera Maricunga (CMM), a Chilean company that is wholly-owned by Kinross since its acquisition of Bema Gold Corporation effective at the end of February 2007.

The Verde and Pancho gold deposits at Maricunga occur in the Maricunga Gold Belt of the high Andes in northern Chile. Since 1980, a total of 40 million ounces of gold have been defined in the belt, (Muntean and Einaudi, 2000). Gold mineralization at Maricunga has been interpreted as porphyry style gold systems and is hosted in the Maricunga volcanic-intrusive complex of Early Miocene age. The porphyries occur within a sequence of intermediate tuffs, porphyries and breccias that are the host rocks to the gold mineralization.

Gold mineralization at Verde is interpreted to be the result of the fracturing and concentration of fluids in the carapace of an intrusive plug or stock. Gold is closely associated with quartz, magnetite, calcite, and garnet stockworks. Gold mineralization at Pancho is characterized as porphyry hosted stockwork and sheeted veins. The veins are subvertical and have a strong, preferred north-westerly strike. The northwest structural control is evident not only at outcrop scale but is also reflected in the northwest alignment of intrusives and the three centers of mineralization in the district, Verde, Pancho and Guanaco.

In September 2002, in response to rising gold prices, Compañia Minera Maricunga (CMM) approved an Exploration Program designed to increase the reserve base of the Maricunga Project to a level sufficient to support resumption of active mining. The drilling completed in 2002-2003 focused largely on definition drilling for a $350 gold pit design.

The exploration program ran from September 2002 to June 2003. During this period, a total of 262 drill holes (51,478 meters) of drilling were completed. The drilling focused on increasing the confidence level of the known mineralization below the current Verde pits as well as increasing the confidence level in the mineralization at the nearby Pancho deposit, located approximately 2.0 km to the northeast. Much of the 2002 – 2003 drilling was diamond drill core, allowing geologists an opportunity to clearly delineate geological and alteration features affecting gold mineralization and recovery.

The information was compiled and incorporated in new mineral resource and mineral reserve estimates for Verde and Pancho. The reserves formed the basis of a detailed engineering study examining the economics of the project assuming a capital investment of $101.1 million to upgrade the existing infrastructure, allowing the plant to process 40,000 tonnes per day of Verde ore and 35,000 tonnes per day of Pancho ore. The capital investment was based on preliminary design and first-principle engineering.

The Maricunga Mine resumed full production in October 2005 at a mining rate of 40,000 tpd. Total production of 233,000 ounces from 14,333,000 tonnes at 0.74 g/t Au is reported for 2007. Only data below the year-end 2007 topographic surface was used to complete the new resource model and reserve estimate.

Both the 2002-2003 and 2006 drill programs were carefully supervised, employing industry best practices and rigid quality management procedures in the collection and management of the field data. In addition, all historic data for the project was subjected to detailed verification programs where the data in the relational database was verified against original logs, survey calculation sheets and original assay certificates. The result is a duly verified database containing data of high quality and free of gross errors and omissions. This database was then used to estimate mineral resource and reserves for the project.

In early 2006, CMM made the decision to drill at Pancho to better define the mineral resource both laterally and at depth. During the course of 2007, a full review of the geologic interpretation and the spatial controls of mineralization was undertaken for both Verde and Pancho resulting in new resource models to support updated Mineral Resource and Mineral Reserve estimates.

In addition to the analytical data, the 2002-2003 drill program was also used to collect carefully controlled metallurgical samples from Verde and Pancho. A total of 58 direct agitation (bottle roll) and 18 column leach test samples were collected and analyzed. The metallurgical samples were carefully selected to ensure adequate sample coverage of various grade bins, recovery classes and lithologies. More metallurgical test work was completed in 2006 and 2007 with the addition of 19 column tests for Pancho. The data compiled to date indicates that gold recovery from the Pancho samples tested is approximately 68% for the sulphide ore and 85% for the oxide ore. This recent test work plus the work completed in 2003 confirms that the gold recovery values used in the 2007 reserve estimate are reasonable.

In 2007, the SART process was tested for the Pancho ore given the higher concentrations of copper present in the ore. Some of the copper minerals are soluble in cyanide solutions and these minerals consume cyanide reagent and contaminate the leach solution with copper metal. A feasibility study and pilot testing program were completed by Idesol and SGS Lakefield of Santiago. The study considers plant design, performance metrics, operating and capital costs and implementation strategy. The reserves defined at Pancho consider the application of SART.

Ore and waste hardness was also tested in 2002-2003. Specific gravity of the various rock types was confirmed through extensive field sampling. In 2006, specific gravity measurements were also performed on more than 1,500 core samples from Pancho.

The resource block models were first updated in 2003 to reflect the lithological models developed from observations taken from drill core. At the same time, a new recovery model was developed based on the visual logging of oxidation. The visual estimates were completed using a well-established scale previously used on the project. In 2006, with the additional core samples, the surfaces defining oxidation levels were updated (oxide, mixed and sulphide).

Statistical and geostatistical analysis of the drill results was performed on the raw, 2.0 meter sample intervals to identify appropriate grade capping factors. Grade capping does not have a significant effect on the resource estimates, resulting in a less than 1% decrease in contained gold. The 2007 work also included compositing of the data using 5 m length. Contact profiles were prepared to determine the search strategies for copper and gold grade interpolation. Directional variograms were modeled to identify anisotropy and search ranges that would guide grade interpolation. The variograms demonstrated excellent continuity, generally exceeding 100 meters in all directions.

Grade interpolation was limited to gold for the Verde deposit while gold; copper and cyanide soluble copper were modeled for Pancho. Ordinary kriging was selected as the primary interpolation method for gold. Both inverse distance and nearest neighbour grade interpolations were also performed to verify the kriged gold grades.

The resultant resource block models were manually verified to ensure that grade interpolation was reasonable. The model was classified as per CIM Guidelines as Measured, Indicated and Inferred based primarily on the search ranges indicated by the variograms and experience gained at Verde.

The block models were used as the basis for pit optimization that was completed with the Whittle algorithm. Optimization work was based on operating costs, adjustment factors and parameters updated in November 2007. The optimum pit shells were imported into Vulcan for final pit design using design parameters approved by a third party geotechnical engineering company.

The final mine designs were used to estimate the mineral resources and reserves was and were prepared by a Qualified Person as required by National Instrument 43-101.

Mineral reserves listed below in Table 1-1 are based on a $550 gold price and constitute the official reserves as of 31, December 2007.

It is Kinross Policy that for all open pit mine projects, the mineral resource must fall within optimized or final design pits that are based on reasonable, long-term, gold price assumptions.

In Table 1-2, mineral resources are based on a $625 gold price and reported exclusive of mineral reserves as of 31, December 2007.

Table 1-1: Maricunga Proven and Probable Mineral Reserves

Deposit	Class	Ore Tonnes (x 1,000)	Grade (Au g/t)	Gold Ounces (x 1,000)
Verde	Proven	71,071	0.85	1,947
	Probable	32,409	0.74	772
	2P	**103,480**	**0.82**	**2,719**
Pancho	Proven	106,627	0.71	2,436
	Probable	69,395	0.58	1,290
	2P	**176,022**	**0.66**	**3,726**
Total		**279,502**	**0.72**	**6,445**

Table 1-2: Maricunga Measured and Indicated Mineral Resources

Deposit	Class	Ore Tonnes (x 1,000)	Grade (Au g/t)	Gold Ounces (x 1,000)
Verde	Measured	19,335	0.73	456
	Indicated	26,987	0.71	619
	M&I	**46,322**	**0.72**	**1,075**
Pancho	Measured	7,585	0.52	128
	Indicated	57,549	0.58	1,071
	M&I	**65,134**	**0.57**	**1,198**
Total		**111,456**	**0.63**	**2,274**

The Maricunga deposits also host an Inferred Mineral Resource of 134.71 million tonnes averaging 0.56 g/tonne Au, given the same $625 US per ounce gold price. Pancho hosts 132.90 million tonnes averaging 0.56 g/tonne Au and Verde hosts 1.81 million tonnes averaging 0.66 g/tonne Au.

The Maricunga Mine is an operating mine with significant infrastructure in place. The author is of the opinion that the mineral reserve estimates presented in this document are sufficient to support the operation for 16 years based on daily production of 44,000 tpd increasing to 49,000 tpd in 2012 as per the Life-of-Mine Plan approved for 2008. The larger reserve base defined in 2007 at Maricunga will require an increase in leach pad facilities. A study for establishing an additional permanent leach pad was completed in 2006. In 2007, CMM undertook some engineering/design work to increase leach pad capacity. Capital costs associated with pad development, the 16 years mine life and the envisaged SART plant have all been considered for this mineral resource and mineral reserve update. This document supports the December 2007 Maricunga Mineral Resource and Reserve Statement.

2.0 INTRODUCTION AND TERMS OF REFERENCE

2.1 Introduction

This Technical Report for the Maricunga Mine has been prepared to support the 2007 Kinross Gold Corporation (Kinross) Annual Mineral Resources and Reserves Statement and has been prepared under the direct supervision of the qualified person, Maryse Bélanger, P. Geo., Director Technical Services, Kinross Americas with contributions from CMM technical staff and Kinross Technical Services.

2.2 Terms of Reference

Unless otherwise noted:

- all units of measurement in the following report are in metric measure;
- all costs are expressed in terms of United States dollars;
- all metal prices are expressed in terms of United States dollars;
- a foreign exchange rate of $1.0 US = 530 Chilean pesos was used.

2.3 Scope of Work

The following Technical Report considered the following:

- regional and local geology, structure, alteration and mineralization;
- sample collection, preparation, security and analysis;
- quality assurance and quality control procedures;
- data entry, verification, management, security and storage;
- block modelling, grade interpolation and resource estimation;
- metallurgical recovery, plant design and performance;
- mine planning, scheduling and reserve estimation;

- leach pad expansion;
- environmental and operational permitting;
- operating and capital cost estimates and
- financial models.

2.4 Sources of Information and Report Basis

This Technical Report is based on verified historical data as well as data collected during a recent field program at Pancho in 2006. All data was collected under the direct supervision of experienced field geologists. Operational data since production resumed in 2005 was also considered in the mineral reserves estimates presented herewith. The updated metallurgical test work, new block models for Verde and Pancho and updated operating and capital costs form the basis of the mineral reserve and mineral resource estimates.

This report has been prepared under the direct supervision of the qualified person with direct contributions from CMM technical staff. A number of other sources of information have been used in the compilation of this report and a complete list of references is provided in Section 21 of this report.

2.5 Field Involvement of the Qualified Person

M. Bélanger has been associated with the Maricunga Mine since November 2003 and has visited the site on numerous occasions during that time frame. In 2006, a total of 75 days were spent at the Maricunga mine site. Regular visits were also made during the course of 2007.

3.0 RELIANCE ON OTHER EXPERTS

This document has been prepared by Kinross Technical Services with input from Compañia Minera Maricunga. The document summarizes the professional opinion of the author and it includes conclusions and estimates that have been based on professional judgement and reasonable care and are based on the information available at the time this report was completed. All conclusions and estimates presented are based on the assumptions and conditions outlined in this report. This report is to be issued and read in its entirety.

The author has relied upon, and believes there is a reasonable basis to rely upon the contribution of other parties as defined below.

Services Provided by Other Parties

During the course of the fieldwork it was necessary to rely on expertise supplied by third party professionals with technical expertise beyond the experience of the field personnel managing the work. Areas impacted by the work of third party consultants are clearly noted. Kinross relied inherently on the conclusions and recommendations of the following third party consultants:

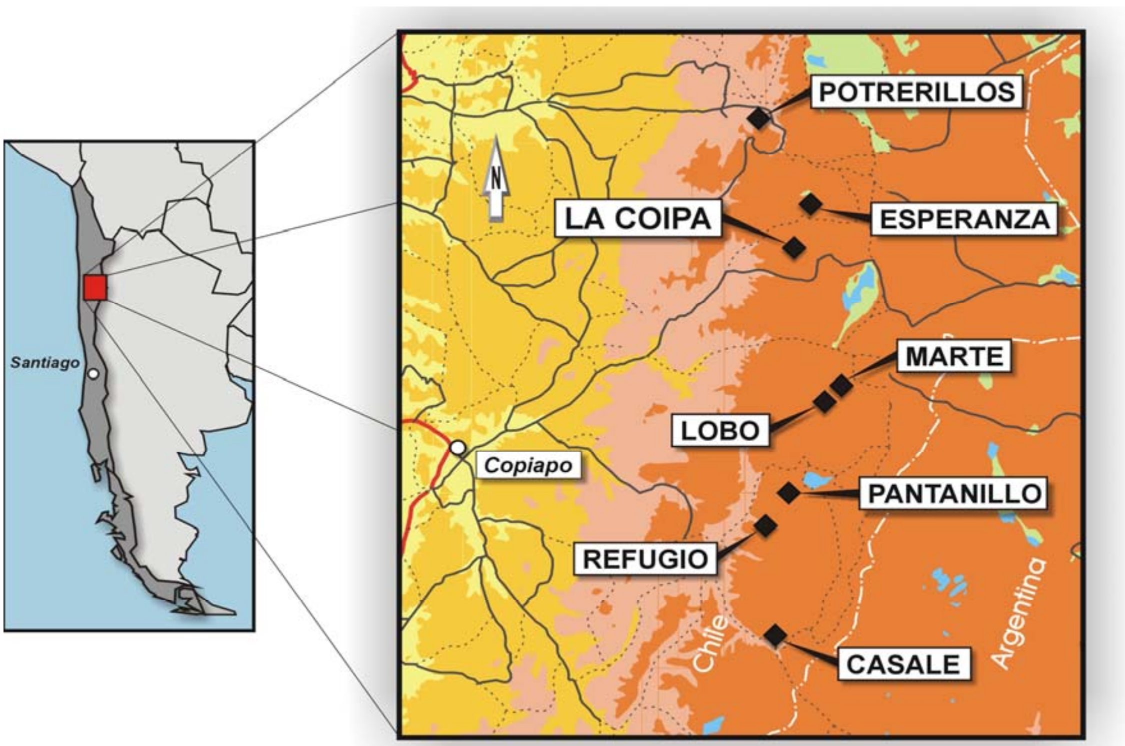
Golder Associates (2003)	Geotechnical Design Parameters
A. Karzulovic and Assoc. Ltda. (2007)	Geotechnical Design Parameters
McClelland Laboratories	Metallurgical Testing Program
Guillermo Contreras and Sons Limitada	Land Survey

4.0 PROPERTY DESCRIPTION AND LOCATION

The Maricunga Property is located in the Maricunga District of Region III in Chile. The property is located 120 km due east of the city of Copiapó at elevations between 4,200 m and 4,500 m above mean sea level. The Maricunga Mine (formerly known as Refugio) is located at 27 degrees 33 minutes south latitude and 69 degrees 18 minutes west longitude.

Figure 4-1 is a generalized location map of the project.

Figure 4-1: Maricunga Location Map



4.1 Claim Status

All surface and mineral claims, surface rights and water rights are maintained in good standing. Mining claims total 8,380 hectares while the exploration properties held by CMM include 5,900 hectares. Chilean attorneys monitor claim status on behalf of CMM annually. No ownership issues with respect to the mineral claims hosting the project's mineral resource and reserve estimates have been identified. The project mineral rights and claims are listed in Table 4-1 below. The position of the CMM claims relative to the main infrastructure and open pit location is shown in Figure 4-2. Figure 4-3 shows the detailed mineral rights and claims.

In Chile, mining claims are obtained through a judicial process that requires all survey information to be filed with the appropriate government agency. All CMM claims were surveyed by Guillermo Contreras and Sons Limitada of Santiago.

4.1.1 Surface Rights

In addition to the mineral claim rights, CMM also holds title to surface rights at Maricunga, providing the land required for the leach pads, waste dumps, Verde pit, Pancho and Guanaco targets. All surface rights affecting the ongoing mining activity have been reviewed. No fatal flaws have been identified that would negatively affect the development of new areas for mining activities.

4.1.2 Water Rights

Water extraction rights, totalling 258 litres per second have been secured by CMM. Permits are in place and maintained. No issues regarding water rights have been identified that would adversely affect the project.

4.1.3 Royalty Payments

The mine production at Maricunga is subject to royalty payments in US dollars based on the realized gold price per ounce. The underlying royalty payments for Maricunga consist of an NSR combined with payment based on Net Operating Margin (NOM). It is effectively a sliding scale. At a gold price of $400 the payments are equivalent to $9.471 per ounce.

4.1.4 Environmental Liabilities

There is no underlying environmental liability affecting CMM.

Figure 4-2: Land Claims and Infrastructure Location

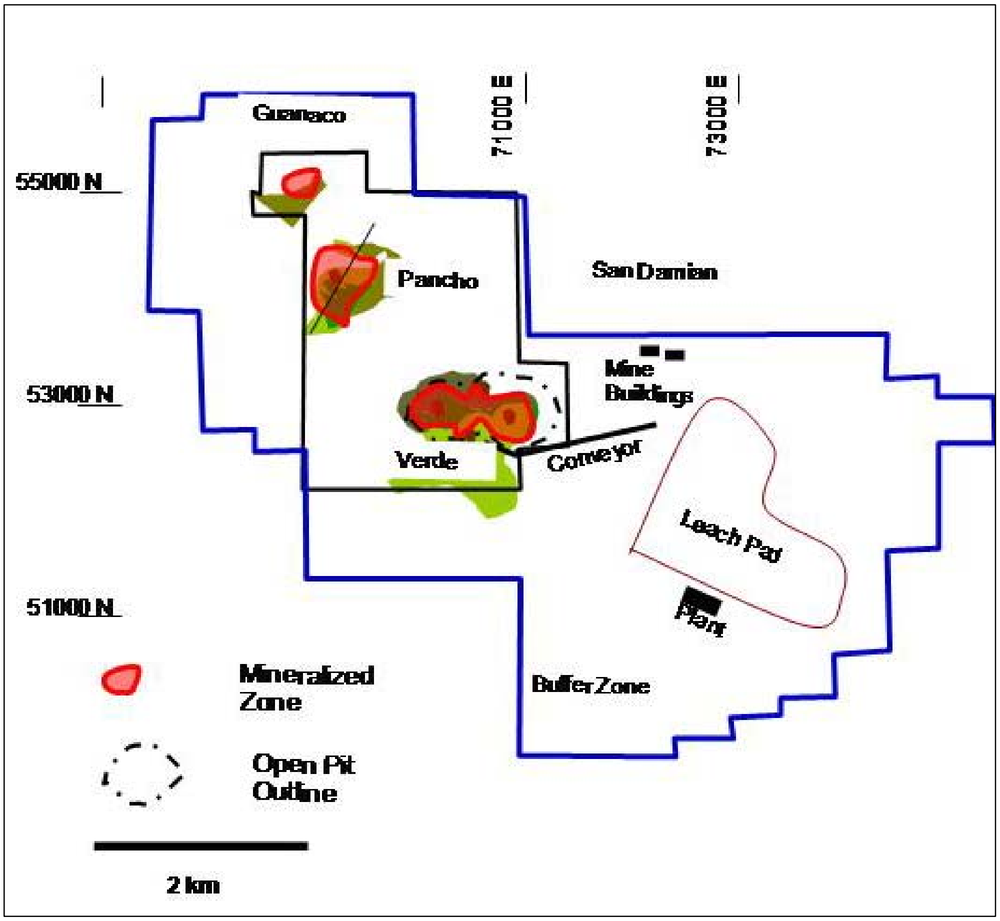


Figure 4-3: Details of Mineral Rights and Claims

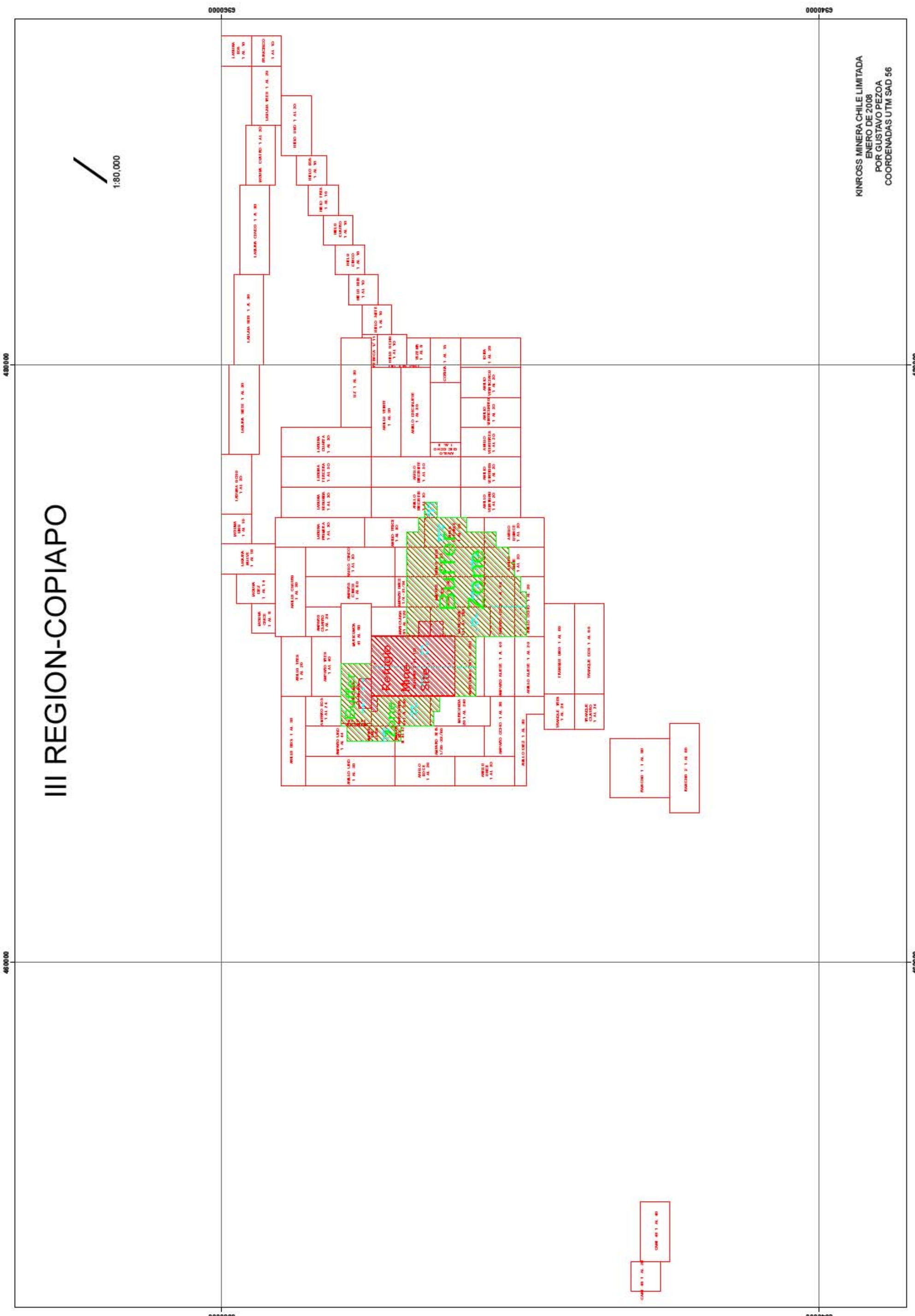


Table 4-1: Mineral Rights and Claims

Maricunga Mine Site Properties		
Claim Name	**Assigned Blocks**	**Hectares**
Maricunga 1-112	1 through 112	560
Maricunga 1-40	17 through 20	20
Maricunga 1-40	27 through 30	20
Maricunga 81-120	97 through 100	20
Maricunga 121-160	121 through 124	20
Maricunga 241-280	261 through 262	10
	TOTAL	**650**
Demasias on Maricunga	1 through 112 Western Boundary	5,6 Aprox.

Buffer Zone Properties		
Claim Name	**Assigned Blocks**	**Hectares**
Maricunga 1-40	1 through 16	80
Maricunga 1-40	21 through 26	30
Maricunga 1-40	31 through 40	50
Maricunga 81-120	93 through 96	20
Maricunga 81-120	113 through 120	40
Maricunga 121-160	125 through 160	180
Maricunga 161-200	161 through 167	35
Maricunga 161-200	171 through 177	35
Maricunga 161-200	181 through 187	35
Maricunga 161-200	191 through 197	35
Maricunga 201-240	201	5
Maricunga 201-240	221 through 223	15
Maricunga 241-280	241 through 260	100
Maricunga 241-280	263 through 280	90
Anillo Seis 1-30	3 through 15	130
Anillo Seis 1-30	18 through 30	130
Anillo Siete 1-20	1 through 5	50
Anillo Siete 1-20	11 through 14	40
Anillo Ocho 1-20	1 through 2	20
Anillo Ocho 1-20	6 through 7	20
Anillo Ocho 1-20	11 through 12	20
Anillo Ocho 1-20	16	10
Anillo Trece 1-20	8 through 10	30
Anillo Trece 1-20	20	10
Anillo Catorce 1-20	1 through 17	170
Anillo Quince 1-20	1 through 2	20
Anillo Dieciseis 1-30	10 through 11	20
Amparo Uno 1-60	45 through 60	80
Amparo Dos 1-26	25 through 26	10
Amparo Seis 1-60	31 through 32	10
Amparo Once 1-60	5 through 30	130
Amparo Once 1-60	35 through 60	130
Amparo Doce 1-36	1 through 36	180
	TOTAL	**1960**

Outside Zone Properties		
Claim Name	**Assigned Blocks**	**Hectares**
Maricunga 81-92	81 through 92	60
HIELO 1 1 AL 20		200
HIELO 2 1 AL 10		100
HIELO 3 1 AL 10		100
HIELO 4 1 AL 10		100
HIELO 5 1 AL 10		100
HIELO 6 1 AL 10		100
HIELO 7 1 AL 10		100
HIELO 8 1 AL 10		100
HIELO 9 1 AL 5		10
FRANCISCO 1 AL 10		100
LAGUNA 1 1 AL 10		100
LAGUNA 2 1 AL 10		100
LAGUNA 3 1 AL 20		200
LAGUNA 4 1 AL 20		200
LAGUNA 5 1 AL 30		300
LAGUNA 6 1 AL 30		300
LAGUNA 7 1 AL 30		300
LAGUNA 8 1 AL 20		200
LAGUNA 9 1 AL 18		180
LAGUNA 10 1 AL 14		130
LAGUNA 11 1 AL 8		80
CAMI 48 1 AL 20		100
CAMI 49 1 AL 40		200
RANCHO 1 1 AL 80		400
RANCHO 2 1 AL 60		300
	TOTAL	**4160**

Exploitation Concessions Located over the Buffer Zone and the Maricunga Mine Site		
Claim Name	**Assigned Blocks**	**Hectares**
TADEO 1 AL 70		350
JUAN 1 AL 70		350
TOMAS 1 AL 70		350
PEDRO 1 AL 18		90
JOSE 1 AL 80		400
SANTIAGO 1 AL 78		390
JUDAS 1 AL 44		220
PABLO 1 AL 42		210
MATEO 1 AL 4		20
MARCOS 1 AL 46		230
	TOTAL	**2610**

Compañía Minera San Damian Exploitation Concessions		
Claim Name	**Assigned Blocks**	**Hectares**
AMPARO 1 1/60		220
AMPARO 2 1/26		120
AMPARO 3 1/40		200
AMPARO 4 1/24		120
AMPARO 5 1/60		290

AMPARO 6 1/60		290
AMPARO 8 1/36		180
AMPARO 9 1/40		200
AMPARO 11 1/60		40
ANILLO 1 1/30		300
ANILLO 2 1/30		300
ANILLO 3 1/20		200
ANILLO 4 1/30		300
ANILLO 5 1/30		300
ANILLO 6 1/30		40
ANILLO 7 1/20		110
ANILLO 8 1/20		130
ANILLO 9 1/20		200
ANILLO 10 1/30		156
ANILLO 11 1/20		200
ANILLO 12 1/20		200
ANILLO 13 1/20		160
ANILLO 14 1/20		30
ANILLO 15 1/20		180
ANILLO 16 1/30		280
ANILLO 17 1/30		300
ANILLO 18 1/5		50
ANILLO 19 1/30		300
ANILLO 20 1/30		300
ANILLO 21 1/20		200
ANILLO 22 1/20		200
ANILLO 23 1/20		200
ANILLO 24 1/20		200
ANILLO 25 1/20		200
CORINA 1/15		150
LAGUNA I 1/30		300
LAGUNA II 1/30		300
LAGUNA III 1/30		300
LAGUNA IV 1/30		300
LIZ 1/30		300
LOIDA 1/20		200
MARICUNGA 41/80		200
MARICUNGA 81/120		60
MARICUNGA 161/200		60
MARICUNGA 201/240		5
MARICUNGA 201/240		5
MARICUNGA 201/240		170
REBECA 1/11		29
TRANQUE 1 1/60		300
TRANQUE 2 1/60		300
TRANQUE 3 1/24		120
TRANQUE 4 1/24		120
YEZENIA 1/8		76
	TOTAL	**9991**

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

Access to the property is via 156 km of a two-lane dirt road connecting with the paved highway C-35 approximately 10 km south of Copiapó. The first 96 km of the dirt road constitute a section of a national highway crossing into Argentina. The road then branches out to the northeast to the mine site. The final 60 km is a private CMM road and it is generally in good to very good condition.

5.2 Climate and Physiography

The property is located 120 km due east of the city of Copiapó at elevations between 4,200 m and 4,500 m above mean sea level. The Maricunga Mine is located in steep, mountainous terrain with slopes up to 35%. The site is largely devoid of vegetation with the exception of the spring-fed marshes found along the valley floors. The climate is arid with an average annual precipitation of 87 mm, most of which is realized as snowfall during the winter months (March through August). Generally, very little precipitation occurs during the summer months (September through February). Local wildlife is sparse.

5.3 Local Resources and Infrastructure

The Maricunga Mine site includes leaching and ADR facilities, power generation, maintenance shops, office facilities and camp. The existing infrastructure also includes an established fresh water supply system, process water supply system, truck shop, warehousing and lay down areas, an in pit crushing and conveying system and a secondary/ tertiary crushing and screening plant. Significant upgrades designed to increase production throughput were completed in 2005. They included the in-pit crushing and conveying system as well as the secondary/tertiary crushing and screening infrastructure. All facilities are in good operating conditions. Surface and water rights are previously discussed in Section 4.0.

Chile features a strong mining culture with well-established support centers in Santiago, Antofagasta and Copiapó, The town of Copiapó is the primary staging and support area for the mine with various well-established contractors.

6.0 HISTORY

David Thomson and Mario Hernandez discovered gold mineralization at Maricunga in 1984. Shortly after their discovery, Hernandez, Thomson, and three other partners acquired the existing claims at Maricunga for Compañia Minera Maricunga (CMR). CMR completed geologic mapping and geochemical sampling, identifying anomalous gold values in three areas: 1) Cerro Verde, 2) Cerro Pancho, and 3) Guanaco. A detailed discussion of exploration activities by previous owners is discussed in Section 10.

In 1985, Anglo American Chile Limitada (Anglo) optioned the property from CMR. Anglo explored the property for three years, returning the claims to CMR in 1988.

In 1989, CMR signed a letter of intent to explore the Maricunga property with Bema Gold Corporation (Bema). Bema commenced exploration fieldwork in October 1989 and from 1989 to 1991, completed 51,765 meters of drilling at Verde with an additional 5,088 meters at Pancho. Bema also commissioned Mineral Resources Development Inc (MRDI) to complete a feasibility study on the project, which indicated positive project economics. In January 1993, Bema exercised its option rights, obtaining a 50% interest in the Maricunga properties. At the same time, CMR sold its remaining 50% interest to Amax Gold Inc. (Amax). Amax (operator) and Bema formed Compañia Minera Maricunga (CMM), a 50/50 joint venture to develop and operate the Maricunga project. From 1993 through 1997, CMM continued developing the project, beginning commercial production in 1996.

The mine first operated from 1996 to 2001, producing more than 920,000 ounces of gold from 46.0 million tonnes of ore. In 1998, Kinross Gold Corporation (Kinross) acquired Amax's 50% interest through a merger agreement.

Mining operations at Maricunga were suspended during the second quarter of 2001, a result of depressed gold prices. Improving gold prices during 2002 prompted CMM to evaluate options for re-opening the mine. Scoping studies quantifying the conditions necessary to re-open the mine were completed. The studies indicated that the project economics required an increase in the existing reserve base. As a result, an exploration program was developed to evaluate the reserve potential at depth at the Verde deposits and the inferred resource at the nearby Pancho deposit, located approximately 2.0 km northwest of the Verde pit.

A 35,000 meter drill program was approved in September 2002 with the objective of adding 20 million tonnes of Measured and Indicated resource at both Pancho and Verde. This mineralization had to be recoverable through open pit mining methods. Follow-up work to further delineate the mineralization, provide representative metallurgical samples and detailed geotechnical information would be contingent on the initial results of the drilling.

The information was collated and incorporated in new mineral resource and mineral reserve estimates for Verde and Pancho. The reserves were based on a detailed engineering study examining the economics of the project assuming a capital investment of $101.1 million to upgrade the existing infrastructure, allowing the plant to process 40,000 tonnes per day of Verde ore and 35,000 tonnes per day of Pancho ore. The capital investment was based on preliminary design and first-principle engineering. Capital cost estimates were prepared and verified by two, independent, North American engineering consultants with significant experience in Latin America.

The Maricunga Mine resumed full production in October 2005 at a rate of 40,000 tpd. Total production of 233,000 ounces from 14,333,000 tonnes at 0.74 g/t Au is reported for 2007. Table 6-1 summarizes the production history of Maricunga.

Table 6-1: Annual Gold Production of the Maricunga Mine

Year	Mined tonnes (x 1,000)	Grade Mined (Au g/t)	Gold Production (ounces)
1996	7,617	1.03	101,276
1997	7,789	1.02	147,085
1998	8,207	0.93	161,046
1999	8,936	0.94	179,465
2000	8,801	0.94	169,832
2001	4,643	0.95	133,947
2002	-	-	26,094
2003	-	-	5,000
2005	6,214	0.87	121,670
2006	14,872	0.74	233,836
2007	14,333	0.74	233,000
TOTAL	**81,412**	**0.88**	**1,512,251**

In early 2006, CMM made the decision to drill at the Pancho deposit to better define the mineral resource both laterally and at depth since the drilling completed in 2003 focused largely on definition drilling for a $350 gold pit design. During 2007, both Verde and Pancho models were refined to better reflect a new geologic interpretation and the mining experience gained over the years. The SART process was tested for the Pancho ore as uneconomic concentrations of copper are present in ore. Some of the copper minerals are soluble in cyanide solutions and these minerals consume cyanide reagent and contaminate the leach solution with copper metal. A feasibility study and pilot testing program were completed by Idesol of Santiago. The study considers plant design, performance metrics, operating and capital costs and implementation strategy. The mineral reserves defined at Pancho consider the application of SART.

7.0 GEOLOGICAL SETTING

The Verde and Pancho gold deposits at Maricunga occur in the Maricunga Gold Belt of the high Andes in northern Chile. Since 1980, a total of 40 million ounces of gold have been defined in the belt, (Muntean and Einaudi, 2000).

Basement rocks in the Maricunga area are Palaeozoic to early Tertiary age. The oldest rocks are the Late Pennsylvanian to Triassic aged rhyolite ignimbrites and breccias of the Pantanosa Formation. This unit has been uplifted along a northerly trending, westerly dipping reverse fault. The Pantanosa Formation is faulted over a package of interlayered redbeds and greenstones of the Monardes and Agua Helada Formations. These formations are Late Jurassic to Early Cretaceous in age. They comprise an east dipping (55°) sequence of strata with a thickness of 900 m. A 200 meter thick sequence of andesitic volcaniclastic sedimentary rocks overlies the redbeds and greenstones. These strata are correlated with Late Cretaceous to Early Tertiary rocks of the Quebrada Seca, Quebrada Paipote, and Las Pircas Formations.

Gold mineralization at Maricunga is hosted in the Maricunga volcanic-intrusive complex of Early Miocene age. These rocks are largely of intermediate composition. Radiometric age dating indicates these rocks are between 22 to 24 Ma in age. (Muntean 1998).

The Maricunga volcanic-intrusive complex is exposed over an area of 12 km^2 and consists of andesitic to dacitic domes, flows, and breccias that are intruded by subvolcanic porphyries and breccias (Muntean 1998). Distinguishing between volcanic and intrusive rocks in the complex is difficult as there are only minor differences between the various units.

7.1 Alteration

7.1.1 Verde

Alteration assemblages observed at Verde and Pancho are generally supportive of porphyry style mineralization but the intensity of the alteration fabric tends to be weak. Potassic alteration has been observed at Verde but generally tends to be rare. Silicification is local and patchy. Propylitic alteration, although variable on small scale, appears ubiquitous on a mine scale and in global terms does not change laterally or vertically. For that reason, primary alteration (including silicification) zonation has not been mapped.

Supergene alteration, which directly affects gold recovery, occurs deeper within fracture and fault zones. Sericite and chlorite are replaced by clay minerals, magnetite by hematite and pyrite by jarosite. The oxidation and the leaching by meteoric waters has penetrated to variable depths within the deposits depending on the fracture intensity and faulting. The supergene alteration is accompanied by the deposition of limonite, manganese oxides, clay, sericite, jarosite along with gypsum in the argillic altered zones.

7.1.2 Pancho

Gold mineralization at Pancho is associated with a central zone of potassic alteration, which is manifested by replacement of mafic minerals, by fine grained, secondary biotite and magnetite. Partial replacement of plagioclase by k-feldspar can also be observed. Muntean (1998) has documented a more restricted core to this potassic zone consisting of magnetite-k feldspar-plagioclase replacement which grades outward into the more widespread secondary biotite zone. Similar to Verde, much of the potassic alteration is obliterated or obscured by a later chlorite overprint.

The intermediate to upper parts of the system are dominated by pyrite-albite-clay-sericite alteration, primarily in the volcanics but also overprinting the upper parts of the potassic zone. This alteration is late and probably has a large supergene component.

In the uppermost parts of the system analysis indicates the presence of hypogene alunite, dickite and pyrophyllite, characteristic of epithermal, high sulfidation alteration. This hypogene assemblage occurs together with strong supergene alunite, kaolinite and other clays.

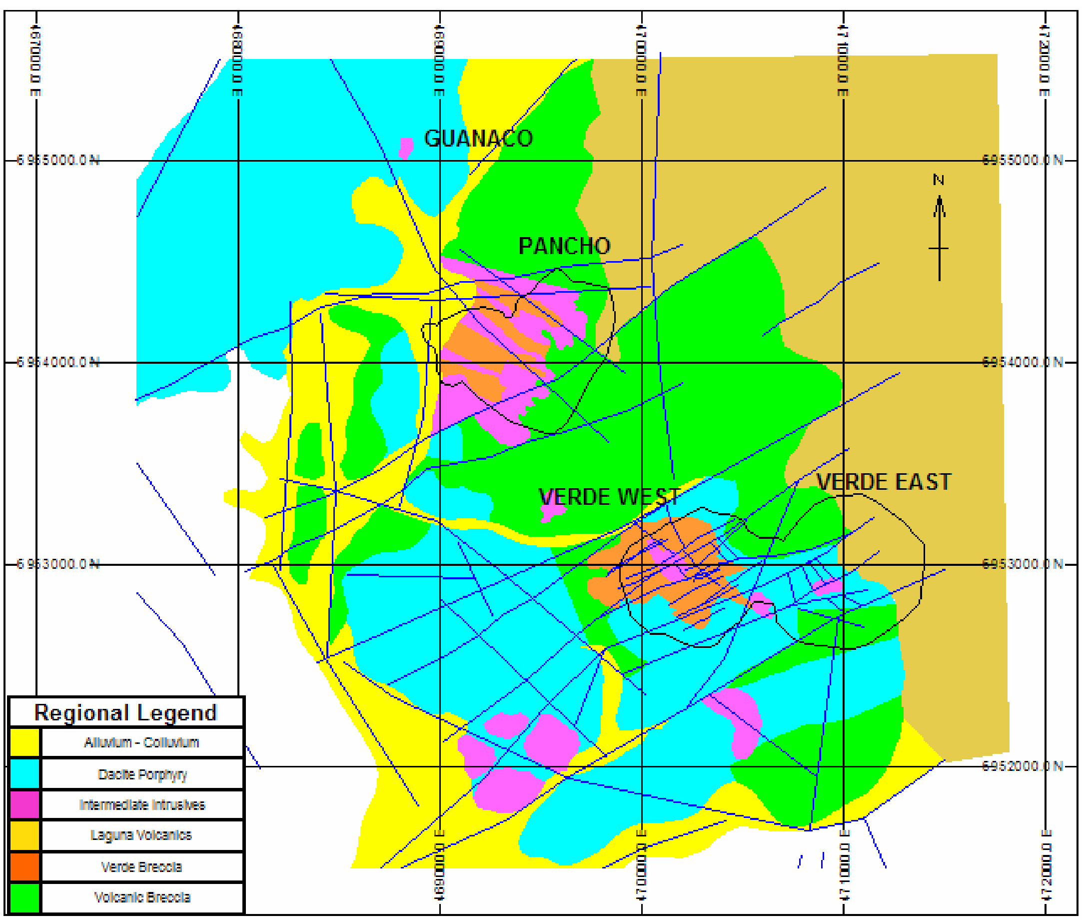
The presence of this high level alteration within 100 - 200 m of the potassic zone suggests strong telescoping of the system.

7.2 Structure

Most of the structural trends affecting the Verde and Pancho deposits are related to fracture systems rather than fault zones. The dominant trends are north-northeast and north-northwest fracture systems that can be readily identified at Verde. At Pancho, the dominant structural trend is northwesterly, reflected not only in mineralized structures but also in late, post mineral structures.

One of the main structural features influencing the Pancho deposit is Falla Guatita fault zone. Field mapping suggests that there may be significant vertical displacement on this structure. The distinctly higher copper grades and presence of potassic alteration in the main zone suggests that the central portion of the Pancho deposit may have been down dropped. Figure 7-1 is a geological map of the Maricunga Area (after Muntean, 2000).

Figure 7-1: Geology of the Maricunga Mine Area



8.0 DEPOSIT TYPES

8.1 Verde

Gold mineralization at Maricunga has been interpreted to be porphyry style gold systems. At Verde West, gold mineralization is centered about an elliptical porphyry plug measuring 175 by 100 m and oriented at N30°W. At Verde East, the porphyry plug measures 130 by 80 m and is oriented at N35°E. The porphyries occur within a sequence of intermediate tuffs, porphyries and breccias that are the host rocks to the gold mineralization. Lithological interpretation at Verde has identified six major lithologic units. These are:

- Post mineral intrusives (barren)
- Mineralized post mineral intrusives
- Verde breccia
- Dacite porphyry
- Dacite tuffs
- Laguna tuff (barren)

The most favourable ore hosts are the Verde Breccia and Dacite Porphyry units. The dacite porphyry is a volcanic to hypabyssal intrusive rock with 20% to 40% plagioclase phenocrysts in a fine-grained matrix. It contains phenocrysts of biotite, hornblende, and sparse quartz. The unit is the best host for stockwork veining, with some portions containing up to 20% quartz-magnetite ± pyrite veinlets.

The Verde Breccia consists of intrusive breccia and/or volcanic tuff breccia. Its geometry suggests an intrusive origin. The unit consists of breccia with angular to rounded clasts, generally matrix supported. The breccia is generally green to greenish gray, which is one of the diagnostic features of the unit. The color of the rock is largely due to chlorite and occasionally epidote. The fragments range from 2 mm to greater than 2 meters in size. Locally the unit is mostly matrix-sized material with only sparse clasts. Clasts in the unit are generally monolithic volcanic fragments, which are often porphyritic with white plagioclase laths up to 5 mm in length. The unoxidized rock typically contains 0.5 to 1% pyrite. Quartz-magnetite veinlets are common in the mineralized portions of the unit. In places, the breccia is cut by fine-grained matrix material that forms clastic-like dikes. These are generally > 2 cm in width. They may have been formed as “fluidized” material injected into fractures at the time of the formation of the unit.

8.2 Pancho

The gold mineralization at Pancho is also described as porphyry style mineralization. It occurs within a sequence of intermediate tuffs, porphyries and breccias that are the host rocks to the gold mineralization. Lithological interpretation based on the recent 2006 drill program has identified six major lithologic units. These are from older to younger rocks:

- Hornfels – They represent less than 5% of the total volume. They were identified at depth in the Pancho porphyry system. They are present in a series of elongated bodies, sub-horizontal and intercalated with intrusive breccias and dioritic porphyry. The hornfels are also characterized by widespread intense silicification.

- Diorite Porphyry - Rocks hypabyssal that represent 60% of the volume of the Pancho intrusive complex. It is the most frequent host of mineralized A, B and T veinlets. The diorite porphyry also shows signs of being intruded by a smaller intrusive phase. The porphyry displays obvious porphyry texture with 20 to 40 % phenocrysts. Phenocrysts alteration pattern include biotite or hornblende being completed altered and replaced by assemblages of chlorite-quartz-sericite-magnetite-hematite. At depth the porphyry clearly shows potassic alteration.

- Intrusive Breccias – The intrusive breccias represent approximately 15% of the volume of material in the Pancho complex. They are generally elongated and sub-horizontal. These breccias are characterized by their fragments of dioritic porphyry. The fragments vary in size for mm to several cm within a matrix which is normally fine-grained. The intrusive breccias are a significant host of gold mineralization with A, B, T veinlets.

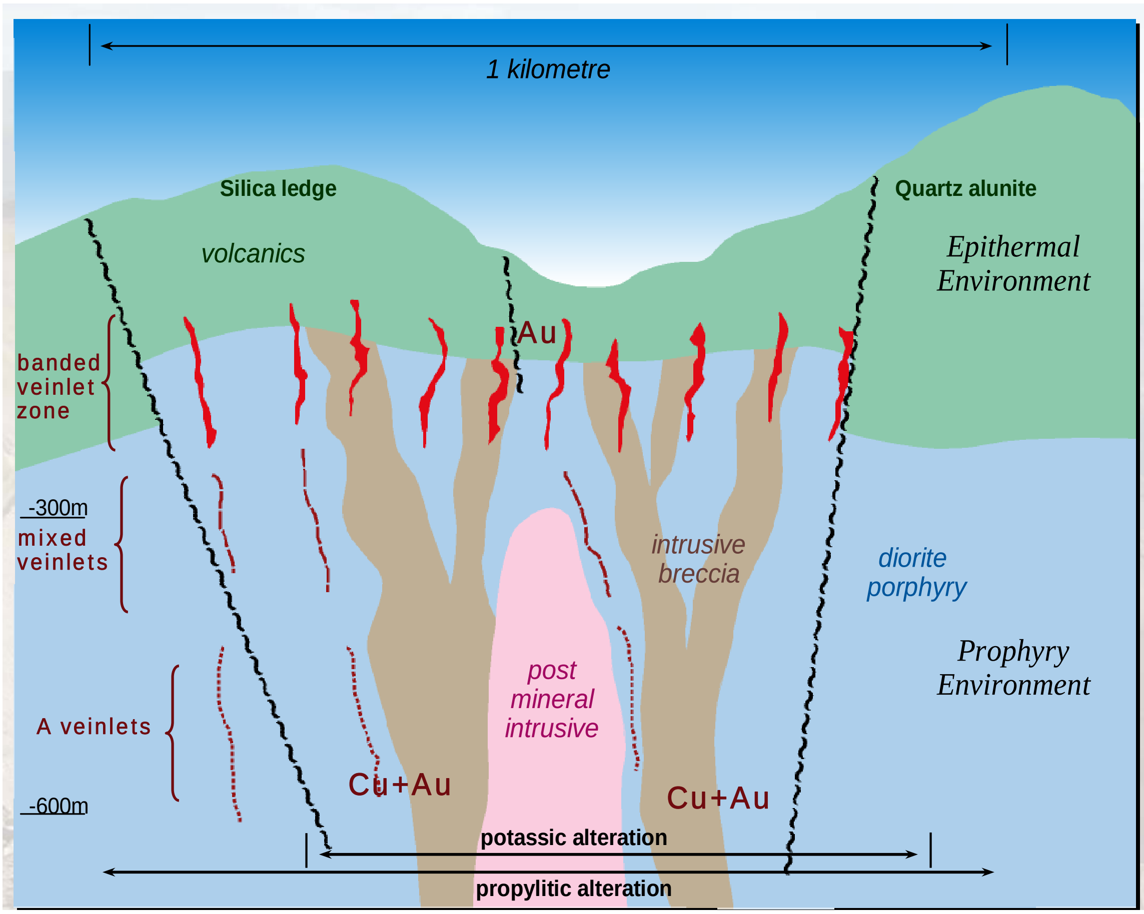
- Diorite Porphyry (2) - This diorite porphyry is defined as a small body identified between two drill sections and found at depths ranging from 300-400 meters. 400 m. In theory, it intrudes the main porphyry but in turn is intruded by the PMI. It is very similar to the main porphyry body in terms of mineralogy, alteration and mineralization. The main difference observed is its primary biotite that is partially altered into chlorite. It also has less veinlets than the main diorite porphyry.

- Dioritic Post Mineral Intrusive – The Post Mineral Intrusive (PMI) identified at Pancho is not a PMI in a strict sense as there is some very low-grade mineralization associated with it.

- Volcanic Breccias – Found in the upper portion of the deposit they represent approximately 15 % of the volume of material explored to date at Pancho. The volcanic breccias are normally sub-horizontal and discordant with the other units described.

Figure 8-1 shows the simplified geology model for Pancho.

Figure 8-1: Simplified Geology for Pancho



9.0 MINERALIZATION

9.1 Verde

Gold mineralization at Verde is interpreted to be the result of the fracturing and concentration of fluids in the carapace of an intrusive plug or stock. Gold is closely associated with quartz, magnetite, calcite, and garnet stockworks. Approximately 80% of the stockwork veins are generally dark grey in color, finely banded, with magnetite. The remaining 20% are principally white quartz veins. Gold mineralization is postulated to have resulted from at least 2 phases of mineralization, the first is a lower grade phase associated with copper and probably the porphyry emplacement event. The second phase is a higher-grade gold only event possibly associated with the structurally emplaced veinlet swarms and northwest trending sheeted veinlet zones more evident in Verde East. Verde East and Verde West cover an area of 1.4 km in length by 700 km in width. Drilling has identified gold mineralization to a depth of a 600 m.

9.2 Pancho

The porphyritic diorite intrusives and intrusive breccias are the main hosts for mineralized veins. Mineralization is also hosted by the volcanic rocks. The mineralization is commonly related to the various geologic features listed below:

- directly associated with the presence of B, A and T veinlets
- in silicified zones, normally micro-granular and a dark grey color, with or without the presence of magnetite
- associated with intrusive breccias
- in contact zones such as the contact between the intrusive breccia with the volcanic breccia or the contact between the intrusive breccia and the diorite porphyry

In the porphyry the veins are present in stockwork or sheeted veins. They are generally subvertical and have a strong, preferred north-westerly strike. The northwest structural control is evident not only at outcrop scale but is also reflected in the northwest alignment of intrusives and the three centers of mineralization in the district, Verde, Pancho and

Guanaco. The mineralization outlined to date at Pancho comprises an area of 800 m by 700 m. Gold porphyry mineralization was identified in drilling at depths exceeding 600 m.

10.0 EXPLORATION

Exploration of the Verde and Pancho deposits has been ongoing since 1984 when the Maricunga Property was identified as a gold prospect in 1984 by David Thomson and Mario Hernandez during a field visit to areas of alteration. The existing claims on the property were acquired by the partnership Companio Minera Maricunga (CMR) consisting of Hernandez, Thomson, and three other partners.

CMR subsequently completed a program of geological mapping, rock chip and geochemical sampling and identified three large areas of alteration with anomalous gold values. These areas were named Cerro Verde, Cerro Pancho and Guanaco.

In December 1985, CMR optioned the Maricunga Property to Anglo American for 3 years. During that period Anglo American accomplished the following:

- Geological mapped the property at a scale of 1:5,000
- Collected 2,161 surface samples over an 8 km^2 area and assayed for gold, silver, copper, molybdenum and zinc.
- Constructed 7.5 km of roads
- Geologically mapped Pancho and Verde at 1:1,000 scale
- Cut 6,000 m trenches on Pancho; collected and assayed 1,682 samples
- Cut 7,500 m of trenches on Verde; collected and assayed 2,350 rock samples
- Drilled 45 percussion drill holes on Pancho to average depths of 50 m (total of 2,234 m), along with 6 diamond drill holes (total of 366 m).
- Drilled 35 percussion holes, also to a depth of 50 m on Verde (total of 1,744 m).

Anglo American's exploration program outlined broad areas of gold mineralization on Pancho, Verde and Guanaco, targeted by geochemical anomalies of +100 ppb gold in the soil.

In December 1988 the property reverted to the owners. In early 1989, CMR solicited bids from twelve mining companies for Maricunga; in July, Bema was chosen and in September a letter of intent was signed. Field work commenced the following month.

Phase I of the Bema program was designed to test the gold mineralized zones on Verde, which showed much greater depths of oxidation than Pancho. Since 1989, drilling has been the primary means of exploration. Drilling on the property is further summarized in Section 11.

11.0 DRILLING

Historically, most of the drilling at Maricunga consisted of reverse circulation drilling. The destructive nature of this drill method made identification of lithology, structure and alteration difficult. The 2002 - 2003 drilling consisted primarily of diamond drill core, providing site geologists with an opportunity to refine the geology model of the deposit.

The preferred orientation of drilling for the Verde deposits is 060 - 240° azimuths. At Pancho, the preferred orientation for the drilling grid of 038 – 218° was established based on the northwest strike of veinlet swarms observed in drill roads. The surface expression of the Pancho system is approximately 700 m by 800 m. The depth for the most recent drill holes was generally determined using a $600 gold pit and the limits of the porphyry mineralization at depth have not been clearly defined as of yet. The mineralization being disseminated in nature also implies that its geometry and size are not limited to a vein. Therefore, the concept of true thickness in drill holes does not apply to Pancho and Verde.

The 2006 exploration campaign followed the same field procedures implemented in 2002-2003. A total of 667 holes (103,392 m) of drilling has been completed on the Verde deposit with an additional 210 holes (56,748 m) completed at Pancho. The drilling has resulted in a drill spacing of approximately 50 x 50 meters at Verde and 75 x 75 meters at Pancho. Table 11-1 summarizes the various drill campaigns at Verde and Pancho.

Table 11-1: Exploration Drill Summary

Company	Year	Verde				Pancho			
		RC (#)	RC (m)	Core (#)	Core (m)	RC (#)	RC (m)	Core (#)	Core (m)
Bema	1989	45	5,060	-	-	-	-	-	-
Bema	1990	231	46,705	31	4,083	-	-	-	-
Bema	1991	-	-	6	1,090	24	5,088	-	-
Subtotal		**276**	**51,765**	**37**	**5,173**	**24**	**5,088**	**-**	**-**
CMM	1993	176	5,060	-	-	-	-	-	-
CMM	1994	-	-	6	4,083	-	-	-	-
CMM	1997	-	-	-	-	15	4,296	-	-
CMM	1998	-	-	18	6,689	-	-	-	-
Subtotal		**176**	**5,060**	**24**	**10,772**	**15**	**4,296**	**-**	**-**
CMM	2003	20	3,154	134	27,468	39	6,710	69	14,146
CMM	2006	-	-	-	-	18	4,012	45	22,496
TOTAL		**472**	**59,979**	**195**	**43,413**	**96**	**20,106**	**114**	**36,642**

It should be noted that all the RC holes drilled in 2006 were effectively pre-collar holes down to a depth of approximately 200 m. The holes were completed with a diamond drill rig to depths averaging 400 m. Figures 11-1 and 11-2 are plan maps of Verde and Pancho showing the completed drill holes as at December 31, 2007.

11.1 Rig Setup and Survey

All proposed drill locations were laid out in plan and on section. The collar coordinates were provided to the mine survey crews who laid out the drill pad locations in the field using total station theodolites. Pad locations were verified prior to construction to ensure access and safety. After pad construction, the mine survey crews established the collar location and marked it in the field. They also established the front sight and back sights necessary to provide the drill direction. After the drill was moved onto the setup and prior to the start of drill, geological staff verified the drill alignment and inclination using a compass. The survey crew later verified alignment and inclination when the drill hole was in progress.

Downhole inclinometry was completed at the end of each hole. Gyroscopic azimuth and inclination readings were taken every 10 meters down the hole to within ten meters of hole bottom and every 50 meters back up the hole as a double check.

11.2 Topographic Base and Survey Audit

In 2002-2003, all field surveys were tied into the established mine grid. Survey data was incorporated into current as-built plans that were updated and maintained in AutoCAD by the survey crews. Guillermo Contreras and Sons Limitada (Santiago), licensed Chilean surveyors, completed a survey audit that verified an approximate 10% of the drill collars using a differential GPS survey system. No significant errors were noted.

11.3 Core Handling, Storage and Security

Drill core was placed in labelled boxes, fitted with lids and transported to the site logging facility by pick-up. All core logging was completed at a facility located at the Maricunga mine site. Only CMM personnel worked with the drill core.

Figure 11-1: Verde Drilling Coverage as December 31, 2007

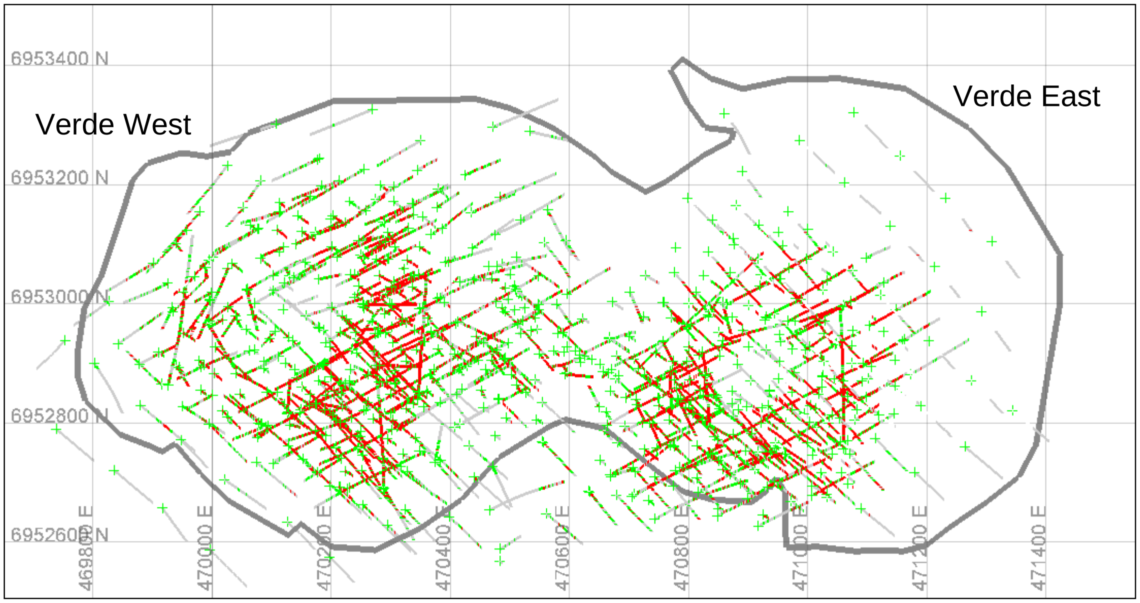
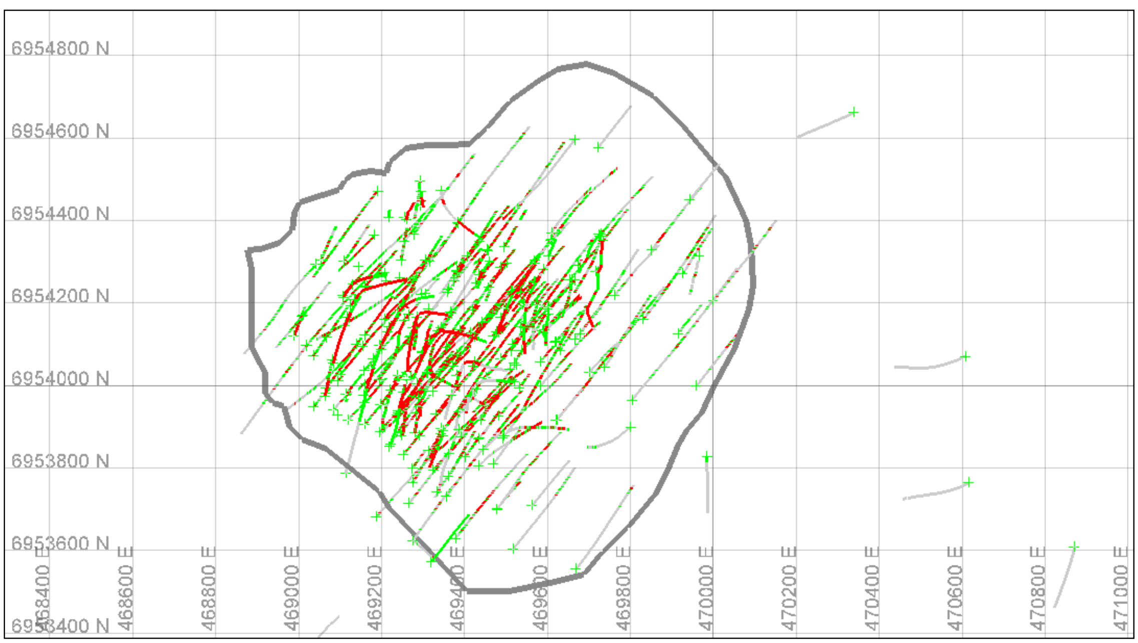


Figure 11-2: Pancho Drill Coverage as at December 31, 2007



12.0 SAMPLE METHOD AND APPROACH

12.1 RC Drilling

CMM provided all of the technical support for the sampling, geologic logging, and drill supervision. Rig geologists and samplers were responsible for the quality/accuracy of each sample. Geologists and samplers typically had up to 15 years experience sampling.

The 2003 drill program adopted a 2.0 meter standard sample length for all samples. The same procedure was followed for the samples in 2006.

When drilling dry, reverse circulation drill cuttings were directed from the cyclone to a Gilson splitter where the sample was reduced to a 6-8 kg sample. Typically, this required a $^{1}/_{8}$ split of the drill cuttings. The hole and cyclone were blown clean after each rod change. Two samples, an original and a duplicate, were produced for each 2.0 meter interval. The samples were weighed, and the weight, number of splits and hole diameter were recorded. The samples were bagged and tagged and prepared for transport to the on-site sample preparation facility. The splitter was air cleaned after each sample.

When sampling wet, a rotating wet sampler replaced the Gilson splitter below the cyclone. Usually ground water flow was not sufficient and make up water was necessary to provide uniform flow to the splitter. As with the dry sample collection, two samples (6 – 8 kg.) were collected in perforated sample bags at the exit port of the wet splitter. These samples were not weighed immediately but were left to drain for a 24 hour period prior to transport to the sample preparation facility.

It should be noted that the drill holes completed in the winter of 2006 did not encounter any water.

Typical chip trays were produced for logging and future reference. In addition, a 100 g, unwashed sample (reference sample) was collected for each 2.0 meter interval. All chip trays and reference samples were placed in permanent storage at site.

The samples are considered representative by the author who has visited the drill sites on many occasions.

12.2 Core Drilling

Drill core was received and laid out for logging and sampling at the core logging facility. The first sample of each core hole was marked from the start of core recovery to the end of the first even meter. From that point, samples were laid out every 2.0 meters downhole. Sample numbers were marked on the core box edges, accounting for number gaps for QA/QC sample insertion. Logged and marked boxes of whole core were transferred to the ALS Chemex sample preparation facility where chain of custody was transferred from CMM to ALS Chemex.

CMM received the sawn core, in the original labelled boxes, from ALS Chemex after splitting. The half-cores were stored shipping containers at the Maricunga site. In all, ten containers were used to store the half-core library.

Core sample rejects are bagged in plastic, labelled and received from the ALS Chemex sample preparation facility. These bags are placed in cubic meter wooden bins. The bins are labelled according to hole number and sample range. The bins are stored beside the core containers and are locked in a covered storage compound.

The samples are representative of the mineralization intercepted at Pancho. The author visited the drill sites at a minimum once a month during the 2006 exploration. The author is satisfied that all established procedures were properly followed by the field staff.

12.3 Logging

Experienced geologists logged the reverse circulation chips and drill core at Maricunga. Reverse circulation chips were typically logged at the drill while core was typically logged at the core logging facility. All logging utilized standardized logging forms and a geological legend developed for the Verde and Pancho deposits. The legend has evolved from historic observation and is consistent with both the regional and local geology. The legend and logging records lithology, alteration, structure, geology, mineralization and oxidation (of pyrite) for each two meter sample interval, reducing the geological descriptions to numeric or alphanumeric codes. Unique features not accounted for in the legend are noted in written comments. The legend stresses oxidation conditions of pyrite, veinlet mineralization, location and amount of magnetite and occurrence of typical porphyry alteration minerals.

In 2002-2003, Golder Associates from Santiago (Golder) identified all holes requiring detailed geotechnical logging that was completed by Golder trained staff. CMM geologists typically recorded both Rock Quality Designation (RQD) and recovery data for each hole. Recoveries averaged better than 95% during the 2003 program. In 2006, average core recovery was 96%.

12.4 Geotechnical Core Logging

Golder managed the geotechnical program. Francisco Carrasco, (Project Manager) and Joaquin Cabello (Assistant Manager) supervised the filed programs and provided the initial training. The program included detailed geotechnical logging of selected drill holes, uniaxial compressive strength (UCS) testing of selected cores, completion of a seven (7) hole oriented core program and cell mapping of existing pit walls and road cuts. Golder also managed a hydrology study including installation of piezometer wells for monitoring water levels.

Golder provided all initial training and completed frequent site visits to maintain quality control of the work and upgrade training of the field staff performing the work.

CMM adopted identical geotechnical logging procedures in 2006 under the supervision of the same geotechnical engineer employed by CMM in 2003.

12.5 Composite Sample Summary

The full composite summary for both Verde and Pancho are shown in Tables 17-4 and 17-5 respectively.

13.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

13.1 Sample Preparation

In 2002-2003, all sample preparation, including core splitting (sawing) was performed and supervised by ALS Chemex personnel. ALS Chemex established, equipped and staffed a portable sample preparation facility at the Maricunga mine for the duration of the program. The sample preparation facility included two rock saws, three drying ovens, two Rhino jaw crushers and a Jones riffle splitter.

After logging and sample mark up, the core boxes were sent to the sample preparation facility. ALS Chemex personnel physically sawed the core. The core splitter randomly selected the split line. Core splitters were trained to maintain the same cut line on contiguous pieces of core wherever possible. After splitting, one half of the core was placed in sample trays along with the sample tag. These were placed in the drying ovens for drying. The other half of the core was fitted back into the core box and returned to CMM for placement in permanent storage.

All samples were dried for 4 – 6 hours at 70° Celsius. For wet reverse circulation samples, the dry time was increased to 24 hours. After drying, sample weight was recorded and the sample was single pass crushed to 90% passing 2 mm using the Rhino Jaw crushers. The crushers were cleaned with air after each sample and cleaned with barren quartz after every five (5) samples. Crush size was monitored after every 20 samples to ensure that the final product met the crush size specification.

After crushing, the entire sample was passed through a Jones riffle splitter, reducing the sample size down to a nominal 1.0 kg size. A preparation duplicate was collected every forty (40) samples. The duplicate was distinguished by an “A and B” designation and treated as per the standard sample procedure. The coarse rejects were returned to the original sample bag, labelled and returned to CMM for storage.

The 1.0 kg samples were sealed in plastic bags, labelled, packed in crates and transported to the primary analytical facility (ALS Chemex – La Serena) for analysis. The laboratory has ISO 9001-2000 accreditation.

13.2 Sample Preparation 2006

In 2006, only core splitting (sawing) was performed at the Maricunga site. All samples were shipped to Actlabs in La Serena where sample preparation was performed using exactly the same procedure as described above. Actlabs also has an ISO 9001-2000 accreditation.

13.3 Analysis

The prepared samples were received at the ALS Chemex's facility in La Serena (the primary analytical lab for the duration of the program) where analyses for Au, Ag, Cu and cyanide soluble Au and Cu analyses were performed.

On receipt of the samples the work order numbers and sample numbers were recorded and the samples were arranged for analysis based on the individual sample numbers. The entire 1.0 kg sample was pulverized to 95% passing -150 mesh using an LM-2 pulverizer. The pulverizer was cleaned with compressed air after each sample. Sieve analyses were performed after every 20th sample to ensure the final product met the stated specification of 95% passing –150 mesh.

After pulverization, the sample was reduced to a 300 gram sample and 700 gram reject. The reject placed in labelled plastic bags and returned to CMM for storage.

The 300 gram sample is homogenized. After homogenization:

- a 50 gram aliquot is selected for fire assay with atomic absorption finish for gold,
- a second aliquot is selected for total digestion atomic absorption analyses for silver and copper,
- a 20 gram aliquot is selected for cold cyanide soluble gold analysis, leached for 4 hours with 60 ml of cyanide solution (0.1% NaOH & 0.5% NaCN).
- A 1 gram aliquot is selected for cold cyanide soluble copper analysis, leached for 2 hours with 20 ml of cyanide solution.

Dr. B. Smee (Smee and Associates Consulting Ltd.) completed operational audits of the La Serena lab for the 2002-2003 program. The operational audits were performed measuring compliance with analytical best practices as well as to NI-43-101 requirements with respect to quality control and quality assurance. Dr Smee did not note any significant problems with this facility, concluding that ALS Chemex's lab and procedures met the highest industry standards.

Initially, ALS Chemex assigned work order numbers at the mine site but in March 2003 a Lab Information Management System (LIMS) was adopted and work order numbers were generated from a centralized ALS Chemex location.

Results containing Au, Ag, Cu or CN Au, CN Cu were sent via digital mail to the Project and Office Managers. Attachments included 1 file per hole, identified by work order number and hole number. The file included internal ALS Chemex blanks and duplicates, while results of ALS Chemex standards were reported monthly in a separate file.

ALS Chemex was also requested to send from a list chosen by CMM 3 batches of pulps (1/40 samples) to Assayers Canada in Vancouver. Dr. Smee audited the Assayers laboratory in February 2003 and found them acceptable to qualify as a referee lab for the project.

13.4 Quality Control / Quality Assurance

13.4.1 Drilling 2002-2003

The original QA/QC program was designed and monitored by Dr. B. Smee. The program included the insertion of one blank, one standard, and one duplicate (randomly selected) for each batch of 40 samples. Standards were not available for insertion for the first 3 weeks of the program. The standards were designed by Dr. Smee and provided by Rocklabs, New Zealand. Dr. Smee selected the standards to best match Maricunga rock types and grades. Standards were designed to monitor gold grade only. The standards were not blind to the analytical lab.

Duplicate samples were randomly selected within the sample batches. Reverse circulation duplicates were taken as an opposite split post cyclone and given a sequential sample number. RC duplicates were not taken below the water table. As the duplicate was generated in the field, no ALS Chemex personnel could identify these samples as duplicates. For core, two duplicates of one interval were taken. The first (d1) was a second split of post crusher product. The second (d2) was the actual other half of sawn core. The intent was to see the variance according to crushing procedure (d1) and also the variance according to natural geological variance + crushing procedure. This would also allude to any bias in core sawing activity. In the core duplicate samples, obviously ALS Chemex sample preparation personnel knew which samples were being duplicated, while ALS Chemex analytical personnel did not.

ALS Chemex also inserted their own blanks, standards and duplicate samples for each sample batch as part of the labs own internal quality management program.

13.4.2 Drilling 2006

The same overall QA/QC procedures were implemented during the 2006 exploration campaign. The results were compiled as soon as received by the site staff. Results for standards were plotted using two standard deviations as lower and upper limits.

The charts were kept updated and any anomalous results were discussed with Actlabs.

13.4.3 Table of Failures

The Senior Geologist extracted the QA/QC sample results from the digital result files based on the sample numbers. If a standard assay fell outside 3 standard deviations of the standard value, the entire batch of 40 samples was rerun. If 2 consecutive standard assays fell within 2 standard deviations of the standard value, a batch rerun was requested. If a blank exceeded 0.05 g/t au, a batch rerun was requested. No reruns were determined from duplicate data.

Monitoring of the QA/QC data required a total of 17 batch reruns (649 samples) based on standards failures. This represented approximately 2% of the total analyses run for the project. Inspection of batch failure results indicates minimal bias with both gold and copper relative to the repeat batch assays. At the end of the project, an additional nine QA/QC batch assays (347 samples) were selected based on standards failures. Three additional batches plus 11 individual samples (127 total) were found to have CNAu at least 0.10 g/t au greater than the total gold assay. These were also rerun for both total and cyanide.

13.4.4 Analysis of QA/QC Data

For the 2002-2003 drilling program all standard, blank and duplicate results were sent to Dr. Smee for analysis. Duplicate results were analyzed for precision and plotted on Thompson Howarth graphs. Dr. Smee found the error rate and duplicate variance normal for porphyry gold deposits. In his final report, (July 20,2003) he concluded the gold assay database to be of commercial quality but raised concerns about eight of the 650 ALS Chemex digital files. All eight files, containing failures, had already been reanalyzed, were determined to have had CMM field labelling errors, or were duplicated assay sets.

The 2006 QA/QC data was compiled as it was received by the geologists at the site. Any anomalous results were flagged and the lab contacted directly to discuss the variations observed. A series of samples representing approximately 10% of the total were also sent to ALS Chemex as an independent check.

13.4.5 Referee Laboratory

Assayers Canada (Assayers), Vancouver, was selected as the referee laboratory for the program. Assayers Canada has achieved Certificates of Laboratory Proficiency from the Standards Council of Canada for precious and base metal analysis. One pulp per ALS Chemex batch of 40 was selected at random, including the CMM standards, blanks and duplicates. Additionally one of the 3 CMM standards was inserted in this sample stream to check Assayers at a frequency of 1 per 40. Three batches of checks (631 samples in total) were sent to Assayers over the duration of the program. Results were forwarded to Dr. Smee who concluded that the two labs essentially agreed on gold and cyanide soluble gold analysis within normal precision limits.

13.4.6 Statement of Author’s Opinion

The author concurs with Dr. Smee’s opinion that the slight variations observed were within normal precision limits.

14.0 DATA VERIFICATION

14.1 General 2002-2003

Kinross Technical Services verified the Initial Database (IDB) to copies provided by Bema Gold Corporation and MRDI.

Original drill logs, assay certificates, survey worksheets and other original data were located and used to verify the data contained in the IDB. Verification of the historical data included:

- Collar location
- Collar azimuth and dip
- Downhole azimuth and dip
- Total gold and copper analyses
- Cyanide soluble gold and copper analyses
- Primary lithology code
- Primary alteration code
- Primary oxidation code

Assay certificates, and original logs were found for approximately 80% of the historical data.

For the 2006 Pancho drill program a system of double entry was established. It included all of the items listed above plus the geology logs as prepared by the field geologists. The final grade database was compiled by Kinross Technical Services (KTS). KTS and CMM assumed that the historical data used in the past was sufficiently free of errors or omissions. Details of the data verification process and database management for 2002-2003 are provided in the following paragraphs.

14.2 Pre-2002 Data Verification

14.2.1 Collar Data

Verification of the historic collar data identified one error of note. Collar coordinates of drill holes completed between 1986 and 1987 did not match the coordinates recorded on original survey sheets. This error was also corrected.

14.2.2 Survey Data

Much of the historical downhole survey data could not be located. The limited downhole survey data that was located was used to verify the data in the IDB. No significant errors were noted.

14.2.3 Lithology, Alteration, Oxidation, Vein and Mineralization Data:

Ten percent of all lithology, alteration, oxidation, vein description and mineralization data for the 1997 – 1998 drill campaign was verified as this data was collected using field practices and logging formats consistent with the 2002-2003 effort.

14.2.4 Analytical Data

Ten percent of the historical analytical results in the IDB were verified against original assay certificates. Rare errors were detected during the verification process but these were almost exclusively related to the recording of results below the detection limit for successive programs.

14.3 2002-2003 Data Verification

14.3.1 Collar Data

Prior to entering collar data into the database, collar locations were visually checked for obvious errors such as transposed northings and eastings. Once entered, collar coordinates and hole lengths were extracted and printed for direct comparison to the original survey certificates. The collar coordinates were compared against the signed surveyors' certificate and the hole length were checked against the geologists' logs. Three independent checks of this information yielded rare errors. Collar information was also digitally checked against the surveyors' final compilation file; this check also yielded rare decimal place errors. Erroneous data was corrected and reloaded. The corrected data was subjected to two additional independent checks.

14.3.2 Survey Data

Prior to entering the downhole survey data into the database it was visually scanned for obvious errors. After entry, the data was extracted and printed for manual verification relative to the original downhole survey data contained in the files and provided by Comprobe. Data verification of 10% of the downhole survey data was completed for the survey data. Two independent checks of this data detected no errors. Visual inspections of the plotted hole traces also failed to identify any obvious errors or miss-plots.

14.3.3 Lithology, Alteration, Oxidation, Vein and Mineralization Data

Lithology, Alteration, Oxidation, Vein Description and Mineralization data collected during the 2002-2003 field season was verified against the original logs. A 10% check of the database yielded an unacceptable amount of errors and led to a 100% check. The 100% check yielded an error rate of 3.5%, with most of the errors related to the recording of alteration. These errors were corrected.

14.3.4 Analytical Data

A random 10% of the analytical data was verified against the original assay certificates. Only rare errors were detected, usually pertaining to how results below detection were entered in the database. Erroneous data was corrected and re-entered.

14.3.5 Geotechnical Data

Ten percent of the geotechnical data collected during the 2002-2003 field program was verified against the original geotechnical logs yielding a significant error rate. As a result, a 100% verification process was initiated. Significant error rates were noted in the fractures-per-meter field. Error rates in the structural measurement and RQD fields were related to format errors rather than data entry errors. These errors were corrected.

The QA/QC Database Manager has certified that the database as transferred to Kinross Technical Services on CD-R media is free of gross errors and omissions and is suitable for estimating resource and reserve estimates.

14.4 Data Entry, Storage, Management and Security

The QA/QC Database Manager was responsible for updating and managing the Project Database (PDB). Only the QA/QC Database Manger could access, update or otherwise modify the PDB. Regular backups were made to CD-R disks over the duration of the program. The final PDB was transferred to Kinross Technical Services to be used in resource modeling.

14.4.1 Data Entry

Collar Coordinates

The mine survey department entered collar coordinates, drill azimuth and inclination into spreadsheets that were provided to the QA/QC Database Manager. The QA/QC Database Manager checked the spreadsheet data against original survey field notes prior to importing the data into the PDB. Cross sections and drill hole composite plans were plotted using Gemcom software and were inspected by the Senior Geologist for collar and directional errors. Final hole depths were verified against the drill log hole depth.

Downhole Survey Data

Comprobe provided the downhole survey data in spreadsheet format to CMM's Office Manager who checked the data for obvious errors. Once satisfied that the downhole survey data was correct, the Office Manager provided the spreadsheet files to the QA/QC Database Manager for import into the PDB.

Lithology, Alteration, Mineralization and Oxidation

The Office Manager and Data Entry Clerks entered data from the field lithology logs into spreadsheets, one for each hole. A double-blind entry system was not used in recording this data. A 100% manual check of this data was performed at the end of the drill program.

Assay Data

Assay results were received as digital files, identified by work order number and hole number. The assay files included individual sample numbers allowing the samples to be matched to the sample intervals. The digital files were imported into spreadsheet files that were verified back to the original assay files prior to being imported into the PDB.

QA/QC Data

The Senior Geologist matched assays to sample numbers on the original log and extracted the QA/QC data from the sample stream into an excel spreadsheet which was used to monitor QA/QC data independently from the sample stream. The QA/QC data was sorted into worksheets for blanks, standards and duplicates. The entire file was cross-checked between the Project Manager’s copy and the Senior Geologist’s copy. Once satisfied that the data was free of error it was sent via email to Dr. Smee for final analysis.

14.4.2 Data Storage

The PDB was established and maintained using Gemcom modeling software. The PDB was backed up on CD-R media on a regular basis for the duration of the field program. Once the PDB was fully updated with all the results collected during the 2002-2003 programs, the data was copied and forwarded to Kinross Technical Services in CD-R format. This database is referred to as the Final Database (FDB).

In addition to the digital files, all available hard copy data was organized and inventoried on a per hole basis. This information was copied in triplicate with one copy forwarded to Kinross Technical Services, one copy forwarded to Bema and a final copy stored at the Maricunga Mine. Original hard copy data was packaged and stored at the CMM office in Copiapó.

14.4.3 Data Management and Security

Only the QA/QC Database Manager had the necessary file access permissions to update the PDB. Only Kinross’ Manager of Technical Services could access and modify the FDB.

15.0 ADJACENT PROPERTIES

The Maricunga Project is located within the Maricunga Gold Belt of northern Chile. Since 1980, a total of 40 million ounces of gold have been defined in the gold belt, (Muntean and Einaudi, 2000).

The Maricunga Gold Belt hosts numerous mineral deposits of economic interest including:

- La Coipa, an epithermal gold-silver mine owned and operated by Kinross
- Marte-Lobo, a gold porphyry deposit that was partially developed by Teck Corporation and,
- Cerro Casale, a large (1.035 billion tonnes), undeveloped, porphyry copper-gold deposit that is a joint venture between Barrick Gold (51%) and Kinross (49%).

No information from adjacent properties has been used in the exploration program or in the estimation of the mineral resource.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

The 2002-2003 Exploration Program included the following metallurgical and physical property test work for both Verde and Pancho:

- cyanide soluble gold analyses
- cyanide soluble copper analyses
- direct agitated cyanidation tests (bottle roll)
- column leach tests
- Bond work index tests
- specific gravity determinations
- Acid Based Accounting (ABA) tests

The 2006 drill program on Pancho also provided representative material to perform additional metallurgical testing. Cyanide soluble gold and copper analyses were systematically completed for all samples. Recent test work included bottle roll and 14 column leach tests. The results based on the 2006 data collection are summarized separately in this section.

Finally, during 2007 the SART process was considered for the Pancho ore as uneconomic concentrations of copper are present in ore. Some of the copper minerals are soluble in cyanide solutions and these minerals consume cyanide reagent and contaminate the leach solution with copper metal. A feasibility study and pilot testing program were completed in 2007. The study considers plant design, performance metrics, operating and capital costs and implementation strategy. The reserves defined at Pancho consider the application of the SART process.

16.1 Metallurgical Test Work 2003

16.1.1 Cyanide Soluble Assays

Both gold and copper cyanide soluble analyses were routinely analyzed for Verde and Pancho. Results were used to guide selection of samples for bottle roll and column leach tests. Cyanide copper analyses were also used to adjust process operating costs during pit optimization of the resource model for Pancho. The mechanics of the Process Cost Adjustment Factor (PCAF) are discussed elsewhere in this report.

16.1.2 Direct Agitated Cyanidation (Bottle Roll) Tests

Bottle roll composite samples were selected from the half core library based on lithology, oxidation state, grade and spatial location. A total of 43 bottle roll samples were selected from the Verde with an additional 15 samples collected from Pancho. The samples were shipped to McClelland Laboratories, Nevada for testing.

The gold grade selection criteria used for selection of the bottle roll tests were:

- Low grade 0.50 to 0.75 g/t
- Mid-grade 0.75 to 1.00 g/t and
- High grade >1.0 g/t

Oxidation state was determined based on the visual logging of oxidation noted in the geological logs. Three distinct oxidation levels were defined.

- Oxide >90% oxidation of sulphide
- Mixed between 10% and 90% oxidation of sulphides and
- Sulphide <10% oxidation of sulphides.

Samples were selected from the four major ore bearing lithologies, post mineral intrusives and Laguna tuffs were not sampled.

Sample selection was restricted within an optimized pit shell generated at a US $400 gold price. Table 16-1 summarizes the Bottle Roll sample selection.

Results of the bottle roll tests show distinct differences in recovery based on the level of oxidation. Oxide ores exhibit higher recoveries than mixed ores which, in turn, demonstrate higher recoveries than the sulphide ores. These findings agreed with the historical metallurgical test work and experience gained from mining in the Verde area. A summary of the bottle roll test results is presented in Table 16-2.

Table 16-1: Summary of Bottle Roll Sample Selection

Deposit		**Gold Grade**		
		Low (0.50 - 0.75)	Medium (0.75-1.00)	High (>1.00 g/t)
Pancho	# of samples	5	5	5
Verde W	# of samples	5	5	11
Verde E	# of samples	9	5	10

Deposit		**Oxidation**		
		Oxide (>90%)	Mixed (10-90%)	Sulphide (<10%)
Pancho	# of samples	6	3	6
Verde W	# of samples	8	5	8
Verde E	# of samples	9	6	9

Deposit		**Lithology**			
		Diorite	Breccia	Mineralized Post Mineral Intrusive	Dacite Porphyry
Pancho	# of samples	6	9	-	-
Verde W	# of samples	-	7	5	9
Verde E	# of samples	-	9	6	9

Table 16-2: Summary of Bottle Roll Results

Deposit	**Oxidation Class**	**Au Recovery (%)**
Verde	Oxide	72.3
	Mixed	63.8
	Sulphide	53.0
Pancho	Oxide	83.0
	Mixed	72.8
	Sulphide	68.6

16.1.3 Column Leach Tests

Results of the bottle roll tests were used to target specific areas of the deposit with the HQ diameter drill holes that were used to collect the sample material necessary for column leach analysis. Composite samples were collected from whole HQ diameter core that was drilled specifically for the metallurgical test program. A total of 10 HQ diameter holes (1,500 meters) of core were drilled at Verde for the purpose of column leach tests. An additional seven (7) samples were collected at Pancho but unlike the Verde samples, the Pancho samples were taken from remaining half cores available at the time of sample selection.

For the Verde samples, column tests were established for three different product sizes (6.3, 9.5 and 15.8 mm). The same sample selection matrix established for the bottle roll tests guided column leach sample selection. Grade of the HQ diameter core was confirmed by cutting a sliver sample for each two-meter interval. Table 16-3 summarizes the Column Leach Sample program.

Table 16-3: Summary of Column Leach Sample Selection

Deposit		Gold Grade		
		Low (0.50 - 0.75 g/t)	Medium (0.75 – 1.00 g/t)	High (>1.00 g/t)
Pancho	# of samples	2	2	3
Verde West	# of samples	2	2	1
Verde East	# of samples	2	2	2
Deposit		**Oxidation**		
		Oxide (>90%)	Mixed (10-90%)	Sulphide (<10%)
Pancho	# of samples	3	1	3
Verde West	# of samples	-	-	5
Verde East	# of samples	3	3	-

Results of the column leach tests matched the bottle roll results. Table 16-4 summarizes the results of the column leach tests completed as part of this study.

Figures 16-1 and 16-2 are plan views of the Verde and Pancho pits showing the original metallurgical sample distribution within the 2002-2003 design pit shells.

Table 16-4: Summary of Column Leach Test Results

Deposit	Ore Type	Crush Size (k_{80} mm)	Au Recovery (%)
West Verde	Sulphide	15.8	44.0
		9.5	53.7
		6.3	57.2
East Verde	Mixed	15.8	70.5
		9.5	73.6
		6.3	79.3
East Verde	Oxide	9.5	77.0
Pancho	Sulphide	9.5	71.6
Pancho	Mixed	9.5	79.8
Pancho	Oxide	9.5	89.0

Figure 16-1: Metallurgical Sample Locations – Verde Pit – 4300 Level

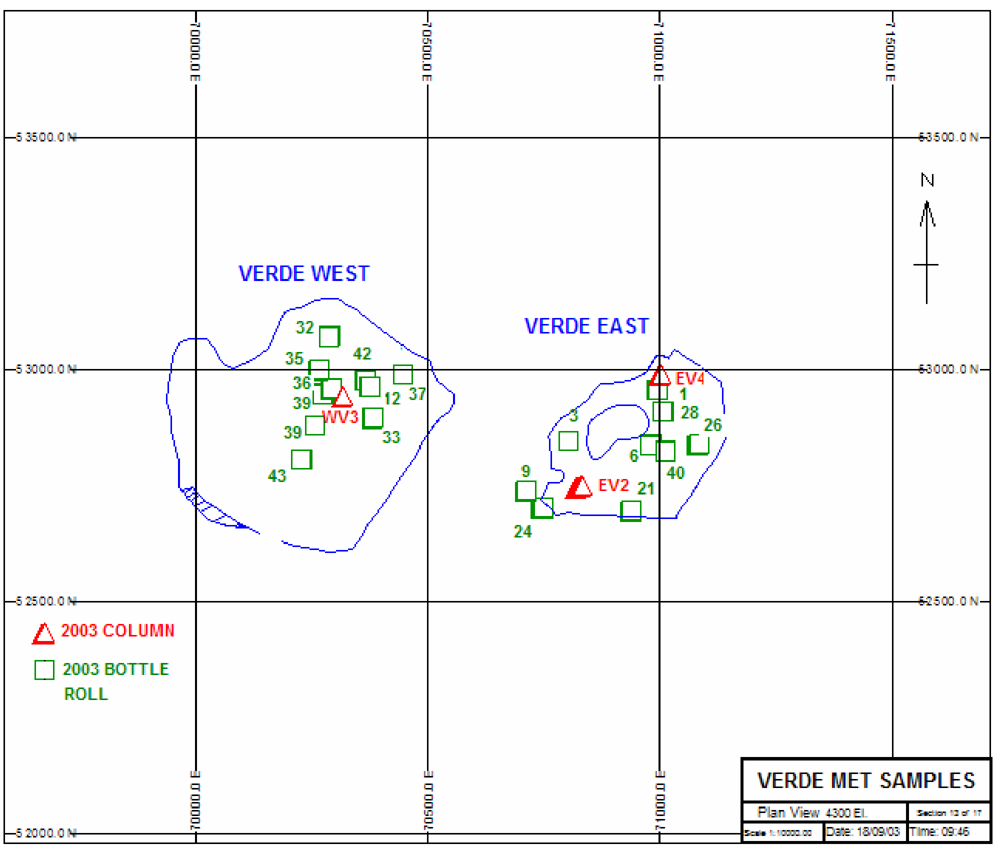
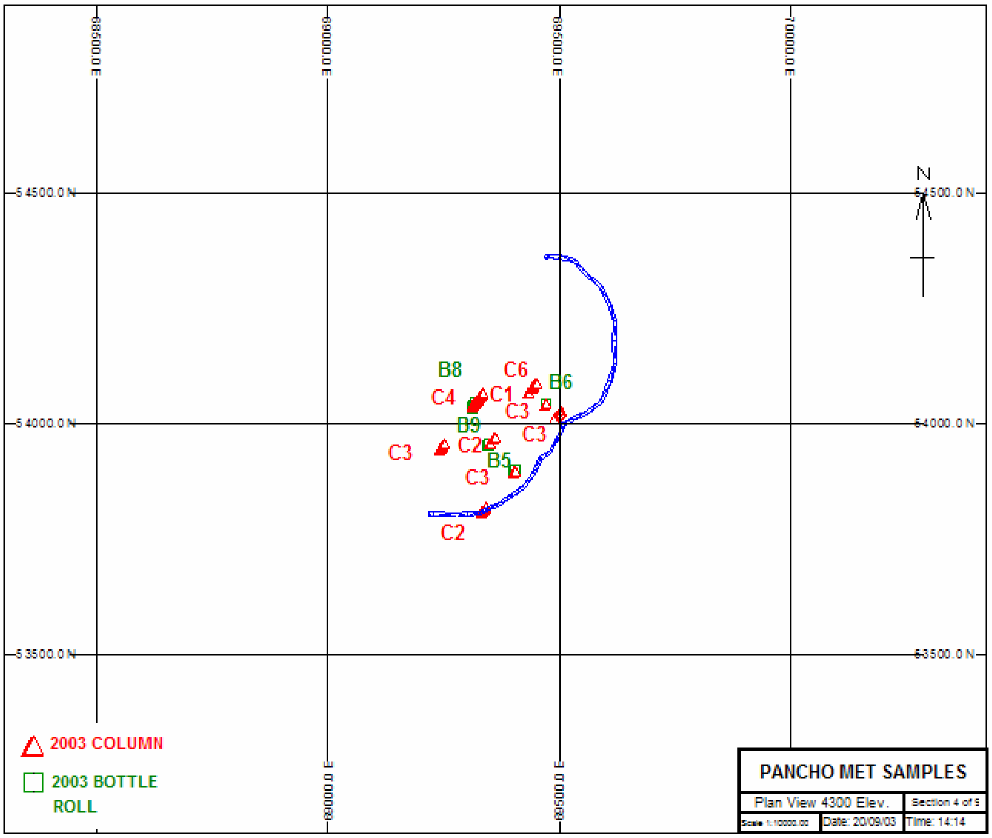


Figure 16-2: Metallurgical Sample Locations – Pancho Pit – 4300 Level



16.1.4 Bond Work Index

Bond Work Index (BWI) samples were collected from the 10, HQ diameter core holes drilled at Verde. Bond work index tests required selection of whole HQ core pieces of approximate length of 0.15 – 0.20 meters. For Verde, such pieces were selected every 4 meters down-hole. All ten Verde HQ diameter holes were sampled in this manner, a total of 296 samples. At Pancho, BWI samples were collected from the standard diameter drill holes. Whole core pieces were taken approximately every 20 meters down hole. A total of 67 samples were selected. All samples were shipped to Hazen Research, Colorado, for analysis.

Results of the BWI tests indicated limited variability in the rock mass therefore; the BWI data was not modeled when completing the Verde and Pancho resource models.

16.1.5 Specific Gravity

As part of the BWI tests above, Hazen completed specific gravity analyses on all the BWI samples submitted. In addition to this data, 1171 samples of half core were selected from the half core library. Samples, roughly 15 cm in length, were selected approximately every 30 meters downhole throughout the entire campaign. The samples were shipped to ACME laboratories, Santiago where they were analysed using the wax dipped immersion method. A summary of the specific gravity results is presented in Table 16-5.

Table 16-5: Specific Gravity Results

Deposit	Unit	SG
Verde	Oxide	2.45
	Mixed	2.52
	Sulphide	2.57
	Laguna Tuff	2.00
	Post Mineral Intrusive	2.60
	Waste Dump Material	2.00
Pancho	Oxide	2.27
	Mixed	2.38
	Sulphide	2.49

Both Pancho and Verde exhibit trends of increasing Specific Gravity with depth, leading to the decision to model the deposit Specific Gravity by the oxidation surface rather than by rock type.

16.1.6 Acid Rock Drainage Study

At the request of the CMM environmental department (Jorge Herrera), ALS Chemex La Serena sent 71 pulp samples to ALS environmental laboratories Santiago for acid based accounting testwork. A subset of 40 of these was proposed for testwork. The forty samples selected were from widely spaced locations in the anticipated pit designs for both Verdes and Pancho. Samples included both ore and waste in major rock types in all redox zones.

In 2007, follow up work included a sample collection program comprising 150 samples. To date partial results are available for 140 samples. Representative samples will be selected for further wet cell analysis work once the dataset is complete.

16.2 Metallurgical Test Work 2006

16.2.1 Introduction

Gold recovery rates for the Verde and Pancho deposits have been estimated from test work conducted at McClelland Laboratories in Reno, Nevada. In 2003, a total of 9 column leach tests were done on 7 Pancho composite samples and 11 column leach tests were done on 11 Verde composite samples. In 2006, an exploration drill program on the Pancho property provided samples for an additional 14 column leach tests at McClelland. Final test data was available by August 2007 and these gold recoveries have been integrated in the Pancho reserve estimate.

16.2.2 Comments on Historical Metallurgical Composite Selection Criteria

The early work on Maricunga metallurgical type classification established that the most predominant metallurgical criteria were grade, oxide state and crush size. The 1998 metallurgical column testing relied on the classification of four groups of sample types largely predicated by the amount of sheeted and stockwork veining as well as silicification. The four groups are outlined below.

- Group 1 – 81.4% recovery – Strong sheeted veining, moderate strong stockwork, and weak matrix silicification.
- Group 2 – 72.9% recovery – Moderate-strong sheeted veining, moderate stockwork, weak matrix silicification.
- Group 3 – 66.1% recovery – Weak-moderate sheeted veining, weak stockwork, and weak to moderate matrix silicification.
- Group 4 – 50.4% recovery – Weak-nil sheeted veining, weak-nil stockwork, and moderate strong silicification.

These groupings were specific to Verde, where East Verde was predominantly represented by Group 2, while West Verde was mostly comprised of Group 3.

In 2003, the methodology for metallurgical sample selection was changed. Of concern was the issue that the head grades of the group 1 to 4 samples were 20 to 40% above average grade of the deposit and it was suspected that higher grade column samples would result in higher gold recoveries. In addition, field geologists had concerns with the classification of silicification and the ability to differentiate between stockwork versus sheeted veining in chips from the predominantly RC based database.

As a result of the assessment of the Maricunga database and regression results and historic results, it was proposed to classify metallurgical column test samples, firstly by oxidation intensity, then by grade, and by elevation where appropriate. Due to questions of database suitability and analysis, a lower priority was given to veining, silicification, argillic alteration and lithology.

In 2003, before the final composites were selected for column tests, McClelland Laboratories in Reno completed bottle roll tests on 14 Pancho “point samples” selected to represent the likely metallurgical domains. Samples were differentiated by grade, lithology and oxidation state and subjected to a 96 hour bottle roll leach at a crush size of 80% passing 1.7 mm. The results showed that the oxidation state is the only reliable predictor of heap leach gold recovery. Head grade and lithology do not significantly affect gold recovery. This correlation was borne out in the column tests and the current Pancho mine plan uses gold recovery values based only on oxide, mixed and sulphide ore designations.

16.2.3 Metallurgical Composite Selection Criteria (2006)

In 2006, an extensive drill campaign was completed on the Pancho deposit and drill core composite samples were shipped to McClelland Laboratories in Reno for metallurgical test work. The first stage of test work was to complete seven column tests on Pancho drill core composite samples crushed to 80% passing 6.3 mm. The objective was to test the variability of gold recoveries from ore in zones of the Pancho pit that had not been tested previously. Composites were selected from single holes where possible to maintain the spatial integrity of the sample. The McClelland columns were 6" diameter x 10' tall and held 68 kg of rock. The column tests were run at a crush size of 80% passing 6.3 mm in order to be compatible with the previous test work completed on the Verde deposit.

The second stage of test work was to select a typical sulphide sample and a typical oxide sample from a multi-hole composite and conduct a crush size variability study to determine the effect of particle size on gold recovery. Crush sizes of 6.3 mm, 10mm and 16 mm were selected. The oxide column leach tests have only just begun and no data is available yet. In addition to the column test work, bottle roll cyanide leach tests were conducted on multiple point samples to determine the variability of gold recovery and copper recovery.

The metallurgical composite selection criteria was to obtain lithotypes with representative oxidation and hypogene alteration combined with representative Au and Cu grades in spatially well distributed areas within the original Pancho $450/Au oz pit. The sample summary table below present the details for 2006 Pancho metallurgical composites (see Table 16-7).

Table 16-6: Pancho Metallurgical Composites 2006

Column Test No.	Sample Ref. No.	Hole	From m	To m	Au g/t	Cu ppm	% Oxidized	Lithology	Hypo Alt str-wk	% py	% veins
P1	1	P2-1930	350	474	0.96	239	0	Porphyry	chl ser pot	1-5	0.5 - 1
P2	2	P2-2240	180	288	0.5	548	5	Breccia	pot,chl.ser	0.5-2	0.5 - 1
P3	3	P9.5-1690	40	104	0.71	698	5	Porphyry	chl ser pot	1-2	0.5
P4	4	P7-2140	80	160	0.62	1366	10	Porph./Breccia	clay,ser,chl	1-3	0.5 - 1
P5	5	P9.5-1840	80	150	0.58	755	2	Porphyry	pot, chl	0.5	0.5 - 1
P7	6a	P7-2140	10	50	0.57	103	100	Breccia	arg,jar	0.5	1-3
P7	6b	P10.5-2090	10	44	0.64	50	100	Porphyry	arg,jar,pot	0.5	0.5
P6	6	P9.5-1840	3	70	0.82	1166	50	Porphyry	chl pot arg	1-1.5	0.5-1
P8	8	P8-2175	140	248	0.63	1322	3	Fine Gr. Porph.	ser chl	0.5-5	0.5 - 2
P9	9	P4.5-1730	380	490	0.72	1041	0	Porphyry	chl sil	0.5	0.5-1
P10	10	P5.5-2000	290	400	0.63	504	0	Porphyry	pot chl ser	0.5	0.5
P11	11	P7-2020	300	390	0.59	1264	100	Porphyry	chl ser pot	0.2	1-2
P12	12	P5.5-2100	400	500	0.57	738	100	Porphyry	chl sil	0.5	0.5-1.5
P13	13	P3.2-1770	350	450	0.52	256	100	Porphyry	chl ser pot	0.2-1	0.2
P14	14	P4.5-1990	300	400	0.79	455	0	Porphyry	qtz-ser-chl	0.2-1	0.2

16.2.4 McClelland Test Work Summary

A total of 13 drill core composites and 43 drill cuttings composites were prepared for heap leach cyanidation testing. Another composite was collected at a later date and was selected for its high copper content. Detailed head analyses and direct agitated cyanidation (bottle roll) tests were conducted on all of the composites. The cuttings composites were evaluated at a nominal 1.7 mm (as-received) feed size. The drill core composites were evaluated at 80% -1.7mm and 80% -75µm feed sizes. Select (3) drill core composites were also evaluated at an 80%-6.3mm feed size. A column percolation leach test was conducted on 10 of the 13 drill core composites, at an 80% -6.3mm feed size. Select (2) drill core composites were also subjected to column leach testing at 80% -15.9mm and 9.5mm feed sizes. Metallurgical results from this testing program are summarized below.

The composites evaluated were representative of the various ore types from the Pancho deposit. Head analysis results showed that the cuttings composites contained between 0.35 and 2.54 gAu/mt ore. The drill core composites contained between 0.44 and 0.95 gAu/mt ore. Total copper content of all of the composites ranged from <0.01 to 0.2% Cu. On average, acid soluble copper and cyanide soluble copper represented 17% and 63%, respectively, of the total contained copper.

The Pancho drill cuttings and core composites generally were amenable to direct agitated cyanidation treatment at an 80%-1.7mm feed size. Gold recoveries from the cuttings composites ranged from 43.5% to 97.4%, and averaged 73.0%. Gold recoveries of less than 60% were obtained from only 4 of the 43 cuttings composites. Gold recoveries obtained from the drill core composites at an 80%-1.7mm feed size ranged from 52.9% to 88.6%, and averaged 74.2%.

The 2140 composite was the only drill core composite exhibiting a gold recovery of less than 60%. Grinding of the drill core composites to 80% -75µm improved the average gold recovery to 83.2%. Bottle roll test cyanide consumptions varied substantially, and tended to increase with increasing copper head grade. Cyanide consumption for the 1.7mm feeds ranged from 0.11 to 3.67 kg NaCN/mt ore, and averaged 1.00 kg NaCN/mt ore. Lime requirements varied substantially, and were very high (>10 kg/mt ore) for seven of the composites tested. Overall, 1.7mm feed bottle test lime requirements ranged from 2.2 to 61.9 kg/mt ore, and averaged 7.3 kg/mt ore.

The 10 drill core composites subjected to column leach testing were amenable to simulated heap leach cyanidation treatment, at the feed sizes evaluated. All 10 composites were evaluated at an 80% -6.3mm feed size. Gold recoveries obtained from the 6.3mm feeds ranged from 55.1% to 89.1%, and averaged 72.0%, in from 71 to 174 days of leaching. Gold recoveries of lower than 70% were obtained from the 1840 composite (62.3%), the 2140 composite (56.5%) and the Copper Column Test (CCT) composite (55.1%).

The Maricunga-High Elevation Variable Crush (RHEVC) and Oxide core composites were both also subjected to column testing at 80% -15.9mm and 80% -9.5mm feed sizes, to evaluate crush size sensitivity of the Pancho ore. Gold recovery increased incrementally (3% - 6%), by crushing from 15.9mm to 9.5mm in size. Gold recoveries obtained from both composites were essentially the same at the 9.5mm and 6.3mm feed sizes.

Column test gold recovery rates for the 2240, Pancho 6, CCT and Oxide composites were rapid, and gold extraction was substantially complete in about 25 days of leaching. Gold recovery rates were slower, and gold extraction progressed at a slow, but significant rate after 75 days of leaching, for the 1690, 1840, 1930 and 2140 composites. The slowest gold recovery rates observed were for the 1840 Shallow and RHEVC composites.

Column test cyanide consumptions generally were high, but were not as strongly correlated to copper head grade, as seen during bottle roll testing. Column test cyanide consumptions tended to increase with increasing leach solution application, as normally expected for column leach tests. Because some of the column test leach cycles were relatively long in duration (>140 days), the resulting cyanide consumptions were relatively high. Average column test cyanide consumption (6.3mm feeds) was 2.39 kg NaCN/mt ore.

Column test lime additions were made based on corresponding bottle roll test lime requirement data. Lime requirements for the drill core composites varied substantially. Initial column test lime additions ranged from 1.8 to 13.0 kg/mt ore. Those lime additions were sufficient for maintaining protective alkalinity during leaching, for most of the composites. Minor pH control problems were encountered during leaching of the 2140 and 2240 composites, and relatively small lime additions were made to the column test barren solutions during leaching, to help control leaching pH. Severe pH control problems were encountered during leaching of the Oxide and CCT composites, and the lime added during leaching for pH control was equivalent to more than the lime added to the ore initially. The highest observed combined lime requirement was 41.1 kg/mt ore, for the Oxide composite 9.5mm column feed. Higher initial lime additions should be evaluated for these ore types to determine whether or not pH control will be a problem during commercial heap leaching.

Overall, reagent consumption data indicate potential concerns during commercial heap leaching of the Pancho ore, with respect to high cyanide consumptions related to copper dissolution, and high lime consumptions related possibly to interaction with sulphates naturally occurring in the ore. Follow-up testing was recommended to better establish base requirements for pH control during heap leaching of the higher lime consuming Pancho ores.

Table 16-7: Pancho and Verde Gold Recovery Estimates

Gold Recovery (%)							
Deposit	**Zone**	**2003 Column Test Data**	**SNC Study**	**2004 $350/oz Pit Design**	**2005 $400/oz Pit Design**	**2006 $475/oz Pit Design**	**2006-2007 Final Column Tests**
Verde	Oxide	77	73	67	73	73	
	Mixed	74	70	67	70	70	
	Sulphide	54	53	48	53	53	
Pancho	Oxide	89	80	80	85	85	86
	Mixed	80	75	75	75	75	77
	Sulphide	72	70	70	70	70	70

Pancho Column Test Work Results

Prior to conducting the column leach tests, bottle roll cyanide leach tests were conducted on the samples to confirm metallurgical response and estimate reagent consumptions. The data is summarized in Table 16-8. Gold recoveries and reagent consumption rates were similar to those established in the 2003 testwork. It is clear that cyanide consumption increases with copper content and lime consumption increases with oxidation state. Gold recovery also increases with oxidation state. It is also clear that gold recovery increases with a finer grind confirming that recovery is sensitive to crush size.

The column leach test data is summarized in Table 16-9 and charts of gold and copper extraction versus time are appended. The data compiled to date indicates that gold recovery from the Pancho samples tested in 2006-2007 is in the region of 69% for the sulphide ore and 86% for the oxide ore. This test work (Table 16-9) plus the work completed in 2003 confirms that the gold recovery values used in the 2007 reserve estimate are reasonable (68% for sulphide, 75% for mixed and 85% for oxide).

Table 16-8: Pancho Bottle Roll Summary 2006

Sample Number	% Oxidized	Lithology	1.7 mm crush, 96 hr bottle roll: Calculated Head Au g/t	Gold Recovery %	Calculated Head Cu ppm	Copper Recovery %	Cyanide Consumption kg/mt ore	Lime Consumption kg/mt ore
P1	0	Porph.	1.04	70	248	19	0.45	2.2
P2	5	Bx.	0.47	79	429	53	1.13	6.1
P3	5	Porph.	0.70	70	631	37	0.98	3.7
P4	10	Porph./Bx.	0.70	53	1,225	51	2.20	2.6
P5	2	Porph.	0.55	62	769	22	0.92	2.4
P6	50	Porph.	0.80	73	1,076	54	1.81	7.1
P7	100	Porph./Bx.	0.64	88	111	10	0.18	7.0
P8	0	Porph./Bx.	0.73	64	765	22	0.83	3.2

Sample Number	% Oxidized	Lithology	75 micron grind, 72 hr bottle roll: Calculated Head Au g/t	Gold Recovery %	Calculated Head Cu ppm	Copper Recovery %	Cyanide Consumption kg/mt ore	Lime Consumption kg/mt ore
P1	0	Porph.	1.04	87	364	18	0.37	2.0
P2	5	Bx.	0.54	80	438	54	1.13	4.9
P3	5	Porph.	0.67	84	675	41	1.46	2.9
P4	10	Porph./Bx.	0.67	64	1,102	46	1.96	2.3
P5	2	Porph.	0.59	83	752	20	0.67	2.4
P6	50	Porph.	0.81	83	1,064	62	2.09	6.2
P7	100	Porph./Bx.	0.65	89	116	14	0.37	5.8
P8	0	Porph./Bx.	0.74	84	742	19	0.61	2.7

Table 16-9: Pancho Column Test Summary 2007 (Part 1)

Column Test No.	Composite	% Oxidized	Lithological Type	Feed Size P_{80} (mm)	Leach/Rinse Time, days	Au Rec. %
P1	1930	0	Porphyry Sulphide	6.3	169	78.8
P2	2240	5	Breccia Sulphide	6.3	108	71.1
P3	1690	5	Porphyry Sulphide	6.3	223	77.6
P4	2140	10	Porphyry Sulphide	6.3	193	56.5
P5	P95-1840	2	Porphyry Sulphide	6.3	175	62.3
P6	Shallow Comp P95-1840	50	Mixed Sulphide	6.3	210	76.6
P7	Pancho 6a+6b Comp	100	Oxide	6.3	84	89.1
P8	RHEVC	0	Porphyry Sulphide	15.9	167	61.3
P9	RHEVC	0	Porphyry Sulphide	9.5	167	67.2
P10	RHEVC	0	Porphyry Sulphide	6.3	167	68.8
P11	Oxide	100	Oxide	15.9	81	82.1
P12	Oxide	100	Oxide	9.5	81	84.8
P13	Oxide	100	Oxide	6.3	81	84.1
P14	Copper Column Test	5	Porphyry Sulphide	6.3	81	55.1

Table 16-10: Pancho Column Test Summary 2007 (Part 2)

Column Test No.	Composite	gAu/mt ore Extracted	Tail Screen	Avg. Head	gCu/mt ore Extracted	Head Assay	NaCN Consumed, kg/mt ore
P1	1930	0.78	0.21	0.99	60	300	2.03
P2	2240	0.32	0.13	0.47	173	600	2.49
P3	1690	0.52	0.15	0.7	173	800	2.82
P4	2140	0.39	0.3	0.69	539	1400	3.71
P5	P95-1840	0.33	0.2	0.5	104	1200	2.12
P6	Shallow Comp P95-1840	0.59	0.18	0.79	600	770	3.46
P7	Pancho 6a+6b Comp	0.57	0.07	0.62	27	100	1.18
P8	RHEVC	0.38	0.24	0.68	162	800	2.19
P9	RHEVC	0.43	0.21	0.68	178	800	2.55
P10	RHEVC	0.44	0.2	0.68	144	800	2.37
P11	Oxide	0.55	0.12	0.67	22	200	2.17
P12	Oxide	0.56	0.1	0.67	38	200	2.23
P13	Oxide	0.53	0.1	0.67	16	200	2.16
P14	Copper Column Test	0.38	0.31	0.75	133	1300	1.52

16.2.5 SART Process Feasibility Study

Uneconomic concentrations of copper are present in the heap leach ore at the Maricunga mine. Some of the copper minerals are soluble in cyanide solutions and these minerals consume cyanide reagent and contaminate the leach solution with copper metal. Unless removed, this copper will decrease the purity of the dore bars and incur penalty refining costs. The copper grade in the leach solution has risen significantly from initial levels of below 50 ppm in 2005 to over 300 ppm in 2007. This copper concentration is expected to increase further as ores from the Pancho deposit are processed. The rising copper level in the Maricunga heap leach solution will need to be addressed to avoid increased lock-up of cyanide by the copper resulting in increased operating costs.

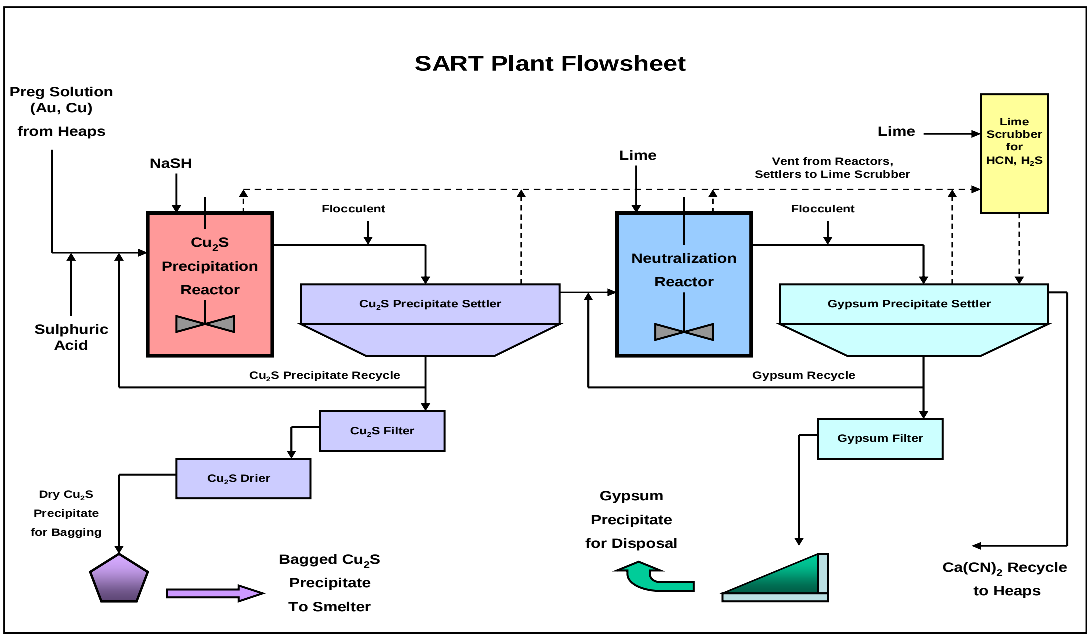
Over the last two years, Kinross has undertaken significant metallurgical testwork and engineering studies to establish a robust process for removing copper from the circuit and recycling the cyanide. In 2005, SGS Lakefield completed preliminary tests confirming that SART technology had the best potential. The SART process was developed in 1997 by SGS Lakefield Research and Teck Corporation and is currently in full-scale commercial application at Newcrest's Telfer Mine in Australia.

The **SART** name is derived from an acronym describing the unit operations in the process, as follows:

- **S**ulphidization
- **A**cidification
- **R**ecycling of precipitate/s
- **T**hickening of precipitate/s

The SART process flowsheet is shown schematically below in Figure 16-3.

Figure 16-3: SART Plant Flowsheet



SART, as originally developed, involves adding chemical sulphide ions, such as sodium hydro-sulphide (NaSH), to an acidified CN solution (pH 10 to pH 5 or below) to precipitate copper, silver and zinc (if present) as metal sulphides and convert cyanide to HCN, under weakly acidic conditions. Other CN solubilized metals like Ni and Co would also be precipitated. HCN is a very soluble gas, and remains in solution until the Cu_2S, Ag_2S or ZnS is removed by thickening and filtration. The HCN solution is then neutralized with lime or caustic soda, prior to recycling to leach.

The overall chemical reactions applied to Cu, Zn and Ag cyanide complexes are:

$$2Cu(CN)_3^{2-} + S^{2-} + 6H^+ \rightarrow Cu_2S + 6HCN$$

$$2Ag(CN)_2^- + S^{2-} + 4H^+ \rightarrow Ag_2S + 4HCN$$

$$Zn(CN)_4^{2-} + S^{2-} + 4H^+ \rightarrow ZnS + 4HCN$$

The above sulphidization reactions occur rapidly in a first stage (primary) precipitation reactor from which sulphide precipitates are fed to a thickener. Recycle of thickened slurry of sulphide precipitates back to the reactor is done until a suitable precipitate particle size develops and sufficient bed depth in the thickener is achieved. A precipitate slurry of 10 – 15% solids may then be drawn off the thickener underflow and washed, filtered and dried. The Cu_2S precipitate (with Zn, Ni and other metal sulphides, if present) is then sold to a smelter as a valuable co-product.

A second reactor is used to neutralize the aqueous HCN in the thickener overflow from pH 5 back to pH 10. Lime or caustic soda may be used to produce $CaCN_2$ or NaCN for recycle to the heap or tank leaching circuit. The neutralization reactions are:

$$2HCN + Ca(OH)_2 \rightarrow CaCN_2 + H_2O$$

$$2HCN + NaOH \rightarrow NaCN + H_2O$$

All three process units (2 reactors and a thickener) are sealed and have vents to a lime scrubber system to capture any evolved HCN or H_2S gases. By acidifying with sulphuric acid in the primary reactor, the SART process leads to formation of a gypsum type precipitate on neutralization of HCN in the neutralization reactor. This gypsum precipitate needs to be removed from the neutralized CN solution prior to recovery of gold on carbon and recycle of barren CN solution to the leach circuit. To achieve this, a gypsum thickener is included in the circuit with the option of recycling the gypsum thickener underflow to enhance precipitation rates and settling / consolidation in the thickener.

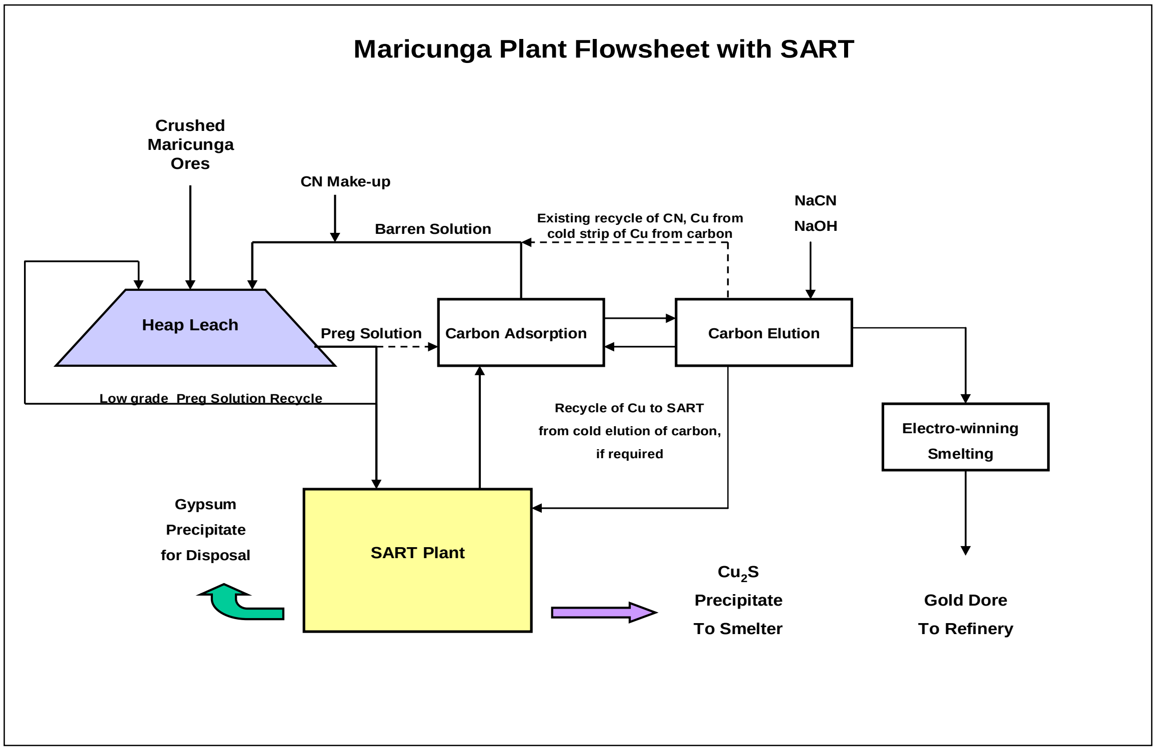
The SART process may be applied to pregnant gold solutions prior to carbon adsorption or on barren solutions after carbon adsorption. For a heap leach operation the inherent leach solution / ore solids separation suits the SART process as clear solutions are fed directly to the primary precipitation reactor. For mill plants the leach slurry may be washed via CCD with the overflow pregnant solution feeding a SART plant prior to carbon adsorption. When the leach slurry feeds a CIP plant, the CIP tails (barren) may be washed in a CCD circuit and then fed to a SART plant.

In summary, the process involves lowering the solution pH to release the cyanide associated with the copper cyanide complex, allowing it to be recycled back to the leach process as free cyanide. Sulphide reagent is added to the acidic solution to precipitate the copper as a solid high-grade copper sulphide precipitate which can be filtered, dried, bagged and shipped to a smelter to generate copper revenue. Lime has to be added back into the solution to neutralize the acid and gypsum (calcium sulphate) will be formed as by-product. This gypsum will be stored in lined ponds.

The location of the SART plant was chosen to be near the existing ADR plant at the base of the heaps in order to maximize the use of existing roads and services. A SART plant of 750 m3/hr capacity was determined as optimal to treat a bleed stream from the leach solution (2,150 m3/hr) from the heaps. In this way, the capital outlay for the plant is minimized and the SART plant can be taken off line without affecting gold production. A copper sulphide precipitate (Cu_2S at about 65% Cu) will be produced at a rate of approximately 5 to 10 tonnes per day and is planned to be sold to the Enami smelter near Copiapo. Figure 16-4 shows the Marigunga modified flowsheet incorporating the SART process.

A process to apply for environmental permitting has begun with preparation of DIA documents leading to a construction permit. This feasibility study has mapped out the DIA application process steps and contains the process technical information that will be required for the application documentation. The installation of a SART plant at Maricunga will involve some changes to the existing operation and a DIA permit will be required. Main changes to the Maricunga operation are: supply of sulphuric acid (about 15 t/day) and NaSH chemicals to the mine, dispatch of copper sulphide precipitate (about 5 t/day) from the mine and disposal of a gypsum product on site in a lined facility at the mine (about 20 t/day). A new EIS for Maricunga will not be required for the plant. The estimated timing for the permitting process following submission of the DIA is expected to be about 8 months and will not be on the project critical path.

Figure 16-4: Modified Maricunga Plant Flowsheet with SART



17.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATE

17.1 Introduction

The Mineral Resource and Reserve Estimates for the Maricunga Project have been prepared under the direct supervision of Maryse Bélanger, P. Geo., Director Technical Services, Kinross Americas.

During the course of 2007, a full review of the geologic interpretation and the spatial controls of mineralization was undertaken for both Verde and Pancho resulting in new resource models to support updated Mineral Resource and Mineral Reserve estimates.

17.2 Interpretation and Grade Domains

17.2.1 Verde

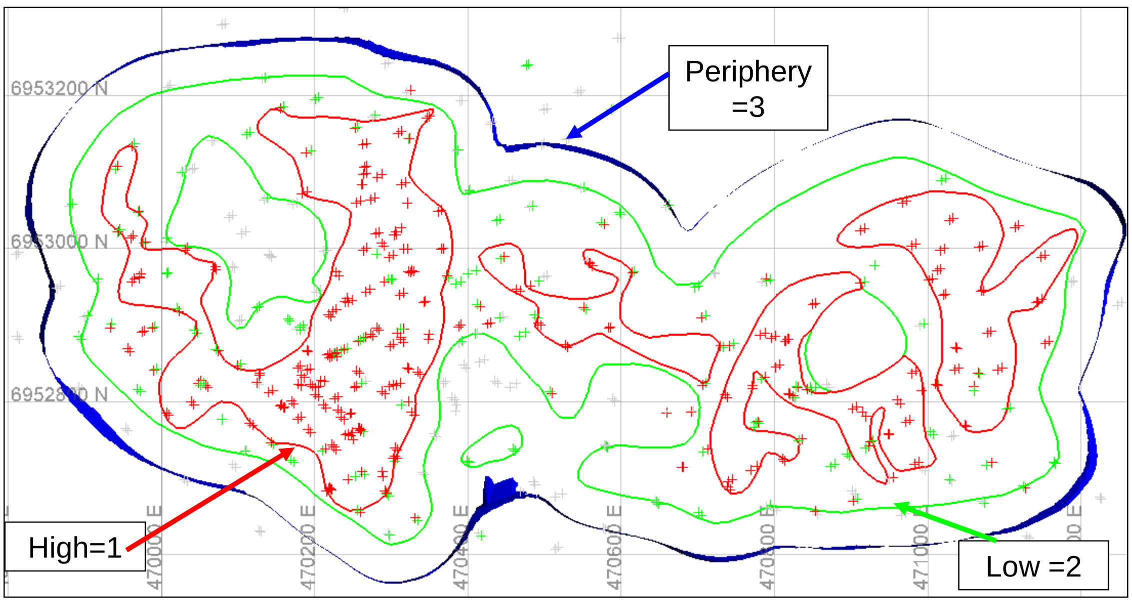
The gold grade model for Verde used to rely on a lithological interpretation completed by the site geologists to discriminate between mineralized and barren lithologic units. Mining experience at Verde has shown that the mineralization tends to cross over all lithologies.

The Verde model update prepared in 2007 incorporates a new interpretation based on a better understanding of the geology and the mineralization and its spatial distribution. Essentially, mining has shown that the gold content of the material mined is often related to its position relative to the known post-mineral intrusive bodies at Verde East and Verde West.

The model update is based on the new interpretation for Verde East and Verde West. The concept and resulting geometry are the same for both open-pit areas. The mineralization and its inner limits are defined within a zone of high-grade material centred about a post-mineral intrusive. The proximity to the post-mineral intrusive is what affects most the overall grade distribution.

The halo surrounding the higher-grade core is slightly more diffuse and incorporates mineralization of lesser tenor. The overall concept also concurs with observations in the mining areas where the material closer to the PMI displays stronger brecciation and more veinlets often resulting in grades exceeding 1 g/t Au. This compares to grades of 0.5 g/t to 0.6 g/t for the rocks located at greater distances and then again decreasing as distances from the PMI become greater. Finally, an external zone of low grade material was modelled broadly based on a cut-off of 0.15 g/t Au. A typical plan view is presented below in Figure 17 -1.

Figure 17-1: Typical Plan View of Verde Domaining (Bench 4280)



The domains model developed from cross sectional and level plan interpretation, identified distinct zone boundaries that are used as soft boundaries during grade interpolation as per the conclusions drawn for the contact profiles (see Section 17.4.2.1).

The Figure above shows the peripheral zone of approx. 0.15 g/t Au, the low-grade mineralization (in green) defined as the material with a gold grade between 0.3 g/t and 0.6 g/t. The high-grade (in red) encompasses the material greater than 0.6 g/t Au. Throughout this report they are referred to as:

- Domain 1 = gold grade greater than 0.6 g/t
- Domain 2 = gold grade between 0.3 g/t and 0.6 g/t
- Domain 3 = low grade at the periphery based on 0.15 g/t

The new interpretation incorporates outlines digitized first from sections and then transposed on plans in order to get the best possible representation of the mineralization in three-dimensions. The outlines were than linked to form solids that were validated in Vulcan. Blast hole plans were also used to guide the interpretation as they provide close-spaced information.

17.2.2 Pancho

The model for Pancho is based on the core samples collected in 2006. Diamond drilling demonstrated that the mineralization crosses over all lithologies and appears centred on a Post-Mineral Intrusive that is in rare instances mineralised. A model similar to Verde was created where three domains were defined. First, 9 sections oriented N40°W were plotted and interpreted with the outlines digitized in Vulcan. Then, 47 plans at 10 m intervals encompassing elevations from 4440 m to 3700 m elevation were interpreted. The outlines obtained in plan and section views were connected to form the solids defining the grade domains. They are also referred to as Domain 1, Domain 2 and Domain 3 corresponding to grade ranges similar to those used at Verde.

- Domain 1 = gold grade greater than 0.6 g/t
- Domain 2 = gold grade between 0.3 g/t and 0.6 g/t
- Domain 3 = low grade at the periphery based on 0.15 g/t

The northwest quadrant of the Pancho deposit shows significant copper content related to the presence of a small chalcocite blanket close to the surface and hypogene porphyry copper mineralization in the form of chalcopyrite at greater depths. A copper domain was interpreted on sections and plans as per the methodology used in defining the gold grade domains. The copper domain of interest is generally located between 4320m and 3840m elevation. The copper grades shown on plans and sections demonstrate that a 200 ppm cyanide soluble copper grade form a continuous domain in three-dimensions. It was therefore the criteria chosen for defining two domains and coding the block accordingly:

- Domain 0 = outside the copper envelope defined by 200 ppm cyanide soluble copper
- Domain 1 = inside the copper envelope

17.3 Oxidation Model and Recovery

Metallurgical recovery is a key aspect in modelling the Verde and Pancho deposits. Significant effort was directed towards establishing a relationship between metallurgical recovery and the level of oxidation visually estimated by the site geologists. Table 17-1 shows the criteria used in defining the material types encountered in the pit.

Table 17-1: Oxidation Codes

Material	Condition	Code
Oxide	greater than 50% oxidation	Block Code Redox = 3
Mixed	between 10% and 50% oxidation	Block Code Redox = 2
Sulphide	less than10% oxidation	Block Code Redox = 1

The original recovery model was developed by interpretation of cross section, longitudinal section and level plan data by the site geologists. The more recent recovery model for Verde integrates all blast hole and mine geology information collected to date. The interpreted surfaces were used as hard boundaries for assigning recoveries in the blocks during pit optimization. The surfaces are highly irregular and similar to the undulations observed in the pit walls at Verde.

The additional core samples available for Pancho allowed for an update of the surfaces marking the different levels of oxidation, namely oxide, mixed and sulphide. One noticeable difference is that the new interpretation includes less mixed material than indicated by the original interpretation completed in 2003.

17.4 Grade Model

17.4.1 Statistical and Geostatistical Analyses

Excluded Data

The following data was not used in completing the original resource block model for Verde:

- blast hole data
- trench data and;
- drill data within the previously mined portion of the Verde pits.

For Pancho some historical data was also excluded. They included trench data and original Anglo American drill holes for which location could not be confirmed. These were part of hole series SHR and SDR.

Unexplained missing data was assigned a zero value for all analyses. Explained missing data was treated as missing data and ignored during processing.

Statistics

Basics statistics were calculated for the original 2 m gold assays. The results for Verde and Pancho gold assays are presented in Table 17-2 and Table 17-3 below.

Table 17-2: Statistics for Uncapped 2 m Gold Assays - Verde

Zone	Number	Mean Au g/t	Std. Dev.	Coef. of Variation	Minimum	Maximum
All	49,789	0.67	0.69	1.03	0	15.94
1	19,153	1.07	0.80	0.75	0	15.94
2	15,274	0.56	0.47	0.84	0	10.2
3	6,908	0.25	0.35	1.39	0	10.63

Table 17-3: Statistics for Uncapped 2 m Gold Assays - Pancho

Zone	Number	Mean Au g/t	Std. Dev.	Coef. of Variation	Minimum	Maximum
All	31,164	0.54	0.66	1.22	0	21.9
1	8,560	0.97	0.86	0.89	0	21.32
2	10,796	0.51	0.37	0.72	0	11.95
3	9,856	0.28	0.53	1.90	0	21.9

It should be noted that the coefficients of variation for gold are low reflecting the disseminated widespread nature of the mineralization.

17.4.2 Grade capping

Capping of gold values affects 0.2% of the total number of assays at Verde. Grade capping was determined based on analysis of the statistics of the three populations created by dividing into the separate grade domains. The levels chosen were 4 g/t Au in the high-grade zone (domain 1) and 2.0 g/t Au in the low-grade zone (domain 2). These capping levels correspond respectively to the 99th percentile and the 98th percentile. In the material to the periphery (domain 3) a cap of 1 g/t was selected effectively reducing the grade of approx. 2.0% of the samples

Similar capping levels were selected for the Pancho domains reflecting the similarity in the grade distributions. The levels chosen were also 4 g/t Au in the high-grade zone (domain 1) and 2.0 g/t Au in the low-grade zone (domain 2). These capping levels correspond to the 99th percentile in both domains. In the material to the periphery (domain 3) a cap of 1 g/t was selected effectively reducing the grade of approx. 2.5% of the samples.

The grade capping was used in combination with restricting the influence of the higher-grade composites. (Please see Sections 17.4.4 and 17.4.5 on Gold Grade Interpolation Verde and Gold Grade Interpolation Pancho for more details).

At Pancho the copper and cyanide soluble copper grades were not capped.

17.4.3 Sample Compositing

Capped assays were composited to build the resource models for Verde and Pancho. All grade interpolation was based on 5 m length composites to account for drill hole deviations.

Statistics for the 5 m composites using gold capped assays are presented in Tables 17-4 to 17-8. Statistics for copper and cyanide soluble copper are also presented for Pancho.

Table 17-4: Statistics for 5 m Gold Composites - Verde

Domain	Number	Mean Au g/t	Std. Dev.	Coef. of Variation	Minimum	Maximum
All	20,577	0.66	0.55	0.84	0	4.00
1	7,901	1.05	0.58	0.55	0	4.00
2	15,274	0.56	0.31	0.55	0	10.20
3	6,908	0.25	0.35	1.39	0	10.63

Table 17-5: Statistics for 5 m Gold Composites – Pancho

Domain	Number	Mean Au g/t	Std. Dev.	Coef. of Variation	Minimum	Maximum
All	12,578	0.52	0.45	0.87	0	4.00
1	3,477	0.94	0.53	0.56	0	4.00
2	4,330	0.51	0.27	0.52	0	2.00
3	4,018	0.27	0.26	0.97	0	1.78

Table 17-6: Statistics for 5 m Copper Composites – Pancho

Domain	Number	Mean Cu ppm	Std. Dev.	Coef. of Variation	Minimum	Maximum
Outside	9,314	303	303	1	0	4,254
Inside	2,592	1,048	1048	1	3	7,820

Table 17-7: Statistics for 5 m Cyanide Soluble Copper Composites – Pancho

Domain	Number	Mean Cu ppm	Std. Dev.	Coef. of Variation	Minimum	Maximum
Outside	7,047	92	184	2	0	2,410
Inside	2,088	469	469	1	3	4650

17.4.4 Contact Profiles

For the updated models, contact profiles were generated to determine cases where hard or soft boundaries should be used during grade interpolation. The graphs were examined and search strategies as described in the grade interpolation sections were adopted.

17.4.5 Variography

Directional semi-variograms, (15 degree increments horizontal, 30 degree increments vertical) were modelled for both Verde and Pancho, examining only the composited drill data below the current topographic surface

The variography focused on the composites for the three zones defined at Verde and Pancho. The models generally indicated isotropic, two-structure models with well defined nugget values that were less than 35% of the modelled sill value.

The models showed good ranges in all directions, typically in excess of 100 meters.

Correlograms also showed good continuity with ranges in excess of 200 m along the vertical.

Table 17-8: Verde Variogram Table

Domain	Variogram Axis Direction	Axis Angles (Azimuth ; Dip)	Nugget	C1	C2	Range	Range 2
Domain 1,2	X (strike)	44;-0 , 127;22	0.15	0.571	0.279	13.4	107.8
	Y (dip)	317;89 , 32;13				23.5	148.3
	Z (across dip)	134;1 , 273;65				9.5	220.8
Domain 3	X (strike)	174;76 , 37;1	0.21	0.545	0.245	32	91.7
	Y (dip)	274;2 , 131;81				32	711.2
	Z (across dip)	185;-14 , 126;-9				23.6	111

Table 17-9: Pancho Variogram Table

Domain	Variogram Axis Direction	Axis Angles (Azimuth ; Dip)	Nugget	C1	C2	Range	Range 2
Domain 1,2	X (strike)	64;18 , 60;-21	0.15	0.702	0.148	50.7	1022.3
	Y (dip)	335;-3 , 329;-3				45.8	550.4
	Z (across dip)	254;72 , 52;69				33	177.8
Domain 3	X (strike)	72;17 , 37;12	0.194	0.639	0.167	87.2	406.4
	Y (dip)	346;-15 , 284;62				35.4	2170.7
	Z (across dip)	295;66 , 133;25				29.7	341

17.4.6 Model Framework

The Verde model is not rotated whereas Pancho is rotated 38° NW around the Z-axis. Both models have sufficient extent to encompass all known mineralization. The model cell size at Verde was set at 10 x 10 x 10 meters (X, Y and Z direction) while at Pancho, the model cell size was increased to 25 x 25 x 10 meters to take into account the wider spaced drilling at depth.

17.4.7 Gold Grade Interpolation Verde

The primary interpolation method for gold was ordinary kriging which used the anisotropy and ranges supported by the best variogram model for each domain. Grade interpolation was verified by inverse distance to the 2nd power and nearest neighbour interpolation. Grade interpolation in Domain 1 did not utilize octant search restrictions but used a soft boundary with Domain 2 composites. A minimum number of 2 samples and maximum of 8 samples were required to estimate the grade of any block. The number of samples from one drill hole that could influence a block grade estimate was limited to four (4).

For Domain 2, soft boundaries were used with both Domain 1 and 3. A minimum number of 2 samples and maximum of 8 samples were required to estimate the grade of any block. The number of samples from one drill hole that could influence a block grade estimate was limited to four (4). The sphere of influence of the composites greater than 0.6 g/t Au was limited to 50 m.

Grade interpolation in Domain 3 utilized octant search restrictions, requiring a minimum of two octants to estimate the grade of a block. A soft boundary approach was applied using samples from both Domain 2 and Domain3. The minimum number of samples per octant was 1 with a minimum of two octants required to estimate the block grade. A minimum of 2 samples and maximum of 10 samples were required to estimate the grade of any block. The number of samples from one drill hole that could influence a block grade estimate was limited to four (4).

Copper grades were not modelled at Verde, as the results of the metallurgical test work originally did not indicate that copper was impacting grade recovery or operating costs. Experience has shown a build up of copper in solution going to the recovery plant.

17.4.8 Gold Grade Interpolation Pancho

The gold grade interpolation at Pancho also included kriging and inverse distance to the 2nd power but did not use octant search restrictions. Domain1 gold grade were interpolated with a soft boundary with Domain 2 samples. A minimum number of 1 samples and maximum of 8 samples were required to estimate the grade of any block. The number of samples from one drill hole was limited to four (4).

For Domain 2, soft boundaries were used with both Domain 1 and 3. A minimum number of 1 samples and maximum of 8 samples were required to estimate the grade of any block. The number of samples from one drill hole that could influence a block grade estimate was limited to four (4).

A soft boundary approach was applied using samples from both Domain 2 and Domain3. The minimum number of samples was 1 and the maximum limited to 8. The number of samples from one drill hole was a maximum of four (4).

Copper Interpolation Pancho

Analysis of the drill data indicated that cyanide soluble copper grades related to the presence of a chalcocite blanket are significant. Therefore, copper grades were interpolated into the Pancho model blocks independently of the gold grade. Kriging was also the primary estimation method. Search ellipsoid anisotropy and ranges were based on the copper variograms for the two copper domains.

The copper grade interpolation included kriging and inverse distance to the 2^{nd} power but did not use octant search restrictions. Domain 0 copper grades were interpolated with a soft boundary incorporating some samples from Domain 1. A minimum number of 1 samples and maximum of 8 samples were required to estimate the grade of any block. The number of samples from one drill hole was limited to four (4).

The interpolation for Domain 1 blocks used a similar approach with the same minimum and maximum number of composites with the effect of a single hole being limited to 4 samples.

Cyanide Soluble Copper Interpolation

The cyanide soluble copper grade interpolation followed exactly the same methodology used for copper.

17.4.9 Densities

The densities that are in the current Verde model were originally based on average values by redox state that were determined at the time of the pre-feasibility study. Since that time, the redox surfaces have been updated with the blasthole data, so a check was completed to ensure that the average values were still reasonable.

The density measurements were coded with the updated redox interpretation and the statistics were compared to the values used in the previous model. Table 17-10 shows that there is no significant difference and therefore the density assignment will remain the same as in previous models. In addition, a review of the data shows that there is insufficient data at this time to produce a spatial density model. Because the variation in the density values by redox code is not significant within the redox type, a mean density value is considered reasonable.

At Pancho the density values were coded using the newly interpreted oxidation surfaces. The statistics were calculated and compared with the previous model. It is, as in the case of Verde, considered reasonable to use a mean density value in the blocks. Table 17-11 shows the comparison between the original model and the model updated in 2007.

Table 17-10: Density Measurements and Oxidation Level for Verde

	Original Values	2007 Checks				
Code	Mean	Mean	No. Data	Min.	Max.	C.V.
Oxide	2.45	2.45	55	2.12	2.63	0.06
Mixed	2.52	2.53	219	2.16	2.69	0.04
Sulphide	2.57	2.57	467	2.10	2.72	0.03

Table 17-11: Density Measurements and Oxidation Level for Pancho

	2003 Values	2007 Checks				
Code	Mean	Mean	No. Data	Min.	Max.	C.V.
Oxide	2.27	2.20	642	1.38	2.73	0.09
Mixed	2.38	2.40	196	2.10	2.63	0.06
Sulphide	2.49	2.50	2,506	1.83	2.90	0.05

17.4.10 Model Checking and Verification

Block model verification included visual inspection of the model after grade interpolation was completed to examine if the grades were consistent with the available raw data. The model was manually reviewed in plan and section using Vulcan visualization tools. Visual inspection did not identify any errors in the model.

17.5 Resource Classification

The mineral resource estimates for Verde and Pancho have bee prepared and classified in accordance with the Canadian Institute of Mining, Metallurgical and Petroleum's ("CIM")

Standards on Mineral Resources and Reserves, Definition and Guidelines Resources classification was based on the variogram ranges for each zone, number of holes found in the search ellipsoid and proximity of the composites to the center of the block being estimated.

17.5.1 Resource Classification Verde

The model update resulted in a better understanding of the mineralization and its controls at Verde. The following conditions had to be met for classifying the blocks in the model.

- Measured Resource – Information from three drill holes found in a 50 m search ellipsoid with the closest composite at a distance of 35 m or less.
- Indicated Resource – Data from two drill holes found within a search of 70 m with the closest composite at a maximum distance of 50 m.
- Inferred Resource – Remainder of the blocks including all the blocks located in Domain 3.

17.5.2 Resource Classification Pancho

The model is now including an epithermal upper portion that is drilled on approximately 35 m and a porphyry system drilled at depth with average drill spacing greater than 75 m in some areas. These considerations and the experience gained while mining at Verde also guided the new classification scheme used for Pancho. The following conditions had to be met for classifying the blocks in the model.

- Measured Resource – Information from three drill holes found in a 75 m search ellipsoid with the closest composite at a distance of 35 m or less.
- Indicated Resource – Data from two drill holes found within a search of 100 m with the closest composite at a maximum distance of 50 m.
- Inferred Resource – Remainder of the blocks.

17.6 Mineral Resource Estimate

The mineral resources estimated using a $625 gold price for Verde and Pancho as of December 31, 2007 are presented in Table 17-5. Mineral resources are reported exclusively of reserves.

Table 17-12: Measured and Indicated Mineral Resources

Deposit	Class	Ore Tonnes (x 1,000)	Grade (Au g/t)	Gold Ounces (x 1,000)
Verde	Measured	19,335	0.73	456
	Indicated	26,987	0.71	619
	M&I	**46,322**	**0.72**	**1,075**
Pancho	Measured	7,585	0.52	128
	Indicated	57,549	0.58	1,071
	M&I	**65,134**	**0.57**	**1,198**
Total		**111,456**	**0.63**	**2,274**

The Maricunga deposits also host an Inferred Mineral Resource of 134.71 million tonnes averaging 0.56 g/tonne Au at the same $525 US per ounce gold price. Pancho hosts 132.90 million tonnes averaging 0.56 g/tonne Au and Verde hosts 1.81 million tonnes averaging 0.66 g/tonne Au.

17.7 Considerations for Reserves

It is Kinross policy that resource estimates for open pit mine projects are reported within optimized or designed pit shells at assumed gold prices that represent reasonable, long-term price projections. Final pit designs have been completed for both Verde and Pancho and as a result, only Proven and Probable Reserves are reported.

17.8 Mineral Reserve Estimate

Kinross Technical Services (KTS) completed the pit optimization for defining the 2007 yearend Mineral Resource and Mineral Reserve estimates. Mineral reserves are reported within actual pit designs while the resources are reported inside optimized pit shells. The results as of December 31, 2007 are summarized and presented in Table 17-13.

The Verde reserve change reflects a higher gold price in conjunction with a new resource model. The increase at Pancho is the result of modifications to the resource model, entirely revised costs as of September 30, 2007 and the application of the SART process. More details on the costs and parameters used for defining the reserves are described in the section below (Section 17.9 Pit Optimizations).

Table 17-13: Proven and Probable Mineral Reserves

Deposit	Class	Ore Tonnes (x 1,000)	Grade (Au g/t)	Gold Ounces (x 1,000)
Verde	Proven	71,071	0.85	1,947
	Probable	32,409	0.74	772
	2P	**103,480**	**0.82**	**2,719**
Pancho	Proven	106,627	0.71	2,436
	Probable	69,395	0.58	1,290
	2P	**176,022**	**0.66**	**3,726**
Total		**279,502**	**0.72**	**6,445**

17.9 Pit Optimizations

17.9.1 Verde

The Verde block model was updated to reflect the change in topographic surface for 2007. A Mining Cost Adjustment Factor (MCAF) was also included to take into account the increased haul distance as the pit will get deeper. The base mining cost used was $1.00 per tonne with an added $0.016 for longer ore haul to crusher and waste dump. Sustaining capital for leach pad development was also included ($0.26 per tonne).

Optimization parameters used in Whittle optimization software are presented in Table 17-14.

17.9.2 Pancho

Pancho ore has variable operating costs, gold recoveries and copper grade. The geological block model contains attributes for copper grade, cyanide soluble grade and ore type. A Process Cost Adjustment Factor based on cyanide soluble copper grade but considering the application of the SART process was defined to optimize the block model. Using 2007 costs and the test work completed in the SART Feasibility Study the following formulas were derived for the different ore types.

Oxide Process +G&A costs = $3.752

Mixed and Sulphide Process +G&A costs = 3.116 + 0.0037*Cu Soluble (ppm) + 1.116 * Cu Soluble (ppm) ^ -0.112

Table 17-14: Optimization Parameters for Verde

Mining Costs		
Unit Mining Costs		$1.000
Mining Recovery		97.50%
Mining Dilution		2.50%
MCAF based on elev		$0.016
Pit Slopes		
Profile 1 (East Verde)	**Azimuth**	**Slope**
	10	44.5
Below 4350 m	65	49
	155	52.5
	275	49
Profile 2 (East Verde)	**Azimuth**	**Slope**
From 4360 to 4430 m	70	44.5
	153	49
Profile 3 (East Verde)	**Azimuth**	**Slope**
Above 4440	0	38
	0	180
Profile 4	**Azimuth**	**Slope**
West Verde	0	51
	90	51
	180	49
	270	49
Process Recoveries		
Oxide	Au	73%
Mixed	Au	70%
Sulphide	Au	53%
Processing Costs		
Process costs $US/t	$3.76	
Gold Prices	$550 and $625	
Selling costs $US/oz	Scaled to gold price	

A Mining Cost Adjustment Factor (MCAF) was also included to take into account the increased haul distance as the pit will get deeper. The base mining cost used was $1.00 per tonne with an added $0.084 for longer ore haul to crusher and waste dump. Another $0.016 was also added per 10 m bench from 4300 m elevation to consider extra haul above or below the pit access road. Sustaining capital for leach pad development was also included ($0.26 per tonne).

The Pancho ore will be processed at a rate of approx. 49,000 tpd over the life of mine. Average heap leach gold recoveries are based on test work results: 85% for oxide ore, 75%

for mixed ore and 69% for sulphide ore. The assumptions for the copper recoveries incorporate the dissolution rates for the different material types. They are 71%, 92% and 60% respectively for oxide, mixed and sulphide material. These rates are used in conjunction with a 95% plant recovery resulting in final recoveries of 67% for oxides, 87% for mixed and 57% for sulphides. Table 17-15 summarizes the optimization parameters for Pancho.

Table 17-15: Optimization Parameters for Pancho

Mining Costs		
Unit Mining Costs		$1.000
Mining Recovery		97.50%
Mining Dilution		2.50%
Pit Slopes		
Profile 1	**Azimuth**	**Slope**
Below 4300 m	70	44.5
	153	49
	210	44.5
	308	52.5
Profile 2	**Azimuth**	**Slope**
Above 4300 m	90	39
	90	39
Process Recoveries		
Oxide	Au	85%
Mixed	Au	75%
Sulphide	Au	68%
Processing Costs		
Process costs $us/t	Block Model PCAF	
Gold Prices	$550 and $625	
Selling costs $US/oz	Scale	

17.10 Pit Designs

For the 2007 reserve estimate final pit designs were prepared for the Verde pits and Pancho. Waste dump designs and overall site layout were also updated.

17.10.1 Verde

Verde pit design work was completed by CMM's mine planning engineers using Vulcan mining software. The software allows the user to input slope design criteria that varies both with azimuth and elevation. The software enables the user to specify the bench face angle and berm width for a given sector of the pit. The slope sectors specified by Golder had two azimuths, a "from" and a "to" (i.e. from 70° ENE to 120° ESE). In Vulcan, these were specified transition azimuths for each slope sector. Vulcan also inserts access ramp segments based on the design parameters and the location specified by the user.

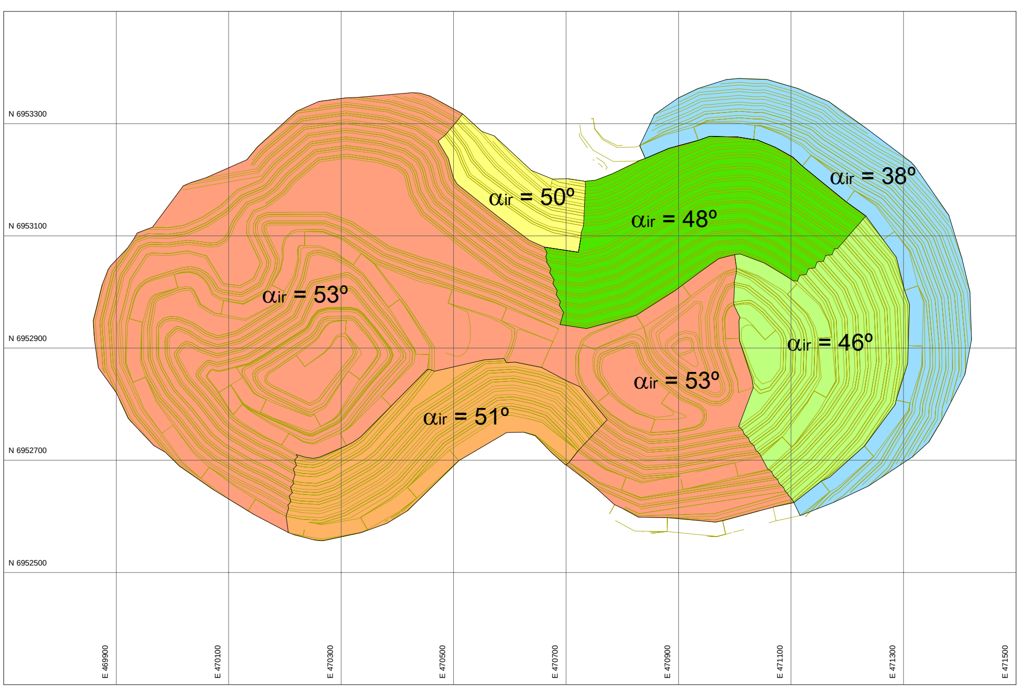
The following sections provide design details for each of the pit designs completed for this study.

East Verde Ultimate Pit Design

Due to the complex nature of the geology in the East Verde pit area a number of different slope sectors had to be used for completing the original pit design. The slope recommendations provided by Golder in 2003 formed the basis of that design.

Since then, mining experience and geotechnical reviews allowed to modify slightly the angles. The most recent work was completed by A. Karzulovic and Ass. Ltda of Santiago in April 2007. Figure 17-3 shows the pit geometry adopted in the yearend 2007 design.

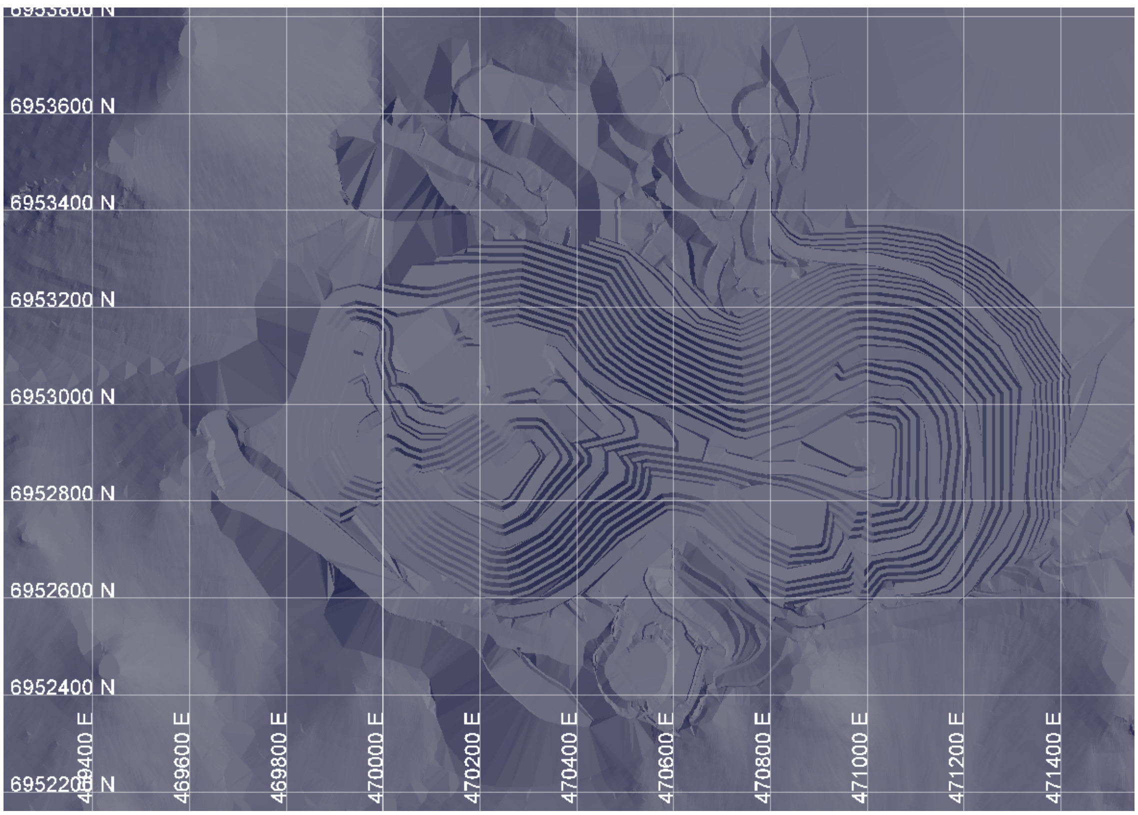
Figure 17-2: Interramp angles for Verde Pit Design



West Verde Ultimate Pit Design

Slight adjustments to the interramp angles were made following the most recent geotechnical review completed in April 2007. An angle of 53 degrees is now used in the Verde Breccia compared to 51 degrees in the original design. In the porphyry material an angle of 53 degrees is also used in the design which corresponds to an increase of 3 degrees relative to the original design.

Figure 17-3: Verde Pit Design Phase 4 with Topography

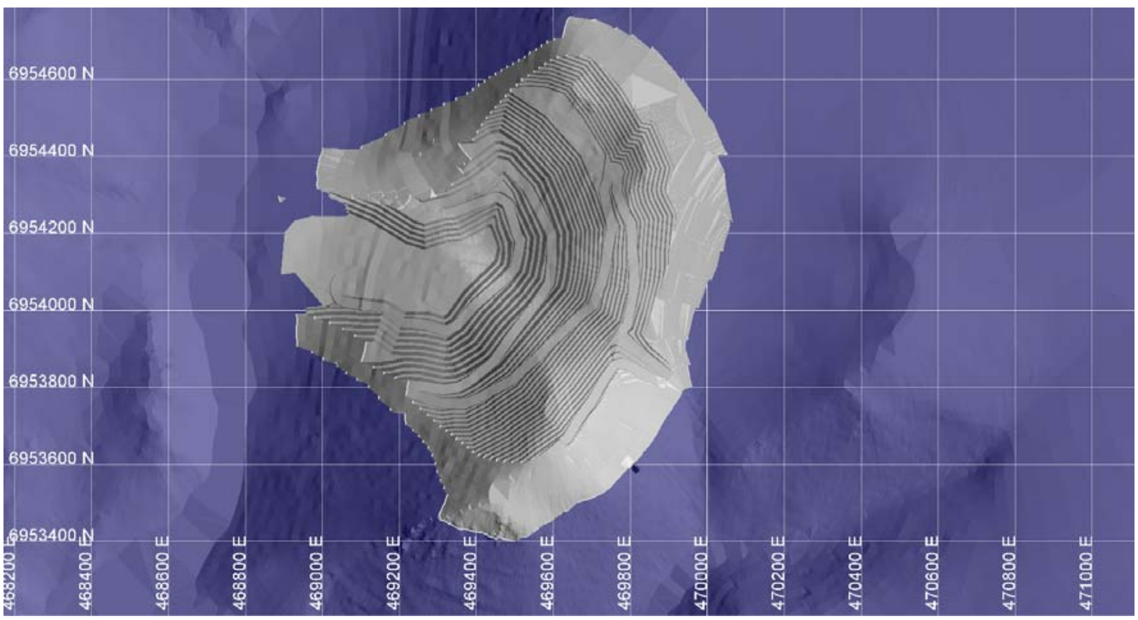


17.10.2 Pancho

The Pancho pit was designed using the same criteria as that used during the 2003 feasibility study. Golder had recommended two slope design criterion and the appropriate rosettes were established in Vulcan. The first rosette controlled the slope design from the pit bottom up to the 4300 elevation where the rock mass was generally considered more competent. Inter-ramp angles varied from 44.5 to 49 degrees in the region southeast to the southwest.

All other inter-ramp slopes were 52 degrees. Above the 4300 elevation, controlled by the second rosette, the rock mass was considered to be weathered and therefore slope angles were shallower. Golder recommended an inter-ramp angle of 38 degrees. More recent geotechnical work by A. Karzulovic and Associates Ltda. of Santiago (April 2007) indicate that the angle in the volcanic material above 4300 m elevation could be increased to 43 degrees. Figure 17-3 shows the pit geometry adopted.

Figure 17-4: Pancho Pit Design with Topography



The author, Maryse Bélanger, has no knowledge of any environmental, permitting, legal, ownership, taxation, political or other relevant issue that would materially affect the current Mineral Resource and Mineral Reserves estimates

18.0 OTHER RELEVANT DATA AND INFORMATION

There is no other data or information relevant to the project that is not covered in other sections of this report.

19.0 INTERPRETATION AND CONCLUSIONS

The Mineral Resource and Reserve Estimates for the Maricunga Project have been prepared under the direct supervision of Maryse Bélanger, P. Geo., Director Technical Services, Kinross Americas.

Ms. Bélanger is satisfied the data used in the estimation of Mineral Resources and Reserves is free of gross errors and omissions and is of suitable quality and sufficient quantity to use in estimating resources and reserves for the Verde and Pancho deposits.

20.0 RECOMMENDATIONS

The Maricunga mine is a producing gold mine for Kinross Gold Corporation. The recommendation for successive work phases is not required.

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22.0 DATE AND SIGNATURE PAGE

The undersigned prepared this Technical report, titled NI-43-101 Technical Report for the Maricunga Gold Mine, dated 31 December 2007. The format and content of the report are intended to conform to Form 43-101F1 of the National Instrument (NI 43-101) of the Canadian Securities Administrators.

“Signed and Sealed”

Maryse Bélanger, P. Geo. 31 March 2008

23.0 ADDITIONAL REQUIREMENTS FOR OPERATING PROPERTIES

23.1 Mining Operations

The Maricunga Mine is an operating mine with significant infrastructure in place. The reserve estimates presented in this document are sufficient to support the operation for 16 years based on daily production of 45,000 tpd.

Maricunga production re-opened in October 2005 and achieved its targeted production rate of 14 million tonnes per year (40,000 tonnes per day) in late 2005. The mine operates two 12-hour shifts per day for 352 days annually allowing for inclement weather interruptions. Final pit design for Verde and Pancho assumed 10 meter bench heights, bench face angles of 65° to 70°, berm widths of 8 to 11 meters, berm interval of 20 meters, inter-ramp angles of 38° to 52.5° and haul road gradient at 10% with a 25 meter road width.

The Maricunga gold recovery process consists of a single line primary crushing, fines crushing (secondary and tertiary), heap leach and adsorption and regeneration (“ADR”) plant operation. The process treats 45,000 tonnes per day of dry Maricunga ore using primary crushing followed by a secondary and tertiary crushing plant. The crushing plant product is approximately 80% passing 10.5 millimetres. A pad type heap leach and an ADR plant are used for gold recovery.

A comprehensive program of metallurgical testing incorporating bottle roll tests and column leach tests of samples obtained from drilling was established to determine gold recovery and reagent consumption data for the remaining Verde resources and the Pancho resource. Based on the recovery estimates by ore type, process recovery over the mine life averaged 69% of contained gold in the plant feed. Life of mine annual gold production is expected to range from 230,000 to 305,000 ounces on a 100% basis.

The increase in reserve as of December 31, 2007 will require an expansion of the heap leach facility. This report considered the findings of a pre-feasibility study level completed by Dave Prins of PrinZ Mining Services based in Santiago, Chile. The increased reserves will result in the need to permit additional leach pad capacity but this is not considered to be a risk, as the existing permitted space is sufficient for the majority of the remaining reserves.

23.1.1 Mobile Equipment

The mine operates as a traditional open pit operation. A partial list of the major mine equipment necessary to achieve the planned production rate includes:

- 3 CAT 994D Front End Loaders
- 12 CAT 785C Haul Trucks
- 1 CAT 773E Haul Truck
- 2 Ingersol Rand 250 mm Rotary Drills
- 1 ECM 720 Rotary Drill
- 2 CAT D9R Bulldozers
- 2 CAT D8R Bulldozers
- 1 CAT D11R Bulldozer
- 2 CAT 14H and 16H Motor Graders
- 1 CAT 834 Wheel Dozer
- 1 50t Water Truck
- 1 CAT 330CL Backhoe

Ancillary equipment include fuel and lube trucks, flatbed trucks, portable light towers, and service and welding trucks.

23.1.2 Leach Pad and Plant Infrastructure

The plant facilities were upgraded with all the work completed by October 2005. Key aspects of the upgrade included establishing a 110 km power line to replace the generated power currently at site. Additional capital expenditures went towards replacing certain components to improve availability and throughput of the crushing and conveying system. All equipment and systems are in place and allow for meeting a daily throughput target of 45,000 tpd.

In 2007 the SART process was tested for the Pancho ore as uneconomic concentrations of copper are present in ore. Some of the copper minerals are soluble in cyanide solutions and these minerals consume cyanide reagent and contaminate the leach solution with copper metal. A feasibility study and pilot testing program were completed by Idesol of Santiago. The study considers plant design, performance metrics, operating and capital costs and implementation strategy. The reserves defined at Pancho consider the application of SART.

23.2 Recoverability

Metallurgical recovery information is discussed in detail in Section 16 of this report.

23.3 Markets

Kinross will continue with current marketing arrangements at Maricunga and does not envision any concerns related to marketing doré.

23.4 Contracts

Smelting, refining, handling and sales charges are within industry standards.

23.5 Environmental Considerations

A reclamation plan for the current mine disturbance was approved in 2002, based on the commitments made in the original environmental impact assessment for the site (1994). The plan addressed physical activities, such as earthworks, but did not address chemical closure of the heap. A closure plan for chemical stabilization of the heap has been prepared and has been submitted to the regulatory authorities in the form of a Declaration of Impact to the Environment (“DIA”). The regulatory agencies are currently considering the Company’s proposed chemical stabilization approach and further discussions with agencies are expected prior to a decision regarding the chemical stabilization plan. The agencies consider submittal of the chemical stabilization DIA as meeting the commitments in the original environmental impact

Future cash outflows for site restoration costs at Maricunga under CICA Handbook Section 3110, as at December 31, 2007, are estimated at approximately US $4.4 million. There is no requirement to post financial assurance to secure site restoration costs in Chile at present.

The increased reserves will result in the need to permit additional leach pad capacity but this is not considered to be a risk, as the existing permitted space is sufficient for the majority of the remaining reserves.

23.6 Capital Costs

Capital requirements were defined based on a mine life of 16 years utilizing the SART process in future years. The capital costs critical items include construction costs for the SART process facility, sustaining capital for continued expansion of the existing leach pad and partial replacement of the mining fleet in year 2015. The breakdown of costs was established by activity area. They are reported in three main categories: mining, processing and others. The costs associated with upgrading some existing equipment in 2008 report to the Other category. The schedule of capital costs is presented in Table 23-1.

Table 23-1: Capital Costs – Maricunga

	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017
Mining										
Stripping										
Project construction										
Equipment	1,900	7,200	13,290	10,650	2,050		13,530	28,900	1,500	4,300
Sustaining	10,233	3,227	10,965	841	755	473	325	583	341	855
Other										
Subtotal	12,133	10,427	24,255	11,491	2,805	473	13,855	29,483	1,841	5,155
Processing										
Project Construction	8,000	2,000	29,387	24,926	5,000	3,000	3,500	3,500	3,500	3,500
Equipment										
Sustaining		1,000				500				500
SART	539	19,704								
Subtotal	8,539	22,704	29,387	24,926	5,000	3,500	3,500	3,500	3,500	4,000
Other										
Project Construction										
Equipment										
Sustaining	14,879	-13,916								
Other										
Subtotal	14,879	-13,916								
Capital Costs	**35,551**	**19,215**	**53,642**	**36,417**	**7,805**	**3,973**	**17,355**	**32,983**	**5,341**	**9,155**

	2018	2019	2020	2021	2022	2023	2024
Mining							
Stripping							
Project construction							
Equipment	980						
Sustaining	325	325	225				
Other							
Subtotal	1,305	325	225				
Processing							
Project Construction	3,500	3,500	3,500	3,500	3,500	3,500	
Equipment							
Sustaining							
SART							
Subtotal	3,500	3,500	3,500	3,500	3,500	3,500	
Other							
Project Construction							
Equipment							
Sustaining							
Other							
Subtotal							
Capital Costs	**4,805**	**3,825**	**3,725**	**3,500**	**3,500**	**3,500**	**0**

Note: All amounts expressed in Table 23-1 are in thousands of dollars.

23.7 Operating Costs

Operating cost estimates were originally prepared based on first-principles and verified to historical mine operating costs available for the project. Operating costs used in this report are consistent with present operating experience. They are updated on a regular basis with a full cost review completed in November 2007.

A detailed 2008 and LOM budget has been developed for the mine based on an actual operating cost history and projected future performance. Current operating costs are similar to most mines of this size and equipment age and are presented in Table 23-2.

Table 23-2: 2007 Actual Operating Costs – Maricunga

Item	2007 Actual Cost
Ktonnes Mined	25,800
Ktonnes to Leach	13,691
Mining cost ($/tonne mined)	$1.09
Mining cost ($/tonne processed)	$2.05
Leaching cost ($/tonne	$3.37
G&A ($/tonne processed)	$1.26

23.8 Economic Analyses

The discounted cash flow analysis indicates that the mine generates a positive cash flow. The details of the economic and sensitivity analyses are considered by Kinross to be confidential information. The economic models are considered as complete, reasonable, and meeting generally accepted industry standards. This information is available and individuals seeking to review these models must request and sign a confidentiality agreement with Kinross Gold Corporation.

23.9 Payback

This is not applicable to the producing operation.

23.10 Mine Life

A complete Life-of-Mine plan was prepared based on the estimated mineral reserves. The mineral reserves are sufficient to support gold production until 2024 (see Table 23-3).

Table 23-3: 2007 Life of Mine Plan

Year	2008	2009	2010	2011	2012	2013	2014	2015
Ore (tonnes)	15,400,000	16,051,200	17,234,805	17,259,980	17,285,155	17,285,155	17,285,155	17,285,155
Strip ratio	0.68	0.97	0.97	0.97	0.93	0.93	0.93	0.93
Waste (tonnes)	10,471,820	15,610,696	16,717,761	16,742,181	16,075,194	16,075,194	16,075,194	16,075,194
Grade (g/t)	0.77	0.82	0.80	0.78	0.80	0.78	0.69	0.64
Recovery (%)	58.2%	56.2%	58.2%	63.9%	83.1%	77.8%	75.7%	74.0%
Recov. ounces	222,753	239,048	257,383	278,076	371,321	335,893	290,088	264,647
Total tonnes	25,871,820	31,661,896	33,952,566	34,002,161	33,360,349	33,360,349	33,360,349	33,360,349

Year	2016	2017	2018	2019	2020	2021	2022	2023	2024
Ore (tonnes)	17,285,155	17,285,155	17,285,155	17,285,155	17,285,155	17,285,155	17,285,155	17,285,155	6,603,551
Strip ratio	0.93	0.93	0.93	0.93	0.93	0.38	0.16	0.23	0.21
Waste (tonnes)	16,075,194	16,075,194	16,075,194	16,075,194	16,075,194	6,646,585	2,767,186	4,023,861	1,372,398
Grade (g/t)	0.58	0.78	0.73	0.65	0.63	0.73	0.63	0.66	0.62
Recovery (%)	71.4%	65.6%	57.9%	58.7%	75.3%	69.8%	72.4%	68.2%	68.0%
Recov. ounces	231,371	285,514	234,528	213,115	262,729	283,548	253,468	251,302	89,768
Total tonnes	33,360,349	33,360,349	33,360,349	33,360,349	33,360,349	23,931,740	20,052,341	21,309,016	7,975,949