GEO INVESTMENT

AMENDED UPDATED PRELIMINARY ECONOMIC ASSESMENT FOR THE BERTA PROJECT INCA DE ORO, III REGION, CHILE

NI 43101

TECHNICAL REPORT

PREPARED BY:

Sergio Alvarado

(CIM; MEMBER OF CHILEAN MINING COMMISSION)

Effective date 24 th of September, 2015

CORO MINING CORP.

DATE AND SIGNATURE PAGE

Project Name: Berta Project

Title of Report: Preliminary Economic Assessment for the Berta Project Inca de Oro, III Region, Chile.

Location: III Region, Chile.

Effective Date of Report: September 24th, 2015

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--- Original Signed---

Signed: "*Sergio Alvarado*"

Sergio Alvarado (CIM, Member of Chilean Mining Commission)

September 24th, 2015

CERTIFICATE OF AUTHOR

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I, Sergio B. Alvarado, Geologist, do hereby certify that:

1. I am a Consultant Geologist, General Manager and partner, who works at Geoinvestment SpA as Project Manager, whose main office is located at Los Militares 5620, Office 511, Piso 5, Las Condes, Santiago de Chile.

2. My professional title is Geologist with the degree of Geology obtained at Universidad Católica del Norte, Chile in 1991 with post graduate studies in resource assessment at the Universidad de Chile in 1997.

3. I am a Competent Person from the Mining Chilean Commission with Registration No. 004.

4. I am registered at the Institute of Chilean Mining Engineers (IMCH) License No. 1939 and the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) License No. 144015.

5. I have practiced my profession continuously since 1985.

6. I have read the definition of "qualified person" contained in the Competency Assessment Commission Resources and Mineral Reserves (Law No. 20,235) in Chile, recognized by the Canadian Securities Administrators (CSA) regulating the standard NI-43-101, which certifies that: because of my education, affiliation to a professional association (as defined in NI 43-101) and relevant past work experience, I am qualified to be a "qualified person" for purposes of NI 43-101. I have reviewed the calculations of many resources and reserves estimations for both, metallic and nonmetallic projects. I have worked in project management of copper and gold open pit mine dealing with all aspect of mining planning and plant construction and operation. I am specialized in geology, geotechnical and engineering having a wide knowledge of the mining industry in Chile and the rest of Latin America. My last reports have been performed for Samsung in Venus Marta and Carola Projects; Dayton in Andacollo project; EPG Partners in Jano and Florida Projects; Samsung and Velasco Pampa Camarones Project; and Grupo Buena Ventura Chacua project in Perú.

7. I am responsible for sections 1-15, 18-20 and 23-26 and for the overall preparation of the "Amended Updated Preliminary Economic Assessment for the Berta Project, Inca de Oro, III Region of Chile" and visited the site project in October 22nd and 23rd, 2012.

8. I have prepared the January 2013, September 2013 and October 2014 reports for the property subject of this Technical Report.

9. To the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

10. The author, Sergio Alvarado, is independent of Coro Mining Corp., of Propipe and of the Berta Project.

11. I have read the National Instruments Rating Mines Commission on Resources and Mineral Reserves, the NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with such forms and instruments.

12. I consent to the filing of this Technical Report to a stock exchange or other regulatory authority and any publication of them for regulatory purposes, including electronic publication in the public company files on their websites, accessible to the public of this Technical Report.

Dated this day of September, 24th of 2015.

"Signed and sealed"

SERGIO ALVARADO CASAS
Geoinvestment
Gerente General y Socio

Sergio B. Alvarado
QUALIFIED PERSON (MINING COMMISSION) Nº 0004

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I, D. Enrique Quiroga, Mining Engineer with graduate degree in Metallurgical undersigned certify that:

1. I am a Mining Engineer, consultant, general manager and senior partner, who works for Q & Q Ltd. as Project Manager, located in Quillapi 7292, Huechuraba, Santiago, Chile.

2. My title of Engineer of Mines was obtained at the University of Chile in 1984 and I did postgraduate studies in Hydrometallurgy and Electrochemistry in the same University in1995.

3. I am a qualified person in Mining as designated by the Mining Commission of Chile, with registration No. 0039. The rating committee Utilization of Resources and Reserves Mining person is a permanent member of the Committee for Mineral Reserves International Information Standards (CRIRSCO)

4. I am enrolled in the Institute of Mining Engineers of Chile (IMCH) License No. 570 and the Colegio de Ingenieros de Chile, Act No. 17971-0

5. I have practiced my profession continuously since 1979.

6. I have read the definition of "qualified person" contained within the jurisdiction of the Evaluation Commission of Mineral Reserves and Resources (Act No. 20235) in Chile, recognized by the Canadian Securities Administrators (CSA), regulating NI Standard -43 to 101, which certifies that due to my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am qualified to be a " qualified person "for the purposes of NI 43- 101 specialty: Mining. I participated in the profile Engineering, basic and detailed some projects still running, and managed mining operations in Chile and Peru, both in the field of large-scale mining and medium mining.

7. I am responsible for sections 13, 16-17 and 21-22 ofthe technical report entitled " Amended Updated Preliminary Economic Assessment for the Berta Project, Inca de Oro, III Region of Chile", with the effective date of 24 th September 2015 and signed on 24 th September 2015 on the Draft Berta.

8. I have had no previous relationship or association with the properties that are the subject of this Technical Report.

9. The author, Enrique Quiroga, is independent of Coro Mining Corp., of Propipe and of the Berta Project

10. I have read the National Instruments Rating Mines Commission on Resources and Mineral Reserves, the NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with such forms and instruments.

11. As of the date hereof, to the best of my knowledge, information and belief, the sections of the foregoing technical report contains all scientific and technical information required to be disclosed to make the technical report not misleading.

12. I consent to the filing of this technical report, a stock exchange or other regulatory authority and any publication by them for regulatory purposes, including electronic publishing in the archives of public companies on their web sites accessible to the public This Technical Report.

Dated this day of 24 September 2015.

"Signed and sealed"

Enrique D. Quiroga
QUALIFIED PERSON (MINING COMMISSION) N° 0039

GEO INVESTMENT

Who We Are

Geoinvestement SpA is a service provider company for mining, giving innovative solutions for mining project development.
We are specialized in geology, geotechnical and engineering having a wide knowledge of the mining industry in Chile and the rest of Latin America.
In the same manner, and as product of our wide experience and contact network, we provide independent consulting on identification and negotiation of high potential mining prospects

Last Works Performed between 2011 and 2013

Contract Name	Client
Estudio trazado Agua de Mar Mejillones - Sierra Gorda, Antofagasta	FNX Quadra
Estudio Geológico y Geotécnico Infraestructura Proyecto Arqueros, Copiapó	Laguna Resources, Kingsgate
Geología y Geotecnia Minas Lautaro Sur y Norte, Copiapò	Minera Comahue
Hidrología e Hidrogeología Cuenca Amolanas	Minera Comahue
NI-43101 Proyecto San Jorge, Mendoza, Argentina	Coro Mining
Estudio trazado Agua de Mar Mejillones - Lomas Bayas, Antofagasta	Propipe S.A.
Estudio Geotecnico Mina Los Cristales, Vallenar	Compañìa Minera CAP
Evaluación Técnica y Económica Proyecto Venus Marta, Copiapó	Samsung
Evaluación Técnica y Económica Proyecto Carola, Copiapó	Samsung
Estudio Hidrológico e Hidrogeológico Proyecto Arqueros, Copiapó	laguna
Estudio Geotecnico y Diseño de Pit Proyecto Teteritas, Copiapó	Laguna Resources, Kingsgate
Estudio Geotecnico y Diseño de Pit Proyecto Chimberos, Copiapó	Laguna Resources, Kingsgate
Geología y Mineralización Proyecto Delirio, Inca de Oro, Copiapó	Emining Ingeniería
Estudio Geotecnico y Diseño de Pit Proyecto Pampita - Rulita, Calama	Compañía Cerro Dominador
Estudio Geotécnico y Diseño de Botaderos, Pampita - Rulita, Calama	Compañía Cerro Dominador
Estudio Geotécnico y Diseño de Botaderos, Pilas de Lixiviación y Tranque de Aguas Pampa Camarones, Arica	Compañía Minera Pampa Camarones
Estudio Geotecnico y Diseño de Pit Proyecto Salamanqueja, Pampa Camarones, Arica	Compañía Minera Pampa Camarones
Estudio Geológico y evaluación del Recurso Potencial Proyecto Jano, Fondo de Inversión	EPG Partners
Estudio Geológico y evaluación del Recurso Potencial Proyecto Florida, Fondo de Inversión	EPG Partners
NI-43101 Evalaución de Recursos y Reservas Proyecto Berta, Inca de Oro, Copiapó	Coro Mining
NI-43101 Actualización de Recursos y Reservas Mina Andacollo, La Serena	Compañía minera Dayton
Estudio Hidrogeológico, Mina Chacua, Perú	Grupo Buena Ventura

DISCLOSURE AND RISKS

The Berta Project is owned by SCM Berta ("SCMB") a Chilean company whose shareholders are Minera Coro Chile Limitada ("MCC") and ProPipe SA ("ProPipe").

In mid-2014, SCMB was presented with the opportunity of acquiring the existingNora SXEW plant from receivership, which would give SCMB the ability to achieve production earlier than otherwise possible, and with a significantly reduced execution risk and cost, by not having to build a standalone SXEW Plant at Berta.

The purchase of the Nora SXEW Plant and the ability to process material from Berta is dependent upon financing. On June 16th 2015, Coro announced that a combined ~$9.0 million convertible debenture and equity financing package (the "Financing") had been provided by Greenstone Resources L.P. ("Greenstone"), subject to certain conditions precedents including but not limited to the approval of Coro Mining Corp.'s ("Coro") shareholders (which was received on July 16th 2015).

SCMB has elected to develop the project in two phases; Phase 1 involves the acquisition and remediation of the Nora SXEW plant and the trucking of high grade ("HG") material from the Berta Sur deposit for 11 months. Phase 2 involves the installation of the Berta crusher, pads and site facilities; the expansion of the Nora plant to 5ktpy of cathode; and the installation of Pregnant Leach Solution ("PLS") and water pipelines between Nora and Berta.

The Financing will provide $7.15 million for the Phase 1 capital expenditures requirements at Berta.**On August 10th 2015 Coro announced the closing of the ofthe first tranche of $5.1 million of the convertible debenture. On September 1st 2015, Coro announced the acquisition of the Nora Plant.**

An additional capital requirement of $12.6 million will be required for the development of Phase 2 at Berta. Using a base case of $2.80/lb copper, it is estimated that Berta will generate approximately $5.9 million in cash flow before and during the Phase 2 development which would be available to fund the Phase 2 capital requirements. SCMB has had advanced discussions with vendors and construction companies and has identified approximately $8.0 million in vendor and construction finance that may be available for Phase 2. SCMB will look to finalize these arrangements or seek alternative financing arrangements for Phase 2 upon completion of the Phase 1 financing.

More detailed engineering studies have not been completed and so the normal progression from Preliminary Economic Assessment ("PEA") to Preliminary Feasibility Study to Feasibility Study has not been followed in respect of making a production decision.

Therefore, investors are cautioned that no mineral reserves have been declared and the level of confidence in the resources, metallurgy, engineering and cost estimation is not at a level normally associated with a project reaching a production decision. This may result in the production rates, copper recoveries and operating costs stated in this PEA not being realized.

Geoinvestment's assessments are preliminary in nature, mineral resources are not mineral reserves and do not have demonstrated economic viability, and there is no assurance the preliminary assessments will be realized. The outcome of this PEA may be materially affected by the closing of the financing, copper pricing, environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. Inferred mineral resources are considered too speculative geologically to have economic considerations applied to them that enable them to be categorized as mineral reserves.

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LIST OF ABBREVIATIONS

Abbreviation	**Unit or Term**
%	Percent
°	Degrees of longitude, latitude, compass bearing or gradient
<	Less than
>	Greater than
AA	Atomic absorption
^{o} C	Degrees Celsius
3D	Three-dimensional
CIL	Carbon-in-leach
cm	Centimeter(s)
cm^3	Cubic centimeter(s)
CuT	Total copper
CuS	Soluble copper
DDH	Diamond drill hole
g	Grams
g/cm^3	Grams per cubic centimeter
g/t	Grams per ton
GPS	Global positioning system
h	Hour(s)
ha	Hectare(s)
in	Inch(es)
IP	Induced polarization
kg	Kilogram(s)
Koz	Thousand ounces
kg/t	Kilograms per ton
km	Kilometer(s)
km^2	Square kilometer(s)
l	Liter(s)
M	Million(s)
Ma	Million year(s)
m	Meter(s)
m/s	Meters per second
m^3	Cubic meter(s)
Mo	Molybdenum
N	North
NSR	Net smelter return
ppb	Parts per billion
ppm	Parts per million
RC	Reverse circulation
s	Second
S	South
SG	Specific gravity
t	Ton(s)
US	United States

Abbreviation	Unit or Term
US$ or $	US dollar(s)
UTM	Universal Transverse Mercator
$/lb	US dollars per pound
W	West

1.0 SUMMARY AND CONCLUSIONS

1.1 INTRODUCTION

Coro, through its subsidiary SCMB retained the services of Geoinvestment SpA ("Geoinvestment") to prepare a mineral resource estimate, PEA and Technical Report, covering its Berta copper property, located in the III Region, Chile. Geoinvestment is aware that this report is intended for disclosure tothe Toronto Stock Exchange, where Coro is listed, giving support to the News Release published on June 16th 2015. The mineral code followed in this report is the Canada Institute of Mining ("CIM") code, 2014 Edition, and this report follows the recommendations of National Instrument 43-101.

Sergio Alvarado, BSc (Hons.) Geology, member of CIM, The Chilean Mining Commission ("CMC") and The Chilean Mining Engineers Institute ("IIMCh") was responsible for the overall preparation of the Technical Report as defined in National Instrument 43-101, Standards of Disclosure for Mineral Projects and in compliance with Form 43-102F1.

In preparing this report, Geoinvestment relied on reports, studies, maps, databases and miscellaneous technical papers listed in the References section of this report. Additional information and data for Geoinvestment's review and studies were obtained from SCMB on site or at Coro´s Santiago office.

1.2 OWNERSHIP

Coro owns all the shares in 0904213 B.C. Ltd (a company incorporated in British Columbia, Canada) which owns all the shares in Sky Dust Holdings Limited ("Sky Dust") (a company incorporated under the BVI Companies Act, 2004). Sky Dust owns all the shares in Machair Investments Ltd ("Machair") (a company incorporated under the BVI Companies Act, 2004).

Machair beneficially owns 100% of Minera Coro Chile Limitada ("MCC"), a limited liability Chilean Company established under the laws of Chile on April 18, 2011. MCC beneficially owns 82% of SCMB (a company incorporated under the laws of Chile on June 4, 2013).

On June 13, 2011 Coro announced, its subsidiary MCC had reached an agreement with a local owner for 506 ha of pending measured and measurable concessions, all registered and in good standing, that protect the main part of the project. The terms of the option were renegotiated in May 2013 reducing the total payments from $6.0 million to $4.0 million for the introduction of a 1.5% NSR on any copper oxides; and further again in 2014 by deferring the $2.5 million and providing a financing option for the final payment due in August 2015. The 2014 amendment also allowed for the deposit to be mined after the payment on August 2014. The financing option allows for the August 2015 payment to be divided into eight quarterly payments of $281,250 plus interest accruing at LIBOR.

Table 1.1: Concessions Option Terms

	Current Terms	**Status**
On June 10th, 2011	$ 200,000	Paid
On June 10th, 2012:	$ 800,000	Paid
On June 10th, 2013:	$ 500,000	Paid
On August 14th,2014:	$250,000	Paid
On August 14th,2015:	$ 2.25 million	
TOTAL	**$ 4.0 million**	
	An NSR of 1.5% on all copper oxides and sulfide production and its by-products	

Additionally to adequately protect the area of interest, MCC has registered approximately 2,400 ha exploration concessions, named Berta 1 to Berta 8.

All concessions are valid according to the Chilean Mining Code. Apart from the option payments and the NSR derived from its execution, no other payment obligations exist on the properties that protect the project.

SCMB have already negotiated 15 lps water rights from the CODELCO Pampa Austral tailings dam which can be used any time from June 2015.

On May 7th 2013, MCC signed a Letter of Intent ("LOI") with ProPipe SA ("ProPipe") whereby ProPipe may earn up to 50% of the shares in a new company called SCMB formed on June 4th 2013, by completing a series of payments, work commitments and project financing, thereby earning percentages of that company as follows;

- Making the $500,000 option payment due on 10th June 2013: 10% earned
- Completing and filing an Environmental Impact Declaration ("EID") by 30th July 2013: 3% earned
- Completing a NI43-101 compliant PEA by September 30th 2013: 5% earned
- Obtaining and structuring project financing on non-recourse basis, at market conditions, with funds available within 6 months of completion of the PEA, for a minimum of 70% of the project cost, including a cost overrun facility, as determined in the PEA. In the event that this financing is for 100% of the project cost, ProPipe will earn 32% of SCMB, for a total shareholding of 50%. If the financing is between 70% and 100% of the required funding, ProPipe will earn a pro-rata shareholding in SCMB. At the minimum 70% level, they would earn 22.4% of SCMB, for a total shareholding of 40.4%. In the event that less than 100% funding is received, ProPipe have the right to earn the corresponding shareholding for the percentage difference in funding, or to assign their right to do so to a third party on the same terms. In the event that they do neither, they must complete such additional work and reports as required by Coro by March 31st 2014, for MCC to obtain the financing required and thus earn the corresponding shareholding.
- In the event that ProPipe do not arrange a minimum of 70% project financing, they must complete a NI43-101 compliant Definitive Feasibility Study ("DFS") for the project by 31st March 2014, and by so doing, will earn an additional 7% shareholding, for a total shareholding of 25% in SCMB. Coro and ProPipe will then seek project financing on a pro-rata basis

- ProPipe will be Operator during the development and construction of the project, thereafter the Operatorship will alternate every 2 years.
- The dates shown above for completion of the various project earn in stages were subsequently extended by mutual agreement of the parties

ProPipe paid the $500,000 option payment due on 10th June 2013 and earned a 10% interest in SCMB. It also earned a further 3% for completion and submission of the Environmental Impact Declaration on November 7th 2013, and approved in October 2014.

In conjunction with the ProPipe agreement, in June 2013, the underlying option agreement with the local owner was transferred from MCC to SCMB, together with the Berta 1-14 exploration claims.

The shareholder’s agreement between ProPipe and MCC has been executed, and ProPipe’s interest in SCMB iscurrently 18% with the completion of the October 2014 PEA.

Propipe and MCC have agreed that Propipe shall have earned a 35% interest in SCMB, once the Phase 1 financing has closed.

1.3 HISTORY & EXPLORATION

There is abundant evidence of superficial copper mineralization in the area; however the oldest mining was directed to the exploitation of superficial narrow Au veins, with copper mining limited to minor exploitation. There is no history of these mining properties prior to Mr Oscar Rojas Garin’s acquisition during the late 1980's. The exploitation at a small-scale mining level was extended to mechanized extraction during the 1980's and 90's, through the development of small open pits and declines. According to existing information (Guiñez and Zamora, 1998) in 1995 a local mining company, developed the Gemela and Carmen oxide bodies producing more than 100,000 t of mineral material at an average grade of 1.68% CuT. If the exploitation of three other small bodies (Salvadora; Berta, San Carlos) is included, the total mineral material extracted at Berta approximates 200,000 t at 1.5% CuT.

Outokumpu (Outokumpu Explorations, 1994) carried out geological, geochemical and geophysical exploration between March and September 1994, completing 48 short air track (DTH) holes and 7 reverse circulation (RC) holes for a total of 2,216 m. These results did not meet Outokumpu minimum target size and therefore the area was returned to the owner.

In 1997 the area was optioned by Empresa Minera Mantos Blancos S. A. ("Mantos Blancos") a subsidiary of Anglo American plc ("Anglo American") (Guinez and Zamora, 1998). During September - December 1997, the area was geologically mapped and, geochemical and geophysical (IP) surveys completed; 42 RC drill holes were completed totaling 4,942 m, and some bulldozer trenches were also dug. The project was deemed not to meet Mantos Blancos' criteria and it was returned to its owner.

In 2005 the properties were optioned by Texas T Minerals through its Chilean subsidiary Faro S.A., then later transferred to Grandcru Resources ("Grandcru"), which initiated exploration works in October 2006 (Adkins, 2008). All previous work was verified and additional exploration carried out, including; geochemistry with new measurements of Cu and Mo content taken from trenches and pits, using a Niton portable XRF equipment; geophysics, consisting of ground magnetometry and radiometry; additional trenching; and finally 9 DDH holes were drilled for 3,311.40 m, with depths between 87 to 932 m. The objective of Grandcru's program was to demonstrate the presence of a porphyry system beneath the breccia and/or other non-outcropping breccia bodies. Results were not considered sufficiently attractive to justify the option payments, and the property was returned to its owner.

In June 2011 the properties were optioned by MCC and subsequently transfered to SCMB. Since then, the potential for Cu (Mo) porphyry style mineralization in the area has been explored via the generation of a topographic base through restitution and ortho-rectification of images with topographical control; geological mapping of outcrops and trenches at 1:2000 scale; systematic rock and soil geochemistry; geophysical studies (IP); and the three successive campaigns of RC drilling totaling 92 drill holes for 18,908 meters. The first two phases of drilling (24 holes: 4,360 m and

32 holes: 10,520 m) were aimed at the exploration of the porphyry system and the third (36 holes: 4,028 m) to provide sufficient information for a resource estimate. Collection of samples from drill core, open pit walls and trenches for metallurgical test work was also undertaken.

1.4 GEOLOGY AND MINERALIZATION

At Berta the evidence for an alteration-mineralization system with Cu and Mo extends over an area of approximately 2.3 km by 1 km, oriented NNE. The elongation of the area is clearly controlled by the Chivato Fault Zone (ZFCH), limiting the mineralization to the W. Notable differences in the geology and alteration-mineralization styles permit the separation of the area into three sectors: Berta Norte, Berta Central and Berta Sur.

Wall rocks comprise tonalite (TON) of medium-coarse equigranular texture, intruded by at least two varieties of porphyry with similar composition: namely,a “Crowded” porphyry (PTC) and a “Fine” porphyry (TFP). The first is volumetrically more abundant, cuts the tonalite showing porphyritic to equigranular textural variations, while the Fine type is younger. Igneous breccia (BXI), with various types of intrusive fragments, semi-rounded in a porphyritic matrix, and hydrothermal breccia (BXH), with angular monomictic clasts, open spaces and sulfide cements, cut the tonalite and Crowded Porphyry, but seem to pre-date the Fine Porphyry.

A NNE elongated belt of tonalite about 1 to 1.5 km wide, is bounded by foliated volcanic rocks, Cretaceous in age to the W and Jurassic to the E. However, these volcanic rocks do not host significant mineralization, except occasional narrow Au veins. Previous geological maps (Outokumpu, 1994, Guiñez and Zamora, 1997) did not recognize rocks with porphyritic textures and in general, only two belts were distinguished; "Fine textured Granodiorite" to the E and "Coarse textured Granodiorite” to the W. Coro mapping has distinguished both at surface and in drilling the porphyry varieties described above and the contact relationship between them, and with the tonalite wall rock.

The most relevant structure corresponds to ZFCH, which can be traced NNE along the western boundary of the area, where it displaces foliated intrusive and volcanic rocks in a belt approx. 50 m wide. A zone of foliated volcanic rocks, 20 to 60 m wide is also mappable along the E contact of the tonalite body with the Jurassic volcanic rocks. NW oriented faults displace the ZFCH as well as the belt of foliated rocks to the E.

A D type vein system, with sulfide filling and a sericitic halo and a predominant NW strike is recognized in Berta Norte. This can be observed at surface in several trenches, with dominant red limonite leached filling, and showing some fault planes parallel to the veins. In the northern part of Berta Central, some of these veins have been determined to have an E-W strike. The breccia bodies also exhibit control by faults varying from E-W in a large part of the Berta Central area to ENE in Berta Sur. As with the D type veins, these structures are pre-mineral.

The development of K-feldspar – biotite ± magnetite ± sericite is the most common alteration at Berta. For descriptive purposes this is named "background potassic alteration". Its intensity increases with further development of K-feldspar as igneous breccia cement and as a strong replacement of the crowded porphyry and tonalite surrounding the breccias. The sericite is preferentially developed in D type veins environment and shows greater development in the Berta Central and Norte areas. Muscovite development is found in some breccia bodies, especially at depth and in general in breccias located towards the western boundaries. Chlorite and variable sericite are best developed in porphyries and breccias, and in the best mineralized areas, the alteration contains "green grey sericite" and is characterized by the absence of magnetite, explaining why magnetic lows coincide with the mineralization. Propylitic halos with abundant chlorite and pyrite are better developed in the northern area. Within the marginal foliated rocks, especially in the west side along the ZFCH, the rocks are strongly replaced by biotite-magnetite, with some albite and actinolite. These minerals also occur as variations of background potassic alteration around the breccias in Berta Sur.

The primary mineralization consists of chalcopyrite with minor variable content of bornite. There is abundant molybdenite in some sectors but with no obvious relationship to Cu sulfides. Mineralization preferentially occurs as breccia filling and cement, to a lesser extent in veins and occasionally in veinlets. Pyrite is very poorly developed in areas of best mineralization, with greater occurrence in the northern part of Berta Central and especially in Berta Norte, where it constitutes the main filling of D type veins. Along the ZFCH, chalcopyrite occursassociated with magnetite mineralization. There is an mineralized-alteration zonation from N to S, with a propylitic border and development of veins and breccias containing pyrite ≥ chalcopyrite (molybdenite) and halos of pervasive replacement of sericite in the north, to a domain of background potassic alteration and mineralization in breccias surrounded by a crackledzone, with chalcopyrite (molybdenite, less bornite) pyrite alteration grading outwards to albite-actinolite in the south. The western boundary is dominated by breccias with muscovite containing only rare Cu mineralization and biotite-magnetite zones with some chalcopyrite that can be traced along the ZFCH. This zoning is also related to a greater abundance of porphyritic rocks toward the central and southern areas and to changes in style and orientation of structures from NW to E-W and, finally, ENE in Berta Sur.

The distribution of limonite at surface shows a direct relationship with alteration as well as with relative abundance of sulfide: yellow to yellow-reddish color predominates in the northern part related to the greater development of D type veins and sericitic alteration, while goethite and scarce jarosite make up the leach cap in the central and southern areas. In situ leaching and oxidation of the sulfides has produced a zone of copper oxides of variable thickness ranging from 30 to 120 m, generated in an environment of scarce pyrite and in poorly reactive rock. It is composed of simple green Cu oxides mineralization, with predominant chrysocolla, and black oxide (mixtures of wad type?), very low clay content, and limonite and predominant goethite. Only in some breccia bodies, mainly those located along the eastern boundary, is there limited development of supergene enrichment with chalcocite thicknesses of 2 to 10 m, invariably oxidized to a combination of hematite, “almagre” and cuprite.

The geology, mineralization and alteration of Berta Sur, corresponding to the sector of the project subject to the initial resource estimate completed in December 2012, comprises an area of 600 x 450 m evaluated according to a grid aligned 340°, perpendicular to the trend of mapped structures and after determining the orientation of mineralized bodies to be 060°. The Cu oxide mineralization is exposed on a 15 m high hill with gentle slopes, being flanked to the N and S by E-W and SW oriented creeks. This mineralization has not been previously mined and its exposure has been aided by trenches dug by Outokumpu, Mantos Blancos and Grandcru.

Berta Central occupies an area of 450 x 500 m. Most of the mineralization outcrops and a part of it have been mined out by artisanal miners. Greater than 1% Cucopper oxide mineralization occurs related to igneous-hydrothermal breccias hosted by tonalite and crowded tonalitic porphyry andcross cut by dykes of barren Fine Tonalite Porphyry. At least eight mineralized breccias bodies were modeled from NW-SE trending, 50 m spaced vertical sections using previous (Outokumpu, Mantos Blancos and Grandcru) and MCCdrill hole data. Mapping and sampling from some open cuts and underground workings as well as from some surface trenches was also digitized and incorporated into the data base.

1.5 METALLURGY

A mineralogicaland chemical characterization and metallurgical leaching test work wasundertaken by Geomet, an independent laboratory in Santiago, Chilefor samples from Berta Sur. A second column testwork program was completed at the Hydrometallurgical Lab of the Universidad de Santiago of Chile Metallurgical Mining Engineering Department (USACH) for samples from Berta Central.

The first campaign at Geomet was performed with the objective of defining the main process variables, such as copper recovery and acid consumption. For the metallurgical tests, SCMBselected three composite samples from Berta Sur, denominated as A, B and C with approximate CuT grades of 0.80%, 0.60% and 0.40%, respectively.

Based on these composites, Geomet performed the metallurgical program designed to obtain mineralogical and physical characterization, preliminary metallurgical test and column leaching test for the three composite samples at two granulometry levels of 100% - 1" (P_{80} = 19 mm), and 100% - ½" (P_{80} = 9 mm), as follows:

1. **Physical Characterization:** This characterization stage comprised: granulometry and humidity analysis at sample reception, specific gravity, and bulk density.

2. **Mineralogical characterization:** Each sample was characterized from a mineralogical point of view, by means of optical microscopy, determining the constituents of mineral material and gangue.

3. **Preliminary metallurgical test:** Preliminary tests were performed, with the objective of obtaining leaching metallurgical parameters, in order to establish the most appropriate experimental conditions for larger scale testing (pilot leaching columns) such as: contaminants determination test, Iso-pH test and Sulfation test.

4. **Column leaching test:** In order to obtain the first metallurgical conceptual engineering level parameters, leaching tests in 4" diameter (100 mm) and 2 meter high columns, for each of the grain sizes, were performed. The irrigation rate was 10 l/hrm^2. Each test was performed in duplicate; therefore, it was required to set up twelve columns in total.Tests were irrigated until completion of the leaching rate of 2 m^3/t, equivalent to 25 leaching days; including daily analysis for Cu, FeT and H^+, during the first eight days, then on an every other day basis, until the completion of irrigation. Thus, for each leaching test 18 samples were taken for kinetic evaluation, including the final drain solution.In order to validate the contaminant elements kinetics, weekly composites were taken and assayed by Inductively Coupled Plasma (ICP) (three in each test).

The most relevant conclusions from the completed study are as follows:

- Material from Berta Sur deposit presented a CuT grade of 0.83% for composite sample A, 0.63% for sample B and 0.39% for sample C.
- The average solubility of the three samples by the sulfuric acid method was 70.1% for composite A, 50.8% for composite B and 37.6% for composite C.
- The average solubility of the three composites by the citric acid method was 55.4% for A, 14.5% for B and 24.8% for C.
- The solubility rates with ferric and sodium bisulfite agent were only performed on composite B, given that it approximates the average grade of the Berta Sur resource. The average solubility rate in ferric environment was 54.5%, while in bisulfite it was 59.5%.
- The fact that the solubility maximizes while using sodium bisulfite (reduction agent), is an indicator of the presence of copper oxides species corresponding to copper wad? ($CuOMnO_2$).
- The head sample mineralogical characterization confirmed that copper would was a major component of the oxide copper species present.
- Results from Iso-pH tests, in terms of total copper extraction were 73% for composite A, 69% for B and 55% for C.
- Net acid consumption from Iso-pH tests were 15.0, 13.8, and 13.0 kg/t, in composites A, B and C respectively, equivalent to rough gross acid consumptions of 22.3, 19.7, and 15.4 kg/t, respectively.
- In terms of chemical kinetics, composite A has the fastest dissolution velocity, followed byB and finally C. Furthermore, composites B and C have kinetic similarities, but they differ greatly from A.
- Sulfation tests showed doses of 17 and 23 for composite A; 12 and 8 kg/t for composites B and C, respectively. Only composite A should use different doses for P_{80} of ¾” and ⅜”.

- In the column leaching tests, the highest copper extraction levels (78-73%) were from composite A P_{80} ¾" as well as ⅜", and B P_{80}⅜". A lower extraction level (61-65%), was for B P_{80} ¾" and C ⅜". Finally, the lowest extraction level (55%) was from sample C, P_{80} ¾".
- Extraction kinetics were identical for each grain size of composite A.
- Composite B shows a distinct difference between each grain size tested (P_{80} ¾" and ⅜"), reaching a difference of 11 points, in terms of copper extraction percentage, at the end of the leaching period.
- Composite C also shows a difference between both sizes, reaching 5.2% difference at the end of the leaching period.
- Net acid consumption varied between 19.0 kg/t (Composite A) and 22.3 kg/t (Composite B).

In order to compare the results obtained by Geomet, representative samples from the Berta Central deposit were extracted and leaching test work wasperformed at the Hydrometallurgical Lab of the Universidad de Santiago of Chile Metallurgical Mining Engineering Department.

According to field studies, Berta Central's mineralogy is similar to that of Berta Sur, tested by Geomet.

Three tests in two meters columns were performed, with the same dimensions as theutilized by Geomet, but with columns' feeding granulometry of 100% -1/2". The sulfuric acid curing dose was 10 kg/t for 24 h at a specific flow of 10 l/hm^2.

Given that the sample extracted from Berta Central has a head grade of1.4% CuT and 1.1% CuS that consumes more sulfuric acid for its higher copper content, it was decided to perform tests at 10, 15 and 20 g/l of sulfuric acid concentration in the leaching solution. Results showed a kinetic behavior very similar to that observed by Geomet, for which the Berta Central minerals are technically feasible to leach, with metallurgical results similar to the achieved by Geomet for Berta Sur, apart from the head grade differences on the samples used for the test work.

Table 1.2 shows a comparison between the metallurgical results obtained by Geomet using the material from Berta Sur and those obtained by USACH treating materialfrom Berta Central. These results corroborates that Berta Sur and Berta Central have a similar metallurgical behavior. For Berta Central's higher grade material, a higher sulfuric acid dose can be added in curing that will result in better metallurgical results.

Table 1.2: Metallurgical Column Test Work for Berta Sur & Berta Central

Column	Sample Location	Head assays		Theoretical % Sol	Actual		Days	NAC kg/t
		% CuT	% CuS		Rec CuT	Rec CuS		
P80 3/8" Comp A Geomet	BDH07-07 Drill Core (Berta Sur)	0.84	0.59	70	**91.0**	**130**	26	21
P80 3/8" Comp B Geomet	Surface trench, partially leached (Berta Sur)	0.66	0.36	55	**68.0**	**126**	28	24
P80 3/8" Comp C Geomet	Surface trench, partially leached (Berta Sur)	0.38	0.14	37	**56.0**	**150**	28	22
P80 1/2" (10 g/L H2SO4) USACH	Berta Central	1.40	1.10	79	**51.5**	**66**	28	22
P80 1/2" (15 g/L H2SO4) USACH	Berta Central	1.40	1.10	79	**80.0**	**113**	28	20
P80 1/2" (20 g/L H2SO4) USACH	Berta Central	1.40	1.10	79	**87.0**	**120**	28	28

Table 1.2 shows that the recovery of soluble copper exceeds 100% in all but one of the columns. This is due to the presence of black oxides in (copper wad?) minerals that did not report to the soluble copper assay during analysis, but is recoverable over the period of the column tests. The columns were stopped at 28 days before the recovery curves went asymptotic. Based on the results of this column test work and the soluble copper component of the deposit from drill hole assays, SCMB estimates that a recovery of 78% of the total copper in the heap leachable material should be achievable in the 60 day leach cycle contemplated for the operation. The ROM material averages 0.20%CuT and 0.12%CuS, and recoveries are estimated to be 75% of the soluble copper which is equivalent to 45% of total copper. This estimate takes into account the proposed blasting pattern of a 5x5m grid on 5m high benches which should result in a grain size slightly better than that from a first stage crusher. Leaching will take place on 7m high pads without liners between lifts, which should also result in additional recovery over time. Benchmarking against other dump leach operations in Chile indicates that they achieve recoveries of between 40 and 50% of total copper.

1.6 MINERAL RESOURCE ESTIMATION

The mineral resources described are located in mining claims originally optioned to MCC and transferred to SCMB, which has rights to acquire 100% of the property. The acquisition of the property is contingent upon making the underlying option payments.

The geology of the Berta Sur and Berta Central deposits are reasonably well understood, in terms of genesis, mineralization controls and structure. Copper oxide mineralization extends to depths of 30 to 100 m with mineralization outcropping at surface and with effectively no overburden. It also has a simple mineralization and gangue mineralogy, excellent response to leaching and fairly continuous Cu grades and sharp contacts with low-grade margin mineralization.

To separate the zones with different statistical behavior, solids were constructed to represent two mineralization types: Oxide Body and Low Grade Oxide Body. Metallurgical test considered copper grades for both types of mineralization.

This Berta report model is based on 22,213 m of drilling, mainly reverse circulation (RC) and mostly drilled by MCCin three stages completed during 2011 and 2012. Other drill holes included in the resource estimate were completed during the 1990's by Mantos Blancos and Outokumpu and diamond drilling completed by Grandcru in 2006 and 2007. Drilling and sampling procedures, sample preparation and assay protocols for all the drilling campaigns were generally acceptable and that available information was used in the resource evaluation without limitation.

The resource estimate was completedat a variety of total copper (%CuT) grades, as shown on Table 1.3

Table 1.3: Resource Estimate

Berta Project Resource Estimate														
Zone	Cutoff	Measured			Indicated			Measured & Indicated			Inferred			
		kt	% CuT	% CuS	kt	% CuT	% CuS	kt	% CuT	% CuS	kt	% CuT	% CuS	
Berta Sur & Central	0.10	16,498	0.34	0.23	8,653	0.23	0.14	25,150	0.30	0.20	4,845	0.24	0.15	
	0.15	13,275	0.39	0.27	5,780	0.27	0.18	19,055	0.36	0.24	3,249	0.30	0.20	
	0.20	10,487	0.45	0.31	3,336	0.35	0.23	13,822	0.43	0.29	2,039	0.38	0.25	
	0.25	8,355	0.51	0.36	1,961	0.44	0.30	10,316	0.50	0.35	1,402	0.45	0.31	
	0.30	6,791	0.56	0.40	1,289	0.52	0.36	8,080	0.56	0.39	932	0.53	0.37	
Berta Sur	0.10	10,972	0.32	0.21	4,423	0.18	0.11	15,394	0.28	0.18	2,105	0.18	0.11	
	0.15	8,853	0.37	0.25	2,800	0.21	0.13	11,653	0.33	0.22	1,296	0.22	0.13	
	0.20	6,892	0.42	0.29	1,332	0.26	0.16	8,225	0.39	0.27	720	0.26	0.16	
	0.25	5,385	0.47	0.33	561	0.31	0.20	5,946	0.46	0.32	343	0.29	0.18	
	0.30	4,288	0.53	0.37	261	0.36	0.24	4,549	0.52	0.36	127	0.33	0.21	
Berta Central	0.10	5,526	0.38	0.26	4,230	0.27	0.17	9,756	0.33	0.22	2,740	0.29	0.19	
	0.15	4,422	0.45	0.31	2,980	0.33	0.22	7,402	0.40	0.27	1,953	0.35	0.24	
	0.20	3,594	0.51	0.36	2,003	0.41	0.27	5,598	0.47	0.33	1,318	0.44	0.30	
	0.25	2,969	0.57	0.40	1,401	0.49	0.34	4,370	0.55	0.38	1,059	0.50	0.34	
	0.30	2,503	0.63	0.45	1,028	0.56	0.39	3,531	0.61	0.43	805	0.57	0.40	

Geoinvestment considered the basis for determining the reasonable prospects for eventual economic extraction of the Berta Sur and Central resources by completing a series of pit optimizations using the Lersch & Grossmann algorithm based on the following technical and economic parameters; mining cost of $2.09/t, processing Cost of $4.74/t ,SXEW cost of $0.102/lb, G&A cost of $0.045/lb , sales & marketing cost of $0.041/lb, metallurgical recovery of 80% (based on results obtained from the metallurgical test work), inter ramp pit slope of 50° , and a variety of copper prices. For a base case using a $3.00/lb copper price, and a 0.1%CuT cut off grade, the optimum pits were determined to contain Measured and Indicated Resources of 17.6 million tons at a grade of 0.37%CuT and an overall stripping ratio of 0.49:1, as detailed in Table 1.4 below.

Table1.4: In Pit Resource Estimate based on $3/lb Cu, 0.1% CuT cutoff

Berta Project In Pit Resource												
Zone	Pit	Measured			Indicated			Measured & Indicated			Waste	Strip
		kt	% CuT	% CuS	kt	% CuT	% CuS	kt	% CuT	% CuS	kt	Ratio
Berta Sur	Berta Sur	8,929	0.35	0.23	1,427	0.19	0.11	10,356	0.33	0.21	2,609	0.25
Berta Central	Trinchera-Salvadora	2,242	0.48	0.30	527	0.47	0.29	2,769	0.48	0.30	2,499	0.90
	Carmen-Gemela	982	0.51	0.36	562	0.38	0.26	1,544	0.47	0.32	1,852	1.20
	Nueva	219	0.43	0.29	295	0.34	0.22	514	0.38	0.25	375	0.73
	Berta II	853	0.37	0.24	150	0.36	0.23	1,003	0.37	0.24	572	0.57
	Chico	900	0.30	0.18	518	0.25	0.14	1,418	0.29	0.17	762	0.54
Berta Sur & Central	**Total**	**14,125**	**0.38**	**0.25**	**3,479**	**0.29**	**0.18**	**17,604**	**0.37**	**0.23**	**8,669**	**0.49**

This Amended updated PEA is further optimizing the project using the new operating parameters shown inTable 1.5. Mine plan also assumes a first phase using a variable cut-off grade in year 1 of between 0.60% and 0.70%CuT, in order to maintain a constant feed to the existing Nora crusher for a period of 11 months, thus postponing part of the capital investment until year 2 of operations. A total of 0.4mt at 0.83%Cu will be mined and trucked to the Nora plant while 1.2mt of lower grade heap leach material and 0.6mt of ROM will be stockpiled for processing in year 2. In addition, the Nora plant will reprocess some spent minerals stockpiles (“Ripios”) from the previous 2009-12 operation at a rate of ~30 tpm of copper cathode during Phase 1 as described in section 17.1.4 of this PEA

In Phase 2, after eleven months, considers all the copper oxide material from the open pits will be treated through a heap leach process with capacity of 1 million tonnes of mineral per year (including crushing, agglomeration and permanent pads), and the processing of 1.2 million tonnes per year of Run of Mine (ROM) material directly onto dump leach pads.

Table 1.5: Design Criteria and Mine Planning

Variable	BERTA	NORA	ROM
Mining Cost (USD/ton)	2.32	2.32	2.32
Hauling (USD/ton)	0.00	0.00	0.00
Processing Cost (USD/ton)	7.91	12.29	1.82
SX-EW Cost (USD/lb)	0.250	0.250	0.250
G&A (USD/lb)	0.090	0.090	0.090
Selling Cost (USD/lb)	0.050	0.050	0.050
Recovery	78.0%	78.0%	45.0%
Selling Price	$3.00	$3.00	$3.00

The final optimized pit contains 7.2 million tonnes @ 0.574%CuT of heap leachable material, 6.63 million tonnes @ 0.20%CuT of ROM and 7.1 million tonnes of waste, as shown below in Table 1.6 by sector. This represents a mining recovery of 89.4% of the heap leach resources and 38.8% of the ROM resources contained in the Berta resource estimate.

Readers are advised that more detailed engineering studies have not been completed for the Berta project and so the normal progression from PEA to Preliminary Feasibility Study to Feasibility Study has not been followed in respect of making a production decision. Therefore, investors are cautioned that no mineral reserves have *been declared and the level of confidence in the resources, metallurgy, engineering and cost estimation is not at a level normally associated with a project reaching a production decision.*

Table 1.6: Pit Optimization by Sector.

Sector	HL Material	CuT%	CuS%	ROM	CuT%	CuS%	Waste	Total
Berta Sur	4.178.240	0,529	0,375	4.175.360	0,203	0,122	1.448.139	9.801.739
Trinchera-S	1.130.880	0,786	0,560	1.096.000	0,186	0,111	2.663.429	4.890.309
Carmen-G	786.240	0,588	0,422	314.880	0,196	0,117	1.931.798	3.032.918
Nueva	223.360	0,567	0,401	205.440	0,209	0,126	271.977	700.777
Berta II	509.760	0,522	0,367	308.160	0,204	0,123	434.526	1.252.446
Chico	395.200	0,492	0,343	533.440	0,196	0,117	434.770	1.363.410
Total	**7.223.680**	**0,574**	**0,407**	**6.633.280**	**0,200**	**0,120**	**7.184.640**	**21.041.600**

1.7 MINING AND PROCESSING

The Project contemplates an open pit mine to extract oxide material from the Berta Sur and Central deposits using mining contractors, followed by crushing, agglomeration and heap leaching of higher grade (>0.3%CuT) material and dump leaching of lower grade (0.1-0.3%CuT) material. The resulting “PLS would then be transported by 6"-54kmpipeline to the Nora SXEW plant for recovery of copper cathode. Water and raffinate would be returned by 10"-54km pipeline from Nora to Berta. Overall material contained in the mine plan developed by Geoinvestment has 7.22 mt of heap leach material, with an average grade of 0.57% CuT and 6.63 mt of dump leach material with an average grade of 0.20%CuT. Annual average material movements represent a strip ratio of approximately 0.52:1 waste: mineral.

This Amended updated PEA also considers a project Phase 1 using a variable cut-off grade in year 1 of between 0.60% and 0.70%CuT, in order to maintain a constant feed to the existing Nora crusher for a period of 11 months.

The Berta mine plan & cathode production schedule is shown on Table 1.7, below;

Table 1.7: Berta Mine Plan

Production Profile		Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Tot
Nora Crushed	Ton	399.258	-	-	-	-	-	-	-	**399.258**
	CuT%	0,83	-	-	-	-	-	-	-	**0,83**
	CuS%	0,61	-	-	-	-	-	-	-	**0,61**
	Rec%	80,97	-	-	-	-	-	-	-	**81,0**
	Cu Cathode, t	2.673	-	-	-	-	-	-	-	**2.673**
Ripios Line	Cu Cathode, t	315								**315**
Berta Crushed	Ton	84.932	1.002.740	1.000.000	1.000.000	1.000.000	846.925	1.000.000	828.737	**6.763.334**
	CuT%	0,55	0,51	0,55	0,50	0,51	0,79	0,61	0,48	**0,56**
	CuS%	0,39	0,36	0,39	0,35	0,36	0,56	0,43	0,34	**0,40**
	Rec%	0,79	0,78	0,78	0,77	0,77	0,78	0,79	0,77	**0,78**
	Cu Cathode, t	366	4.025	4.271	3.838	3.945	5.241	4.783	3.074	**29.544**
Berta ROM	Ton	109.353	1.537.653	1.163.006	975.679	598.340	490.499	470.580	603.613	**5.948.725**
	CuT%	0,18	0,20	0,21	0,19	0,19	0,19	0,21	0,20	**0,20**
	CuS%	0,11	0,12	0,13	0,11	0,11	0,11	0,13	0,12	**0,12**
	Rec%	45	45	45	45	45	45	45	45	**45**
	Cu Cathode, t	90	1.387	1.101	812	501	408	440	541	**5.280**
Total Cu	Cu Cathode, t	3.444	5.412	5.372	4.650	4.446	5.650	5.223	3.615	**37.812**
Stockpiled Material										
Berta ROM	Ton	499.882	499.882	499.882	499.882	499.882	499.882	499.882	499.882	**499.882**
	Cu Cathode, t	490	490	490	490	490	490	490	490	**490**
Berta Leach	Ton	1.090.174	732.799	957.612	650.361	254.530	(0)	115.396	- 0	**- 0**
	Cu Cathode, t	4.036	2.281	2.922	1.896	742	- 0	330	- 0	**- 0**

Equipment and support facilities for this schedule have been costed on a mining contractor basis. The main installations for maintenance will be composed of a maintenance shop dimensions for a fleet of 3 trucks of 25 ton capacity, 2 front end loaders, 1 bulldozers, 1 grader, 1 wheeldozer, 1 water truck and 1 drill rig, representing the most cost effective option.

1.8 INFRASTRUCTURE

At the Nora Plant, power supply will be obtained from the existing electrical grid through a local distributor EMELAT that has confirmed connection feasibility point to the existing power line. At the Berta mine site power will be supplied by 1.75Mw diesel generators.

Water will be sourced from the CODELCO owned Pampa Austral tailing dams, located 10km north of Nora Plant. The Berta mine site water requirement will be supplied by 10"-54km pipeline.

Sulphuric acid may be sourced from CODELCO´s Potrerillo smelter located 85km to the northwest of Berta mine site or from ENAMI´s Paipote Smelter located 110 km to the south.

1.9 ENVIRONMENTAL AND SOCIAL ISSUES

The Evaluation Commission of the Atacama Region of Chile, part of the Chilean Environmental Evaluation Service (in Spanish, "SEA"), has approved the EID of the Berta copper project and has emitted the corresponding Resolution of Environmental Qualification (in Spanish, "RCA") on 14 October 2014. The RCA notification is attached in Annex 1 of chapter 28.

The corresponding RCA for the Nora SXEW plant was granted in July 31, 2008.

1.10 ECONOMIC AND FINANCIAL ANALYSIS

Operating Costs

Operating cost estimates reflect the current market environment in northern Chile for contract mining, crushing, sulphuric acid, power supply, cathode production by SXEW, and transportation of PLS and water, and are shown on Table 1.8 below.

Principal operating cost components are sulphuric acid at $94/t and power at $222/MW for Berta (generators) and $117/MW for Nora (connected to grid).

Table 1.8: Life of Mine Operating Costs

	$´000			$/lb		
Operating Costs	**Phase 1**	**Phase 2**	**LOM $m**	**Phase 1**	**Phase 2**	**LOM $m**
Mining	2,653	38,700	41,353	0.40	0.50	0.50
Processing	5,478	71,334	76,811	0.83	0.93	0.92
Transport	2,276	2,942	5,218	0.35	0.04	0.06
G&A	1,143	7,762	8,906	0.17	0.10	0.11
Cash Costs C1	**11,549**	**120,739**	**132,288**	**1.75**	**1.57**	**1.59**

Capital Costs

SCMB is in the process of closing the acquisition of the Nora plant and obtaining $7.15M of debt financing, to complete the following capital expenditures in Phase 1;

Area No	AREA TITLE	TOTAL $´000	PHASE 1 $´000	PHASE 2 $´000	Rest of LOM $´000
10	NORA PLANT PURCHASE & STARTUP	5,761	6,467	219	-925
20	BERTA CONSTRUCTION	6,375		6,375	
30	NORA EXPANSION	1,324		1,324	
40	PIPELINE PLS & RAFF/WATER	3,773	107	3,666	
50	Other Owner Cost	5,807	574	1,319	3,914
	GRAND TOTAL	23,040	7,148	12,903	2,989

Pre- Financing Financial Analysis

The Project has been evaluated on both a pre-tax basis and after all Chilean taxes and a 1.5% royalty due to the Berta claim owner at a base case copper price of $2.80/lb and for sensitivity, at prices of $2.60/lb and $3.00/lb as shown on Table 1.9. The project economics contemplated by this Amended updated PEA are summarized on Table 1.10

Table 1.9: Berta Economic Evaluation Summary

Cu Price	$2.60/ lb		$2.80/ lb		$3.00/ lb	
NPV ($ millions)	Pre tax	After tax	Pre tax	After tax	Pre tax	After tax
5%	42.3	32.3	55.2	41.8	68.1	52.1
8%	35.1	26.8	**46.4**	**35.2**	57.7	44.3
10%	31.1	23.7	41.5	31.5	51.8	39.9
IRR	62%	56%	**83%**	**75%**	106%	98%

Table 1.10: Summary Economics

	Revised Mine Plan		
	Phase 1	**Phase 2**	**LOM**
Copper Price	US$2.80/lb		
Copper Production	2,988	34,833	37,821
Duration	11 months	7 years	8 years
Cash Costs	$1.75/lb	$1.57/lb	$1.59/lb
CAPEX ($million)	$7.15	$12.6	$23.0[(1)]
Pre-tax:			
NPV (8%)	$46.4 million		
IRR	83%		
After-tax			
NPV (8%)	$35.2 million		
IRR	75%		

Readers are advised that more detailed engineering studies have not been completed for the Berta project and so the normal progression from PEA to Preliminary Feasibility Study to Feasibility Study has not been followed in respect of making a production decision. Therefore, investors are cautioned that no mineral reserves have been declared and the level of confidence in the resources, metallurgy, engineering and cost estimation is not at a level normally associated with a project reaching a production decision.

1.10.1 RECOMMENDATIONS

Sufficient metallurgical test work has been completed for a PEA. However, a detailed assessment of the mine plan and testing of specific samples based on the early years of production is recommended in phase 1.

For Berta Central, which will be exploited towards the end of the mine life in this plan, further drilling is necessary to investigate if more HG material is available for continuing the strategy differing initial capital. Also test work is necessary to confirm the anticipated metallurgical performance.

There is some potentially available dump material within trucking distance of the Nora plant which should be evaluated as feed for the plant in early stage Phase 1 and when Berta is being developed.

An alternative to the diesel generators proposed for mine site power supply could include solar power generation, similar to those currently being built in the area, and this should be evaluated.

Despite the execution of initials agreements, itis recommended that SCMB should conclude a sulphuric acid contract with either of the smelters located in the region.

2.0 INTRODUCTION AND TERMS OF REFERENCE

2.1 INTRODUCTION

Coro, through its subsidiary SCMB retained the services of Geoinvestment to prepare a mineral resources estimate and a PEA covering its Berta Copper property, located in the III Region, Chile. It is intended for disclosure to the Toronto Stock Exchange, where Coro is listed. The mineral code followed in this report is the CIM code and this report follows the recommendations of the National Instrument 43-101.

Sergio Alvarado, BSc (Hons.) Geology, member of CIM, CMC and IIMCh, was responsible for the overall preparation of the Technical Report as defined in National Instrument 43-101 Standards of Disclosure for Mineral Projects and in compliance with Form 43-102F1.

In preparing this report, Geoinvestment relied on reports, studies, maps, databases and miscellaneous technical papers listed in the References section of this report. Additional information and data for Geoinvestment's review and studies were obtained from SCMB on site or at SCMB´s Santiago office.

2.2 TERMS OF REFERENCE

The scope of work included an initial review of the available information, assistance regarding aspects of sample quality; interpretation (together with the MCC's geological team) and preparation of the geological model, resource estimate and the preparation of the Report.

Sergio Alvarado, Consulting Geologist, completed the initial site visit on October 22nd and 23rd 2012. In this visit, besides the familiarization with the geology and site conditions, the core yard was visited and Quality Control aspects were discussed. Also Mario Orrego, Geostatistics specialist, attended the site visit.

Database validation, preparation of vertical geological interpretation solids modeling and geostatistical analysis of the drill hole data were conducted. An assessment of the quality of these data relative to industry standard practiceswas also made.

Geoinvestment is not an associate or affiliate of Coro nor of any associated company, or any joint-venture company. Geoinvestment's fees for this Technical Report are not dependent in whole or in part on any prior or future engagement or understanding resulting from the conclusions of this report. These fees are in accordance with standard industry fees for work of this nature, and Geoinvestment's previously provided estimates are based solely on the approximate time needed to assess the various data and reach appropriate conclusions. This report is based on information known to Geoinvestment as of May 30th 2015.

2.3 CONTRIBUTORS TO THE REPORT

The report is based on Geoinvestment "Updated Geology and Mineral Resources Estimates for Berta Project, Inca de Oro, III Region, Chile" completed in September 2013 that is also based on Propipe "Geology and Mineral Resources Estimates for Berta Project, Inca de Oro, III Region, Chile" completed in January 2013

Mr. Sergio Alvarado, is the QP responsible for the overall preparation of the report and the geology, resource estimate and geotechnical study sections. Mr Enrique Quiroga, member of Chilean Mining Commision, is the QP responsible for those sections relating to mining, metallurgy and process design, engineering and cost estimation. Other contributors who supplied data and information for the report are as follows:

- Jaime Simpson, a metallurgical engineer with more than 25 years of experience and employed by ProPipe as Development and Research Manager.
- Victor Araya, a civil mechanical engineer with over 15 years of experience in project development and management, and a board member of ProPipe and partner companies.
- Oscar Rosas, a metallurgical engineer with more than 27 years of experience and General Manager of SCMB

Table 2.1 Amended Updated PEA Contributors

Table 2.1: Amended Updated PEA Contributors

Section	Participant(s)
Overall / Introduction	Sergio Alvarado
Geology	Sergio Alvarado
Resource estimate	Sergio Alvarado
Mine Plan and Schedule	Enrique Quiroga
Site Geotechnical	Sergio Alvarado
Power and Water Supply	Enrique Quiroga/ Victor Araya
On-Site and Off-Site Infrastructure	Enrique Quiroga/ Victor Araya
Site Earthworks	Sergio Alvarado
Metallurgy test work	Enrique Quiroga / Jaime Simpson
Metallurgy interpretation and Modeling	Enrique Quiroga / Jaime Simpson
Processing	Enrique Quiroga / Jaime Simpson
Environmental & Social issues	IAL Consultores / BORDOLI & Consultores EIRL – Sergio Alvarado
Capital Costs	Enrique Quiroga / Victor Araya
Operating Costs	Enrique Quiroga / Oscar Rosas
Project Implementation	Enrique Quiroga / Victor Araya
Marketing	Coro
Financial Analysis	Coro - Oscar Rosas

3.0 RELIANCE ON OTHER EXPERTS

Geoinvestment has relied on a legal opinion of the status of SCMB and its mining properties provided by SCMB's lawyers, Bofill, Mir and Alvarez Jana of Santiago, Chile.

Geoinvestment has relied on the environmental studies completed by IAL Consultores and BORDOLI & Consultores EIRL and on the Resolution of Environmental Qualification (in Spanish, "RCA") issued by the Evaluation Commission of the Atacama Region of Chile, part of the Chilean Environmental Evaluation Service (in Spanish, "SEA") for the environmental status of the project.

The results and opinions expressed in this report are reliant upon the aforementioned information being current, accurate, and complete as of the date of this report, and the understanding that no information has been withheld that would affect the conclusions made herein. Geoinvestment reserves the right, but will not be obliged, to revise this report and conclusions if additional information becomes known to Geoinvestment subsequent to the date of this report. Geoinvestment does not assume responsibility for Coro's actions in distributing this report.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 PROPERTY DESCRIPTION

The Berta project is located in an area that contains evidence of mineralization of copper oxides and sulfides, partly explored and exploited in the past, and covering an area of about 6 km^2. The area covered by Coro'sexploration comprises an area of 2 km N-Sby 1 km E-W. The mining property that protects the project totals 4,000 hacovering and surrounding the area of interest. Figure 4.1 shows an overview of the project area for illustrative purposes.

Figure 4.1: The Mining Property (North to the Top)

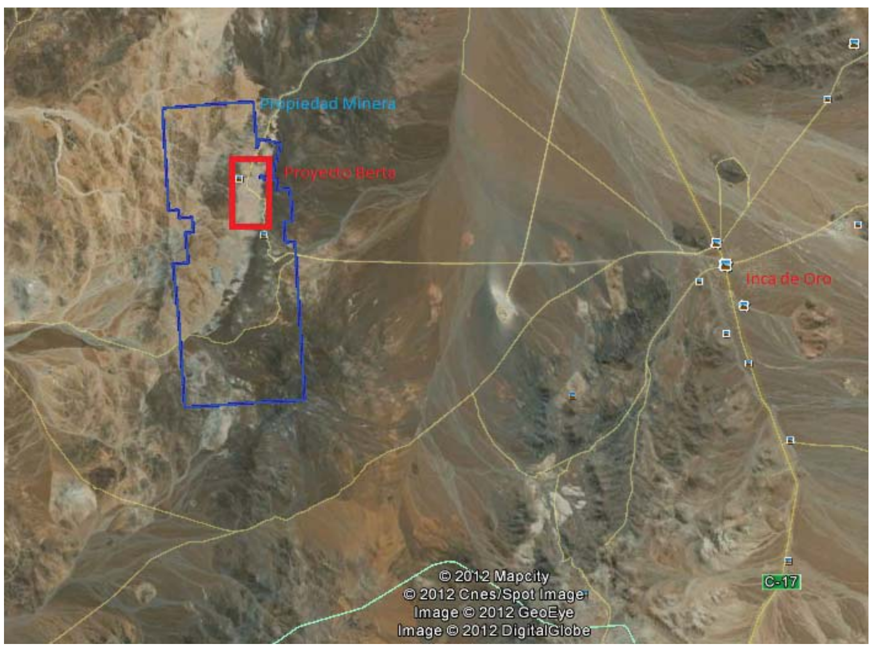
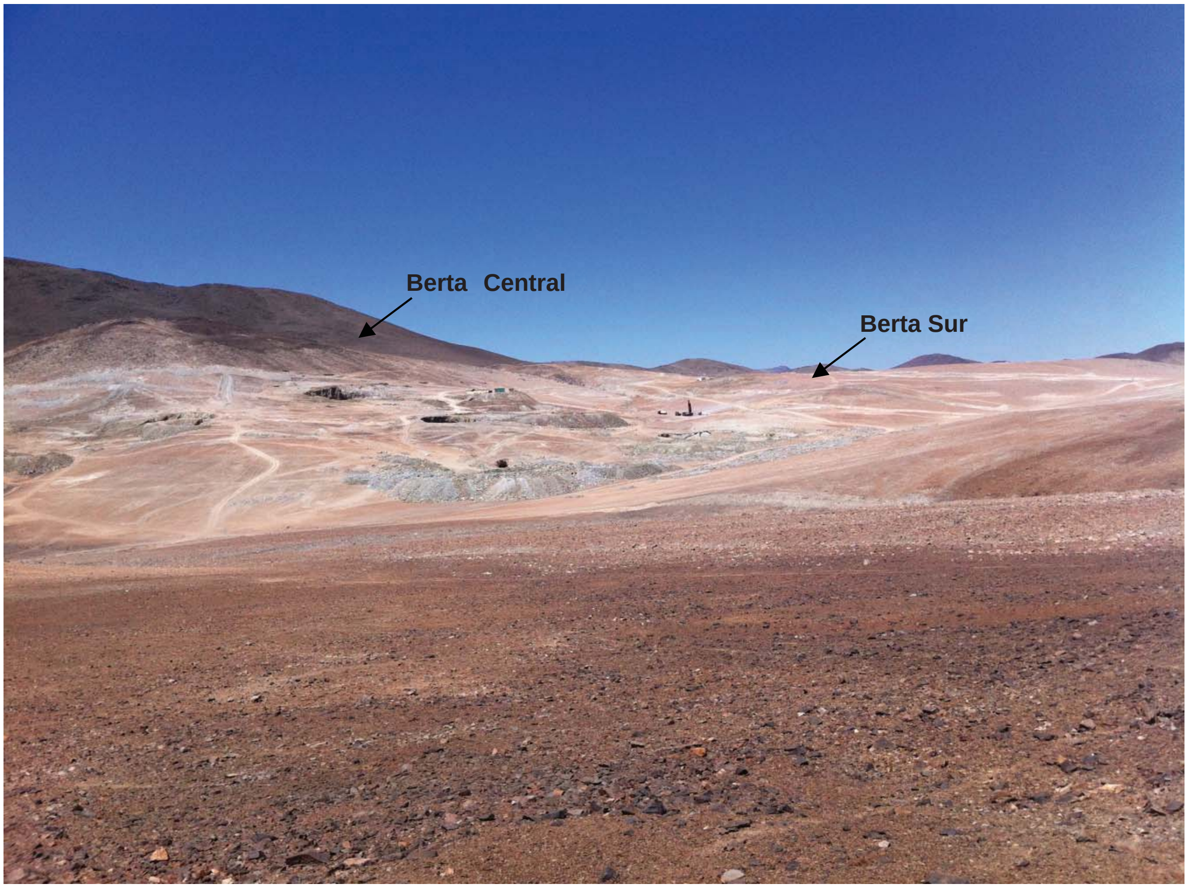


Figure 4.2 shows a panoramic view of the Berta Project, looking SE. Pits and dumps at Berta Central are in the central part and Berta Sur is locatedin the small hills to the right. Note drill rig for scale.

Figure 4.2: Berta Project panoramic view looking southeast



4.2 LOCATION

The Property is located in Chañaral Province, III Region, Northern Chile, at the approximate latitude 26º43'S and longitude 70º03'W, approximately 20 km Wof the village of Inca de Oro, at an elevation of 1700 m. It is situated about 750 km Nof Santiago, 75 km NNE of Copiapó, and 70 km SEof the port of Chañaral (Figure 4.3). The UTM coordinates of the center of the Property are approximately 395,000 E and 7,044,100 N, UTM Zone 19-J, Provisional South American 1956 datum.

The project is located approximately 33 km SEof AngloAmerican´s Manto Verde SXEW operation that produces 60,000 t of Cu per year. Codelco's El Salvador mine, with a production of 69,000 t Cu is located about 68 km NEof Berta. TheInca de Oro

(PanAust Limited) and Santo Domingo (Capstone Mining Corp.) development projects are located 15 km Eand approximately 30 km NE respectively. The project is located in a mining region that also contains numerous operations of small and medium mining of Cu and Au, several of which supply minerals to the state mining company ENAMI, which has a processing plant in the town of El Salado, located about 41 km NW of Berta (Figure 4.3).

Figure 4.3: Location Map (To the Top)

4.3 PROPERTY TITLE IN CHILE

Chile's mining policy is based on legal provisions that were enacted as part of the 1980 constitution. These were established to stimulate the development of mining and to guarantee the property rights of both local and foreign investors. According to the law, the state owns all mineral resources, but exploration and exploitation of these resources by private parties is permitted through mining concessions, which are granted by the courts. The concessions have both rights and obligations, as defined by the Constitutional Organic Law on Mining Concessions (JGRCh, 1982) and the Mining Code (JGRCh, 1983). Concessions can be mortgaged or transferred, and the holder has full ownership rights and is entitled to obtain the rights of way for exploration (*pedimentos*) and exploitation (*mensuras*). In addition, the concession holder has the right to defend his ownership against state and third parties. A concession is obtained by a claim application and includes all minerals that may exist within its area. Mining rights in Chile are acquired in the following stages:

- Pedimento: A pedimento is an initial exploration claim whose position is well defined by UTM coordinates which define north-south and east-west boundaries. The minimum size of a pedimento is 100 ha and the maximum is 5,000 ha, with a maximum length-to-width ratio of 5:1. The duration of validity is for a maximum period of 2 years; however, at the end of this period, and provided that no overlying claim has been staked, the claim may be reduced in size by at least 50% and renewed for an additional 2 years. If the yearly claim taxes are not paid on a pedimento, the claim can be restored to good standing by paying double the annual claim tax the following year. New pedimentos are allowed to overlap with pre-existing ones; however, the underlying (previously staked) claim always takes precedence providing the claim holder avoids letting the claim lapse due to lack of payments, corrects any minor filing errors and converts the pedimento to a manifestación within the initial 2-year period.

- Manifestación: Before a pedimento expires, or at any stage during its two year life, it may be converted to a manifestación. Within 220 days of filing a manifestación, the applicant must file a "Request for Survey" (Solicitud de Mensura) with the court of jurisdiction, including official publication to advise the surrounding claim holders, who may raise objections if they believe their pre-established rights are being encroached upon. A manifestación may also be filed on any open ground, without going through the pedimento filing process.
- Mensura: Within 9 months of the approval of the "Request for Survey" by the court, the claim must be surveyed by a government-licensed surveyor. Surrounding claim owners may be present. Once surveyed, presented to the court and reviewed by the National Mining Service (SERNAGEOMIN), the application is adjudicated by the court as a permanent property right (a mensura), which is equivalent to a "patented claim".

At each of the stages of the claim acquisition process, several steps are required (application, "publication", "inscription payments", notarization, tax payments, "*patente* payment", lawyers' fees, publication of the extract, etc.) before the application is finally converted to a "declaratory sentence" by the court constituting the new mineral property. A full description of the process is documented in Chile's Mining Code (JGRCh, 1983).

Many of the steps involved in establishing the claim are published weekly in Chile's official mining bulletin for the appropriate region. At the *manifestación* and *mensura* stages, a process for opposition from conflicting claims is allowed. Most companies in Chile retain a mining claim specialist to review the weekly mining bulletins and ensure that their land position is kept secure.

4.4 COMPANY OWNERSHIP AND AGREEMENTS TERMS

Coro is a British Columbia company incorporated under the Business Corporations Act of B.C., incorporated on September 22 2004, with a registered office at Suite 1280, 625 Howe Street, Vancouver, British Columbia, Canada, V6C 2T6.

Coro owns all the shares in 0904213 B.C. Ltd (A company incorporated in British Columbia, Canada) which owns all the shares in Sky Dust Holdings Limited ("Sky Dust") (A company incorporated under the BVI Companies Act, 2004). Sky Dust owns all the shares in Machair Investments Ltd ("Machair") (A company incorporated under the BVI Companies Act, 2004).

Machair beneficially owns a 100% of Minera Coro Chile Limitada ("MCC"), a limited liability Chilean Company established under the laws of Chile on April 18, 2011.MCC currently beneficially owns 82% of Sociedad Contractual Minera Berta ("SCMB") (a company incorporated under the laws of Chile on June 4, 2013), which will reduce to 82% upon submission of this PEA.

On June 13, 2011 Coro announced it had reached an option agreement with a local owner for 506 ha of pending, measured and measurable concessions, all existing and registered that protect the main part of the project. They are listed in Table 4.1 andare shown in Figure 4.4. The terms of the option were renegotiated in May 2013 and again in June 2014 and areshown inTable 4.1The underlying option terms were modified in 2013 reducing the total payments from $6.0 million to $4.0 million in return for the introduction of a 1.5% NSR on all copper production, and again in 2014 by deferring the $2.5 million final payment and providing a financing option for the final payment due in August 2015. The 2014 amendment also allowed for the deposit to be mined after the payment on August 2014. The financing option allows for the August 2015 payment to be divided into eight quarterly payments of $281,250 plus interest accruing at LIBOR.

Table 4.1: Property Option Terms

	Current Terms	Status
On June 10th, 2011	$ 200,000	Paid
On June 10th, 2012:	$ 800,000	Paid
On June 10th, 2013:	$ 500,000	Paid
On August 14th,2014:	$250,000	Paid
On August 14th,2015:	$2.25 million	
TOTAL	**$ 4.0 million**	
	An NSR of 1.5% on all copper oxides and sulfide production and its by-products	

Additionally to adequately protect the area of interest, Coro has registered approximately 4,000 ha exploration concessions, named Berta 1 to Berta 14. These properties are shown in Figure 4.4.

All concessions are valid according to the Mining Code of Chile. Apart from the option payments and the NSR, no other payment obligations exist on the properties that protect the project.

The cost of maintaining the mining property ("the Property"), in terms of annual fees payable each March totals $6,600.The outstanding transaction and surveying costs to achieve measured exploitation concession status is estimated to be $75,000.

The exploitation claims allow the owner to exploit the minerals in the subsurface. In the project there is no reference to the existence of easements or other commitments with third parties that may affect the development of a mining operation in the future.

On May 7th 2013, MCC signed a Letter of Intent with ProPipe whereby ProPipe may earn up to 50% of the shares in a new company called SCM Berta, formed on June 4th 2013, by completing a series of payments, work commitments and project financing, thereby earning percentages of that company as follows;

- Making the $500,000 option payment due on June 10, 2013: 10% earned
- Completing and filing an Environmental Impact Declaration ("EID") by July 30, 2013: 3% earned
- Completing a NI43-101 compliant PEA by September 30, 2013: 5% earned

- Obtaining and structuring project financing on non-recourse basis, at market conditions, with funds available within 6 months of completion of the PEA, for a minimum of 70% of the project cost, including a cost overrun facility, as determined in the PEA. In the event that this financing is for 100% of the project cost, ProPipe will earn 32% of SCM Berta, for a total shareholding of 50%. If the financing is between 70% and 100% of the required funding, ProPipe will earn a pro-rata shareholding in SCM Berta. At the minimum 70% level, they would earn 22.4% of SCM Berta, for a total shareholding of 40.4%. In the event that less than 100% funding is received, ProPipe have the right to earn the corresponding shareholding for the percentage difference in funding, or to assign their right to do so to a third party on the same terms. In the event that they do neither, they must complete such additional work and reports as required by Coro by March 31 2014, for Coro to obtain the financing required and thus earn the corresponding shareholding.
- In the event that ProPipe do not arrange a minimum of 70% project financing, they must complete a NI43-101 compliant Definitive Feasibility Study (“DFS”) for the project by March 31 2014, and by so doing, will earn an additional 7% shareholding, for a total shareholding of 25% in SCM Berta. Coro and ProPipe will then seek project financing on a pro-rata basis.
- ProPipe will be Operator during the development and construction of the project, thereafter the Operatorship will alternate every 2 years.
- The dates shown above for completion of the various project earn in stages were subsequently extended by mutual agreement of the parties, and a shareholder’s agreement signed by the parties, has replaced the LOI.

ProPipe paid the $500,000 option payment due on June 10th 2013 and earned a 10% interest in SCMB. ProPipe earned an additional 3% interest in SCMB by completing and submitting the EID and will earn, an additional 5% interest was earned upon the submission of October 2014 PEA, taking its interest to 18%.

SCMB is in the process of financing the acquisition of the Nora solvent extraction and electro winning (“SXEW”) plant, located 4km north of the town of Diego de Almagro and 42km north of Berta. **On August 10th 2015 Coro announced the closing of the ofthe first trench of $5.1 million of the convertible debenture. On September 1st 2015, Coro announced the acquisition of the Nora Plant.**

In conjunction with the ProPipe agreement, in June 2013, the underlying option agreement with the local owner was transferred from MCC to SCM Berta, together with the Berta 1-14 exploration claims.

Geoinvestment has not reviewed in detail the mineral titles or agreements to assess the validity of the stated ownerships of the mining, exploration, water and land concessions, and relied on the documentation supplied by SCMB lawyers.

4.5 LAND TENURE

A summary land tenure map is presented in Figure 4.4. The Property is comprised of two groups of concessions:

Group I: Six optioned exploitation concessions, covering 206 ha and one exploration concession, covering 300 ha as shown Table 4.2. The mining license fees to date have been paid and the claims are in good standing. Exploration concession Chivato is superimposed on some of the above mentioned optioned concessions.

Table 4.2: Optioned Concessions (Group I)

Concession	National Record	Surface (ha)	Registration Data				Type of Concession
			Page	#	Year	Office	
Berta 1/20 (1/14)	031020 887-6	70	33	14	1967	Diego de Almagro	Mining
Salvadora 1/3	031020885-K	15	205	70	1936	Diego de Almagro	Mining
Salvadora 1/5	031022188-0	25	224	45	1994	Diego de Almagro	Mining
Elisabeth 1/8	031022305-0	68	56	16	1996	Diego de Almagro	Mining
Miguel 1/10	031023114-2	10	274	236	2003	Diego de Almagro	Mining
San Carlos 1/3	031022303-4	18	164	31	1995	Diego de Almagro	Mining
Chivato	031020430-2	300	2418	1808	2010	Diego de Almagro	Exploration
Total		**506**					

- Group II: Eight registered exploration concessions Berta 1 to 8, covering approximately 2,400 ha totally covering the Group I concessions. The purpose of staking these concessions is to establish a second layer of protection to the Group I concessions, and to secure some free areas. Some of these concessions are overlapping other third party concessions, which retain priority according to the Chilean law.

4.6 SURFACE RIGHTS

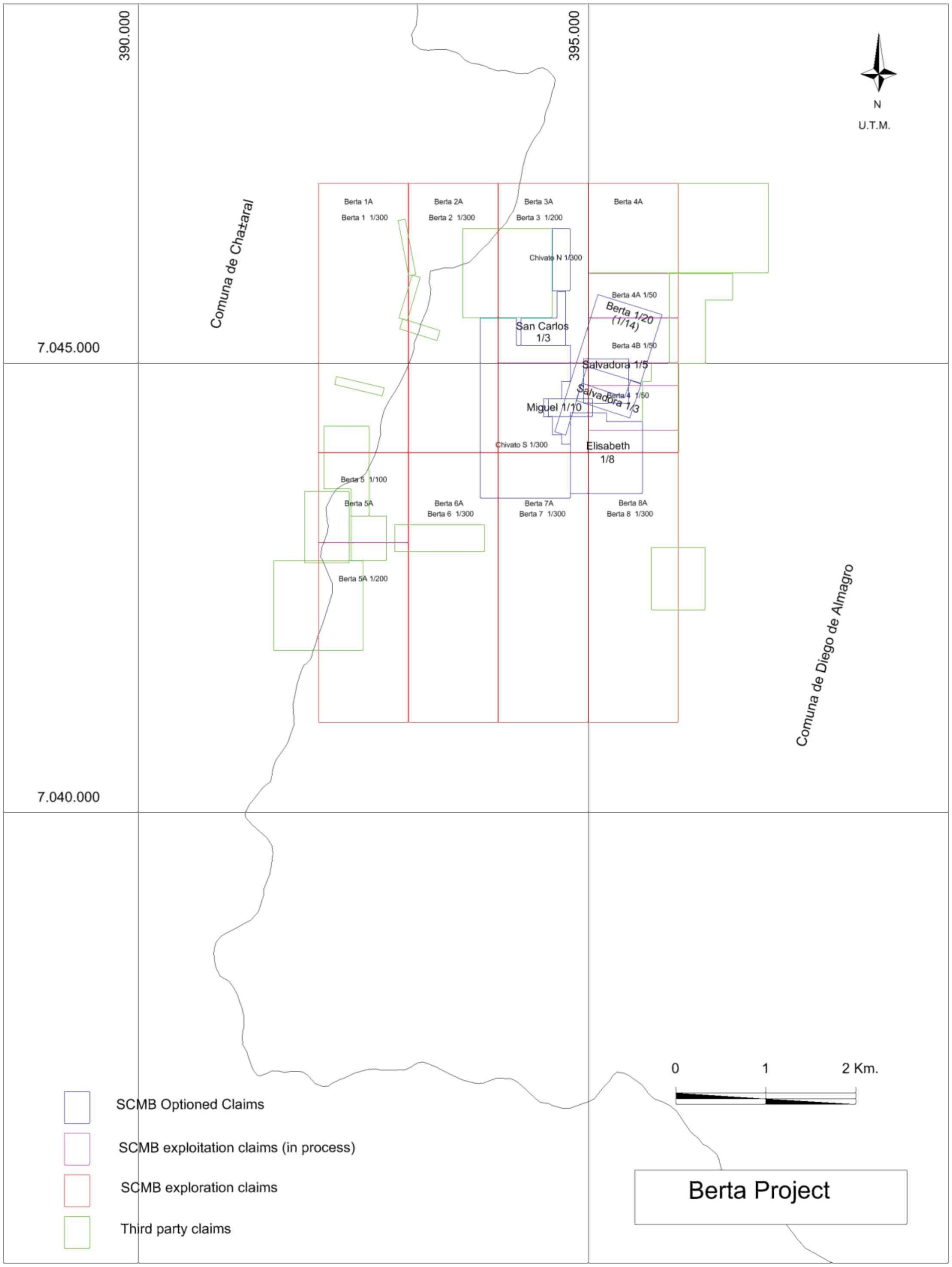
SCMB is currently assessing surface land rights in the Berta Property area. However, in accordance with the Chilean Mining Code, any titleholder of a mining concession, whether for exploration or exploitation, shall have the right to establish an occupation easement over the surface land, as required for the exploration or exploitation of its concession. In the event that the surface property owner is not agreeable to grant the easement voluntarily, the titleholder of the mining concession may request said easement before the Courts of Justice who shall grant it upon determination of the compensation for losses as deemed fit.

The Nora plant surface property is subject to a lease agreement between the National Land Agency of Atacama and SCMB.

4.7 WATER RIGHTS

SCMB has investigated the ownership of the water rights in the area. The most important source is the CODELCO Salvador Pampa Austral tailings dam and SCMB has reached an agreement with CODELCO for the use of15 lps from June 2015 which is sufficient to fulfil plant requirements.

Figure 4.4: SCMB Land Tenure Map



5.0 ACCESS, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The Property can be accessed by road from Santiago via Copiapó - Inca de Oro through the Pan American highway and the C-17 paved highway (844 km). A good gravel road, C-217, connects Inca de Oro with the Property (15 km). Details of the route to the Project are presented in Table 5.1 and Figure 4.3. Other gravel roads link the Property with Chañaral (100 km) and Diego de Almagro (50 km).

Table 5.1: Access Routes to the Berta Property from Santiago

Route	Distance (km)	Number Route	Drive Time (hours)	Conditions
Santiago to Copiapó	737	5	10	Asphalt highway
Copiapó to Inca de Oro	92	C-17	1	Asphalt highway
Inca de Oro to Berta	15	C-217	0.5	Gravel road
TOTAL	**844**		**11.5**	

Another alternative to reach the Property is by plane from Santiago toCopiapó, the capital of the III Region, and one of the main Chilean mining centers. Copiapó airport is served by numerous daily flights, which connect Copiapó with Santiago, La Serena, Antofagasta and Calama. The airport is located 55 km west of Copiapó and 17 km south of Caldera, along the Pan American highway.

5.2 PHYSIOGRAPHY

The Propertyis located at an average elevation of 1,700 meters, ranging from 1,500 meters on the plain to 1,900 meters in the hillsthat form the Sierra del Chivato Nuevo whichformsthe eastern limitof a larger physiographic unit known as the Cordillera de la Costa.

The topography is characterized by smooth shaped hills typical of a mature landscape, surrounding a wide pass that extends from NNE-SSW to N-S, draining from Berta to both N and S.

The Property, with its shallow relief, offers good alternatives for siting of heap leach pads, and wastedumps. It is also favorable for the location of mine and processing facilities.

5.3 CLIMATE, VEGETATION AND FAUNA

The Property is located in the intermediary depression of the Atacama Desert. The normal desert climatein areas between 1,000 and 2,000 m.a.s.l., is characterized by a very low relative humidity, virtual lack of precipitation, practically no oceanic influence and clear skies during the whole year[1].

The average annual precipitation and evapotranspiration rates do not exceed 10 mm/year (IDICTEC, 1994). Due to the extreme aridity, there is practically no natural vegetation or fauna in the area, with the exception of occasional insects, lizards and small mammals.

The Atacama Desert along the Pacific coast of Chile and Peru is one of the driest, and possibly oldest, deserts in the world. A detailed study conducted between 1994 and 1998 (McKay et al., 2003), determined that the average air temperature was 16.5°C and 16.6°C in 1995 and 1996, respectively. The maximum air temperature recorded was 37.9°C, and the minimum was -5.7°C. Annual average sunlight was 336 W/m^2 and 335 W/m^2 in 1995 and 1996, respectively. Winds averaged a few meters per second, with strong *föhn*[2] winds coming from the west exceeding 12 m/s.

Between 1994 and 1998 there was only one significant rain event of 2.3 mm, possibly as rainfall from a heavy fog, which occurred near midnight local time. It is of interest that the strong El Niño of 1997-1998 brought heavy rainfall to the deserts of Peru, but did not bring significant rain to the central Atacama Desert in Chile. Dew occurs frequently following high levels of night time relative humidity, but is not a significant

[1] www.meteochile.cl

[2] Föhn winds: A föhn wind or foehn wind occurs when a deep layer of prevailing wind is forced over a mountain range. As the wind moves upslope, it expands and cools, causing water vapor to precipitate out. This dehydrated air then passes over the crest and begins to move downslope. As the wind descends to lower levels on the leeward side of the mountains, the air heats, as it comes under greater atmospheric pressure creating strong, gusty, warm and dry winds.

source of moisture in the soil or under stones. Groundwater also does not contribute to surface moisture.

At Inca de Oro, a typical location with this kind of climate in the III Region, located 15 km east of the Property, the average temperature difference between day and night is 12°C, the average daily precipitation rate is 15 mm, and the relative humidity reaches 40%[3]. A SSW wind direction predominates 80% of the time, with velocities of 1.5 m/s to 4.0 m/s, averaging 2.8 m/s.**Error! Reference source not found.**Table 5.2 shows average meteorological parameters at several meteorological stations in the III Region[4].

Table 5.2: Average Meteorological Parameters in Selected Stations (II and III Region)

Copiapó (Chamonate)	(27° 18' S - 70°25' W / 291 m.a.s.l.)												Total
	J	F	M	A	M	J	J	A	S	O	N	D	
Ave. Temp. (°C)	15.1	14.9	13.9	12.1	10.3	8.7	8.6	9.4	11.2	12.7	14.1	14.8	12.2
Low Temp. (°C)	5.1	5.5	4.4	2.2	0.7	-0.5	-0.9	-0.9	0.4	1.7	2.8	3.6	2.0
High Temp. (°C)	24.1	24.1	23.6	23.0	22.1	20.6	20.9	21.5	22.6	23.7	24.2	24.4	22.9
Precip. (mm)	0.0	0.0	0.0	0.0	0.0	0.1	0.3	0.6	0.5	0.1	0.1	0.0	1.7

Source: www.atmosfera.cl

5.4 LOCAL RESOURCES AND INFRASTRUCTURE

Inca de Oro, Diego de Almagro, and Chañaral are small towns (populations below 20,000), which mostly provide labor for the fishing or mining industry. These towns are able to support basic needs (food, accommodations, communications, fuel, hardware, labor) for early stages of exploration. More advanced projects must be serviced from Copiapó, Antofagasta, La Serena or Santiago.

Table 5.3: Local Population

Population	
Chañaral	13,543
Diego de Almagro	18,589
Inca de Oro	900

[3] www.meteochile.cl
[4] www.atmosfera.cl

A power line linking Copiapó to Diego de Almagro passes 15 km to the Eof the Property. Cellular communication is possible from various high points in the vicinity of the Property.

The closest port facility is at Barquito, adjacent to the town of Chañaral, 70 km NW of the Property, which is used by Codelco's Salvador mine for exporting its production.

Thereare no active streams and nor identified underground water resources on the Property. The Salado River, which drains very brackish waters from the Salar de Pedernales, contains underground waterways and was used in the past as drainagefor the concentrator tailings from El Salvador. Also there are some wells around Inca de Oro, which reportedly have reduced flows.

As mentioned in Section 5.4 0, SCMBhas not yet acquired any surface or water rights in the Property area.

5.5 AN OVERVIEW OF CHILEAN MINING

As the result of natural advantages and a favorable legislation, the Chilean mining sector has become particularly attractive for investors, thus reaching a boom of mining exploration by the mid of the eighties. Mining in Chile, especially copper and gold, is experiencing an important growth on the basis of the following natural, technological and administrative measures:

- Chile has the largest reserves of copper with a quarterof the world's total according to US Geological Survey (USGS) data.
- Gold reserves account for 7% of the world's total and the development of important new projects such as Pascua Lama, Cerro Casale, Lobo-Marte and El Morro, will putChile into the top ten largest producers of gold in the world.
- Large deposits permit massive mining, low-cost modern technology such as open-pit extraction, use of big trucks, shovels and conveyor belts, leaching technologies, etc.
- A very good road system.
- The country has well qualified and first-class human resources.

- Clear regulations and contractual guarantees through Decree Law 600 facilitate the participation of foreign investors.
- The country's economic and political stability creates a favorable margin for the development of mining activities.
- Confidencein the fulfillment of contracts is normaland customs formalities are expeditious.

Investments made to date have allowed copper production to grow from about 3.5 million tonnes in the 1990's to 5.2 million tonnes in 2011.

In the case of gold, the production has been rather cyclical during the last decades, varying between 38t toover 50t. Table 5.4 shows Copper, Molybdenum and Gold production. It is however forecasted that this figure will increase from the current 40t to at least 90t when the Pascua Lama and Cerro Casale projects startup.

It should be noted that during the last decades non-metallic mining has also expanded. The production of nitrates, iodine, sulfuric acid, lithium carbonate and calcium, sodium chloride and boric acid, among others, has significantly increased.

Table 5.4: Copper, Molybdenum and Gold Production

Year	Copper million ton	Molybdenum million ton	Gold(1) kg
2001	4,766	33,492	42,673
2002	4,617	29,466	38,688
2003	4,909	33,374	38,954
2004	5,413	41,883	39,986
2005	5,321	48,041	40,447
2006	5,360	43,277	40,753
2007	5,557	44,912	41,527
2008	5,330	33,687	39,162
2009	5,389	34,925	40,834
2010	5,419	37,185	38,417
2011	5,263	40,889	45,137

Source: Cochilco, (1) Sernageomin.

5.5.1 LARGE-SCALE MINING

Due to the quality and size of mining deposits, most of the large-scale mining companies of the world have mining assets in Chile. In addition, some of these companies have established important offices in Santiago to manage their businesses in South America.

The large-scale mining companies that produce copper, gold and silver, both public and private, formed the Mining Council (Consejo Minero de Chile A.G.) in 1998, to fill a gap that was missing according to the industry. The Consejo's purposes are, firstly, maintaining close relations with the government; and secondly, increasing esteem for mining by the Chilean people due to the importance the sector has in the country's economy.

In this context, it should be mentioned that local and world mining show a strong trend to concentration of properties and to "giant models" of exploitations. This is reflectedin the merger of companies and acquisition of companies and properties by the largest mining corporations.

5.5.2 MEDIUM- AND SMALL-SIZED MINING

There are a number of medium- and small-sized producers in Chile, both of copper and other products. According to Sociedad Nacional de Minería (Sonami) that comprisessmall- and medium-sized miners, medium-sized mining is asector exploiting between 300 and 8,000 tpd mineral (100,000-3,000,000 tpy). By applying this to a worksite that is representative of copper mining, a medium-sized mining company produces up to 50,000 tpy of fine copper.

An important institution for small- and medium-sized miners is the Empresa Nacional de Minería (Enami), which main purpose is fostering the development of medium - and small - sized operations. This includes funding resource definition, providing advicein the preparation and assessment of projects, training and assigning credit in order to support the commissioning of viable projects, including support to equipment, development of worksites, working capital and emergencies.

Enami also purchases minerals and concentrates from medium and small-sized producers paying a price that reflects the international price after deducting the costs of processing. Mineral purchased isprocessed in its own plants (Manuel A. Matta, José A. Moreno, El Salado and Vallenar) to obtain concentrates which are smelted at the Paipote smelter, or cathode.

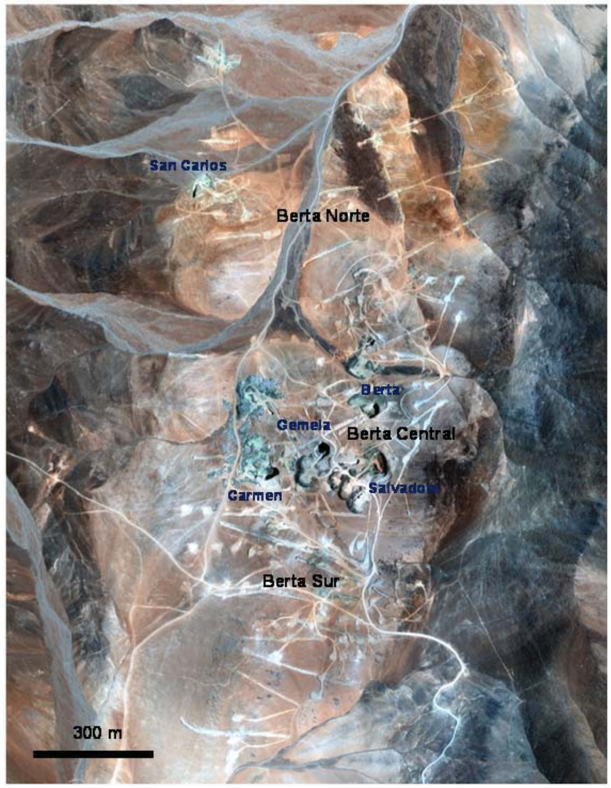
5.5.3 MINERAL RESOURCE DATA

Over the last 25 years, new geologic data in Chile have been generated at an increasingly rapid pace by state agencies, universities and private industry. This progress is largely driven by governmental mapping and industry mineral exploration programs. New digital geological, lithotectonic, geophysical and hydrogeological maps are constantly being produced by the Chilean state geological agency SERNAGEOMIN and a project started in 1999, the Multinational Andean Project (MAP). MAP is the result of collaboration between the Canadian International Development Agency, the Geological Survey of Canada and the National Geoscience Agencies of Chile, plus Argentina, Bolivia and Peru, which will continue to help in the understanding of the metallogeny of Chile (and other parts of South America) and assist in the future development of mineral resources.

6.0 HISTORY

There is abundant evidence of superficial copper mineralization in the area; however the oldest mining was directed to the exploitation of superficial narrow Au veins, with copper mining limited to minor exploitation. There is no history of these mining properties prior to Mr. Oscar Rojas Garin’s acquisition during the late 80's. The exploitation at a small-scale mining level was extended to mechanized extraction during the 1980's and 90's through the development of small pits and declines (Figure 6.1) According to the existing information (Guiñez and Zamora, 1998) in 1995 a local mining company, developed the Gemela and Carmen oxide bodies) producing more than 100,000 t of mineral at an average grade of 1.68% CuT. If the exploitation of three other small bodies (Salvadora; Berta, San Carlos) is included, the total mineral material extracted at Berta approximates 200,000 t at 1.5% CuT.

Figure 6.1: Berta Project, main explored areas and old mine workings (North to The Top)



Outokumpu (Outokumpu Explorations, 1994) carried out geological, geochemical and geophysical exploration between March and September 1994, completing 48 short airtrack holes and 7 reverse circulation (RC) holes for a total of 2,216 m. The details of this work are shown in Table 6.1 with trenches and drill holes shown on Figure 6.2

These results did not meet Outokumpu minimum target size and therefore the area was returned to the owner.

Table 6.1: Summary of exploration work done at Berta Project

Company	Years	Exploration Works	Reference
Outokumpu	Mar – Sep 1994	Surveying and geologic mapping (1.5 x 1.1 km area) 1:1,000 scale. Bulldozer trenching, sampling (11 trenches, 3 km total). Geochemistry and 48 shallow drill holes (15 to 25 m depth; 1,093 m; 817 rock samples). Geophysics: 7 electromagnetic lines surveyed (5,850 m). Drilling: 7 RC holes totaling 1,123 m.	Outokumpu, 1994
Cia. Minera San Rafael	1995	Reported by Mantos Blancos, some drilling at Carmen and Gemela Breccias subsequently mined by ramp and pit.	Guiñez and Zamora, 1998
Mantos Blancos	Sep – Dec 1997	Surveying on a 2 x 1.1 km area, including mine works and previous trenches and collar holes. 1:1,000 geological mapping. Trench sampling at 5 m intervals. IP spectral geophysics along 7 EW 900 m long cross-sections (6.3 km). Drilling: 4,942 m in 42 RC holes.	Guiñez and Zamora, 1999
Grandcru	Oct 2006 – Jul 2008	Surveying. Systematic analysis of trenches by means of portable XRF Niton equipment. Ground magnetics and rodiometrics. Drilling of 3,311.40 m in 9 DDH holes.	Adkins, 2008
MCC.	Jul 2011 – Aug 2012	Image orthorectification and contour restitution at 1:5,000 scale. Ground surveying. IP/R Geophysics. Rock-soil geochemistry 100 x 100 grid. 1:2,000 geological mapping and 18,908 m in 92 RC drilling holes, including the infill 50 x 50 m grid at Berta Sur.	This report

In 1997 the area was optioned by Mantos Blancos (Guinez and Zamora, 1998). During September - December 1997, the area was geologically mapped and, geochemical and geophysical (IP) surveys completed; 42 RC drill holes were completed totaling 4,942 m, and some bulldozer trenches were also dug (Figure 6.1)

The project was deemed not to meet Mantos Blancos' criteria and it was returned to its owner.

In 2005 the properties were optioned by Texas T Minerals through its Chilean subsidiary Faro S.A., then later was transferred to Grandcru Resources, which initiated exploration works on October 2006 (Adkins, 2008). All previous work was verified and additional exploration carried out, including; geochemistry with new measurements of Cu and Mo contenttaken from trenches and pits, using a Niton portable XRF equipment; geophysics, consisting of ground magnetometry and radiometry; additional trenching; and finally 9 DDH holes were drilled for 3,311.40 m, with depths between 87 to 932 m.The objective of Grandcru's program was to demonstrate the presence of a porphyry system beneaththe breccia and/or other non-outcropping breccia bodies

Figure 6.1 and Table 6.1). Results were not considered sufficiently attractive to justify the option payments, and the property was returned to its owner.

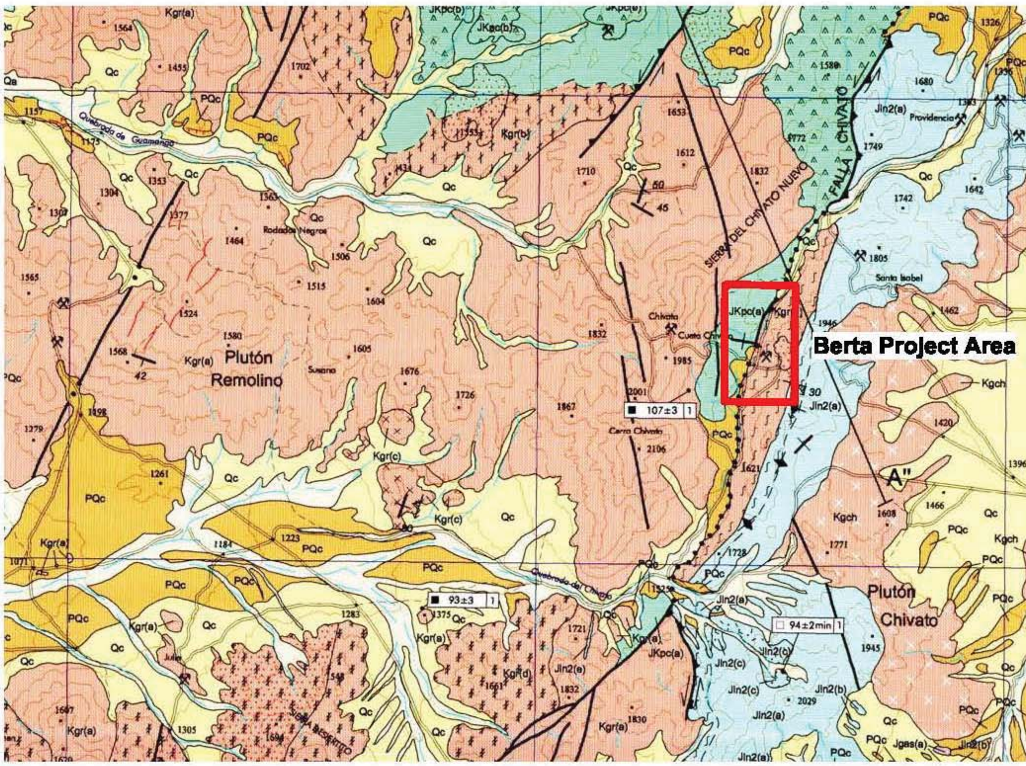
In June 2011 the properties were optioned by Coro through its Chilean subsidiary MCC. Since then, the potential for Cu (Mo) porphyry style mineralization in the area has been explored via the generation of a topographic base through restitution and ortho-rectification of images with topographical control; geological mapping of outcrops and trenches at 1:2000 scale; systematic rock and soil geochemistry; geophysical studies (IP); and the three successive campaigns of RC drilling totaling 92 drill holes for 18,910 meters. The first two phases of drilling (24 holes: 4,360 m and 32 holes: 10,520 m) were aimed at the exploration of the porphyry system and the third (36 holes: 4,028 m) to provide sufficient information for a resource estimate at Berta. Collection of samples from drill core and trenches for metallurgical test work was also undertaken. The summary of work carried out and results obtained by MCC are shown in Table 6.1 and described in greater detail in chapters 9.0 to 11.0 of this report.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

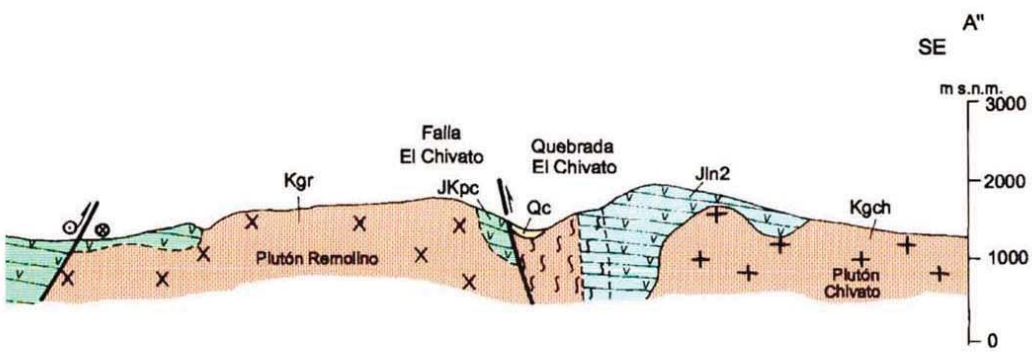
7.1 REGIONAL SETTING

The geological framework of the project area is taken from the SERNAGEOMIN Quebrada Salitrosa 1:100,000 scale mapsheet (Lara and Godoy, 1998). In addition, there are various studies of the region related to intrusive activity and structure, including some radiometric data (Grocott, et al., 1994; Dallmayer, et al., 1996; Grocott and Taylor, 2002), (Figure 7.1).

Figure 7.1: Berta Project geological setting [A) geologic map; B) Section] (North to the Top)



A)



B)

Source: Lara and Godoy (1998)

The Chivato fault (ZFCH; Godoy et al., 1997; Lara and Godoy, 1998; Grocott and Taylor, 2002) is the main geological feature. It is a fault zone striking N-NE sub-parallel to the Atacama fault system, which is located some 30 km to the W(Lara and Godoy, 1998). The ZFCH places Cretaceous volcanics in contact with a ~100 Ma Cretaceous granodiorite and with Jurassic volcanics. The fault zone is recognizable in the field as a 50m wide band of foliated intrusive and volcanic rocks, however, the primary fault zone itself exhibits fault breccia and provides evidence of East verging reverse movement (Figure 7.1).

The oldest rocks correspond to the Jurassic La Negra volcanic formation, composed of a massive sequence of andesitic lava with interbedded volcanic breccias, volcarenites as well as occasional calcareous beds. In this unit, rhyolite to dacitic and andesitic domes are also recognized. These rocks are discordantly overlain by andesitic lavas, tuffs and sandstones interbedded with calcareous sedimentary rocks,and andesitic-dacitic domes and dykes of the Punta de Cobre formation of lower Cretaceous age (Lara and Godoy, 1998). The age of both units and their regional correlation was established based on stratigraphic relationships, some age dating, fossil contents and minimum ages derived from intrusions with radiometric data (Figure 7.1).

The Jurassic volcanics to the E and SE of the ZFCH are intruded by syenogranites, reddish monzogranite; granodiorite and leucocratic granite of the Agua del Sol pluton and its equivalent Chivato pluton to the N, made up of pyroxene quartz monzodiorite aged ~150 Ma. To the W of the ZFCH the Remolinos pluton occurs (Figure 7.1) formed of amphibole tonalites; microdiorite; biotite-amphibole granodiorite; hornblende quartz diorite aged 110-90 Ma (Lara and Godoy, 1998).

The ZFCH crosscuts a band of mylonite developed in La Negra formation andesite and tonalites of the Remolinos pluton. The foliation is oriented NNW, sub vertical or strongly inclined to E (Godoy et al., 1997; Lara and Godoy, 1998). The kinematic indicators show a left lateral displacement both along strike and down dip. The contact relationships indicate the development of this deformation zone on the margins of the Remolino pluton during its emplacement. The ZFCH reverse fault is later and is the expression of crustal shortening which uplifts the eastern block of Jurassic rocks over the Cretaceous volcanics. It is probably derived from the reactivation of a basin edge fault during the upper Cretaceous (Figure 7.1).

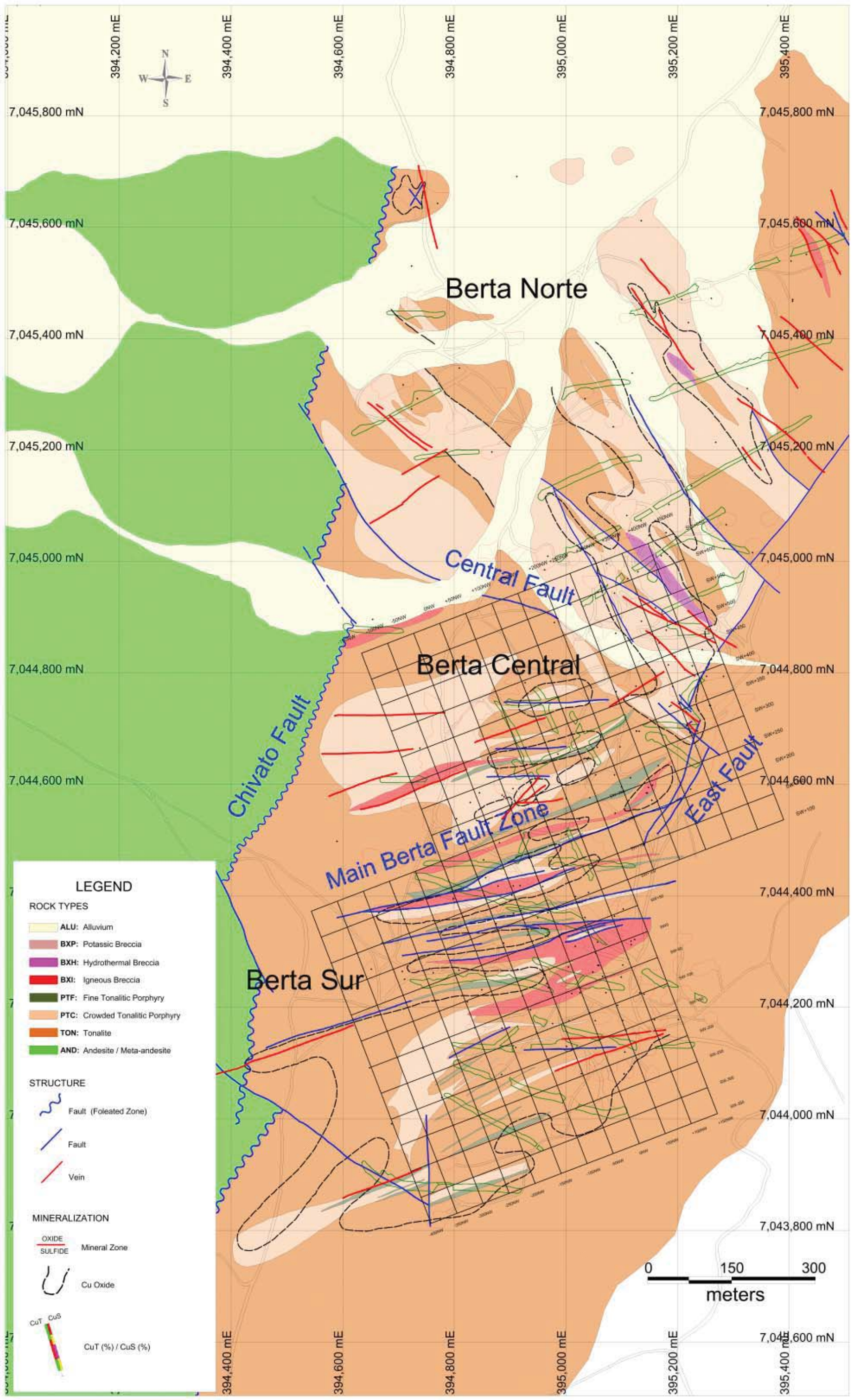
Iron Oxide Copper Gold (IOCG) type Fe, Cu-Fe-Au and Cu-Ag mineralization; Au veins; and Cu-Au porphyries are present in the Berta district. Major IOCG type deposits, which represent the extremes of the spectrum with Fe (Bella Ester, Rodados Negros) and Fe-Cu ± Au; (Manto Verde) are located westward in the Atacama fault system, while Cu-Au porphyries are located eastwards in the Inca de Oro district and also related to Au vein systems. The ZFCH particularly controls the location of several districts, apart from Berta, with Au veins and IOCG type bodies. In general, mineralization events occurred in the Cretaceous period between 120-100 million years (Ma) and approximately 90 Ma.

7.2 LOCAL GEOLOGY

At Berta the evidence for an alteration-mineralization system with Cu and Mo extends over an area of approximately 2.3 km by 1 km, oriented NNE. The elongation of the area is clearly controlled by the ZFCH, limiting the mineralization to the W. Notable differences in the geology and alteration-mineralization styles permit the separation of the area into three sectors: Berta Norte, Berta Central and Berta Sur (Figure 7.2 and Figure 7.3) presents Berta Project Panoramic view to the E, showing the three main mineralized sectors: Berta Norte, Central and Sur. Note that most of the old mining workings and dumps are in the Berta Central area.

Figure 7.2: Berta Project Panoramic view (to the E)

Figure 7.3: Berta Project Geology, simplified from the 1:2,000 scale map



Wall rocks comprise tonalite (TON) of medium-coarse equigranular texture, intruded by at least two varieties of porphyry with similar composition: namely, "Crowded" porphyry (PTC) and "Fine" porphyry (TFP). The first is volumetrically more abundant, cuts the tonalite showing porphyritic to equigranular textural variations, while the Fine type is younger. Igneous breccia (BXI), with various types of intrusive fragments, semi-rounded in a porphyritic matrix, and hydrothermal breccia (BXH), with angular monomictic clasts, open spaces and sulfide cements, cut the tonalite and Crowded Porphyry, but seem to pre-date the Fine Porphyry (Figure 7.3) A NNE elongated belt of tonalite about 1 to 1.5 km wide, is bounded by foliated volcanic rocks, Cretaceous to the W and Jurassic to the E. However, these volcanic rocks do not host significant Cu mineralization, except occasional narrow Au veins. Previous geological maps (Outokumpu, 1994; Guiñez and Zamora, 1997) did not recognized rocks with porphyritic textures and in general, only two belts were distinguished; "Fine textured Granodiorite" to the E and "Coarse textured Granodiorite” to the W. MCCmapping has distinguished both at surface and in drilling the porphyry varieties described above and the contact relationship between them, and with the tonalite wall rock.

The most relevant structure corresponds to ZFCH, which can be traced NNE along the western boundary of the area, where it displaces foliated intrusive and volcanic rocks in a belt approx. 50 m wide (Figure 7.3). A zone of foliated volcanic rocks of 20 to 60 m wide is also mappable along the E contact of the tonalite body with the Jurassic volcanic rocks. NW oriented faults displace the ZFCH as well as the belt of foliated rocks to the east.

A D type vein system, with sulfide filling and a sericitic halo and a predominant NW strike is recognized in Berta Norte. This can be observed at surface in several trenches, with dominant red limonite leached filling, and showing some fault planes parallel to the veins. In the northern part of Berta Central, some of these veins have been determined to have an E-W strike. The breccia bodies also exhibit control by faults varying from E-W in a large part of the Berta Central area to E-NE in Berta Sur. As with the D type veins, these structures are pre-mineral.

The development of K-feldspar – biotite ± magnetite ± sericite is the most common alteration at Berta. For descriptive purposes this is named "background potassic alteration". Its intensity increases with further development of K-feldspar as Igneous breccia cement and as a strong replacement of the Crowded porphyry and tonalite surrounding the breccias. The sericite is preferentially developed in D type veins environment and shows greater development in the Berta Central and Norte areas. Muscovite development is found in some breccia bodies, especially at depth and in general in breccias located towards the western boundaries. Chlorite and variable sericite are best developed in porphyries and breccias, and in the best mineralized areas, the alteration contains "green grey sericite" and is characterized by the absence of magnetite, explaining why magnetic lows coincide with the mineralization. Propylitic halos with abundant chlorite and pyrite are better developed in the northern area (Figure 7.4). Within the marginal foliated rocks, especially in the west side along the ZFCH, the rocks are strongly replaced by biotite-magnetite, with some albite and actinolite. These minerals also occur as variations of background potassic alteration around the breccias in Berta Sur.

Figure 7.4: Typical propylitic type altered Tonalite and Crowded Tonalite Porphyry, crosscut by jarosite replaced, sub-parallel,pyrite veinlets.

The primary mineralization consists of chalcopyrite with minor variable content of bornite. There is abundant molybdenite in some sectors but with no obvious relationship to Cu sulfides. Mineralization preferentially occurs as breccia filling and cement, to a lesser extent in veins and occasionally in veinlets. Pyrite is very poorly developed in areas of best mineralization, with greater occurrence in the northern part of Berta Central and especially in Berta Norte, where it constitutes the main filling of D type veins. Along the ZFCH, chalcopyrite occurs associated with magnetite mineralization. There is an ore-alteration zonation from N to S, with a propylitic border and development of veins and breccias containing pyrite ≥ chalcopyrite (molybdenite) and halos of pervasive replacement of sericite in the north to a domain of background potassic alteration and mineralization in breccias surrounded by a crackled zone, with chalcopyrite (molybdenite, less bornite) >> pyrite, alteration grading outwards to albite-actinolite in the south. The western boundary is dominated by breccias with muscovite containing only rare Cu mineralization and biotite-magnetite zones with some chalcopyrite that can be traced along the ZFCH. This zoning is also related to a greater abundance of porphyritic rocks toward the central and southern areas and to changes in style and orientation of structures from NW to E-W and, finally, E-NE in Berta Sur (Figure 7.3)

The distribution of limonite at surface shows a direct relationship with alteration as well as with relative abundance of sulfide: yellow to yellow-reddish color predominates in the northern part related to the greater development of D type veins and sericitic alteration, while goethite and scarce jarosite make up the leach cap in the central and southern areas (Figure 7.5). In situ leaching and oxidation of the sulfides has produced a zone of copper oxides of variable thickness ranging from 30 to 120 m, generated in an environment of scarce pyrite and in poorly reactiverock. It is composed of simple green Cu oxides ores, with predominant chrysocolla, and black oxide (mixtures of wad type?), very low clay content, and limonite and predominant goethite. Only in some breccia bodies, mainly those located along the eastern boundary, is there limited development of supergene enrichment with chalcocite thicknesses of 2 to 10 m, invariably oxidized to a combination of hematite, “almagre” and cuprite.

Figure 7.5: Berta Norte sector looking East-Southeast

Figure 7.5 shows characteristic ridges controlled by prominent, NW trending "D" type veins. Note the sharp contact between foliated Jurassic volcanics in black and the cretaceous Tonalite in more light grey to white colors. Drill rig for scale.

7.3 BERTA SUR GEOLOGY

This section provides details of the geology, mineralization and alteration of Berta Sur. This sector comprises an area of 600 x 450 m evaluated according to a grid aligned 340°, perpendicular to the trend of mapped structures and after determining the orientation of mineralized bodies to be 060°. The Cu oxide mineralization is exposed on a 15 m high hill withgentle slopes, being flanked to the N and S by E-W and SW oriented creeks. This mineralization has not been mined and its exposure has been aided by trenches dug by Outokumpu, Mantos Blancos and Grandcru. For the resource evaluation of the sector, Coro has completed geological mapping of trenches and outcrops; rock and soil geochemistry; and three campaigns RC drilling for 66 RC holes totaling 11,622 m. The surface map of rocks and structures is shown in Figure 7.6 and a cross-section with rock types and mineralization is shown in Figure 7.7

Figure 7.6: Berta Sur Geology

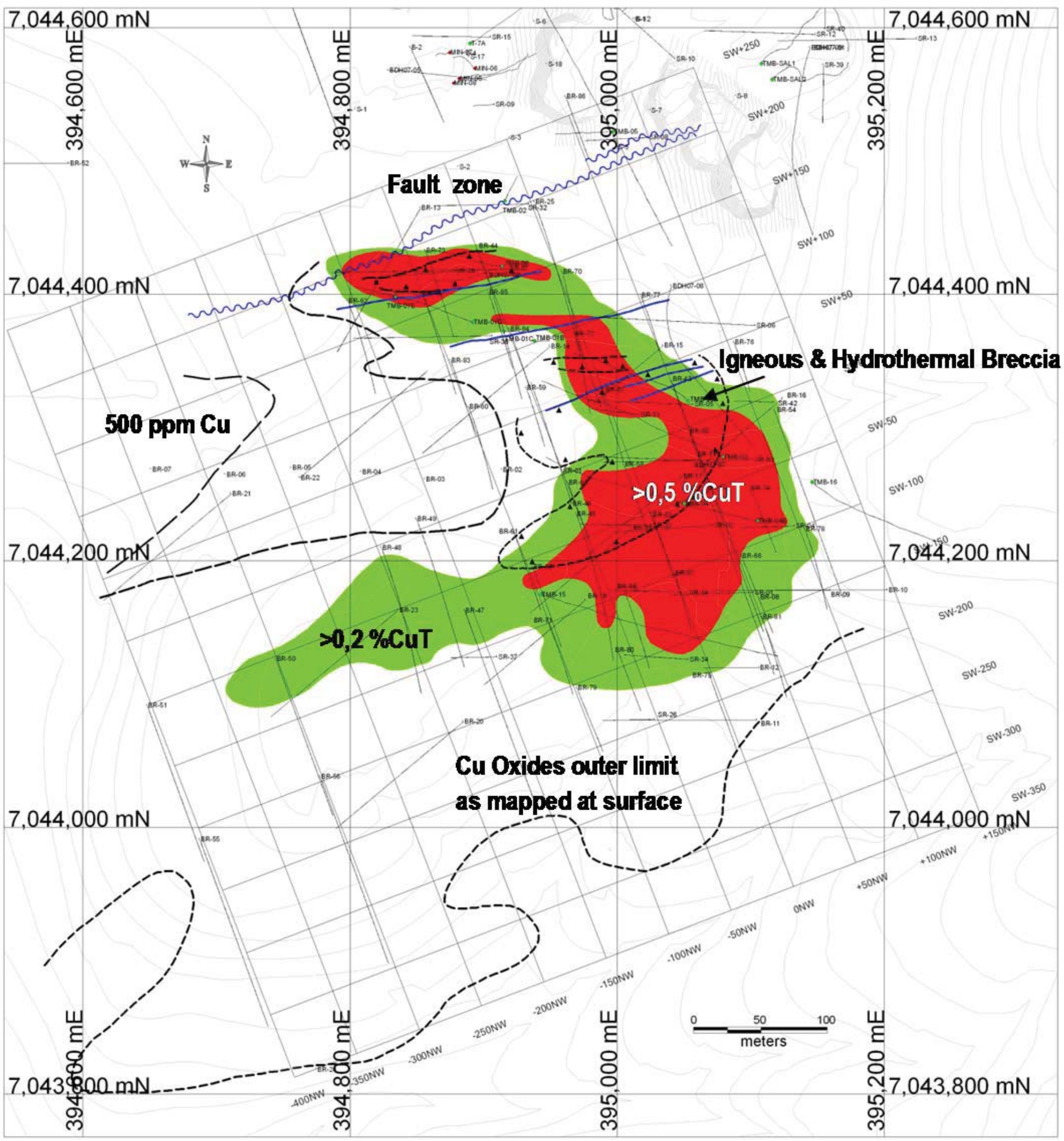
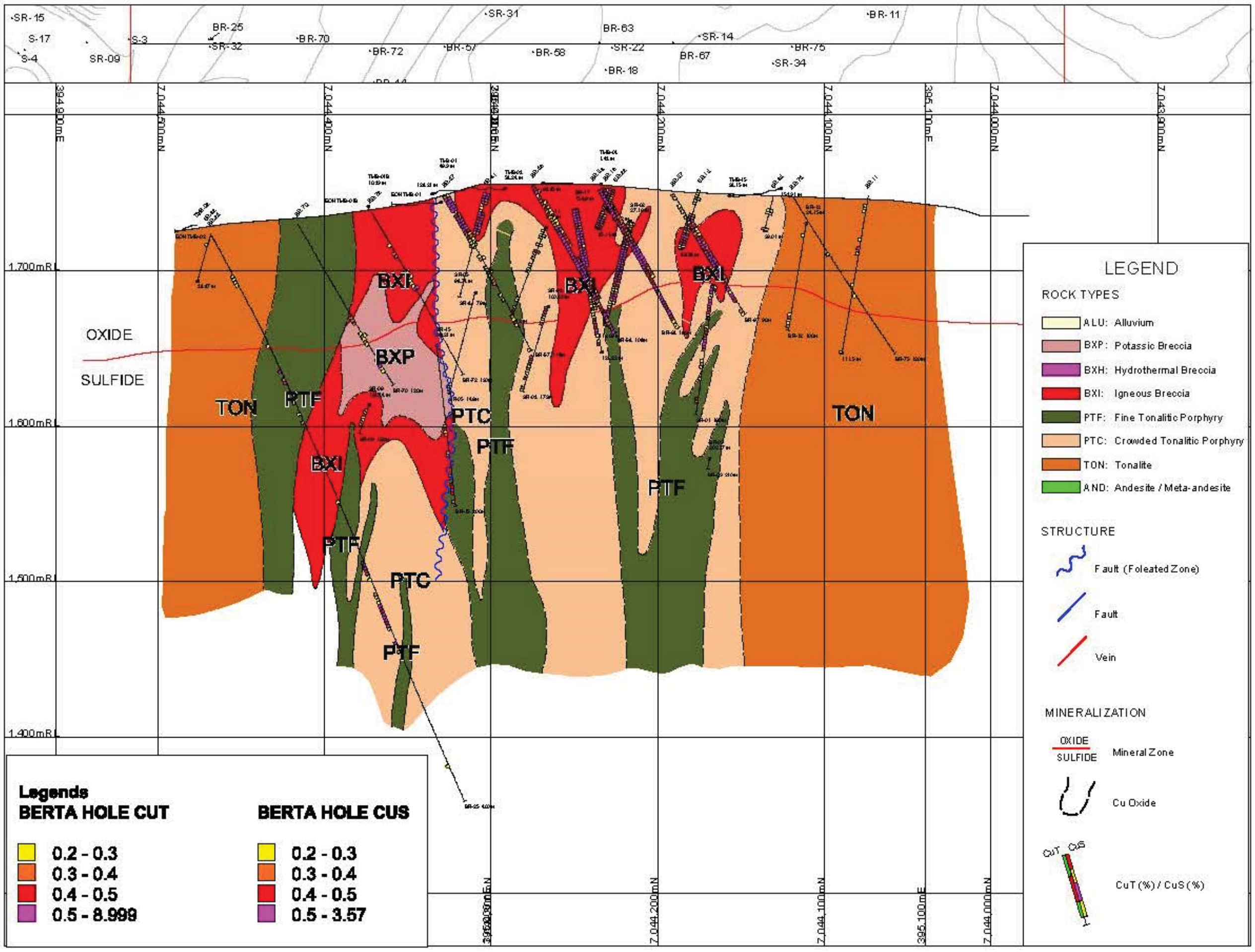


Figure 7.7: Berta Sur Typical Cross Section



Berta Sur: typical cross section displaying main rock types, structures and mineralization zones

7.3.1 LITHOLOGY

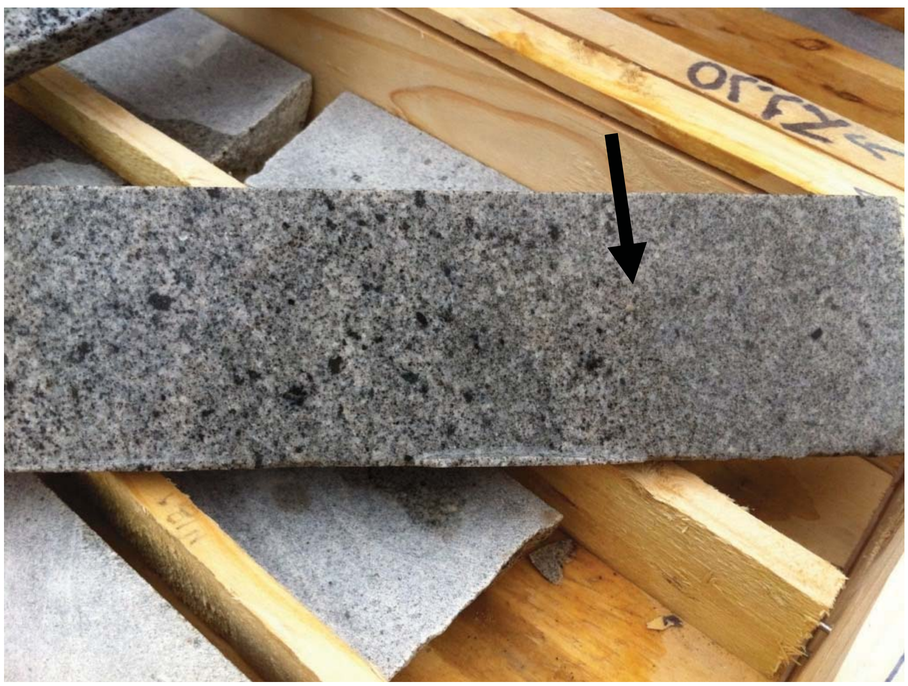
Berta Sur is formed ofa body of Cu sulfides and oxides, hosted by breccias and wallrock composed of tonalite, Crowded porphyry, Fine porphyry, and some andesitic dykes (Figure 7.6). The tonalite (TON) (Figure 7.8) is the oldest rock unit and is exposed at the east and south side of the sector. It has equigranular monzogranite textures varying from coarse to fine grain, also showing slightly porphyritic textures; the color of outcrops is light gray stained with limonite of brownish tones. It is composed of plagioclase, hornblende, biotite plus some K-feldspar and less quartz. Normally the hornblende is replaced by secondary biotite-magnetite and also by some actinolite and chlorite aggregates. The plagioclase shows varying degrees of albitization. In outcrops and in drilling it is possible to observe tonalite intruded by both Crowded and Fine Porphyry.

Figure 7.8: Tonalite (TON) as observed in core samples

Note biotite replaced mafics anda subtle foliation. Limonite coatings are jarosite and lesser goethite. Some disseminated chalcopyrite and pyrite.

The Crowded porphyry (PTC) (Figure 7.9) is of tonalitic composition and is composed of abundant phenocrysts (> 70-80%) of plagioclase > K-feldspar, plus biotite, hornblende and some quartz eyes, the color is gray to pinkish-gray. Some phenocrystals larger than the average (2 - 3 mm) correspond to biotite and plagioclase. The scarce matrix contains plagioclase, K-feldspar, interstitial quartz and some biotite. The PTC together with the breccia occupies the greater part of the mineralized area in Berta Sur (Figure 7.6), shows a clear intrusive relationship with TON (Figure 7.10) and is cut by Fine porphyry dykes and some andesitic dykes observed in drilling.

Figure 7.9: Tonalitic Crowded Porphyry (PTC)



Note abundant feldspar, hornblende and biotite phenocrysts. Contact with intruding Fine Tonalite Porphyry to the right (see arrow)

Figure 7.10: PTC intruding TON

The Fine porphyry (PTF) has a tonalitic composition and is characterized by a fine population of ~1 mm plagioclase phenocrystals, some feldspar, hornblende and biotite, normally very little altered, in a fine feldspar matrix with minor quartz (Figure 7.11). Large “books" of biotite ranging from 3 to 5 mm dominate the grey to dark grey rock texture. PTF dykes crosscut the breccia mineralization at Berta Central, related in some cases with fault zones and D type veins oriented E-W and E-NE (Figure 7.6). In the mineralized body of Berta Sur, the PTF is mainly distributed toward the southeast half, like a trunk with branches forming dykes extending NE towards the mineralized breccia and PTC sector, and consequently forming the limit of the mineralized body toward the SE. PTF dykes associated with major faults that displace the mineralization toward the north have been mapped explaining why several vertical holes drilled along an E-NE line intersected only these dykes that interrupt the continuity of the body toward the north. The PTF is post-mineral, although in some holes it is seen to be related to late stage, pyrite dominant sulphide introduction.

Figure 7.11: Fine Tonalitic Porphyry (PTF)

Note the prominent biotite “books” and the fine feldspar-biotite phenocrysts

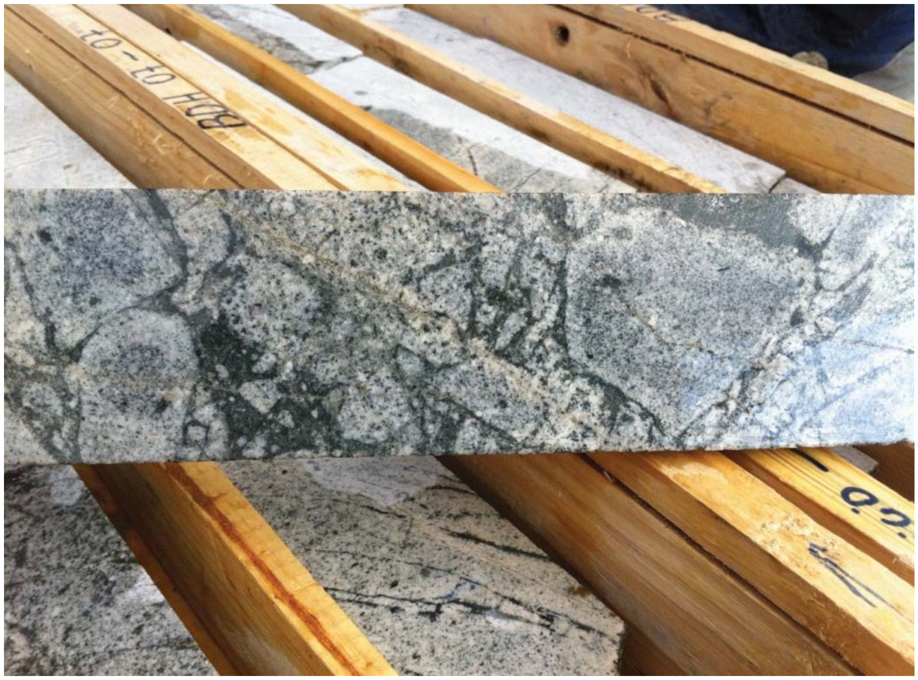
Andesitic dykes (PAN), a black rock of fine texture and composed of plagioclase with some biotite and magnetite, has been observed in some drill holes. It occurs in fault zones as dykelets of just a few cm of width, and is post-mineral.

The breccias contain the greatest amount of high grade Cu mineralization in Berta Sur. They form an ENE elongated body measuring approximately 200 x 100 m, hosted by TON and PTC and cut by PTF dykes to the SE (Figure 7.6). It comprises Igneous breccias (BXI, Figure 7.12), characterized by sub-rounded clasts of TON and PTC with intense alteration of K-feldspar > sericite-biotite in a matrix of igneous material with quartz, feldspar, and some biotite. The Hydrothermal breccias (BXH,Figure 7.13) contain angular to sub-rounded clasts of PTC, with a serictic matrix and in some cases of muscovite and K-feldspar with Cu sulfides, and variable occurrence of minor pyrite and molybdenite. It is also possible to observe "grey-green sericite" and chlorite as BXI matrix. Even though the contact with the PTC or TON wallrock tends to be faulted, in some cases there is gradation from sub-rounded breccia fragments to a puzzle type and/or crackled wallrock. Contact relationships and textural variations show that the BXI in Berta Sur developed in contact areas between PTC and the TON, and that the BXH wasoriginally BXI with a greater degree of alteration and mineralization. The PTF dykes are post breccia emplacement.

Figure 7.12: Typical Igneous Breccia (BXI)

Note the sub-rounded fragments of Tonalitic Crowded Porphyry (PTC), all affected by potassic alteration

Figure 7.13: Hydrothermal Breccia (BXH)



Note that therotated fragments of background potassic alteredPTC are superimposed by K-Feldspar-Sericite alteration invading fragments from the matrix, this contains green-grey sericite (fine chlorite-biotite-sericite-feldspar)

7.3.2 STRUCTURE

The likely extension of the foliated rocks that control the eastern contact of the TON with Jurassic andesites in Berta Sur is interpreted from imagery and 1: 2 000 scale mapping. However, detailed mapping of trenches and outcrops did not accurately identify their location, implying that alteration-mineralization events originated later, and have masked their presence. The intersection of the NE oriented zone of foliated rocks with NW oriented faults is interpreted to control the location of the mineralized breccia bodies of Berta Sur. In this area, the trenches expose some bodies of tonalite strongly replaced by biotite and magnetite, in parts with breccia textures (Figure 7.6).

The breccia body that hosts high-grade mineralization in Berta Sur is controlled by vertical ENE oriented faults which formthe main system of pre-mineral structures. In drill holes, dykes of PAN are related to fault zones but they have not been observed at surface.

A fault system which can be traced over some 150 m of width, oriented ENE, limits the breccias and the mineralized body to the north and causing the dismemberment of the breccia body into at least two sections, limited by faults. The post mineral PTF dykes are related to these faults. The fault zones have several cm of gouge, with crushed rock, altered to sericite. To the S and SE the breccia body shows intrusion contacts with PTC and TON oriented ENE. This orientation is also the dominant one for PTF and PTC dykes and some D-type veins hosted by the TON, defining the low-grade halo surrounding the main mineralizedbody to the SW.

Towards the southern part of the body some faults of NW and N-S strike have been mapped, which cause a slight displacement of porphyry dykes and D type veins.

7.3.3 ALTERATION

TON, PTC and breccia are affected by background potassic alteration. This is characterized by stable biotite and replacement of amphiboles by secondary biotite. The K-feldspar is also stable and magnetite is often associated. Minor replacement of ferromagnesian by chlorite and of feldspar by sericite, also characterizes this alteration. There is no obvious link with the copper mineralization, however as evidenced by potassically altered clasts in the Igneous and Hydrothermal breccias confirming they postdate the background potassic alteration. Breccias with coarse grained magnetite and biotite are exposed to the E, outside the main area of mineralization and close to the contact with the foliated rocks.

Evidence of Ca-Na alteration predating or marginal to the background potassic alteration is demonstrated by development of albite on the margins of feldspars and actinolite. The occurrence of epidote is very restricted. The best development of Ca-Na alteration is toward the margins, in the foliated volcanics and intrusives which have been intensely replaced by biotite-magnetite, actinolite and albite.

The main phase of breccia formation, alteration and mineralization is related to potassic alteration characterized by strong replacements of K-Feldspar, biotite and to the development of green sericite as cement of hydrothermal breccias. It is also characterized by the absence of magnetite, which is reflected in relative magnetic lows in the ground mag RTP. This alteration is related to Cu sulfide mineralization and to Mo in some cases. Incipient development of greisen with coarse muscovite is present in the deeper parts of the breccia bodies, and some of the Mo mineralization is related to this phase.

Phyllic alteration with replacement of the rock by sericite, accompanied by pyrite, is observed in the halos of veins D type and is common in the fault zone limiting the Berta Sur mineralization to the north. Although this alteration is moderately pervasive, the width of the halos is small in relation to the faults and veins, grading rapidly outwards to the original texture of the rock. There is no relationship between the Cu mineralization and this type of alteration; however, the relatively higher pyrite content produces a thicker leached zone in areas affected by it.

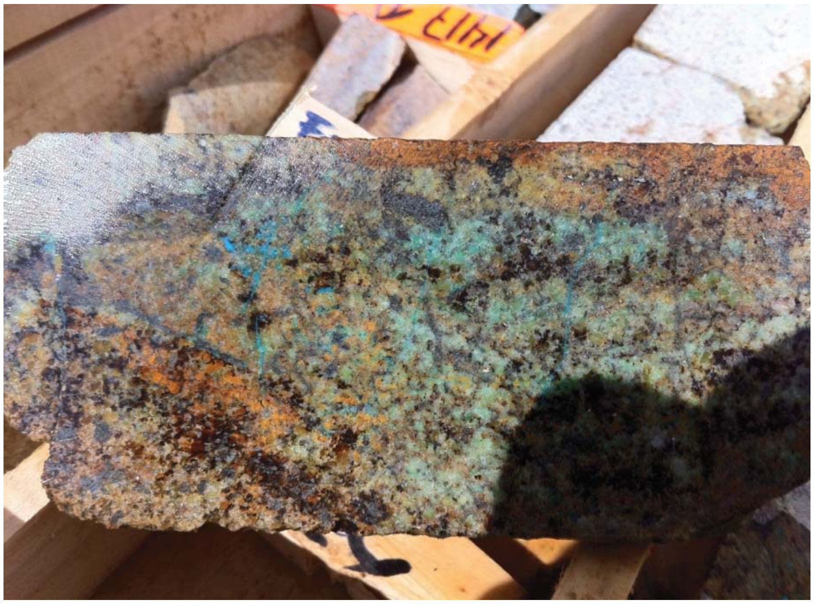
7.3.4 MINERALIZATION

The primary mineralization consists of Cu sulfide, dominated by chalcopyrite that occurs as breccia cement and disseminations in the breccias and their contact zones with PTC and TON. Occasionally bornite has been observed in the hypogene zones of Berta Sur breccias. There is gradation in the chalcopyrite percentage from higher grade centers (> 0.5% Cu) controlled by zones of greater permeability and potassic alteration in the breccia bodies, toward areas of lower Cu grades on the margins. This variation is truncated in the N by faults cutting the body, while in the S and SW part, there is an abrupt decrease to grades approximating 0.2% Cu, due to reduced dissemination of chalcopyrite. The mineralization is controlled by the distribution of the breccias and by the NE oriented pre-mineral structures. There is a close relationship between potassic alteration, with K-feldspar and grey green sericite, and the presence of chalcopyrite. The form of the sulfide mineralization is irregular however, such that good grade Cu oxides overly equivalent grade sulfides in some areas, in others this relationship does not exist and in contrast, an abrupt decrease of good grade oxides to underlying low grade sulfides is suggestive of an inverted cone-shape or root to the mineralization. These relationships, coupled with variable shape of the breccia bodies and variations in alteration with depth permits an interpretation of the deeper parts of the system (see Figure 7.7, Figure 7.14 and Figure 7.15)

Figure 7.14: PTC affected by pervasive potassic background alteration

Figure 7.14 shows PTC affected by pervasive potassic background alteration with biotite and K-Feldspar superimposed in the most brecciated zones by strong K-feldspar rich alteration related to chalcopyrite and green-grey sericite

Figure 7.15: Green oxide mineralization



Example of green oxide mineralization, mostly chrysocolla and black oxide (wad?) in tonalite host rock

Molybdenite mineralization is related to the introduction of muscovite, and occurs in veins of coarse quartz/Mo, and K-feldspar/ Mo. This mineralization does not have a direct link with the occurrence of chalcopyrite and as a result it is not possible to correlate high Cu grades with Mo. The relationship with at least three events shows that Mo mineralization is pre-, syn-, and post-, the main episode of copper mineralization.

In Berta Sur the Cu oxide zone extends from30 to 100 m in depth. The top of sulfides (TS) generally follows the current topography, except in areas of faulting and in areas of greater pyrite abundance related to PTF dykes (Figure 7.7). In the oxide zone, the main copper mineral is black oxides (copper wad?), identified as such in mineralogical studies during metallurgical testing, and chrysocolla. In logging these minerals are identified as black oxides and green oxides and occur as disseminations, fracture filling and in breccia matrix. The gangue minerals are primarily biotite, sericite and feldspar, with minor clays and carbonates as confirmed by the leach test work. Goethite is the most common iron oxide. The oxide body hosting the bulk of the Berta

Sur averages 60 m in thickness and occupies an area of approximately 200 x 140 m elongated in E-NE direction.

No mineralogical zonation within the oxide deposit was noted, at least with the limitations of logging RC cuttings. Both green and black oxides occur in similar proportions in the mineralization, both vertically and horizontally as confirmed by mapping and mineralogical studies. For resource estimation purposes a natural zonation was established based on the intensity of mineralization, without distinguishing separate mineral species, with the outer limits of the mineralization being determined from logging of cuttings. This lack of mineralogical variation in the oxides is a true reflection of the simple conditions of primary chalcopyrite mineralization and potassic alteration of the breccias, PTC and TON. The almost complete absence of pyrite and the unreactive nature of the wall rocks have resulted in an oxide zone with minimal mineralogical complications for metallurgy.

7.3.5 GEOCHEMISTRY

The interpreted maximum extension of copper oxides at surface is generally coincident with plus 500 ppm values in rocks and soils,outlining an area of 500 x 250 m, oriented NE. A second anomaly is related to the linear breccias confined by fault zone immediately to the N. There are no significant Mo values related to the principal copper anomaly.

The Berta Sur body is exposed at surface. Trenches, small pitsand drill holes confirm that the area of > 0.5 %Cu extends over 200 m x 140 m surface at elongated ENE, within a larger zone of > 0.2 % Cu measuring 400 x 200 m. The higher grade zone of mineralization coincides with the breccia bodies, and their contacts with PTC and TON and the more intense potassic alteration without magnetite. The lower grade halo corresponds to PTC and TON with background potassic alteration and PTF dykes.

There are no anomalous Au values in Berta Sur. The geochemical data for portable XRF collected by Grandcru showed a consistent Mo anomaly coincident with the Cu mineralization, however, there is no verification of this from the AAS results obtained by Coro.

7.4 BERTA CENTRAL GEOLOGY

This area of the Project is located immediately north of the previously described Berta Sur sector. It was evaluated via a 450 x 500m grid corresponding to the extension of the Berta Sur grid. Although the breccia bodies that make up the majority of the Berta Central deposits are roughly circular in plan view, they are aligned along the same 340^{o} structural system mapped in Berta SurThese structures are faults, D type veins and post mineral late porphyry dykes.The breccia bodies containing copper mineralization grading greater than 1%CuT are surrounded by halos of lower grade mineralization which together form areas of more than 100 x 100m whose continuity has been demonstrated by drilling, trenching and inspection of old workings.

The largest old open pit workings exploited the Carmen, Gemela and Salvadora bodies.In the first two, a more than 100m long decline provided access to underground stopes at a depth of up to 60m below surface At Salvadora, a 20m deep pit selectively exploited high grade zones some of which were further mined underground via pits and adits to a depth of up to 33m from surface. Only shallow workings of 10-20m depth occur at Berta II and Chico, while the Trinchera and Nueva bodies have not been mined, but were defined by drilling and trenching

In order to evaluate the Berta Central mineralized bodies, Coro has mapped the outcrops, trenches and old workings and compiled all the previous information from the Outokumpu, Mantos Blancos and Grandcru exploration. In particular, the geologic controls to mineralization have been established on the same basis of lithology-alteration-mineralization as defined for the adjacent Berta Sur deposit. The interpretation was completed on vertical sections oriented perpendicular to the principal structures, i.e. NW-SE, along the above mentioned grid. The re-logging of the Grandcru diamond drill core was particularly helpful in this regard. Figure 7.16 shows the location of the mineralized bodies, breccias and principal structures, while Figure 7.17 is a representative cross section showing structure, rock types and mineralized zones.

Figure 7.16: Summary geology and mineralized bodies defined at Berta Central

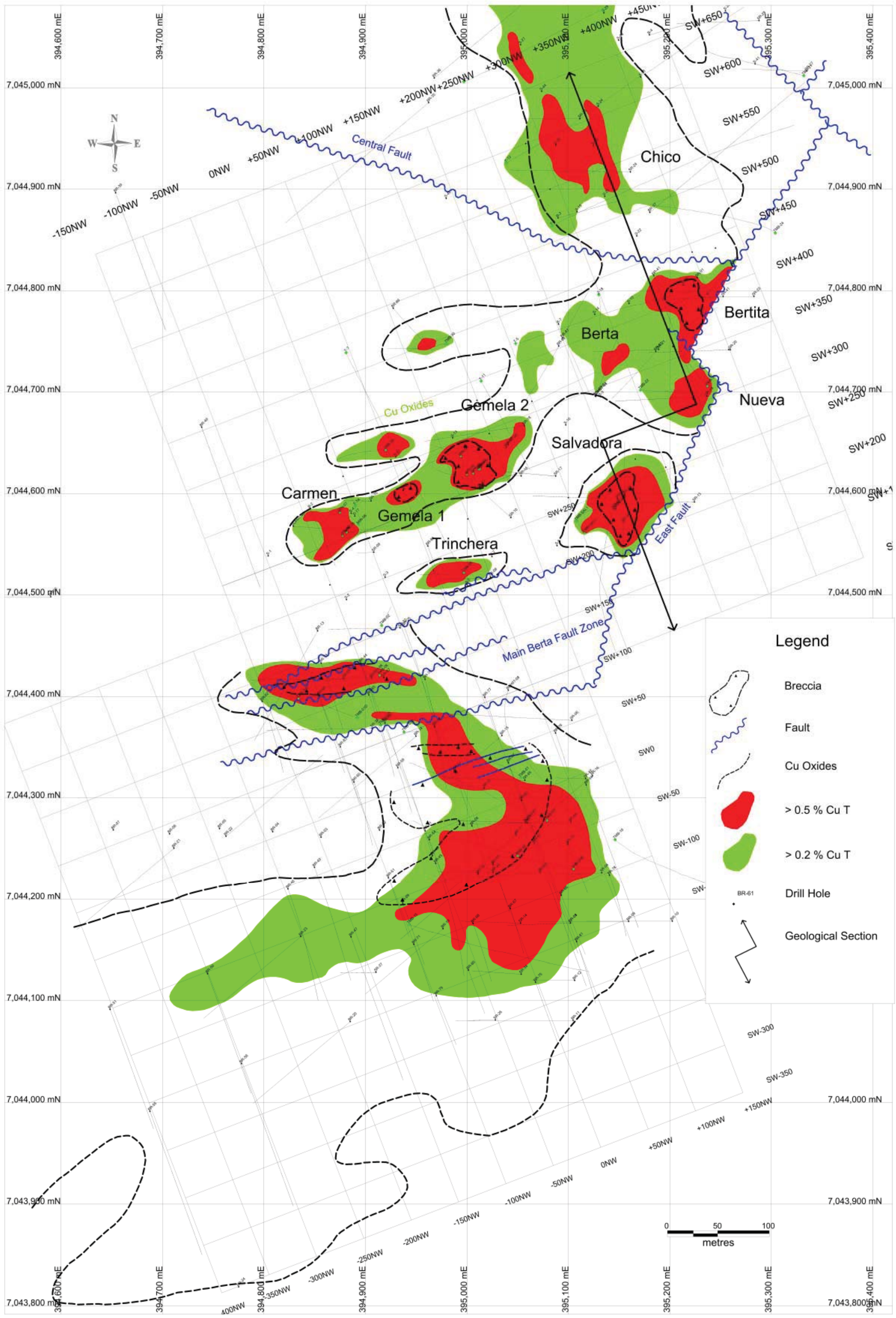
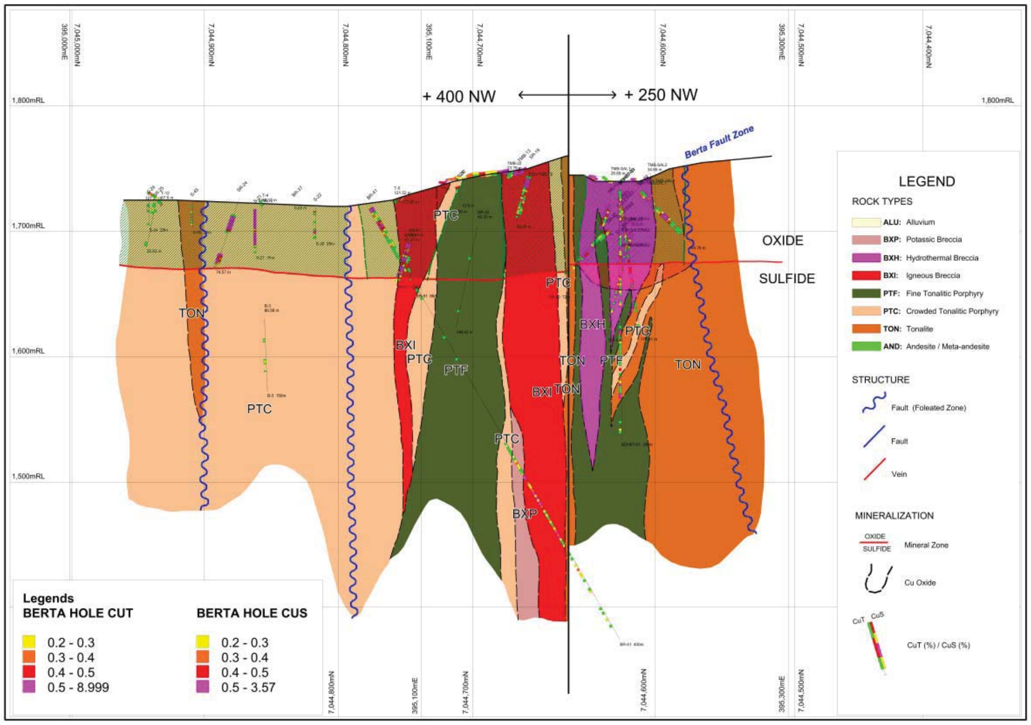


Figure 7.17: Section through Berta Central showing main lithologies, structure and mineralization (location at Figure 7.16)



7.4.1 LITHOLOGY

Unlike Berta Sur, Berta Central consists of several structurally controlled breccia bodies The breccia bodies are essentially cylindrical in form as defined in old workings and drilling in the oxide zone; there is much less information available for their depth continuation and hypogene mineralization. The breccias, more hydrothermal in nature than those at Berta Sur, are hosted by PTC and to a lesser degree by TON. Post mineral PTF dykes occur more frequently in the south of the área, between the Carmen-Gemela and Salvadora breccias. These dykes strike WSW-ENE and are observed to cross cut mineralized breccias generating a false idea of the termination of the breccias at depth in drilling. The mineralogical and textural characteristics of the various intrusives at Berta Central are identical to those described in section 7.3.1 for Berta Sur.

The hydrothermal breccias (BXH) host most of the copper mineralization at Berta Central. They comprise subrounded fragments of PTC and lesser TON cemented by limonite and copper oxides with quartz, biotite and sericite. In contrast to the Berta Sur breccias, they have greater sericite and hematite contents derived from the oxidation of chalcocite. The wall rock to the breccias is dominantly PTC which also forms most of the clasts. In some outcrops and drill core, the breccia matrix contains green sericite and K-feldspar which selectively replace the clasts and wall rocks especially in áreas of greater brecciation. While the form of the bodies is generally cylindrical, with diameters of 20-50m, the structural control which aligns the bodies results in corridors of breccia such as that between Carmen and Gemela which extends for some 250m. Internal complexity derived from sub horizontal branching of breccias away from the main structures occurs at various scales as shown in the photograph in Figure 7.18.

Figure 7.18: Close up of hydrothermal breccia in outcrop. Note the zones of subhorizontal brecciation

7.4.2 STRUCTURE

Three principal fault zones have been defined at Berta Central; East Fault, Main Berta Fault Zone ("MBFZ") and Central Fault. The first of these corresponds to the western limit of the zone of foliated rocks and mylonites which limit the Berta mineralization to the east and which controls the contact between the TON and the volcanics. Some of the trenches in the Berta II and Nueva sectors expose this fault which is evidenced by gouge filled fault planes. In the same sectors, left lateral displacements of 20-40m by NW oriented faults has been mapped, with some of these faults controlling D-type veining with sericitic halos. The continuation of the East Fault can be seen in the east wall of the Salvadora pit, and its further southward continuation is inferred by lack of outcrop succeeded by NNE oriented fault planes exposed in road cuts in the northern part of Berta Sur. In this area, the East Fault may also have been interrupted by the MBFZ

The MBFZ consists of various sub-parallel faults which have controlled the introduction of systems of D-type veins and PTF dykes.The southern part of the MBFZ is exposed in road cuts and trenches in the NW part of Berta Sur where it has a width of 100m and has been inferred over a distance of 350m, oriented WSW-ENE.This unmineralized fault zone is marked by conspicuous sericitic alteration and contains a few faulted slices of Berta Sur mineralization that have been caught up in it. One of these slices is well exposed and defined by drilling and forms part of the Berta Sur resource.The details of this fault as it limits Berta Sur has been described in section 7.3.2.

The Central Fault is interpreted as forming the northern limit of Berta Central and is similar to the MBFZ in that it is inferred from rare outcrops and trench exposures of sericitised and fractured rocks as well as from abrupt changes in orientation of other structures, rock types and mineralization styles. It is possible to trace it over some 500m from the west side of the project areawhere it shows left lateral movements of 100-150 m (Figure 7.3) This change corresponds to the limit between Berta Central, with its mineralized breccias interconnected by faults; and Berta Norte, where D-type veins predominate, as has been further described in section 7.2. Although the Central

Fault doesn't outcrop in the Berta Central area, it can be interpreted crossing the project areain a WNW-ESE direction displacing the Chivato Fault, and perhaps having a similar effect to the East Fault, although this has not yet been verified in the mapping to date (Figure 7.3) Dykes of PTF, oriented WSW-ENE and varying in width from metres to dozens of metres, are important structures at Berta Central. They are related to the MBFZ and extend to the Carmen-Gemela-Salvadora corridor, although their occurrence farther north is much less notable. In part, they interrupt the lateral and depth continuity of the mineralized breccias, as observed in the deeper drill holes completed by GrandcruHowever their high biotite content has acted as a neutralizerfor the lateral migration of copper in solution in the oxide zone, such that the low grade halo in the Carmen and part of the Salvadora bodies extends to the otherwise unmineralized PTF

7.4.3 ALTERATION

The principal alteration types at Berta Central are very similar to those described in section 7.3.3 for Berta Sur. Replacement of feldspars and mafic minerals in the PTC and TON by sericite is the dominant alteration related to mineralization of the breccias. The sericite is generally fine grained and associated with chlorite and biotite, and unlike in Berta Sur, there is direct relationship between sericitization, brecciation intensity and Cu & Cu-Mo mineralization in the hypogene sulphide zone.

The background potassic alteration commonly affects the PTC and TON and consists of biotite and K-feldspar. Its intensity is notably greater in the margins of the breccias and can be followed within the structural corridors that control their emplacement. Here, a phase of K-feldspar alteration is clearly observable in outcrop and drill core, and is characterised by intense replacement and invasión of both the wall rocks and breccia clasts, in some cases resulting in their total replacement. The presence of green sericite as part of the breccia matrix in areas of strong potassic alteration and sulphide mineralization confirms a relationship between mineralization and alteration.

7.4.4 MINERALIZATION

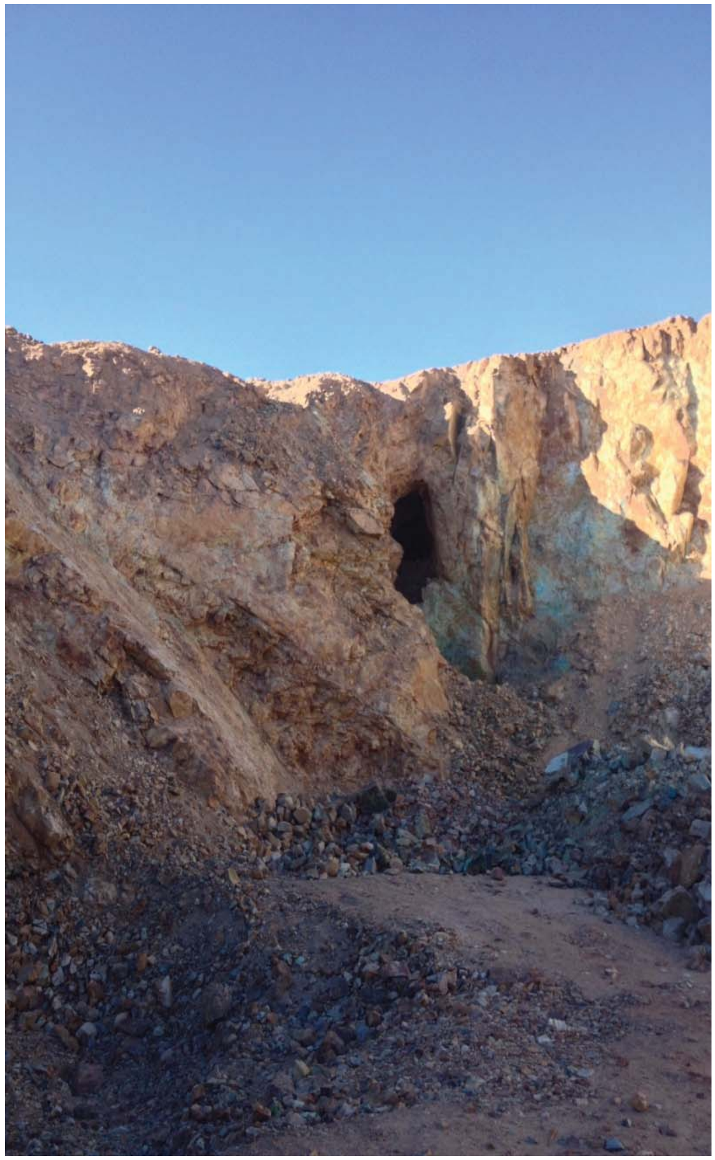
The copper mineralization at Berta Central is distributed both within and around the margins of the various hydrothermal-igneous breccia bodies. The bodies are approximately cylindrical with diameters of 30 – 50m, although they may be slightly ovoid with major axes oriented N-S to NE-SW. Cu-Mo mineralization has been shown by drilling to extend to depths of 400m; however most of the old workings and drill holes have been confined to the oxide zone which extends to 60 – 70m.While the breccias themselves commonly have grades in excess of 1%CuT, significant mineralization grading greater than 0.2%CuT forms a halo around and between the breccias. In these halos, the mineralization occurs in small bodies of breccia and crackling, in fractures, in PTF dykes and within fault zones. The previously mentioned fault controlled corridors have been recognised over 300m between Trinchera and Salvadora and over 250m between Carmen and Gemelas.

The known breccia bodies have been added to by two newly defined bodies and the characteristics of each are as follows:

- **Trinchera:** Hydrothermal breccia mineralized with copper oxides exposed at surface in trenches T-7 and TMB-05, as well as intersected in hole BR-86. It is interpreted as a NE oriented body measuring 80 x 20m, partially covered by dumps that conceal its lateral extent.
- **Salvadora:** Corresponds to one of the old mines previously exploited and consisting of an open pit and irregular underground workings (Figure 7.19). The breccia measures 50 x 30m, elongated in a N-S direction. It is limited to the east by the East Fault, and below 70m has a 20m thick zone of secondary enrichment. A portion of the mineralization exposed in the base of the pit comprises copper oxides and “almagre” derived from the oxidation of chalcocite
- **Carmen:** Measuring 30m in diameter, it forms part of the Carmen-Gemela corridor. It was exploited by underground workings and a glory hole to a depth of 70m. The breccia is not exposed at surface and it can be seen to be controlled by ENE oriented structures in underground workings. A good part of the mineralization is fractured controlled within an argilised and fractured PTF dyke

- **Gemela:** Defined by Outokumpu as Gemela 2, it was exploited together with Carmen. It measures 50m in diameter and has good oxide mineralization in the bottom of the pit at 70m depth. It is a hydrothermal breccia unaffected by late PTF dykes. La brecha The Gemela 1 breccia is exposed at surface in small outcrops among the dumps and igneous-hydrothermal textures. Its true depth extension is unknown, since the underground workings that cut it were developed in post mineral PTF dykes. In addition to Gemela 1, a small outcrop of mineralized breccia occurs 30m to the north where it is exposed in trenches T-1 and TMB-09, and which may represent additional lateral mineralization along the Carmen-Gemela corridor.
- **Berta:** A cluster of bodies made up of Berta, Bertita and Nueva extending over 250 x 150m. Berta has been mined on a small scale via a decline and a glory ole (Figure 7.20). It measures some 20m in diameter and comprises an irregular igneous-hydrothermal breccia hosted by strongly potassically altered PTC
- **Bertita:** A hydrothermal breccia located 40m NE of Berta , it has not been mined, notwithstanding the presence of attractive copper oxide mineralization exposed at surface and intersected by Outokumpu, Mantos Blancos and Coro drill holes
- **Nueva:** A 40m diameter breccia body partially exposed in trench TMB-13 and intersected at depth by hole BR-41. Two of the older holes also partially intersected the mineralization, but were drilled along post mineral PTF dykes or faults. The resource of this body and Bertita may be currently underestimated due to the lack of systematic information.
- **Chico:** A body composed of breccias and “D-type” veins, typical of Berta Norte, of which it is really the southernmost member, and is located immediately north of the Central Fault. A small working exposes high grade mineralization in the margins of prominent NW oriented “D-type” veins. Additionally, several trenches and short Outokumpu holes intersected oxide mineralization, which in conjunction have defined a zone of 100 x 50m, within a 200 x 150m low grade halo.

Figure 7.19: Salvadora pit looking to the NE



Note the old workings following high grade structures exposed in the pit. Also, the structures dipping towards the East Fault which control the mineralization

Figure 7.20: East wall of the Berta pit

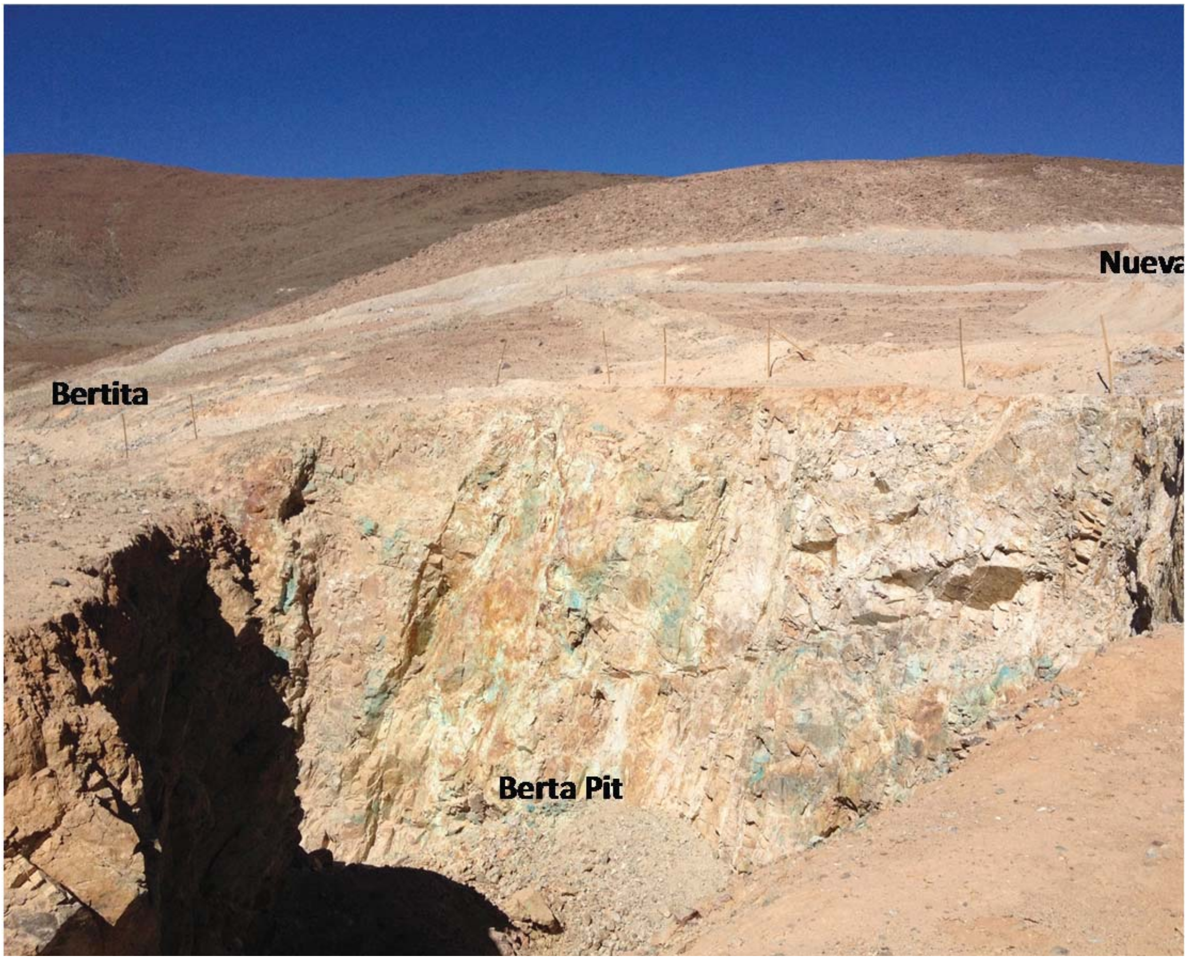


Figure also shows the approximate location of the unmined Nueva and Bertita breccias

For the purposes of evaluation, all of these bodies were interpreted in NW-SE oriented sections. All of the trench and drill hole assay information and geology derived from 1:2000 scale mapping werethen utilized to generate the boundaries of the oxide mineralization to be evaluated. In the same way, the material previously mined from the Carmen, Gemela 2, Salvadora and Berta pits was calculated and subtracted from the resource inventory.

Chalcopyrite is the dominant sulphide mineral in the hypogene zone, with occasional bornite; pyrite contents are typically very low. Molybdenite is also present in the breccias but shows no direct relationship with the copper minralization. Most of the mineralization occurs as cement in the hydrothermal breccias where it occurs together with sericite and quartz, as well as green sericite. Copper grades decrease from the centres of the breccia bodies towards the margins, where the mineralization occurs as disseminations, sulphide dominant veinlets and as halos to K feldspar and sericite.

The oxide zone reaches depths of 60-70m consistently in Berta Central. Green copper oxides are dominant, with chrysocolla being the dominant species, together with atacamite and copper sulphates. As in Berta Sur, a part of the oxides are present as black copper oxides (copper wad?).The copper oxides are present as matrix filling, disseminated in highly altered clasts, and filling open spaces and fractures. As with the sulphides, the oxides are coarse grained and bear a tight relationship with the breccias. It is estimated that the higher grade portion of the breccias contained grades in excess of 2%CuT and this was probably the grade of the material extracted from the old workings.It is noteworthy that the composite sample taken from the various breccia bodies and exploration drives for metallurgical testwork averaged 1.4%CuT, which confirms the high grade nature of the Berta Central bodies

No lateral or vertical zonation of mineralogy has been noted with the distribution of chrysocolla and black oxides (wad?) being very constant. However, in the Salvadora body, a small remnant chalcocite blanket is preserved in part by structures, towards the eastern boundary of the current pit. This has been partially oxidised to almagre and cuprite.

7.4.5 GEOCHEMISTRY

Although the rock geochemistry of Berta Central was affected by the numerous dumps and old workings, the sampling of surface trenches by Mantos Blancos and Outokumpu demonstrates the continuation of both the higher grade bodies and their low grade halos. The corridors linking the various breccia bodies are clearly established both by surface sampling and by mapping.

The surface anomalies of Cu and Mo coincide well with the breccias. There are no significant Au contents at Berta.

7.5 METALLOGENY

The Berta mineralized system is located in a belt of intrusives and IOCG, porphyry and Au vein style mineralization with ages close to 100 Ma. The Chivato fault is a reactivated primary structure which controlled the emplacement of the Remolinos pluton and is the metallotect that determined the location of mineralization over more than 50 km.

The various types of porphyry and breccias together with at least two phases of potassic alteration, phyllic alteration associated with the D type veins, evidence of Na-Ca alteration and incipient greisen, and finally the chalcopyrite with molybdenite mineralization together imply a system of porphyry copper type (Dilles, et al., 2000; Seedorf et al., 2008). The relatively high Mo content and absence of Au are inconsistent with the general pattern shown by Au rich porphyries correlated in age located to the E of Berta, near Inca de Oro. However, this difference in by-product is normal in porphyry belts even at the scale of districtsor clusters (Rivera et al., 2004).

8.0 DEPOSIT TYPES

As previously described, Berta corresponds to a porphyry copper system. In detail, Berta Sur exhibits evidence of alteration and mineralization which is comparable with that observed in the deepest parts of such systems. In particular the textural variations from TON to a crystal rich PTC, the background potassic alteration, the development of muscovite and greisen", and the Ca-Na alteration, are typical characteristics of the roots of porphyry systems (Dilles, et. al., 2000; Seedorf et. al., 2008). These characteristics also explain the lack of development of significant mineralization at depth. Rather the mineralization decreases in grade with depth or has a root-like shape becoming narrower in depth.

The interpretation at a local level shows variations from north to south, with the development of propyllitic alteration and NW oriented D type vein system with sericitic halo in the north passing to zones of breccia and the development of porphyry with potassic alteration in the south. This suggests a relatively deeper level of erosion toward the south and east, produced by NW oriented block faulting, which have segmented the porphyry system. The mineralization and alteration at Berta represents the exhumed roots of a porphyry system whose location was controlled by ZFCH and related structures, in particular by their recent movements of reverse type.

The copper oxide mineralization at Berta extends to depths of 30 to 100 m with mineralization outcropping at surface and with effectively no overburden. It has a simple mineral material and gangue mineralogy, excellent response to leaching and fairly continuous Cu grades and sharp contacts with low-grade margin mineralization. These favorable conditions are due to oxidation of the hypogene mineralization with simple alteration and mineralogy: dominant chalcopyrite hosted in breccias, porphyry and tonalite affected by potassic alteration. The lack of pyrite and unreactive hostrock has allowed the generation of in-situ oxidation, with only minor Cu re-mobilization and migration, without the formation of significant supergene sulfides.

9.0 EXPLORATION

Berta has been subject to surface and drilling exploration campaigns by Outokumpu, Mantos Blancos (Anglo American), Grandcru and MCC. Outokumpu completed trenching, shallow percussion drilling and 6 RC holes. Mantos Blancos discovered and evaluated the Berta Sur mineralization through trenching and RC drilling. Grandcru searching for a deep porphyry system drilled 9 DDH holes on the property. MCC, completed 3 phases of RC drilling, and covered the whole project area with rock geochemistry and geophysics. Work carried out is summarized in this section.

9.1 SURVEYING, IMAGE AND TOPOGRAPHIC CONTOUR BASE

Notwithstanding the fact that all the work performed in the area had their own topographic bases and controls of the previous surveys, MCC carried out a verification of all drill hole collars, mine workings and other significant points. Topography and restitution through images ortho-rectification works were as follows:

- A topographical survey was completedwith ISO 9001-2000 certified Geodimeter Total Station, by STD Servicios Topográficos. All points were linked to the planimetric UTM coordinate system, obtained at the Instituto Geográfico Militar (IGM) or SERNAGEOMIN survey markers. Allmeasurements were compensated for the sector’s magnetic declination.

- All drill hole collars were surveyed and their coordinates input into a database.

- It also included surface trenches, mine workings and dumps, and roadsresulting in a 1 m contour line survey of Berta Sur and its surroundings for mine planning purposes.

- As an exploration base, MCC requested that TerraAnálisis S.A. produced an ortho-rectification of a high resolution satellite image and the restitution of the entire Berta Project area with contour lines every 5 m.

- Surface topography for the deposit's NW end was needed to be reconstructed from contour lines every 5 m. This surface created from 2D data is far away from the detail obtained for the rest of the deposit, and generates a problem of different dimensions at the limit between both surfaces; nevertheless, this error does not generate major distortions in calculations and its solution is to complete the detailed topographical survey. In addition, for the estimated resources calculation by block model, the small tonnage extracted from pits by artisinal mining was subtracted.

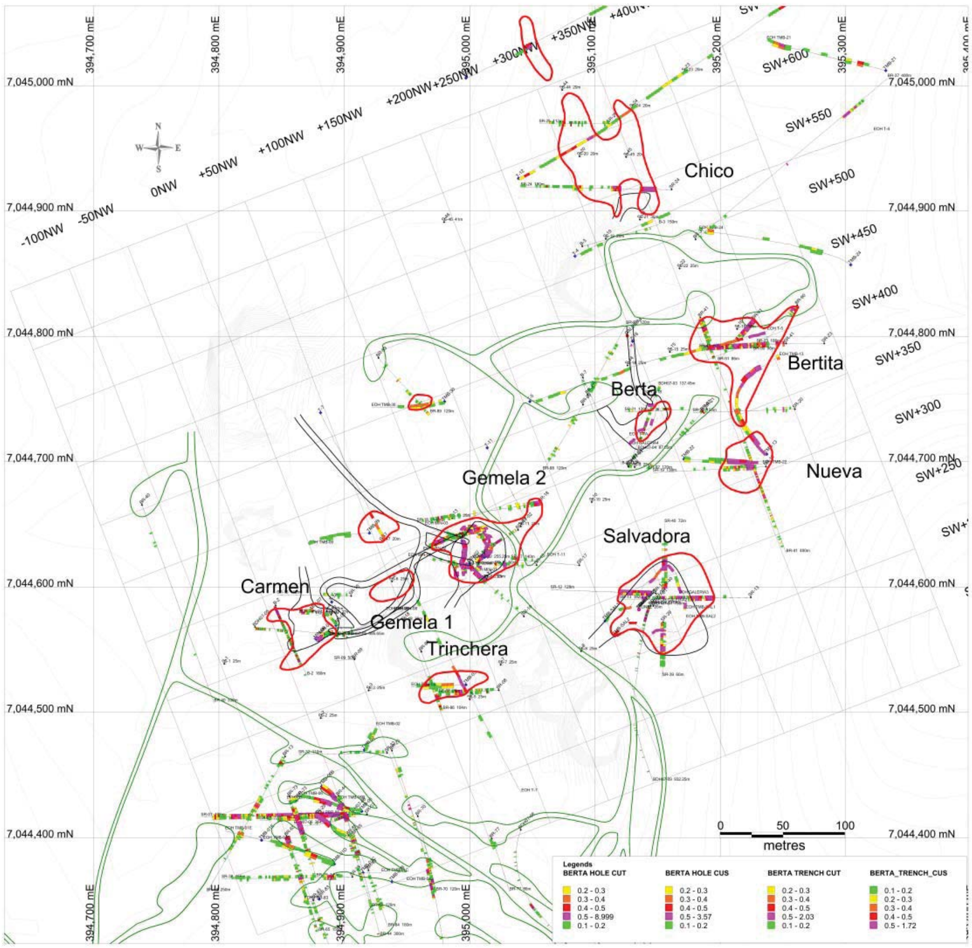
9.2 SURFACE SAMPLING

Available surface sampling information was recovered from Mantos Blancos and to a lesser extent from Outokumpu and Grancru report (described in section 7.3.5).

Mantos Blancos (Guiñez and Zamora, 1998) sampled Berta Sur with ten trenches. These were excavated with bulldozer following a 290° azimuth, exposing the rock and generating a central cut from which samples were taken at 5 m regular intervals. Samples were assayed for CuT and CuS by Absortion Atomic Spectroscope (AAS). Although the control material and background of QA/QC standard procedures are not available, the quality was tested through the collection of samples used for metallurgical tests from various trenches for the different CuT grade ranges. These second samples showed very similar CuT and CuS results to the samples reported by Mantos Blancos used in this report. Figure 9.1 shows the Berta Sur trench locations with the Mantos Blancos' CuT and CuS results used in this report.

For BertaCentral as for Berta Sur, there was a sampling of trenches and channels totaling 36 trenches / channels with 782 samples in 3,229 linear m., located at surface, in pits and underground.Although MCChas not completed check sampling of these trenches and channels, their results are deemed to be reasonable based on comparison with nearby drill holes and with their mapped geology (Figure 9.1)

Figure 9.1: Berta Central area showing surface trench CuT – CuS results from Mantos Blancos



Grandcru sampled the same trenches with a portable Fluorecence X Ray (XRF), reporting very different values to the AAS results obtained from the original trench samples. Notwithstanding, the orders of magnitude are correlated to the mineralization limits. Some additional trenches were excavated by Grandcru, to the south, also sampled with portable XRF. However, the Grandcru geochemical data was not utilized for the evaluation presented in this report.

MCC completed grid rock and soil geochemistry for the entire project area, with samples assayed for Cu and Mo by AAS

9.3 SURFACE GEOLOGIC MAPPING

The complete project area was previously mapped by Outokumpu and Mantos Blancos, at a 1:2,000 scale, by conventional methods. MCCinitiallycarried out a geological scouting, characterizing some points with data of rock-alteration-mineralization and then followed up with a systematic mapping of trenches, mine workings and outcrops, locating observation points with GPS at a 1:2,000 scale. Each data point was characterized by rock type, structure,alteration and mineralization with each mineral intensity estimated in a qualitative manner on a scale from trace to abundant, (1 to 5, respectively) and occurrence type. Units and criteria were the same as those used in the subsequent logging and re-logging of drill holes. The resulting map is shown in Figure 7.6.

9.4 GEOPHYSICS

Mantos Blancos completed seven EW oriented 900m long Dipole-Dipole IP/resistivity totaling 6.3 km, reportedly encountering a NS oriented IP anomaly, but MCC does not have a copy of this data. Grandcru completed a ground magnetics survey which has been utilized byMCC.

MCC explored the Berta Project area in October 2011, with Off-Set Pole-Dipole IP/Resistivity, carried out by Zonge S.A. The interpretation of the 3D modeled IP together with magnetometry was used for the design of the subsequent drilling phase. It was concluded that the geophysics did not define the Berta Sur mineralization, which does, however coincide with a magnetic low and an IP high, both of which extend well beyond the limits of the mineralization. Other IP anomalies on the property were found to be unrelated to mineralization,

9.5 GEOCHEMISTRY

At Berta, geochemical sampling has comprised trench sampling by Outokumpu and Mantos Blancos together with portable XRF assaying in the aforementioned trenches and some additional trenches dug by Grandcru. MCC covered the project area with a regular 100 x 100 m grid of rock/soil samples assayed for Cu and Mo.

The Berta Surand Central trenches exposed the oxide mineralization and aided in geological mapping and sampling. In particular, trench samples collected by Mantos Blancos and validated by MCC's metallurgical sampling, have been used in the evaluation exercise. Their origin and quality has already been discussed in section 9.2 of this report.

10.0 DRILLING

A total of 29,377 m of drilling, of which 85% corresponds to RC drilling has been completed at Berta, as shown in Tabla 10.1.

Tabla 10.1: Summary of drilling campaigns at Berta Sur and Berta Central as compared to the total Berta Project

				Berta Sur			Berta Central			Total Berta Project		
Berta Drillhole Database	Companny	Date	Type	Number of Holes	Avg Depth (m)	Total Meters	Number of Holes	Avg Depth (m)	Total Meters	Number of Holes	Avg Depth (m)	Total Meters
1	Outokumpu	Mar-Sept 1994	RC/DTH	4	25	100	28	44	1221	55	40	2216
2	Mantos Blancos	Sep-Dec 1997	RC	19	112	2,126	16	121	1930	42	118	4942
3	Grand CRU	Feb-Jul 2007	DDH	3	435	1,305	6	335	2007	9	368	3311
4	CORO Phase 1	Jul-Aug 2011	RC	23	181	4,160				24	182	4360
5	CORO Phase 2	Mar-Jun 2012	RC	14	300	4,198	3	379	1136	32	329	10520
6	CORO Infill	Jul-Aug 2012	RC	29	112	3,264	7	109	766	36	112	4028
TOTAL (Holes)				92		15,153	60		7,060	198		29,377

The first drilling campaign was carried out in 1994 by Outokumpu, with short percussion (DTH) holes and 7 RC holes located almost entirely in Berta Central and Norte. The second RC drilling campaign was completed in 1997 by Mantos Blancos with a total of 4,942 m of which 2,126 m was completed in Berta Sur and 1,930m in Berta Central. In 2007 Grandcru completed nine DDH holes totaling 3,312 m, of which 3 holes for 1,305 m were located in Berta Sur and 6 holes for 2,007m in Berta Central (Tabla 10.1). Mantos Blancos interpreted mineralization controls to be following N-S main structures and accordingly oriented their drilling at Berta Sur at 270°. MCC's subsequent detailed mapping showed that these holes were mostly drilled sub parallel to the structures controlling the mineralization.

MCC completed two RC exploration campaigns at Berta and a third one of resource drilling on a 50 m x 50 m regular grid at Berta Sur. All three programs were carried out by Perfomin Ltda. The first phase was undertaken in 2011 totaling 4,160 m and drilling oriented from 230 to 270°, vertical, with depths from 100 to 250 m (Tabla 10.1). The second phase, commenced after the mapping and geophysical and geochemical data integration included 4,198 drilled meters in Berta Sur, with depths from 250 to 400 at 160° azimuth. Finally, infill grid drilling was completed, spaced at 50 m x 50 m, with holes oriented 160°, inclination -60° and depths from 80 to 120 m.

For this resource update 82 additional drill holes were included totaling 10,721.15 drilled meters and 5,965 samples with 1 and 2 meters supports.

Field operation and especially the procedures for drill hole sampling, recovery control and geological logging was performed by a consulting company, Geominco Ltda. Field control included holes depth control, measurement of hole deviation, taking and cleaning of cuttings for geological logging, as well as the collection of basic sample information such as serial numbering, depth interval and weight data for the original samples, as well as for samples sent to the laboratory.

In the grid drilling stage, the deviation of most of the holes was surveyed, a service performed by Perfomin, using a Reflex Gyro digital micro-gyroscope. In order to ensure the quality some holes were surveyed twice, with no significant differences found. Surveying was controlled by Geominco.

Cuttings from MCC holes, especially those from phase two and the infill holes were logged under standard methodologies, from samples obtained after the process of splitting and storage in plastic containers. An initial description of rock types, and intensity of mineralization and alteration was subsequently validated against assay values and all logs have been stored in Excel spreadsheets.

11.0 SAMPLE PREPARATION AND SECURITY

11.1 RC SAMPLE COLLECTION

RC drilling samples were collected by MCC; according to the following standard procedure (see procedure's photographs Figure 11.1):

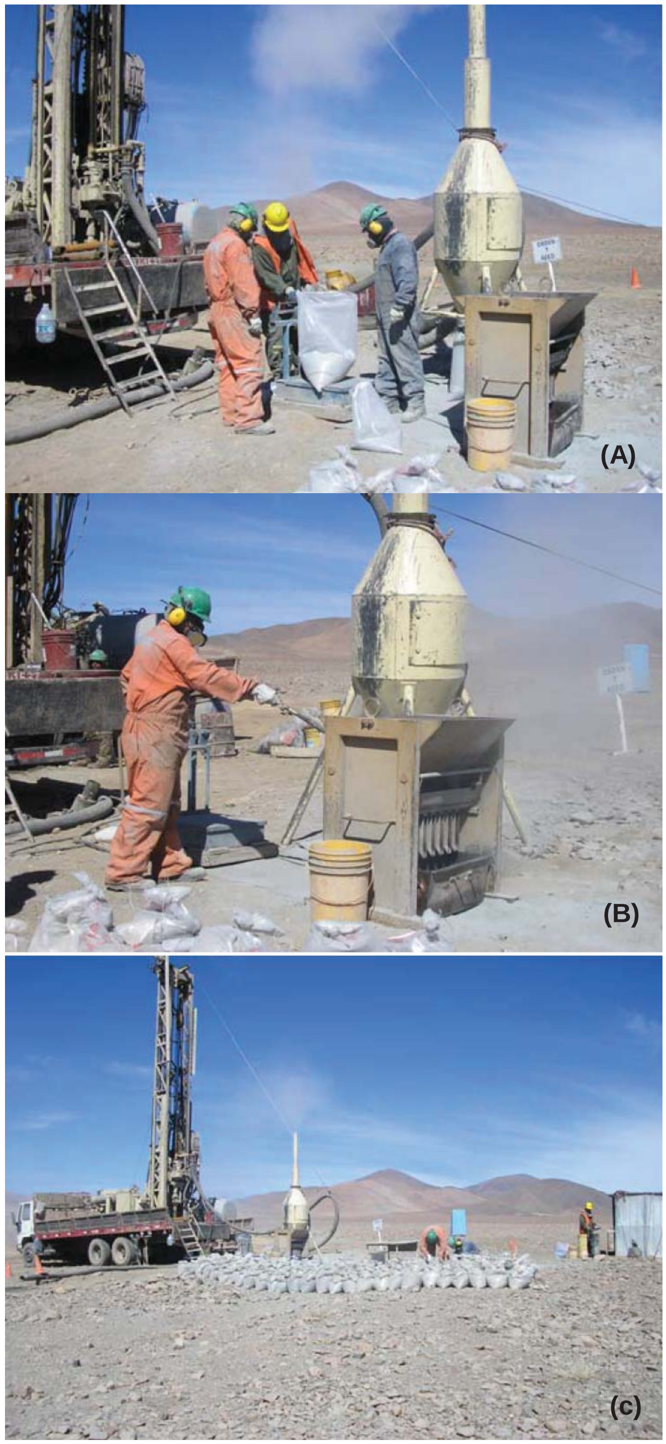
- Samples were collected at 2 m intervals, in suitable plastic bags, using a cyclone provided by the contractor. The serial numbering and depths of the samples were verified in paper control spreadsheets, as well as in numbered ticket book of sample cards.
- Samples were weighed to measure recoveries, by comparing the individual sample weights to the theoretical material weight (70-80 kg), a task performed manually by contractor's personnel and supervised by Geominco personnel. The scale was calibrated through standard weights every 10 m; the sample was re-weighed to ensure the process quality.
- Successive splits, with a mechanical splitter, were made by the contractor's personnel under the supervision of Geominco personnel. In Phases One and Two half of the sample was stored on site, while in the final grid drilling phase only the last splits of approximately 10 kg were collected, one of them being sent to the laboratory for preparation and analysis and the other being stored on siteas backup.
- In the first splitting process, a 1 kg sample was extracted for geological logging, with part of it kept as separate coarse and fine cuttings in a plastic chip tray.
- The splitter and equipment utilized in the sampling process were cleaned after every sample, using compressed air.
- The sample collection and splitting operation was supervised by a Shift chief from Geominco, by the drill holegeologist and by an MCC field assistant.

- Samples were sealed, organized and counted in inventory for laboratory shipment. Transport was carried out under MCC supervision until its reception at Analytical Assay (3A) in Copiapó.

Operational reports on each campaign were issued. Drill hole location, depths and sampling were coordinated on a daily basis between Geominco's shift chief and MCC's field assistant.

Geoinvestment is of the opinion that the sample collection methods and security protocols described for the Berta project were done to acceptable industry standards.

Figure 11.1: Sampling collecting and weighing (A); Splitting (B) and Checking (C) on site



11.2 SAMPLE PREPARATION AND ANALYSIS

Samples were prepared in 3A's laboratories according to the following procedure:

- After receiving the samples the numbers were verified against the laboratory orders.
- Drying of humid samples.
- Weighing.
- Sieving and crushing of the fraction over -10# mesh. Assuring the sample is under -10 # mesh.
- Weighing for fines losses verification.
- Splitting in rotary splitter to obtain a 500 g sample.
- Pulverizing of the 500g sample to -150# mesh, dividing it into three envelopes: two sample pulps of 125 g and one of 250 g. with one 125g sample, Pulp1, sent for assay.
- Laboratory preparation coarse rejects are in storage and the envelopes of pulps 2 and 3 were retrieved by MCC.

In order to prepare the batches for analysis, MCC supervisors received the pulp samples and prepared batches of 40 samples on average, inserting reference and duplicates materials according to a previously assigned numbering system. A new order for chemical analysis was then prepared.

The CuT and Mo assay at 3A was performed according to the following procedure:

- Weighing 1.0 g of sample in a 100 ml flask, with a minimum precision of 0.1 mg.
- Addition of 50 ml of *Agua Regia* (Regia Water, $3H_2O + 2HNO_3 + 6HCl$).
- Digest in bain-marie to 85-90° C for three hours. If molybdenum is not to be analyzed, one hour digestion is sufficient.

- Cool down with cold water current. Dilute to the level of the flask with $AlCl_3*6H_2O$ solution (17.88g/l); only if Fe Mn or Mo is to be analyzed. If those elements are not to be analyzed, dilute only with distilled water.
- Analysis by Atomic Absorption Spectometry (AAS).

For CuT and Mo, the detection limit is 0.001%.

CuS assay by sulfuric acid leaching method and AAS determination is made in 3A under the following procedure:

- Weighing in an analytical scale with 0.1 mg precision 1,000 mg and transfer it into a 100 ml flask.
- Leaching; add 50 ml of 5% v/v sulfuric acid solution at room temperature, shake at 130 rpm for 60 minutes.
- Phases separation; dilute to 100 ml and separate the liquid and solid phases.
- The solid phase can be used to determine the sequential cyanide soluble copper, i.e. CuCN.
- If so, wash the solid phase separated by filtration, with 20 ml of de-ionized water; dilute the liquid phase to 100 ml.
- Analysis by Atomic Absorption Spectometry (AAS).
- Results with three decimals detection limit 0.001%, or in ppm with no decimals, detection limit 10 ppm.

CuS limit detection limit is 0.001%

Geoinvestment is of the opinion that the sample preparation methods and assaying protocols described for the Berta project were done to acceptable industry standards.

11.3 METALLURGICAL SAMPLE COLLECTION

At Berta Sur three samples were collected for metallurgical leach tests. The criteria and material obtaining that constitute those samples are as follows:

- Materials were collected to characterize high grade (~0.8 % CuT), medium grade (~0.6 %CuT) and low grade (~0.4 %CuT) oxide mineralization in different rock types, especially breccias, tonalites and Crowded porphyry. They were denominated samples A, B and C.

- Sample A was obtained as a composite from selected sections of Grandcru's core samples. For that, samples were cleaned and stored in bags. In addition, a photographic registerwas kept and the detailed geological logging completed. The weight was 211.1 kg and average grade from drilling data was 0.86 %CuT, with no available CuS assays.

- Samples B and C were composited from selected sections of the Mantos Blancos' trenches. The intention was to reconstruct the original 5 m intervals, with known grade of CuT and CuS. Trenches were cleaned and samples were obtained bearing in mind the requirement of the final average values. These were photographed and mapped in detail. Sample B was obtained from trenches MB-03, 015 and 04B had an average grade of 0.63 %CuT and 0.45 %CuS, for a 252.5 kg weightSample C has a weight of 251.5 kg with an average grade of 0.44 %CuT and 0.30 %CuS

At Berta Central two metallurgical samples were collected from prospecting trenches and old mine workings. Because no obvious variation exists in host rock type, alteration and mineralization, the selection and collection procedure for both samples was as follows:

Rock samples were collected from the Berta-Bertita-Nueva sector (sample BC-A) and from Gemela- Salvadora sectors (sample BC-B). Composites were obtained from approximately 12 kg individual samples collected from 16 sectors.

Sample BC-A was obtained mainly from Berta and Trench TMB-13, as well as from some other representative shallow workings and roads cuts. Sample BC-B was obtained from small pits from Gemela 1, Trench TMB-05 and from the Salvadora pit walls.

All samples were located with geological control on site; walls and rock exposures were cleaned and rock fragments collected at each sample point. A geological description and photograph was stored from each sample site.

Samples from both Berta Sur and Berta Central were stored in plastic bags, labeled and sent for metallurgical tests.

11.4 DENSITY MEASUREMENTS

In order to obtain density measurements characterizing the Berta Sur and Central mineralized rocks, test-samples were obtained from Grandcru's DDH core samples. Sample selection criteria and laboratory tests are as follows:

- 16 test-samples were selected from Grandcru's DDH coresamples. Different ranges of grade, rock type and mineralization were selected.
- Each selected piece was logged in detail and photographed.
- They were then sent to Calama's Rock Mechanics certified laboratories, for the corresponding unit weights.
- The utilized method was the weight-volume ratio, with previously kerosene waterproofed samples precision weighed in air and then, weighed submerged in water. The variability of the weights is considered to be acceptable.

For oxide samples in the range of 2.47 and 2.62 g/cm^3, the average density was 2.51 g/cm^3. For sulfides, it was 2.63 g/cm^3 in a range of 2.56 to 2.66 g/cm^3.

11.5 QUALITY CONTROL AND QUALITY ASSURANCE (QA/QC)

The complete list of drill holes used in the Berta Sur and Central resource estimate can be found inTabla 11.1

Around 81% of the total length drilled included in this first resource estimate for Berta Sur, (11,622 m), and 34,5% for Berta Central (3,688 m) has Quality Assurance / Quality Control (QA/QC) information. Some of the older holes are over 20 years old and it was not possible to access their data. The procedures used to assure and control the sampling and chemical analysis quality of the drilling performed by MCCwere supervised by Alan Stephens, President & CEO, Coro Mining Corp, and VP Exploration Sergio Rivera, Coro Mining Corp, both Qualified Persons. This review of MCCQA/QC procedures wassupervised by Sergio Alvarado, Qualified Person external to Coro.

11.5.1 SAMPLING PROCEDURE

During the two day site visit by Geoinvestmentin October 2012, the existence of the drill chip trays for logging, as well as the corresponding labeled and sealed bags of sample rejectswasverified. These are storedby drill hole in a secure location, as indicated in

Figure 11.2. Recoveries for the 2 m samples averaged 68 kg for the 2011 campaign (holes BR-01 to BR-24) and 77.5 kg for the 2012 campaigns (holes BR-25 to BR-85), which in the latter caserepresents recoveries of 98%. For the first case there is not enough data to calculate the recovery percentage.

Samples sent to the laboratory corresponds to 12.5% of the total 2 m recovered during drilling, representing the portion obtained on site by three successive divisions on a simple two stage splitter. 50% of the material on the first division was discarded and on the third division two 12.5% samples were obtained from the total: one for chemical analysis and the other was combined with the 25% reject or was used as duplicate.

Sample shipment was made in 70 sample batches, to the Andes Analytical Assays in Santiago Chile, where they were prepared according to the 3A7.5P4T1R1 lab protocol and tested for copper (Cu) and molybdenum (Mo) according to 2A-AAAS1E01/03 procedures. Some samples were assayed for gold (Au) also. Results were entered in the MCCdatabase by remote direct transfer, without human intervention.

Almost 15% (2,126 m), of the drilling included in the initial resource estimate, was completed by Mantos Blancos in the 1990's. Work was directed by Richard Zamora, Chief Geologist of the Manto Verde mine at the time. One of the authors of the resource estimate report issued in January 2013participated in that campaign and can testify that the job was performed under very high quality standards and sampling protocols, especially given that the objective was to add copper oxide resources to Manto Verde operations. Finally, this resource estimate includes nine diamond drill holes completed by Grandcru. It was verified on site that these are in a very good state of conservation and that they were sampled using a diamond saw (see Figure 11.3)

Figure 11.2: Drilling rejects of drill holes completed by MCC at Berta project

Figure 11.3: Core Sample of drilling performed by Grandcru at Berta project

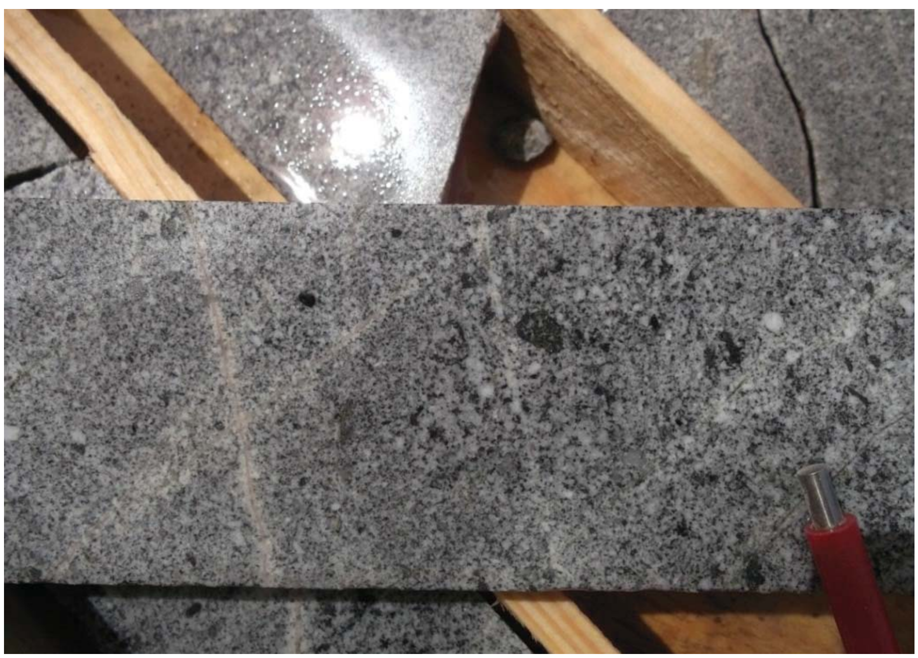


Tabla 11.1: List of Drill holes included in Berta Surand Berta Central resource estimate

ID	Este	Norte	Cota	Az	Dip	Largo	Company	QA/QC
BDH07-01	395146.525	7044584.844	1732.800	0	-90	199.00	Grancru	
BDH07-02	395038.688	7044645.459	1729.959	237	-80	255.25	Grancru	
BDH07-03	395127.081	7044695.748	1736.536	20	-60	137.45	Grancru	
BDH07-04	395126.496	7044696.082	1736.647	30	-80	87.05	Grancru	
BDH07-05	394828.928	7044567.843	1712.270	95	-80	395.65	Grancru	
BDH07-06	394,906,031	7,044,418,000	1,731,320	248,000	-80,000	264.45	Grancru	
BDH07-07	395,057,531	7,044,276,000	1,751,840	0	-90,000	250.00	Grancru	
BDH07-08	395040.771	7044406.209	1730.232	200	-80	790.45	Grancru	
BDH07-09	395144.525	7044584.844	1732.500	200	-80	932.25	Grancru	
S-1	394804.302	7044538.671	1714.686	0	-90	25.00	Outokumpu	
S-2	394,881,344	7,044,495,500	1,721,720	0	-90,000	25.00	Outokumpu	
S-3	394919.514	7044517.488	1719.513	0	-90	25.00	Outokumpu	
S-4	394886.055	7044580.646	1712.454	0	-90	25.00	Outokumpu	
S-5	395,000,125	7,044,510,500	1,728,060	0	-90,000	25.00	Outokumpu	
S-6	394938.031	7044604.375	1718.912	0	-90	25.00	Outokumpu	
S-7	395,024,781	7,044,538,000	1,730,330	0	-90,000	25.00	Outokumpu	
S-8	395,088,875	7,044,548,500	1,736,670	0	-90,000	25.00	Outokumpu	
S-9	395131.663	7044695.558	1735.346	0	-90	25.00	Outokumpu	
S-10	395097.447	7044667.686	1732.601	0	-90	25.00	Outokumpu	
S-11	395040.918	7044648.283	1728.128	0	-90	25.00	Outokumpu	
S-12	395013.672	7044606.358	1728.450	0	-90	25.00	Outokumpu	
S-13	394986.058	7044654.819	1722.955	0	-90	25.00	Outokumpu	
S-14	395126.006	7044776.607	1718.957	0	-90	25.00	Outokumpu	
S-15	395159.755	7044787.414	1719.937	0	-90	25.00	Outokumpu	
S-16	395213.469	7044805.681	1723.953	0	-90	15.00	Outokumpu	
S-17	394889.595	7044577.883	1712.759	0	-90	27.00	Outokumpu	
S-18	394947.633	7044572.523	1720.905	0	-90	30.00	Outokumpu	
S-19	395108.619	7044877.732	1712.526	0	-90	20.00	Outokumpu	
S-20	395087.681	7044943.417	1714.552	0	-90	20.00	Outokumpu	
S-21	395135.827	7044893.039	1717.871	0	-90	35.00	Outokumpu	
S-22	395167.447	7044853.985	1719.139	0	-90	25.00	Outokumpu	
S-23	395171.307	7045010.627	1728.804	0	-90	20.00	Outokumpu	
S-24	395129.566	7044981.834	1725.149	0	-90	20.00	Outokumpu	
S-44	395073.795	7044996.528	1712.339	0	-90	20.00	Outokumpu	
S-45	395124.825	7044942.944	1723.096	0	-90	20.00	Outokumpu	
S-46	394979.623	7044890.992	1700.738	0	-90	41.00	Outokumpu	
S-47	394929.832	7044636.091	1717.963	0	-90	20.00	Outokumpu	
B-1	395012.672	7044606.258	1727.900	0	-90	150.00	Outokumpu	
B-2	394845.298	7044585.176	1712.097	156	-70	180.00	Outokumpu	
B-3	395089.938	7044871.690	1708.693	74	-65	150.00	Outokumpu	
B-7	395090.373	7044766.942	1720.861	156	-60	153.00	Outokumpu	
SR-01	395,102,156	7,044,176,500	1,754,910	268,000	-67,000	160.00	Mantos Blancos	
SR-02	395,072,219	7,044,226,500	1,752,820	266,000	-52,000	120.00	Mantos Blancos	
SR-03	395,102,344	7,044,276,000	1,752,240	269,000	-49,000	172.00	Mantos Blancos	
SR-04	395,132,938	7,044,226,000	1,759,300	268,000	-60,000	84.00	Mantos Blancos	
SR-05	395,057,125	7,044,317,500	1,743,370	272,000	-57,000	146.00	Mantos Blancos	
SR-06	395,103,469	7,044,376,500	1,742,640	273,000	-50,000	192.00	Mantos Blancos	
SR-07	394,918,125	7,044,418,500	1,731,420	269,000	-48,000	198.00	Mantos Blancos	
SR-08	395,023,250	7,044,517,500	1,729,950	267,000	-60,000	96.00	Mantos Blancos	
SR-10	395043.402	7044576.405	1733.132	312	-61	124.00	Mantos Blancos	
SR-11	395,059,500	7,044,275,500	1,751,740	272,000	-46,000	114.00	Mantos Blancos	
SR-12	395148.525	7044594.844	1725.500	272	-49	128.00	Mantos Blancos	
SR-13	395224.296	7044591.717	1762.431	269	-57	192.00	Mantos Blancos	
SR-14	395,052,906	7,044,176,500	1,751,180	272,000	-49,000	90.00	Mantos Blancos	
SR-15	394905.554	7044592.764	1715.576	271	-58	112.00	Mantos Blancos	
SR-16	395053.525	7044617.344	1732.199	270	-48	50.00	Mantos Blancos	

Table 11.1 (continued)

ID	Este	Norte	Cota	Az	Dip	Largo	Company	QA/QC
SR-17	395087.553	7044617.302	1734.989	275	-67	140.00	Mantos Blancos	
SR-18	395056.724	7044667.136	1728.835	261	-46	144.00	Mantos Blancos	
SR-19	395236.466	7044695.960	1752.899	267	-49	138.00	Mantos Blancos	
SR-20	395259.603	7044741.823	1746.179	268	-57	150.00	Mantos Blancos	
SR-21	395188.977	7044742.047	1735.142	268	-57	120.00	Mantos Blancos	
SR-22	395,025,812	7,044,226,500	1,752,060	271,000	-46,000	78.00	Mantos Blancos	
SR-23	395282.888	7044794.242	1734.587	270	-60	108.00	Mantos Blancos	
SR-24	395161.143	7044917.008	1726.830	271	-48	180.00	Mantos Blancos	
SR-25	395111.355	7044969.497	1722.618	270	-60	112.00	Mantos Blancos	
SR-26	395,028,125	7,044,081,000	1,741,280	269,000	-49,000	60.00	Mantos Blancos	
SR-29	394,878,781	7,044,417,500	1,734,400	265,000	-50,000	72.00	Mantos Blancos	
SR-31	395,017,250	7,044,309,500	1,749,500	277,000	-61,000	76.00	Mantos Blancos	
SR-32	394,934,125	7,044,467,500	1,723,140	269,000	-50,000	110.00	Mantos Blancos	
SR-34	395,053,469	7,044,126,000	1,747,780	274,000	-58,000	92.00	Mantos Blancos	
SR-37	394,910,406	7,044,128,000	1,753,100	269,000	-45,000	50.00	Mantos Blancos	
SR-38	394,906,531	7,044,369,000	1,738,230	269,000	-48,000	156.00	Mantos Blancos	
SR-39	395154.525	7044571.844	1731.500	181	-43	60.00	Mantos Blancos	
SR-40	395155.525	7044598.844	1701.500	0	-44	72.00	Mantos Blancos	
SR-41	395252.521	7044794.174	1731.398	267	-46	100.00	Mantos Blancos	
SR-42	395,117,031	7,044,319,000	1,745,460	272,000	-60,000	60.00	Mantos Blancos	
BR-01	394,958,906	7,044,267,500	1,753,780	0	-90,000	100.00	Coro Mining Phase 1	
BR-02	394,913,906	7,044,268,000	1,750,960	0	-90,000	100.00	Coro Mining Phase 1	
BR-03	394,856,281	7,044,261,500	1,748,890	0	-90,000	100.00	Coro Mining Phase 1	
BR-04	394,808,594	7,044,266,500	1,744,940	0	-90,000	100.00	Coro Mining Phase 1	
BR-05	394,756,219	7,044,270,000	1,741,560	0	-90,000	100.00	Coro Mining Phase 1	
BR-06	394,707,031	7,044,264,500	1,739,410	0	-90,000	100.00	Coro Mining Phase 1	
BR-07	394,651,531	7,044,268,500	1,738,390	0	-90,000	100.00	Coro Mining Phase 1	
BR-08	395,102,156	7,044,176,500	1,754,910	0	-90,000	200.00	Coro Mining Phase 1	
BR-09	395,159,406	7,044,178,000	1,763,100	270,000	-65,000	210.00	Coro Mining Phase 1	
BR-10	395,202,469	7,044,178,000	1,765,810	270,000	-65,000	200.00	Coro Mining Phase 1	
BR-11	395,104,688	7,044,080,500	1,747,810	270,000	-65,000	250.00	Coro Mining Phase 1	
BR-12	395,106,250	7,044,120,000	1,752,260	270,000	-65,000	100.00	Coro Mining Phase 1	
BR-13	394852.872	7044464.686	1727.593	210	-60	250.00	Coro Mining Phase 1	
BR-14	394,949,844	7,044,361,000	1,738,330	230,000	-60,000	250.00	Coro Mining Phase 1	
BR-15	395,034,531	7,044,361,500	1,736,590	230,000	-70,000	200.00	Coro Mining Phase 1	
BR-16	395,123,812	7,044,322,000	1,745,590	230,000	-60,000	200.00	Coro Mining Phase 1	
BR-17	395,048,688	7,044,263,500	1,752,740	230,000	-60,000	200.00	Coro Mining Phase 1	
BR-18	395,011,125	7,044,225,000	1,752,420	230,000	-60,000	200.00	Coro Mining Phase 1	
BR-19	394,976,531	7,044,171,000	1,751,210	230,000	-60,000	200.00	Coro Mining Phase 1	
BR-20	394,885,000	7,044,079,500	1,750,790	230,000	-60,000	250.00	Coro Mining Phase 1	
BR-21	394,711,281	7,044,250,500	1,739,740	230,000	-60,000	250.00	Coro Mining Phase 1	
BR-22	394,762,562	7,044,262,500	1,741,400	50,000	-60,000	250.00	Coro Mining Phase 1	
BR-23	394,836,375	7,044,163,000	1,757,680	230,000	-60,000	250.00	Coro Mining Phase 1	
BR-25	394938.065	7044469.344	1723.507	160	-60	400.00	Coro Mining Phase 2	QA/QC
BR-37	395180.284	7044877.269	1722.866	50	-60	400.00	Coro Mining Phase 2	QA/QC
BR-40	394738.627	7044665.362	1706.251	160	-60	336.00	Coro Mining Phase 2	QA/QC
BR-41	395184.161	7044814.706	1720.601	160	-60	400.00	Coro Mining Phase 2	QA/QC
BR-42	395,040,969	7,044,334,500	1,742,130	160,000	-59,300	400.00	Coro Mining Phase 2	QA/QC
BR-43	395,066,344	7,044,255,500	1,751,140	160,000	-60,000	300.00	Coro Mining Phase 2	QA/QC
BR-44	394895.406	7044432.923	1730.741	160	-60	300.00	Coro Mining Phase 2	QA/QC
BR-45	394,969,250	7,044,235,500	1,753,190	160,000	-60,450	300.00	Coro Mining Phase 2	QA/QC
BR-46	394,966,094	7,044,243,000	1,753,360	340,000	-60,090	200.00	Coro Mining Phase 2	QA/QC
BR-47	394,885,875	7,044,163,000	1,757,180	160,000	-60,000	300.00	Coro Mining Phase 2	QA/QC
BR-48	394,823,750	7,044,210,000	1,751,180	160,000	-59,430	300.00	Coro Mining Phase 2	QA/QC
BR-49	394,849,781	7,044,231,500	1,750,030	160,000	-59,690	300.00	Coro Mining Phase 2	QA/QC
BR-50	394,744,469	7,044,126,500	1,750,990	160,000	-60,000	250.00	Coro Mining Phase 2	QA/QC

Table 11.1 (continued)

ID	Este	Norte	Cota	Az	Dip	Largo	Company	QA/QC
BR-51	394,648,406	7,044,091,500	1,732,290	160,000	-60,000	250.00	Coro Mining Phase 2	QA/QC
BR-54	395,117,656	7,044,313,000	1,745,110	160,000	-60,000	300.00	Coro Mining Phase 2	QA/QC
BR-55	394,687,719	7,043,991,000	1,739,790	160,000	-60,000	300.00	Coro Mining Phase 2	QA/QC
BR-56	394,777,875	7,044,038,000	1,754,960	160,000	-60,000	298.00	Coro Mining Phase 2	QA/QC
BR-57	394,987,938	7,044,327,000	1,748,200	160,000	-59,140	114.00	Coro Mining Infill	QA/QC
BR-58	395,005,000	7,044,272,500	1,754,730	160,000	-60,910	108.00	Coro Mining Infill	QA/QC
BR-59	394,931,938	7,044,330,000	1,745,680	160,000	-60,520	98.00	Coro Mining Infill	QA/QC
BR-60	394,888,719	7,044,315,500	1,745,260	160,000	-59,260	126.00	Coro Mining Infill	QA/QC
BR-61	394,922,656	7,044,221,500	1,753,120	160,000	-59,790	120.00	Coro Mining Infill	QA/QC
BR-62	394,798,375	7,044,395,000	1,730,910	160,000	-59,310	150.00	Coro Mining Infill	QA/QC
BR-63	395,025,969	7,044,234,500	1,752,010	160,000	-58,800	102.00	Coro Mining Infill	QA/QC
BR-64	394,963,000	7,044,259,000	1,753,210	160,000	-60,770	96.00	Coro Mining Infill	QA/QC
BR-65	394854.150	7044400.790	1738.221	160	-60	150.00	Coro Mining Infill	QA/QC
BR-66	395,093,031	7,044,204,000	1,753,650	160,000	-59,980	120.00	Coro Mining Infill	QA/QC
BR-67	395,042,719	7,044,190,500	1,749,350	160,000	-60,400	90.00	Coro Mining Infill	QA/QC
BR-68	395,006,031	7,044,176,000	1,748,510	160,000	-59,760	120.00	Coro Mining Infill	QA/QC
BR-69	394,936,781	7,044,195,500	1,754,260	160,000	-59,150	120.00	Coro Mining Infill	QA/QC
BR-70	394958.720	7044416.940	1729.500	160	-60	120.00	Coro Mining Infill	QA/QC
BR-71	394,947,750	7,044,155,500	1,752,350	160,000	-58,940	120.00	Coro Mining Infill	QA/QC
BR-72	394,967,969	7,044,370,500	1,737,890	160,000	-59,350	120.00	Coro Mining Infill	QA/QC
BR-73	394856.400	7044432.770	1732.208	160	-60	140.00	Coro Mining Infill	QA/QC
BR-74	395,099,562	7,044,254,500	1,752,010	160,000	-59,040	100.00	Coro Mining Infill	QA/QC
BR-75	395,067,750	7,044,118,000	1,747,390	160,000	-59,220	120.00	Coro Mining Infill	QA/QC
BR-76	395,087,406	7,044,364,000	1,738,060	160,000	-60,410	80.00	Coro Mining Infill	QA/QC
BR-77	395017.650	7044399.120	1731.433	160	-60	86.00	Coro Mining Infill	QA/QC
BR-78	395,140,531	7,044,223,500	1,758,970	160,000	-59,700	120.00	Coro Mining Infill	QA/QC
BR-79	394,969,906	7,044,105,000	1,744,580	160,000	-59,450	100.00	Coro Mining Infill	QA/QC
BR-80	395,001,562	7,044,130,000	1,744,590	160,000	-59,380	84.00	Coro Mining Infill	QA/QC
BR-81	395,108,156	7,044,158,000	1,754,530	160,000	-59,690	120.00	Coro Mining Infill	QA/QC
BR-82	395,053,688	7,044,297,500	1,747,870	160,000	-61,860	100.00	Coro Mining Infill	QA/QC
BR-83	394875.330	7044350.320	1742.046	160	-60	120.00	Coro Mining Infill	QA/QC
BR-84	394919.407	7044373.856	1738.220	160	-60	100.00	Coro Mining Infill	QA/QC
BR-85	394903.440	7044401.560	1734.551	160	-60	120.00	Coro Mining Infill	QA/QC
BR-86	394961.500	7044548.290	1727.232	160	-60	104.00	Coro Mining Infill	QA/QC
BR-87	395094.150	7044753.280	1727.996	30	-60	120.00	Coro Mining Infill	QA/QC
BR-88	395090.530	7044745.410	1727.821	210	-60	120.00	Coro Mining Infill	QA/QC
BR-89	394927.370	7044782.670	1714.122	140	-60	120.00	Coro Mining Infill	QA/QC
BR-90	395261.410	7044824.680	1729.732	220	-60	96.00	Coro Mining Infill	QA/QC
BR-91	395227.120	7044812.880	1726.253	220	-60	86.00	Coro Mining Infill	QA/QC
BR-92	395186.150	7044740.150	1735.920	220	-60	120.00	Coro Mining Infill	QA/QC
					Total	22,213		

11.5.2 QUALITY CONTROL

The MCC Phase 1, 2 and infill drilling campaigns included in the Berta Sur resource estimate, totaling 11,622 m or 81%, have quality control such as standards, duplicates and blanks. This figure corresponds to 3,688 m and 34.5% for Berta Central.

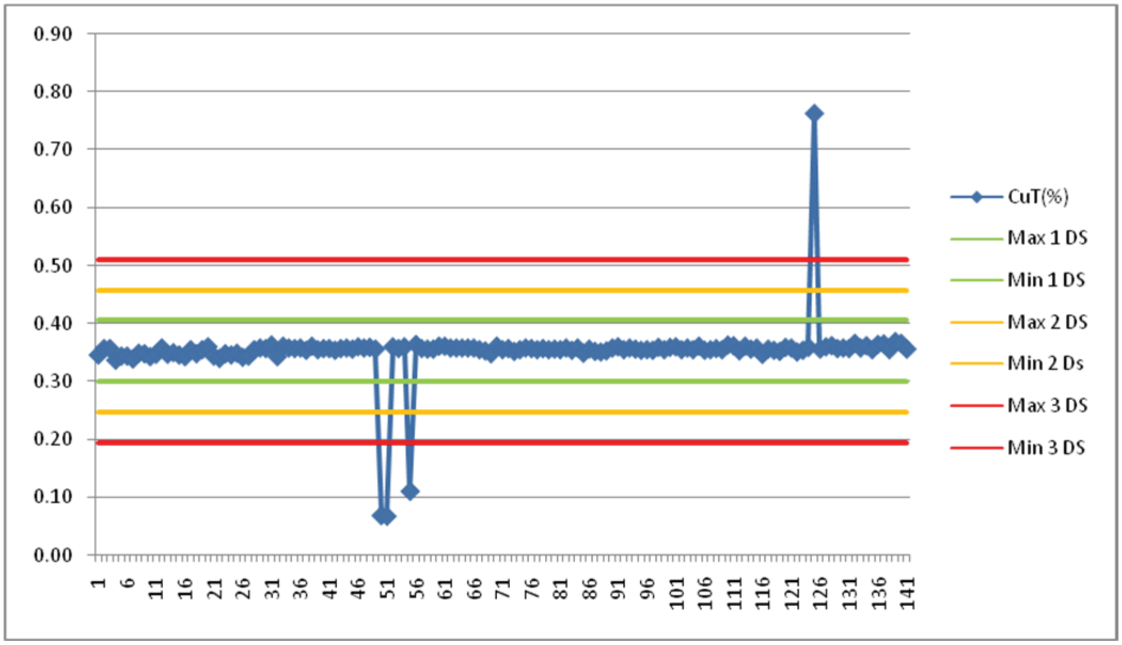
11.5.2.1 STANDARDS

In total, fourteen (14) different standards where utilized by MCC in all Berta drilling campaigns, which were prepared specially for this task, These, which represent different grades of copper, where inserted at the request of the geologist on duty according to the estimated copper grade observed at the drillings logging stage. In total, 491 standards where utilized, representing 4.2% of the drilled meters or 8.4% of the total samples sent to the laboratory, considering that each one comprises a 2 m sample.

The amount of different inserted standards, in most cases, is sufficientto perform a formal statistical analysis. Nevertheless, in the case of examples such as GBM-301-7 and GBM-303-5, with 5 and 6 data respectively, it is not possible to obtain a graph that shows a minimum data sequence between first, second or third standard deviation. These eleven (11) elements represent 130 samples or 260mdrilled, a little over 2% of MCC total drilled meters for Berta. In any event, visually the values are considered very acceptable. For GMB-301-7 case the mean is 0.5942 with a standard deviation of 0.045, which signifies that there are at least four data points within the first standard deviation. For the GMB-303-5 case the mean is 0.6213 with a standard deviation of 0.02, which signifies that there are five data points within the first standard deviation.

This first data visual analysis shows a tendency observed in all MCC drilling for Berta QA/QC database. The great majority of the values obtained for the used standards fall within the first standard deviation. A typical example is GMB-995-4 standard (Figure 11.4), with 141 data that represent 1,669 samples or 3,338 meters drilled, meaning 29% of the drilling campaign.

Figure 11.4: Standard GBM-995-4 distribution used in Berta



This graph is also a good example of the few inconsistencies found in the obatined values for the standards used at Berta. There is no chance of the existence of a series of events that permits suspicion of a severe analytical problem, but there are some very low values, that suggest a possible error in the standard physical identification or its incorrect input into the database. To the four inconsistencies in the GMB-995-4 standard, corresponding to batches GEL-148(2), GEO-159(1) and GEL-23(1), are added one in the GBM-309-2 (batch GEL-310), GBM-396-1 (batch GEL-107), GBM-905-12 (batch GEL-23) and GBM-998-4 (batch GEL-23). These 8 isolated data points, outside the third standard deviation, represent 95 samples or 190 m of drilling, meaning 1.6% of the meterage completed by MCCfor Berta, which is considerd totally acceptable. Batch GEL-23 is mentioned three times, which needs to be reviewed.

Standard GBM-307-13 shows an atypical distribution to the rest of the statistical population, initially with a squence of ten data points with a tendency to low values between the first and second standard deviation, and thenat the end, another sequence of 28 values above the mean (Figure 11.5). Values are very low, but are well over the analytical limit. GMB-307-13 standards were analyzed between GBM-396-4, 999-4, 995-4, 309-2 and 3961, which show very favourable tendencies, normally within the first standard deviation. GBM-307-13 deviation cannot be clearly explained, for which it is recommended to evaluate the importance of the involved samples for the resource estimate, in order to decide if an eventual re-analysis is required.

Figure 11.5: Standard GBM-307-13 distribution used in Berta

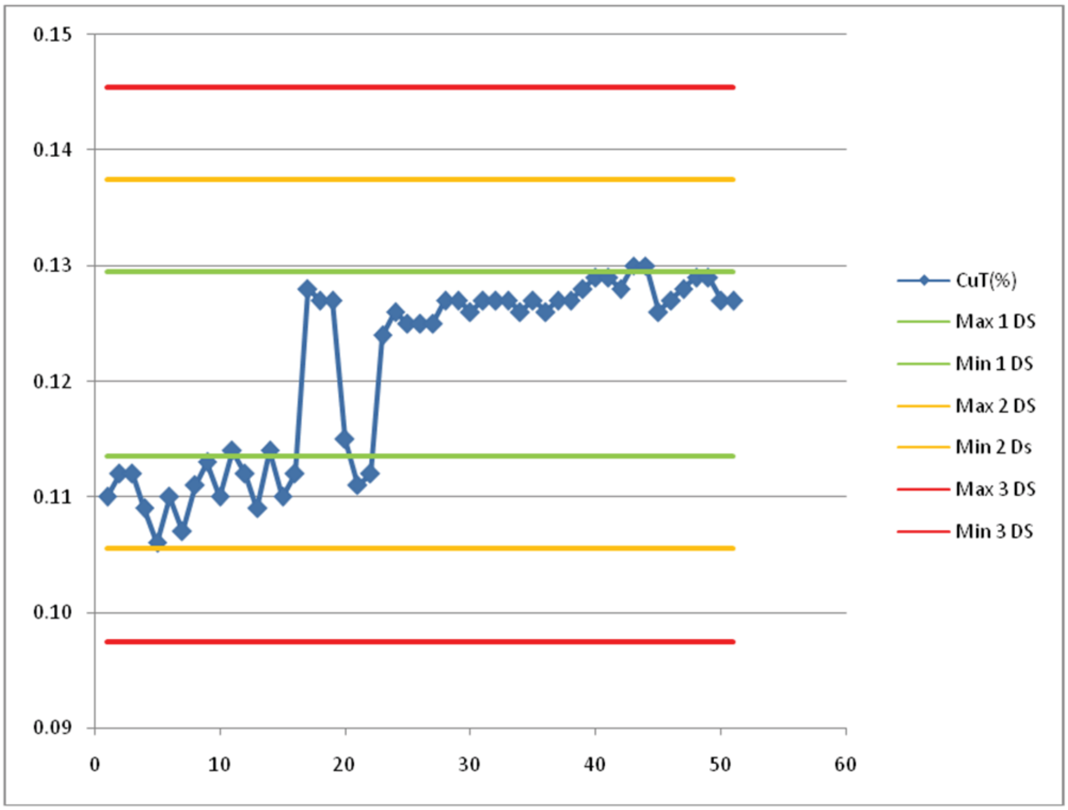


Figure 11.6: Standard GBM-394-4 distribution used in Berta

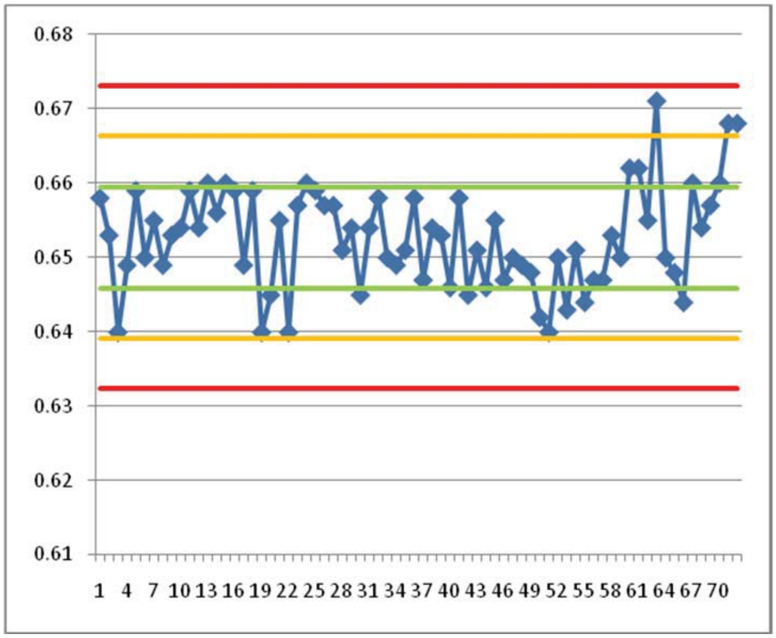
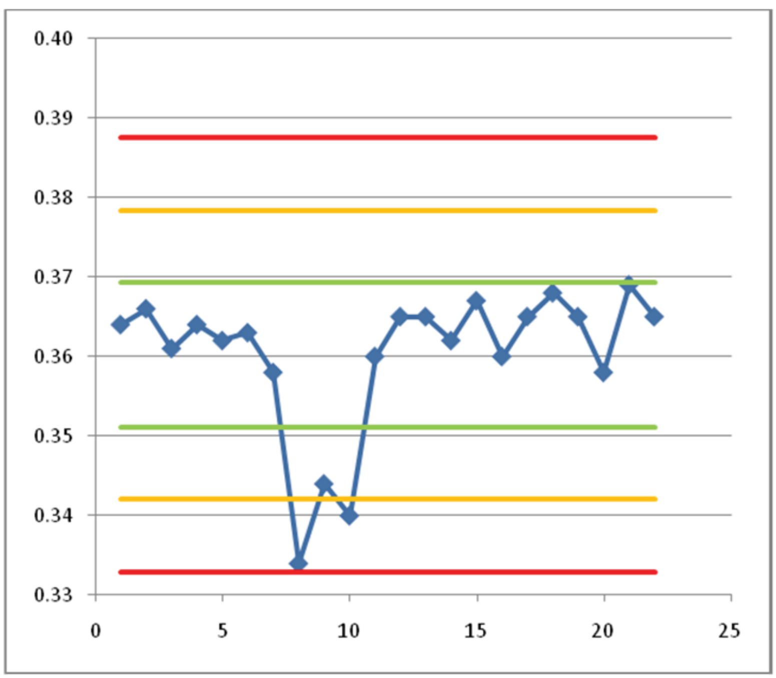


Figure 11.7: Standard GBM-908-10 distribution used in Berta



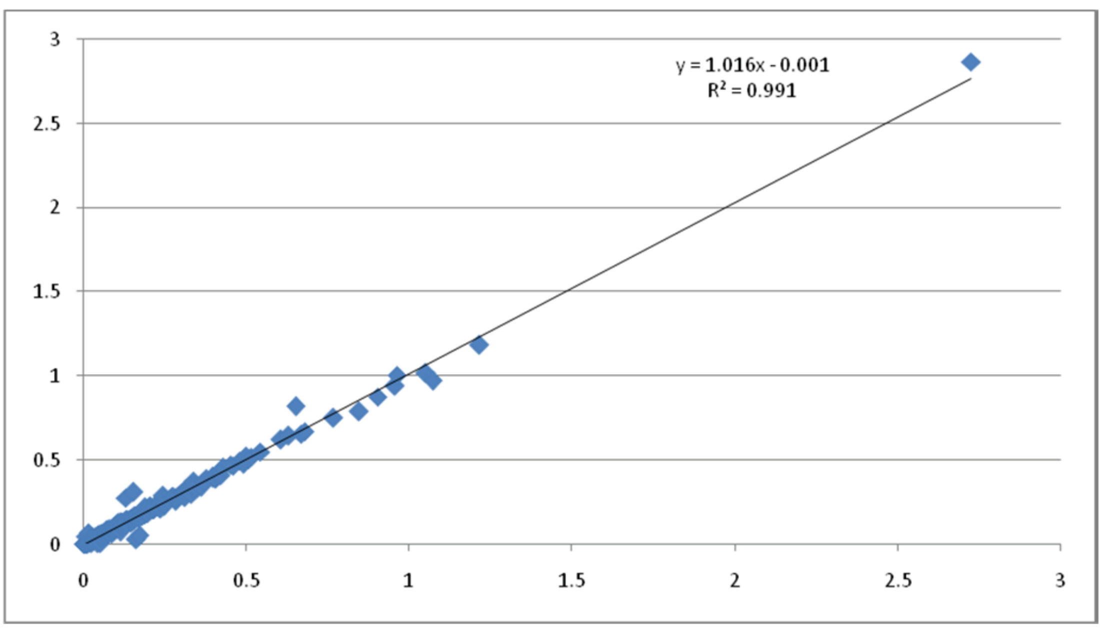
11.5.3 DUPLICATES

Berta duplicates base is varied, with field, core and coarse duplicates for copper (Cu), molybdenum (Mo) and gold (Au), although regrettably they do not have continuity between 2011 and 2012 campaigns. The most complete, but with the lesser amount of data, is the 2011 drilling campaign, which is due to the initial analysis for by-product metals.

11.5.3.1 FIELD DUPLICATES

Field duplicates include 489 data, which is equal to 8.4% of the samples, considering that samples were taken every 2 m and that values of samples under the detection limit were eliminated from the study. The difference between the mean of the original copper sample and the one from the duplicate is 0.00099, that is, it is under the detection limit, while the correlation between both values define a straight line with a 1.0161 slope, starting very close to the origin (-0.001), Figure 11.8.

Figure 11.8: Correlation between original sample and field duplicates values



Correlation between minimum and maximum values obtained from comparing the orginal and duplicates values (Figure 11.9), also define a straight line with a slope close to 1 (1.0425) starting very close to the origin (+0.002). The relative error shows significant error percentages of almost 150%, but that in general they ten to be concentrated in the lower grade mean values, less than 0.11% Cu (Figure 11.10) The absolute relative error value cumulative curve shows that 81% of the data present less than 10% error (Figure 11.11). About the base for this graph and establishing a reject of 20%, 39 data points or 8% of the population are above this error percentage, which is considered very acceptable. With a 10% reject criteria, 90 data points or 20% of the data are rejected. Considering 15% as the reject criteria, then 53 data points or 11% of the population are rejected, which is also considered very acceptable.

The previous analysis indicates that the duplicates control is within the acceptable error range. Some isolated error 100-150% values, corresponding to the batches GEL-437, 148, 292, 309 and 23, are possibly related to physical error in the range or incorrect sample assigning, for which it is suggested they be reviewed with the objective of improving this analysis results.

Figure 11.9: Correlation between minimum and maximum values

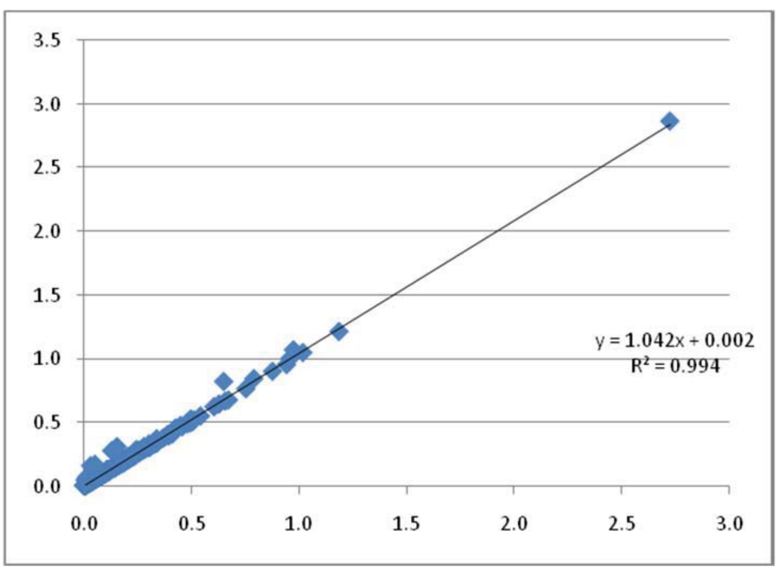


Figure 11.10: Relative error dispersion according to mean grade

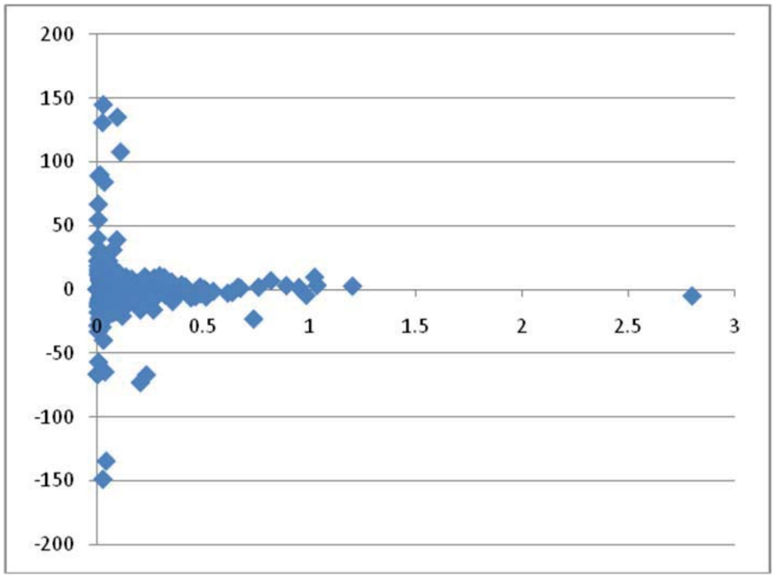
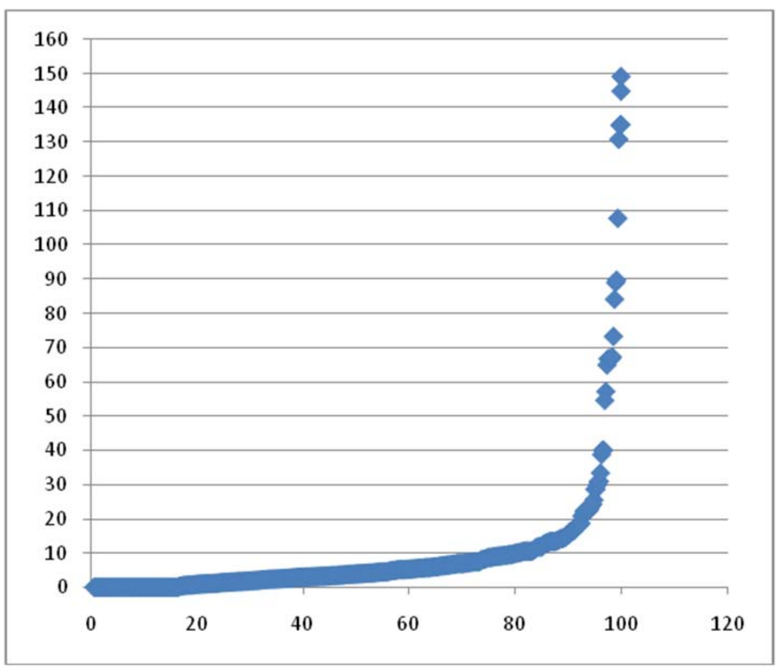


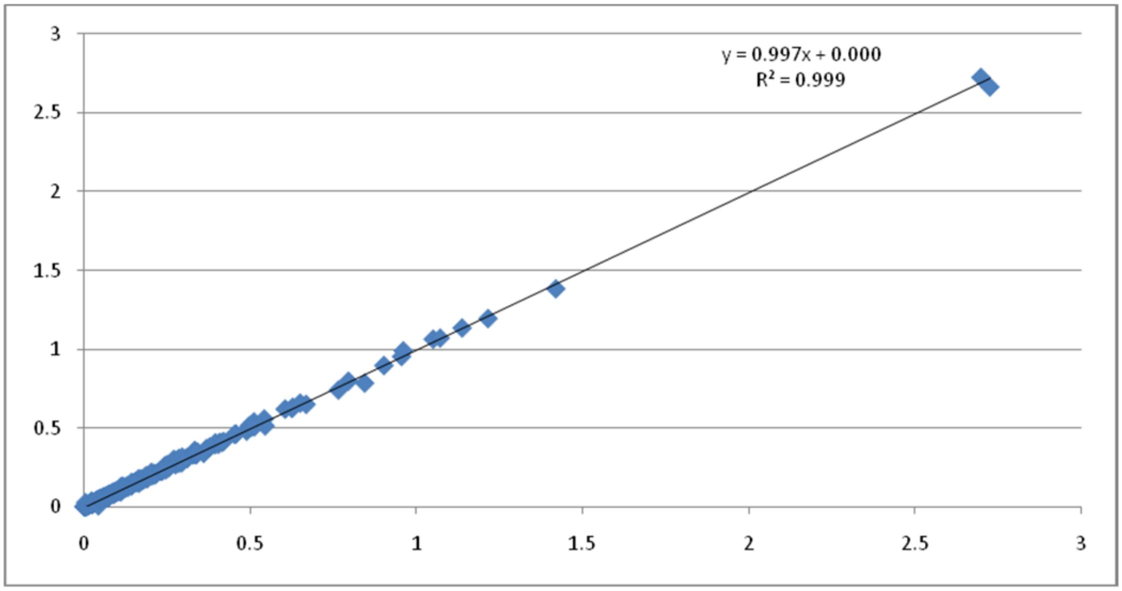
Figure 11.11: Absolute relative error value cumulative curve for field duplicates



11.5.3.2 CORE DUPLICATES

Core duplicates population has 506 data points, which represents 8.7% of the samples sent to analysis in the Berta project. The difference between the original and core duplicate value, in absolute value, is 0.00035, under the 0.001% Cu detection limit. Correlation between both values define a curve with a 0.997 slope which starts very close to the origin, in +0.0008 (Figure 11.12)

Figure 11.12: Correlation between original sample and core duplicates values



Correlation between minimum and maximum values obtained from comparing the orginal and core duplicates values (Figure 11.13), also define a straight line with a 1.0168 slope starting very close to the origin (+0.0011). The relative error shows punctual errors up to140%, but that in general they tend to be concentrated in the lower grade mean values, less than 0.025% Cu (Figure 11.14). The absolute relative error value cumulative curve shows that 92% of the data points present less than 10% error). About the base for this graph and establishing a reject of 20%, 21 data points or 4.1% of the population are above this error percentage, which is considered very acceptable. With a 10% reject criteria, 42 data points or 8.3% of the data are rejected, which is also considered very acceptable.

The previous analysis concludes that the core duplicate control is within the very acceptable error range, better than thatobtained for the field duplicates. Some isolated error 100-140% values, corresponding to the batches GEL-148 (2) and 120, possibly are related to physical error in the range or incorrect sample assigning, for which it is suggested they be reviewed with the objective of improving this analysis results.

Figure 11.13: Correlation between minimum and maximum values

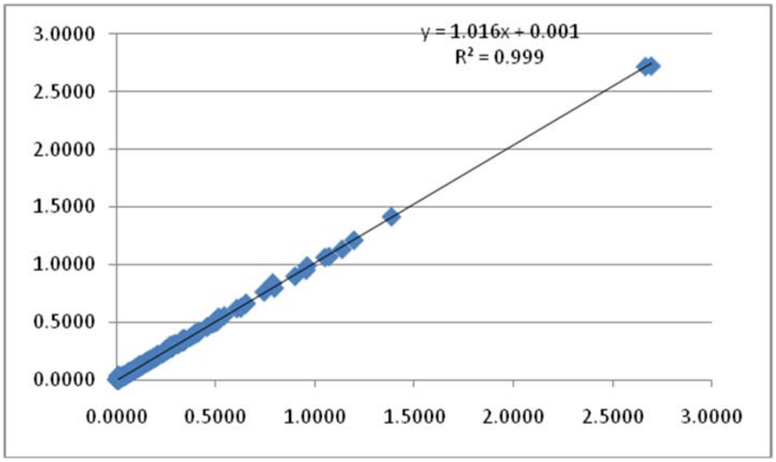


Figure 11.14: Relative error dispersion according to mean grade

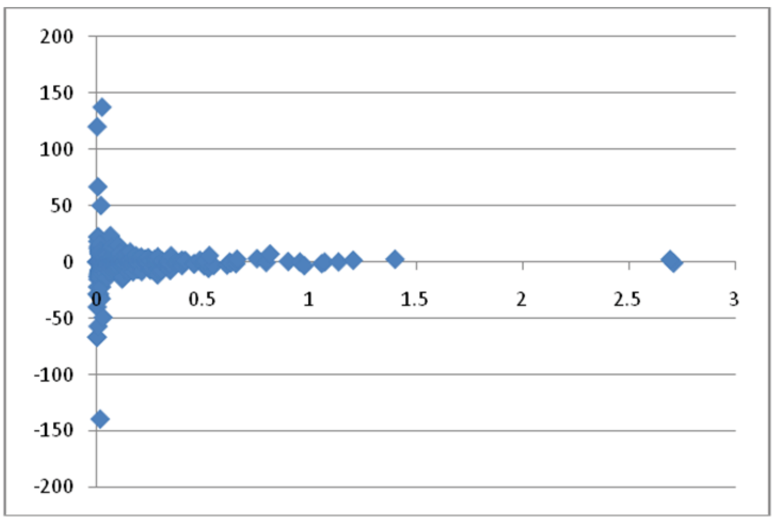
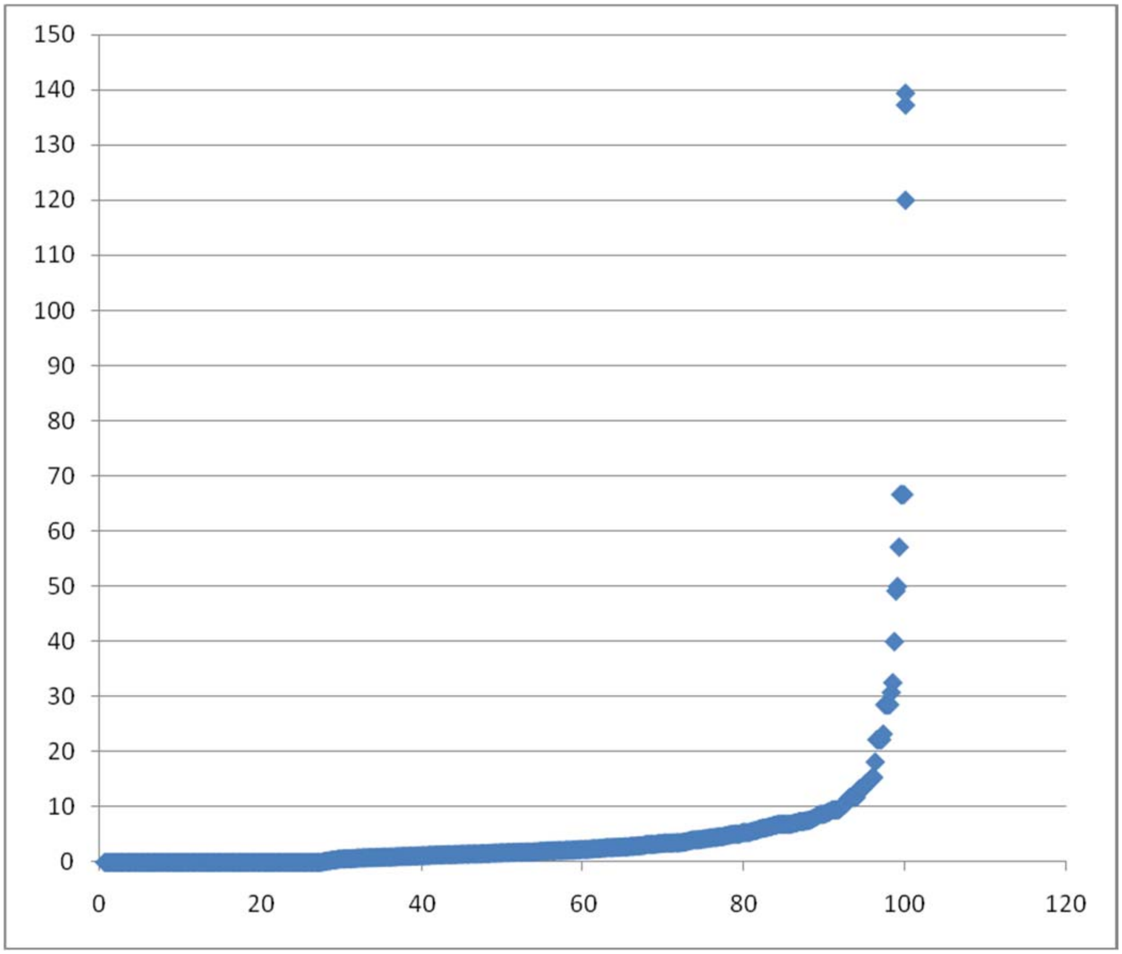


Figure 11.15: Absolute relative error value cumulative curve for core duplicates



11.5.3.3 BLANKS

Only during first drilling campaign of 2011 were blanks inserted in the sampling sequence. Regrettably, this practice was not carried on during 2012 campaigns, during which only laboratory duplicates were used. The first ones represent 0.8% of the project total samples amount, which are insufficient to perform a complete statistical analysis. In the second case, the corresponding values have not been reported, such that it is not possible to complete this item.

12.0 DATA VERIFICATION

The steps taken by Geoinvestments to verify the data in the technical report are as follows;

(a) The data verification procedures applied by the qualified person

- The overall integrity and internal consistency of the database was checked when preparing the data for estimation.
- The nine diamond drill holes completed by Grandcru were verified to be in a very good state of conservation and had been sampled using a diamond saw.
- Geoinvestment verified the existence of the drill chip trays for logging, as well as the corresponding labeled and sealed bags of sample rejects from the MCC drilling campaign.
- Geoinvestment's analysis of the standards, blanks and duplicates employed by MCC described in Section 11, has verified that results are within acceptable error ranges. Very little adjustment was necessary, following the visit, to adjust some items of the database found to be incorrect. All problems were corrected with the prompt help of the site personnel.
- Geoinvestment verified that all MCC samples were sent to recognized, industry standard assay laboratories in Chile.

(b) any limitations on or failure to conduct such verification, and the reasons for any such limitations or failure;

- Information from campaigns of previous operators is not well supported by hard copy documentation. However, one of the contributors to the first Berta resource estimate technical report issued in January 2013 participated in the original Mantos Blancos (a subsidiary of Anglo American plc) drilling campaign and verified that this program's drilling, sample collection and assaying was performed under very high quality standards and sampling protocols. The Mantos Blancos drilling represents ~15% of all of the drilling on the property.
- No other limitations were placed on Geoinvestment's verification of the database and there was no failure to conduct such verification

(c) the qualified person's opinion on the adequacy of the data for the purposes used in the technical report.

- In general, the database is considered adequate and in accordance with international standards. MCC continues to maintain an orderly database and filing systems with all the relevant information separated by drill hole

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Sections 13.1 and 13.2 summarize the initial metallurgical test work for samples from Berta Sur, completed at Geomet SA, an independent metallurgical laboratory located in Santiago, Chile. Section 13.313.3summarizes the additional metallurgical test work for samples from Berta Central completed at Hydrometallurgical Lab of the Universidad de Santiago of Chile Metallurgical Mining Engineering Department

13.1 BERTA SUR METALLURGICAL TESTING

Mineral and chemical characterization and a campaign of metallurgical leaching test work were undertaken, with the objective of defining the main process variables, such as copper recovery and acid consumption. For the metallurgical tests, SCMBselected three composite samples from the Berta Sur deposit, denominated as A, B and C with approximate CuT grades of 0.80%, 0.60% and 0.40%, respectively.

Based on these composites, Geomet performed the metallurgical program designed by SCMB.

The work program developed by Geomet was the following:

13.1.1 HEAD SAMPLE MECHANICAL PREPARATION

For performing the leaching tests, MCC sent Geomet three composite samples of 200 kg each, with preparation consisting of material crushing, from the received granulometry of 100% - 3", to two granulometry levels of 100% - 1" (P_{80} = 19 mm), and 100% - ½" (P_{80} = 9 mm), by means of a jaw crusher.

The crushed material was homogenized and split in a Jones cutter, according to the metallurgical program requirements, obtaining an approximately 5 kg sample from each particle size, for the corresponding granulometric analysis.

An approximately 5 kg sub sample was crushed to 100% - 10# mesh, from which another approximately 320 g were obtained for pulverization to 100% - 150# mesh, to complete head sample chemical characterization.

13.1.2 HEAD SAMPLE CHEMICAL ANALYSIS

The 5 kg increment with granulometry 100% - ½", was divided through three binary splits until obtaining an approximately 320 g sample. This sample was pulverized to 100% -150#, by means of a concentric rings pulverizer, in order to perform the CuT, CuS, FeT, Al, Mg, Cl and AAC (Analytic Acid Consumption) analysis. While determining the CuS, the level of the following contaminants (FeT, Fe^{+2}, H^{+}, Al, Mg, Mn, Ca, SO_4) was also determined.

In order to determine the level of contaminants, a 31 element ICP analysis was completed on each head sample.

13.1.3 HEAD SAMPLE MINERALOGICAL CHARACTERIZATION

Each sample was characterized from a mineralogical point of view, by means of optical microscopy, determining the constituents ofmineral materialand gangue. This characterization was performed by Mrs Franco Barbagelata of MAM Limited.

13.1.4 HEAD SAMPLES PHYSICAL CHARACTERIZATION

Samples were physically characterized, before starting the process of size reduction to specific granulometries. This characterization stage comprised: granulometry analysis of both granulometries, sample humidity at sample reception, specific gravity, and bulk density at both granulometries.

- **Granulometry**:

Assaying for %CuT and %CuS,in dry conditions of the ten fractions obtained (1", ¾", ½", ¼", 6#, 10#, 35#, 65#, 100# and -100# mesh) was completed.

- **Sample Humidity:**

Humidity was determined at the time of arrival of materials at the Geomet laboratories. Material humidity was calculated through the determination of dry and humid weights, obtaining the humidity as the difference between the dry and humid weights, expressing this difference as a percentage of the dry weight.

- **Specific Gravity:**

Material specific gravity was determined three times, using the pycnometer technique.

- **Bulk Density:**

Material bulk (or apparent) density was determined in a 50 liter rectangular base parallelepiped, which was filled with the material to test, leveling the top with a wooden ruler. Then, the weight of the content was determined, calculating the material density by dividing the mass by the recipient volume. This determination was performed for both granulometries at the moment of reception.

This was also determined at each column leaching test, to the filling humidity, given that the humid material was weighed, determining the initial filling height. At the moment of unloading the column, the final height and humid unloaded material weight were measured, thus determining the material bulk density at the moment of unloading.

13.1.5 PHASE I. PRELIMINARY METALLURGICAL TESTS

At the beginning of the metallurgical program, preliminary tests were performed, with the objective of obtaining leaching metallurgical parameters, in order to establish the most appropriate experimental conditions for larger scale testing (pilot leaching columns).

- **Contaminant Determination Test:**

This test has as objective to perform a first estimation of the contaminants equilibrium in a comprehensive leaching-SX process. In this test 10 g of 100% - 150 # mesh material, were subjected to agitation leaching, with acid solution (1 N). After 24 hours, the solution was filtered and analyzed for Cu, FeT, Fe^{2+}, H^+, Al, Mg, Mn, Ca and SO_4.

- **Iso pH (Bottle Roll) Test:**

These tests were performed using material 100% -10 # mesh, in a 48 h period, using 1 kg of material and 33% Cp (solids percentage). Tests were performed in a 10 liter capacity plastic container, which turns over a roller at 55 r.p.m., specially designed for such work. The leaching solution was kept at all times at a 1.5 pH, achieved through the constant and permanent adding of acid, reported as net and rough acid consumption.

Pulp samples were taken at a rate of 2, 4, 6, 8, 10, 24, 48 and 72 h, to keep records of the copper extraction kinetics and acid consumption. At the end of the leaching period, pulp was filtered and washed obtaining rich and washed solutions. The mineral residue was dried, weighed and assayed for CuT, FeT, Al, Mg and Mn. The dissolution of the contaminants, Al, Mg, and Fe (expressed as a percentage or in solubilized kg/t), was also determined from the resulting solution.

Finally, based on weights, solution volumes and chemical analysis, the metallurgical balance was completed.

Of the three samples, two were leached for 48 h, while the third was leached for 72 h Main contaminant elements, Fe, Al, and Mg were tested in all final rich solutions.

- **Sulfation Tests:**

They are used to determine the acid dose to employ in the column leaching tests. This experiment integrates one set of four sulfation tests which, according to Geomet’s proposal, take a 24 h -36 h resting time and a determined pivot point from the previous background (Iso-pH Tests).

From the results of the four sulfation tests, the cure acid dose was determined, based on the principle of using the least amount of acid after which there is acid remaining. This amount of acid was used to cure the material before the column leaching stage.

13.1.6 PHASE II. COLUMN LEACHING TESTS

Column leaching tests have the objective of obtaining the first metallurgical parameters, for the Project's conceptual engineering level estimate.

The metallurgical program included performing leaching tests in 4" diameter (100 mm) and 2 meters high columns, for each of the grain sizes. The irrigation rate was 10 $l/h/m^2$. Each test was performed in duplicate; therefore, it was required to set up twelve columns in total.

Tests were irrigated until completion of the leaching rate of 2 m^3/t, equivalent to 25 leaching days; including daily analysis for Cu, FeT and H^+, during the first eight days, then on an every other day basis, until the completion of irrigation. Thus, for each leaching test 18 samples were taken for kinetic evaluation, including the final drain solution.

In order to validate the contaminant elements kinetics, weekly composites were taken and assayed by ICP (three in each test).

13.2 RESULTS

Results of the different study stages were as follows:

13.2.1 HEAD SAMPLES CHEMICAL CHARACTERIZATION

Chemical characterization of the head was comprised of CuT, CuS, FeT, Al, Mg, Mn, Cl and AAC (Analytic Acid Consumption), for each of the three samples, as well as for each of the two utilized grain size levels. In the determination of CuS (Sulf.), in addition to the soluble copper, the contaminant element levels (FeT, Fe^{+2}, H^+, Al, Mg, Mn, Ca and $SO_4^{=}$) were determined. Results are shown in Tabla 13.1while for purposes of quantifying the contaminant elements, an ICP analysis was performed, with results shown inTable 13.2

Tabla 13.1: Head Chemical Characterization

ELEMENT	Unit	Sample A		Sample B		Sample C	
		3/4"	3/8"	3/4"	3/8"	3/4"	3/8"
CuT	%	0.83	0.84	0.60	0.66	0.40	0.38
CuS (Sulf.)	%	0.58	0.59	0.29	0.36	0.15	0.14
CuS (Citric)	%	0.50	0.43	0.07	0.11	0.10	0.09
CuS (Fe^{+++})	%	-	-	0.314	0.377	-	-
CuS (Bisulfite)	%	-	-	0.350	0.404	-	-
FeT	%	2.25	2.07	2.03	2.05	1.80	2.02
Al	%	4.94	6.11	8.50	5.89	7.61	6.92
Mg	%	0.25	0.25	0.43	0.37	0.50	0.59
Mn	%	2.07	1.55	0.92	0.92	0.89	0.94
Cl^-	%	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
A.A.C.	kg/ton	75.64	87.94	87.22	87.91	80.20	72.54
Fe Sol.	%	0.16	0.14	0.14	0.17	0.82	0.21
Fe^{++}	%	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Al Sol.	%	0.08	0.10	0.10	0.11	0.12	0.13
H^+	gr/L	42.02	48.08	48.19	42.58	42.01	40.00
Mg Sol.	%	0.02	0.02	0.03	0.04	0.04	0.04
$SO_4^=$	%	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1
Mn Sol.	%	0.05	0.03	0.02	0.02	0.00	0.00
Ca Sol.	%	0.22	0.20	0.29	0.25	0.21	0.19

Table 13.2: ICP Analysis Results

Element	Unit	Sample A		Sample B		Sample C	
		3/4"	3/8"	3/4"	3/8"	3/4"	3/8"
Al	%	1.24	1.29	1.43	1.54	1.52	1.57
Ca	%	0.45	0.45	0.55	0.6	0.62	0.58
Fe	%	2.94	2.82	2.72	2.81	2.53	2.57
K	%	0.14	0.13	0.16	0.18	0.17	0.2
Mg	%	0.450	0.450	0.580	0.640	0.800	0.780
Na	%	0.07	0.07	0.08	0.09	0.1	0.1
S	%	0.02	0.02	0.02	0.03	0.02	0.03
Ti	%	0.03	0.03	0.07	0.08	0.09	0.08
Cu	gr/t	8,212	8,182	5,629	6,318	3,647	3,559
Mn	gr/t	867	635	414	416	394	376
Mo	gr/t	571	658	98	131	35	42
P	gr/t	451	465	581	604	619	559
Zn	gr/t	91	84	52	64	36	36
Sr	gr/t	55	59	57	59	82	81
Ba	gr/t	41	33	173	273	63	95
V	gr/t	39	39	53	55	48	48
La	gr/t	31	35	<10	<10	<10	<10
As	gr/t	24	18	20	33	<5	6
Co	gr/t	23	17	13	12	12	10
Tl	gr/t	15	11	<10	<10	<10	<10
Y	gr/t	9	8	8	9	7	7
Pb	gr/t	7	4	19	50	4	3
Cr	gr/t	6	4	5	5	6	5
Sb	gr/t	6	< 5	< 5	14	< 5	< 5
Li	gr/t	4	4	5	6	7	7
Ni	gr/t	3	3	3	3	2	3
Cd	gr/t	2	2	2	2	1	1
Ag	gr/t	1	2	<1	3	<1	<1
Sc	gr/t	1	4	3	7	2	3

In total four types of soluble copper assay were performed, with results on solubility rates shown in Table 13.3

Table 13.3: Solubility Rates

ELEMENT	Sample A		Sample B		Sample C	
	3/4"	3/8"	3/4"	3/8"	3/4"	3/8"
Citric	60.24	51.19	11.67	16.67	25.00	23.68
Sulfuric	69.88	70.24	48.33	54.55	37.50	36.84
Ferric	-	-	52.33	57.12	-	-
Bisulfite	-	-	58.33	61.21	-	-

Based on the chemical characterization results on the three samples and at each grain size level, it can be stated that regarding the CuT grade they showed the desired grades for metallurgical tests: 0.83% for sample A, 0.63% for sample B and 0.39% for sample C.

Regarding the solubility ratio a high variability in respect of the utilized CuS analysis method was observed, thus the average solubility rate for both granulometries in sulfuric acid showed values of 70.1% for sample A, 50.8% for sample B and 37.6% for sample C. The same rate, but with citric method showed values of 55.4% for sample A, 14.5% for sample B and 24.8% for sample C.

Solubility rates in ferric and sodium bisulfite media were only performed on sample B material, with average material solubility rates on ferric of 54.5%, and bisulfite, 59.5%.

In conclusion, the solubility rate that maximized copper extraction is sodium bisulfite, i.e., in a reducing media. This situation indicates that a part of the oxidized copper would be copper wad ($CuOMnO_2$), a species that is solubilized in reducing media ($FeSO_4$), according to the following reaction:

$$2CuOMnO_2 + 7H_2O + 6H_2SO_4 + 4FeSO_4 \rightarrow 2CuSO_4 + 2MnSO_4 + 2Fe_2(SO_4)_3 + 20H_2O$$

It should be noted that the presence of copper wad would inferred only through the realization of the CuS analysis with sodium bisulfite, a situation that will be further confirmed through the respective mineralogical characterization.

Regarding the Analytic Acid Consumption (A.A.C.), on average for the studied grain sizes, the results were 81.8 kg/t on sample A, 87.6 kg/t on sample B and 76.4 kg/t on sample C. These results cannot be interpreted under any point of view as values to be obtained in the leaching tests, given the big differences in test realization conditions between them.

ICP analyses do not show presence of significant concentration of elements, nevertheless, it is convenient to keep in mind the following: Ca concentration is higher on samples C and B than sample A, which will signify a higher acid consumption on those samples, although the overall obtained values are moderate (< 0.63%). Sulfur presence is very low, which indicates that the material is mostly oxidized. There is an important content of Mn which would indicates the presence of copper wad, which is higher in sample A. The molybdenum content is also important and it is mostly concentrated on sample A. Samples B and C have higher phosphorus content than sample A, whose values are also significant.

13.2.2 HEAD SAMPLES MINERALOGICAL CHARACTERIZATION

For the three analyzed samples mineralogy corresponds mostly to copper oxidized species. For its part, the insoluble copper mineral determined in the samples was generally named as "copper wad?", but it may correspond to other species that may contain copper, manganese, silicon, etc., whose identification can only be performed through EDAX or QUEMSCAN analysis.

Samples show similar mineralogy regarding nonmetallic and opaque minerals, with the principal minerals, such as chalcopyrite and covellite, are at trace levels. Lower concentration levels of magnetite, hematite and limonite, as well as pyrite, were observed at trace levels. Nonmetallic minerals present are clay chlorite, quartz, plagioclase, tourmaline and sericite.

Table 13.4 shows the copper species mineralogy summary, while the general summary is shown in Table 13.5

Table 13.4: Copper Species Mineralogy Summary

Mineral	Formula	Comp-A (weight %)	Comp-B (weight %)	Comp-C (weight %)
Copper Wad	Oxides of Cu, Mn, Si, etc	2.16	2.88	1.51
Chrysocolla	$CuSiO_3.2H_2O$	0.94	0.30	0.25
Malachite	$Cu_2CO_3(OH)_2$	0.15	-	-

Table 13.5: Mineralogical Characterization Summary

Mineral	Formula	Comp-A (weight %)	Comp-B (weight %)	Comp-C (weight %)
Chalcopyrite	$CuFeS_2$	0.004	0.01	0.01
Copper Wad	Oxides of Cu, Mn, Si, etc.	2.16	2.88	1.51
Chrysocolla	$CuSiO_3. 2H_2O$	0.94	0.30	0.25
Malachite	$Cu_2CO_3(OH)_2$	0.15	-	-
Psilomelane	$(Ba,H_2O)_2Mn_5O_{10}$	-	-	-
Pyrite	FeS_2	0.03	0.06	0.05
Magnetite	Fe_3O_4	0.06	0.68	0.22
Hematite	Fe_2O_3	0.65	0.46	0.34
Limonite	FeO(OH)	0.61	0.48	0.32
Titania	TiO_2	0.05	-	-
Clay	$Al_4(Si_4O_{10})(OH)_3$	7.38	8.39	11.51
Chlorite	$(Mg,Al)_3(AlSi_3O_{10})(OH)_2Mg_3(OH)_6$	0.76	0.53	0.70
Illite	$(K,H_2O)Al_2((Al,Si)Si_3O_{10})(OH)_2$	2.57	-	-
Amphibolite	$(X,Y)_{7-8}(Z_4O_{11})_2(OH)_2$	0.44	-	-
Actinolite	$Ca_2(Mg,Fe^{++})5Si_8O_{22}(OH)_2$	0.68	0.63	1.44
Apatite	$Ca_5(PO_4)_3(Cl)$	0.22	0.28	0.20
Calcite	$CaCO_3$	0.24	0.37	0.20
Epidote	$Ca_2Al_2Fe\ Si_3O_{12}(OH)$	0.78	0.73	2.16
Biotite	$K(Mg, Fe)_3(Al\ Si_3O_{10})(OH, F)_2$	1.11	0.97	1.18
Quartz	SiO_2	52.31	55.70	51.36
Plagioclase	$(Ca,Na)(Al,Si)AlSi_2O_8$	5.73	7.31	8.56
Feldspar	$KAlSi_3O_8$	11.99	10.06	8.24
Pyroxene	$(Mg,Fe)_2\ Si_2O_6$	-	-	-
Sericite	$(Na,Ca)(Mg,Fe,Li)_3Al_6B_3Si_6O_{27}(OH)_4$	10.33	9.72	11.33
Total		**100.00**	**100.00**	**100.00**

13.2.3 HEAD SAMPLES GRANULOMETRY ANALYSIS

Head granulometry analysis was performed for each sample and grain size level utilized, as well as CuT and CuS analysis for each granulometry fraction..

13.2.4 HEAD SAMPLES PHYSICAL CHARACTERIZATION

Table 13.6 shows the summary of physical characterization performed on the head samples

Table 13.6: Physical Characterization Summary

Composite	P_{80}	Angle of Repose		Bulk Density	
		Dry (º)	Agglomerated (º)	Dry (g/ml)	Agglomerated (g/ml)
A	3/4”	29	39	1.533	1.846
A	3/4”	29	39	1.533	1.846
A	3/8”	31	38	1.400	1.492
A	3/8”	31	38	1.400	1.492
B	3/4”	29	42	1.455	1.636
B	3/4”	29	42	1.455	1.636
B	3/8”	31	39	1.273	1.448
B	3/8”	31	39	1.273	1.448
C	3/4”	25	35	1.379	1.630
C	3/4”	25	35	1.379	1.630
C	3/8”	33	39	1.348	1.499

The three composites showed no humidity at receipt of samples.

13.2.5 ISO-PH (BOTTLE ROLL) TESTS

Iso.pH tests were performed using material with size 100% -10# mesh, for 48-72 hours at a constant pH of 1.5.

Table 13.7 shows the obtained results.

Table 13.7: Iso-pH Tests Results Summary

Composite	Acid consumption			Cu Recovery (%)	
	Net kg/t	Gross kg/t	Unit kg/kg	Analyzed Head (C.A.)	Calculated Head (C.C.)
A	15.03	22.25	4.75	68.77	73.11
B	13.79	19.74	7.15	61.71	69.45
C	12.98	15.38	9.87	48.35	55.49

Iso-pH tests results were completely compatible with the solubility rate results of each assayed sample. Thus, composite A showed a higher calculated head copper extraction (73%), then composite B (69%) and finally composite C (55%). (Composite B was leached for 72 hours, while composites A and C were leached for 48 hours).

Net acid consumption was 15.0, 13.8, and 13.0 kg/t in composites A, B and C, respectively; equivalent to gross acid consumption of 22.3, 19.7, 15.4 kg/t, respectively.

Figure 13.1 shows the copper dissolution kinetics for each sample, where it can be observed that sample A has the fastest dissolution rate, followed by sample B and finally sample C, a situation illustrated more clearly in

Figure 13.2, which shows a comparison of the three samples kinetics. It can be observed that there are similarities between samples B and C kinetics, but they have significant differences with respect to sample A kinetics. Figure 13.3 shows acid consumption.

Figure 13.1: Iso-pH Tests Copper Dissolution Kinetics

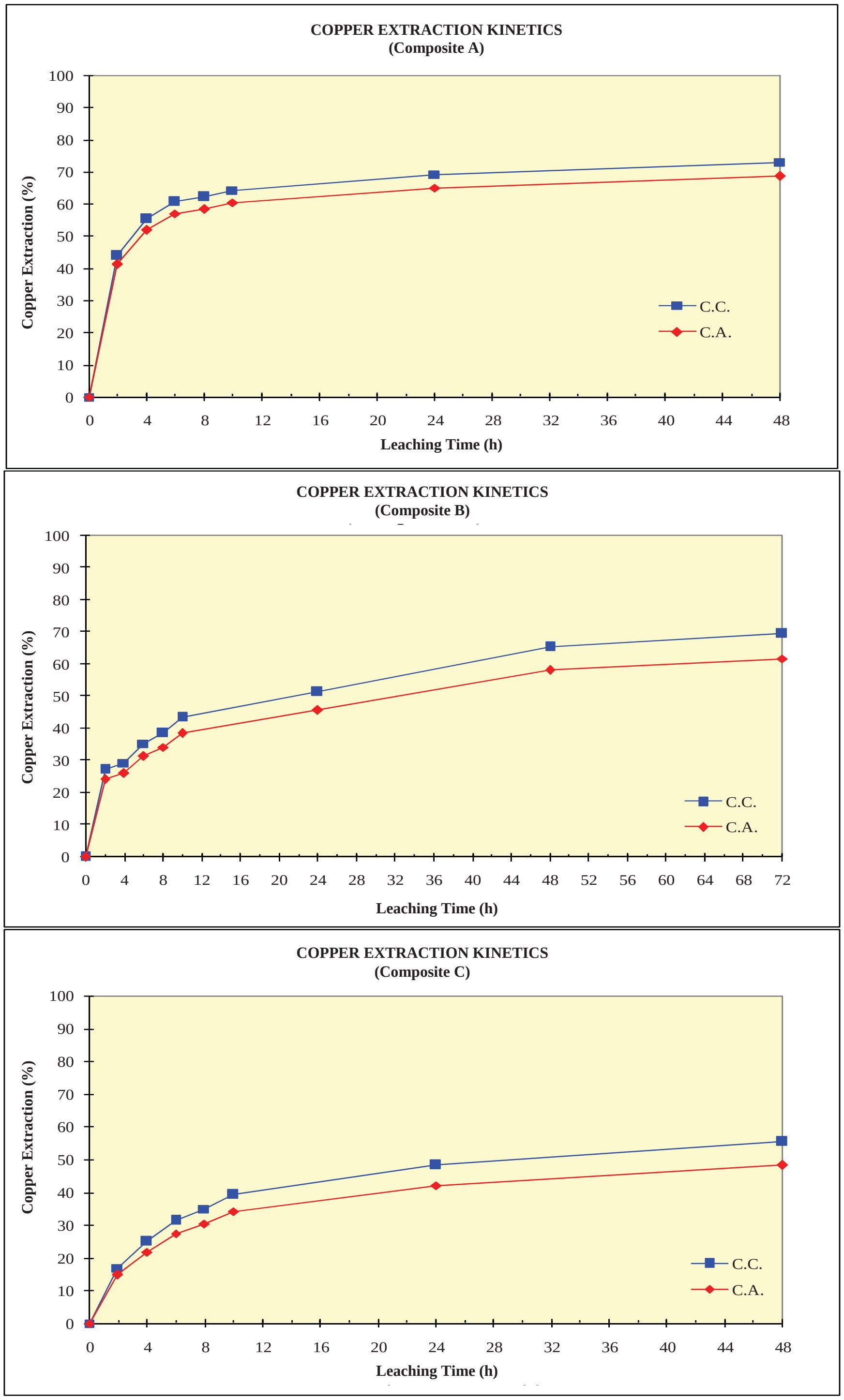


Figure 13.2: Iso-pH Tests Copper Extraction Kinetics

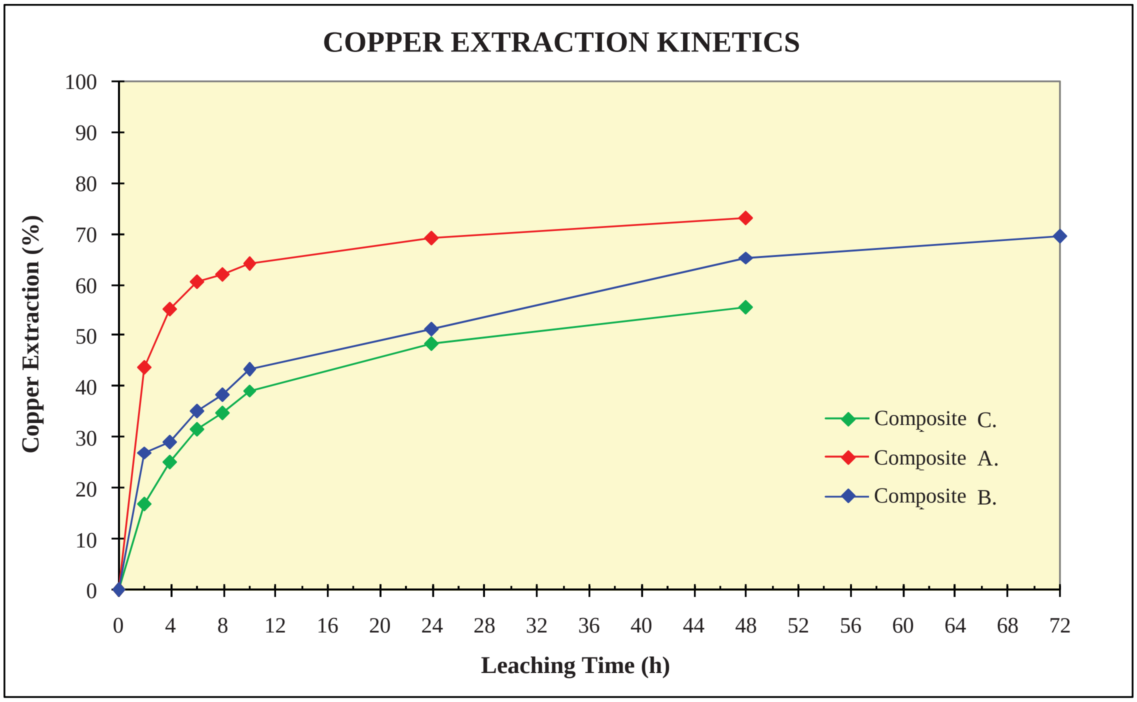
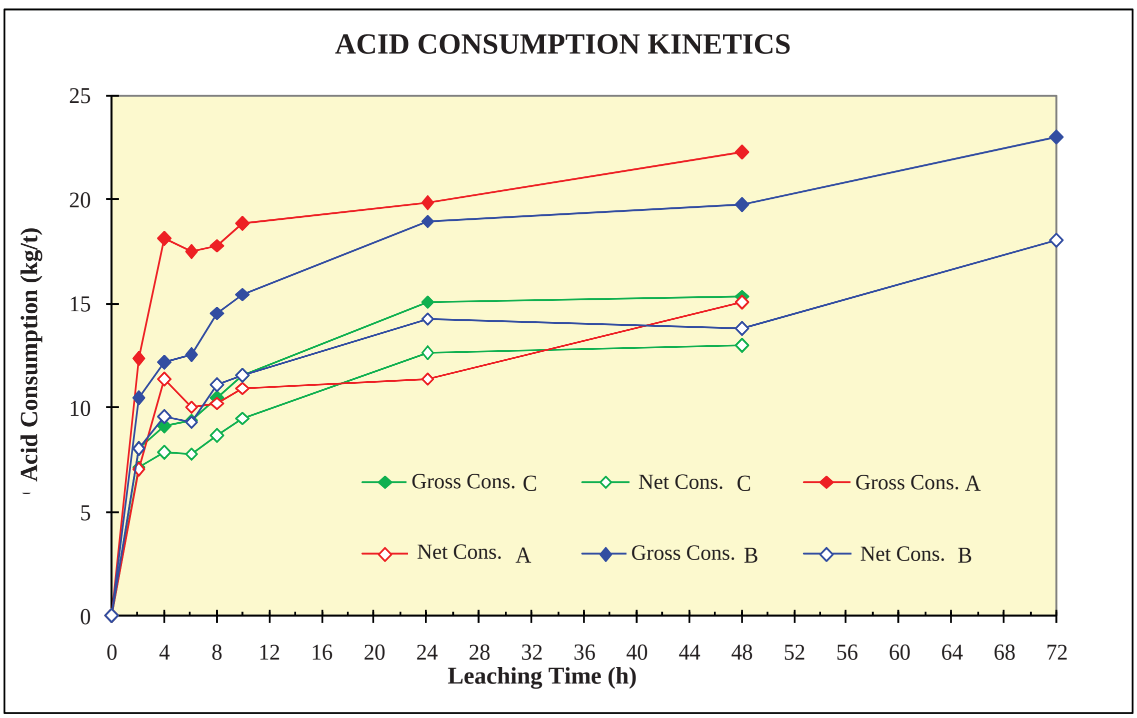


Figure 13.3: Iso-pH Tests Acid Consumption Kinetics



13.2.6 SULFATION TESTS

Taking as reference the values for gross acid consumption obtained in the Iso-pH tests, the acid ranges to use in the sulfation tests were defined, taking as central or pivot point, the Iso-pH tests values. The dose values to assay are shown in Table 13.8

Table 13.8: Acid Dose to assay in Sulfation Tests

Dose Nº	Acid Dose to assay (kg/t)					
	Comp. A		Comp. B		Comp. C	
	¾"	3/8"	¾"	3/8"	¾"	3/8"
Dose 1	12	12	12	12	8	8
Dose 2	17	17	17	17	14	14
Dose 3	23	23	23	23	20	20
Dose 4	29	29	29	29	26	26

Tests were performed according to the methodology stated in section 13.1.5 obtaining the values shown in Table 13.9. In order to minimize acid consumption, Geomet was requested to decrease the obtained value from composite A tests, to a lower value also shown in Table 13.9 Accordingly, composites A and B used a curing dose of 12 kg/t in both grain sizes, while composite C used 8 kg/t, also for both granulometries.

Table 13.9: Acid Dose for Curing

Curing Dose	Acid Dose to Cure (kg/t)					
	Comp. A		Comp. B		Comp. C	
	¾"	3/8"	¾"	3/8"	¾"	3/8"
Dose from Sulf. Test	17	23	12	12	8	8
Dose to use in Curing	12	12	12	12	8	8

13.2.7 COLUMN LEACHING TESTS

Column leaching tests were performed for each of the three samples at granulometries of P_{80} ¾" and ⅜". Furthermore, each test was performed in duplicate, using 2 m high and 8" in diameter columns, with a leaching rate of 2 m^3/t.

Table 13.10 shows the summary of obtained results, while the results are analyzed as follows:

13.2.7.1 COPPER DISSOLUTION

Figure 13.4 shows the copper dissolution kinetics for all performed tests, with three well defined extraction levels identified. The first of them, for extraction values between 78 and 73%, corresponds to composite A, for P_{80} of ¾" as well as ⅜", and composite B for P_{80} of ⅜".

The second level, for copper extraction values between 61 and 65%, correspond to composite B for P_{80} of ¾" and composite C for P_{80} of ⅜". Finally, the third level with copper extraction value of 55% is confined exclusively to composite C, for P_{80} of ¾".

Figure 13.5 shows copper extraction tests kinetics for composite A, where it can be observed that there is no great difference in terms of extraction for material with P_{80} of ¾" or ⅜". All tests presented the same kinetic profile.

Table 13.10: Column Leaching Test Results Summary

	Parameters	Unit	Samples											
HEAD	Mineral Composite	Id	A	A	A	A	B	B	B	B	C	C	C	C
	Granulometry (Grain Size)	P_{80} Inch	3/4	3/4	3/8	3/8	3/4	3/4	3/8	3/8	3/4	3/4	3/8	3/8
	Total Copper Grade	%	0.83	0.83	0.84	0.84	0.60	0.60	0.66	0.66	0.40	0.40	0.38	0.38
	Soluble Copper Grade	%	0.58	0.58	0.59	0.59	0.29	0.29	0.36	0.36	0.15	0.15	0.14	0.14
	Solubility Grade	%	69.88	69.88	70.24	70.24	48.33	48.33	54.55	54.55	37.50	37.50	36.84	36.84
	-100 # Mesh	%	8.1	8.1	11.0	11.0	7.0	7.0	8.7	8.7	7.6	7.6	8.8	8.8
OPER. CONDITION	Irrigation Type	-	Cont.	Cont.	Cont.	Cont.	Cont.	Cont.	Cont.	Cont.	Cont.	Cont.	Cont.	Cont.
	Bed Height	m	2	2	2	2	2	2	2	2	2	2	2	2
	Cure Acid Dose	kg/t	12	12	12	12	12	12	12	12	8	8	8	8
	Repose Time	days	2	2	2	2	2	2	2	2	2	2	2	2
	Leaching Time	days	31	31	31	31	31	31	31	31	31	31	31	31
	Leaching Rate On	m^3/t	2.01	2.05	2.11	2.02	2.05	2.04	2.06	2.03	2.00	2.02	2.01	2.08
	Leaching Rate Off	m^3/t	1.90	1.93	2.00	1.83	1.86	1.95	1.93	1.90	1.90	1.92	1.89	1.97
	Irrigation Solution	-	Ref. Art	Ref. Art	Ref. Art	Ref. Art	Ref. Art	Ref. Art	Ref. Art	Ref. Art	Ref. Art	Ref. Art	Ref. Art	Ref. Art
	Irrigation Solution Acidity	g/l	10	10	10	10	10	10	10	10	10	10	10	10
RUBLES	Total Copper Grade	%	0.24	0.24	0.23	0.23	0.23	0.20	0.17	0.17	0.18	0.18	0.14	0.14
	Impregnation Humidity	%	11.17	10.62	12.95	10.47	10.68	10.34	11.70	9.43	8.74	9.63	12.91	9.52
	Bed Compacting	%	10.10	8.00	11.00	9.00	9.00	10.50	8.50	9.50	8.50	8.50	9.00	9.50
	Wieght Loss	%	3.89	1.68	3.57	3.00	2.00	1.94	1.87	2.01	1.79	1.69	1.74	1.45
RESULTS	Metallurgical Accounting	%	104.3	108.1	119	109.4	99.1	94.2	94.4	92.6	98.3	100.3	92.4	90.3
	Copper in Rich Solution	g/l	3.52	3.81	4.03	3.96	2.23	2.12	2.58	2.58	1.41	1.43	1.40	1.39
	Free Acidity in Rich Solution	g/l	2.42	1.78	0.81	1.25	2.12	2.39	1.62	2.00	2.34	2.81	2.40	2.71
	Copper Extraction (CuT)	%	72.9	74.3	78.0	76.2	62.4	65.1	72.9	76.6	54.8	55.9	60.6	60.5
	Gross Acid Consumption	kg/t	27.7	29.6	31.7	30.1	28.6	28.1	29.5	28.6	23.6	22.8	23.5	23.8
	Net Acid Consuption	kg/t	18.0	19.3	19.6	19.2	22.8	22.1	22.5	21.6	20.3	19.4	20.2	20.4

Figure 13.4: Copper Extraction Kinetics

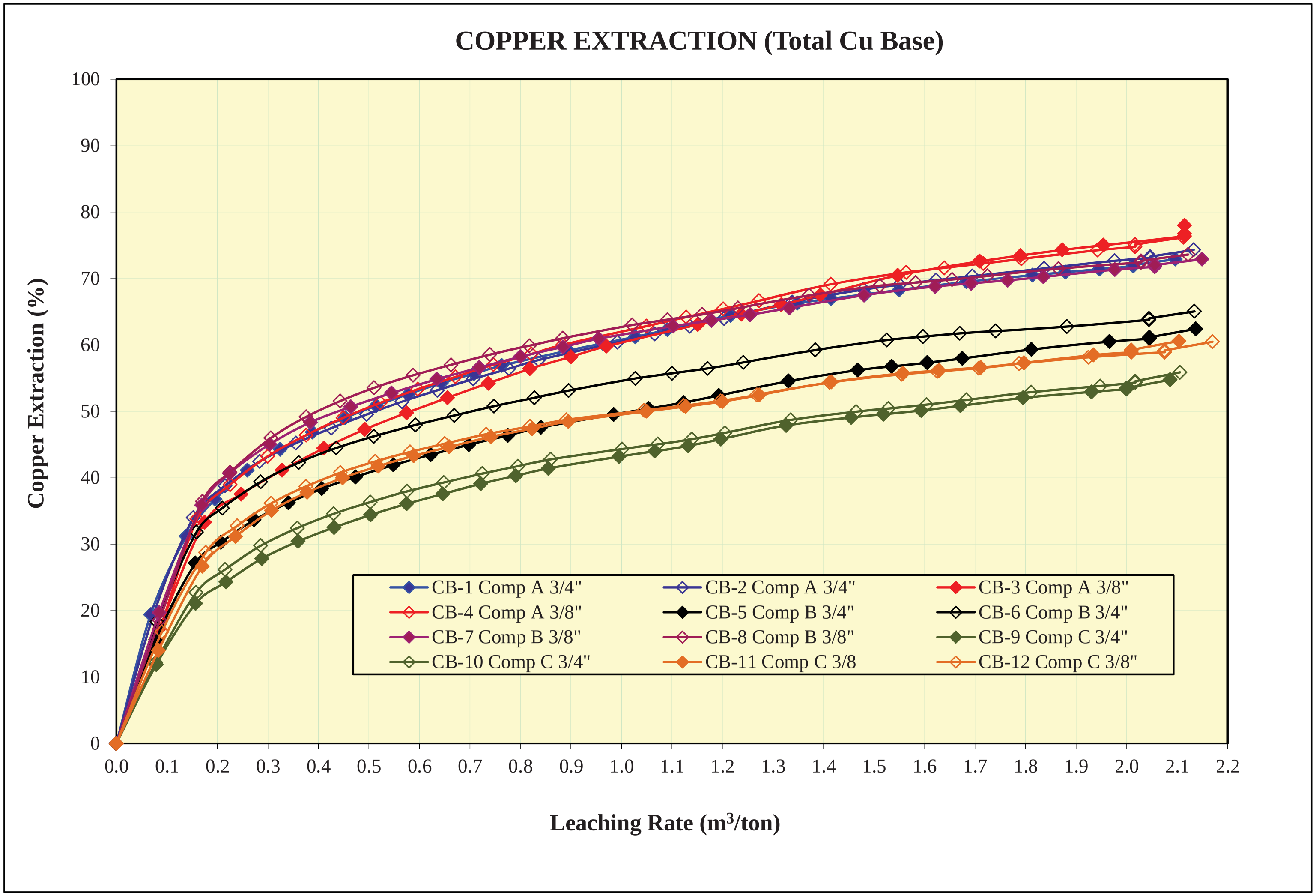


Figure 13.5: Composite A Copper Extraction Kinetics

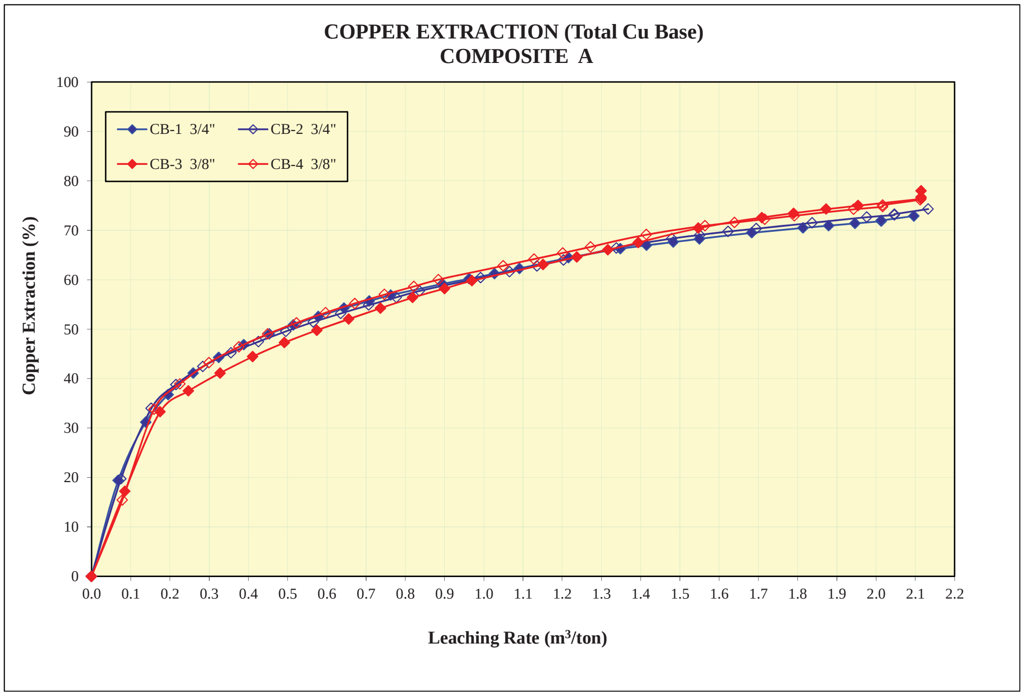


Figure 13.6 shows the copper extraction kinetics for composite B, where the kinetic differences for P_{80} of ¾" or ⅜" are compared. On average and for the end of the irrigation cycle, the difference in terms of copper extraction percentage for both grain sizes was 11 points.

Figure 13.7 shows the copper extraction kinetics for composite C, where the kinetic differences for P_{80} of ¾" or ⅜" are compared. On average and for the end of the irrigation cycle, the difference in terms of copper extraction percentage for both grain sizes was 5.2 points.

Based on this, leaching should be performed at a P_{80} of ⅜", given that it is possible to obtain higher copper extraction values on composites B and C. It should be highlighted that the three composites kinetic profiles did not fully reach an asymptotic level during the leach period.

Figure 13.6: Composite B Copper Extraction Kinetics

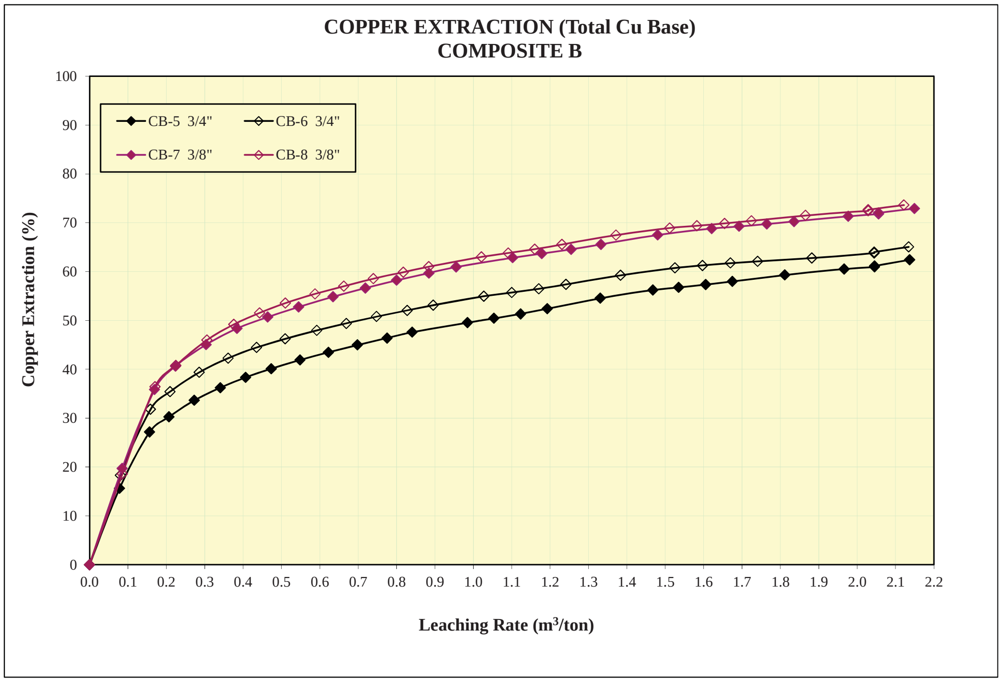
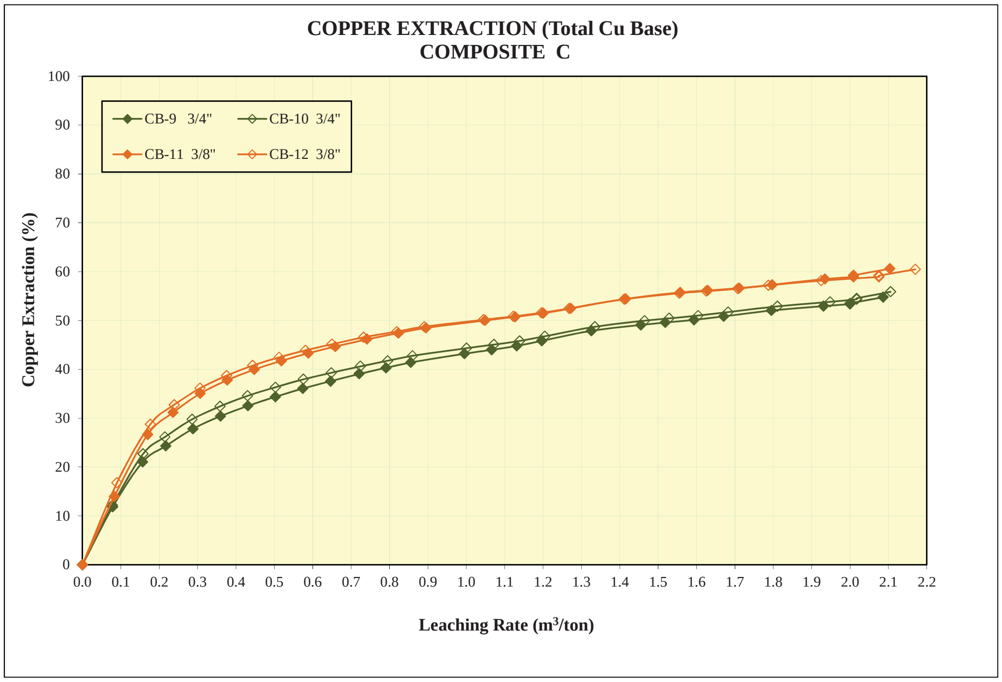


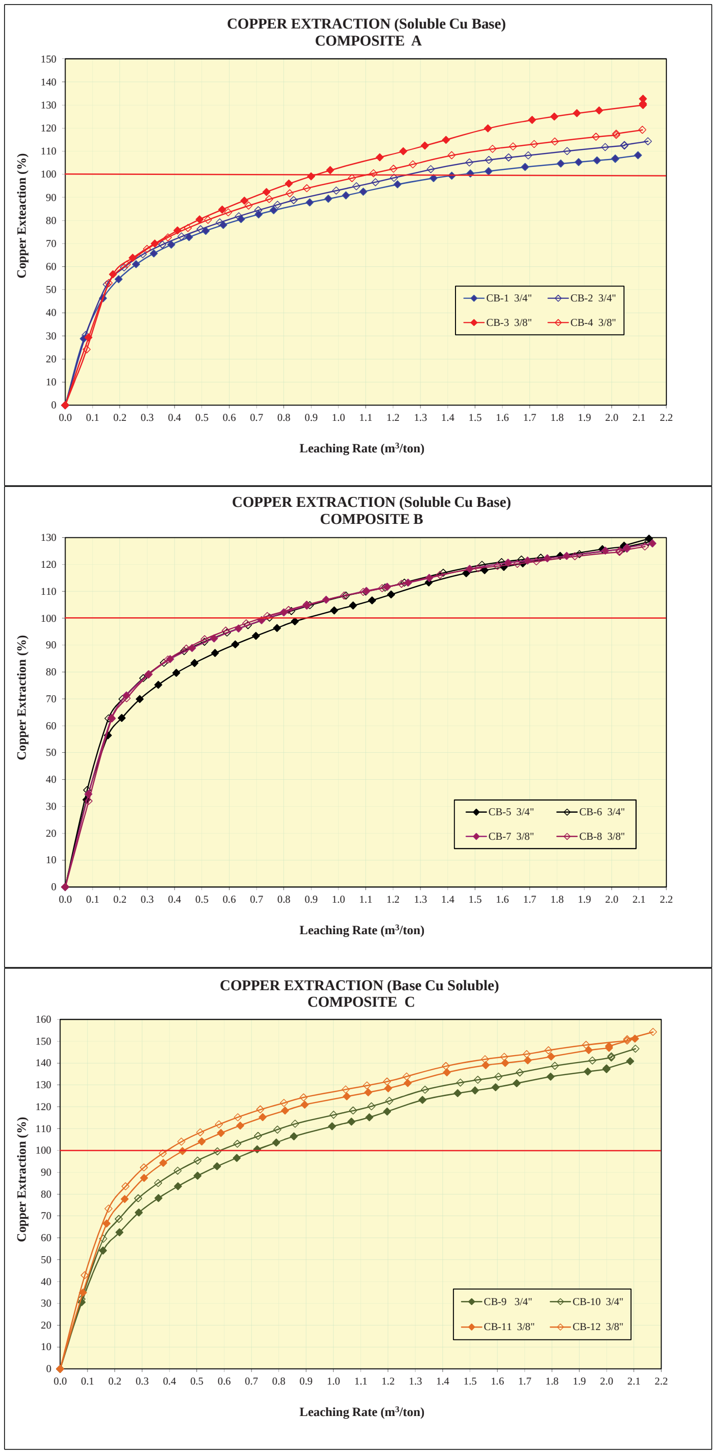
Figure 13.7: Composite C Copper Extraction Kinetics



Regarding copper extraction, in terms of soluble copper grade, all samples greatly exceeded 100% extraction, as shown in Figure 13.8.

This is due to the assaying method for CuS already commented on in 13.2.1.As mentioned, oxidized copper species present at Berta would correspond to copper wad ($CuOMnO_2$), which is dissolved more efficiently in a reducing media, which in turn is the reason why CuS should be assayed in future with the sodium bisulfite method, rather than sulfuric acid.

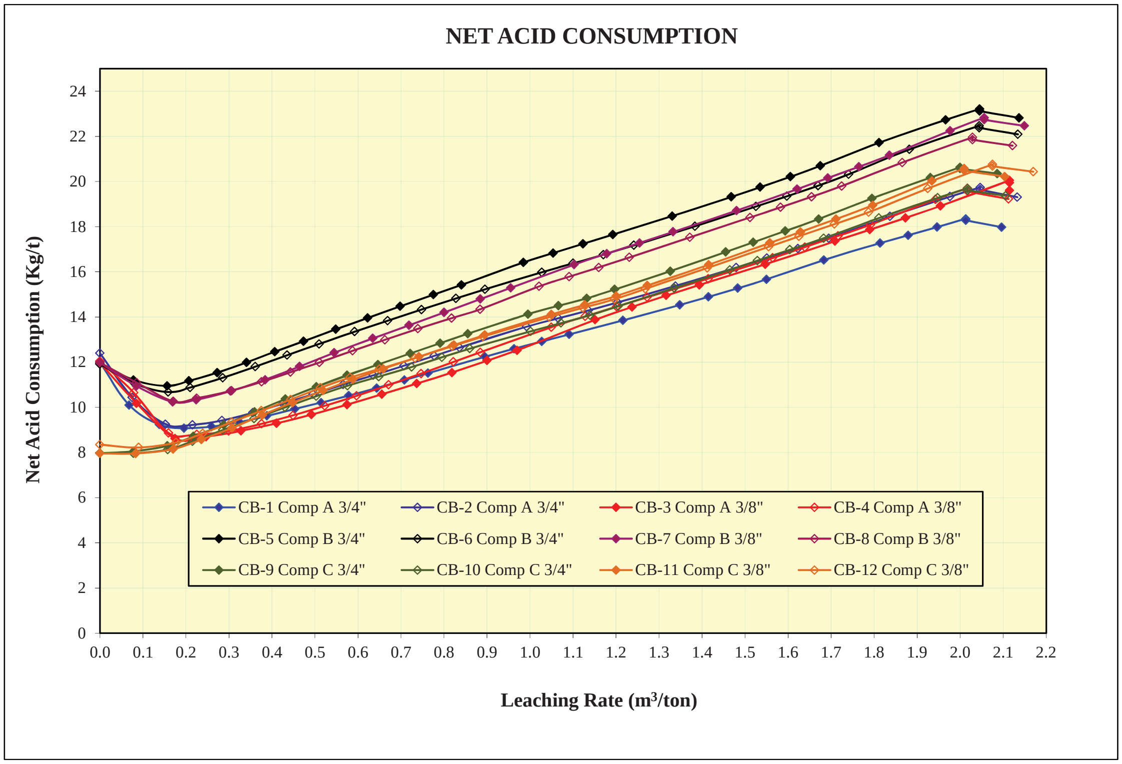
Figure 13.8: Copper Extraction of Soluble Copper



13.2.7.2 ACID CONSUMPTION

Figure 13.9 shows the net acid consumption kinetics, obtained from all performed tests, showing a linear increment during all the irrigation cycle.

Figure 13.9: Net Acid Consumption Kinetics



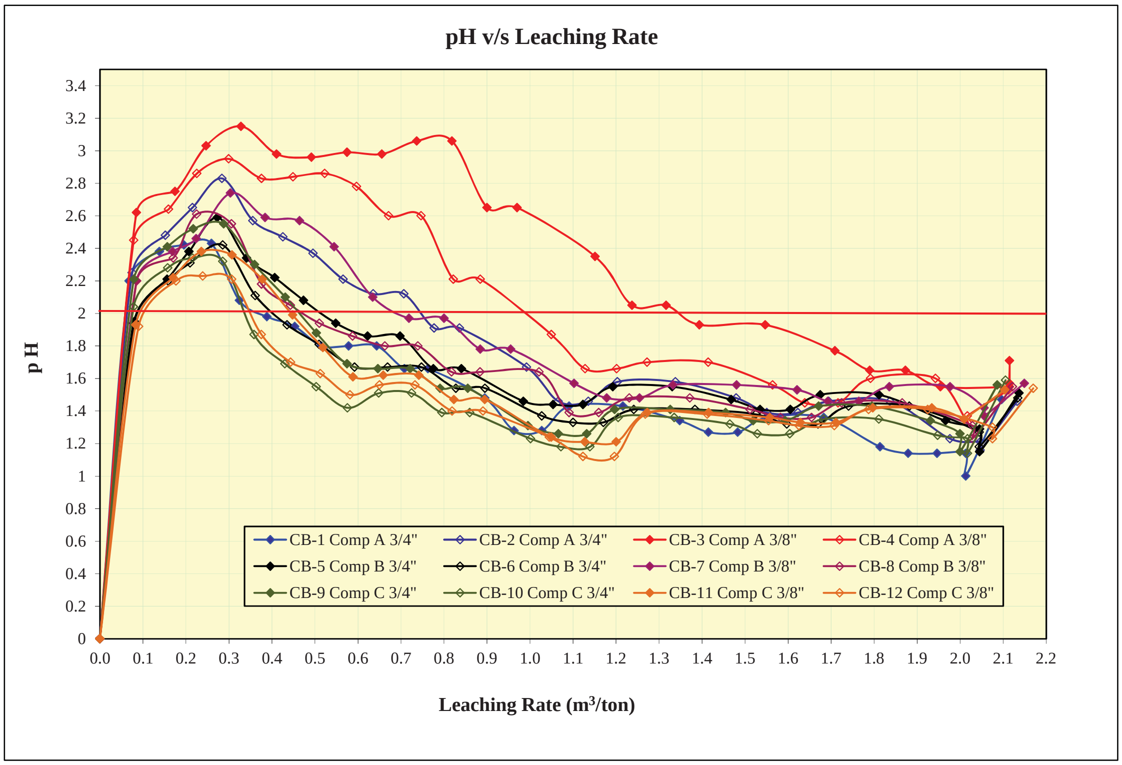
Composite A in its two grain sizes, showed a similar net acid consumption (average 19.0 kg/t). Composite B in its two grain sizes, showed a similar net acid consumption (average 22.3 kg/t). Composite C also showed similar net acid consumption for its two grain sizes (average 20.0 kg/t).

Consequently, net acid consumption varied from 19.9 kg/t (comp A) to 22.3 kg/t (comp B).

13.2.7.3 EFFLUENT PH EVOLUTION

Figure 13.10 shows the effluent pH evolution during the irrigation cycle. It is observed that composite A with P_{80} of ⅜", maintained practically half of the irrigation cycle with a pH > 2.0, and then finished with a 1.6 pH. For composites B and C this situation was less intense, with most of the irrigation cycle having a pH < 2.0, and then finishing with pH values from 1.2 to 1.6.

Figure 13.10: pH Evolution of the Effluent

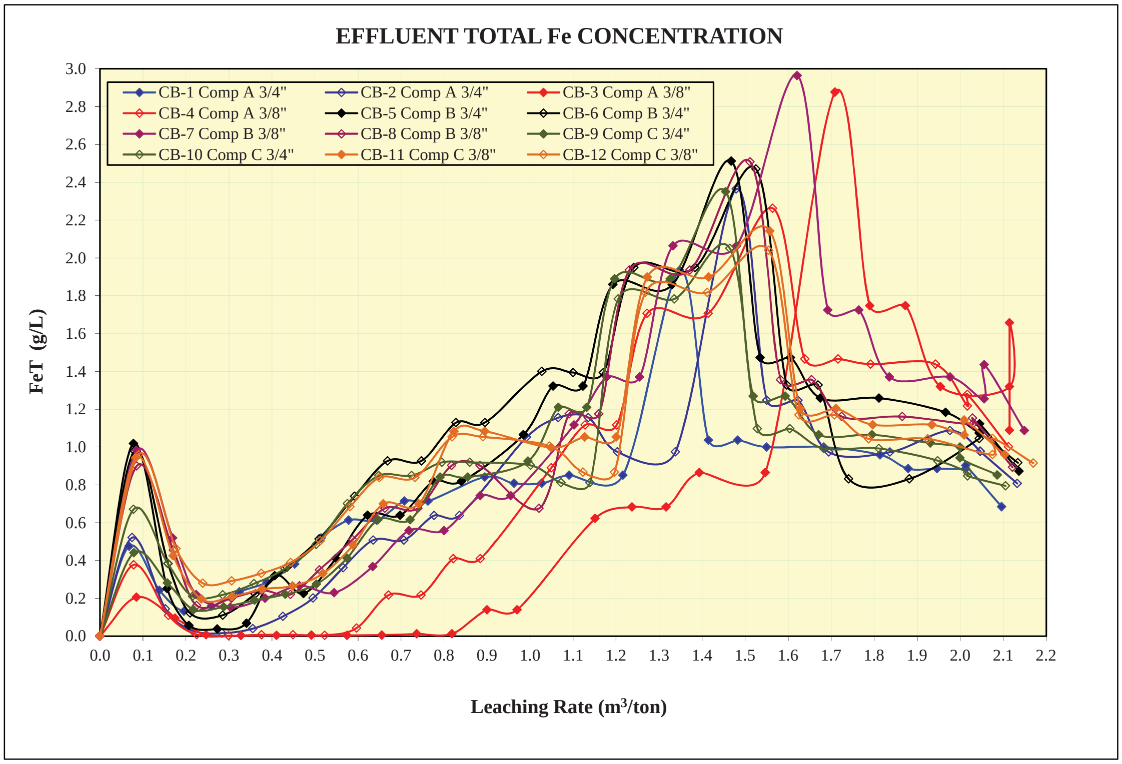


Composite A could have been cured at a higher dose, as the sulfation test indicated (see point 13.2.6), which would have produced a higher copper extraction level, however, net acid consumption would had also have increased. A decision on final dose rates is subject to an economic analysis.

13.2.7.4 EFFLUENT FE CONCENTRATION EVOLUTION

Figure 13.11 shows the evolution of Fe concentration in the effluent solution, which was governed by the solution acidity (pH). During the leaching cycle, when pH was maintained over 2.0, Fe concentration was low, and started to increase slowly when the effluent solution acidity increased.

Figure 13.11: Fe Evolution on the Effluent



13.2.8 LEACH RESIDUE MINERALOGICAL COMPOSITION

On A, B and C composites, for P_{80}⅜" grain size, corresponding to tests CB-3, CB-7 and CB-11, a leach residue mineralogical characterization was carried out with copper oxide mineralogy summarized on Table 13.11 and a complete summary presented on Table 13.12

Table 13.11: Oxide Species in Leach Residue

Copper Oxide	Comp-A (weight %)	Comp-B (weight %)	Comp-B (weight %)
Copper wad	0.18	0.03	0.07
Chrysocolla	0.16	-	0.02

Table 13.12: Leach Residue Mineralogical Characterization Summary

Mineral	Formula	Comp-A (weight %)	Comp-B (weight %)	Comp-C (weight %)
Chalcopyrite	$CuFeS_2$	0.01	0.004	0.01
Chalcocite	Cu_2S	0.01	0.004	0.004
Covelite	CuS		0.004	0.004
Bornite	Cu_5FeS_4	0.01	0.004	-
Copper Wad	Oxides of Cu, Mn, Si, etc.	0.18	0.03	0.07
Chrysocolla	$CuSiO_3.2H_2O$	0.16	-	0.02
Pyrite	FeS_2	0.01	0.01	0.01
Magnetite	Fe_3O_4	0.06	0.02	0.10
Hematite	Fe_2O_3	0.63	0.33	0.25
Limonite	FeOOH	0.83	0.18	0.21
Titania	TiO_2	0.08	0.02	0.07
Clay	$Al_4(Si_4O_{10})(OH)_3$	10.86	5.54	6.62
Chlorite	$(Mg,Al)_3(AlSi_3O_{10})(OH)_2Mg_3(OH)_6$	1.19	1.12	2.03
Amphibolite	$(X,Y)_{7-8}(Z_{411})_2(OH)_2$X:Na,K,CaY:Al, F^{+3}, Fe^{+2}, Mg, Mn,Ti,Cr,LiyZnZ:Si,Al	0.77	0.61	-
Actinolite	$Ca_2(Mg, Fe^{2+})_5Si_8O_{22}(OH)_2$	0.36	1.11	1.20
Apatite	$Ca_5(PO_4)_3$ (Cl)	0.13	-	0.16
Calcite	$CaCO_3$	0.20	0.63	0.44
Epidote	$Ca_2Al_2FeSi_3O_{12}(OH)$	1.37	1.28	1.73
Biotite	$K(Mg,Fe)_3(AlSi_3O_{10})(OH,F)_2$	1.21	2.06	1.84
Quartz	SiO_2	56.56	53.04	52.20
Plagioclase	$(Ca, Na)(Al,Si)AlSi_2O_8$	8.92	14.91	19.00
Feldspar	$KalSi_3O_8$	7.78	9.90	7.28
Pyroxene	$(Mg, Fe)_2Si_2O_6$	2.03	0.77	0.86
Sericite	$(Na,Ca)(Mg,Fe,Li)_3Al_6B_3Si_6O_{27}(OH)_4$	6.66	8.44	5.89
TOTAL		**100.00**	**100.00**	**100.00**

13.3 BERTA CENTRAL METALLURGICAL TESTING

In order to compare the results obtained by Geomet with the metallurgical tests from Berta Central, representative samples from the deposit were extracted and leaching assays were performed at the Hydrometallurgical Lab of the Universidad de Santiago of Chile Metallurgical Mining Engineering Department.

Mineralogy is similar to that of the samples studied by Geomet.

Three tests in two meters columns were performed, with the same dimensions as those utilized by Geomet, but the columns' feeding granulometry was 100% -1/2". The sulfuric acid curating dose was 10 kg/t for 24 h at specific flow of 10 l/hm^2.

Given that the sample extracted from Berta Central has a 1.4% CuT and 1.1% CuS head grade that requires more sulfuric acid for its higher copper content, it was decided to perform tests at 10, 15 and 20 g/l of sulfuric acid concentration in the leaching solution. Results showed a kinetic behavior very similar to that observed by Geomet, for which the Berta Central minerals are technically feasible to leach, with metallurgical results similar to the achieved by Geomet, apart from the head grade differences on the samples utilized for assays.

13.3.1 LEACHING TESTS WITH 10 G/L ACID CONCENTRATION IN LEACHING SOLUTIONS

On Figure 13.12 it can be observed the extraction kinetics achieved with Berta Central material at a granulometry 100% under ½", leached at a specific flow of $10l/hm^2$, with a 2 m column height and a 10 g/l sulfuric acid concentration in the leaching solution.

Figure 13.12: Copper Extraction Kinetics 10 g/l of H2SO4

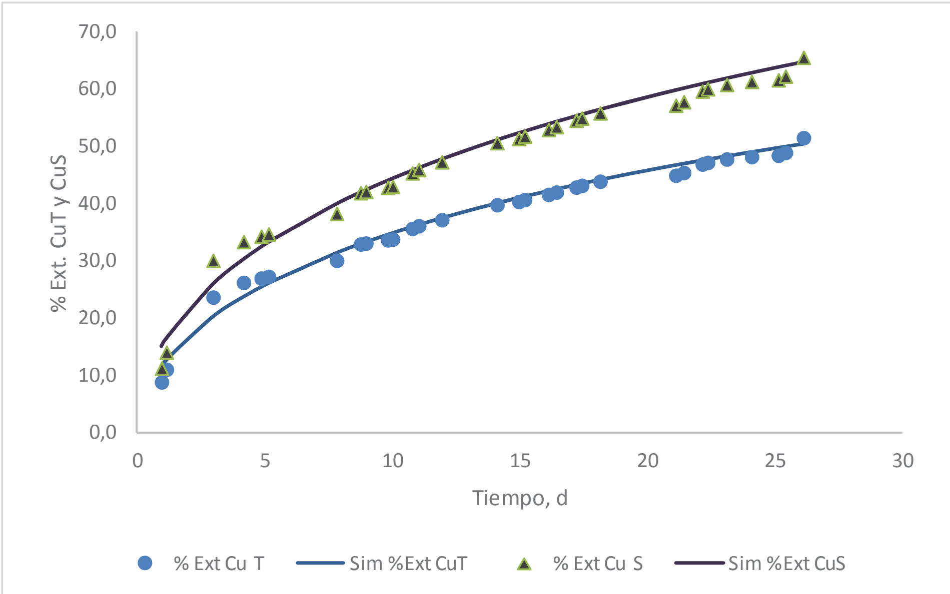
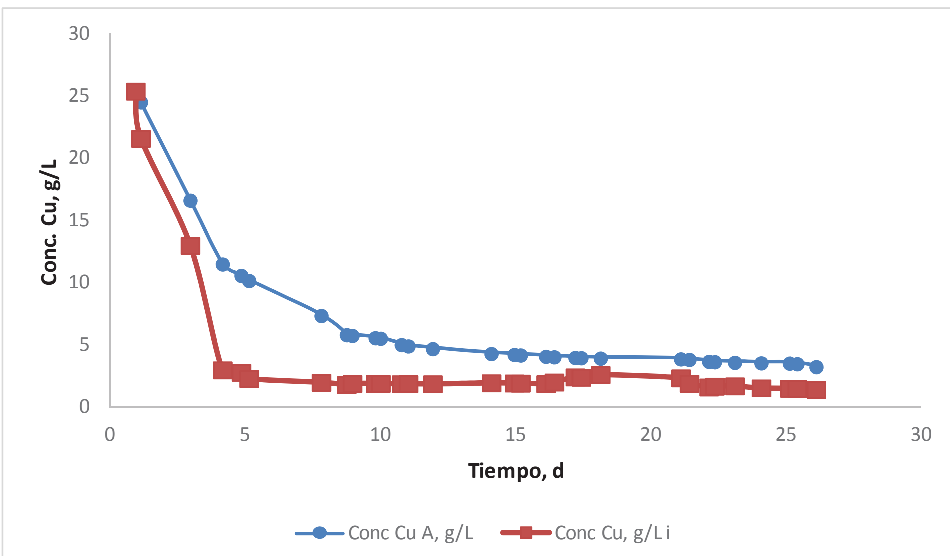


Figure 13.13 shows the variation of copper concentration in the solution product of leaching over the time. The figure shows the instant concentration and accumulated concentration (equivalent to PLS).

Figure 13.13: Copper Concentration in time (10 g/l of H2SO4)



Net acid consumption was 22.11 kg of sulfuric acid per ton (kg/t). It shouldbe noted that this particular curve shows a slow kinetic compared to the results obtained by Geomet, but this is due to the noticeable grade difference that in this case was 1.4% CuT and 1.1% CuS. The observed kinetic, with a strong and fast drop of the instant concentration, added to the very low sulfuric acid concentrations in the product solution, evidences the lack of acid in the conditions of the studied leaching system. Assays at higher sulfuric acid concentration confirm this situation.

13.3.2 LEACHING TESTS WITH 15 G/L ACID CONCENTRATION IN LEACHING SOLUTIONS

On Figure 13.14 it can be observed the extraction kinetics achieved with Berta Central mineral at a granulometry 100% under ½", leached at a specific flow of 10l/hm^2, with a 2 m column height and a 15 g/l sulfuric acid concentration in the leaching solution.

Figure 13.14: Copper Extraction Kinetics with 15 g/l of H2SO4

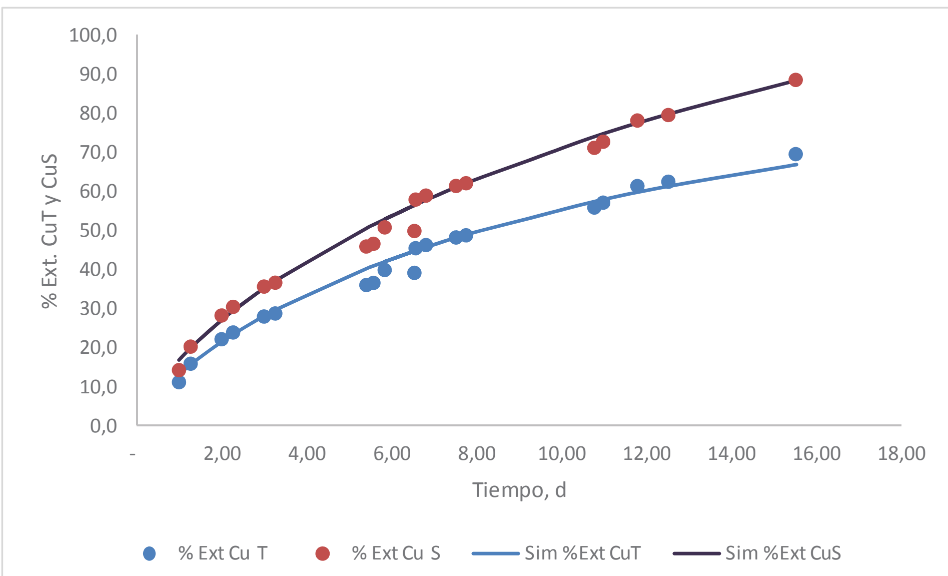
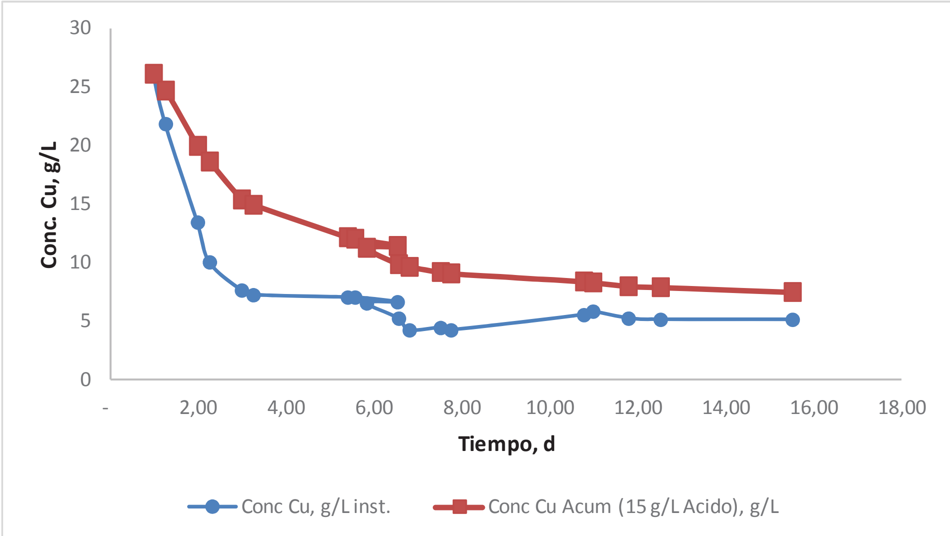


Figure 13.15 shows the variation of copper concentration on the solution product of leaching over the time. The figure shows the instant concentration and accumulated concentration (equivalent to PLS).

Figure 13.15: Copper Concentration in time (15 g/l of H2SO4)



Net acid consumption was 19.6 kg/t. The obtained kinetics shows a comparable behavior to the obtained by Geomet and shows how the extraction improves over the time. From the concentration results it can be observed that the accumulated concentration to 16 days was 7.5 g/l.

13.3.3 LEACHING TESTS WITH 20 G/L ACID CONCENTRATION IN LEACHING SOLUTIONS

On Figure 13.16 it can be observed the extraction kinetics achieved with Berta Central mineral at a granulometry 100% under ½", leached at a specific flow of 10l/hm^2, with a 2 m column height and a 20 g/l sulfuric acid concentration in the leaching solution.

Figure 13.16: Copper Extraction Kinetics with 20 g/l of H2SO4

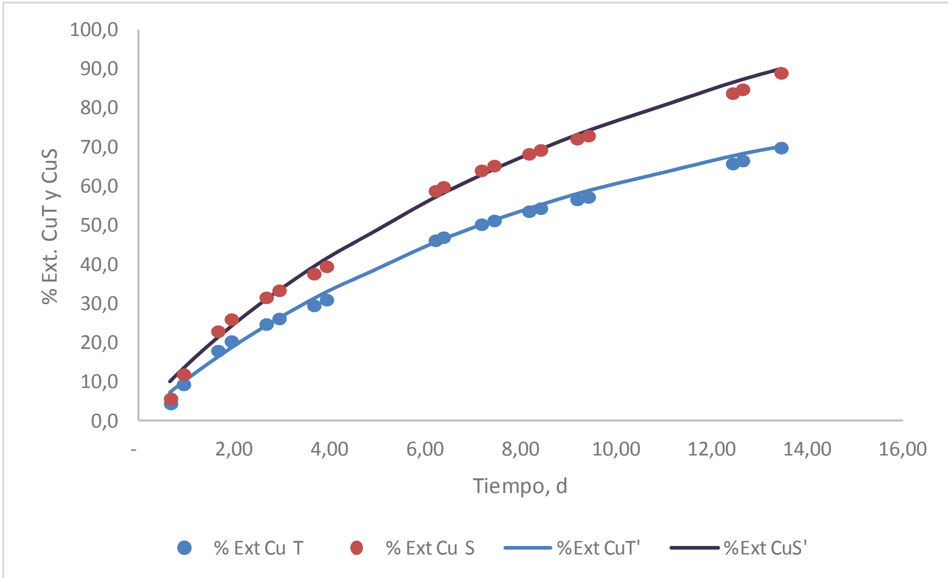
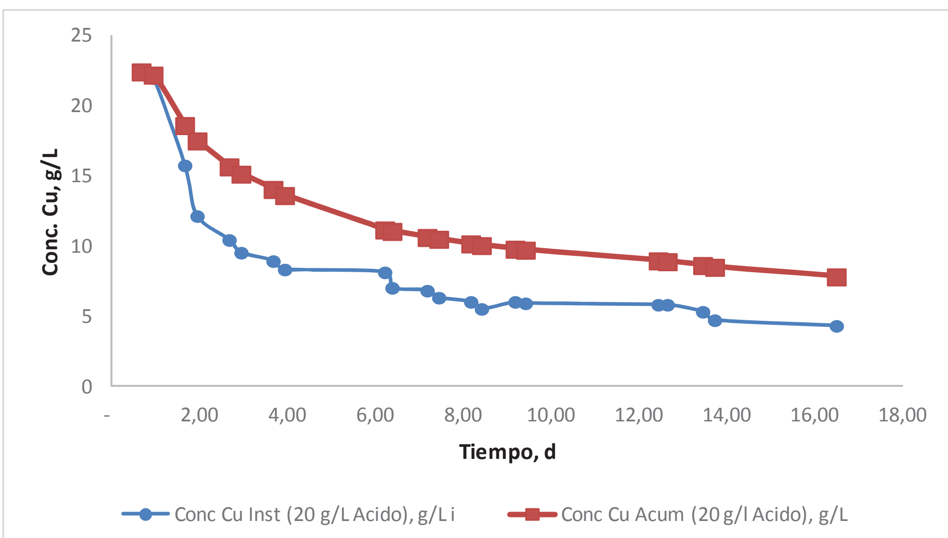


Figure 13.17 shows the variation of copper concentration on the solution product of leaching over the time. The figure shows the instant concentration and accumulated concentration (equivalent to PLS).

Figure 13.17: Copper Concentration in time (20 g/l of H2SO4)



Net acid consumption was 27.9 kg/t. Although technical results evidence an improvement in respect to the 15 g/l leaching, this higher net acid consumption need to be revised to check cost linked to this reactant. In order to make a decision it is recommended to make a trade off study, to determine if the kinetics rise is more beneficial than the acid consumption rise.

13.3.4 RESULTS ANALYSIS

Figure 13.18 shows a comparison on the three performed tests.

As it can be observed in the figure, the kinetic with the conditions utilized by Geomet is the one that shows the lowest performance, this occurs because of the higher head grade of Berta Central minerals that forces to use a higher quantity of acid to accelerate the kinetics, as shown by the curves at higher acidity. This behavior is not out of the ordinary, on the contrary, it is expected when the grade increases, and normally it is decided to increase the leaching time and maintain the acid in the irrigation or curing, or more sulfuric acid is added in the curing in order to maintain the leaching time (generally based on the sulfuric acid costs and the leaching space available).

Figure 13.18 CuT Extraction % Assays Comparison

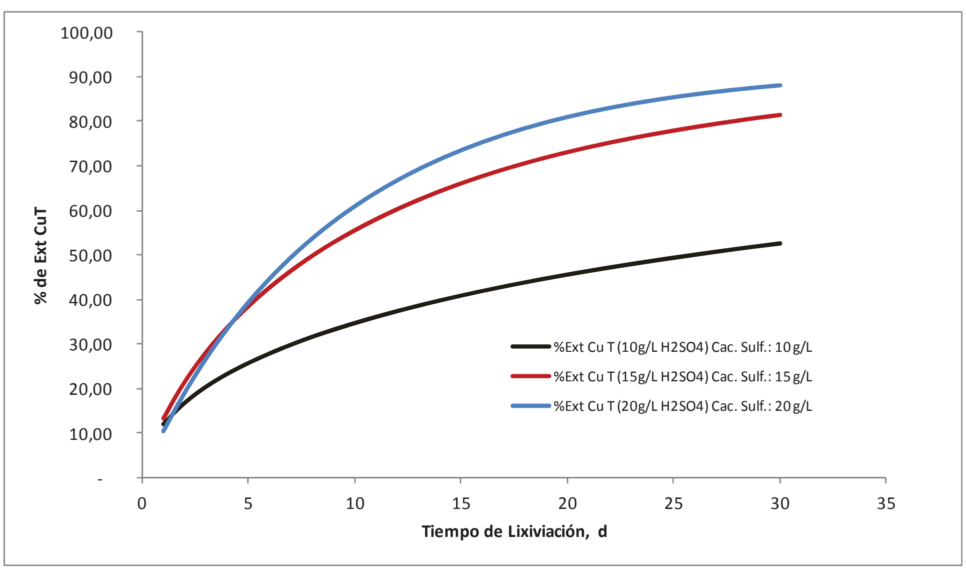
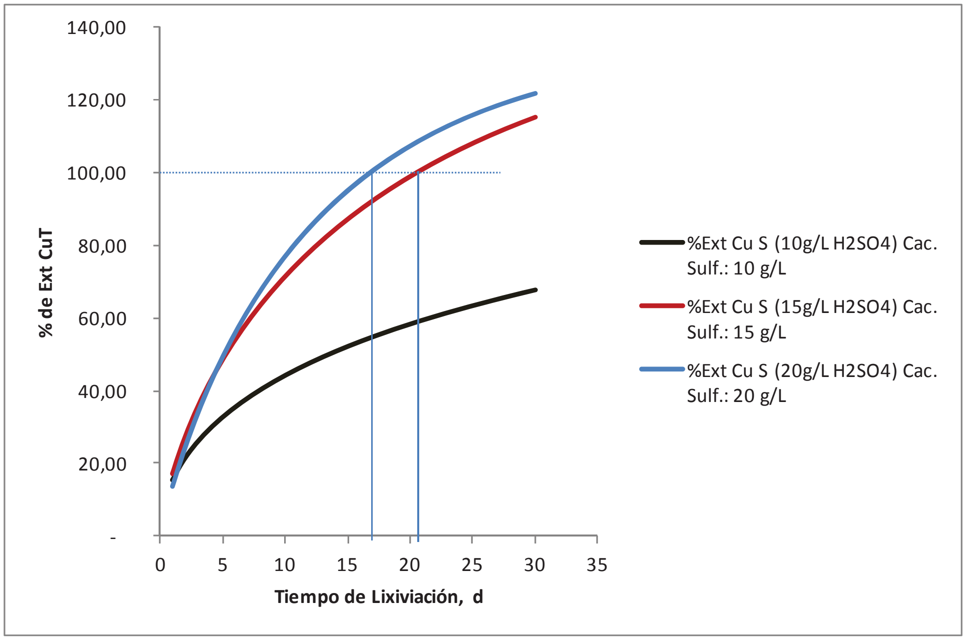


Figure 13.19 shows the leaching kinetics, represented by the Cu extraction percentage in terms of soluble copper versus the leaching time.

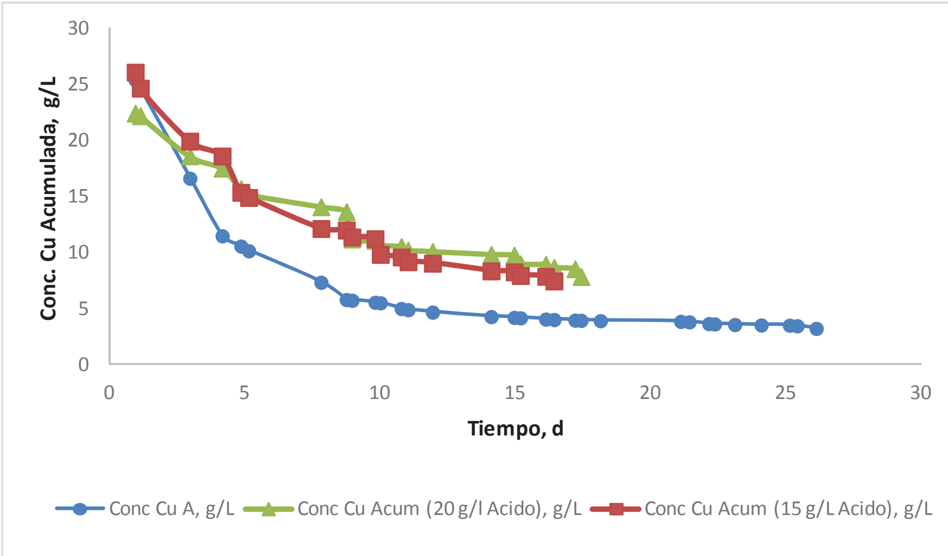
Figure 13.19: Soluble Cu Extraction % in Time



It shouldbe noticed that for 15 and 20 g/l of sulfuric acid in the leaching solutions, the 100% extraction is exceeded at the 21 and 16 days of irrigation; this is explained because minerals that require presence of oxidant agents are being leached (Cuprite, Copper Wad, chalcocite and other secondary sulphides). These results are very similar to those obtained by Geomet. Using 10 g/l of acid for these grades will require longer leaching times to achieve the same extraction level.

Finally, in Figure 13.19 it can be observed the comparison between the accumulated concentrations in the product solution.

Figure 13.19: Accumulated Concentration in Time



In Figure 13.19 the difference between the accumulated concentrationscan be observed. It is clear that for a 10 g/l of H_2SO_4 final concentrations are very low, nevertheless, the difference between the final concentrations on the assays with 15 and 20 g/l of sulfuric acid are not big. Concentration observed in the first leaching hours can be increased with an increase of the sulfuric acid dose.

13.4 BERTA SUR AND CENTRAL METALLURGY CONCLUSIONS

Table 13.13 shows a comparison between the metallurgical results obtained by Geomet using the material from Berta Sur and those obtained by USACH treating material from Berta Central. These results corroborate that Berta Sur and Berta Central have a similar behavior. It should be considered that for Berta Central higher grades, a higher sulfuric acid dose can be added in curing that will produce better metallurgical results.

Table 13.13: Metallurgical Column Test Work / Berta Sur & Central Comparison

Column	Sample Location	Head assays		Theoretical % Sol	Actual		Days	NAC kg/t
		% CuT	% CuS		Rec CuT	Rec CuS		
P80 3/8" Comp A Geomet	BDH07-07 Drill Core (Berta Sur)	0,84	0,59	70	**91,0**	**130**	26	21
P80 3/8" Comp B Geomet	Surface trench, partially leached (Berta Sur)	0,66	0,36	55	**68,0**	**126**	28	24
P80 3/8" Comp C Geomet	Surface trench, partially leached (Berta Sur)	0,38	0,14	37	**56,0**	**150**	28	22
P80 1/2" (10 g/L H2SO4) USACH	Berta Central	1,40	1,10	79	**51,5**	**66**	28	22
P80 1/2" (15 g/L H2SO4) USACH	Berta Central	1,40	1,10	79	**80,0**	**113**	28	20
P80 1/2" (20 g/L H2SO4) USACH	Berta Central	1,40	1,10	79	**87,0**	**120**	28	28

The most relevant conclusions for the Berta Sur study are as follows:

- Material from Berta deposit presented a CuT grade of 0.83% for composite sample A, 0.63% for sample B and 0.39% for sample C.
- The average solubility of the three samples by the sulfuric acid method was 70.1% for composite A, 50.8% for composite B and 37.6% for composite C.
- The average solubility of the three composites by the citric acid method was 55.4% for A, 14.5% for B and 24.8% for C.
- The solubility rates with ferric and sodium bisulfite agent were only performed on composite B, given that it approximates the average grade of the Berta Sur resource. The average solubility rate in ferric environment was 54.5%, while in bisulfite it was 59.5%.
- The fact that the solubility maximizes while using sodium bisulfite (reduction agent), is an indicator of the presence of copper oxides species corresponding to copper wad? ($CuOMnO_2$).
- The head sample mineralogical characterization confirmed that copper wad would a major component of the oxide copper species present.
- Results from Iso-pH tests, in terms of total copper extraction were 73% for composite A, 69% for B and 55% for C.

- Net acid consumption from Iso-pH tests were 15.0, 13.8, and 13.0 kg/t, in composites A, B and C respectively, equivalent to rough gross acid consumptions of 22.3, 19.7, and 15.4 kg/t, respectively.
- In terms of chemical kinetics, composite A has the fastest dissolution velocity, then B and finally C. Furthermore, composites B and C have kinetic similarities, but they differ greatly from A.
- Sulfation tests showed doses of 17 and 23 for composite A; 12 and 8 kg/t for composites B and C, respectively. Only composite A should use different doses for P_{80} of ¾" and ⅜".
- In the column leaching tests, the highest copper extraction levels (78-73%) were from composite A P_{80} ¾" as well as ⅜", and B P_{80}⅜". A lower extraction level (61-65%), was for B P_{80} ¾" and C ⅜". Finally, the lowest extraction level (55%) was from sample C, P_{80} ¾".
- Extraction kinetics were identical for each grain size of composite A.
- Composite B shows a distinct difference between each grain size tested (P_{80} ¾" and ⅜"), reaching a difference of 11 points, in terms of copper extraction percentage, at the end of the leaching period.
- Composite C also shows a difference between both sizes, reaching 5.2% difference at the end of the leaching period.
- Net acid consumption varied between 19.0 kg/t (Composite A) and 22.3 kg/t (Composite B).

The most relevant conclusions for Berta Central study are as follows:

- Samples from Berta Central have a similar kinetic behavior to that observed in the tests performed by Geomet.
- It is technically feasible to leachBerta Central material with acid solutions, achieving net acid consumption from 20 to 22 kg of acid / t.

- Total dissolution of soluble copper is achieved between 16 and 18 days of leaching.
- In 30 days of leaching extractions achieved are 53, 81 and 88% when leaching with sulfuric acid solutions with 10, 15 and 20 g/l respectively.
- It is recommended to perform tests with higher curing acid dose, leaching with irrigation solutions with concentrations of sulfuric acid less than 15 g/l.

Table 13.13 shows that the recovery of soluble copper exceeds 100% in all but one of the columns. This is due to the presence of black oxides copper (copper wad?) minerals that did not report to the soluble copper assay during analysis, but is recoverable over the period of the column tests. The columns were stopped at 28 days before the recovery curves went asymptotic. Based on the results of this column test work and the soluble copper component of the deposit from drill hole assays, SCMB estimates that a recovery of 78% of the total copper in the heap leachable material should be achievable in the 60 day leach cycle contemplated for the operation. The ROM material averages 0.20%CuT and 0.12%CuS, and recoveries are estimated to be 75% of the soluble copper which is equivalent to 45% of total copper. This estimate takes into account the proposed blasting pattern of a 5x5m grid on 5m high benches which should result in a grain size slightly better than that from a first stage crusher. Leaching will take place on 7m high pads without liners between lifts, which should also result in additional recovery over time. Benchmarking against other dump leach operations in Chile indicates that they achieve recoveries of between 40 and 50% of total copper.

14.0 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

Geoinvestment generated a resource block model that integrates all the available information in order to determine the grade-tonnage curve for the Berta deposit, together with the estimation of the total exploitable economical resources.

The block model includes only the estimation of the oxide zone of the Berta Sur and Central deposits

The following products were obtained from this work:

- Data validation.
- 3D Model of the mineralized body.
- Resource block model.

14.2 WORK METHODOLOGY

The process started with the review and validation of the data base, in order to have reliable data and results in the modeling and estimating processes. Samples were then regularized through compositing by bench height. The fixed length used in the compositing process was 2.5m.

Given that mining parameters such as bench height and extraction equipment are known, it was decided to change the block size from 2.5 m to 5.0 m.Prior to this, the existence of support change or dilution effect as a product of this decision was studied, with no consequences nor alterations attributable to these concepts being detected.

Mineral body envelopes were generated and received from MCC, corresponding to the realization of control level plans every 5.0 m in order to obtain more precise solids that better represented the mineral distribution.

The selected estimation method was Ordinary Kriging.

14.3 RECEIVED INFORMATION

Original information from drill holes was received at the beginning of the study (October, 2012).

The Berta Sur and Berta Central drill hole database was closed-off as of October 1st, 2012, as is detailed in Table 14.1

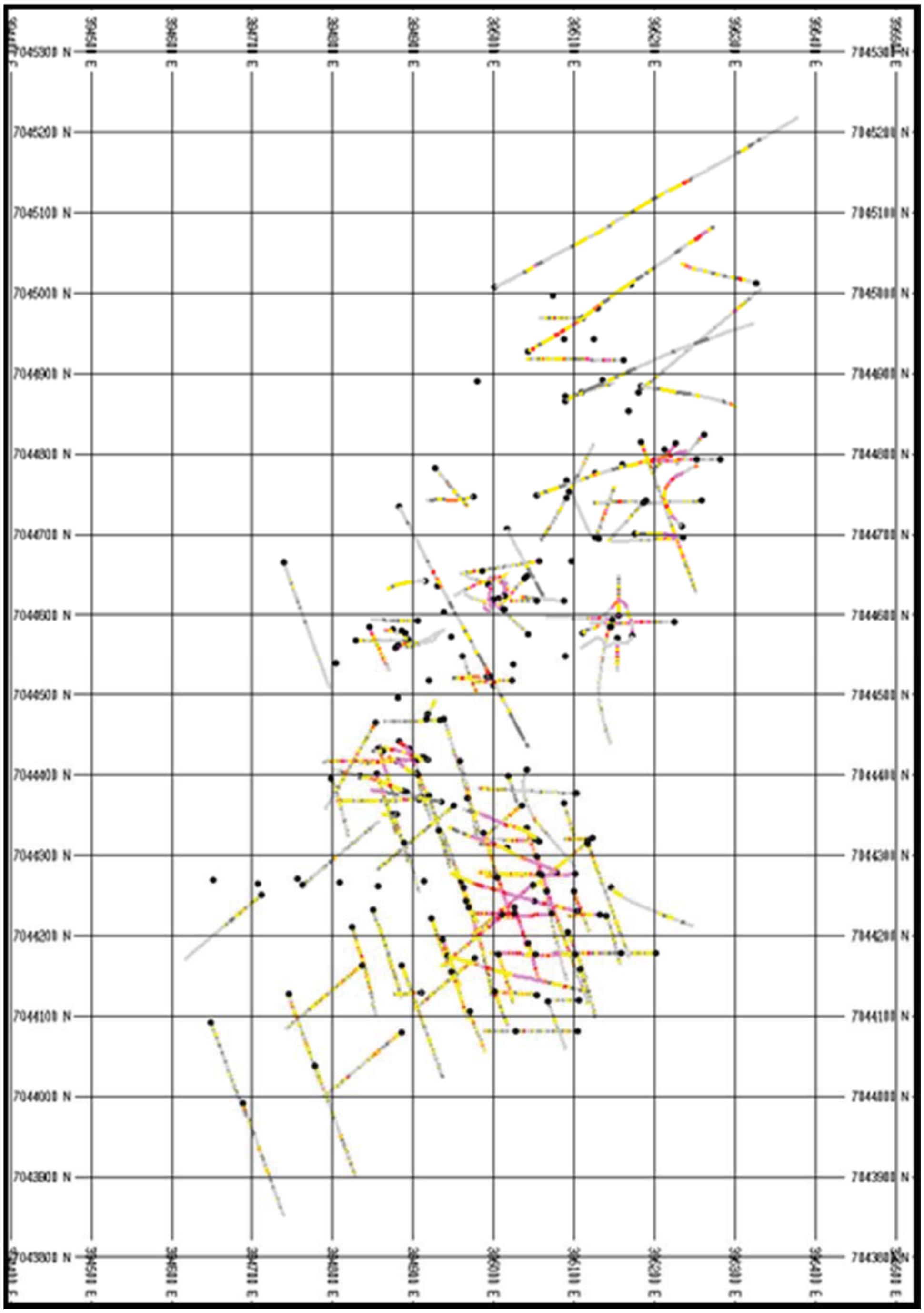
Table 14.1: Berta Sur and Berta Central Drillhole and Trench Database

Berta Drill hole Database (BDD)	Company	Type	Start Drill hole	End Drill hole	Number of Drill holes
1	Outokumpu	Reverse Circulation	S-01	S-47	32
2	Mantos Blancos	Reverse Circulation	SR-01	SR-42	35
3	Grand CRU	Diamond Drill	BDH07-01	BDH07-09	9
4	CORO Phase 1	Reverse Circulation	BR-01	BR-23	23
5	CORO Phase 2	Reverse Circulation	BR-25	BR-56	18
6	CORO Infill	Reverse Circulation	BR-57	BR-92	35
7	Mantos Blancos	Trenches	TMB-01	TMB-30	16
		Trenches	PBR-73	PBR-85	3
		Trenches	MIN-01	MIN-09	9
		Trenches	T-3	T-12 and Z-8	8

Topography was provided by MCC. Theavailable information covers an area of 595,000 m^2 between N7.043.840 – N7.045.220 North coordinates and E394.600 and E395.380 East coordinates. The drilled zone describes a rectangle of 1,385 m (X) by 780 m (Y).

The deposit was drilled in an approximated grid of 50 m by 50 m, having isolated sectors within the drilling distribution where the grid is 12.5 m by 12.5 m. This is shown in Figure 14.2 Figure 14.2

Figure 14.1: Berta Sur and Berta Central Drilled Area



14.4 INFORMATION REVIEW AND VALIDATION

Drill hole database structure and formats are summarized below. Database review and validation is included.

14.4.1 COLLAR TABLE

Collar ASCII table structure and features are listed in Table 14.2

Table 14.2: Collar ASCII Table Structure

File name:	Header.prn
# of records:	102
# of columns:	7

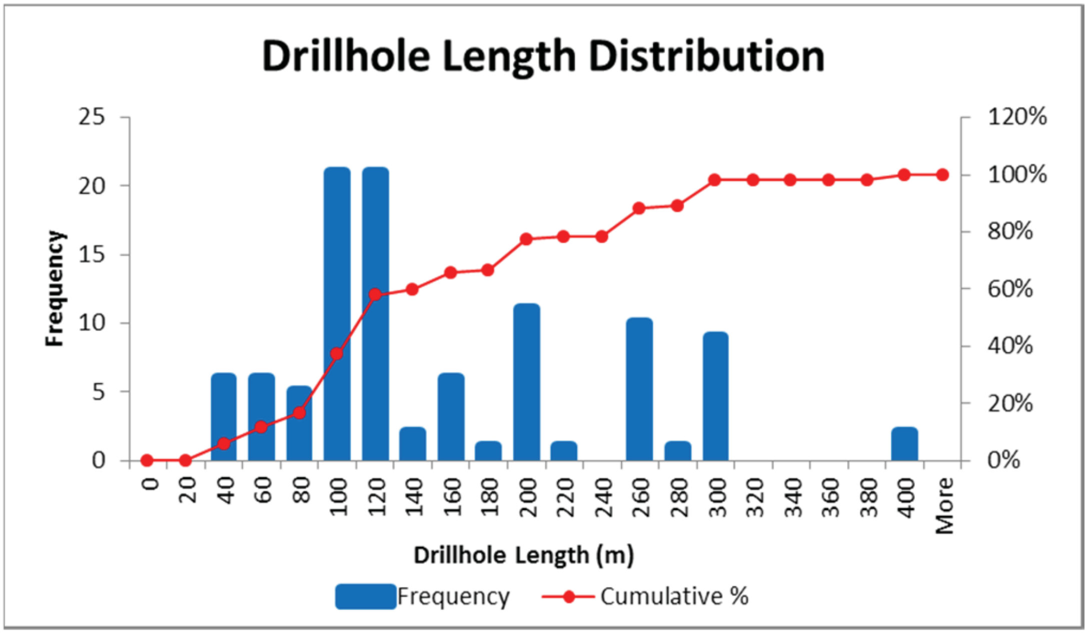
Column	Field	Observations
1	ID	Hole identification
2	UTM_E	East coordinate
3	UTM_N	North coordinate
4	z	Elevation coordinate
5	Azimuth	Drill hole azimuth angle
6	DIP	Drill hole dip angle
7	Length	Drill hole length (m)

Table 14.2 includes information from 91 drill holes and 11 trenches, corresponding to 14,362.45 drilled meters and 938.6 meters sampled respectively. Drill holes and trenches have different lengths, as it is shown in Table 14.3 and Figure 14.2

Table 14.3: Drill Hole and Trenches Length Statistics

	Length (m)
# of Drill holes	102
Min.	21.4
Max.	400
Mean	150
Std. Dev.	85
Variance	7.265
Coeff. of Var.	0.57

Figure 14.2: Berta Sur Drilled Area



14.4.2 SURVEY TABLE

Survey ASCII table structure and its characteristics are enumerated in Table 14.4

Table 14.4: Survey ASCII Structure

File name:	Survey.prn
# of records:	102
# of columns:	6

Column	Field	Observations
1	ID	Hole identification
2	From	From sample interval (m)
3	To	To sample interval (m)
4	Al	Sample length
5	Azimuth	Drill hole azimuth angle
6	DIP	Drill hole dip angle

Table 14.4 includes information from 91 drill holes and 11 trenches. There are two sorts of drill holes, surveyed downhole and nonsurveyed down hole (Table 14.5).

Table 14.5: Survey Statistics

Drill hole Executor	Drill hole Prefix	Drill hole Type	# of Holes not surveyed downhole	# of Holes surveyed downhole	Total
Grand CRU	BDH	Diamon Drill	1	1	2
Outokumpu	S	Reverse Circulation	4	0	4
AngloAmerican	SR	Reverse Circulation	19	0	19
CORO	BR	Reverse Circulation	30	36	66
AngloAmerican	TMB	Trench	11	0	11
TOTAL			65	37	102
			64%	36%	100%

As shown in the Table 14.5, more than half (64%) of the drill holes have no downhole surveys, particularly those fromMantos Blancos (Anglo American) campaign. No records were found to be inconsistent when reviewing and uploading data.

14.4.3 ASSAY TABLE

Assay ASCII table structure and its characteristics are in Table 14.6

Table 14.6: Assay Table Structure

File name:	Assays.prn
# of records:	7,414
# of columns:	8

Column	Field	Observations
1	ID	Hole identification
2	From	From sample interval (m)
3	To	To sample interval (m)
4	Al	Sample length (m)
5	CuT	T Cu grade (%); -1 = T Cu grade not measured
6	CuSH	S Cu grade (%); -1 = S Cu grade not measured
7	Mo	Mo grade (%); -1 = Mo grade not measured
8	TAOX	Solub. Ratio (S Cu / T Cu); -1 = solubility not calculated

Table 14.6 includes information from 91 drill holes and 11 trenches (7,229 and 185 samples respectively). Samples were assayed for total copper (%CuT), acid soluble copper (%CuS), and %Mo variables.

For geological modeling, %CuS grade, and type of copper oxides wereused to define domains, while capping of total and soluble copper grades was not considered necessary since the distribution of grades in variable notes the existence of outliers.

One inconsistency was found in the original assay file, where %CuS was greater than %CuT in trench TMB-01 in the interval 80m – 85m.

Samples have different lengths, as shown in Table 14.7 and

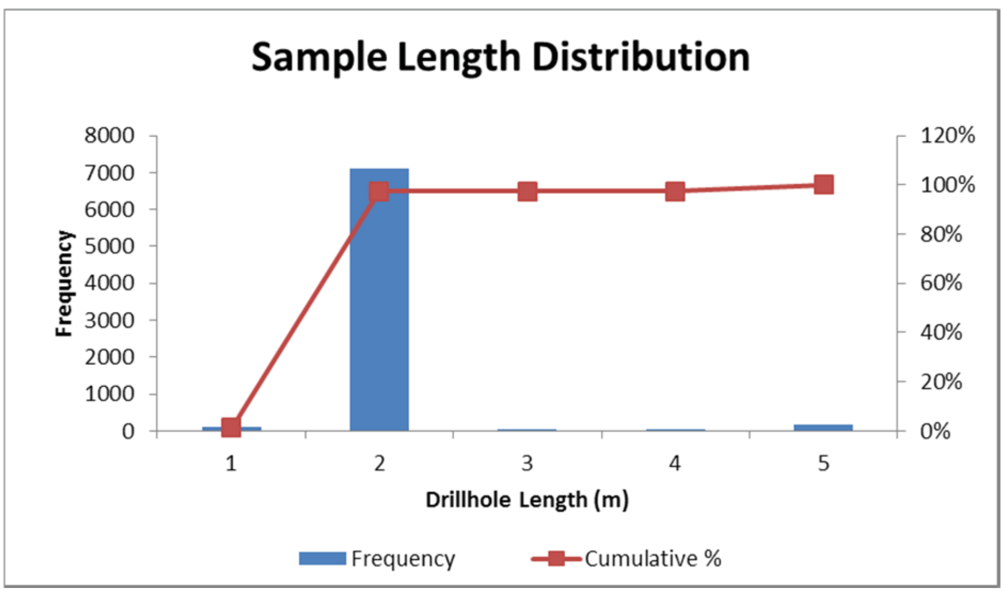
Figure 14.3: Sample Length Histogram

.

Table 14.7: Sample Length Statistics

	Length (m)
# of Samples	7,414
Min.	1
Max.	5
Mean	2.06
Std. Dev.	0.49
Variance	0.24
Coeff. of Var.	0.24

Figure 14.3: Sample Length Histogram



Most of the samples (90%) have length values close to 2 m.

When Collar and Assay tables were compared, the following trench length differences were detected (Table 14.8).

Table 14.8: Collar vs. Assay Tables Drill Hole Lengths

Hole ID	Collar Table Drill hole Length	Assay Table Drill hole Length	Difference
TMB-01	95	100.78	5.8
TMB-01D	60	60.85	0.9
TMB-02	20	21.42	1.4
TMB-03	150	150.23	0.2
TMB-04B	55	55.18	0.2
TMB-06	55	57.5	2.5
TMB-15	185	185.27	0.3
TMB-16	115	117.39	2.4

These small differences present in the trenches are due to the transformation of data to be taken to drilling format.

Because of this, the collar table reports 15,287.45 drilled meters, and assay table reports 15,301.07 assayed meters (99.9% of the total drilling).

14.4.4 TOPOGRAPHY CONTOUR LINES

Original topography provided from 1m contour lines, presented no errors and it was possible to generate a 3D triangulation, validated and adjusted, for the project area.

14.5 RESOURCE ESTIMATION

14.5.1 BLOCK MODEL DEFINITION

From the validated grade data, corresponding to 15,301.07 samples of total copper, the resource estimateof the Berta deposits was carried out.

The estimation was executed by Minesight version 7.0-4, using geostatistical methods available in both software packages.

The block model box was determined by considering the following factors:

- To completely contain the mineralization geological models.
- To provide enough work space for pit optimization.

Thus the Berta Surand Berta Central block model box represents an area of 775 x 1,450 and a 575m depth, as it appears Figure 14.4: Berta Sur Block Model Box

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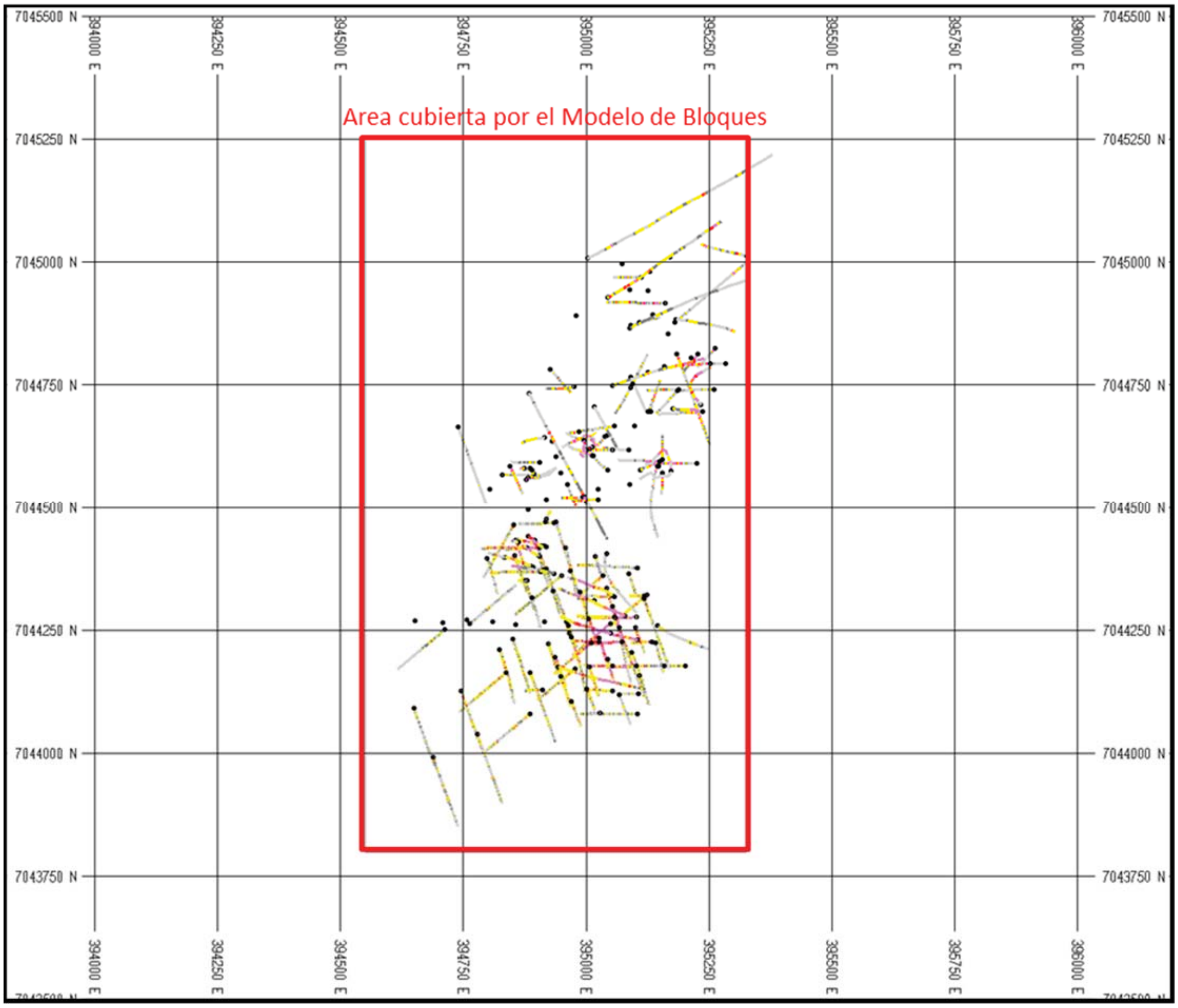
Figure 14.4: Berta Sur Block Model Box



Selected block model size was 5x5x5 meters in order to match the bench height and proposed mining equipment, this change from the first resource estimate does not represent a major variation given the nonoccurrence of phenomenon caused by the support change and geological dilution effect.

Block model properties for Berta Sur and Berta Centralare summarized in Table 14.9

Table 14.9: Berta Sur and Berta Central Block Model Properties

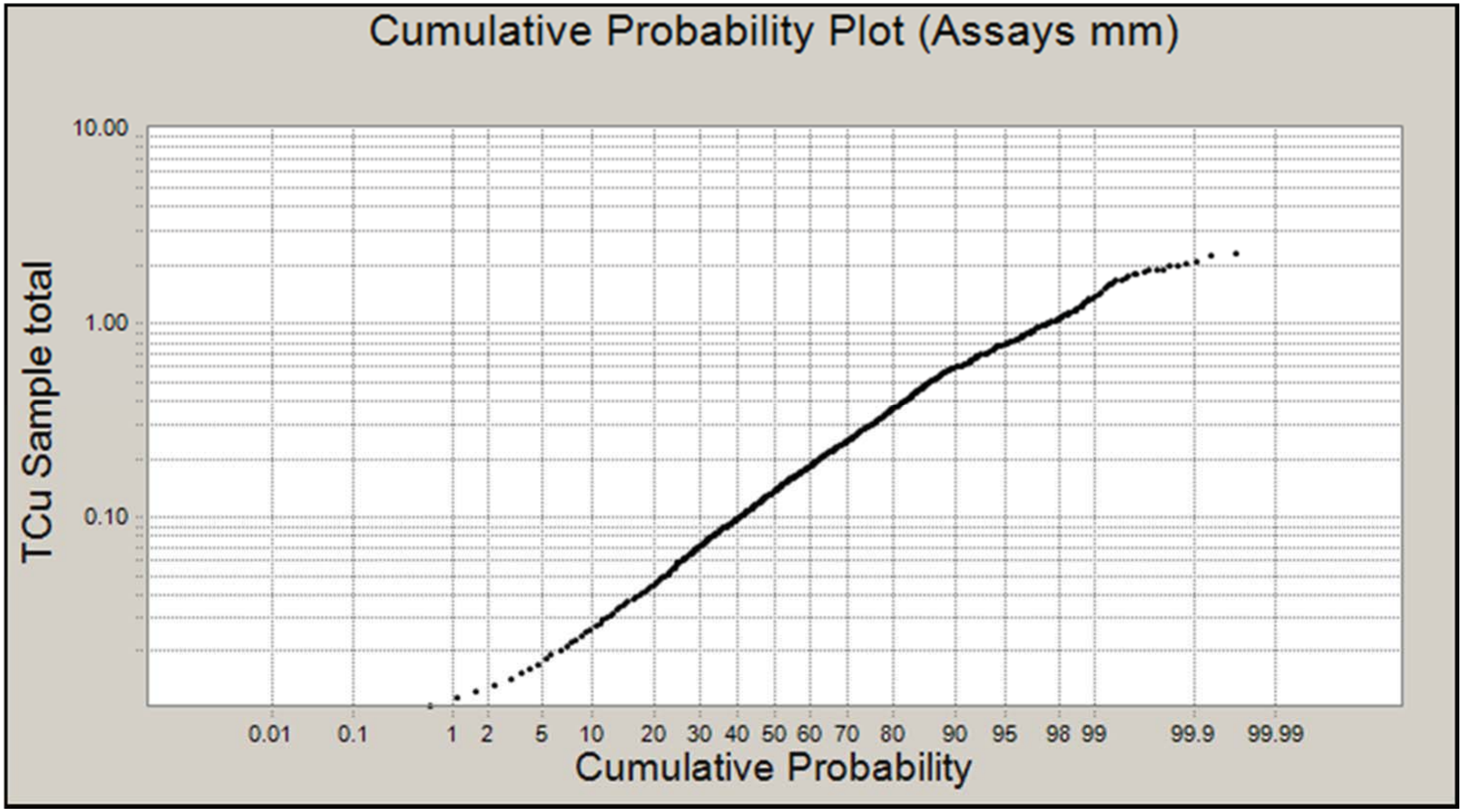
	Min. Coordinate (m)	Max. Coordinate (m)	Length (m)	# of Blocks	Block Size
East	394,550	395,325	775	155	**5**
North	7,043,800	7,045,250	1,450	290	**5**
Elevation	1,250	1,825	575	115	**5**
			Total # of Blocks	**5,169,250**	

14.5.2 SAMPLE CAPPING

Because of the variability of the deposit and the presence of occasional high grades, it was considered necessary to examine the grade distribution of the copper samples, in order to evaluate the necessity for grade capping.

This was performed by plotting the cumulative probability of the total Cu samples, as shown in Figure 14.5: Sample Capping

Figure 14.5: Sample Capping



After reviewing the results, it was decided not to cap %CuT grades, as there was no clear evidence of out of range high values within the distribution. Moreover, the slight break in the distribution at around 1.8% %CuT is located on over 99% of the total distribution, thus further validating this decision.

14.5.3 COMPOSITING

Great part of the data is represented by a 2 meter support; the vertical dimension of the selected block corresponds to 5 m, besides its dimension on the X-Y plan are also 5 m. On the other hand, horizontal distribution of samples mainly located in trenches and channel samples, does not allow performing a compositing according to the bench height method, given that the data of trenches and channels from open pit and underground are lost. For this reason it was decided to perform a compositing though the fixed length method (2.5 m), thus correcting the problem generated before. The fixed length of 2.5 m corresponds to the fact that great part of the original sample support is 2 m, therefore distribution and statistics of the studied variables do not suffer major alterations and/or modifications out of the expected. Means are maintained and lower variance and coefficient of variation are observed.

A total of 12,988 m of samples with grades higher than 0.01 %CuT are available for the resource estimation, when regularizing those samples, the data amount reduces to 9,670 composites higher than 0.01 %CuT, from which only 4,427 composites higher than 0.01 %CuT are located in the estimating zone denominated Oxides Zone (OXI Zone),

Figure 14.6: Zones of Estimation

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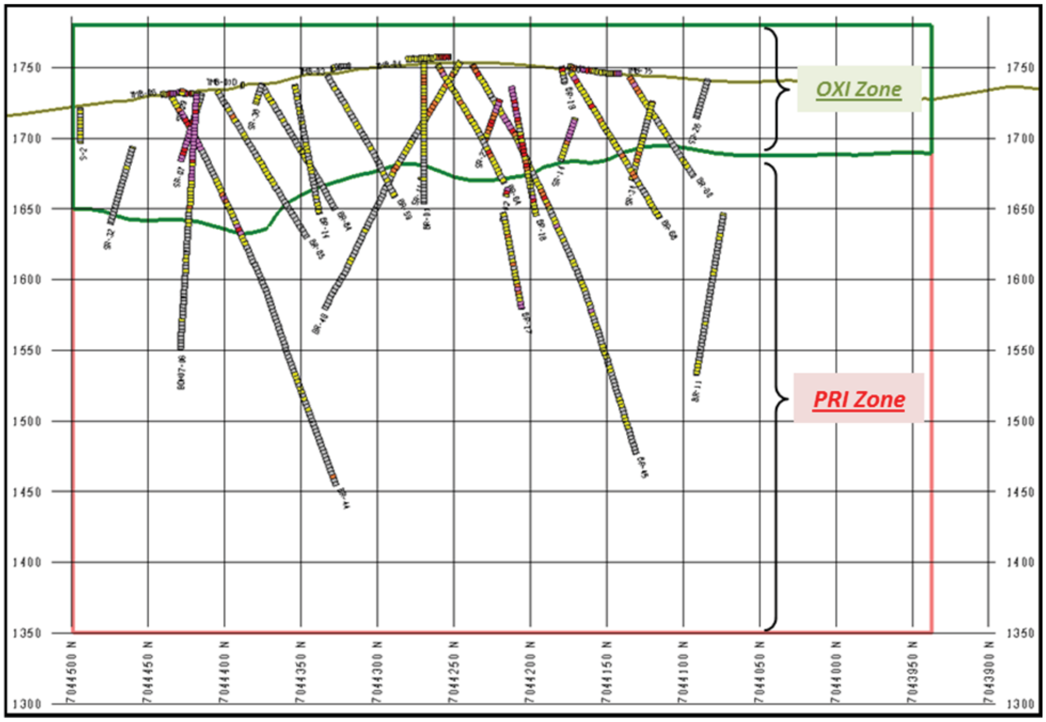
Figure 14.6: Zones of Estimation



Composites were calculated from the 4,533 samples with capped total copper greater than 0.01% missing and nonassayed intervals, stopes and samples with length lower than 1 m, not considered. In this way 3,103 composites were obtained.

14.5.4 DOMAIN DEFINITION

14.5.4.1 MINERALIZED BODY

There are six mineralized bodies at Berta, as defined by MCC. Fromsouth to north the bodies are:

- Berta Sur
- Trinchera-Salvador
- Carmen-Gemela
- Nueva
- Berta II
- Chico

These bodies were constructed from section interpretations (every 50 m) delivered by MCC and reinterpreted in level plans every 5 m with the intention to mark in more detail the continuity of the mineralization recognized at surface. Obtained solids were corrected according to the topography in the upper part and by the surface that marks the limit between the oxides – sulphides mineralization in the lower part. As a consequence only solids contained in the oxides zone of the mineralization were evaluated. See Figure14.7 and Figure 14.8

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Figure 14.7: Mineralized Bodies 3D View

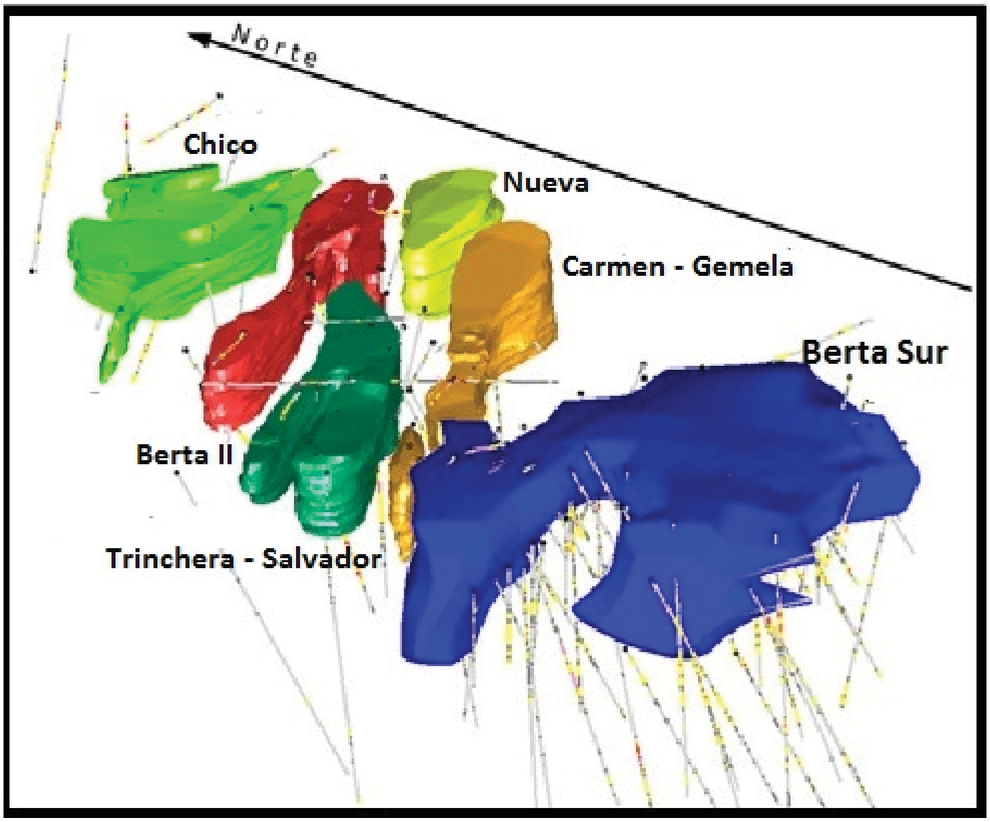
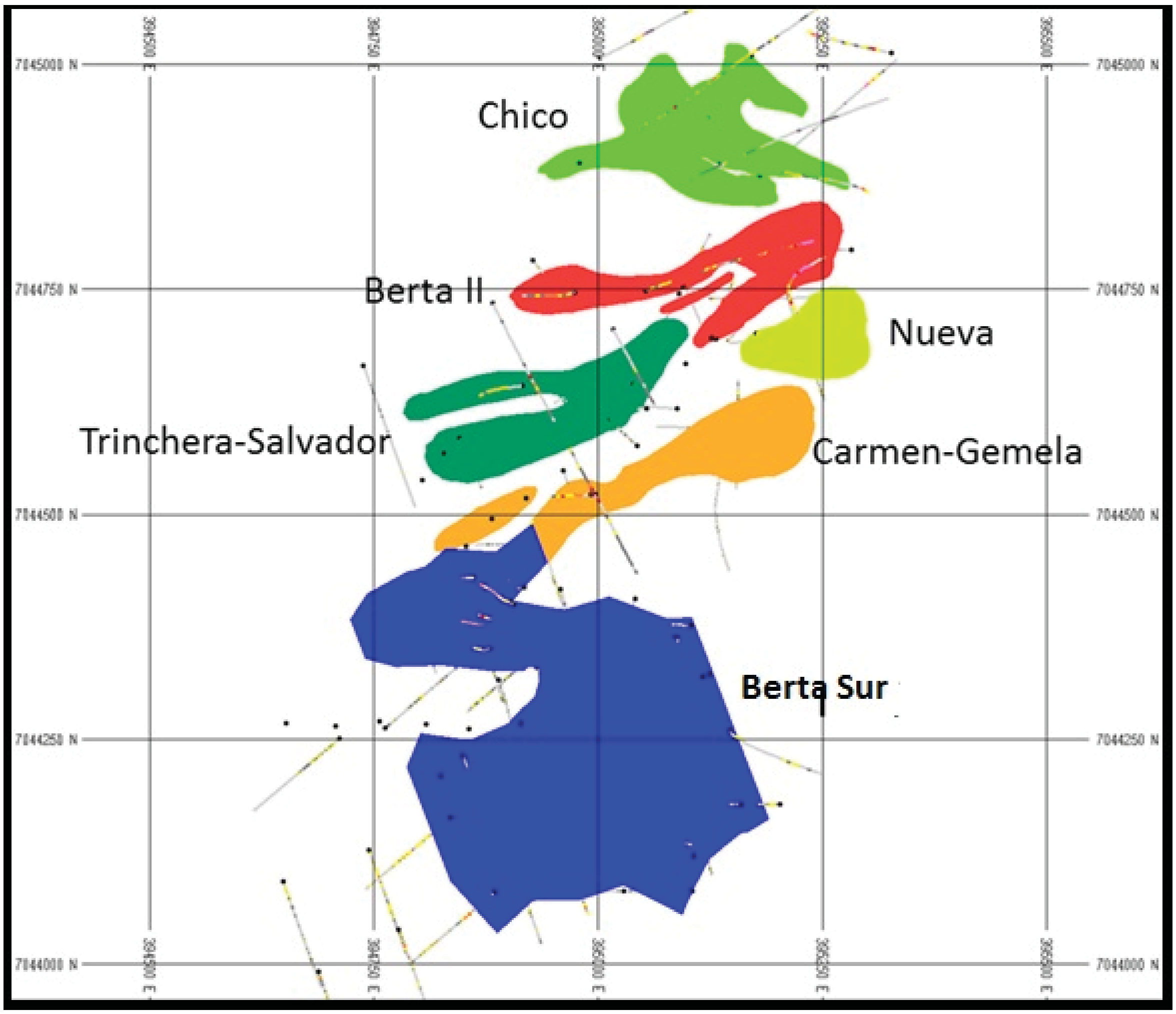


Figure 14.8: Mineralized Bodies Plan View



14.5.4.2 MINERALIZED ZONES

The section based geological interpretation included a top of sulfide/base of oxide which was used to generate a 3D surface forming the base to the oxide zone resource estimate. The geological interpretation of distribution of copper oxide species, %CuS grade, and variations in solubility ratio were utilized to define two geological solids above the base of oxides, namely the Oxide Body (Zone 1) and the Low Grade Oxide Body (Zone 2).

Figure 14.9: Berta Sur and Berta Central Mineralized Body

Figure 14.9: Berta Sur and Berta Central Mineralized Body

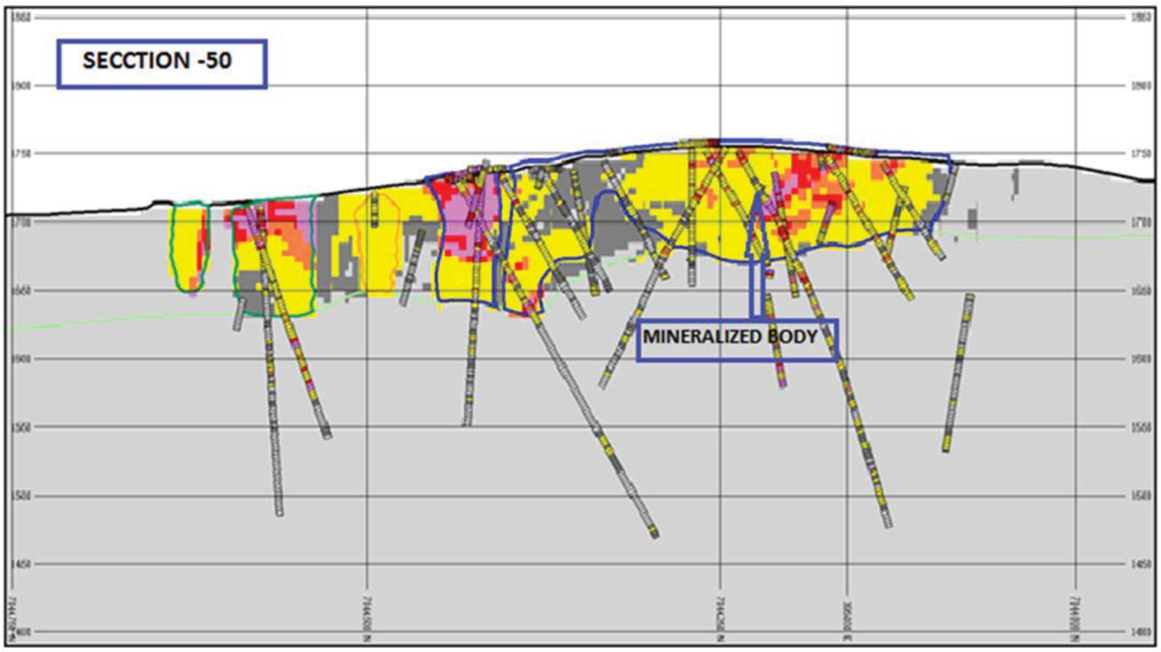


Table 14.10: Estimation Domains

Estimation Domain	Deposit	Mineralized Zone	Composite and Block Model Code
1	Berta	Oxides Mineralized Body	1
2		Oxides Low Grade Body	2

Finally, the composite and block code assignments by mineralized zone were made considering the location of its centroid, in relation to the mineralized solid.

14.6 EXPLORATORY STUDY

To characterize the statistics of the samples and composites, it was necessary to carry out an exploratory study consisting of:

1) Basic statistics
2) Histograms
3) Proportional Effect

14.6.1 SAMPLE BASIC STATISTICS

Once the database wasreviewed and validated, basic statistics were calculated for %CuT and %CuS grades, as follows (Table 14.11):

Table 14.11: CuT & CuS Sample Statistics

	Zone	# of Samples	Min	Max	Mean	Std. Dev.	Variance	Coeff. Of Var
Cu T	1	5,688	0.010	8.700	0.338	0.476	0.226	1.41
	2	1,948	0.010	1.570	0.122	0.122	0.015	1.43
	Total	7,636	0.010	8.700	0.430	0.430	0.185	1.57
Cu S	1	3,812	0.010	5.250	0.296	0.296	0.087	1.34
	2	767	0.010	0.890	0.094	0.094	0.009	1.43
	Total	4,579	0.010	5.250	0.279	0.279	0.078	1.43

(Note): -1 (not assayed samples) were excluded from this analysis

From the previous table the following can be concluded:

- Samples located in themineralized area represent 29.6% of the total samples.
- Mean solubility ratio for the samples is 0.70.

14.6.2 COMPOSITE BASIC STATISTICS

Once the bench-height composites werecalculated, basic statistics were calculated for %CuT and %CuS grades, as follows (Table 14.12):

Table 14.12: CuT & CuS Composite Statistics

	Zone	# of Samples	Min	Max	Mean	Std. Dev.	Variance	Coeff. Of Var
Cu T	1	4,461	0.010	6.000	0.305	0.390	0.152	1.28
	2	1,244	0.010	0.996	0.07	0.092	0.008	1.30
	Total	6	0.010	6.000	0.254	0.361	0.13	1.42
Cu S	1	3,103	0.010	3.150	0.208	0.263	0.069	1.26
	2	439	0.010	0.651	0.056	0.083	0.007	1.47
	Total	3,542	0.010	3.150	0.189	0.253	0.064	1.33

(Note): -1 (not assayed samples) were excluded from this analysis

As expected, sample and composite grade trends are similar, with amean solubility ratio of 0.69 for the composite data.

The difference in the number of samples with %CuT and %CuS assays, suggested determining the %CuS grade indirectly from the %CuT estimation and the subsequent calculation of %CuS in function of the relationship %CuT - %CuS. For this case, this relationship corresponds to a 2nd order polynomial regression.

14.6.3 COMPOSITE GRADE HISTOGRAMS

Next, %CuT and %CuS distributions are presented. Below, analysis for total copper is shown in Figure 14.10:

Figure 14.10: %CuT Grade Histogram (Blue: Oxide Body Zone 1, Green: Low Grade Oxide Body Zone 2)

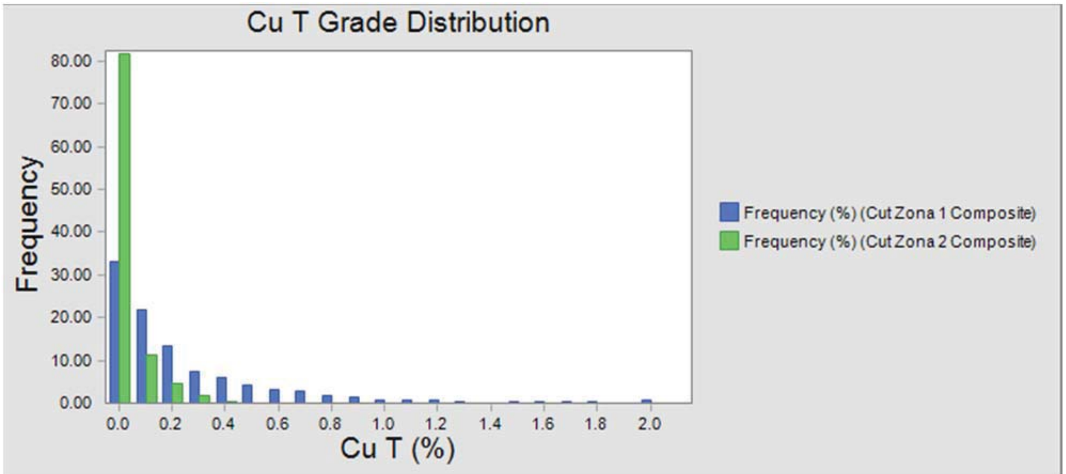


Table 14.13: %CuT Grade Distribution

Cutoff (%CuT)	Item	Count	Acum. Percent	# of Samples	Average	Std. Deviation
0.00	CuT	5,705	100.00	2,483	0.254	0.361
0.10	CuT	3,222	56.48	1,121	0.414	0.414
0.20	CuT	2,101	36.83	653	0.558	0.450
0.30	CuT	1,448	25.38	349	0.700	0.479
0.40	CuT	1,099	19.26	269	0.812	0.499
0.50	CuT	830	14.55	189	0.931	0.521
0.60	CuT	641	11.24	145	1.045	0.543
0.70	CuT	496	8.69	123	1.163	0.565
0.80	CuT	373	6.54	78	1.300	0.590
0.90	CuT	295	5.17	63	1.420	0.610
1.00	CuT	232	4.07	38	1.549	0.629
1.10	CuT	194	3.40	39	1.646	0.644
1.20	CuT	155	2.72	34	1.772	0.664
1.30	CuT	121	2.12	18	1.919	0.682
1.40	CuT	103	1.81	6	2.019	0.692
1.50	CuT	97	1.70	15	2.055	0.698
1.60	CuT	82	1.44	17	2.147	0.722
1.70	CuT	65	1.14	13	2.276	0.761
1.80	CuT	52	0.91	12	2.408	0.798
1.90	CuT	40	0.70	-	2.576	0.841
2.00	CuT	34	0.60	34	2.689	0.864

Figure 14.11: %CuS Grade Histogram (Blue: Oxide Body Zone 1, Green: Low Grade Oxide Body Zone 2)

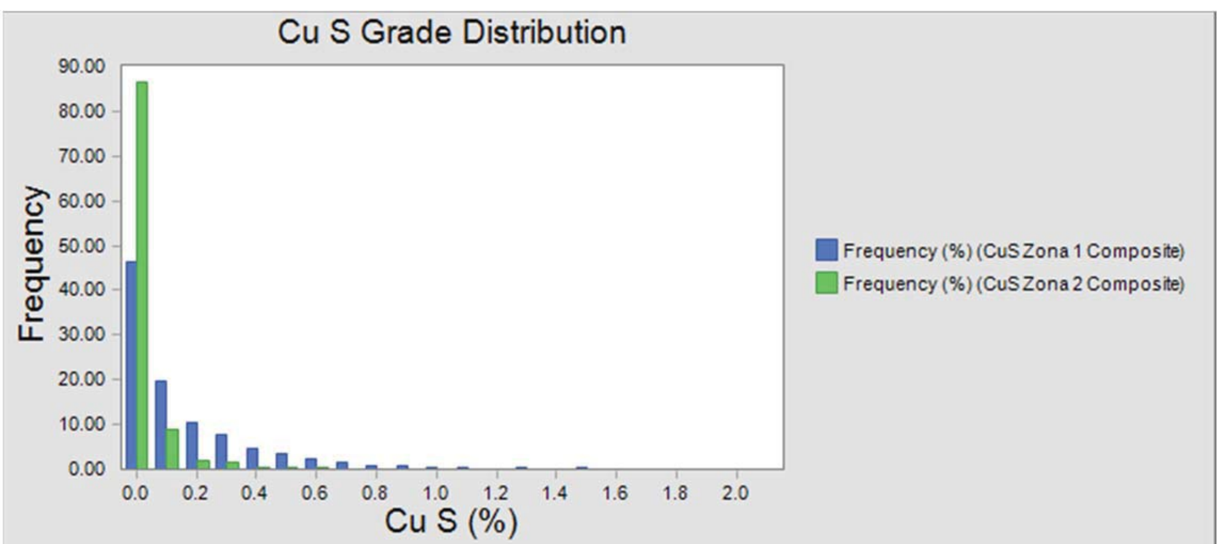


Table 14.14: %CuS Grade Distribution

Cutoff (%CuS)	Item	Count	Acum. Percent	# of Samples	Average	Std. Deviation
0.00	CuS	3,542	100.00	1,823	0.189	0.253
0.10	CuS	1,719	48.53	650	0.346	0.288
0.20	CuS	1,069	30.18	329	0.471	0.303
0.30	CuS	740	20.89	247	0.572	0.315
0.40	CuS	493	13.92	148	0.684	0.332
0.50	CuS	345	9.74	110	0.788	0.349
0.60	CuS	235	6.63	78	0.899	0.374
0.70	CuS	157	4.43	52	1.028	0.399
0.80	CuS	105	2.96	24	1.167	0.424
0.90	CuS	81	2.29	27	1.260	0.441
1.00	CuS	54	1.52	15	1.413	0.470
1.10	CuS	39	1.10	8	1.555	0.484
1.20	CuS	31	0.88	5	1.663	0.487
1.30	CuS	26	0.73	6	1.739	0.497
1.40	CuS	20	0.56	2	1.854	0.515
1.50	CuS	18	0.51	6	1.895	0.528
1.60	CuS	12	0.34	4	2.078	0.566
1.70	CuS	8	0.23	3	2.282	0.600
1.80	CuS	5	0.14	1	2.605	0.530
1.90	CuS	4	0.11	-	2.803	0.337
2.00	CuS	4	0.11	4	2.803	0.337

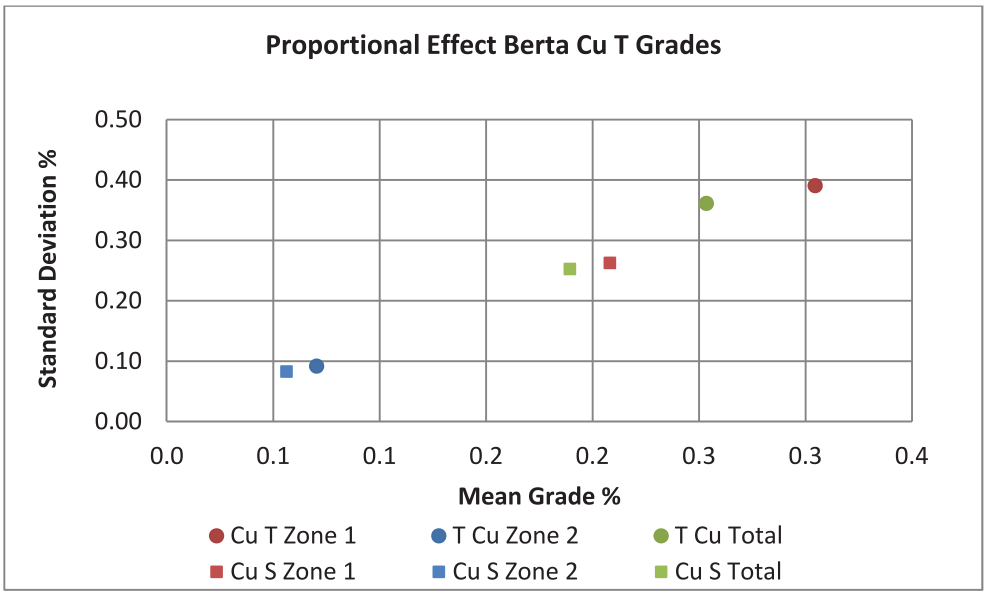
It is deduced from these results that over 71% of the composites have grades greater than 0.10 %CuT, and a cumulative mean grade of 0.39 %CuT.

Finally, a significant number of samples have grades between 0.1 and 0.4 %CuT grades, representing 47.7% in the Berta Sur deposit.

14.6.4 PROPORTIONAL EFFECT

In order to confirm the geostatistical treatment of the Oxide Body (Zone 1), and Low Grade Oxide Body (Zone 2), the proportional effect was analyzed. This study was performed for total copper grades. The results are indicated Figure 14.12: Proportional Effect

Figure 14.12: Proportional Effect



These results confirm the existence of different copper grade populations within the Berta Sur data, and support the separate domain estimation.

14.7 VARIOGRAPHY

Variography was considered for Berta Sur and Berta Centraldeposits, because the estimation was performed by the "Ordinary Kriging" method.

Separate variograms were completed within the Oxide Body and the Low Grade Oxide Body, because of their different grade distributions (Figure 14.13 and Table 14.16 14.14)

Variography was performed for total copper with the Covariance variogram type chosen.

New variograms were determined from the new amount of available compositings, generally and particularly the used anisotropy directions are maintained, only changing some scopes from the first structures, those change are irrelevant given that the data variance is not changed radically. That being said, it was possible to use the variographic plan utilized before, and it is left for later evaluation to perform a more detailed study after validating the directions and distributions proposed for the mineralization distribution.

The nugget effect was deduced from the Downhole variogram, because it provides more information along the drill holes. Table 14.15 and Table 14.16 summarizes the parameters used in the experimental variography:

Table 14.15: Oxide Body (Zone 1) %CuT Covariance Parameters

Direction	# of Lags	Lag (m)	Lag Tolerance	Azm.	DIP	Azm. Tol. Angle	DIP Tol. Angle	Horiz. Band (m)	Vert. Band (m)
Major Axis	20	10	5	293	-81	22.5	22.5	50	50
Secondary Axis	20	10	5	59	-5	22.5	22.5	50	50
Vertical Axis	20	10	5	150	-7	22.5	22.5	50	50

Table 14.16: Low Grade Oxide Body (Zone 2) %CuT Covariance Parameters

Direction	# of Lags	Lag (m)	Lag Tolerance	Azm.	DIP	Azm. Tol. Angle	DIP Tol. Angle	Horiz. Band (m)	Vert. Band (m)
Major Axis	15	10	5	293	-81	30	30	40	50
Secondary Axis	15	10	5	59	-5	30	30	40	50
Vertical Axis	15	10	5	150	-7	30	30	40	50

From these results, the following modeled variograms were derived:

Figure 14.13: %CuT Semi – Variogram Model (Covariance) for Oxide Body (Zone 1)

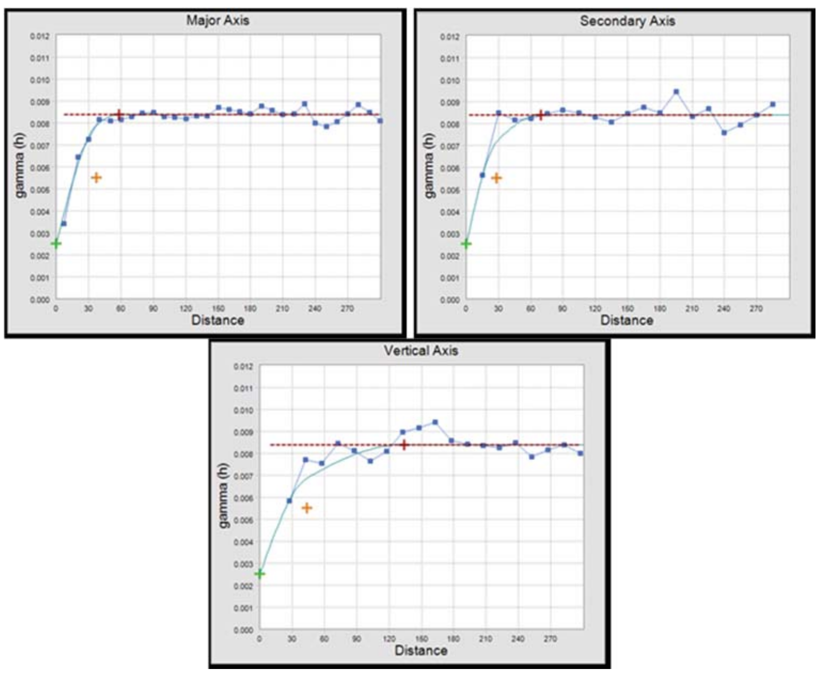
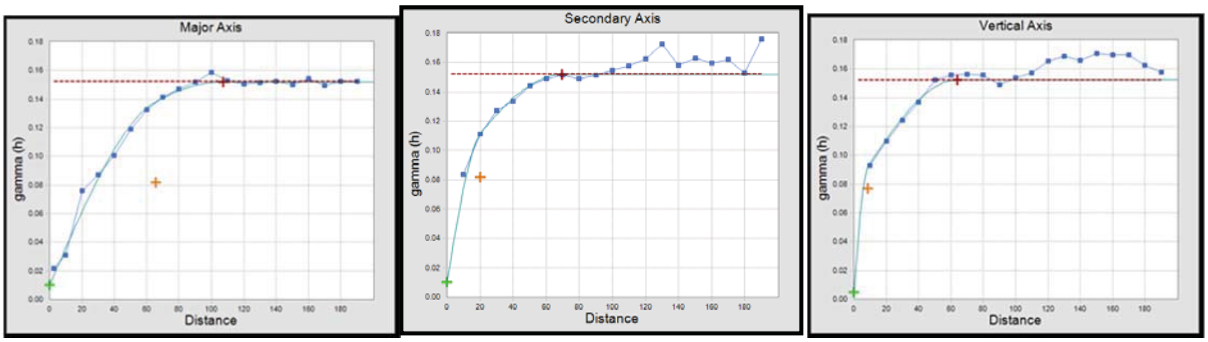


Figure 14.14: %CuT Semi – Variogram Model (Covariance) for Low Grade Oxide Body (Zone 2)



The variographic model is summarized as follows, Table 14.17

Table 14.17: Variographic Model Table 14.1:

Zone	Tvar	Mod	C_0	C_1	C_2	Range 1 (m) Structure 1	Range 2 (m) Structure 1	Range 3 (m) Structure 1	Range 1 (m) Structure 2	Range 2 (m) Structure 2	Range 3 (m) Structure 2	C_0 / Sill	Var
1	Cov	Esf	0.0100	0.072	0.0700	66	20	8	107	70	64	0.21	0.1520
2	Cov	Esf	0.0025	0.003	0.0029	38	28	43	58	70	134	0.42	0.0084

14.8 DENSITY

Specific gravity was calculated from the arithmetic mean of 16 samples of drill core provided by MCC, with a mean density of 2.56

Table 14.18: Density

Kerosene Unit Price = 0.871	Net Weight	"Kerosened" Wieght		Net Weights (grams)		Kerosened Samples (grams)		Kerosene		Unit Weight
Sample Number	Air	Air	Submerged	Air	Submerged	Air	Submerged	Weight (grams)	Volume (cm^3)	(g/cm^3)
BDH07-06-12,38-12,55-12	698.00	721.99	411.93	698.00	415.48	721.99	411.93	23.99	27.54	**2.47**
BDH07-06,23-23,11-12	419.54	430.69	247.02	419.54	248.67	430.69	247.02	11.15	12.80	**2.46**
BDH07-06-43,7-43,86-12	455.42	470.17	270.28	455.42	272.46	470.17	270.28	14.75	16.93	**2.49**
BDH07-06-62,1-62,33-12	839.84	869.91	508.30	839.84	512.75	869.91	508.30	30.07	34.52	**2.57**
BDH07-06-67,75-67,9-12	512.31	529.74	313.83	512.31	316.41	529.74	313.83	17.43	20.01	**2.62**
BDH07-06-81-81,17-12	638.23	659.97	381.15	638.23	384.37	659.97	381.15	21.74	24.96	**2.51**
BDH07-07-12,28-12,4-12	449.50	462.72	266.35	449.50	268.31	462.72	266.35	13.22	15.18	**2.48**
BDH07-07-24,84-25,1-12	937.72	968.57	558.10	937.72	562.67	968.57	558.10	30.85	35.42	**2.50**
BDH07-07-42,8-42,97-12	587.31	608.27	347.09	587.31	350.19	608.27	347.09	20.96	24.06	**2.48**
BDH07-07-56,8-56,98-12	638.81	657.07	385.54	638.81	388.24	657.07	385.54	18.26	20.96	**2.55**
BDH07-07-100,5-100,74-12	936.92	971.57	574.94	936.92	580.07	971.57	574.94	34.65	39.78	**2.63**
BDH07-07-116,84-117,06-12	942.72	970.85	583.50	942.72	587.67	970.85	583.50	28.13	32.30	**2.66**
BDH07-07-140,7-140,93-12	949.86	980.57	585.45	949.86	590.00	980.57	585.45	30.71	35.26	**2.64**
BDH07-07-227,07-227,32-12	974.04	1,005.27	600.14	974.04	604.77	1,005.27	600.14	31.23	35.86	**2.64**
BDH07-08-365,47-365,64-12	653.19	673.10	403.26	653.19	406.21	673.10	403.26	19.91	22.86	**2.64**
BDH07-08-371-371,2-12	985.08	1,015.95	596.20	985.08	600.77	1,015.95	596.20	30.87	35.44	**2.56**

14.9 BLOCK MODEL DIMENSION AND GRADE ESTIMATION

A total of 12,988 m of samples with grades higher than 0.01 %CuT are available for the resource estimation, when regularizing those samples, the data amount reduces to 9,670 composites higher than 0.01 %CuT, from which only 4,427 composites higher than 0.01 %CuT are located in the estimating zone denominated Oxides Zone (OXI Zone)

Grade elements existing in these block models, are:

1) Total copper in percent (%CuT)
2) Soluble copper in percent (%CuS, in function of the relationship %CuT - %CuS)

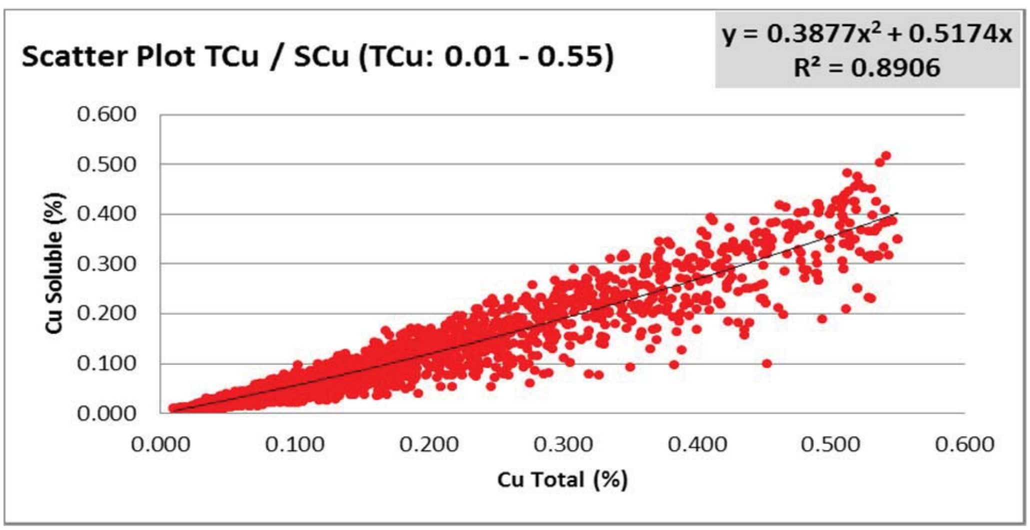
14.9.1 ESTIMATION DOMAINS AND ESTIMATION PLANS

The estimation plan considered on this occasion keeps the same parameters as the previous estimation, given that it is intended to obtain similar and comparable results, despite the change of the block size.

%CuT estimations were performed by Ordinary Kriging. In all the cases, %CuS was calculated in function of the %CuT – %CuS relationship. For this purpose, the relationship %CuT - %CuS was studied by constructing scatterplots according to three ranges %CuT grades.

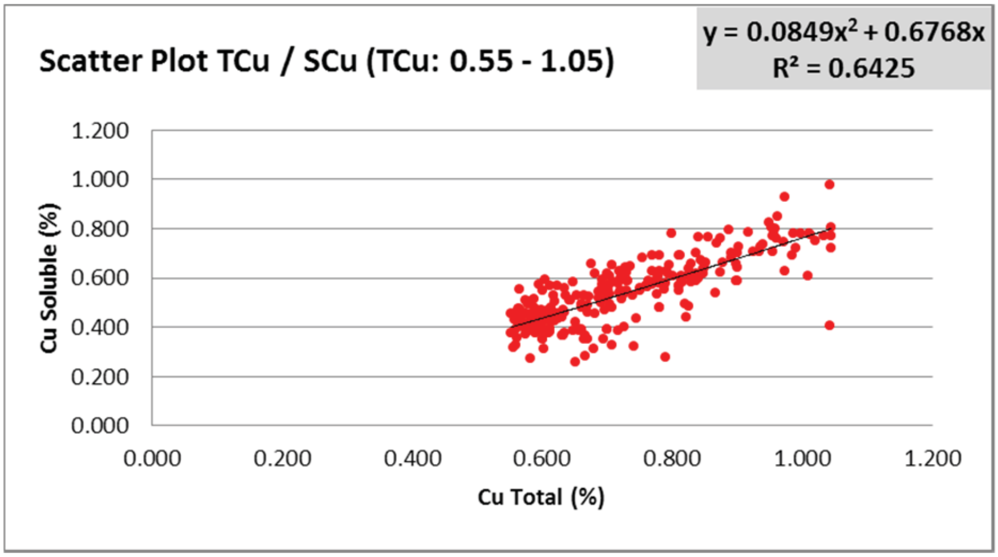
- %CuT: 0,01% to 0.55% (Figure 14.15)
- %CuT: 0.55% to 1.05% (Figure 14.16)
- %CuT: 1.05% to 5.00% (Figure 14.17)

Figure 14.15: Scatter Plot for Range %CuT 0.01 – 0.55 %CuT



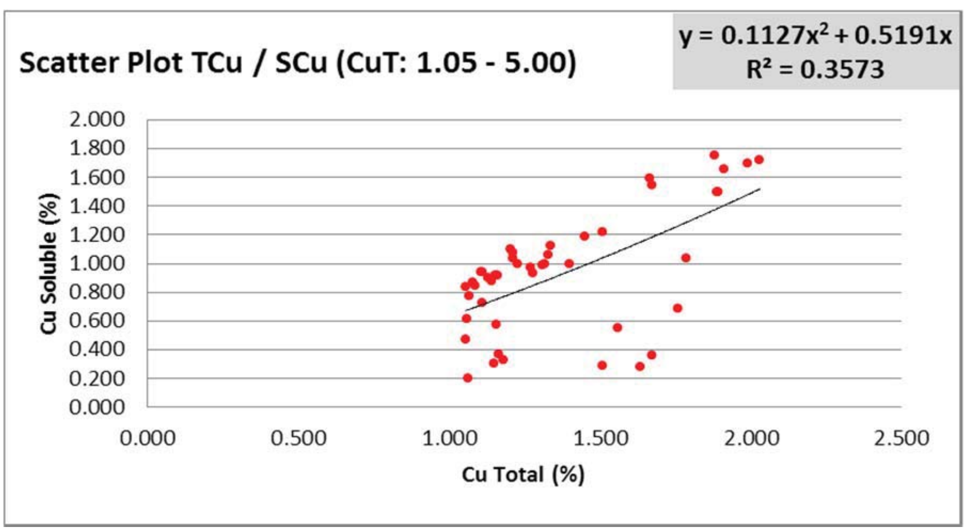
Regression curve corresponds to a polynomial adjustment, in this case the ratio to determine the %CuS corresponds to: %**CuS= (0.3877 * %CuT^2) + (0.5174 * %CuT)**

Figure 14.16: Scatter Plot for Range %CuT 0.55 – 1.05 %CuT



Regression curve corresponds to a polynomial adjustment, in this case the ratio to determine the %CuS corresponds to: %**CuS= (0.0849 * %CuT^{2}) + (0.6768 * %CuT)**

Figure 14.17: Scatter Plot for range %CuT 1.05 – 5.00 %CuT



Regression curve corresponds to a polynomial adjustment, in this case the ratio to determine the %CuS corresponds to: %**CuS= (0.1127 * %CuT^{2}) + (0.5191 * %CuT)**

This method was used because 25% of the samples have no %Cu Sassays. Thus the estimation was completed for the%CuT grade and the %CuS and the solubility ratio were then derived from the %CuT grade.

For %CuT grade, estimation was completed separately for each of the Oxide and Low Grade Oxide domains, as shown in the following figures (Figure 14.18: Section -50, Estimation Domains

to Figure 14.29)

Figure 14.18: Section -50, Estimation Domains

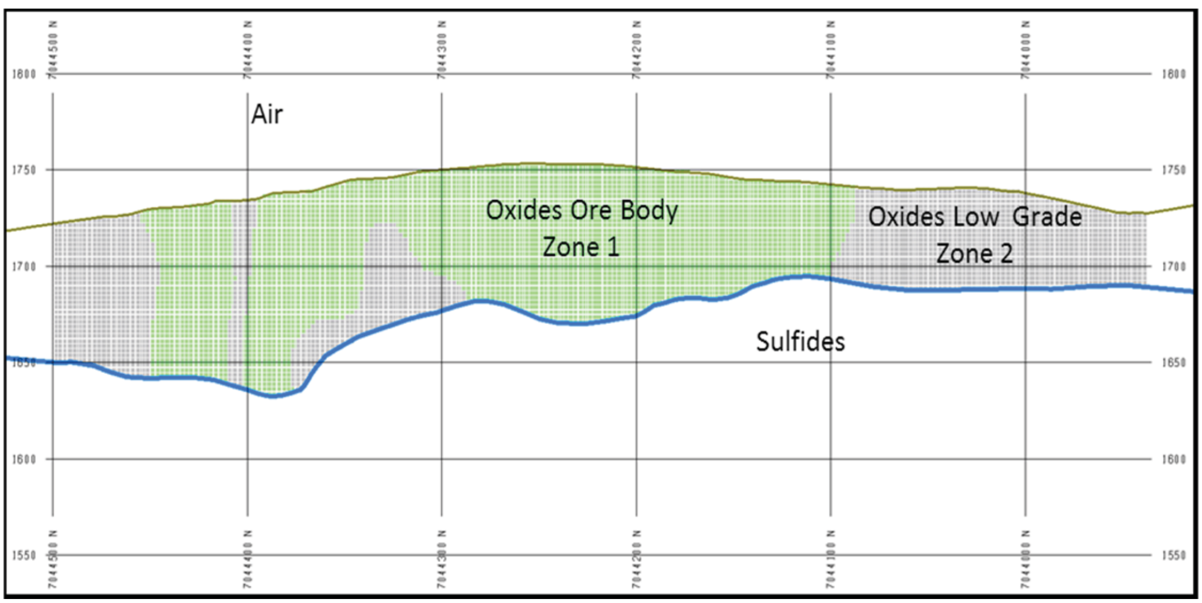
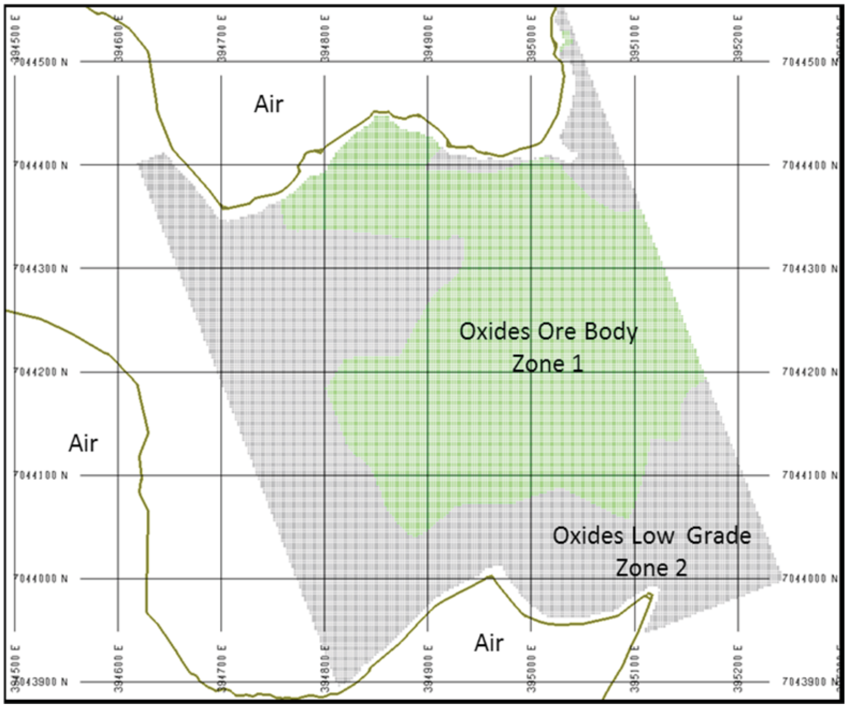


Figure 14.19: Plan 1,730, Estimation Domains



The estimation of each of these domains was performed only with composites belonging to the same population.

See Table 14.19 summarizes the resource estimation plans:

Table 14.19: Estimation Plan

	Zone 1	Zone 2
Estimation Method	OK	OK
Min. # of Composites	2	2
Max. # of Composites	16	16
Max. # of Composites for Hole	4	4
Search Direction	-	-
Search Type	No Octants	No Octants
Search Radius (m)	144 m (Major dir)	84 m (Major dir)
	102 m (Secondary dir)	82 m (Secondary dir)
	76 m (Vertical dir)	44 m (Vertical dir)

14.10 RESOURCE CATEGORIZATION

The resource categorization was established according to the quality of the estimation. This methodology guarantees that all blocks were estimated, at least, with two composites.

Measured and indicated resources require composites coming from, at least, two different drill hole and restricted search radii of 0 m to 35 m (measured) and 0 m to 70 m (indicated).

Any block that does not meet these conditions, is assigned tothe inferred category.

Table 14.20 shows the criteria considered for the resource categorization:

Table 14.20: Resource Categorization Criteria

# Drill holes	Dist / Comp			Classification	Code
> = 4	0.00	-	35.00	Measured	1
> = 4	35.00	-	70.00	Indicated	2
> = 2		>	70.01	Inferred	3
2	0.00	-	35.00	Indicated	2
2	35.00	-	70.00	Inferred	3
2		>	70.01	Inferred	3
1	0.00	-	35.00	Inferred	3
1	35.00	-	70.00	Inferred	3
1		>	70.01	Inferred	3

14.10.1 RESOURCE INVENTORY

The grade – tonnage curve determined in the previous report considered only the body denominated Berta Sur, which defined the following resource inventory shown in Table 14.21

Table 14.21: Total Tonnage-Grade Curves Berta Sur

BERTA [OXI Zone - Ordinary Kriging]												
	Measured Resources			Indicated Resources			Measured + Indicated Resources			Inferred Resources		
Cutoff	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol
0,10	10.672	0,318	0,212	7.725	0,169	0,100	18.397	0,255	0,165	6.465	0,163	0,095
0,15	8.498	0,367	0,249	4.250	0,206	0,125	12.748	0,314	0,207	3.705	0,193	0,115
0,20	6.736	0,418	0,287	1.814	0,253	0,157	8.550	0,383	0,259	1.363	0,229	0,139
0,25	5.254	0,473	0,330	691	0,306	0,196	5.945	0,454	0,314	265	0,271	0,169
0,30	4.170	0,525	0,371	261	0,367	0,243	4.431	0,516	0,364	21	0,318	0,204
0,35	3.423	0,569	0,407	126	0,415	0,283	3.548	0,564	0,402	2	0,368	0,243
0,40	2.850	0,608	0,439	60	0,463	0,323	2.910	0,605	0,436			
0,45	2.372	0,646	0,469	29	0,507	0,361	2.400	0,644	0,468			
0,50	1.933	0,684	0,500	12	0,559	0,405	1.945	0,684	0,499			

The new resource inventory including the Berta Central bodies is shown in the following Tonnage – Grade curve in Table 14.22

Table 14.22: Total Tonnage-Grade Curves Berta Sur and Berta Central

BERTA + BERTA CENTRAL [OXI Zone - Ordinary Kriging]												
	Measured Resources			Indicated Resources			Measured + Indicated Resources			Inferred Resources		
Cutoff	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol
0,10	16.498	0,341	0,229	8.653	0,225	0,141	25.150	0,301	0,199	4.845	0,242	0,154
0,15	13.275	0,393	0,268	5.780	0,274	0,175	19.055	0,357	0,240	3.249	0,300	0,195
0,20	10.487	0,451	0,312	3.336	0,348	0,230	13.822	0,427	0,292	2.039	0,377	0,252
0,25	8.355	0,510	0,357	1.961	0,437	0,297	10.316	0,496	0,346	1.402	0,446	0,305
0,30	6.791	0,564	0,400	1.289	0,524	0,364	8.080	0,558	0,394	932	0,534	0,373
0,35	5.647	0,613	0,439	892	0,613	0,433	6.539	0,613	0,438	695	0,608	0,430
0,40	4.757	0,658	0,475	664	0,696	0,497	5.420	0,663	0,478	586	0,650	0,463
0,45	3.978	0,704	0,511	531	0,764	0,549	4.509	0,711	0,516	479	0,702	0,503
0,50	3.262	0,754	0,550	429	0,833	0,601	3.691	0,763	0,556	362	0,775	0,558

The following tables

(Table 14.23 to Table 14.28) showdetails of calculations of measured, indicated and inferred resources for each body.

Table 14.23: Measured, Indicated and Inferred Resources Berta Sur

Recursos Berta Sur [OXI Zone - Ordinary Kriging]

	Measured Resources			Indicated Resources			Measured + Indicated Resources			Inferred Resources		
Cutoff	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol
0,10	10.972	0,319	0,213	4.423	0,181	0,108	15.394	0,279	0,183	2.105	0,182	0,108
0,15	8.853	0,365	0,247	2.800	0,214	0,130	11.653	0,329	0,219	1.296	0,218	0,133
0,20	6.892	0,420	0,288	1.332	0,260	0,162	8.225	0,394	0,268	720	0,256	0,159
0,25	5.385	0,475	0,331	561	0,314	0,202	5.946	0,460	0,319	343	0,291	0,184
0,30	4.288	0,526	0,372	261	0,363	0,240	4.549	0,517	0,365	127	0,328	0,212
0,35	3.505	0,572	0,409	118	0,415	0,282	3.623	0,567	0,405	22	0,376	0,249
0,40	2.939	0,610	0,440	57	0,462	0,322	2.996	0,607	0,438	3	0,439	0,302
0,45	2.458	0,646	0,470	25	0,512	0,365	2.483	0,645	0,469	1	0,469	0,328
0,50	2.009	0,685	0,500	11	0,561	0,407	2.020	0,684	0,500			

Table 14.24: Measured, Indicated and Inferred resources Trinchera-Salvador

Recursos Trinchera-Salvador [OXI Zone - Ordinary Kriging]

	Measured Resources			Indicated Resources			Measured + Indicated Resources			Inferred Resources		
Cutoff	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol
0,10	1.161	0,408	0,283	1.241	0,278	0,183	2.402	0,341	0,231	844	0,397	0,268
0,15	1.004	0,452	0,316	872	0,343	0,231	1.876	0,402	0,276	721	0,443	0,302
0,20	862	0,498	0,351	587	0,426	0,294	1.450	0,469	0,328	574	0,513	0,354
0,25	763	0,534	0,379	429	0,502	0,353	1.192	0,522	0,370	477	0,571	0,398
0,30	674	0,568	0,406	360	0,546	0,388	1.034	0,560	0,400	409	0,620	0,436
0,35	601	0,597	0,430	313	0,578	0,414	914	0,591	0,425	354	0,666	0,471
0,40	528	0,628	0,456	276	0,606	0,436	804	0,621	0,449	316	0,702	0,498
0,45	458	0,660	0,481	239	0,633	0,459	697	0,651	0,473	278	0,739	0,527
0,50	384	0,695	0,509	197	0,667	0,486	581	0,686	0,501	229	0,797	0,569

Table 14.25: Measured, Indicated and Inferred resources Carmen-Gemela

Recursos Carmen-Gemela [OXI Zone - Ordinary Kriging]												
	Measured Resources			Indicated Resources			Measured + Indicated Resources			Inferred Resources		
Cutoff	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol
0,10	1.997	0,443	0,305	954	0,341	0,226	2.951	0,410	0,280	306	0,191	0,119
0,15	1.535	0,540	0,376	632	0,452	0,306	2.167	0,514	0,356	149	0,273	0,177
0,20	1.240	0,627	0,442	439	0,576	0,396	1.679	0,614	0,430	76	0,369	0,249
0,25	1.020	0,714	0,508	351	0,664	0,461	1.372	0,701	0,496	48	0,454	0,316
0,30	872	0,789	0,565	299	0,733	0,513	1.171	0,775	0,552	39	0,495	0,349
0,35	755	0,861	0,620	229	0,858	0,606	985	0,860	0,617	33	0,529	0,377
0,40	653	0,937	0,679	181	0,987	0,701	834	0,948	0,683	26	0,568	0,410
0,45	552	1,032	0,750	153	1,089	0,776	705	1,044	0,756	22	0,596	0,433
0,50	471	1,128	0,822	129	1,203	0,858	600	1,144	0,830	16	0,640	0,469

Table 14.26: Measured, Indicated and Inferred resources Nueva

Recursos Nueva [OXI Zone - Ordinary Kriging]												
	Measured Resources			Indicated Resources			Measured + Indicated Resources			Inferred Resources		
Cutoff	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol
0,10	234	0,409	0,279	504	0,285	0,185	739	0,325	0,215	617	0,363	0,246
0,15	202	0,455	0,312	423	0,317	0,208	625	0,361	0,241	571	0,383	0,260
0,20	178	0,491	0,340	319	0,362	0,241	497	0,408	0,277	411	0,463	0,321
0,25	148	0,546	0,382	251	0,399	0,270	398	0,454	0,312	358	0,499	0,349
0,30	125	0,596	0,421	162	0,469	0,325	286	0,524	0,367	296	0,546	0,386
0,35	107	0,642	0,457	115	0,527	0,372	222	0,583	0,413	270	0,568	0,403
0,40	91	0,688	0,493	69	0,628	0,453	160	0,662	0,476	230	0,600	0,429
0,45	79	0,730	0,525	53	0,693	0,506	131	0,715	0,517	173	0,659	0,476
0,50	69	0,768	0,553	44	0,736	0,539	113	0,755	0,548	114	0,754	0,550

Table 14.27: Measured, Indicated and Inferred resources Berta II

Recursos Berta II [OXI Zone - Ordinary Kriging]

	Measured Resources			Indicated Resources			Measured + Indicated Resources			Inferred Resources		
Cutoff	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol
0,10	1.188	0,326	0,218	421	0,238	0,151	1.609	0,303	0,200	447	0,149	0,086
0,15	943	0,379	0,256	294	0,287	0,186	1.238	0,357	0,240	219	0,173	0,101
0,20	776	0,423	0,290	184	0,354	0,236	960	0,410	0,279	23	0,236	0,144
0,25	629	0,469	0,326	119	0,428	0,292	747	0,463	0,320	8	0,272	0,170
0,30	513	0,514	0,361	78	0,507	0,355	591	0,513	0,360	0	0,303	0,192
0,35	413	0,560	0,398	59	0,570	0,404	472	0,561	0,399			
0,40	332	0,606	0,435	47	0,619	0,443	379	0,607	0,436			
0,45	271	0,646	0,467	37	0,670	0,484	309	0,649	0,469			
0,50	215	0,692	0,503	31	0,709	0,514	246	0,694	0,504			

Table 14.28: Measured, Indicated and Inferred resources Chico

Recursos Chico [OXI Zone - Ordinary Kriging]

	Measured Resources			Indicated Resources			Measured + Indicated Resources			Inferred Resources		
Cutoff	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol	Kton	% Cu Tot	% Cu Sol
0,10	945	0,296	0,195	1.110	0,203	0,124	2.055	0,246	0,156	527	0,200	0,122
0,15	738	0,344	0,230	759	0,240	0,149	1.497	0,291	0,189	292	0,260	0,162
0,20	538	0,407	0,277	474	0,279	0,177	1.012	0,347	0,230	234	0,282	0,178
0,25	410	0,465	0,322	251	0,329	0,214	661	0,413	0,281	168	0,304	0,194
0,30	320	0,519	0,365	130	0,381	0,255	449	0,479	0,333	60	0,352	0,232
0,35	265	0,560	0,397	59	0,453	0,314	324	0,541	0,382	15	0,459	0,318
0,40	214	0,604	0,433	33	0,515	0,366	247	0,592	0,424	11	0,495	0,348
0,45	161	0,663	0,480	24	0,552	0,397	185	0,649	0,469	5	0,598	0,434
0,50	115	0,738	0,538	16	0,591	0,430	131	0,720	0,525	3	0,648	0,474

14.10.2 RESOURCE ESTIMATION VALIDATION

The results were validated by comparing the blocks with the composites, using the two following methods:

1) Statistical validation
2) Graphical validation

14.10.2.1 STATISTICAL VALIDATION

Table 14.29 presents a summary of the basic statistics of the composites versus the blocks, for each grade element:

Table 14.29: Statistical Validation %CuT and %CuS

%CuS	Composites	Blocks
# of Composites / Blocks	2.226	2.226
Minimum	0,010	0,015
Maximum	2,620	1,899
Mean	0,304	0,302
Standard Deviation	0,311	0,278
Variance	0,097	0,077
Coeff. of Var.	1,022	0,921

%CuS	Composites	Blocks
# of Composites / Blocks	1.882	1.882
Minimum	0,010	0,010
Maximum	1,756	1,392
Mean	0,207	0,211
Standard Deviation	0,221	0,195
Variance	0,049	0,038
Coeff. of Var.	1,072	0,925

From these results, it is concluded that there is an acceptable global bias in the dataset, and that the differences between the mean composite and block grades are due to the high grade treatment in the interpolation (block grades were underestimated)

14.10.2.2 GRAPHICAL VALIDATION

In order to visually examine their trends, blocks and samples were displayed in the same plot. Some plans and sections were included in this analysis as shown in Figure 14.20.

Figure 14.21: Berta Sur and Berta Central %CuT – Section +50

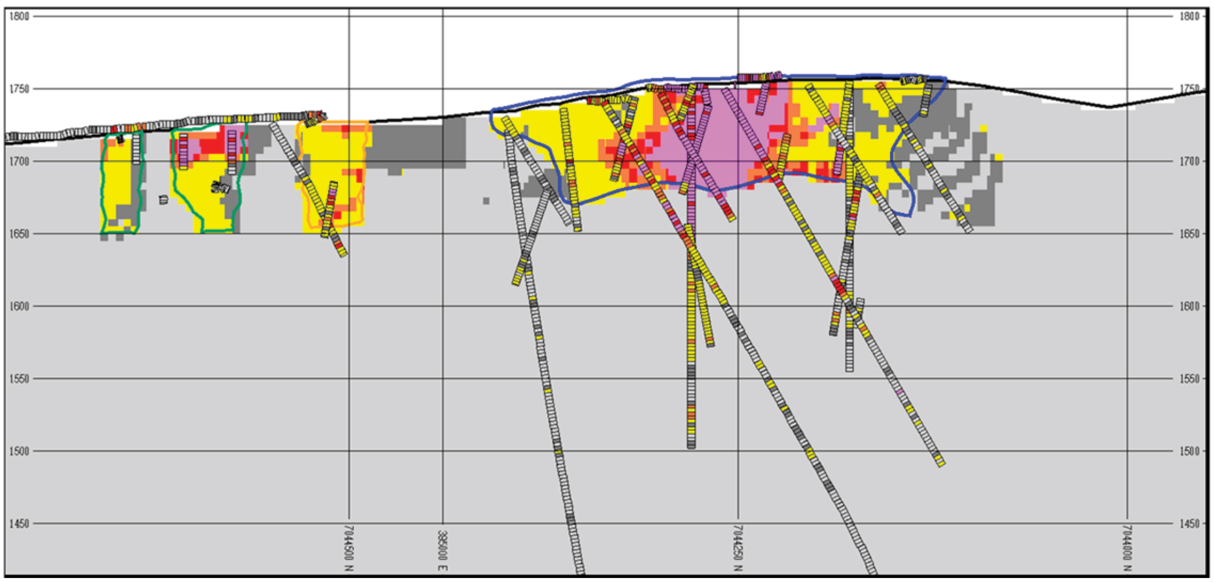


Figure 14.22: Berta Sur and Berta Central %CuT – Section +350

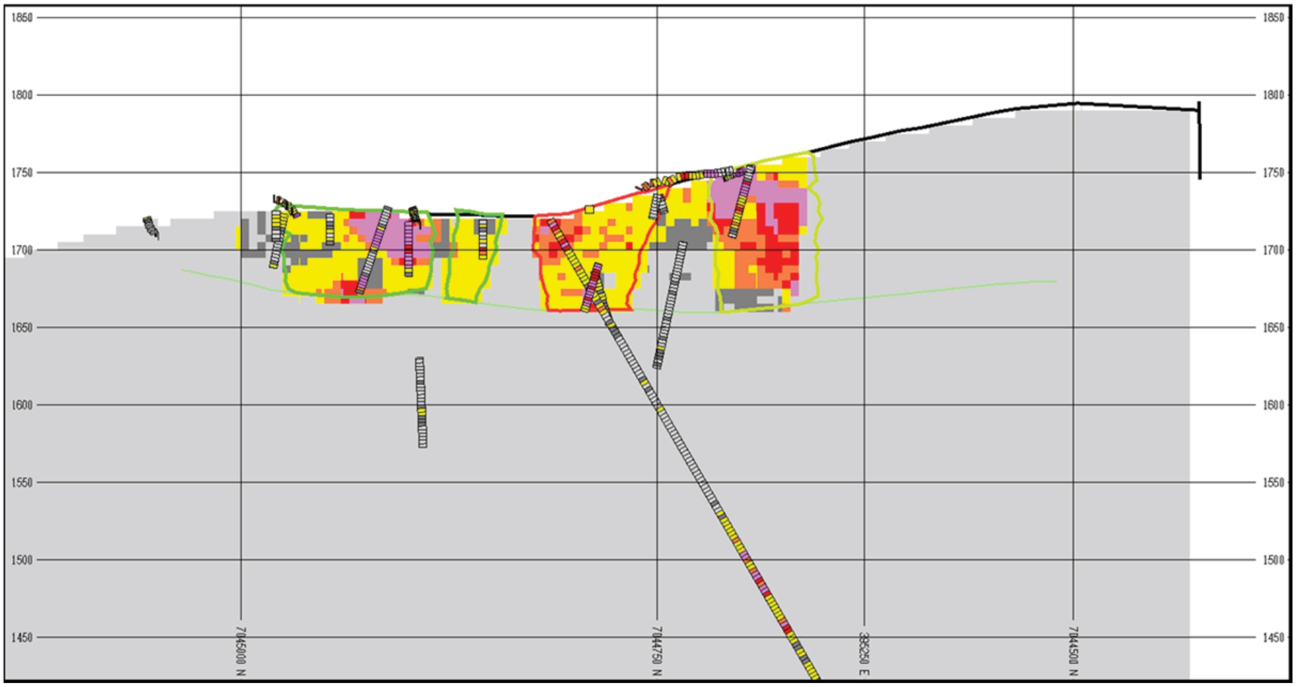


Figure 14.23: Berta Sur and Berta Central %CuT – Plan 1,730m elevation

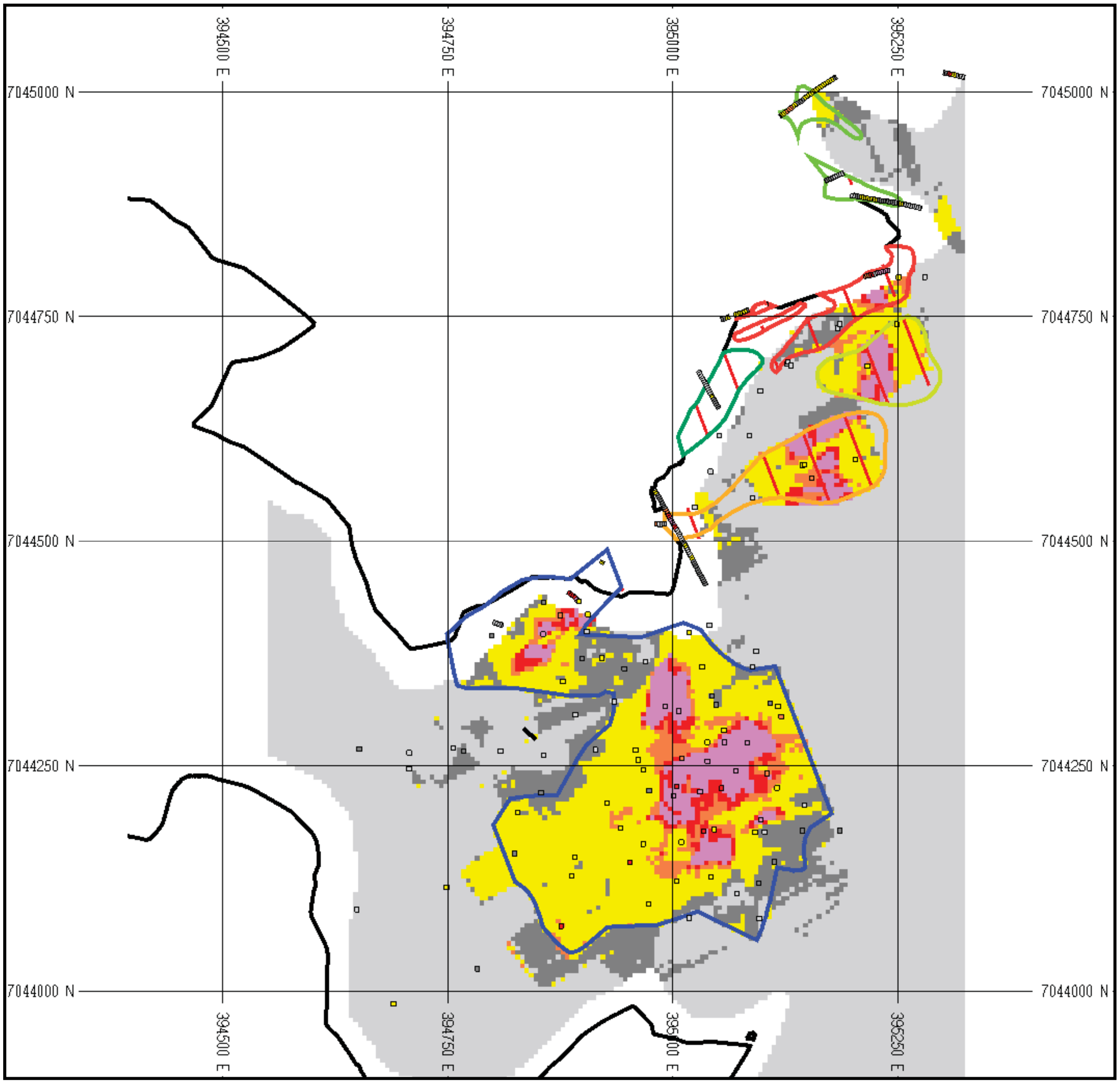


Table 14.30 below, shows the resource estimate including the Berta Sur deposit, and the Berta Central deposits which comprise 5 individual but adjacent deposits which have been subject to prior small scale open pit mining. The resource estimate was completed at a variety of total copper ("%CuT") grades.

Table 14.30: Mineral Resources

Berta Project Resource Estimate														
Zone	Cutoff	Measured			Indicated			Measured & Indicated			Inferred			
		kt	% CuT	% CuS	kt	% CuT	% CuS	kt	% CuT	% CuS	kt	% CuT	% CuS	
Berta Sur & Central	0.10	16,498	0.34	0.23	8,653	0.23	0.14	25,150	0.30	0.20	4,845	0.24	0.15	
	0.15	13,275	0.39	0.27	5,780	0.27	0.18	19,055	0.36	0.24	3,249	0.30	0.20	
	0.20	10,487	0.45	0.31	3,336	0.35	0.23	13,822	0.43	0.29	2,039	0.38	0.25	
	0.25	8,355	0.51	0.36	1,961	0.44	0.30	10,316	0.50	0.35	1,402	0.45	0.31	
	0.30	6,791	0.56	0.40	1,289	0.52	0.36	8,080	0.56	0.39	932	0.53	0.37	
Berta Sur	0.10	10,972	0.32	0.21	4,423	0.18	0.11	15,394	0.28	0.18	2,105	0.18	0.11	
	0.15	8,853	0.37	0.25	2,800	0.21	0.13	11,653	0.33	0.22	1,296	0.22	0.13	
	0.20	6,892	0.42	0.29	1,332	0.26	0.16	8,225	0.39	0.27	720	0.26	0.16	
	0.25	5,385	0.47	0.33	561	0.31	0.20	5,946	0.46	0.32	343	0.29	0.18	
	0.30	4,288	0.53	0.37	261	0.36	0.24	4,549	0.52	0.36	127	0.33	0.21	
Berta Central	0.10	5,526	0.38	0.26	4,230	0.27	0.17	9,756	0.33	0.22	2,740	0.29	0.19	
	0.15	4,422	0.45	0.31	2,980	0.33	0.22	7,402	0.40	0.27	1,953	0.35	0.24	
	0.20	3,594	0.51	0.36	2,003	0.41	0.27	5,598	0.47	0.33	1,318	0.44	0.30	
	0.25	2,969	0.57	0.40	1,401	0.49	0.34	4,370	0.55	0.38	1,059	0.50	0.34	
	0.30	2,503	0.63	0.45	1,028	0.56	0.39	3,531	0.61	0.43	805	0.57	0.40	

Geoinvestment's assessments are preliminary in nature, mineral resources are not mineral reserves and do not have demonstrated economic viability, and there is no assurance the preliminary assessments will be realized. The outcome of this PEA may be materially affected by the closing of the financing, copper pricing, environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. Inferred mineral resources are considered too speculative geologically to have economic considerations applied to them that enable them to be categorized as mineral reserves.

14.11 DETERMINATION OF ECONOMIC ENVELOPE

Geoinvestment considered the basis for determining the reasonable prospects for eventual economic extraction of the Berta Sur and Central resources by completing a series of pit optimizations using the Lersch & Grossmann algorithm based on the following technical and economic parameters; mining cost of $2.09/t, processing Cost of $4.74/t ,SXEW cost of $0.102/lb, G & A cost of $0.045/lb , sales & marketing cost of $0.041/lb, metallurgical recovery of 80% (based on results obtained from the metallurgical test work), inter ramp pit slope of 50 o , and a variety of copper prices.

The pit limit was obtained through the optimization process based on the Lersch & Grossmann algorithm. This kind of optimization is performed for a range of prices which gives value to each of the blocks and determines their capability of being extracted if their economic value allows it.

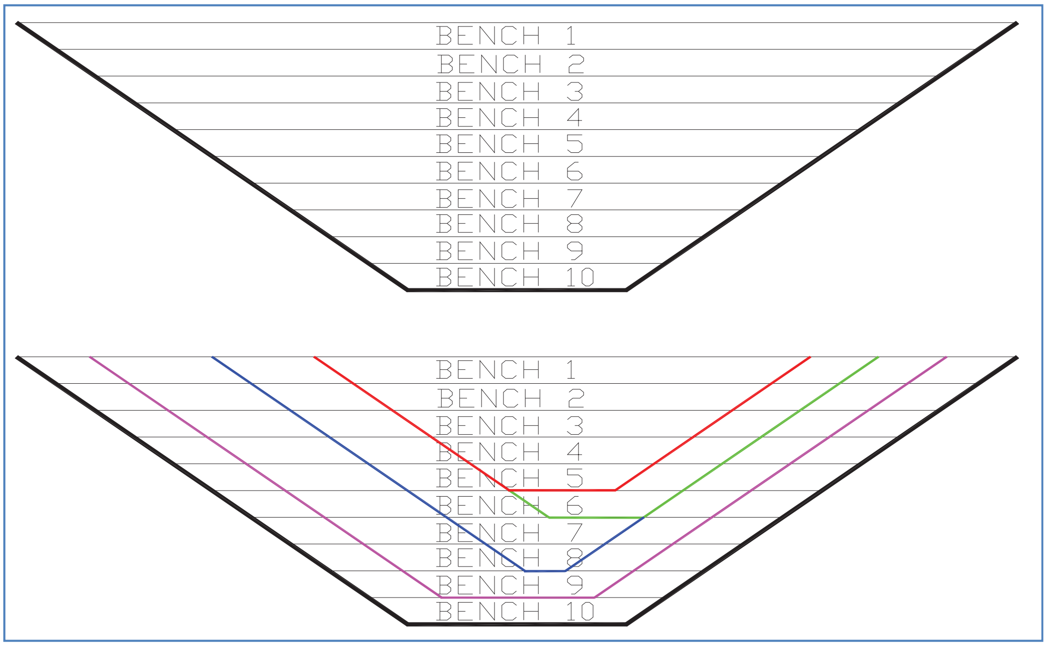
Once each of these pits have been defined, a mine plan to preliminary standards is evaluated in accordance with the imposed restrictions of production. Two kinds of plans are prepared:

“Best Case” Plan: the extraction is made for each expansion (every price is an expansion). This produces the highest NPV, but may be not operational, since frequently the required minimum mining widths are not present.

"Worst Case" Plan: complete banks are extracted, only considering the final pit chosen. This alternative is operative, but it is not optimal in NPV terms, since it could extract much of the waste before exposing the mineralised material.

Since neither of the cases is realistic, it is usual to take an intermediate case for the final pit design and its intermediate phases. These cases are shown in Figure 14.23

Figure 14.24: Worst Case" (up) y "Best Case" (down)



The Berta in Pit resource is unchanged from that published in the Updated Geology and Mineral Resource estimate for the Berta Project with an effective date of September 10th 2013. For a base case using a $3.00/lb copper price, and a 0.1%CuT cut off grade, the optimum pits were determined to contain Measured and Indicated Resources of 17.6 million tons at a grade of 0.37%CuT and an overall stripping ratio of 0.49:1, as detailed in Table 14.31

Table 14.31: In Pit Resource estimate based on $3/lb Cu, 0.1% CuT cutoff

Berta Project In Pit Resource													
Zone	Pit	Measured			Indicated			Measured & Indicated			Waste	Strip
		kt	% CuT	% CuS	kt	% CuT	% CuS	kt	% CuT	% CuS	kt	Ratio
Berta Sur	Berta Sur	8,929	0.35	0.23	1,427	0.19	0.11	10,356	0.33	0.21	2,609	0.25
Berta Central	Trinchera-Salvadora	2,242	0.48	0.30	527	0.47	0.29	2,769	0.48	0.30	2,499	0.90
	Carmen-Gemela	982	0.51	0.36	562	0.38	0.26	1,544	0.47	0.32	1,852	1.20
	Nueva	219	0.43	0.29	295	0.34	0.22	514	0.38	0.25	375	0.73
	Berta II	853	0.37	0.24	150	0.36	0.23	1,003	0.37	0.24	572	0.57
	Chico	900	0.30	0.18	518	0.25	0.14	1,418	0.29	0.17	762	0.54
Berta Sur & Central	**Total**	**14,125**	**0.38**	**0.25**	**3,479**	**0.29**	**0.18**	**17,604**	**0.37**	**0.23**	**8,669**	**0.49**

Readers are advised that more detailed engineering studies have not been completed for the Berta project and so the normal progression from PEA to Preliminary Feasibility Study to Feasibility Study has not been followed in respect of making a production decision. Therefore, investors are cautioned that no mineral reserves have been declared and the level of confidence in the resources, metallurgy, engineering and cost estimation is not at a level normally associated with a project reaching a production decision.

14.12 MINERAL RESOURCE MODEL - PRELIMINARY ANALISYS

The cost structure that was applied to the resources is shown Table 14.32 and metallurgical restrictions were applied, based on a project operating plan, such that oxide material grading above 0.3 %CuT were assumed to be crushed and heap leached while material grading between 0.1 %CuT and 0.3 %CuT would be sent directly to the ROM pile.

Production Capacities

The following production capacities were defined:

- Mine capacity: 7,500 tpd
- Crushing capacity: 1,000,000 tpy
- Plant output capacity: 5,000 tpy cathode.

Table 14.32: Design Criteria and Mine Planning

Economic Parameters	HL to Crusher	ROM
Mining Cost ($/ton)	2.32	2.32
Processing Cost ($/ton)	6.99	1.63
SX-EW Cost ($/lb)	0.33	0.33
G&A ($/lb)	0.09	0.09
Selling ($/lb)	0.15	0.15
Recovery	78,0%	45,0%
Selling Price	$3,0	$3,0

In applying to the model the geotechnical constraints and parameters, a final pit design was obtained that is shown Table 14.33

Table 14.33: Pit Optimization by Sector

Sector	HL Material	CuT%	CuS%	ROM	CuT%	CuS%	Waste	Total
Berta Sur	4.178.240	0,529	0,375	4.175.360	0,203	0,122	1.448.139	9.801.739
Trinchera-S	1.130.880	0,786	0,560	1.096.000	0,186	0,111	2.663.429	4.890.309
Carmen-G	786.240	0,588	0,422	314.880	0,196	0,117	1.931.798	3.032.918
Nueva	223.360	0,567	0,401	205.440	0,209	0,126	271.977	700.777
Berta II	509.760	0,522	0,367	308.160	0,204	0,123	434.526	1.252.446
Chico	395.200	0,492	0,343	533.440	0,196	0,117	434.770	1.363.410
Total	**7.223.680**	**0,574**	**0,407**	**6.633.280**	**0,200**	**0,120**	**7.184.640**	**21.041.600**

15.0 MINERAL RESERVES ESTIMATE

No mineral reserves have been declared for the Berta project.

16.0 MINING METHODS

Mining at Berta would be carried out by open pit methods using contract miners.

16.1 MINE PLAN DESIGN

The Berta development strategy consists of deferring capital in an initial Phase 1 comprising trucking high grade material from the mine to the Nora plant at the rate of 35 ktpm, so as to produce 250 tpm cathode for a period of at least 6 months, and the Phase 2 development of pads and crusher at the mine site and delivery of PLS from Berta to Nora for the remainder of the mine life.

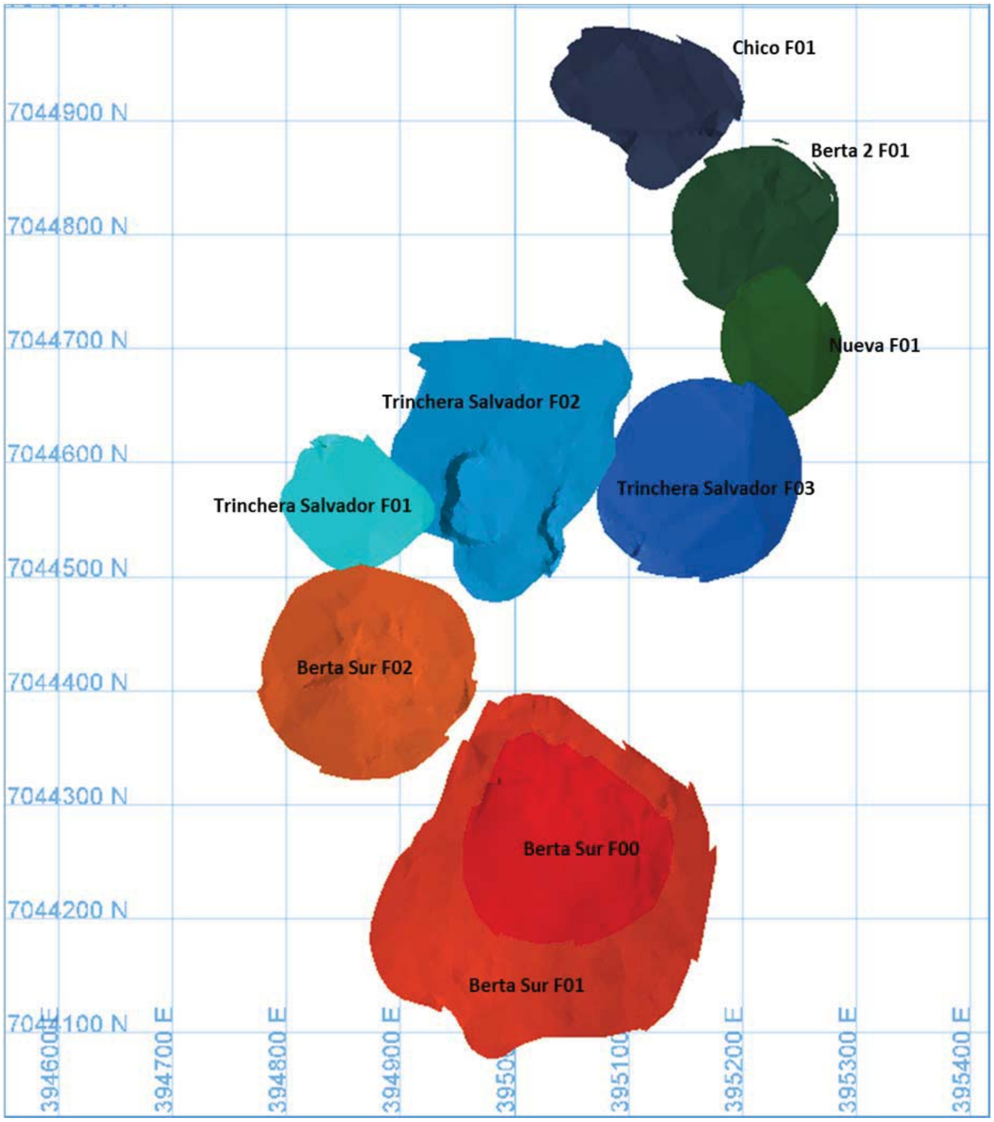
Table 16.1 shows the economic parameters that were developed for this staged strategy, with the Nora column showing the Phase 1 parameters and the Berta column, Phase 2.

Table 16.1: Economic Parameters

Variable	BERTA	NORA	ROM
Mining Cost (USD/ton)	2.32	2.32	2.32
Hauling (USD/ton)	0.00	0.00	0.00
Processing Cost (USD/ton)	7.91	12.29	1.82
SX-EW Cost (USD/lb)	0.250	0.250	0.250
G&A (USD/lb)	0.090	0.090	0.090
Selling Cost (USD/lb)	0.050	0.050	0.050
Recovery	78.0%	78.0%	45.0%
Selling Price	$3.00	$3.00	$3.00

In accordance with these parameters, a phase exclusively for feeding the Nora plant, named Berta Sur Phase 00 was designed as shown on Figure 16.1, which also shows all of the mine development phases.

Figure 16.1: Berta Mine Phase Designs



BS: Berta Sur

TS: Trinchera – Salvadora

NV: Nueva

B2: Berta Dos

CH: Chico

The input data for the mine plan was the tonnage per bench of each one of the phases. An Excel spreadsheet was used, in which percentages of the tonnage of each bench were discounted while, distributing the waste tomineral materialratio in the same proportion to the whole bench.

The time periods used took into account any pre-stripping period and plant filling start up phases.

- For the pre-stripping, the smallest unit of time was a month which will allow for definition of the ramp-up of loading and hauling equipment.
- For the first year of production, once any pre-stripping period is finished, the smallest unit of time is a month. This will demonstrate the build-up of exposed material available to feed the crusher, for which ramp-up period of three months has been considered.

The main controls in the development of the mining plan can be summarized as follows:

- Extraction sequence: the extraction sequence is given by the sequence that comes firstly from the optimization process, subsequently including access ramps and operational considerations with respect to the minimum mining widths between each expansion.
- Movement and production capacity: the following parameters to prepare the mine plan were defined the capacity Production phase 1 and phase 2 shown on Table 16.2: Production Capacity and Phase 1 and Phase 2

Table 16.2: Production Capacity and Phase 1 and Phase 2

Production Capacity	Phase 1	Phase 2
Mine Production	7.5KTPD	7.5KTPD (non-limiting)
Crushing Capacity	40kTPM	83KTPM / 1MTPA (non-limiting)
Cathodes	250 TPM	400TPM / 5,000TPA (main limiting)

The mine production plan was developed to smooth the equipment requirements and utilizations during prestripping andfirst year of production, taking into consideration the distances between the pits as this affects the sequencing.

- Intermediate Stockpiles: three intermediate stockpiles are planned: one for medium grades material (MG), other for low grade material (LW) and the third for ROM, which aims to maintain in inventory for phase 1 and also in case of any unfavourable operational conditions.
- Materials type and locations: material above 0.65%CuT will be sent for crushing at Nora, material between 0.3%CuT and 0.65%CuT will be sent to HG or MG stockpiles depending on amount of material exposed and available capacity. ROM material, above 0.1%CuT will be also stockpiled.

Prestripping Plan

There was no need to develop a pre-stripping plan because the mineralized material is exposed at surface.

Production Plan

Using a variable cut off grade in Year 1 of between 0.60 and 0.70%CuT, it is feasible to maintain a constant feed to the Nora crusher for a period of 11 months, thus postponing part of the initial capital investment until Year 2 of operations.

The initial investment includes the preparation of areas for stockpiling, the first to contain 500kt @0.60%CuT and the second to contain 700kt @ 0.40%CuT, material which will be mined during the first 12 months of operation

An area for the ROM which will contain at least 600kt during the first year should be prepared.

The production plan was made on annual basis and includes the recovery of a small amount of copper from reprocessing spent ore stockpiles or “ripios”.Table 16.3 shown the summary of the annual production plan for the life of the mine (LOM). Figure 16.2 and 16.3 shows graphs of the production plans.

Table 16.3: Extraction Plan and Production by year

Production Profile		Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Total
Heap Leach at Nora	Tonnes	399,258	-	-	-	-	-	-	-	**399,258**
	CuT%	0.83	-	-	-	-	-	-	-	**0.83**
	CuS%	0.61	-	-	-	-	-	-	-	**0.61**
	Rec%	80.97	-	-	-	-	-	-	-	**81,000**
	Cu Cathode, t	2,673	-	-	-	-	-	-	-	**2,673**
Ripios Line	Cu Cathodes, t	315,000								**315,000**
Heap Leach at Berta	Tonnes	84,932	1,002,740	1,000,000	1,000,000	1,000,000	846,925	1,000,000	828,737	**6,763,334**
	CuT%	0.55	0.51	0.55	0.50	0.51	0.79	0.61	0.48	**0.56**
	CuS%	0.39	0.36	0.39	0.35	0.36	0.56	0.43	0.34	**0.40**
	Rec%	0.79	0.78	0.78	0.77	0.77	0.78	0.79	0.77	**0.78**
	Cu Cathode, t	366,000	4,025	4,271	3,838	3,945	5,241	4,783	3,074	**29,544**
Berta ROM	Tonnes	109,353	1,537,653	1,163,006	975,679	598,340	490,499	470,580	603,613	**5,948,725**
	CuT%	0.18	0.20	0.21	0.19	0.19	0.19	0.21	0.20	**0.20**
	CuS%	0.11	0.12	0.13	0.11	0.11	0.11	0.13	0.12	**0.12**
	Rec%	45	45	45	45	45	45	45	45	**45**
	Cu Cathode, t	90,000	1,387	1,101	812,000	501,000	408,000	440,000	541,000	**5,280**
Total Cu	Cu Cathode, t	3,444	5,412	5,372	4,650	4,446	5,650	5,223	3,615	**37,812**
Stockpiled Material										
Berta ROM	Tonnes	499,882	499,882	499,882	499,882	499,882	499,882	499,882	499,882	**3999,056**
	Cu Cathode, t	490,000	490,000	490,000	490,000	490,000	490,000	490,000	490,000	**3920,000**
Berta Leach	Tonnes	1,090,174	732,799	957,612	650,361	254,530	-	115,396	-	**2710,698**
	Cu Cathode, t	4,036	2,281	2,922	1,896	742,000	-	330,000	-	**1083,135**

Readers are advised that more detailed engineering studies have not been completed for the Berta project and so the normal progression from PEA to Preliminary Feasibility Study to Feasibility Study has not been followed in respect of making a production decision. Therefore, investors are cautioned that no mineral reserves have been declared and the level of confidence in the resources, metallurgy, engineering and cost estimation is not at a level normally associated with a project reaching a production decision.

Figure 16.2: Extraction Plan LOM

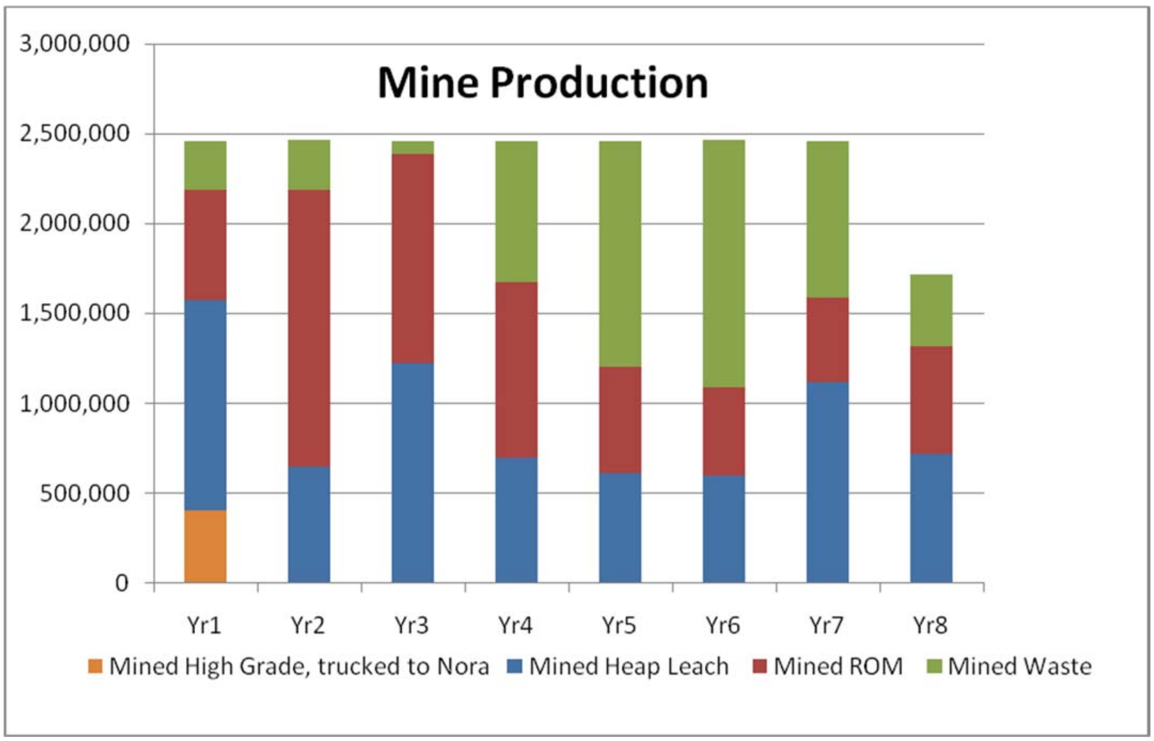
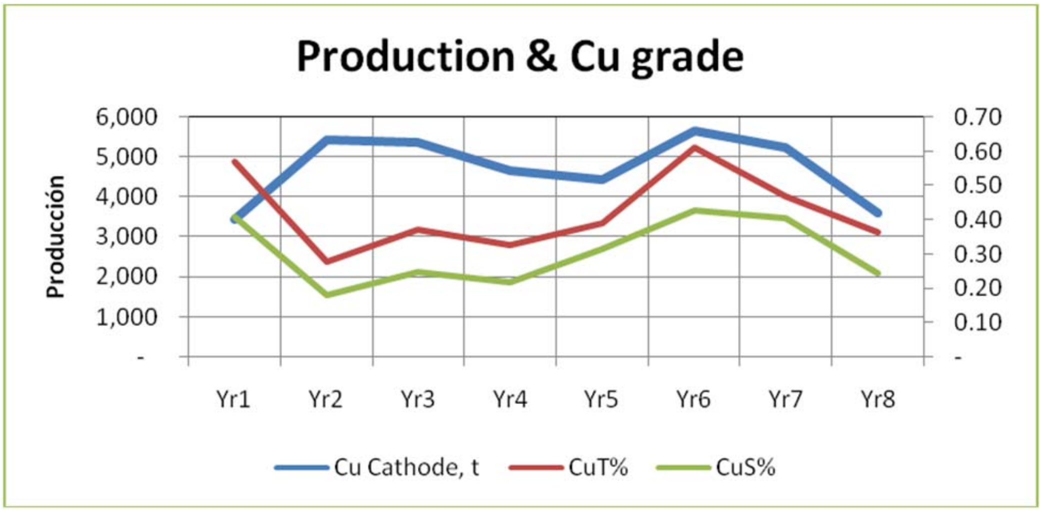


Figure 16.3: Production Plan LOM – Cathodes and CuT%



16.2 MINE PLAN DESIGN

16.2.1 MINE PLAN BACKGROUND

The input data for the mine plan was the tonnage per bench of each one of the phases. An Excel spreadsheet was used, in which percentages of the tonnage of each bench were discounted while, distributing the waste tomineral materialratio in the same proportion to the whole bench.

Background and Considerations To be updated

The time periods used took into account any pre-stripping period and plant filling start up phases.

- For the pre-stripping, the smallest unit of time was a month which will allow for definition of the ramp-up of loading and hauling equipment.
- For the first year of production, once any pre-stripping period is finished, the smallest unit of time is a month. This will demonstrate the build-up of exposed material available to feed the crusher, for which ramp-up period of three months has been considered.

The main controls in the development of the mining plan can be summarized as follows:

- Extraction sequence: the extraction sequence is given by the sequence that comes firstly from the optimization process, subsequently including access ramps and operational considerations with respect to the minimum mining widths between each expansion.
- Movement and production capacity: the following parameters to prepare the mine plan were defined the capacity Production phase 1 and phase 2 shown on Table 16.4

Table 16.4: Production Capacity and Phase 1 and Phase 2

Production Capacity	Phase 1	Phase 2
Mine Production	7.5KTPD	7.5KTPD (non-limiting)
Crushing Capacity	40kTPM	83KTPM / 1MTPA (non-limiting)
Cathodes	250 TPM	400TPM / 5,000TPA (main limiting)

The mine production plan was developed to smooth the equipment requirements and utilizations during prestripping andmineral materialstacking for first year of production, taking into consideration the distances between the pits as this affects the sequencing.

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- Materials type and locations: minerals over cut-off grades of 0.65%CuT will be sent to the crushing at Nora plant, minerals over cut-off grades of 0.3%CuT will be sent to HG or MG stockpiles depending on amount of material exposed and available capacity. ROM material, cut-off 0.1%CuT will be also stockpiled.

Prestripping Plan

There was no need to develop a pre-stripping plan because the mineralized material is exposed at surface.

Production Plan

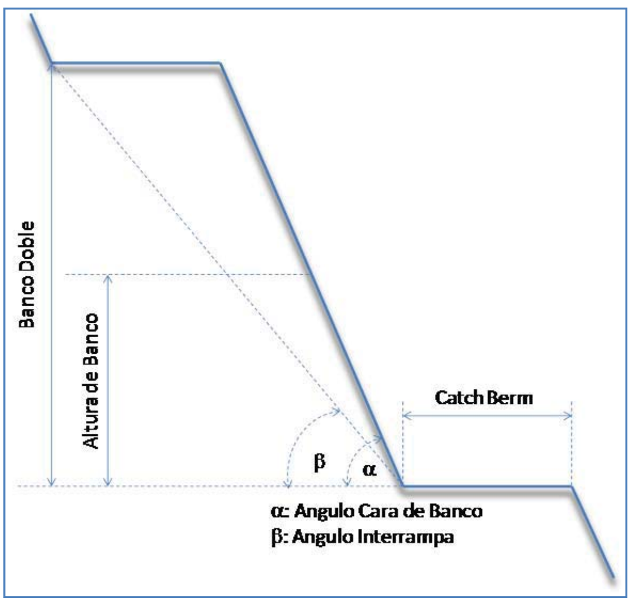
The production plan was made on annual basis. Table 16.3 shows the summary of the annual production plan for the life of the mine (LOM). Figure 16.2 and 16.3 shows graphs of the production plans.

16.2.2 GEOTECHNICAL STUDY

The conclusions of a geotechnical report completed for the Berta Sur pits by Geoinvestment in November 2013, incorporates the following design criteria Figure 16.4

- Bench Height: 5m (double o quadruples)
- Bench Face Angle: 75°
- Inter ramp Angle: 45° - 50°
- Catch Berm: 5.7m

Figure 16.4: Open pit Geometric Parameters



The main conclusions and recommendations of the geotechnical report are:

- The rock mass of the Phase I and Phase II Berta Sur pits, according to the core logging of the three drill-holes that are located in both pits, calculated by MRMR presents rocks that have regular to very good geotechnical quality.
- The geotechnical units and qualities defined by logging of the drill-holes corroborate the units observed in the field.

- The overall slope angles of 50º for the pit walls are within the accepted ranges for MRMR geotechnical qualities for the rock in Berta Sur Pits.
- Based on the stability analysis results, slopes with bench face angles of 75º and overall angle of 50º are stable for both Regions of design.
- Security Factors are all greater than FS>1.0, both for static and pseudo-static analyses.
- It is recommended to carry out sampling for laboratory tests to further characterize the rocky massif and component structures resistance.
- It is necessary to complete 3 or 4 drill holes with the objective of further defining the geotechnical qualities of the rocks, including sampling and laboratory analysis of rock mass resistance and more detailed structural interpretation
- It is recommended to maintain a permanent geological, geotechnical and structural pits control of the focusing on the Berta Sur structures, via mapping. This will allow for improved operational performance during the mine's life.
- During the operation drilling and blasting (pre-cut and load density for the different types of rocks) must be strictly controlled, because this work is fundamental to conserve the slopes of Berta Sur Pits

Final Pit Design

The final pit design was made using the following geometric parameters shown in Table 16.5

Table 16.5: Pit Geometric Parameters

Sector	Catchberm	Rampa	Banco Op.	Banco Final
Berta Sur	5.7	10	5	10
Carmen / Gemela	5.7	7	5	20
Trinchera / Salvador	5.7	7	5	20
Nueva / Berta II / Chico	5.7	7	5	20

The assumed parameters are justified through experience of other operations with similar characteristics.

16.3 HAULAGE DISTANCES

16.3.1 ROADS DESIGNS AND DUMPS

The roads were designed assuming accessing each of the pits with a maximum gradient of 10%.

With respect to the dumps, the existing drawings were utilised.

The designed roads comply with the maximum gradient and necessary width for safe traffic of the equipment.

The main roads, accesses and dumps are shown in Figure 16.5 and 16.6

Figure 16.5: Access to Dumps (North to the Top)

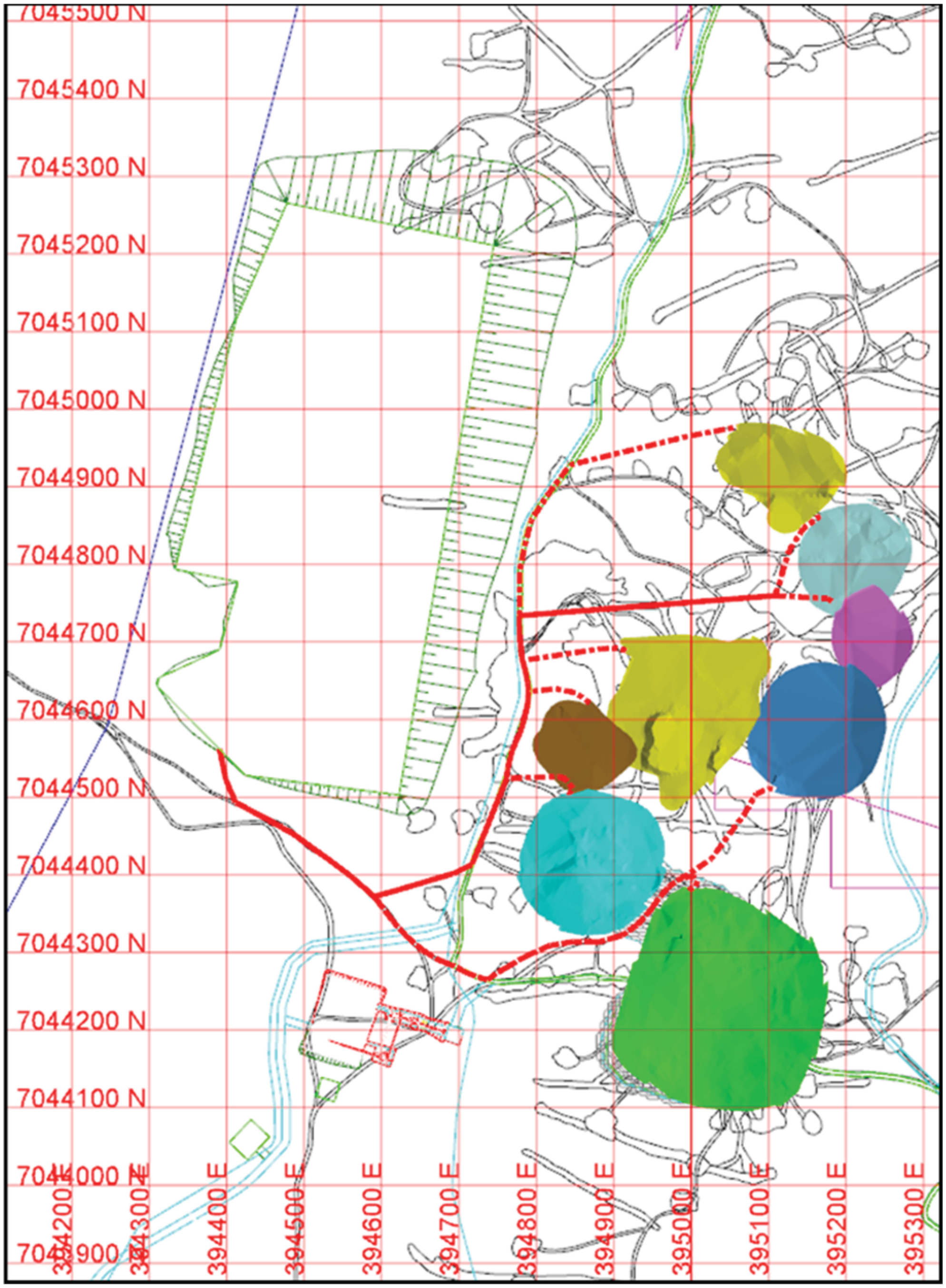


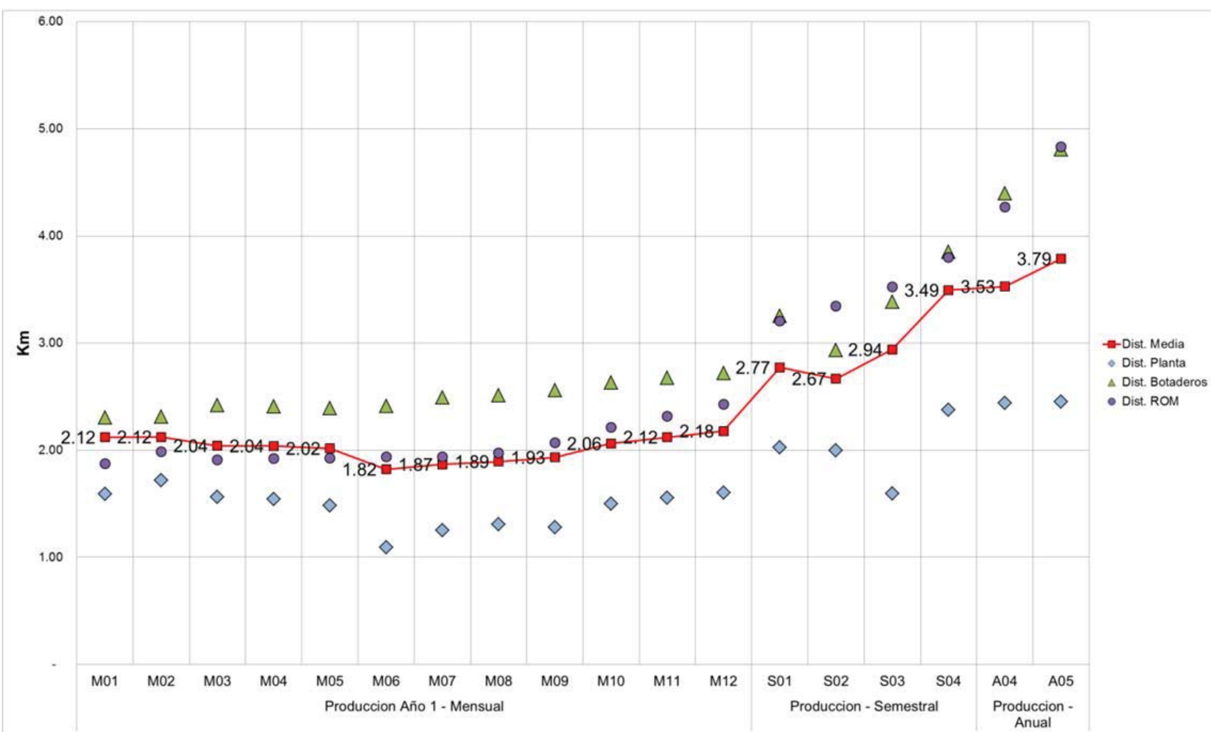
Figure 16.6: Access to Crushing Plant (North to the Top)

16.3.2 HAULAGE DISTANCES

The haulage distances were calculated considering the following:

- Horizontal distance in bench: distance travelled by the truck from the loading point until exit ramp of the bench.
- Exiting the pit phase: distance travelled by the truck from the entering the off ramp of the bench until the exit from the pit phase.
- Access to main road: distance travelled by the truck from the exit of the pit phase to the road that leads to either crushing plant, stockpile or dump.
- Route by main road: distance travelled by the truck by a route considered as the main road where other roads converge from other benches or phases. The Figure 16.7 shows HaulageDistances by Period

Figure 16.7: Average Haulage Distances by Period



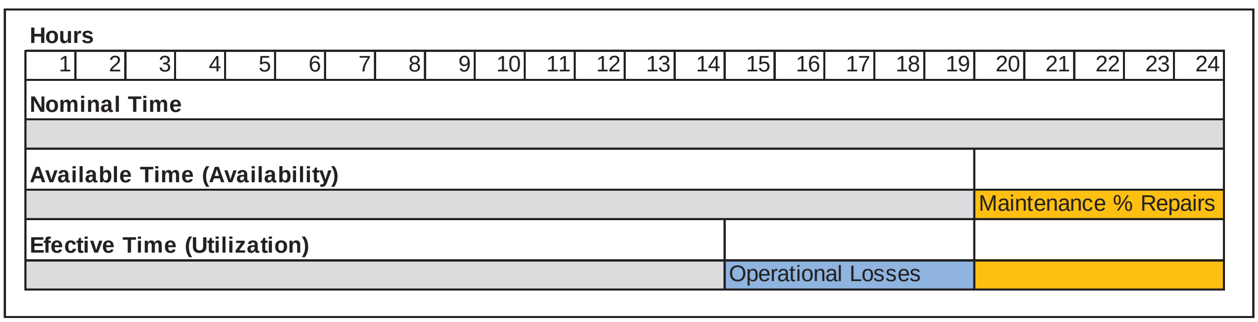
16.3.3 CALCULATION OF EQUIPMENT

This chapter's aim is to indicate the equipment necessary to fulfil the designed mine plan. Capacities and equipment types are shown, with actual models and brands included for reference only.

16.3.4 FLEET CALCULATION

To calculate the fleet size and composition, the traditional methodology based on the effective time to perform each of the activities, was used. In this respect, the availability and utilisation rates were determined and Figure 16.8 shows time distribution for this calculation:

Figure 16.8: Non Operative and Operative Time Distribution



- Availability: the total percentage of hours in a period in which one unit is available for its work. The "unavailable" hours results from scheduled maintenances and repairs due to equipment breakdowns.
- Utilization: the total percentage of hours with respect to the available hours, in which the equipment is effectively working. The "non-utilization" times include shift meetings, meals, shift changes, delays for blasting and other operational delays.

The availability and utilization rates for different periods have been assumed according to the experience of the consultant and values obtained from similar operations. In addition, an efficiency factor for each unit's operation has been included, thus ensuring the maximum fleet needed to implement the plan and comply with the required production plan. These should be re-evaluated once formal quotations and proposals from different suppliers are available.

16.3.5 DRILLING AND BLASTING

The drilling and blasting should be evaluated together, as drill hole diameter and explosive charge factors that ensure correct fragmentation, must be reconciled. Geotechnical and rock mechanics data is used to make a correct estimation of equipment and cost.

According the geotechnical information of the deposit, two material types, mineral material and waste have been defined.

Drilling

Two drilling diameters, 51/2" and 6 1/2", based on bench height and the geotechnical characteristics of similar deposits were assumed. Table 16.6 shows the data that has been assumed to calculate drilling and blasting criteria.

Table 16.6: Operation Parameters for Drilling

DRILLING			
ITEM	UNITS	ORE	WASTE
Diameter	in	5 1/2"	6 1/2"
Diameter	mm	140	165
Density	mt/m3	2,56	2,56
Bench Height	m	5,00	5,00
Burden	m	4,0	5,0
Spacing	m	4,0	5,5
Subdrill	m	0,8	1,0
Hole length	m	5,8	6,0
Stemming	m	1,5	2,0
Hole volume	bcm	80,0	137,5
Specific drilling	m3/m	13,8	22,9
Tonnes/hole	mt	204,8	352,0
Overdrill factor	%	10%	10%
Specific drilling	mt/m	31,8	52,8
Compressive strength	Mpa	60	60
Instantaneous velocity	m/hr	40	40

Atlas Copco ROC L8 Figure 16.9 type equipment has been assumed based on the operational performance of similar operations in the III Region.

Improvements at this stage could be related to the assumed values for the rate of penetration and the incorporation of a differentiation with gravels.

Figure 16.9: Atlas Copco ROC L8 Driller

Table 16.7 shows the drilling productivities calculation and Table 16.8 indicates the units required to accomplish the first year of the plan

Table 16.7: Drilling Equipment Productivities LOM

DRILLING									
Heap Leach + ROM (DRILL 5 1/2)	**un**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**
DIAMETER	mm	140	140	140	140	140	140	140	140
VOLUMETRIC ESPECIFIC DRILLING	m3/m	19	19	19	19	19	19	19	19
DENSITY	t/m3	3	3	3	3	3	3	3	3
ESPECIFICA DRILLING	t/m	50	50	50	50	50	50	50	50
PENETRATION SPEED	m/h	15	15	15	15	15	15	15	15
OPERATIONAL EFICIENCY	%	95%	95%	95%	95%	95%	95%	95%	95%
PENETRATION SPEED (OPERATIONAL)	m/h	14	14	14	14	14	14	14	14
PRODUCTIVITY	t/h	708	708	708	708	708	708	708	708
AVALABILITY	%	90%	90%	90%	90%	90%	90%	90%	90%
UTILIZATION	%	85%	85%	85%	85%	85%	85%	85%	85%
HR/SHIFT	hr	12	12	12	12	12	12	12	12
SHIFT/DAY	ea	2	2	2	2	2	2	2	2
DAYS	d	360	360	360	360	360	360	360	360
PRODUCTION	kt	4.677	4.677	4.677	4.677	4.677	4.677	4.677	4.677
WASTE (DRILL 6 1/2)									
DIAMETER	mm	165	165	165	165	165	165	165	165
VOLUMETRIC ESPECIFIC DRILLING	m3/m	23	23	23	23	23	23	23	23
DENSITY	t/m3	3	3	3	3	3	3	3	3
ESPECIFICA DRILLING	t/m	59	59	59	59	59	59	59	59
PENETRATION SPEED	m/h	15	15	15	15	15	15	15	15
OPERATIONAL EFICIENCY	%	95%	95%	95%	95%	95%	95%	95%	95%
PENETRATION SPEED (OPERATIONAL)	m/h	14	14	14	14	14	14	14	14
PRODUCTIVITY	t/h	836	836	836	836	836	836	836	836
AVALABILITY	%	90%	90%	90%	90%	90%	90%	90%	90%
UTILIZATION	%	85%	85%	85%	85%	85%	85%	85%	85%
HR/SHIFT	hr	12	12	12	12	12	12	12	12
SHIFT/DAY	ea	2	2	2	2	2	2	2	2
DAYS	d	360	360	360	360	360	360	360	360
PRODUCTION	kt	5.526	5.526	5.526	5.526	5.526	5.526	5.526	5.526

Table 16.8: Required Units of Drilling Equipment LOM

Drilling Diesel Driller		Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Heap Leach + ROM (DRILL 5 1/2)									
PRODUTION	t	2.183.599	2.183.017	2.387.818	1.668.429	1.202.508	1.082.895	1.585.977	1.316.954
PRODUCTIVITY	kt	4.677	4.677	4.677	4.677	4.677	4.677	4.677	4.677
Nº UNIT	ea	0,47	0,47	0,51	0,36	0,26	0,23	0,34	0,28
WASTE (DRILL 6 1/2)									
PRODUCCION	ton	270.428	83.320	19.862	235.679	375.455	413.356	260.415	118.582
PRODUCTIVIDAD	kt	5.526	5.526	5.526	5.526	5.526	5.526	5.526	5.526
Nº UNIDADES	ea	0,05	0,02	0,00	0,04	0,07	0,07	0,05	0,02
Nº UNIT (DRILL 5 1/2)	**ea**	**0,47**	**0,47**	**0,51**	**0,36**	**0,26**	**0,23**	**0,34**	**0,28**
Nº UNIT (DRILL 6 1/2)	**ea**	**0,05**	**0,02**	**0,00**	**0,04**	**0,07**	**0,07**	**0,05**	**0,02**

Table 16.9 indicates the calculation of explosives and blasting products required, based on the consultant's experience and on operations with similar characteristics and conditions.

Table 16.9: Operational Parameters for Blasting

BLASTING			
ITEM	UNITS	ORE	WASTE
Diameter	in	5,50	6,50
Diameter	mm	140	165
Diameter of explosive load	in	5,50	6,50
Diameter of explosive load	mm	140	165
Rock Density	mt/m3	2,56	2,56
Explosive density	mt/m3	0,80	0,80
Bench Height	m	5,0	5,0
Burden	m	4,0	5,0
Spacing	m	4,0	5,5
Subdrill	m	0,8	1,0
Hole length	m	5,8	6,0
Stemming	m	1,5	2,0
Length of explosive column	m	4,3	4,0
Energy distribution factor	%	70	60
Explosive/meter	Kg/m	12,3	17,1
Explosive/hole	Kg	53	69
Hole volume	bcm	80	138
Total rock	mt/m3	205	352
Powder Factor	gr/mt	259	196

It has been assumed that the base load will be HEAVY ANFO (HANFO), and for loading a column, ANFO NORMAL. Any holes making water will be loaded with an emulsion of 60% HEAVY ANFO

Table 16.10 shows the consumption of explosives and diesel required for the designed plan.

Table 16.10: Explosives Consumption

BLASTING	Unit	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	TOTAL
Heap Leach Blasting	t	2183599	2183017	2387818	1668429	1202508	1082895	1585977	1316954	13611198
ANFO	60%	339	339	371	259	187	168	246	204	2114
EMULSION	40%	226	226	247	173	124	112	164	136	1409
ACCESORIES	10%	57	56	62	43	31	28	41	34	352
Waste Blasting	t	270428	277732	66208	785597	1251518	1377855	868050	395272	5292660
ANFO	60%	32	33	8	92	147	162	102	46	623
EMULSION	40%	21	22	5	62	98	108	68	31	415
ACCESORIES	10%	5	5	1	15	25	27	17	8	104
TOTAL HL + Waste										
ANFO	t	371	372	379	351	334	330	348	251	2736
EMULSION	t	247	248	252	234	223	220	232	167	1824
ACCESORIES	t	62	62	63	59	56	55	58	42	456

16.3.6 LOADING AND HAULING FLEET

The loading and hauling should be analysed together to determine the compatibility of the units. The aspects to evaluate are:

- Physical Match: the dimension of the equipment must be compatible. The height of the dump trucks must be such that the loading equipment is capable of that height to load them.
- Productivity Match: the number of passes required to load a truck must be taken in account. High productivity is considered a match of 3 to 4 passes. It is planned to use 25ton dump trucks and 3.7yd3 loaders which results in medium productivity, as they require 5 passes to load a truck.

As for drilling and blasting, the GRAVEL type material has an important effect in the productivity calculations. This material has a low-density value, which affects the performance of both the loaders and the trucks. This should be evaluated at a later stage.

Loading Fleet

Front end loaders of 3.7yd3, such as the one shown in Figure 16.10 are considered to be appropriate equipment given the pit characteristics and the levels of required productivity.Table 16.5 and Table 16.6 show the productivities per month and the required units, respectively, for the first year of production, are summarised. The same information for the remaining years is also shown in Table 16.11 and Table 16.12

Figure 16.10: Loader Type CAT 962Kde 3.7 y d3

Table 16.11: Productivities of Loading Fleet LOM

LOADING									
FEL 4 YD^3 - Heap Leach Material	**un**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**
FEL CAPACITY	yd3	3,7	3,7	3,7	3,7	3,7	3,7	3,7	3,7
FEL CAPACITY	m3	2,8	2,8	2,8	2,8	2,8	2,8	2,8	2,8
FILLING FACTOR	%	90%	90%	90%	90%	90%	90%	90%	90%
SPONGY DENSITY	t/m3	1,8	1,8	1,8	1,8	1,8	1,8	1,8	1,8
HUMIDITY	%	2%	2%	2%	2%	2%	2%	2%	2%
WET EFECTIVE LOAD	t	4,6	4,6	4,6	4,6	4,6	4,6	4,6	4,6
DRY EFECTIVE LOAD	t	4,5	4,5	4,5	4,5	4,5	4,5	4,5	4,5
LOAD CICLE (1 PASS + 15seg)	min	0,5	0,5	0,5	0,5	0,5	0,5	0,5	0,5
Nº TIMES	cu	5	5	5	5	5	5	5	5
DRY LOADED TRUCK	t	22	22	22	22	22	22	22	22
WET LOADED TRUCK	t	23	23	23	23	23	23	23	23
TRUCK LOAD TIME	min	2	2	2	2	2	2	2	2
HANDLING TIME	min	1	1	1	1	1	1	1	1
TOTAL LOADING TIME	min	3	3	3	3	3	3	3	3
EFECFTIVE PRODUCTION	t/h	415	415	415	415	415	415	415	415
OPERATIONAL EFICIENCY	%	95%	95%	95%	95%	95%	95%	95%	95%
OPERATIONAL EFICIENCY	t/h	394	394	394	394	394	394	394	394
AVAILABILITY	%	92%	92%	92%	92%	92%	92%	92%	92%
UTILIZATION	%	87%	87%	87%	87%	87%	87%	87%	87%
HR/SHIFT	h	12	12	12	12	12	12	12	12
SHIFT/DAY	cu	2	2	2	2	2	2	2	2
DAYS	d	360	360	360	360	360	360	360	360
PRODUCTION	kt	2.724	2.724	2.724	2.724	2.724	2.724	2.724	2.724
FEL 4 YD^3 - WASTE									
FEL CAPACITY	yd3	3,7	3,7	3,7	3,7	3,7	3,7	3,7	3,7
FEL CAPACITY	m3	2,8	2,8	2,8	2,8	2,8	2,8	2,8	2,8
FILLING FACTOR	%	90%	90%	90%	90%	90%	90%	90%	90%
SPONGY DENSITY	t/m3	1,8	1,8	1,8	1,8	1,8	1,8	1,8	1,8
HUMIDITY	%	2%	2%	2%	2%	2%	2%	2%	2%
WET EFECTIVE LOAD	t	4,6	4,6	4,6	4,6	4,6	4,6	4,6	4,6
DRY EFECTIVE LOAD	t	4,5	4,5	4,5	4,5	4,5	4,5	4,5	4,5
LOAD CICLE (1 PASS + 15seg)	min	0,5	0,5	0,5	0,5	0,5	0,5	0,5	0,5
Nº TIMES	cu	5	5	5	5	5	5	5	5
DRY LOADED TRUCK	t	22	22	22	22	22	22	22	22
WET LOADED TRUCK	t	23	23	23	23	23	23	23	23
TRUCK LOAD TIME	min	2	2	2	2	2	2	2	2
HANDLING TIME	min	1	1	1	1	1	1	1	1
TOTAL LOADING TIME	min	3	3	3	3	3	3	3	3
EFECFTIVE PRODUCTION	t/h	415	415	415	415	415	415	415	415
OPERATIONAL EFICIENCY	%	95%	95%	95%	95%	95%	95%	95%	95%
OPERATIONAL EFICIENCY	t/h	394	394	394	394	394	394	394	394
AVAILABILITY	%	92%	92%	92%	92%	92%	92%	92%	92%
UTILIZATION	%	87%	87%	87%	87%	87%	87%	87%	87%
HR/SHIFT	h	12	12	12	12	12	12	12	12
SHIFT/DAY	cu	2	2	2	2	2	2	2	2
DAYS	d	360	360	360	360	360	360	360	360
PRODUCTION	kt	2.724	2.724	2.724	2.724	2.724	2.724	2.724	2.724

Table 16.12: Required Units of Loading Fleet LOM

LOADING FRONTUP LOADERS		Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
FEL 3,7 YD3 Heap Leach + ROM									
PRODUCTION	t	2.183.599	2.183.017	2.387.818	1.668.429	1.202.508	1.082.895	1.585.977	1.316.954
PRODUCTIVITY	kt	2.724	2.724	2.724	2.724	2.724	2.724	2.724	2.724
PERFORMANCE	tpd	6.066	6.064	6.633	4.635	3.340	3.008	4.405	3.658
N° UNITS	ea	0,80	0,80	0,88	0,61	0,44	0,40	0,58	0,48
	tph	394							
FEL 3,7 YD3 WASTE									
PRODUCTION	t	270.428	83.320	19.862	235.679	375.455	413.356	260.415	118.582
PRODUCTIVITY	kt	2.724	2.724	2.724	2.724	2.724	2.724	2.724	2.724
PERFORMANCE	tpd	751	231	55	655	1.043	1.148	723	329
N° UNITS	ea	0,10	0,03	0,01	0,09	0,14	0,15	0,10	0,04
TOTAL FEL 3,7 YD^3	**ea**	**0,9**	**0,8**	**0,9**	**0,7**	**0,6**	**0,5**	**0,7**	**0,5**

Haulage Fleet

To obtain the required productivity match with the loading equipment, it has been decided to use 25ton dump trucks. These trucks may be used both in mining operations and on highways. Figure 16.11 shows a type of truck with the necessary characteristics for Berta.

Figure 16.11: 25 ton Truck 6x4

Table 16.13 and Table 16.14 show the productivity results and required units, for the remaining years.

Table 16.13: Productivity of Haulage Equipment LOM

HAULING									
TRUCK - FEL 4 YD³ (HL TO CRUSHER)	**un**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**
NOMINAL CAPACITY	t	23	23	23	23	23	23	23	23
HUMIDITY	%	0	0	0	0	0	0	0	0
CAPACITY (DRY)	t	22	22	22	22	22	22	22	22
TRANSLATION TIME	min	4	3	5	6	5	7	7	6
LOAD, DISCHARGE AND HANDLING	min	3	3	3	3	3	3	3	3
CYCLE	min	7	7	9	10	8	10	10	9
EFECTIVE PROCUCTION	t/h	196	200	158	141	171	129	134	146
OPERATIONAL EFICIENCY	%	95%	95%	95%	95%	95%	95%	95%	95%
OPERATIONAL PRODUCTION	t/h	186	190	150	134	163	123	127	139
AVAILABILITY	%	92%	92%	90%	90%	90%	90%	87%	87%
UTILIZATION	%	86%	86%	86%	86%	86%	86%	86%	86%
HR/SHIFT	h	12	12	12	12	12	12	12	12
SHIFT/DAY	cu	2	2	2	2	2	2	2	2
DAYS	d	360	360	360	360	360	360	360	360
PRODUCTION	kt	1.270	1.300	1.001	895	1.087	819	824	897
TRUCK - FEL 4 YD³ (ROM)									
NOMINAL CAPACITY	t	23	23	23	23	23	23	23	23
HUMIDITY	%	0	0	0	0	0	0	0	0
CAPACITY (DRY)	t	22	22	22	22	22	22	22	22
TRANSLATION TIME	min	4	5	7	8	8	9	9	10
LOAD, DISCHARGE AND HANDLING	min	3	3	3	3	3	3	3	3
CYCLE	min	8	8	10	11	11	12	12	13
EFECTIVE PROCUCTION	t/h	177	170	130	118	121	112	109	100
OPERATIONAL EFICIENCY	%	95%	95%	95%	95%	95%	95%	95%	95%
OPERATIONAL PRODUCTION	t/h	168	162	124	112	115	107	103	95
AVAILABILITY	%	92%	92%	90%	90%	90%	90%	87%	87%
UTILIZATION	%	86%	86%	86%	86%	86%	86%	86%	86%
HR/SHIFT	h	12	12	12	12	12	12	12	12
SHIFT/DAY	cu	2	2	2	2	2	2	2	2
DAYS	d	360	360	360	360	360	360	360	360
PRODUCTION	t	1.151	1.104	827	750	769	713	667	614
TRUCK - FEL 4 YD³ (WASTE)	**un**								
NOMINAL CAPACITY	t	23	23	23	23	23	23	23	23
HUMIDITY	%	0	0	0	0	0	0	0	0
CAPACITY (DRY)	t	22	22	22	22	22	22	22	22
TRANSLATION TIME	min	5	5	7	6	8	8	9	10
LOAD, DISCHARGE AND HANDLING	min	3	3	3	3	3	3	3	3
CYCLE	min	8	9	10	10	11	11	12	13
EFECTIVE PROCUCTION	t/h	164	157	135	139	122	121	109	100
OPERATIONAL EFICIENCY	%	95%	95%	95%	95%	95%	95%	95%	95%
OPERATIONAL PRODUCTION	t/h	156	150	128	132	116	115	103	95
AVAILABILITY	%	92%	92%	90%	90%	90%	90%	87%	87%
UTILIZATION	%	86%	86%	86%	86%	86%	86%	86%	86%
HR/SHIFT	h	12	12	12	12	12	12	12	12
SHIFT/DAY	cu	2	2	2	2	2	2	2	2
DAYS	d	360	360	360	360	360	360	360	360
PRODUCTION	kt	1.066	1.023	856	885	778	772	667	615

Table 16.14: Required Units of Hauling Fleet LOM

HAULING 25t TRUCK		Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
MINE TO PROCESS									
PRODUCTION	t	1.574.364	645.365	1.224.812	692.750	604.169	592.396	1.115.396	713.341
PRODUCTIVITY	t	1.270	1.300	1.001	895	1.087	819	824	897
N° UNITS	ea	1,24	0,50	1,22	0,77	0,56	0,72	1,35	0,79
MINE TO ROM									
PRODUCTION	t	609.235	1.537.653	1.163.006	975.679	598.340	490.499	470.580	603.613
PRODUCTIVITY	t	1.151	1.104	827	750	769	713	667	614
N° UNITS	ea	0,53	1,39	1,41	1,30	0,78	0,69	0,71	0,98
WASTE TO DUMP									
PRODUCTION	t	270.428	277.732	66.208	785.597	1.251.518	1.377.855	868.050	395.272
PRODUCTIVITY	t	1.066	1.023	856	885	778	772	667	615
N° UNITS	ea	0,25	0,27	0,08	0,89	1,61	1,79	1,30	0,64
TOTAL TRUCK / FEL 3.7 YD3	**ea**	**2,0**	**2,2**	**2,7**	**3,0**	**2,9**	**3,2**	**3,4**	**2,4**

16.3.7 SUPPORT FLEET EQUIPMENT

The required support equipment has been estimated based on the loading and hauling equipment and the nature of the deposit and its environment.

The support equipment required for Berta is:

- Bulldozer: will be used for the preparation of production faces and maintenance of the dumps. Based on the mine plan, it is estimated that one bulldozer will be required for every two loaders, plus one for each dump in operation.
- Wheel-dozer: will be responsible for maintaining both the loading points and the haulage routes. Given its mobility, a half a unit per loader is required.
- Grader: will be used for road maintenance roads. One unit is needed for every eight trucks based on the transport distances.
- Sprinkler Truck: will be used for road maintenance and dust control. One 20,000lt sprinkler truck is required per every 20,000 TPD moved.

A summary of the mine fleet is shown in Table 16.15 below:

Table 16.15: Required Units of Haulage Equipment

MINE FLEET SUMMARY		Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
DRILLING									
DRILL 5 1/2	h	4.034	4.033	4.411	3.082	2.222	2.001	2.930	2.433
DRILL 6 1/2	h	423	130	31	369	587	646	407	185
TOTAL DRILLING		**4.457**	**4.163**	**4.442**	**3.451**	**2.809**	**2.647**	**3.337**	**2.618**
N° Units		1	1	1	1	1	1	1	1
LOADING									
FEL 4 YD3	h	7.785	7.189	7.638	6.040	5.006	4.746	5.857	4.554
N° Units		1	1	1	1	1	1	1	1
HAULING									
25t Truck	h	17.479	18.669	23.388	25.591	25.424	27.611	29.039	20.918
N° Units		3	3	3	3	4	4	4	4
SUPPORT FLEET EQUIPMENT									
BULLDOZER - 1 Unit	h	3.892	3.595	3.819	3.020	2.503	2.373	2.929	2.277
WHEELDOZER - 1 Unit	h	3.892	3.595	3.819	3.020	2.503	2.373	2.929	2.277
Grader - 1 Unit	h	2.185	2.334	2.924	3.199	3.178	3.451	3.630	2.615
Sprinkler Truck - 1 Unit	h	3.033	3.041	3.033	3.033	3.033	3.041	3.033	2.116

17.0 RECOVERY METHODS

The Berta Project involves the exploitation of an open pit mine and the design of facilities for the production of 5,000 copper tonnes per year of high concentration Pregnant Leaching Solution (PLS), which will be transported to Nora Plant where this solution will be processed in the SXEW Plant, thus producing high purity copper cathodes, 99.99%. Berta mine site facilities will be located 15km west of the village Inca de Oro at an elevation of 1,700m.a.s.l. and the existing Nora Plant is located 42 km north from Berta.

In this Amended updated PEA, the mine plan assumes a first phase using a variable cut-off grade in year 1 of between 0.60% and 0.70%CuT, in order to maintain a constant feed to theexisting Nora crusher for a period of 11 months, thus postponing part of the capital investment until year 2 of operations. A total of 0.4mt at 0.83%Cu will be mined and trucked to the Nora plant while 1.2mt of lower grade heap leach material and 0.6mt of ROM will be stockpiled for processing in year 2. In addition, the Nora plant will reprocess some spent mineral material stockpiles (“Ripios”) from the previous 2009-12 operation at a rate of ~30 tpm of copper cathode during Phase 1 as described in section 17.1.4 of this chapter.

Phase 2, after eleven month, considers all the copper oxide material from the open pits will be treated through a heap leach process with capacity of 1 million tonnes of mineral material per year (including crushing, agglomeration and permanent pads), and the processing of 1.2 million tonnes per year of Run of Mine (ROM) material directly onto dump leach pads.

Figure 17.1, 17.2 and 17.3 shows the general flow diagram of Berta Plant and Nora plant

Figure 17.1: General flow Diagram of Berta Project Phase 2 sacar camiones

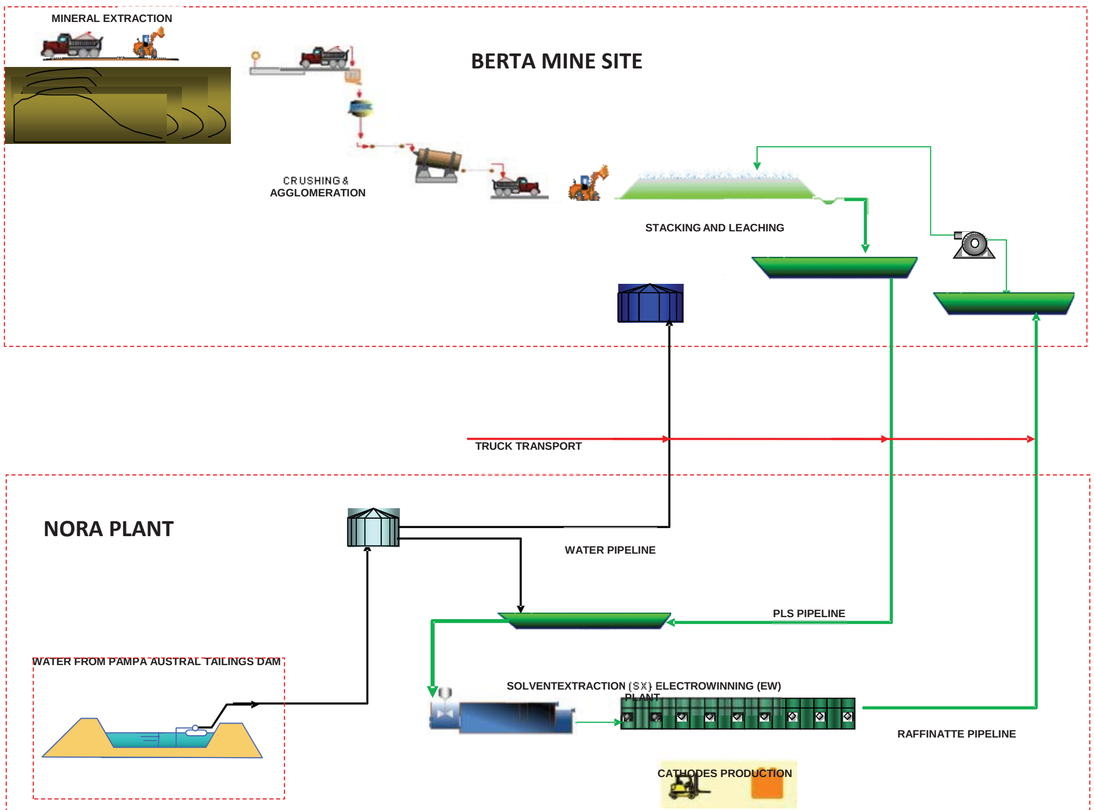


Figure 17.2: Berta mine site layout

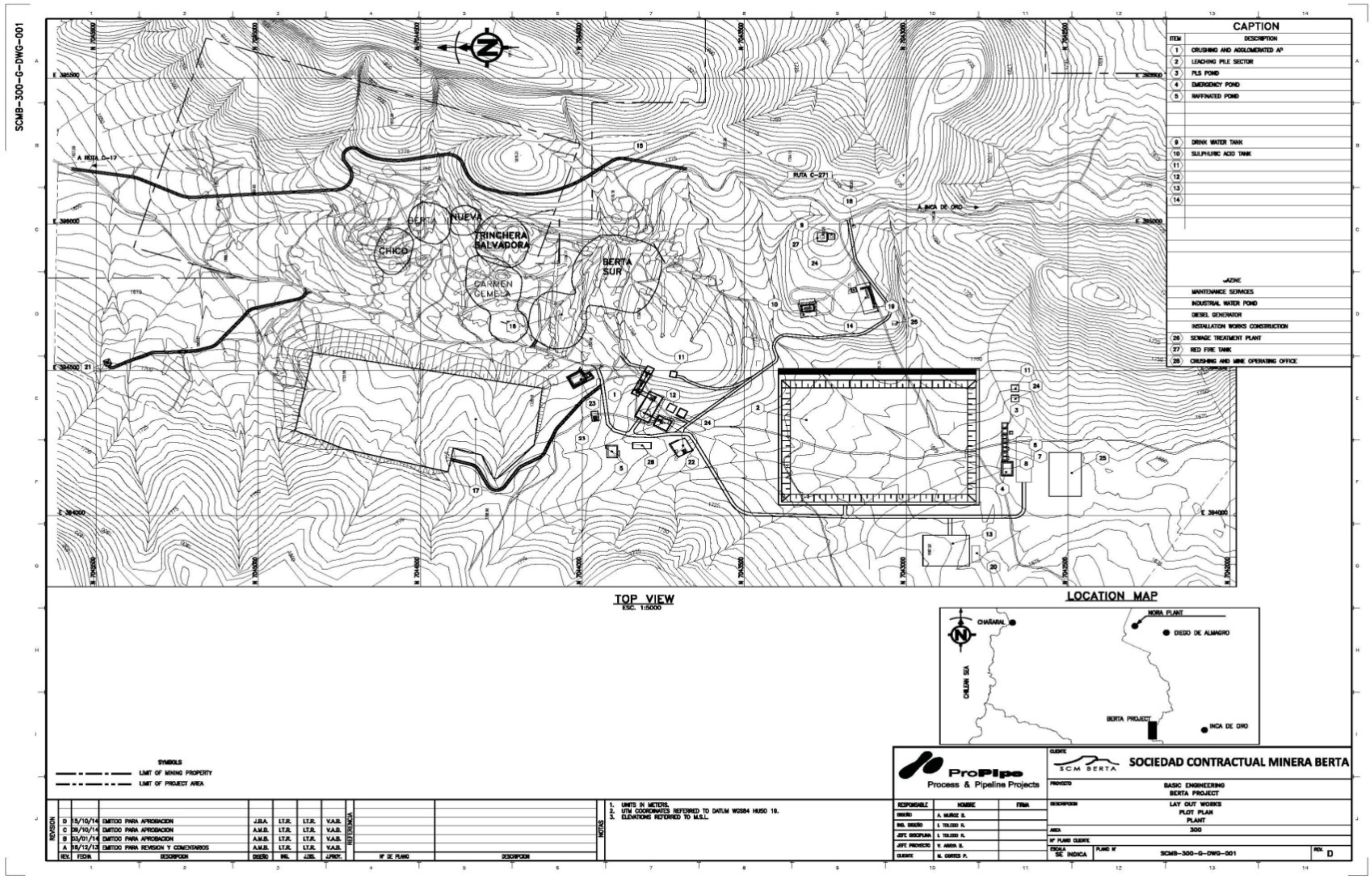
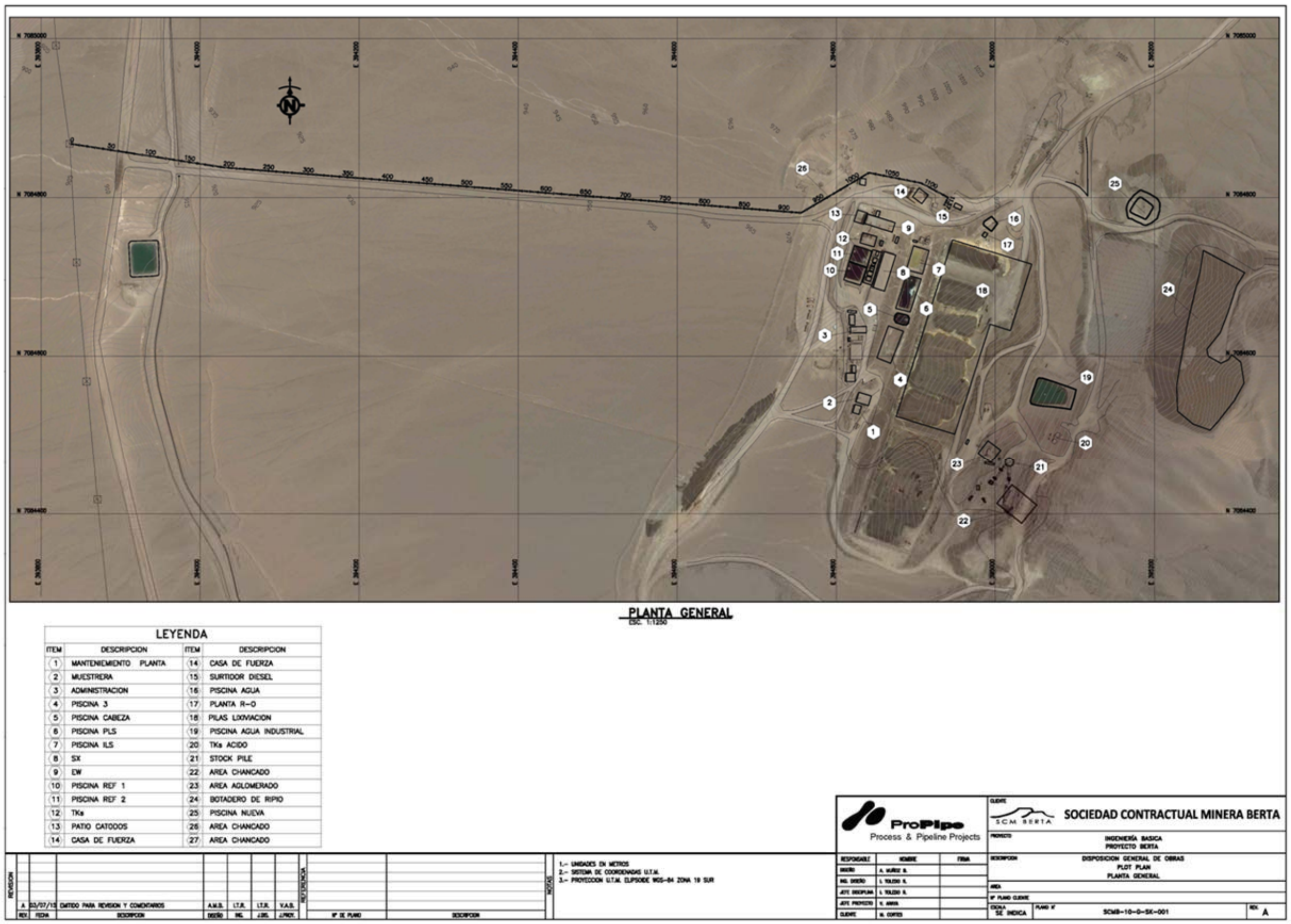


Figure 17.3: Nora Plant layout



17.1 RECOVERY METHODS

17.1.1 NORA PLANT FACILITIES

The Nora plant was built in 2009 and was running till 2012 when its former Owner SCM Trinidad ("Trinidad") went to bankruptcy because of a lack of ore. Nora Plant comprises a 750ktpy crushing circuit and a 3ktpy SXEW plant with associated heap leach pads, spent mineral material stockpiles, piping, PLS ponds etc., together with certain mining properties and surface rights. SCMB is acquiring all of these physical assets, which have been maintained in good condition since 2013 when the plant closed, free of debts, liabilities and liens.

Phase I involves the transportation of HG mineral material from Berta to the Nora plant while Phase II involves the leaching of mineral material at Berta, and the transportation of PLS to Nora for further processing. The two phased approach reduces the initial capital expenditure associated with the project.

The Nora plant is located 60km north of Berta and the Company has secured it through a promise and sale agreement with the Liquidator in the bankruptcy process.

17.1.1.1 TRANSPORTATION OF HG MINERAL MATERIAL - PHASE 1

The Company is anticipating using 30 tonne trucks to transport the HG mineral material to Nora. HG from Berta will be delivered at a rate of ~40,000 tonnes per month for the first 11 months. This will require ~1,333 trips per month or ~45 trips per day. Based on 5 trips per day the Company will require 9 trucks to meet the delivery requirements.

17.1.1.2 CRUSHING AND AGGLOMERATION - PHASE 1

The Crushing Plant at Nora is a three stage circuit, with the final tertiary stage in close circuit with a cutoff at ¾”, which yields approximately 80% passing a ½” sieve, which is the size sent to Agglomeration and then to stacking and leaching process. Figure 17.4 shows crushing plant Layout and Figure 17.5 a process calculation sheet. Trinidad the previous operator ran one 12-hr shift and based on this the crusher had a capacity of 360,000 tpy or 30,000 tonnes per month. To achieve the requiring crushing capacity of 40,000 tonnes per month, the Company would need to run the crusher at more than a 12-hour shift per day. The past production profile is also supportive of the ability to crush the amount of mineral material required.

Figure 17.4: Nora Crushing Plant

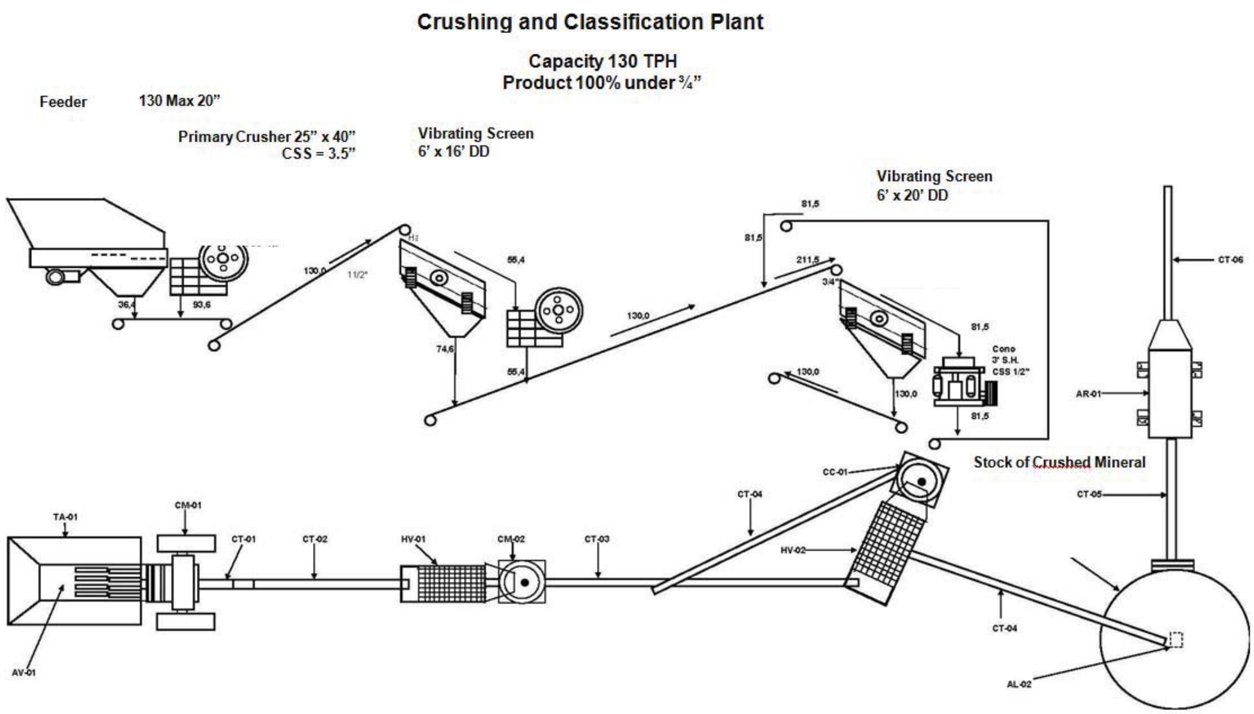


Figura 17.5: Nora Crushing process calculation sheet

CRUSING PLANT CALCULATIONS — PAGE 2

Secondary Crusher — Jaw

CSS 1 ¼"	55,4	
Max	64,0	86,5%
Size	TPH	% passing
6"	55,4	100,0%
5"	55,4	100,0%
4"	55,4	100,0%
3"	55,4	100,0%
2"	49,8	90,0%
1 ½"	44,3	80,0%
1"	23,8	43,0%
¾"	16,1	29,0%
½"	11,6	21,0%
3/8"	8,3	15,0%
¼"	5,0	9,0%
#4	3,9	7,0%
#10	1,7	3,0%
#40	0,6	1,0%
#100	0,3	0,5%

Results 1st and 2nd Stage

Size	TPH	% passing
3"	130,0	100,0%
2"	124,5	95,7%
1 ½"	118,9	91,5%
1"	79,6	61,2%
¾"	60,7	46,7%
½"	47,4	36,5%
3/8"	36,2	27,9%
¼"	18,5	14,2%
#4	15,7	12,1%
#10	7,5	5,8%
#40	3,0	2,3%
#100	0,6	0,5%

Tertiary Crusher — Cone 3' S.H.

CSS ½"	81,5	
Max	100,0	81,5%
Size	TPH	% passing
3"	81,5	100,0%
2"	81,5	100,0%
1 ½"	81,5	100,0%
1"	81,5	100,0%
¾"	69,3	85,0%
½"	55,4	68,0%
3/8"	41,6	51,0%
¼"	17,9	22,0%
#4	13,0	16,0%
#10	7,3	9,0%
#40	3,3	4,0%
#100	1,2	1,5%

Product

Size	TPH	% passing
¾"	130,0	100,0%
½"	102,9	79,1%
3/8"	77,8	59,8%
¼"	36,4	28,0%
#4	28,8	22,1%
#10	14,9	11,4%
#40	6,3	4,8%
#100	1,9	1,4%

At the Nora plant, the mineral from the crushing plant goes directly to the agglomerator drum, which was designed to treat 135 tons of mineral per hour. The drum is 1.8 meters in diameter and 5.8 meters long, and was designed for a residence time of 39 seconds, by allowing the slope to be adjusted between 5º and 11º. In this stage the mineral is treated with sulphuric acid and thensent to the leaching pads in dump trucks.Based on the requirement to treat ~40,000 tonnes per month of material at a rate of 135 tonnes per hour then the Agglomeration drum would have to run for ~300 hours in a given month or ~12.5 hours per day.

The next Table 17.1, shows the equipment list of the Crushing and Agglomeration plant at Nora.

Table 17.1: Crushing & Agglomeration Equipment List

Equipment	TAG	Description	Weight Kg	Power HP	Total PowerHP	Qty.	Total WeightKg
PlantEquipment							
FeedingChute	TA-01	20mts3	9.500	0,0	0,0	1	9.500
MainFeeder	AV-01	HorizontalVibratingwithGrid1.312 mmx2.490 mmRMENA	4.859	20,0	20,0	1	4.859
PrimaryCrusher	JC-01	JawSKET25"x40"	28.600	100,0	100,0	1	28.600
BeltCT-01UnderPrimary	CT-01	24"x8mts RMENA	2.240	10,0	10,0	1	2.240
BeltCT-02toSecondaryVibrating Screen	CT-02	24"x18mts RMENA	4.320	15,0	15,0	1	4.320
SecondaryVibratingScreen	HV-01	3'x10'DD RMENA	2.800	20,0	20,0	1	2.800
SecondaryCrusher	CM-02	JawSANLAND PEF-0507	8.600	100,0	100,0	1	8.600
BeltCT-03toTertiaryVibrating Screen	CT-03	24"x20mts RMENA	4.800	15,0	15,0	1	4.800
TertiaryVibratingScreen	HV-02	4'x10'DD RMENA	7.840	25,0	25,0	1	7.840
TertiaryCone	CC-01	SANLAND 3'S.H.PYB-900	11.000	100,0	100,0	1	11.000
BeltCT-04Return	CT-04	24"x18mts RMENA	3.096	15,0	15,0	1	3.096
BeltCT-05 producttoPile	CT-05	24"x35mts RMENA	9.030	20,0	20,0	1	9.030
Feeder underPile	AL-02	Vibrating700x 1490	1.150	6,0	6,0	1	1.150
Correa CT-06a Agglomerator	CT-06	24"x25mts RMENA	6.000	10,0	10,0	1	6.000
Agglomerator	AR-01	1.800mmDiameter x5.500mmlargoRMENA	12.500	40,0	40,0	1	12.500
BeltCT-07totruck	CT-07	24"x16mts RMENA	3.840	15,0	15,0	1	3.840
Electric andControlRoom		Complete,electric junctions and controlroom	2.500	0,0	0,0	1	2.500
TotalEquipmentPlant					**511,0**		**122.675**

17.1.1.3 LEACHING AREA - PHASE 1

There is currently ~30,000 m^2 of pads lined at NORA. With the crushing and agglomeration circuit producing 40,000 tonnes of mineral material per month (1,333 tonnes per day) this is the equivalent of 740 m^3 (using a density of 1.8:1). On the basis of a 3m height of the leach pads then a leaching area of 250 m^2 per day is required. Over the course of a 60 day leach cycle the physical space requirements are ~15,000 m^2 versus the ~30,000 m^2 available. This will provide sufficient space to have pads that are being unloaded and loaded and that are being drained and washed and pipes being armed and disarmed.

The plan is to have 5 heaps under irrigation, one heap in washing process, one heap in loading process, one heap in unloading process, and open space for loading and unloading job equivalent to the space of two heaps. Heaps will be irrigated with raffinate (from extraction stages two and three of solvent extraction plan) and ILS (intermediate leach solution). Each of these solutions has its own containment pool. The rich pregnant leach solution, or PLS, is conducted to the PLS pond and the intermediate solution to the ILS pond. ILS is obtained when the heaps are almost exhausted, and therefore copper concentration is lower than PLS. Solutions percolate into perforated pipes installed at the base of the heaps and then are conducted through sedimentation channelsto the solvent extraction plant and containment pools which capacity is shown in Table 17.2

Table 17.2: Nora Leaching Pools Capacity

Nora Leaching Pools	Design m^3	Operation m^3
PLS	1869	1558
HeadPLS	174	148
ILS	1430	1215
Raffinate1	873	742
Raffinate2	910	774
Emergency	2282	1939
Plant WaterReservoir	2989	2541
Pampa Austral WaterReservoir	2000	1700

Once the exhausted heaps are drained and washed, the spent ores, also called "Ripios", are loaded and placed at Ripios waste dump. Fig 17.6 shows the actual sites facilities.

Figure 17.6: Actual sites facilities at Nora

17.1.1.4 SOLVENT EXTRACTION AND ELECTRO WINNING - PHASE 1 & 2

The Nora SXEW plant will be expanded from its existing 3,000tpy capacity to 5,000tpy of high purity copper cathodes per year.

The reception of the PLS solution will be held in the existing PLS pond (30m wide x25m long x 3m deep), and from this pond the PLS solution will be pumped to the SX stage, which consists of five mixer-decanters tanks made of reinforced fiber glass material placed on concrete foundations. These are located in a container of 15m x 48m.

The five mixer-decanter wells and pipeline and valves configuration allow the circuit to operate the circuit in series or parallel to achieve greater operational flexibility. Figure 17.7 shows the mixer-decanter equipment.

Figure 17.7: Mixer-Decanter Tank

The process of concentration and purification of solutions named Solvent Extraction (SX) is based on a reversible chemical reaction of ionic exchange generated by contacting a rich solution in the element of interest (Cu2+) and an organic solution, both immiscible with each other:

[Cu+2 / SO4-2] PLS Solution + 2 [R – H]Organic Phase $\rightleftarrows$ [R2 – Cu]Organic Phase + [2H+ / SO4-2]Raffinate Solution

Extraction

The metal element of copper in the PLS solution in ionic form (Cu2+) when contacting with the organic phase carries the element of interest to the organic phase and this releases a hydrogen (H+), which turns into an increase in the acidity of the aqueous resulting phase, now with a lack of copper, and typically named raffinate.

The raffinate solution will be sent to the existing raffinate ponds at Nora (2 solution pools of 20m wide x 20m high x 3m deep), and from this point the raffinate solution will be loaded on water trucks for its transport back to Berta.

Based on the reversibility of the chemical reaction of the extraction, it is possible to re-extract the extracted metal by another aqueous phase (Poor Electrolyte) from the organic phase, by the simple mechanism of pH incensement or acidity of the system (H+), thereby meaning that the chemical reaction is reversed:

[R2 - Cu] Organic Phase + [2H+ - SO4-2] Spent Electrolyte $\rightleftarrows$ [Cu+2 - SO4-2] Loaded Electrolyte + 2 [R - H] Organic Phase

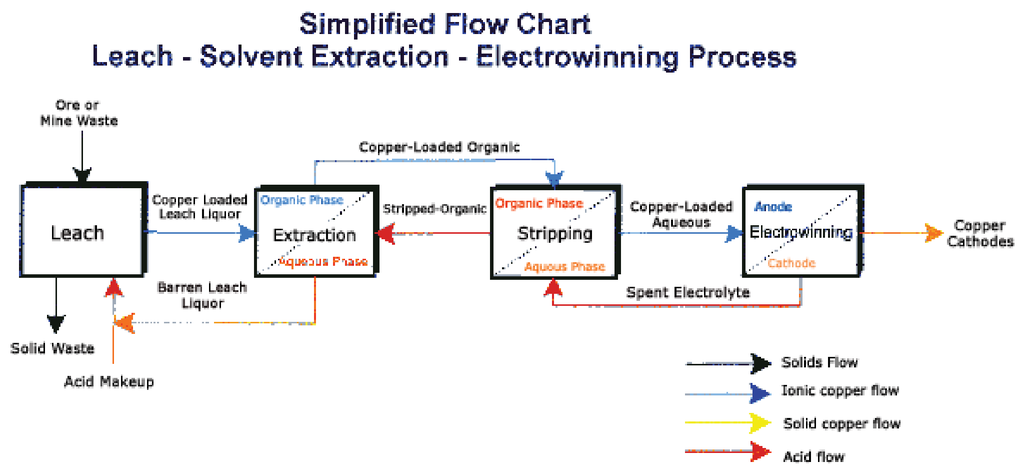
Re-extraction

Thus in the so-called phase of re-extraction the organic loaded obtained on the previous extraction step is contacted with a solution named Poor Electrolyte that returns from EW to SX, and by its high acidity condition produces the inverse reaction of re-extraction, so, the copper of the organic phase is transferred to the aqueous phase which in this case is the same electrolyte.

EW will take place in an electrolytic chamber of 28 cells of vinyl-ester polymer concrete (polymer concrete), with capacity of 30 permanent stainless steel cathodes and 31 anodes of alloy lead/calcium/tin in each cell, an electric circuit fed by two transformers and two DC rectifier generate the electric current needed to generate the electrolytic deposition of copper at the cathode of stainless steel.

The Figure 17.8 shows a general scheme of the SX and the transference of copper to the EW process for the production of copper cathodes.

Figure 17.8: SX EW copper Transfer Process



Figures 17.9 and 17:10 shows Nora Plant Tank Farm and EW building.

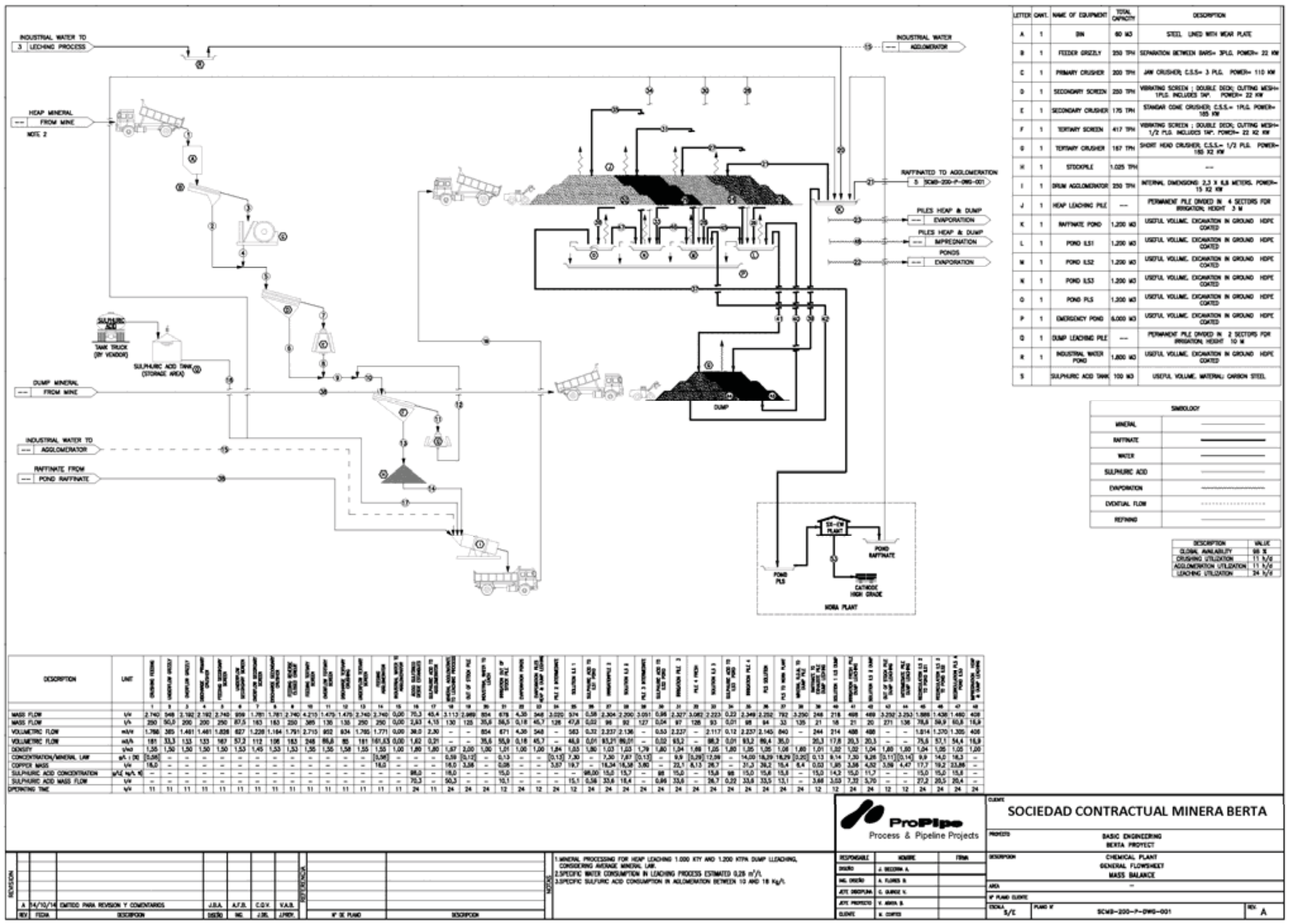
Figure 17.9: Nora Tank Farm

Figure 17.10: Nora EW Building

17.1.2 BERTA SITE FACILITIES (PHASE 2)

Figure 17.11: shows the general flow diagram of Berta plant and its respective mass balances.

Figure 17.11: Flow Diagram of Berta plant/Mass Balances



17.1.2.1 CRUSHING

The material from Berta open pits, with an average grade of 0.57% CuT, will be trucked to the crushing plant.

The plant considers three stages of crushing in reverse closed circuit. The feeding of the crushing process is made directly from mine truck to a primary hopper, and grizzly vibrating screening, which separates the material over 3" sending it to the primary jaw crushing; and then to the conveyor belt that feeds the secondary stage of crushing. The material under 3" goes directly to the secondary crushing feeding conveyor.

The secondary crushing stage involves a double deck screener and a standard type cone crusher. In this stage the primary material product feeds the screener that sends the 1" oversize to the secondary crusher; and the 1" undersize material to the third crushing feed conveyor.

The tertiary crushing stage comprises 2 double deck screeners and 2 short head cone type tertiary crushers in reverse closed circuit. The secondary crushing product and the material from the tertiary crusher recirculation, feeds tertiary screeners. The material sizing more than ½” will feed the tertiary crushers and then recirculated to the 2 tertiary screeners. The final crushing product sizing less than ½” will be sent through a conveyor belt to a stockpile with 1,025 tonnes of capacity.

To control emissions, the plant considers the use of dust suppressants of dry haze type and chutes in all the mineral transfers, and covers for all the plant conveyor belts.

Figure 17.12 and 17.13 shows the Flow diagram of Crushing circuit.

Figure 17.12: Flow Diagram Primary and Secondary Crushing

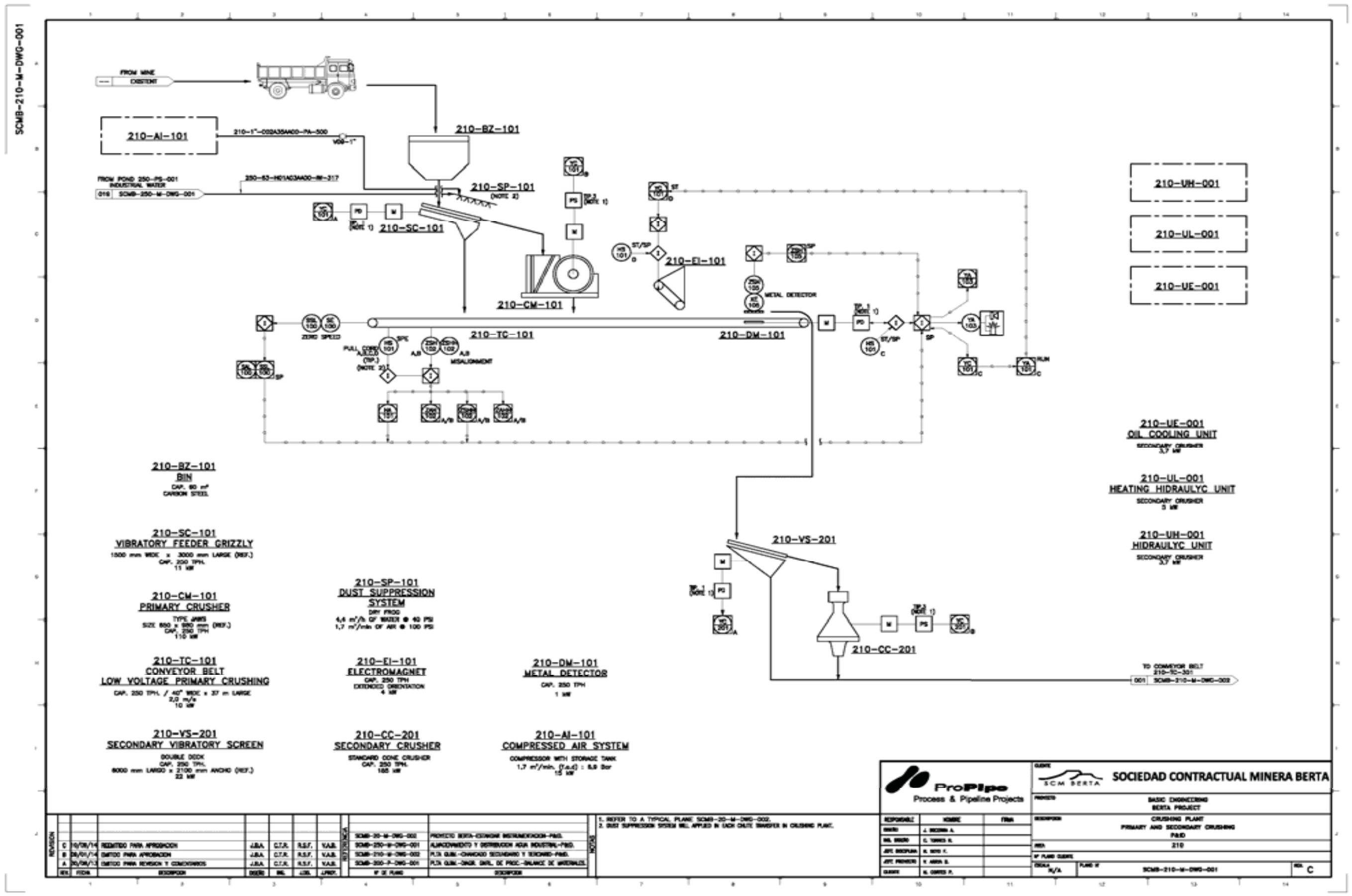
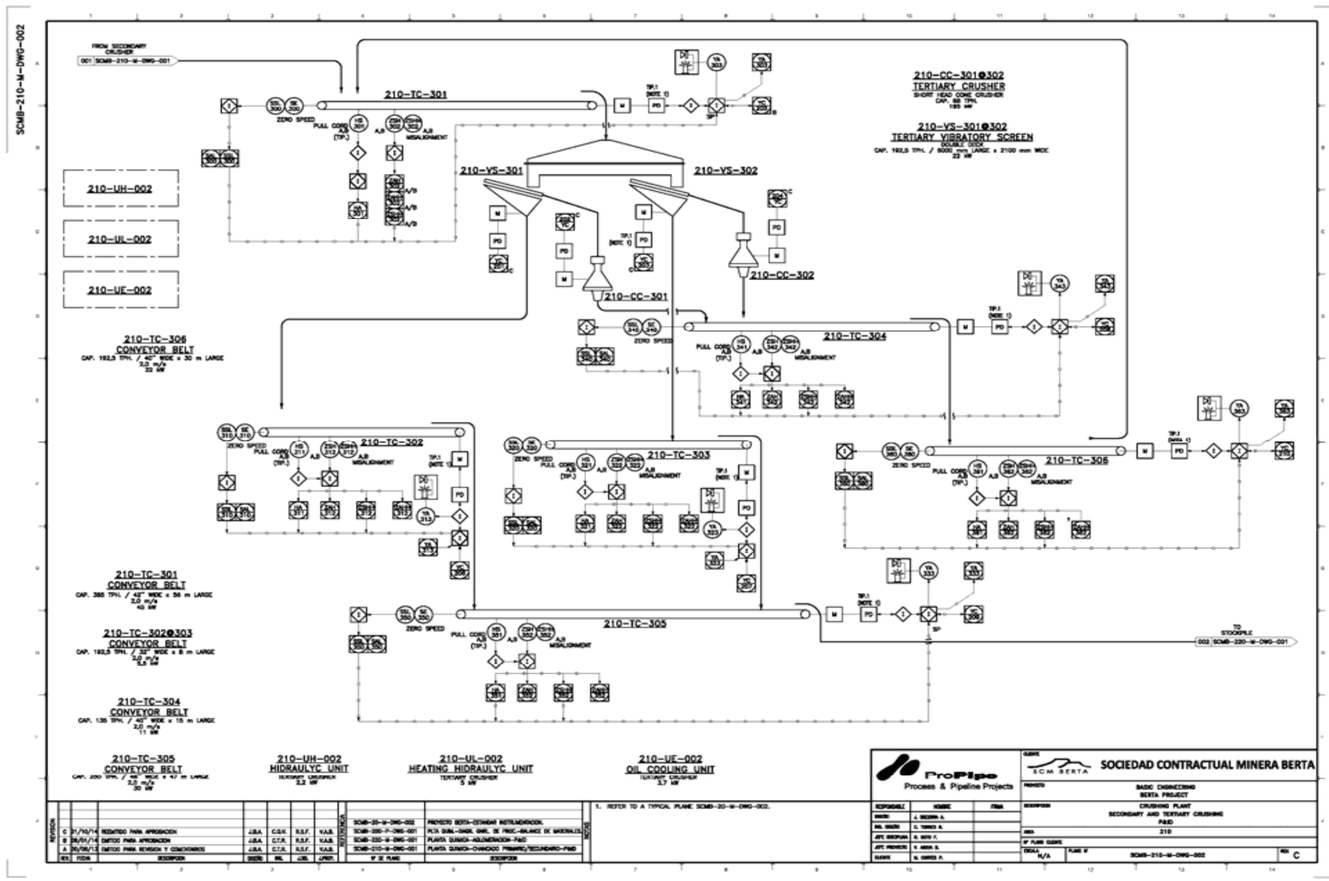


Figure 17.13: Flow Diagram Secondary and Tertiary Crushing



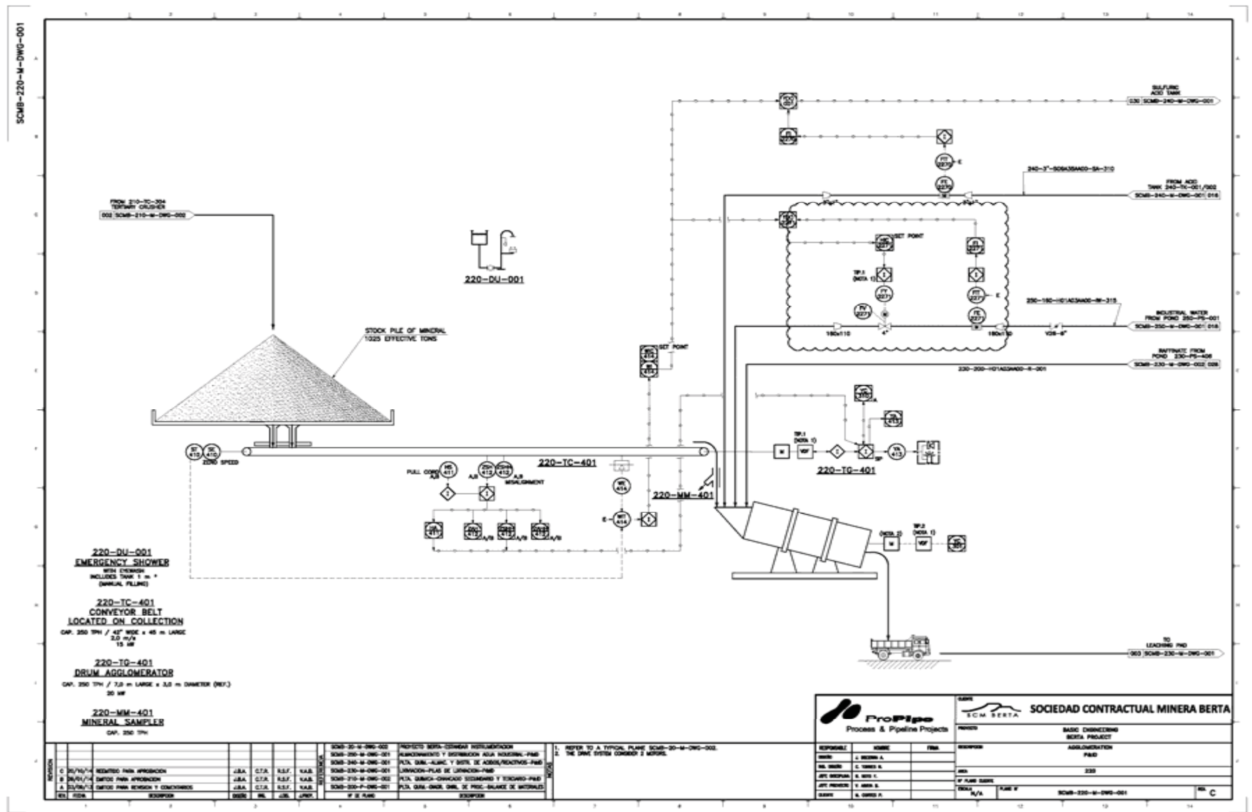
17.1.2.2 AGGLOMERATION

For this stage of the process, there will be an agglomeration drum with 250t/h of capacity, to mix the oxide material with sulphuric acid and water.This results in the leaching and agglomeration of the particles, with fines adhering to larger particles, thus improving the permeability of the heap.

Transport period and repose time of the agglomerated material, before irrigation in the leaching stage, allow the cohesion of fine and gross particles to be optimal for the mechanical resistance during leaching process.

Figure 17.14 shows the Flow Diagram Agglomeration Process

Figure 17.14: Flow Diagram Agglomeration Process



17.1.2.3 STACKING AND LEACHING

Oxide material grading with more than 0.30%CuT are treated through heap leaching and those less than 0.30%CuT are treated directly in the dump leach area. Leaching involves an area of 24 hectares (600m long and 400m wide) for the construction of the heaps which will be properly compacted and lined with a 300g/m^2 geotextile and covered with a 1.5mm thick HDPE carpet.

In the heap leaching stage, the agglomerated material will be placed in a 3m high truncated pyramid form, with an average grade of 0.57%CuTanticipating to recover 78% of the total copper available. The 60 days leaching cycle is divided in 4 sub cycles of 15 days each, every cycle generates an ILS solution that will serve to the next cycle.

Irrigation of heaps will be made with droppers preferably, but there will be flexibility to switch to sprinkler irrigation if the process allows it. The irrigation will be made under the concept of counter-current circuit with the inner generation of 3 ILS solutions, to reach a final solution of PLS of 18g/l of copper. The irrigation rate is estimated at between 10l/h/m^2 and 12l/h/m^2.

Heap irrigation cycle begins once is over the load of the new mineral module. This mineral is irrigated for 15 days with ILS solution Nº3 and the drainage of this module is PLS solution, then the same module is irrigated during 15 days with ILS solution Nº2 and the drainage solutions are channelled to the ILS solution pool Nº3, then this is irrigated during 15 days with ILS solution Nº1 and the drainage is channelled to ILS solution pool Nº2 and finally this is irrigated during 15 days with raffinate solution and the drainage is channelled to ILS pool Nº1.

ROM leaching process will be done in a 40 day cycle divided in 2 sub cycles of 20 days each. The stockpile generated for this process will be 10m high and the recovery is estimated at 45% of the total copper available. The irrigation cycle of the new stockpile will be irrigated with ILS solution Nº1 and the drainage solution will be sent to ILS pool Nº2, the second cycle of irrigation correspond to the washing stage and will be made with raffinate solution and the drainage flow will be sent to ILS pool Nº1.

PLS solution of $35m^3/h$ at a concentration of 18g/l de Cu^{2+}P will be the product of heap and ROM leaching processes. The PLS will be sent by tanker trucks to SX EW of Nora Plant, and from there it will return an equal volume of raffinate solution that will be stored in the raffinate pond at Berta.

There will be 4 passive wells in the downstream area of leaching stockpile and solutions pools to detect any leakage of solutions, similarly a fifth monitoring well upstream of the leaching stockpile will be installed, which will be the reference well.

Figure 17.15, 17.16 and 17.17 shows the flow diagrams of the leach circuit

Figure 17.15: Flow Diagram of Leaching Plant –ILS Reffinate and Emergency Pond

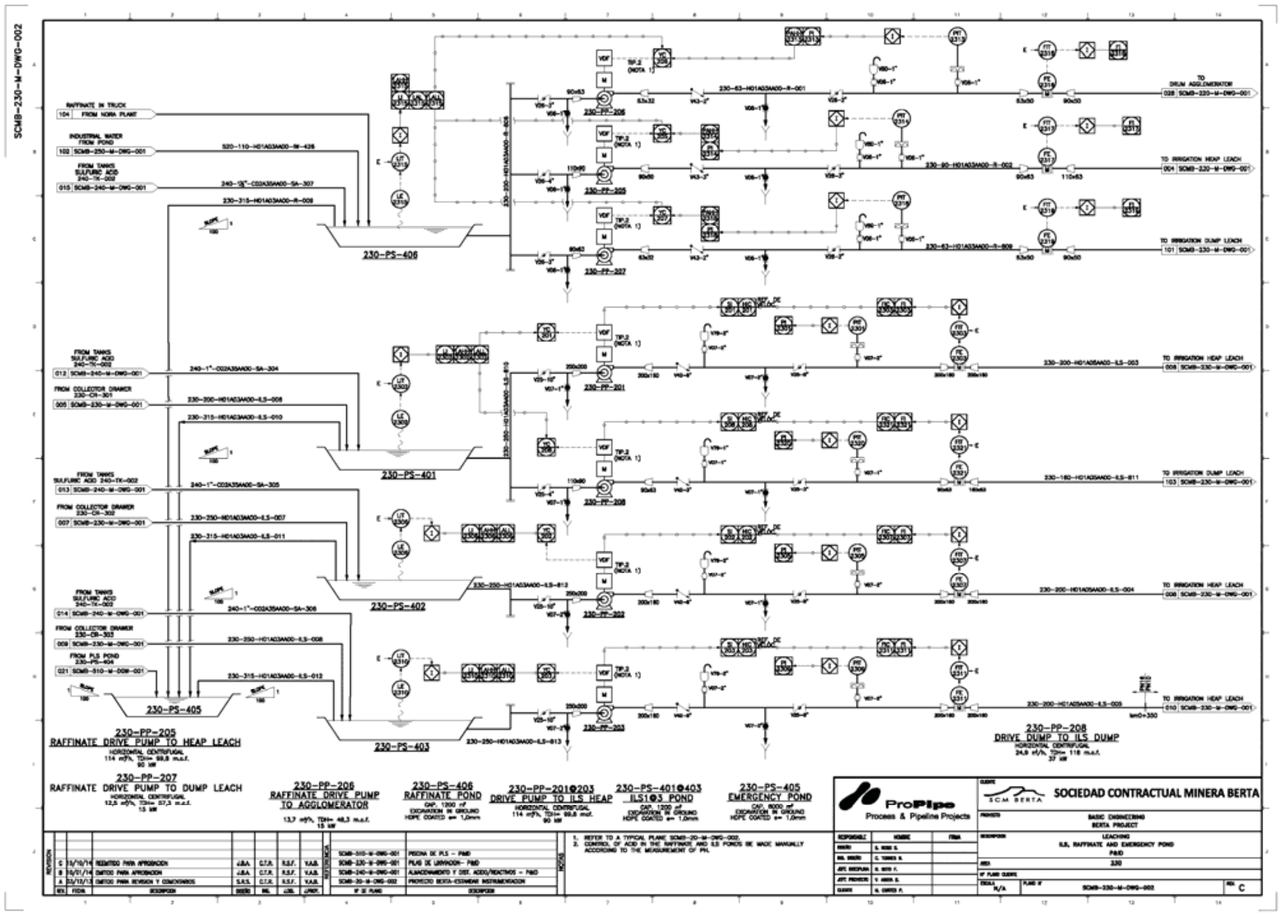


Figure 17.16: Flow Diagram of Leaching Plant –Solution Transport System

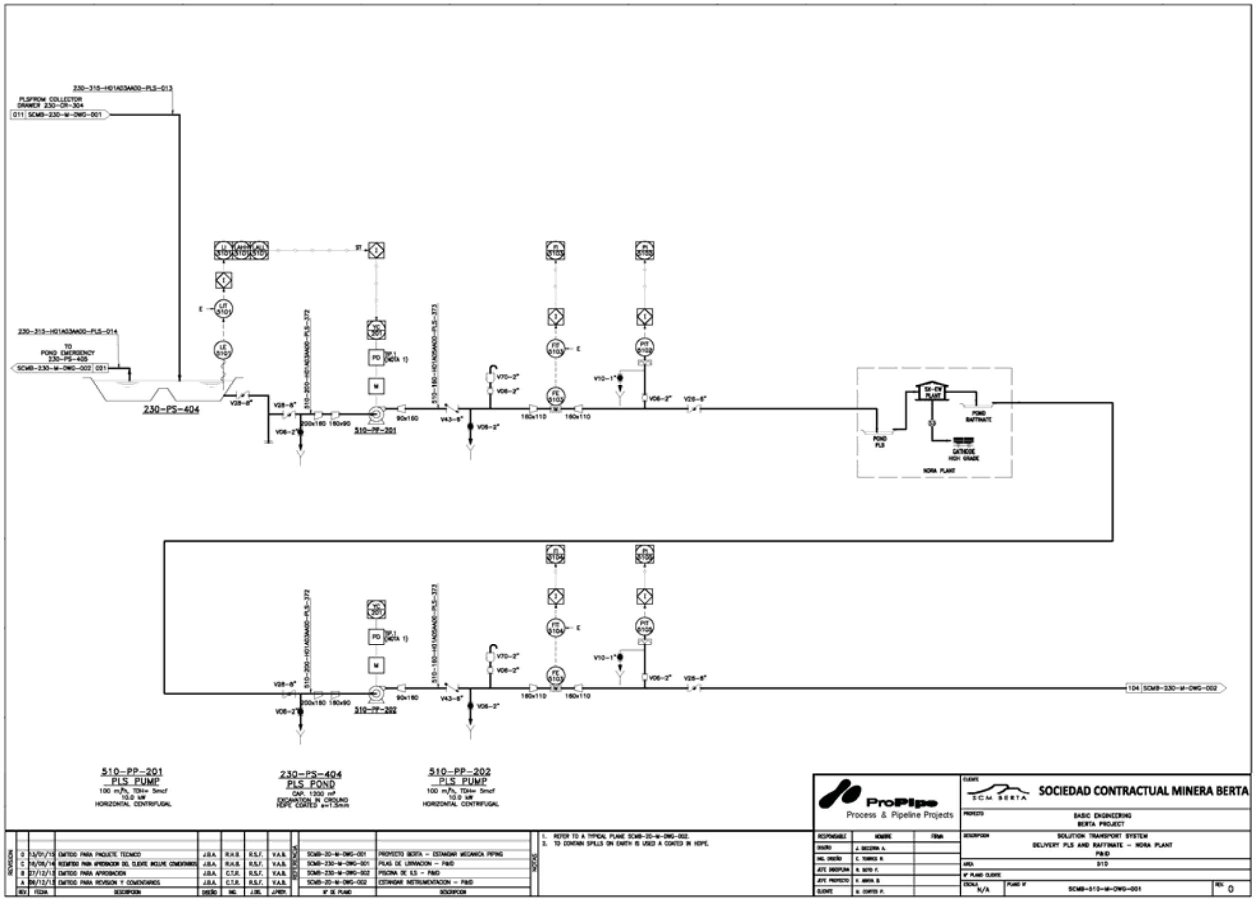
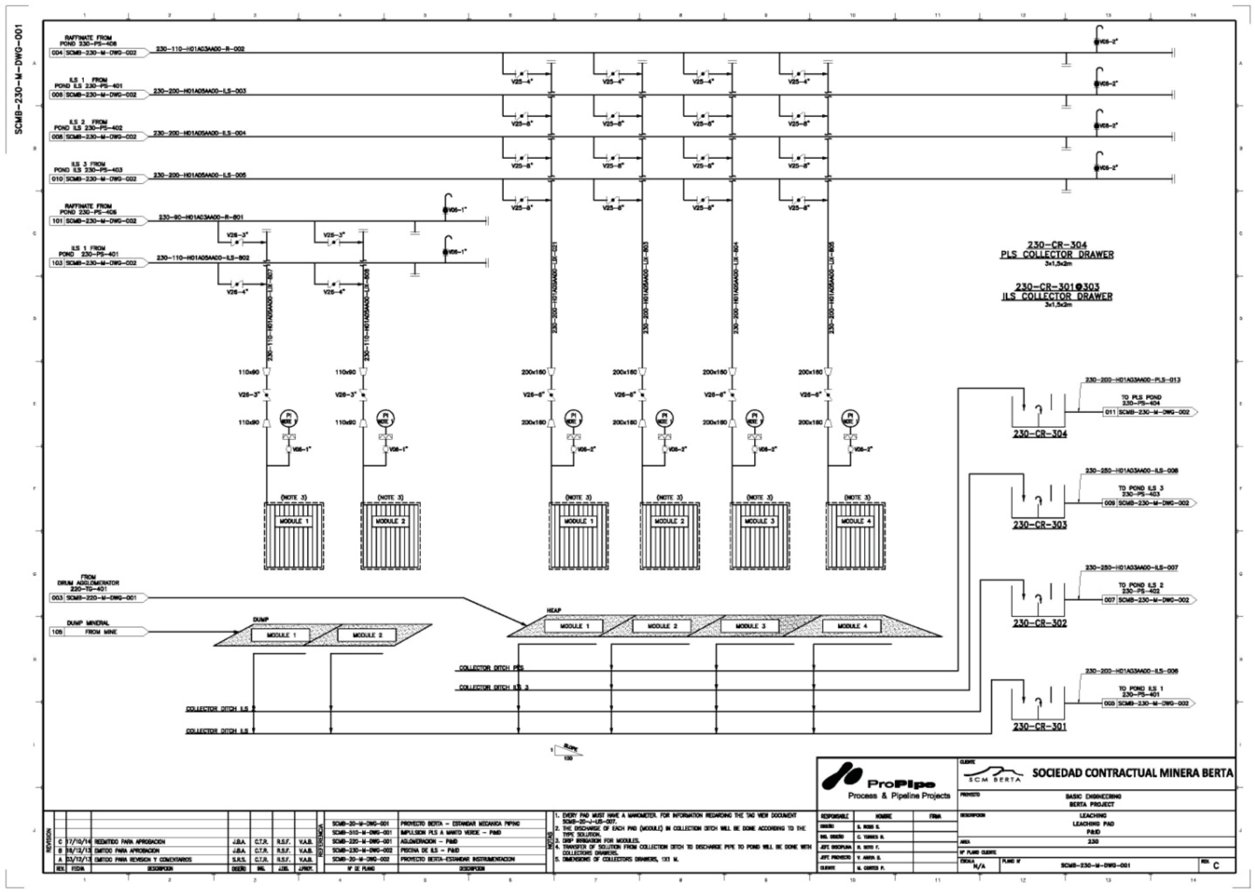


Figure 17.17: Flow Diagram of Leaching Plant – Leaching PAD



18.0 PROJECT INFRASTRUCTURE

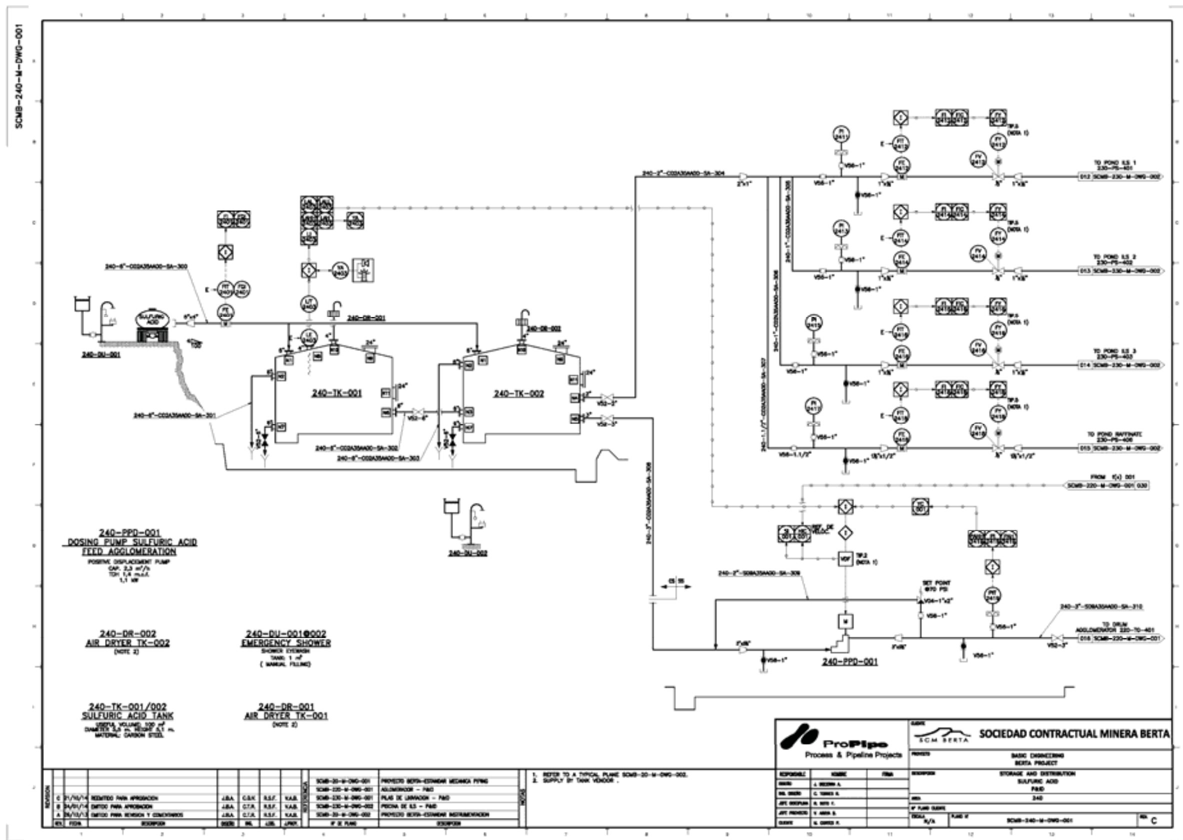
18.1 SULPHURIC ACID SUPPLY AND INTERNAL DISTRIBUTION

Sulphuric acid supply is made directly to Berta at 5 trucks per day (149tpd), which will be received in two carbon steel tank of 100m^3 each, representing supply autonomy for 3 days. Tanks will be mounted on lined foundation pond with perimeter parapets with capacity of 1.1 times the volume of a tank. In case of spillage it has a secure access gateway that allows for closing of the valve without risk of exposing operators to contact with sulphuric acid. The tank has all the facilities to perform maintenance activities.

Trucks with sulphuric acid will be received in a platform located above the tank, specially prepared to perform a downloading through a fast and secure connexion. The acid is discharged in a carbon steel pipeline that feeds the tanks of reception and distribution, through a manifold that allows for loading of either tank.

The acid distribution to the agglomeration stage is via a variable speed pump, and leaching ponds via a gravitational carbon steel pipe. Figures 18.1 shows the Sulphuric Acid Storage and Distribution System.

Figure 18.1: Storage and Distribution of Sulphuric Acid



18.2 WATER SUPPLY AND PLS & RAFFINATE PIPELINES SYSTEM

Industrial water supply will come from the Pampa Austral tailings dam (TPA), which feeds a water pond located in front of Nora Plant facilities. From this pond, water will betransported to Berta at 10.8l/s, equivalent to 33 trips per day over a distance of 60km by road. The industrial water deposit at Berta will be made directly into the industrial water pond and from there will be distributed to the consumption points that are raffinate and agglomeration ponds.

The Base Case assumes the supply of water from the Pampa Austral Tailings Dump (TPA) to Nora using the existing infrastructure.

For the transport of raffinate and wáter from Nora to Berta, a single pipeline is considered through which would be pumped water for 3 days per week and raffinate for 4 days a week.

The Berta installations would include a storage pond for the process waters to ensure continuous operation.The course of the pipeline has been designed as shown on the following Figure 18.2:

Figure18.2: PLS and Water/Raff pipeline

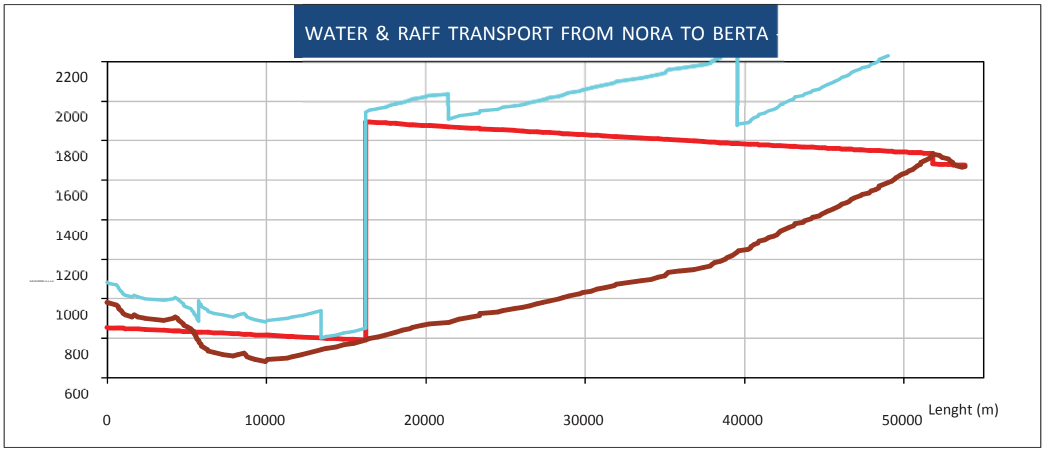
The hydraulic dimensions of the pipeline have been calculated as follows:

Water/Raffinate pipeline design

- A low pressure gravitational pipeline would be installed between Nora and the valley of the Rio Salado, using an 8" diameter HDPE pipe
- After crossing the valley, a tank with pumps would be installed to pump solutions to Berta by means of a 6" Steel pipe lined with HDPE.
- The pumping system would be connected to the SIC electricity grid

The following Figure 18.3 shows the hydraulic profile for the wáter/raffinate pipeline, with a design flow of 89 m3/h (35 m3/h water + 54m3/h raffinate pro-rata) in order to satisfy the demands of the project.

Figure18.3: Water/Raff Pipeline Hydraulic profile



The estimated capital and operating costs for this system of pumping water and raffinate is shown in Table 18.1

Table 18.1: Water/Raff Capex and Opex

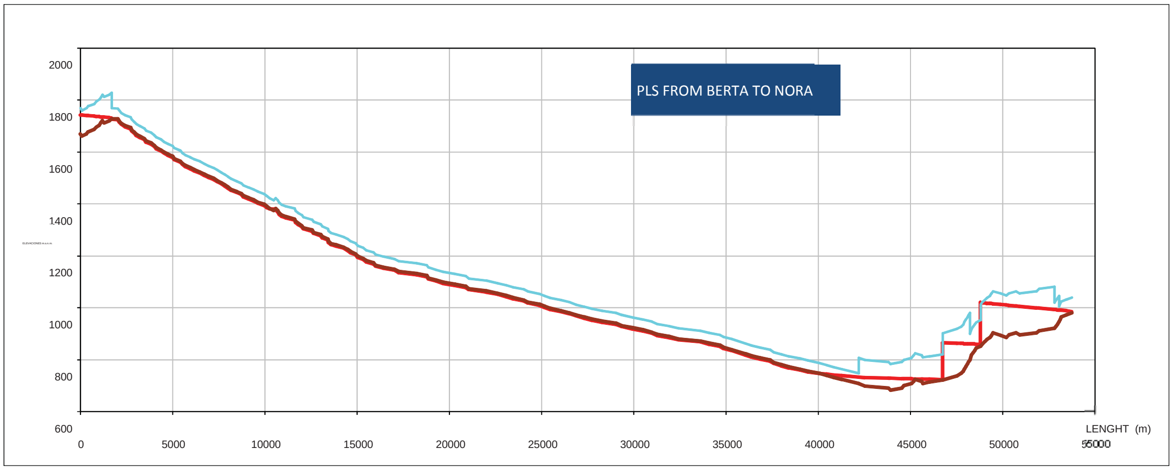
Capex		kUSD
Pipeline supply & installation		2,413
Pump stations	193	
Indirect costs	415	
Total Water/Raffinate		**3,021**
Opex (power from grid)		
	USD/m3	0.82

PLS pipeline design

The transport of PLS from Berta to Nora considers the same course as the wáter/raffinate pipeline, with the section from Berta to the Rio Salado valley comprising a gravitational 6” diameter HDPE pipe followed by an 8” diameter HDPE pipe, to maintain low velocity while crossing the valley. Once the valley has been crossed, 2 low pressure pump stations connected to the grid will be employed to pump the PLS up to the Nora plant through HDPE piping.

The following figure 18.4 shows the profile of the PLS pipeline with a design flow of 54 m3/h.

Figure 18.4: PLS Pipeline Hydraulic Profile



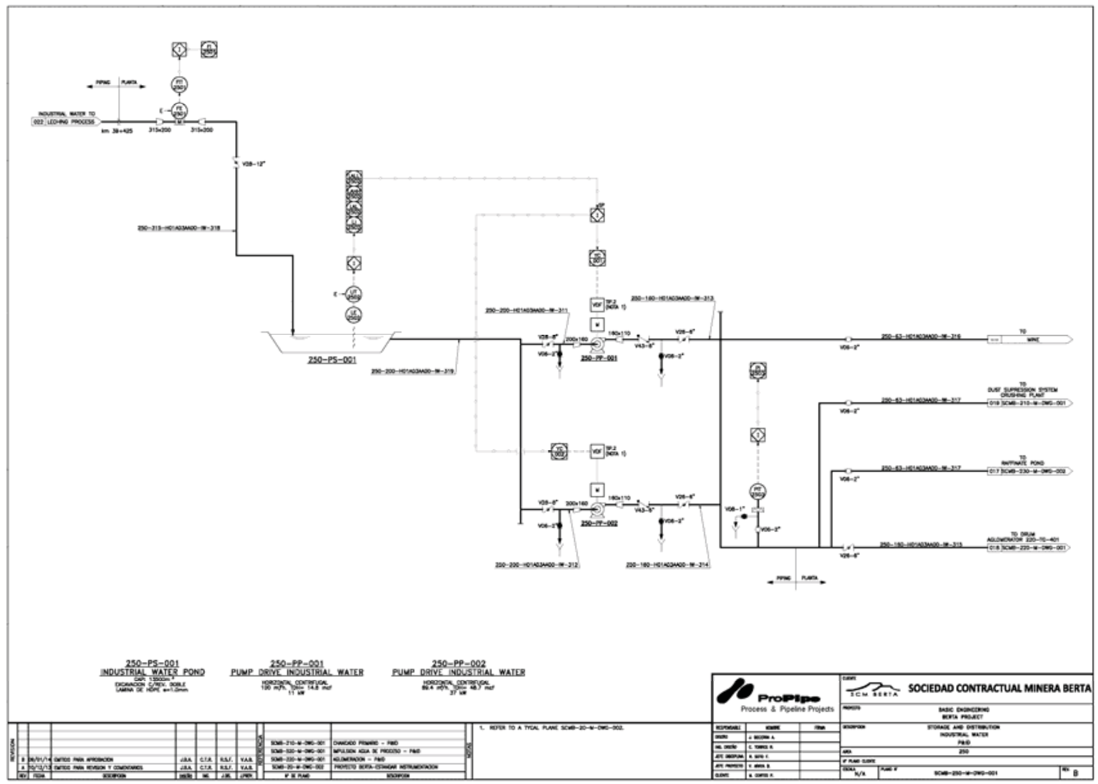
The estimated capital and operating costs for this system of pumping PLSis shown in Table 18.2

Table 18.2: Capex and Opex

Capex	kUSD
Pipeline supply and installation	562
Pump Stations	35
Indirect costs	119
Total PLS	**716**
Opex (Berta with generators &Nora connected to the grid)	
USD/m3	0.40

Figure 18.5 shows the Storage and Distribution of Industrial Water

Figure18.5 Storage and Distribution of Industrial Water



18.3 POWER SUPPLY

The power requirement for Berta site is 1.4MW, which includes the operations of crushing, agglomeration, leaching, and other lower consumption requirements. Table 18.3 shows the Berta site power requirements.

Table 18.3: Power Requirement for Berta Project

Process	Requirements	Quantity/Capacity Generation
Crushing / Agglomeration	1,000 Kw	2 Generators de 600 KW
Leaching	300 Kw	1 Generators de 400 KW
Other services	100 Kw	1 Generators de 150 KW

The project considers that Nora Plant will be connected to the central energy grid with a power requirement of 2.0 MW for the SXEW process.

18.4 OTHER FACILITIES

The project includes additional facilities that are already built at the Nora Plant such as metallurgical laboratory, maintenance workshop, warehouse, dining facilities and general offices.

At the Berta mine site, the project considers the following facilities, based on modified containers for the Berta operations:

- Administration Offices
- Dining Room
- Sample Preparation Room
- Storage of Materials
- First Aid Room
- Plant Maintenance Room
- Changing Room
- Fuel and Lubrication Station
- Waste Storage

19.0 MARKET STUDIES AND CONTRACTS

This analysis is based on a projected annual capacity of 5,000 metric tonnes per year of ASTM B115 Grade 1 quality cathode. There are no contracts currently in place or required for the sale of copper cathode for this project.

19.1 RECOMMENDED MARKETING STRATEGY

The projected production volume is too small to justify a direct marketing effort, and it is recommended that full production be tendered to a select group of buyers. This strategy will ensure immediate market entry and expedited payment with little or no credit risk in an increasingly competitive regional market.

Metal offtakershave generally been eager to source new production from non-integrated producers in order to displace higher cost imported cathode, and to provide more competitive prices. Consequently, new cathode producers haveobtained reasonable terms under long-term off-take contracts from major Asian, US and/or Europe buyers, usuallyfor 100% of production.

These contracts include provisions for very prompt payment facilities. With a suitable counterparty these contracts could also be tied to bridge loans, senior or subordinated project debt facilities and/or corporate or project equity purchase agreements.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

This section of the preliminary study report:

- States SCM Berta's environmental philosophy;
- Summarizes the Chilean legal requirements for environmental protection;
- Outlines the baseline data collection results; and
- Summarizes possible impacts to the environment and the mitigation plans for project development, operation and closure.

20.1 ENVIRONMENTAL PHILOSOPHY

20.1.1 ENVIRONMENTAL POLICY

SCM Berta´s policy is to meet or exceed all environmental requirements established by Chilean Law. In the same manner, SCM Berta will require that all of its contractors and employees comply with SCM Berta's policy on environmental regulations. The following environmental policy will be adopted for the Berta Project:

- All applicable environmental laws will be given due regard during the planning and implementation of facilities' construction and operation;
- Management and employees at all levels will be kept aware of the project's environmental responsibilities through proper training;
- Procedural reviews and risk assessment will be conducted periodically to identify any aspect which could cause environmental damage during facility construction and operations to permit reasonable corrective actions to be taken in a timely manner; and
- At all organization levels, the environmental factors will be considered in the decision making process.

20.1.2 ENVIRONMENTAL MANAGEMENT SYSTEM

Below are the key elements of SCM Berta's Environmental Management System:

- Environmental issues are considered in the early stages of projects. MMDS (Mining and Minerals for Sustainable Development), IFC (International Finance Corporation) and WBG (World Bank Group) guidelines and codes of best practice are taken into account for any of SCM Berta's projects;
- Environmental issues will be incorporated into project procedures wherever relevant;
- Environmental objectives will be included in the key performance indicators for projects;
- Monitoring programs will be established to provide early warning of any deficiencies in the project's safeguards;
- A review of management procedures will be made periodically to confirm their suitability for complying with the environmental responsibilities;
- Audits will be conducted to verify environmental compliance at appropriate intervals;
- An environmental incident reporting system will be established and reports issued in a timely manner; and
- Procedures will be established to assure fluid communications with government agencies.

20.1.3 ENVIRONMENTAL PERMITTING FRAMEWORK

As set out in the Article 10 of the Law 19.300 about General Bases of the Environment, modified by the Law 20.417 which creates the Ministry of the Environment, and in the article 3 of the Supreme Decree Nº 95/01 that modifies the Regulation of the Environmental Impact Assessment System:

> *"Projects or activities likely to cause environmental impact, in any of their phases, must be subject to Environmental Impact Assessment, corresponding to the letter i):*
>
> *i) Mining development projects, including coal, oil and gas, comprising exploration, exploitation, processing plants and waste disposal .*
>
> *Mining development projects are those who'saim is the extraction or beneficiation of one or more mineral deposits, whosemining capacity is greaterthan five thousand tonnes per month"*

According to the foregoing, and given that Berta is a mining project that contemplatesan extraction rate higher than 5,000 tonnesper month, the project wassubmitted to the Environmental Impact Assessment System viaan Environmental Impact Declaration.

The Lawspecifies a deadline for the EIA approval process, whichnormally requires at least 60 days from the date the DIA is submitted. A period of 90 days has been estimated according to an analysis of the administrative schedule, with no considerable delays. The approval process is completed with an official Environmental QualifyingResolution (RCA), which approves the DIA and establishes obligations for future environmental management of the project.

On November 7th 2013, SCM Berta submitted the Environmental Impact Declaration (EID) and the Evaluation Commission of the Atacama Region of Chile, part of the Chilean Environmental Evaluation Service (in Spanish, "SEA"), has approved the EID of the Berta copper project and has emitted the corresponding Resolution of Environmental Qualification (in Spanish, "RCA") on 24 October 2014.

20.2 BASELINE SURVEYS

20.2.1 GENERAL INFORMATION

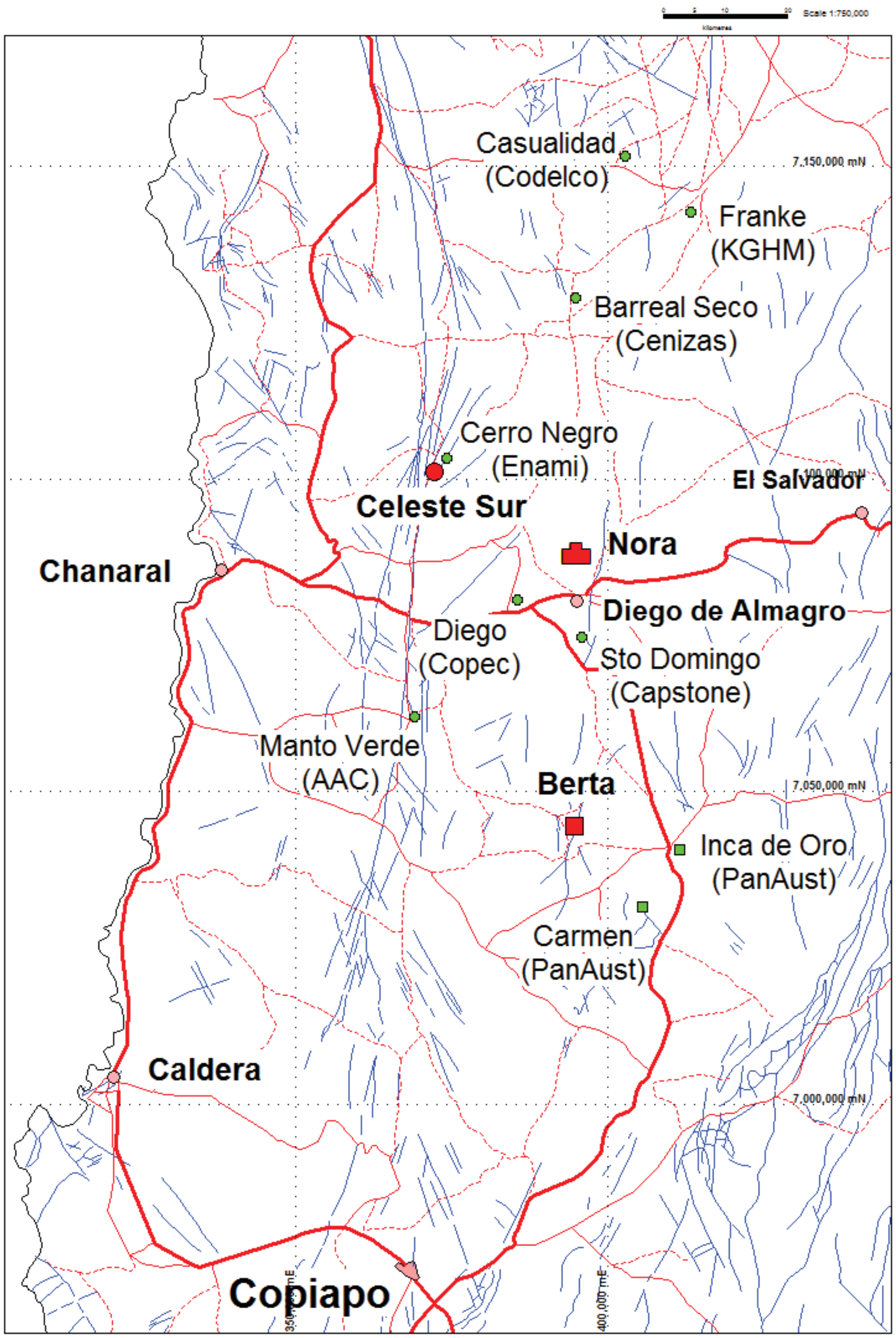
For the Berta project, definitive environmental studies have been completed, and provided a detailed description of the mine site environmental conditions such as climate and meteorology, hydrology, hydrogeology, archaeology, flora and fauna, and social background of Diego de Almagro and Inca de Oro.

The baseline studies for the DIA related to Berta Project are based upon studies completed by IAL Consultores Ltda. The dust emission model was performed Jorge BordoliIngenieríaAmbiental Ltda.

The Property is located in Chañaral Province, III Region, Northern Chile, at the approximate latitude 26º43’S and longitude 70º03’W, approximately 20 km West of the village of Inca de Oro, at an elevation of 1700 m. It is situated about 750 km North of Santiago, 75 km North-Northeast of Copiapó, and 70 km South east of the port of Chañaral (Figure 19.1). The UTM coordinates of the center of the Property are approximately 395,000 E and 7,044,100 N, UTM Zone 19-J, Provisional South American 1956 datum.

The project is about 33 km east of AngloAmerican´sManto Verde operation that produces 60,000 t of Cu per year. Codelco’s El Salvador mine, with a production of 69,000 t Cu is located about 68 km northeast of Berta. The Inca de Oro (PanAust Limited) and Santo Domingo (Capstone Mining Corp.) development projects are located 15 km east and approximately 30 km northeast respectively. The project is located in a mining region that also contains numerous operations of small and medium mining of Cu and Au, several of which supply mineral material to the state mining company ENAMI, which has a processing plant in the town of El Salado located about 41 km northwest of Berta (Figure 20.1).

Figure 20.1: Location Map (North to the Top)



20.2.2 CLIMATE

The Property is located in the intermediary depression of the Chilean Atacama Desert. The normal desert climate, present in areas between 1,000 and 2,000 m.a.s.l., is characterized by a very low relative humidity, virtual lack of precipitation, practically no oceanic influence and clear skies during the whole year[5].

The average annual precipitation and evapotranspiration rates do not exceed 10 mm/year (IDICTEC, 1994). Due to the extreme aridity, there is practically no natural vegetation or fauna in the area, with the exception of occasional insects, lizards and small mammals.

The Atacama Desert along the Pacific Coast of Chile and Peru is one of the driest, and possibly oldest, deserts in the world. A detailed study conducted between 1994 and 1998 (McKay et al., 2003), determined that the average air temperature was 16.5°C and 16.6°C in 1995 and 1996, respectively. The maximum air temperature recorded was 37.9°C, and the minimum was -5.7°C. Annual average sunlight was 336 W/m^2 and 335 W/m^2 in 1995 and 1996, respectively. Winds averaged a few meters per second, with strong *föhn*[6] winds coming from the west exceeding 12 m/s.

Between 1994 and 1998 there was only one significant rain event of 2.3 mm, possibly as rainfall from a heavy fog, which occurred near midnight local time. It is of interest that the strong El Niño of 1997-1998 brought heavy rainfall to the deserts of Peru, but did not bring significant rain to the central Atacama Desert in Chile. Dew occurs frequently following high levels of night time relative humidity, but is not a significant source of moisture in the soil or under stones. Groundwater also does not contribute to surface moisture.

At Inca de Oro, a typical location with this kind of climate in the III Region, located 15 km east of the Property, the average temperature difference between day and night is

[5] www.meteochile.cl

[6] Föhn winds: A föhn wind or foehn wind occurs when a deep layer of prevailing wind is forced over a mountain range. As the wind moves upslope, it expands and cools, causing water vapor to precipitate out. This dehydrated air then passes over the crest and begins to move downslope. As the wind descends to lower levels on the leeward side of the mountains, the air heats, as it comes under greater atmospheric pressure creating strong, gusty, warm and dry winds.

12°C, the average daily precipitation rate is 15 mm, and the relative humidity reaches 40%7. A SSW wind direction predominates 80% of the time, with velocities of 1.5 m/s to 4.0 m/s, averaging 2.8 m/s.Table 20.1 shows average meteorological parameters at several meteorological stations in the III Region[8].

Table 20.1: Average Meteorological Parameters in Selected Stations (II and III Region)

Copiapó (Chamonate)	(27° 18' S - 70°25' W / 291 m.a.s.l.)												
	J	F	M	A	M	J	J	A	S	O	N	D	Total
Ave. Temp. (°C)	15.1	14.9	13.9	12.1	10.3	8.7	8.6	9.4	11.2	12.7	14.1	14.8	12.2
Low Temp. (°C)	5.1	5.5	4.4	2.2	0.7	-0.5	-0.9	-0.9	0.4	1.7	2.8	3.6	2.0
High Temp. (°C)	24.1	24.1	23.6	23.0	22.1	20.6	20.9	21.5	22.6	23.7	24.2	24.4	22.9
Precip. (mm)	0.0	0.0	0.0	0.0	0.0	0.1	0.3	0.6	0.5	0.1	0.1	0.0	1.7

Source: www.atmosfera.cl

20.2.3 LOCAL RESOURCES AND INFRASTRUCTURE

Inca de Oro, Diego de Almagro, and Chañaral are small towns (populations below 20,000), which mostly provide labor for the fishing or mining industry. These towns are able to support basic needs (food, accommodations, communications, fuel, hardware, labor) for early stages of exploration. More advanced projects must be serviced from Copiapó, Antofagasta, La Serena or Santiago.

Table 20.2: Local Population

Population	
Chañaral	13,543
Diego de Almagro	18,589
Inca de Oro	900

[7] www.meteochile.cl
[8] www.atmosfera.cl

A power line linking Copiapó to Diego de Almagro is 15 km to the east of the Property. Cellular communication is possible from various high points in the vicinity of the Property.

The closest port facility is at Barquito, adjacent to the town of Chañaral, 70 km southeast of the Property, which is used by Codelco's Salvador mine for exporting its production.

In the project area there are no active streams and nor identified underground water resources. The Salado River, which drains very brackish waters from the Salar de Pedernales, contains underground waterways and was used in the past as a download for the concentrator tailings from El Salvador, is located about 84 km northof the project (Figure 21.1) Also there are some wells around Inca de Oro, which reportedly have reduced flows.

SCM Bertais the process of acquiring water rights from CODELCO´s Pampa Austral tailings dam.

20.2.4 AIR QUALITY

The activity being developediscopper mining. The location is approximately 15km from the closest populated area, Inca de Oro (Comuna deDiego de Almagro). Theproject areahas notbeen declared as latent or saturated zone referring to affected emissions of air quality standards.

Concerning the emissions generated by the project, these will be mainly characterized by the emissions of particulate matter, both in their gross and fine fraction (MP10 y MP2,5) due to truck traffic on the access road to the site of the project. Apart from the coming and going vehicle traffic of the project, we developed an estimation of the rockextraction, which in its implementation considers activities of preparing the ground, construction of the facilities, operation to extractand process the mineralised material, and finally, the closure of the project.

Therefore the impact on air quality were analysed for each of thesestages and the results indicate that the impact on the closest community is well below the legallimits.

20.2.5 SOILS

The desert soilsare characterised by a weak development profile, in which it ispossible to find mixtures of alluvial and colluvial sediments, which can be saline, presenting a coarsetexture in the surfacehorizon, brown to dark brown in colour, and without structure. Taxonomically, these soils belong to the Order of Entisols, which are soilswith a very limited development.

It is also possible to find soils with greater grade of development, evidenced by the presence of a B horizon, generally of weak expression but with structure present. Taxonomically these Aridsolsoilsshow some evidence of evolution. Their essential characteristic is a deficit of permanent or semi-permanent moisture. Due to this lack of moisture, soils of this type have excess of salt and/or sodium that can severely limit the growth of crops.

The annual average soil temperature is between 15ºC and 22ºC and the difference between the average temperature of summer soil and winter soil is more than 6ºC, at 50cm of depth or at a lithic or para-lithic contact, whichever is shallower (Soil Survey Staff, 2006).

The soils in the area of the project can be divided in two main groups: soils associated with hills, and soils not associated with hills (depositional). Thesoils present in the mine area correspond to soils associated with hills. These soils have noagricultural use, due to their stonyness, lowhumidity, poordevelopment, irregular surface relief and lack of organic matter.

Once the project has received its environmental approval, a change in land use (PermisoAmbiental Sectorial 96) needs to be applied for and granted.

20.2.6 HYDROLOGY

The Atacama Region, in contrast to more northerly basins in Chile,and due to its geological and geomorphological characteristics,corresponds to a rain-fed system.

The project is located in the QuebradaGuamangahydrographic sub basin, which is characterised by an exoreic environment with sporadic and episodic runoff.

The creeks present in the area are intermittent and observation of their courses during the field program demonstrated that they do not present a dynamic of dramatic changes in their behaviour. Rather, climatic episodes could produce minor runoff but not a significant flow.

Therefore, and as observed in the field, the creekscan be activated with concentrated rain in a short period of time, but theseevents are rare. However, theiroccurrence generates mudflows that make the soil impermeableandreduces theirfiltration capacityso that the rechargeof thewater table is minimal. Further evidence for this concentrated pluvial activity is the presence of poorly sorted alluvial gravels

Based on the climatic and hydrological data obtained in both summer and winter, the influence of precipitation and therefore, the hydrology of the area, has little importance to the development of the project

20.2.7 VEGETATION AND FLORA

Themine area presents very difficult conditions for developmentof vegetationalthoughon occasionsea mistsdo reach the area. Even so, is the obviouslack of water due to scarce precipitationcombined withintensesolar radiation, high temperatures during the day, and surface rockiness and soil characteristics results in a limited diversity of species.

In the mine area only nine species were found, mainly associated with impermanent watercourses, distinguishing 15 different vegetationalformations, described in the chart of organic lands. None of the species found in the area are ina category of conservation as defined by the SEIA. Nor are any of themarelistedin the decree Nº68/2009 as native species.

Based on the above, it is concluded that the project will not generate significant impact onthe vegetation and flora, nor is there anyrequirement to seeka special permit.

20.2.8 FAUNA

Two reptile, four birdand two mammal speciescomprisethe fauna registered inthe area during the baseline survey.Analysis of theirgeographic distribution indicates that none of these species has a distribution restricted to the Atacama Region; however, the speciesHomonotagaudichaudii is found only in the III and IV regions.

Of the 36 vertebrate species potentially present in the area, three are citedin some of the categories of threatened species by the Regulation of the Classification of Wild Species (Reglamento de Clasificación de EspeciesSilvestres - RCES, Supreme Decree MINSEGPRES y MMA).

According to the Regulation of the Hunting Law, nine potentially recordable species in the sector are in category of threat level in the north of Chile.Thereptilesare considered species of low mobility and are under the threat category; thereforea rescue plan must be implemented so that they can be relocatedprior to the construction of the project.

The study area is not located within any of the Priority Sites cited for the Atacama region.

It is recommended that measures of protection that include a ban on hunting by workers and contractors (especially those with conservation issues) be implemented, that domestic animals be restricted fromconstruction sites and that measures are taken to restrict the access of wild fauna into the project area. Additionally, there should be special attention paid to the management of household waste in order to not attract animals from outside the area.

20.2.9 CLOSURE AND ABANDONMENT STAGE

The following is a description of the project closure and abandonment stages. The purpose of this description is to provide a summary of activities necessary to completethe environmental impact assessment.

The finalstage of the project comprises the execution of a planned closure and abandonment of the mine and process plant and their associated facilities. A progressive closure should be considered (closure should be planned fromconstruction day 1). Closure includes all the surface remediation, equipment demobilization, demolition of facilities, and monitoring of the physical and chemical stability of all components (mine, dumps and abandonment areas). Abandonment refers to the status of the project area after completion of the closure activities.

It is important to highlight that the material described in this section is presented in terms of design and handling criteria, with general guidelines to provide the project with the necessary flexibility to implementthe changes inherent to the regional development and the economic status at the time in which such activities are completed.

The main objective of the closure plan is to prevent any adverse effect to either the environment or public safety once the project lifeis completed, to mitigate the environmental impacts that may occur during the closure activities and, to the greatest extent reasonably practicable, recover the site to its original condition.

Table 20.3 shows the main design specifications, closure activities and environmental management criteria for this area.

Table 20.3: Closure Activities and Environmental Management Criteria

Works	Design Specifications	Main Closure Activities	Main Environmental Management Criteria
Open Pit	▪ Slope of the pit in safe structural condition ▪ The perimeter will be fenced with berms of approximately 2 m height ▪ The Company will build a system to bypass the surface water runoff and avoid the entry of water to the excavation. ▪ The siteis furnished with special areas equipped as watchtower, having the required protective structures. ▪ The accesses will be closed with restricted safety fences ▪ The perimeter will have the required signposting to warn people of the area's potential hazards.	▪ If required, earthworks associated with sloping and leveling. ▪ Waste loading, hauling and placement for berm construction purposes. ▪ Sign posting and fencing. ▪ If required, the use of explosives can be considered to improve the slope stability. ▪ Inspections. ▪ Acid Drainage Control	▪ Irrigation for dust control purposes during execution of closure activities.
Heap Leaching	▪ Leached gravel will be left onthe last processed mineral material heaps.	▪ If required, earthworks associated with sloping and leveling ▪ Inspections ▪ Monitoring	▪ Dust Control during execution of closure activities.
Ponds	▪ Ponds will be abandoned with the impervious membranes and will be filled in. ▪ Prior to filling the ponds, these can be used to eliminate the process remnant solution and also to isolate hydrocarbon soils	▪ Solutions and soils polluted with hydrocarbon will be transported to the ponds ▪ If required, earthworks associated with sloping and leveling ▪ Inspections ▪ Monitoring	▪ Adequate containment of solution for ponds handling and transport

Works	Design Specifications	Main Closure Activities	Main Environmental Management Criteria
Building and main production facilities	▪ All the buildings such as crushed mineral material stockpile, shops, administration facilities and any minor building will be dismantled or demolished. ▪ Major production equipment such as tanks, crushers, conveyor belts will be dismantled and sold ▪ Concrete foundations will be covered with alluvial material.	▪ Handling of solution stocks during the last years of operation ▪ Tank drainage and cleaning ▪ Dismantling of buildings, facilities and equipment. ▪ Storage and transport of equipment, materials, and commercial value solutions ▪ Disposal of wastes and debris at the industrial waste dumps ▪ Land grading. ▪ Inspections and monitoring	▪ Containment of solution during the handling and transport. ▪ Fire risk prevention during the storage of solutions. ▪ Dust control during the earthworks for demolishing and grading purposes. ▪ Adequate practices of waste handling and disposal.
Roads	▪ The necessary roads will be maintained to access the main areas and to facilitate the monitoring activities during the abandonment stage. ▪ The unused roads will have their access blocked.	▪ Earthworks as required ▪ Inspections and monitoring.	▪ Irrigation for dust control purposes.

Works	Design Specifications	Main Closure Activities	Main Environmental Management Criteria
Industrial Solid wastes	▪ Once the infrastructure is removed the salvage and special waste handling yards will be dismantled and the related materials will be sold or sent to authorized disposal sites and the perimeter fencing removed. ▪ Foundations will be covered and the ground will be graded. ▪ Soil will be assessed for potential contamination and if necessary, both the salvage and special waste handling yards will be rehabilitated. ▪ Closure of the special waste deposit shall comply with the applicable rules and legislation.	▪ Handling of material stocks during the last years of operation. ▪ Inspections and monitoring	▪ Suitable waste handling and classification ▪ Dust control during the filling and grading procedures.
Domestic wastes	▪ Once the mine closure tasks are completed no more wastes will be disposed at the sanitary landfill. ▪ The sanitary landfill closure must observe the applicable rules and legislation.	▪ Landfill final covering. ▪ Fencing will be removed. ▪ Restoration, revegetation	▪ Dust and leach control

21.0 CAPITAL AND OPERATING COST

21.1 CAPITAL COST ESTIMATE

21.1.1 SUMMARY

The Berta project contemplates an open pit mine to extract oxide material from the Berta Sur and Central deposits using contractmining, followed by crushing, agglomeration and heap leaching of higher grade (>0.3%CuT) material and dump leaching of lower grade (0.1-0.3%CuT) material. Overall material contained in the mine plan developed by Geoinvestments has 7.22 mt of heap leach material, with an average grade of 0.574% CuT and 6.63 mt of dump leach material ("ROM") with an average grade of 0.20%CuT. A total of twelve 2m column tests have been completed on material from Berta Sur at Geomet SA and three 2m columns from Berta Central material at the Hydrometallurgy Laboratory of the University of Santiago de Chile, and this testwork was used to estimate recoveries of 78% of total copper for the heap leach and 45% of total copper for the ROM.

The Amended updated mine plan assumes using a variable cut-off grade in year 1 of between 0.60% and 0.70%CuT, in order to maintain a constant feed to the Nora crusher for a period of 11 months, thus postponing part of the capital investment until year 2 of operations. A total of 0.4mt at 0.83%Cu will be mined and trucked to the Nora plant while 1.2mt of lower grade heap leach material and 0.6mt of ROM will be stockpiled for processing in year 2. In addition, the Nora plant will reprocess some spent mineral material stockpiles ("Ripios") from the previous 2009-12 operation at a rate of ~30 tpm of copper cathode during Phase 1.

The project will have a mine life of 8 years producing 37,821 tonnes of copper cathode.

The capital cost estimate for the mining operation, process plant, purchase of the Nora SXEW plant, and project infrastructure has a base date of 1st Quarter 2015, is expressed in United States dollars and is estimated to an accuracy of ± 30% as shown in Table 21.1 Also the SCM Berta is adopting a phased build out of the project to utilize cash flow from the project and minimize financing costs. Phase 1 capital costs are $7.15M which includes the purchase of the existing Nora plant, remediation and refurbishment costs and initial start-up and working capital requirements. Phase 2$12.9Mwhich includes the installation of the Berta crusher, pads and site facilities; the expansion of the Nora plant to 5ktpy of cathode; and the installation of PLS and water pipelines between Nora and Berta.

Apart from $23M of Phase 1 and 2, it was Also considered $2.25M for option payments, $2.2 for closure plant and $1.3 for sustaining capital over the project LOM. Table 21.4 shows a detailed total capital expenditure year by year.

Table 21.1: Capital Cost Estimate Summary

Area No	AREA TITLE	TOTAL $´000	PHASE 1 $´000	PHASE 2 $´000	LOM $´000
10	**NORA PLANT PURCHASE & STARTUP**	**5.761**	**6.467**	**219**	**- 925**
	Startup Costs	746	670	77	-
	Working Capital Requirements	-	925		- 925
	Remediation and Refurbishment	551	551	-	-
	Met and Additional Drilling	359	359	-	-
	NORA Purchase	3.629	3.629	-	-
	First Fill	476	333	143	-
20	**BERTA CONSTRUCTION**	**6.375**		**6.375**	**-**
21	**Mineral reception**	**116**		**116**	**-**
	construction Platform	81		81	-
	civil works	35		35	-
22	**Crushing**	**2.065**		**2.065**	**-**
	purchase equipment	1.294		1.294	-
	Civil Construction Works	386		386	-
	mounting equipment	193		193	-
	electrical assembly	135		135	-
	Commissioning	58		58	-
24	**Agglomeratation**	**804**		**804**	**-**
	purchase equipment	384		384	-
	Civil Construction Works	267		267	-
	mounting equipment	76		76	-
	electrical assembly	53		53	-
	Commissioning	23		23	-
26	**Leaching**	**1.746**		**1.746**	**-**
	Purchase equipment and materials	288		288	-
	Earthmoving	686		686	-
	Liner	377		377	-
	construction Pools	343		343	-
	Piping installation	51		51	-
27	**On Site Infrastructure**	**1.188**		**1.188**	**-**
	Access Roads	135		135	-
	offices	62		62	-
	ponds	639		639	-
	generators	352		352	-
28	**Indirect Construction**	**455**		**455**	**-**
	Admin Building	173		173	-
	Freight and Insurance	24		24	-
	Parts PEM and Capital	54		54	-
	Commissioning and PEM (Engineering)	105		105	-
	Contingencies	100		100	-
30	**NORA EXPANSION**	**1.324**		**1.324**	**-**
	Plant expansion	1.000		1.000	-
	Start up	224		224	-
	Contingencies	100		100	-

40	**PIPELINE PLS & RAFF/WATER**	**3.773**	**107**	**3.666**	**-**
	Engineering	69	69		
	Environmental processing	58	38	20	
	Pipes and Piping	1.190		1.190	-
	Equipment Supply	80		80	-
	Temporary facilities	73		73	-
	Pipeline Construction	1.696		1.696	-
	Construction Pumping Stations	148		148	-
	Admin Building	275		275	-
	Startup	76		76	-
	Contingencies	109		109	-
50	**OTHER OWNER COST**	**5.807**	**574**	**1.319**	**3.914**
	Sustaining Capex	1.308		163	1.144
	Option Payments	2.299	574	1.156	570
	Closure Costs	2.200		-	2.200
	GRAND TOTAL	**23.040**	**7.148**	**12.903**	**2.989**

In addition to the $18M included in the above table, $2.5M in sustaining and closure capital and $2.25m in option payments are includedin the financial model, bringing the total capital investment in the project to $23M.

21.1.2 MINING CAPITAL COST ESTIMATE

The exploitation of the deposit will be performed by mining contractor in an all inclusive contract basis, so no capital expenditure is considered in this estimation.

Other mine capital cost will be addressed as the following items:

- Mine offices (included in Berta infrastructure capital cost);
- Maintenance shop (provided by mine contractor);
- Explosives magazine and supply (provided by explosives contractor);
- Fuel storage and supply (provided by fuel supplier);
- Road between the maintenance shop and the mine (included in Berta infrastructure capital cost);

Also as per the mine plan, no pre-stripping initial capital is considered.

21.1.3 PROCESS PLANT AND INFRASTRUCTURE CAPITAL COST ESTIMATE

A. CAPITAL COST ESTIMATE SECTIONS

The capital cost for the mine, process plant and infrastructure estimate was assembled in accordance with a breakdown into the following facilities:

Process Plant

The estimate allows for the crushing plant at the mine site; and agglomerator, on-pad stacking system, leaching pad and ponds at Berta site, and the cost of acquiring and expanding the Nora SXEW plant.

Utilities and Reagents

Utilities and reagent systems include most of systems that are already included in Nora SXEW Plant such as: compressed air, raw and firewater distribution, potable water treatment and distribution, sewerage treatment, hot water, process water storage and distribution, diluent storage and distribution and reagent storage, mixing and distribution,offices and amenities, laboratory facilities, workshop facilities, warehouse and stores facilities.

Infrastructure

Infrastructure costs include site development of roads, surface water management, power distribution and information systems and internal communications.

Indirect Costs

The estimate allows for Engineering, Procurement, Construction and Management ("EPCM") costs, temporary construction facilities and services, construction equipment hire, freight and logistics and vendor installation and commissioning representatives.

Owners Costs

The owner’s costs allowed for pre-production operating costssuch as:mining option remain payments, first fill, working capital, head office overheads, licence fees, insurances, recruitment costs and operator training expense.

These facilities are further broken down into areas, as may be seen in Table 21.1: Capital Cost Estimate Summary

B. CAPITAL COST ESTIMATE PARAMETERS

Exchange Rates

The estimate is expressed in United States (US) dollars. Conversion of all quoted foreign currencies was based upon the foreign exchange rates as at the estimate base date. These are shown in Table 21.2

Table 21.2: Capital Cost Foreign Exchange Rate Summary

Country	Unit of Currency	Exchange Rate
Chile	Peso	615

Escalation

Capital costs have a base date of 1st Quarter 2015. No allowance was made for escalation during the construction phase of the project.

Working Capital

The estimate includes an allowance for working capital that includes operating costs for two months of operation. Copper sales are made in advance.

Deferred and Sustaining Capital

Deferred and sustaining capital including rehabilitation costs were estimated .

Project Contingency

Project contingency provides for the risk of changes in scope or reasonable expectations embedded in the estimate - changes which arise from outside or unpredictable circumstances.

These include, but are not limited to, exchange rate variations from the estimate basis, escalation on materials and equipment, escalation of field construction labor costs above the base line escalation of Q12015, abnormalities in industrial relations, market conditions, inclement weather, or adverse political or regulatory developments.

Project contingency as defined above was estimated by SCMB at $200,000.

Accuracy Provision

The component estimates were developed at bare cost (excluding accuracy provision).

An accuracy provision allowance was then allocated to each area and element of the direct and indirect costs to reflect the level of definition available in the scope of work.

The purpose of the accuracy provisions is to make allowance for uncertain elements of costs to cover such factors as:

- Accuracy of material and labour rates;
- Accuracy of productivity expectations;
- Accuracy of equipment budget pricing;
- Lack of direct knowledge of local contractor capabilities; and
- Lack of direct knowledge of local permitting methods, procedures and outcomes.

The sum of the estimated bare costs and accuracy provisions is the total estimated cost for the project.

C. CAPITAL COST ESTIMATE STRUCTURE

The capital cost estimate was structured into the following major categories - direct costs and indirect costs.

Direct costs included:

- Supply of permanent equipment and materials and fixed and mobile equipment;
- Labour to undertake and manage the construction, installation and service activities. This includes wages and salaries with loadings for site labour,

supervision and management, including associated expenses such as accommodation, messing and travel, and home and/or satellite office management expenses;

- Supply of construction support facilities and services such as water, site roads, lay down areas and including maintenance of these facilities;
- Contractors and suppliers mark-up and profit; and
- Freight and shipping expenses for permanent and temporary equipment and materials.

Indirect costs included:

- EPCM services, together with construction supervision and commissioning of the plant;
- Supply of temporary equipment, materials, consumables and small tools; and
- Initial capital spares for permanent equipment and first fill of reagents and consumables.

Local Supply, Construction Rates and Capabilities:

Pricing was established based on local contractors for earthworks, concrete, and structural-mechanical-piping (SMP) works.

The labor rates used are considered an “all-in” rate expressed in US Dollars and include:

- Base labor rate, payroll burdens, overtime premiums;
- Meals and accommodation for all supervision and labor;
- Small tools and consumables, including safety consumables;
- Contractor supervision and overheads;
- Construction equipment;
- Contractor’s temporary facilities;

- Contractors insurances; and
- Mark up and profit.

21.1.4 FIRST FILLS

The quantities for first fill reagents, consumables, fuels and lubricants were estimated according to process requirements. Pricing is based on quotations from several suppliers.

Wear steel for the crushing plant is assumed to be available as consignment stock in the warehouse.

Table 21.3: First Fill Estimate

Summarizes the first fill costs

Table 21.3: First Fill Estimate

First Fill	Qty	Price delivered $/t	Cost $	Source/Comment
Sulfuric Acid	1,000	94	94,000	initial stock
Extractant (t)	20	12,000	240,.000	Inc´Nora, so reposition
Diluent (t)	7	1,390	9,730	Inc´Nora, so reposition
Diesel Fuel (t)	45	700	31,500	tankage
Cobalt Sulfate (t)	1	4,200	4,200	4 week's use
Laboratory consumables (lot)	1	91,000	91,000	initial stock
Guar (t)	1	6,000	6,000	4 week's use
TOTAL First Fill			**476,430**	

21.1.5 DEFERRED AND SUSTAINING CAPITAL COSTS

Deferred capital costs are those that can be delayed until after the commencement of operations.

Sustaining capital is defined as the annual capital required for replacing equipment items that have served their useful life.

A. DEFERRED CAPITAL COSTS

No deferred costs are considered in the project.

B. SUSTAINING CAPITAL COSTS

Average annual sustaining capital costs were estimated at $163,000 per year based on consultant experience in this project size.

C. CLOSURE COSTS

The pit closure will include:

- The slopes of the pit walls will be assessed and adjusted if necessary to attain safe long-term structural conditions;
- Placement of a 2 m high berm around the pit perimeter to limit access and to reduce erosion;
- Construction of drainage diversion channels to direct surface run-off away from the pit and to prevent it from entering the pit;
- A watchtower will be installed to allow safe monitoring of the pit after closure; and
- Access to the pit will be limited by safety fencing and safety signs will warn of potential hazards in the area.

All costs for waste dump construction are included in the mine operating costs for the project. Deposits of sterile and waste dumps will remain in the area at mine closure.

The slopes of the waste dumps will be assessed, angles of repose adjusted if necessary and additional measures put in place to attain a long-term safe condition.

Material will be left in place on the last processed heaps and the process ponds will be abandoned with the impervious membranes left in place and the ponds filled in.

The plant, equipment and infrastructure will be decommissioned and offered for sale. The cost of relocation will be covered by the purchaser. Concrete foundations will be covered with alluvial material.

Electric overhead lines including power poles and cable will be dismantled and sold.

Haul roads and mine site access roads will be left in place to allow ongoing access for monitoring activities when the mine site is abandoned. Unused roads will be blocked off.

Once the process plant and infrastructure has been removed the salvage and special waste handling yards will be dismantled and salvage material and special wastes sold or disposed of in authorized landfills. Closure of the special waste handling yard will comply with statutory requirements.

Soils in these areas will be assessed for potential contamination and if necessary both the salvage and special waste handling yards will be rehabilitated.

Perimeter fences will be removed, foundations covered and the ground graded.

Once the mine closure tasks are completed no more waste will be disposed of at the sanitary landfill. Closure of the sanitary landfill will also comply with statutory requirements.

Once the mine closure has been completed in year 8, a 2 year monitoring and inspection program will be put in place.

Total plant closure costs were estimated by SCMB at $2.2M including contingency.

This included:

- Adjusting the final bank grade of the waste dumps for long term stability;
- Decommissioning and filling in of the process ponds;
- Dismantling and demolition as required of the process plant and associated infrastructure;
- Dismantling the overhead electric power lines; and
- Dismantling and rehabilitation of the salvage and special waste handing yards and rehabilitation of these areas in accordance with statutory requirements.

Table 21.4 shows the total capex for the 8 years LOM.

Table 21.4: Total Capital Expenditure

Area No	AREA TITLE	TOTAL $´000	Yr1 $´000	Yr2 $´000	Yr3 $´000	Yr4 $´000	Yr5 $´000	Yr6 $´000	Yr7 $´000	Yr8 $´000
10	**NORA PLANT PURCHASE & STARTUP**	**5,761**	6,467	219						-925
20	**BERTA CONSTRUCTION**	**6,375**		6,375						
30	**NORA EXPANSION**	**1,324**		1,324						
40	**PIPELINE PLS & RAFF/WATER**	**3,773**	107	3,666						
50	**Other Owner Cost**									
	Sustaining Capex	1,308	163	163	163	163	163	163	163	163
	Option Payments	2,300	574	1,156	570					
	Closure Costs	2,200								2,200
	GRAND TOTAL	**23,040**	**7,311**	**12,903**	**733**	**163**	**163**	**163**	**163**	**1,438**

21.2 OPERATING COST ESTIMATE

21.2.1 SUMMARY

The operating costs for the Berta Project were estimated in US dollars to an accuracy of +/-30% and the base date for the estimate was the 1st Quarter 2015. The operating costs were broken into the following cost centres:

- Mine;
- Processing; and
- General and administration.

The estimated mining, processing and general and administration (G&A) operating costs, the mining schedule and the expected annual copper production over the life of the mine are shown in Table 21.5. The total operating cost is shown in dollars per tonne of oxide material processed ($/t) and converted into cents per pound of copper cathode produced (c/lb copper)

Table 21.5: Total Cash Cost (C1) and Production Summary

OPERATIONAL SUMMARY		ANNUAL								
Annual		**Yr1**	**Yr2**	**Yr3**	**Yr4**	**Yr5**	**Yr6**	**Yr7**	**Yr8**	**TOTAL**
Mine										
Nore Direct Ore		399	-	-	-	-	-	-	-	399
Heap Leach Ore	kt	1.175	645	1.225	693	604	592	1.115	713	6.763
Dump Leach Ore	kt	609	1.538	1.163	976	598	490	471	604	6.449
Waste Total	kt	270	278	66	786	1.252	1.378	868	395	5.293
Total Mined	kt	2.454	2.461	2.454	2.454	2.454	2.461	2.454	1.712	18.904
Strip Ratio waste/(Ore+Dump)		0,2	0,1	0,0	0,5	1,0	1,3	0,5	0,3	0,4
NORA Leach										
Treatment	kt	399	-	-	-	-	-	-	-	399
Total Copper Grade	%	0,76	-	-	-	-	-	-	-	-
Copper Recovery	% CuT	0,56	-	-	-	-	-	-	-	
Total Acid Consumption	Kg/t	22	22	22	22	22	22	22	22	
RIPIOS Line										
Treatment	kt	345	-	-	-	-	-	-	-	345
Total Copper Grade	%	0,20	-	-	-	-	-	-	-	
Copper Recovery	% CuT	50	-	-	-	-	-	-	-	
Heap Leach										
Treatment	kt	85	1.003	1.000	1.000	1.000	847	1.000	829	6.763
Total Copper Grade	%	0,55	0,51	0,55	0,50	0,51	0,75	0,61	0,48	
Copper Recovery	% CuT	7	78	78	77	77	78	79	77	
Total Acid Consumption	Kg/t	22	22	22	22	22	22	22	22	
Dump Leach										
Treatment	kt	109	1.538	1.163	976	598	490	471	604	5.949
Total Copper Grade	%	0,22	0,20	0,21	0,19	0,19	0,18	0,21	0,20	
Copper Recovery	% CuT	45	45	45	45	45	45	45	45	
Total Acid Consumption	Kg/t	12	12	12	12	12	12	12	12	
Production										
Nora Direct Production	t	2.673	-	-	-	-	-	-	-	2.673
Ripios Line	t Cathodes	315	-	-	-	-	-	-	-	315
Heap Production	t Cathodes	366	4.027	4.271	3.839	3.947	5.244	4.783	3.075	29.553
Dump Production	t Cathodes	90	1.387	1.101	812	501	408	440	541	5.280
Payable Production	**t Cathodes**	**3.444**	**5.414**	**5.372**	**4.651**	**4.448**	**5.652**	**5.223**	**3.615**	**37.821**
Payable Production	**M lb**	**7,6**	**11,9**	**11,8**	**10,3**	**9,8**	**12,5**	**11,5**	**8,0**	**83,4**
Prices										
Copper Price	us$/lb	3	246	114	130	146	162	178	194	1.173
Copper Price	us$/lb	2,6	2,6	2,6	2,6	2,6	2,6	2,6	2,6	
Revenues										
Copper Sales	Mus$	19,7	31,0	30,8	26,7	25,5	32,4	29,9	20,7	216,8
Credit	Mus$	- 0,0	- 0,0	- 0,0	- 0,0	- 0,0	- 0,0	- 0,0	- 0,0	- 0,1
Total Revenue	**Mus$**	**19,7**	**31,0**	**30,8**	**26,6**	**25,5**	**32,4**	**29,9**	**20,7**	**216,6**
Operating Cost										
Mining	Mus$	3,2	6,2	4,9	6,0	6,2	5,9	4,8	4,3	41,4
	$/lb	0,43	0,52	0,41	0,58	0,63	0,47	0,42	0,54	0,50
Processing	Mus$	8,2	9,9	9,2	8,9	8,9	8,6	9,7	8,9	72,4
	$/lb	1,08	0,83	0,78	0,87	0,90	0,69	0,85	1,12	0,87
ROM_RIPIOS Cost	Mus$	0,9	2,2	1,7	1,4	0,9	0,8	0,8	0,9	9,6
	$/lb	0,11	0,19	0,14	0,14	0,09	0,06	0,07	0,12	0,12
G&A	Mus$	1,3	1,4	1,1	1,0	1,0	1,1	1,1	1,0	8,9
	$/lb	0,17	0,12	0,09	0,10	0,10	0,09	0,09	0,12	0,11
Cash Cost C1	**Mus$**	**13,6**	**19,7**	**16,8**	**17,4**	**17,0**	**16,4**	**16,3**	**15,1**	**132,3**
	$/lb	**1,79**	**1,65**	**1,42**	**1,69**	**1,73**	**1,31**	**1,42**	**1,90**	**1,59**

The mining cost centre includes all costs associated with contractor operated mining of oxide material from the Berta Sur and Central pits.

The processing cost centre includes all costs associated with the production of up to 5,000 t/y of copper cathode including crushing, agglomeration, stacking on a heap leach pad, leaching of up to 1 Mt/y ore; and the recovery of copper metal by the SX-EW process in Nora Plant, and total transport of water, PLS and Raffinate by pipeline from Nora to Berta.

All common administration and support services not attributable to mining or processing have been collected in the General and Administration cost centre. This includes general management, accounting, procurement, purchasing and warehousing, security, safety and training.

Mining costs make up 31% and processing represent 62% of the total unit cost of $1,50/lb of copper. General and administration costs remain essentially constant over the life of the operation and represent approximately 7% of the total unit cost.

The mining cost centre is dominated by mining operational contract that considers drilling, blasting, loading, hauling and labor totaling more than 90% of the mine cost.

Sulfuric acid (16%) and power (13%) are the largest processing operating costs.

Services including product transportation, freight and personnel costs (accommodation, catering and R&R transport) are the major components of the general and administration costs.

The operating cost estimate excludes currency and commodity price fluctuations and duties and taxes.

21.2.2 BASIS OF OPERATING COST ESTIMATE

A. BASIS OF ESTIMATE – MINING

Information included in the model was obtained from various sources including equipment suppliers, similar operating mines, mining contractors and empirical calculations.

Mine operating costs were calculated annually over the 8 year life of mine. The basis used in developing these costs were the required equipment operating hours, the unit rates applied to the different types of equipment, personnel requirements and unit costs for materials, services and labor.

The mining costs include all mining operations required to feed oxide material to the primary crusher and the removal and haulage of waste to the disposal site. All costs associated with operation and maintenance of the mine is included. Contractors have been allowed for blasting operations and mining equipment maintenance.

B. BASIS OF ESTIMATE – PROCESSING

A detailed operating cost model was used to estimate costs incurred in the process plant, administration and the supporting services and infrastructure required for the operation.

Annual operating costs have been estimated for reagents, labor, power, contractors, consumables and miscellaneous items. The life of mine production schedule was applied to unit costs on a $/t oxide materialand $/t copper basis where appropriate to calculate annual operating costs.

Operating cost estimates are based on the operating plans prepared for the processing functions. The oxide material processing schedule and the process design criteria provided the basis for the estimation of reagent, power, contractor and consumable costs. The estimates were derived for each of the cost centers using the methods detailed below.

Reagent unit costs were based on supplier quotations including freight to site.

Fuel costs were based on $0,71/l for diesel delivered to site as it was quoted for similar mining companies in the area.

Operating and maintenance consumable costs were based on supplier quotations and information derived from Propipe's database and experience with similar heap leach–SX–EW operations.

C. BASIS OF ESTIMATE – G&A

The general and administration (G&A) costs include all costs not associated with either mining or processing and these were provided by SCM Berta. The G&A operating costs include labor, vehicles and other administration costs.

Labor on-costs have been allowed for at 20% of base annualized salaries. Other personnel costs including R&R travel, clothing and accommodation and messing are collected under "Other Costs" in the G&A estimate.

21.2.3 MINING OPERATING COSTS

A. GENERAL

The mining operating cost estimate was based for a maximum total of 7,500 t/d, for 365 days per year.

Mine operating costs were calculated annually for all the mine life periodson all inclusive mine service contract.

The basis used in developing these costs were the required equipment operating hours, the unit rates applied to the different types of equipment, personnel requirements, and unit costs for materials, services and labor. Also a 10% profit contract has been allowed in the Mine G&A cost item.

Other costs, such as water, mine and labour for managing the mine contract has been included under Other Owner Costs item.

A summary by year for mining operating costs is presented in Table 21.6

Table 21.6: Summary of Mine Operating Cost

Berta 5Kt Copper Leach Project MINING OPERATING COST SUMMARY		Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	COST
DRILLING	$k	165	130	112	99	132	95	98	104	935
BLASTING	$k	439	440	448	416	396	391	413	297	3.241
LOADING	$k	678	510	437	384	539	391	396	415	3.751
HAULING	$k	1.414	1.177	1.214	1.122	2.370	1.999	1.728	1.679	12.702
AUXILIARY EQUIPMENT	$k	576	459	399	354	574	444	411	413	3.631
	$/t material	0,26	0,21	0,17	0,21	0,48	0,41	0,26	0,31	0,27
	c/lb	5,34	4,21	3,66	3,25	5,41	4,07	3,80	4,77	4,30
	$/ton mined	0,23	0,19	0,16	0,14	0,23	0,18	0,17	0,24	0,19
LABOUR	$k	1.093	1.334	1.649	1.945	878	1.136	1.403	1.052	10.490
	$/t material	0,50	0,61	0,69	1,17	0,73	1,05	0,88	0,80	0,77
	c/lb	10,12	12,22	15,12	17,84	8,28	10,42	12,98	12,15	12,41
	$/ton mined	0,45	0,54	0,67	0,79	0,36	0,46	0,57	0,61	0,53
MINE G&A incl´ Contractor profit	$k	614	615	614	614	614	615	614	428	4.726
	$/t material	0,28	0,28	0,26	0,37	0,51	0,57	0,39	0,33	0,35
	c/lb	5,68	5,64	5,63	5,63	5,78	5,65	5,67	4,95	5,59
	$/ton mined	0,25	0,25	0,25	0,25	0,25	0,25	0,25	0,25	0,25
TOTAL Mine Service Contract	$k	4.979	4.666	4.873	4.933	5.502	5.070	5.064	4.388	39.475
	$/t material	2,28	2,14	2,04	2,96	4,58	4,68	3,19	3,33	2,90
	$/lb	0,46	0,43	0,45	0,45	0,52	0,47	0,47	0,51	0,47
	$/ton mined	2,03	1,90	1,99	2,01	2,24	2,06	2,06	2,56	2,09
Owner Mining Costs	$k	670	511	524	527	561	535	535	495	4.359
	$/t material	0,31	0,23	0,22	0,32	0,47	0,49	0,34	0,38	0,32
	c/lb	0,06	0,05	0,05	0,05	0,05	0,05	0,05	0,06	51,58
	$/ton mined	0,27	0,21	0,21	0,21	0,23	0,22	0,22	0,29	0,23
TOTAL MINING COSTS	$k	5.649	5.177	5.397	5.461	6.064	5.605	5.599	4.883	43.834
	$/ton mined	2,30	2,10	2,20	2,23	2,47	2,28	2,28	2,85	2,32

B. BASIS AND CONSUMPTION

Fuel

Fuel costs correspond to prices at the mine site, and were estimated at $0.7/l. Consumption for each item of equipment was based upon information from vendors and similar cases as shown in Table 21.7

Trucks fuel consumption represents an average given the characteristics of the hauling profiles.

Table 21.7: Fuel Consumption

Equipment Item	Consumption l/h
Drill rig	20
Trucks	67
Front end loader	80
Track dozer	53
Wheel dozer	41
Motor grader	25
Water truck	41

Operations Supplies & Consumables

Operations supplies and consumables include drilling steels (bits, rods, and stabilizers), tires for trucks and auxiliary equipment; wear parts for loading and auxiliary equipment, etc. These items are described separately for each one of the different units of equipment.

C. EQUIPMENT HOURLY COST

Table 21.8, shown Steel Consumption Cost

Table 21.9: Lubricants Consumption Cost

And Table 21.10 shown provide a summary of the hourly costs of the main mine equipment.

Table 21.8: Steel Consumption Cost

Equipment Item	Steel Consumption $/h
Drill rig	12
Trucks	3
Front end loader	1.5
Track dozer	0.5
Wheel dozer	0.5
Motor grader	0.5

Table 21.9: Lubricants Consumption Cost

Equipment Item	Lubricants Consumption $/h
Drill rig	2.63
Trucks	5.13
Front end loader	5.86
Track dozer	2.00
Wheel dozer	2.37
Motor grader	1.79

Table 21.10: Tire Consumption Cost

Equipment Item	Tire Consumption $/h
Trucks	1,00
Front end loader	2,00
Wheel dozer	0,73
Motor grader	0,73
Water truck	1,00

A summary of the mine consumptioncost is presented in Table 21.11

Table 21.11: Mine Fleet Consumption Costs

Berta 5Kt Copper Leach Project										
Mine Consumables & Material Costs		Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Drilling	KUS$	165	130	112	99	132	95	98	104	935
Combustible	KUS$	78	60	50	44	61	42	44	46	**426**
Lubricants	KUS$	14	11	9	8	11	8	8	9	**79**
Steels	KUS$	60	46	39	34	47	32	34	36	**326**
Maintenance	KUS$	13	13	13	13	13	12	13	13	**104**
LOADING	KUS$	678	510	437	384	539	391	396	415	3751
Combustible	KUS$	561	415	348	302	436	307	309	324	**3003**
Lubricants	KUS$	58	43	36	31	45	32	32	33	**310**
Steels	KUS$	15	11	9	8	12	8	8	9	**79**
Tires	KUS$	20	15	12	11	15	11	11	11	**106**
Maintenance	KUS$	25	26	32	32	32	34	36	37	**253**
HAULING	KUS$	1414	1177	1214	1122	2370	1999	1728	1679	12702
Combustible	KUS$	1105	908	941	865	1914	1603	1378	1337	**10050**
Lubricants	KUS$	119	98	101	93	206	173	149	144	**1083**
Steels	KUS$	70	57	59	55	121	101	87	84	**634**
Tires	KUS$	23	19	20	18	40	34	29	28	**211**
Maintenance	KUS$	97	95	93	91	88	88	85	86	**723**
SUPPORT FLEET EQUIPMENT	KUS$	576	459	399	354	574	444	411	413	3631
Combustible	KUS$	493	378	323	281	485	363	333	334	**2990**
Lubricants	KUS$	21	20	18	16	26	19	18	19	**157**
Steels	KUS$	5	4	4	4	7	6	5	5	**40**
Tires	KUS$	4	4	4	4	6	5	5	5	**37**
Maintenance	KUS$	53	52	50	50	50	50	50	50	**408**

Explosives Magazine

An estimate of the total explosives requirements is shown in Table 21.12

Table 21.12: Explosives Consumption

Explosives consumption (g/t)	186
Total rock moved (kt/year)	2,350
Explosives (kg/year)	436.7
Explosives (kg/month)	36.4
Explosives (kg/week)	9.1

D. BLASTING

Blasting costs were based upon the blasting parameters presented in the corresponding section. Explosives considered were Anfo, and emulsion for wet holes that has been estimated as 30% of the total.

Explosive prices were $650/t of Anfo and $630/t of emulsion. With these parameters the average explosive cost results as shown in Table 21.13: Explosives Cost

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Table 21.13: Explosives Cost

BLASTING	Unit	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Oxide Material Blasting	t	2183599	2183017	2387818	1668429	1202508	1082895	1585977	1316954
ANFO	60%	0,102	0,102	0,102	0,102	0,102	0,102	0,102	0,102
EMULSION	40%	0,066	0,066	0,066	0,066	0,066	0,066	0,066	0,066
ACCESORIES	10%	0,017	0,017	0,017	0,017	0,017	0,017	0,017	0,017
TOTAL EXPLOSIVES COST	US$/t	0,184	0,184	0,184	0,184	0,184	0,184	0,184	0,184
TOTAL EXPLOSIVES COST	US$	401700	401593	439269	306928	221216	199212	291760	242270
Waste Blasting	t	270428	277732	66208	785597	1251518	1377855	868050	395272
ANFO	60%	0,077	0,077	0,077	0,077	0,077	0,077	0,077	0,077
EMULSION	40%	0,050	0,050	0,050	0,050	0,050	0,050	0,050	0,050
ACCESORIES	10%	0,013	0,013	0,013	0,013	0,013	0,013	0,013	0,013
TOTAL EXPLOSIVES COST	US$/t	0,139	0,139	0,139	0,139	0,139	0,139	0,139	0,139
TOTAL EXPLOSIVES COST	US$	37683	38700	9226	109469	174392	191996	120958	55079
TOTAL Oxide + Waste	**US$**	**439383**	**440294**	**448495**	**416397**	**395609**	**391208**	**412718**	**297349**
TOTAL	**US$/t**	**0,179**	**0,179**	**0,183**	**0,170**	**0,161**	**0,159**	**0,168**	**0,174**

E. MINE G&A COSTS

Mine General and Administrative costs include all the items related with mine personnel (camp, transport, etc.) that have been estimated at $0.03/t of mined material. Also 10%, or $0.22/t, has been allowed as contractor profit.

F. MANPOWER COSTS

Manpower costs were estimated separated from the corresponding to the mine contractor unit operations, according to the mine labor estimates that are described in the next section, applying company cost per person were based on experience from similar mining projects in Chile.

A summary of the labor costs per period of the mine plan is shown in Table 21.14

A. OWNER MINE COST

Owner mine costs were considered for considering the administration of the mining contract and the cost of supplying water to the mine site including the transport.

A summary of the labor costs per period of the mine plan is shown in Table 21.15

Table 21.14: Mining Labor Costs

Berta 5kt/y Copper Leach Project Mining Contract Labor	Unit	Labor Cost US$	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
MINE CONTRACT LABOR										
CONTRACT ADMINISTRATOR	ea	50.182	1	1	1	1	1	1	1	1
SHIFT BOSS	ea	50.182	4	4	4	4	4	4	4	4
OPERATORS	ea	18.491	56	56	56	56	56	56	56	56
MAINTENANCE										
HEAD OF DEPARTMENT	ea	50.182	1	1	1	1	1	1	1	1
THECNICIANS	ea	18.491	2	2	2	2	2	2	2	2
TOTAL	ea		**64**	64	64	64	64	64	64	64
LABOR COST	**KUS$**		**1374**	**1374**	**1374**	**1374**	**1374**	**1374**	**1374**	**1374**
LABOR COST	**KUS$**		**1.374**	**1.374**	**1.374**	**1.374**	**1.374**	**1.374**	**1.374**	**1.374**
	US$/t		**0,44**	**0,54**	**0,67**	**0,79**	**0,35**	**0,46**	**0,57**	**0,61**

Table 21.15: Owner Mining Costs

Berta Owner Mine Cost	Unit	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Mine Water Compsumption	m3/d	119	98	79	67	149	115	93	87
	m3/t Mined	0,014	0,014	0,014	0,014	0,014	0,014	0,014	0,014
	m3/y	43.546	35.781	28.871	24.471	54.211	42.030	33.917	31.581
Water Cost ($0,0/m3)	KUS$								
Water Transport Cost	KUS$	156	24	24	24	24	24	24	24
Total Mine water Cost	**KUS$**	**156**	**24**	**24**	**24**	**24**	**24**	**24**	**24**
Owner Mine Labor									
Contract Manager & Administrator	ea	2	2	2	2	2	2	2	2
Labor Cost	**KUS$**	**194**	**194**	**194**	**194**	**194**	**194**	**194**	**194**
TOTAL Cost		**350**	**218**	**218**	**218**	**218**	**218**	**218**	**218**
		0,14	**0,09**	**0,09**	**0,09**	**0,09**	**0,09**	**0,09**	**0,13**

B. COST PER UNIT OPERATION

The cost per unit operation and period of the plan was calculated using the above described parameters. A summary of these estimates is shown in Table 21.16.

Table 21.16: Cost per unit operation

Berta 5Kt Copper Leach Project MINING OPERATING COST SUMMARY		Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	COST
DRILLING	$k	165	130	112	99	132	95	98	104	935
BLASTING	$k	439	440	448	416	396	391	413	297	3.241
LOADING	$k	678	510	437	384	539	391	396	415	3.751
HAULING	$k	1.414	1.177	1.214	1.122	2.370	1.999	1.728	1.679	12.702
AUXILIARY EQUIPMENT	$k	576	459	399	354	574	444	411	413	3.631
LABOUR	$k	1.093	1.334	1.649	1.945	878	1.136	1.403	1.052	10.490
MINE G&A incl´ Contractor profit	$k	614	615	614	614	614	615	614	428	4.726
TOTAL Mine Service Contract	$k	4.979	4.666	4.873	4.933	5.502	5.070	5.064	4.388	39.475
Owner Mining Costs	$k	670	511	524	527	561	535	535	495	4.359
TOTAL MINING COSTS	$k	5.649	5.177	5.397	5.461	6.064	5.605	5.599	4.883	43.834
	$/ton mined	2,30	2,10	2,20	2,23	2,47	2,28	2,28	2,85	2,32

C. MINE OPERATIONS LABOR

Mine operations manpower was estimated as a function of the number of equipment items in the fleet for every year of the mine plan.

The number of operators was calculated assigning four operators to each unit. In the case of trucks and auxiliary equipment, a 0.9 factor has also been applied to account for the fleet availability. A 0.8 factor was used for auxiliary equipment.

A 15% of extra labor force was considered for absenteeism, including vacations, sickness, etc.

All the units operations will be carried out by mine contractor and a 12% profit has been charged in General & Administration costs.

21.2.4 PROCESSING OPERATING COSTS

A. SUMMARY

A summary breakdown of operating costs by area is provided in Table 21.17

Table 21.17: Process Plant Operating Cost Summary

Berta 5ktpy Copper Leach Project										
Process Opex Summary		**Yr1**	**Yr2**	**Yr3**	**Yr4**	**Yr5**	**Yr6**	**Yr7**	**Yr8**	**TOTAL**
Total Costs ($million)	**$´000**	**$ 9,06**	**$ 12,17**	**$ 10,88**	**$ 10,36**	**$ 9,79**	**$ 9,39**	**$ 10,50**	**$ 9,88**	**$ 82,03**
	$/lb	**$ 1,19**	**$ 1,02**	**$ 0,92**	**$ 1,01**	**$ 1,00**	**$ 0,75**	**$ 0,91**	**$ 1,24**	**$ 0,98**
Processing	**$´000**	**8.197**	**9.923**	**9.220**	**8.939**	**8.859**	**8.604**	**9.733**	**8.943**	**72.419**
	$/lb	**1,08**	**0,83**	**0,78**	**0,87**	**0,90**	**0,69**	**0,85**	**1,12**	**0,87**
Crushing & Agglomeration	**$´000**	**1.166**	**1.706**	**1.704**	**1.704**	**1.704**	**1.568**	**1.704**	**1.552**	**12.808**
	$/lb	**0,15**	**0,14**	**0,14**	**0,17**	**0,17**	**0,13**	**0,15**	**0,19**	**0,15**
Labor		39	39	39	39	39	39	39	39	308
Power		128	435	434	434	434	367	434	359	3.024
Materials		169	351	350	350	350	296	350	290	2.507
Maintenance		48	100	100	100	100	85	100	83	716
Operational contract		782	782	782	782	782	782	782	782	6.253
Stacking & Leaching	**$´000**	**5.391**	**5.942**	**5.257**	**5.209**	**5.195**	**4.687**	**5.247**	**4.479**	**41.407**
	$/lb	**0,71**	**0,50**	**0,44**	**0,51**	**0,53**	**0,38**	**0,46**	**0,56**	**0,50**
Labor		39	39	39	39	39	39	39	39	308
Power		255	449	449	449	449	449	449	449	3.395
Water		73	265	148	148	148	125	148	123	1.178
Sulfuric Acid		1.001	2.074	2.068	2.068	2.068	1.751	2.068	1.714	14.812
Hauling & Stacking Service		617	1.279	1.275	1.275	1.275	1.080	1.275	1.057	9.134
Operational Raw Materials		174	361	360	360	360	305	360	298	2.580
Operational Contract		471	471	471	471	471	471	471	471	3.769
Chemical Analisys Service		253	253	84	84	84	84	84	84	1.013
DIRECT - Ore Transport		2.276	0	0	0	0	0	0	0	2.276
PLS-Raffinate Transport		232	751	363	315	301	382	353	245	2.942
SX-EW	**$´000**	**1.640**	**2.275**	**2.259**	**2.026**	**1.961**	**2.349**	**2.782**	**2.912**	**18.204**
	$/lb	**0,22**	**0,19**	**0,19**	**0,20**	**0,20**	**0,19**	**0,24**	**0,37**	**0,22**
Labor		39	39	39	39	39	39	39	39	308
Power		931	1.461	1.447	1.253	1.199	1.523	1.407	974	10.195
Reagents		183	288	286	248	237	301	850	1.412	3.805
Operational Contract		487	487	487	487	487	487	487	487	3.896
ROM and RIPIOS	**$´000**	**867**	**2.246**	**1.661**	**1.418**	**929**	**790**	**764**	**936**	**9.611**
	$/lb	**0,11**	**0,19**	**0,14**	**0,14**	**0,09**	**0,06**	**0,07**	**0,12**	**0,12**
Water		52	227	96	80	49	40	39	50	632
Sulfuric Acid		350	1.734	1.312	1.101	675	553	531	681	6.937
Power		94	154	154	154	154	154	154	154	1.173
Materiales de Operación		370	131	99	83	51	42	40	51	868

B. REAGENTS

Reagent consumption estimates are based on:

- Column leach test work and process modeling;
- Process plant mass balance; and
- Typical operating parameters for SX-EW plants.

Budget quotes from suppliers which include freight to site have been used for major plant reagents.

Reagent unit costs and consumptions are provided in Table 21.18

Table 21.18: Reagent Unit Costs and Consumptions

Reagents	Unit Cost $/t	Comsumption	Remarks
Sulfuric Acid	94		Tariff includes transport
Extractant (LIX984N)	12,500		
Diluent (escaid 110)	1,300		
Water	0,0	0.18 m3/t Ore 0.10 m3/t ROM	Agreement with Codelco Pampa Austral Tailings Dump
Cobalt Sulfate		0.29 kg/t	Costed as 10% of Extractant and Diluent Reagent
Guar		0.21 kg/t	

Extractant consumption is based on historical performance of a number of Chilean SX-EW operations provided by a major extractant supplier.

Diluent consumption is made on the same basis as extractant consumption and this includes an allowance for evaporation losses.

Cobalt sulfate consumption is proportional to the electrolyte bleed required to maintain an iron concentration in electrolyte of 1.5 g/l and a cobalt concentration in electrolyte of 150 ppm based on an expected PLS iron concentration of about 7 g/L.

Guar consumption is based on the standard industry application rate of 0.15-0.20 kg/t of cathode copper and remains constant throughout the mine life reflecting the constant copper production of 5,000t/y.

C. LABOR

Labor costs include salary and cost charges for managerial, technical, supervisory, operating and maintenance staff at the plant.

Processing labor estimates have been developed assuming 12 hour shifts from Monday to Thursday. Annualized salaries were based on similar mining projects in Chile.

The mine supervision includes a mine contract manager and mine planner. The processing supervision also considers a plant contract manager aa chief metallurgist. In addition, there are 6 administrative and safety staff.

D. POWER

100% of Berta mine site power will be provided by diesel generators at 0.222 $/kWh. Nora power requirements will be supplied from the grid via a 4km - 23kV line from Diego de Almagro at 0.117$/kWh.

E. CONSUMABLES

Operating Consumables

Major operating parts and materials that will be consumed in processing include:

- Feeder liners;
- Crusher liners;
- Screen decks;
- Heap leach irrigation equipment;
- Cathode mother plates; and
- Lead anodes.

The expected costs in $/t of heap leach material of crushing circuit wear parts such as feeder liners, crusher liners, and screen decks are based on a similar heap leach operations in Chile.

The heap leach irrigation system has been designed so that it can be moved from cell to cell as required. An allowance for emitter, piping and valve replacement has been allowed for in the estimate.

Each cathode mother plate is handled by crane and stripped as part of the tank-house cathode 7 days harvesting cycle. The mother plates are sometimes damaged as a result of this handling and also due to chemical attack. Allowance has been made for this normal wear and tear and also for the replacement of damaged edge strips.

Lead anodes have lead alloy blades welded to a “steer horn” shaped copper hanger bar encapsulated in lead. Although they are not handled as often as the cathode mother plates the blades deteriorate over time due to lead oxide spalling and an allowance has been made for lead anode blade replacement each year.

Where specific information was unavailable the operating parts and materials have been factored off actual costs from a heap leach – solvent extraction – electrowinning plant of similar complexity.

Maintenance Consumables

Maintenance consumable costs comprise maintenance parts and materials and these have been estimated as a percentage of the direct installed capital cost (% factor).

The magnitude of the factors applied are based on typical industry values and the maintenance consumable costs derived by this method have been checked against an operation of the same size and complexity and found to be consistent. The following factors have been applied to each discipline.

- Wear steel, operating parts & materials, Crusher 0.4 $/t heap leach material.
- Operating parts & materials, Liners, Piping, Heap Leach 0.36 $/t heap leach material and 0,09 $/t ROM material.

F. MISCELLANEOUS

The main contract in this item refers to the service contract that provides labor, equipment and tools for operation and maintenance support.

The processing operations group at Berta mine site will consist of 39 operators. Each crushing, agglomeration, stacking and heap irrigation installation, monitoring and maintenance will be undertaken by a crew of 10 on each shift. The SX-EW at Nora Plant including cathode stripping will be operated by a crew of 5 on each shift.

The maintenance group will consist of 15 personnel including 4 maintenance supervisors, 8 mechanical tradesmen and 3 electrical-instrument tradesmen who will be responsible for the process plant and infrastructure.

A manning schedule for processing personnel was developed assuming that the majority of the workforce will be sourced from Diego de Almagro and Inca de Oro Localities.

Other small contract are also considered as the surveyor service for the mine control and monitoring, laboratory service for chemical analysis, watching and surveillance contract, and vehicles rental for operation supervisors.

21.2.5 GENERAL AND ADMINISTRATION COSTS

A. SUMMARY

The General and Administration costs cover those costs that cannot be directly assigned to mining or processing. The annual General and Administration operating costs have been estimated by major cost driver:

- Labor;
- Other G&A costs; and
- Commercialization.

The labor, expenses, accommodation & catering costs, vehicles, transport and exporting freight components of the general and administration cost estimate were developed based on experience from similar mining projects in Chile.

The total estimated cost is about US$ 1.1 million per year or approximately 11 c/lb copper produced.

A summary by year for mining operating costs is presented in Table 21.19

Table 21.19: General & Administration and Total Operating Cost Summary

Berta 5kt/y Copper Leach Project GENERAL & ADMINISTRATION COSTS		Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	TOTAL
LABOUR	$k	**273**	**273**	**273**	**273**	**273**	**273**	**273**	**273**	**2.185**
	$/t Oxide	0,13	0,13	0,11	0,16	0,23	0,25	0,17	0,21	0,16
	$/lb	0,04	0,02	0,02	0,03	0,03	0,02	0,02	0,03	0,03
OTHER G&G AND EXPENSES	$k	**576**	**625**	**298**	**280**	**274**	**305**	**294**	**254**	**2.904**
	$/t Oxide	0,26	0,29	0,12	0,17	0,23	0,28	0,19	0,19	0,21
	$/lb	0,08	0,05	0,03	0,03	0,03	0,02	0,03	0,03	0,03
COMMERCIALIZATION	$k	**426**	**505**	**503**	**474**	**466**	**514**	**497**	**433**	**3.817**
	$/t Oxide	0,19	0,23	0,21	0,28	0,39	0,47	0,31	0,33	0,28
	$/lb	0,06	0,04	0,04	0,05	0,05	0,04	0,04	0,05	0,05
TOTAL	$k	**1.275**	**1.403**	**1.073**	**1.027**	**1.013**	**1.092**	**1.064**	**959**	**8.906**
	$/t Oxide	0,58	0,64	0,45	0,62	0,84	1,01	0,67	0,73	0,65
	$/lb	0,17	0,12	0,09	0,10	0,10	0,09	0,09	0,12	0,11

B. LABOR

Labor costs include salary and cost charges for managerial and administration staff at the site and also comprises a small management team leading accounting, marketing and sales and procurement groups in the Santiago office.

C. OTHER G&A COSTS

Expenses cover those general and administration costs for the site other than labour and vehicles including:

- Consumables;
- Services;
- Personnel costs; and
- General expenses.

Consumables include work clothes and safety equipment, tools and stationery, safety, subscriptions and publications and environmental consumables.

Services include product transportation services and exporting freight, specialized services such as industrial cleaning, auditing and accounting services, IT support and telephony services.

Personnel costs include accommodation, recreation, catering services and crew rotation transportation to Diego de Almagro, Copiapó or/and Santiago. These services will be provided by a contractor and the costs charged monthly at agreed rates.

General expenses include office expenses, travel and training, insurances, licenses, legal and tax expenses and consulting fees.

D. COMMERCIALIZATION

Commercialization includes all the expenses for selling cathodes such as: insurance, administrative and legal consulting and transport to port.

22.0 ECONOMIC ANALYSIS

The Amended Updated PEA includes a revised open pit mine plan, new operating and capital costs and financial analysis for the Berta project which contemplates the production of an average of 4,700ktpy of copper cathode for a period of 8 years.

The primary assessment criterion of project viability was determined by specific calculations of the Net Present Value (NPV) and Internal Rate of Return (IRR) using project cash flows and Base Case at a constant $2.8/lb copper price over the life of mine.

22.1 REVISED MINE PLAN AND ECONOMICS

- Mine life: 8 years
- Total copper production: 37,821 tonnes (83.3 million lbs)
- Copper price: $2.80/lb, flat
- Average cash operating costs of $1.59/lb Cu
- Capital costs: $23 million (with an accuracy of +/- 25%, including $0.2 million in project contingency, $2.5 million in sustaining and closure capital
- Pre-tax NPV(8%): $46.4 million, IRR: 83%
- After tax NPV(8%): $35.2million, IRR: 75%

22.2 FINANCIAL MODEL DETAILS

- Base Case at $2,8/lb copper price over the life of mine
- Energy price $0.117/Kwh at Nora Plant and $0.222/Kwh at Berta site (by diesel generation);
- Capital Cost Details Table 22.1:

Table 22.1: Capital Cost

Area No	AREA TITLE	TOTAL $´000	Yr1 $´000	Yr2 $´000	Yr3 $´000	Yr4 $´000	Yr5 $´000	Yr6 $´000	Yr7 $´000	Yr8 $´000
10	**NORA PLANT PURCHASE & STARTUP**	**5,761**	6,467	219						-925
20	**BERTA CONSTRUCTION**	**6,375**		6,375						
30	**NORA EXPANSION**	**1,324**		1,324						
40	**PIPELINE PLS & RAFF/WATER**	**3,773**	107	3,666						
50	**Other Owner Cost**									
	Sustaining Capex	1,308	163	163	163	163	163	163	163	163
	Option Payments	2,300	574	1,156	570					
	Closure Costs	2,200								2,200
	GRAND TOTAL	**23,040**	**7,311**	**12,903**	**733**	**163**	**163**	**163**	**163**	**1,438**

- Operating Cost Details Table 22.2:

Table 22.2: Operating Cost Details

Berta 5ktpy Copper Leach Project Process Opex Summary		Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	TOTAL
Total Costs ($million)	$´000	$ 9,93	$ 14,42	$ 12,54	$ 11,77	$ 10,72	$ 10,18	$ 11,26	$ 10,82	$ 132,29
	$/lb	$ 2,09	$ 1,59	$ 1,46	$ 1,63	$ 1,64	$ 1,27	$ 1,47	$ 1,83	$ 1,59
Mining	$´000	5.527	5.428	5.325	5.325	5.325	5.339	5.325	3.760	41.353
	$/lb	0,73	0,45	0,45	0,52	0,54	0,43	0,46	0,47	0,50
Processing	$´000	9.064	12.169	10.880	10.357	9.789	9.394	10.497	9.880	82.029
	$/lb	1,19	1,02	0,92	1,01	1,00	0,75	0,91	1,24	0,98
General & Adm	$´000	1.275	1.403	1.073	1.027	1.013	1.092	1.064	959	8.906
	$/lb	0,17	0,12	0,09	0,10	0,10	0,09	0,09	0,12	0,11

22.3 PRODUCTION SCHEDULES

The project has an annual production rate of cathode copper of approximately 5,000 t/y. The mining rate varies with the copper grade processed in order to maintain this cathode production rate throughout the 8 year mine life.

A. MINING AND PROCESSING

The annual tonnages of mined and processed oxide materialare shown in Table 22.3

Table 22.3: Mine Production Schedule

Production Profile		Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Tot
Nora Crushed	Ton	399.258	-	-	-	-	-	-	-	**399.258**
	CuT%	0,83	-	-	-	-	-	-	-	**0,83**
	CuS%	0,61	-	-	-	-	-	-	-	**0,61**
	Rec%	80,97	-	-	-	-	-	-	-	**81,0**
	Cu Cathode, t	2.673	-	-	-	-	-	-	-	**2.673**
Ripios Line	Cu Cathode, t	315								**315**
Berta Crushed	Ton	84.932	1.002.740	1.000.000	1.000.000	1.000.000	846.925	1.000.000	828.737	**6.763.334**
	CuT%	0,55	0,51	0,55	0,50	0,51	0,79	0,61	0,48	**0,56**
	CuS%	0,39	0,36	0,39	0,35	0,36	0,56	0,43	0,34	**0,40**
	Rec%	0,79	0,78	0,78	0,77	0,77	0,78	0,79	0,77	**0,78**
	Cu Cathode, t	366	4.025	4.271	3.838	3.945	5.241	4.783	3.074	**29.544**
Berta ROM	Ton	109.353	1.537.653	1.163.006	975.679	598.340	490.499	470.580	603.613	**5.948.725**
	CuT%	0,18	0,20	0,21	0,19	0,19	0,19	0,21	0,20	**0,20**
	CuS%	0,11	0,12	0,13	0,11	0,11	0,11	0,13	0,12	**0,12**
	Rec%	45	45	45	45	45	45	45	45	**45**
	Cu Cathode, t	90	1.387	1.101	812	501	408	440	541	**5.280**
Total Cu	Cu Cathode, t	3.444	5.412	5.372	4.650	4.446	5.650	5.223	3.615	**37.812**
Stockpiled Material										
Berta ROM	Ton	499.882	499.882	499.882	499.882	499.882	499.882	499.882	499.882	**499.882**
	Cu Cathode, t	490	490	490	490	490	490	490	490	**490**
Berta Leach	Ton	1.090.174	732.799	957.612	650.361	254.530	(0)	115.396	- 0	**- 0**
	Cu Cathode, t	4.036	2.281	2.922	1.896	742	- 0	330	- 0	**- 0**

B. PLANT PERFORMANCE

The processing plant will produce approximately 5,000 t/y cathode copper. The average copper recoveries assumed for this study and the base case economic model are 78% CuT for heap leach material and 45% CuT for dump leach material.

22.4 CASH FLOW ANALYSIS

As shown in Table 22.4, The Base Case operating cash flow peaks at $14 million in the 2nd year and reached its minimum value of $6.1 million in the 8th year, which is the last operating period.

Regarding taxation, on September 29th 2014 a tax reform bill was approved by Congress and signed by the President.This resulted in the tax rate increasing from 20% to 25% in 2015 and 27% by 2017. However, the regulations accompanying this law have not yet been published and other changes to the Chilean tax system are being considered, so SCMB elected to retain a flat 27% tax rate for the effects of this PEA.

Table 22.4: Summary Cash Flow Analysis (Base Case)

Table 2.5 - US$ millions		Phase I	Phase II							LOM
Operating Cash flow per period	**0**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	
		(11 months)	(13 months)	(12 months each period)						**TOTAL**
Net Revenue		18,1	35,7	32,6	28,3	27,0	34,4	31,7	22,0	229,8
Operating Costs (incl. inventory)		14,0	20,9	17,3	16,7	16,1	15,8	16,9	14,6	132,3
Operating Cash Flow		4,2	14,8	15,4	11,6	10,9	18,5	14,9	7,4	97,5
Paid Taxes		-	-	1,8	2,7	2,9	5,0	3,9	1,9	18,3
Net Cash Flow		4,2	14,8	13,6	8,8	8,0	13,5	10,9	5,4	79,2
	Phase I	**Phase II**								
CAPEX	4,1	11,4	-	-	-	-	-	-	-	15,4
Working Capital & Startup	2,5	0,2	-	-	-	-	-	-	- 0,9	1,8
Sustaining CAPEX	-	0,2	0,2	0,2	0,2	0,2	0,2	0,2	0,2	1,3
Option Payments	0,6	1,2	0,6	-	-	-	-	-	-	2,3
Closure Costs	-	-	-	-	-	-	-		2,2	2,2
Total Outflows	7,1	12,9	0,7	0,2	0,2	0,2	0,2	0,2	1,5	23,0
Net Cash Flow After Tax	- 7,1	- 8,7	14,0	13,4	8,7	7,8	13,4	10,8	3,9	56,2

Readers are advised that more detailed engineering studies have not been completed for the Berta project and so the normal progression from PEA to Preliminary Feasibility Study to Feasibility Study has not been followed in respect of making a production decision. Therefore, investors are cautioned that no mineral reserves have been declared and the level of confidence in the resources, metallurgy, engineering and cost estimation is not at a level normally associated with a project reaching a production decision.

Table 22.5 provides a summary of the Base Case economic evaluation at discount rates of 5%, 8% and 10% for NPV and IRR. Also it provides a sensitivity analysis regarding copper price.

Table 22.5: Berta Economic Evaluation Summary

Cu Price	$2.60/ lb		$2.80/ lb		$3.00/ lb	
NPV ($ millions)	**Pre tax**	**After tax**	**Pre tax**	**After tax**	**Pre tax**	**After tax**
5%	42.3	32.3	55.2	41.8	68.1	52.1
8%	35.1	26.8	**46.4**	**35.2**	57.7	44.3
10%	31.1	23.7	41.5	31.5	51.8	39.9
IRR	62%	56%	**83%**	**75%**	106%	98%

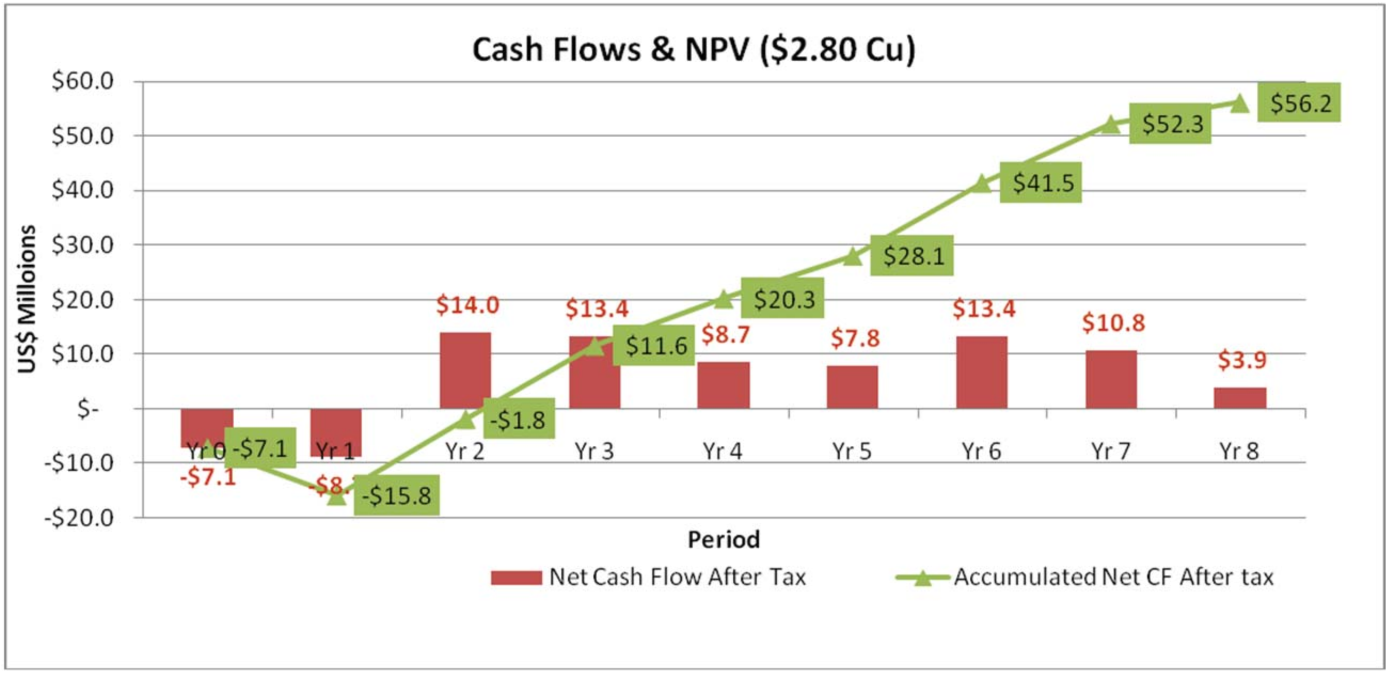
Table 22.6 sets out the revised mine plan and economics contemplated by the Amended Updated PEA and Figure 22.1: Economic Evaluation Summary: Cash flow & NPV

Table 22.6: Summary Economics

	Revised Mine Plan			Prior
	Phase 1	**Phase 2**	**LOM**	**LOM**
Copper Price	US$2.80/lb			US$3.00/lb
Copper Production	2,988	34,833	37,821	38,400
Duration	11 months	7 years	8 years	8 years
Cash Costs	$1.75/lb	$1.57/lb	$1.59/lb	$2.03/lb
CAPEX ($million)	$7.15	$12.6	$23.0(1)	$20.3
Pre-tax:				
NPV (8%)	$46.4 million			$34.3m
IRR	83%			55.2%
After-tax				
NPV (8%)	$35.2 million			$26.6m
IRR	75%			46.9%

(1)Includes closure costs and sustaining CAPEX not included in Phase 2

Figure 22.1: Economic Evaluation Summary: Cash flow & NPV



23.0 ADJACENT PROPERTIES

There are no adjacent properties which are material for the present resource evaluation of Berta.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 BERTA SUR PIT GEOTECHNICAL DESIGN

SCM Berta commissioned a preliminary geotechnical design of the Berta Sur pit from GEOINVESTMENT SPA.

In the following sections the available information, the resistant properties, the results and conclusions of the completed analysis and studies, and the recommendations, are presented.

Available Information

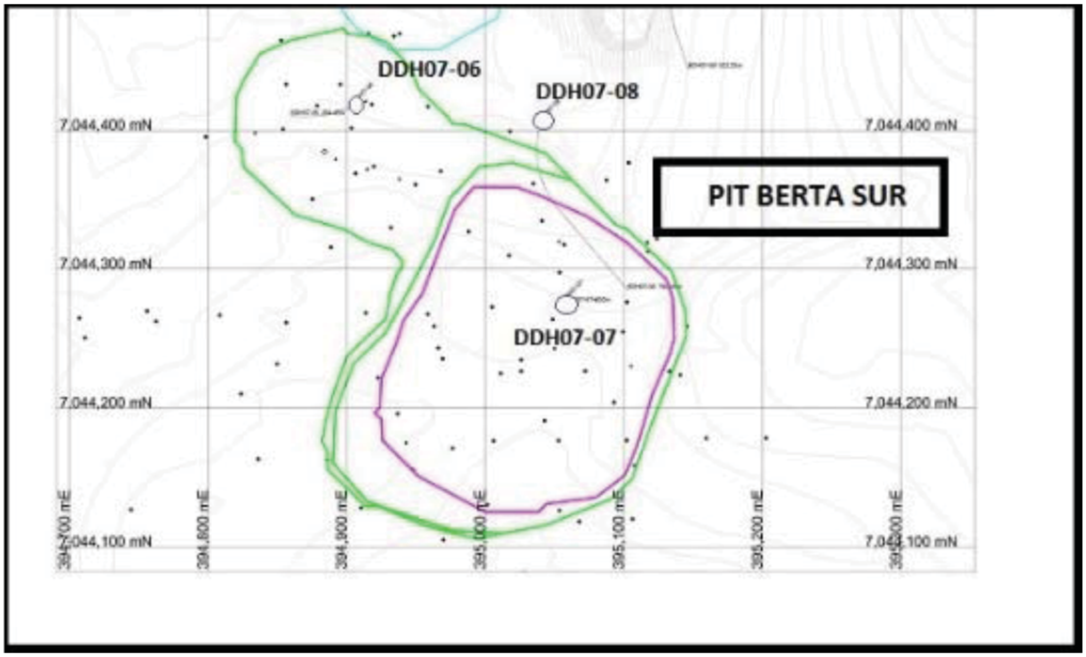
The available information to carry out the study consisted of:

- Summary of the regional and local geology
- Preliminary geological model
- Block model
- Updated topography of the project area
- Record of drilling for geotechnical purposes

Based on the available information, the analysis process began with the implementation of diverse approaches for obtaining pit slope angles The methodologies used in the analysis, are as follows;

The diamond drill holes drilled in Berta Sur, shown in Figure 24.1: DDH-7-06, DDH-7-07 y DDH-7-08, already cut to determine the copper grade, were geotechnically logged using the GSI methodology of HOEK and MARINOS 2001.. The core used to estimate GSI, although previously cut, produced very favourable results indicating good geotechnical qualities for the rocks. These positive results should be corroborated with an analysis of whole core which would probably result in more favourable conclusions.

Figure 24.1: Location of Diamond drill holes in Berta Sur



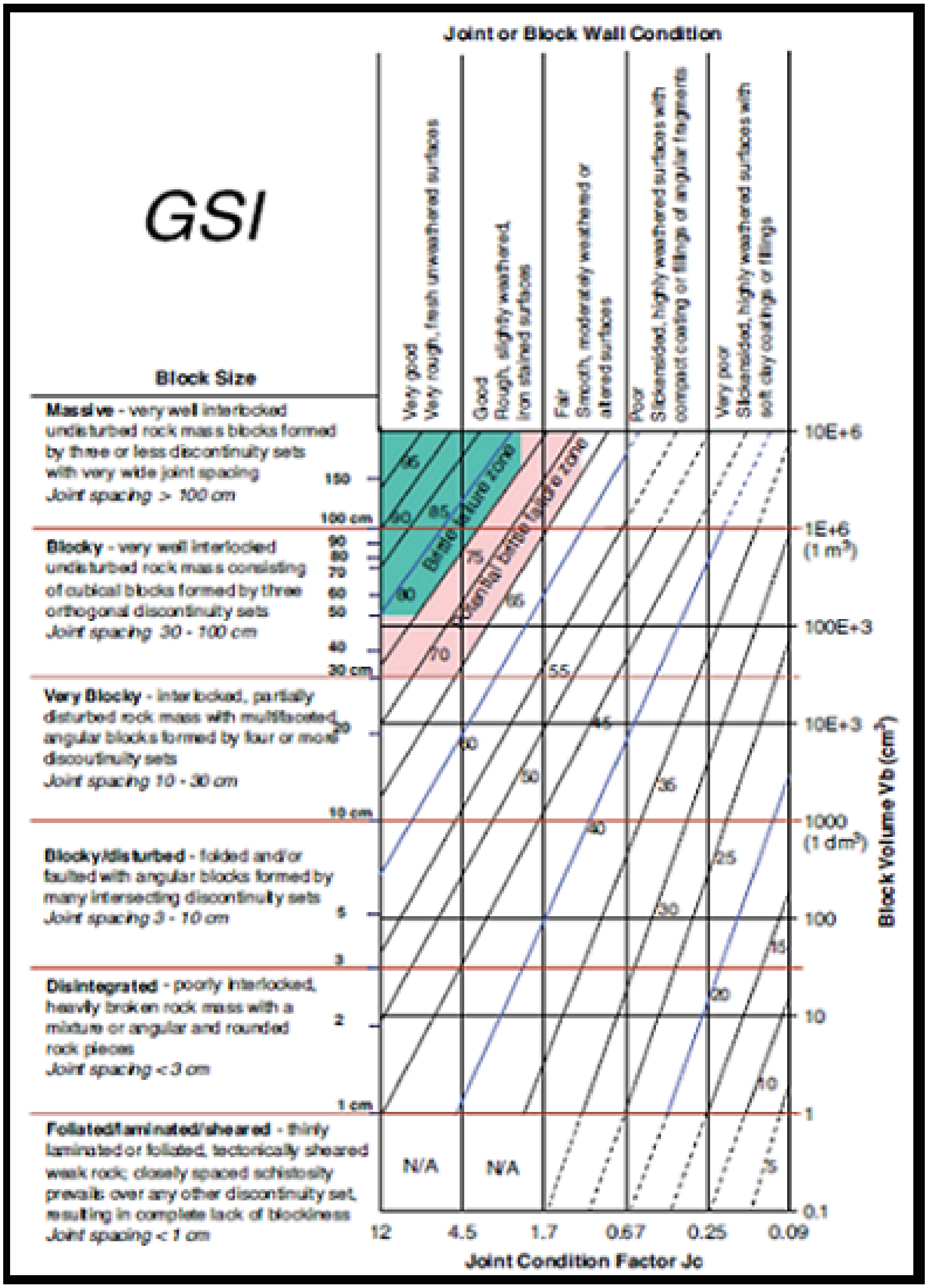
The GeologicalStrengthIndex (GSI) and its use in the failure criteria of Hoek-Brown has been featured in several articles by Hoek (1994), Hoek. et al. (1995) and Hoek-Brown (1997), associated with hard rock masses and equivalent to the RMR (Rock Mass Rating) system. From 1998 to date the GSI system with the aim of including rock masses of poor quality (Hoek at al 1998; Marinos and Hoek, 2000 and 2001), has been developed Graph Nº1.

The GSI provides a system to estimate the reduction in resistance that a rock mass would present under different geological conditions and is obtained from the combination of two fundamental geological parameters; the structure of the rock mass and the conditions of the discontinuities. In practice, it is usual to define the GSI in ranges of ±15 point. The classification is made according to the following criteria, shown in Table 24.1

Table 24.1: Geotechnical Quality of the Rock Mass, Based on GSI

Rock Quality	Class	GSI
Very Poor	V	0-20
Poor	IV	21-40
Regular	III	41-60
Good	II	61-80
Very Good	I	81-100

Figure 24.2: Graph Nº 1 GSI ranges, modified by Cai et al



For this case, the GSI values obtained by lithology for every drill hole logged geotechnically and their average values by lithology and mineral zone, are summarized in Table 24.2

Table 24.2: GSI values by Lithology and Mineral Zone

BERTA SUR GSI		
GEOTECNICAL UNIT	Oxides	Sulphides
VCB (CARBONATE VEIN)		80
TON (TONALITE)	52	78
PTF (FINE PORPHYRY)	52	78
PTC (CROWDED PORPHYRY)	49	75
BXI (IGNEOUS BRECCIA)	49	76
PAN (SANDESITIC DYKES)		72

Additionally, it is possible to establish a relationship between the GSI and the RMR, which has been established empirically. If the water condition rating is 15 and the discontinuity orientation is 0, it can be approximately observed that the following equation is fulfilled:

$$GSI = RMR_L - 5$$

$$RMR_L = GSI + 5$$

The results obtained for RMR L, are shown in Table 24.3

Table 24.3: Obtained values of RMR L

BERTA SUR RMR L		
GEOTECNICAL UNIT	Oxides	Sulphides
VCB (CARBONATE VEIN)		85
TON (TONALITE)	57	83
PTF (FINE PORPHYRY)	57	83
PTC (CROWDED PORPHYRY)	54	80
BXI (IGNEOUS BRECCIA)	54	81
PAN (ANDESITIC DYKES)		77

Table 24.4: Rock Mass Quality according to RMR index

Class	Quality	RMR	Cohesion Kg/cm^2	Friction Angle
I	Very Good	100 -81	> 4	> 45°
II	Good	80 - 61	3 - 4	35° - 45°
III	Regular	60 - 41	2 - 3	25° - 35°
IV	Poor	40 - 21	1 -2	15° - 25°
V	Very Poor	< 20	< 1	< 15°

Summarizing, the oxides zone has values between 54 and 57, which means rock of regular geotechnical quality, while sulphides are between 77 and 85, meaning rocks of good and very good geotechnical quality.

To the RMR (Rock Mass Rating) values or in situ score for each design region the necessary adjustments to take into account of weathering, structural orientation, forces and blasting effects, were applied, thus obtaining the MRMR (Mining Rock Mass Rating) values shown in Table 24.5, from which Table 24.6 showing the respective overall slope angles was derived.

The adjustment factors to be used in this case are:

m= weathering = 1

o= orientation = 0.95

t= blasting = 0.94

e= forces = 1 **With overall adjustment of 0.893**

Table 24.5: Adjusted values MRMR

BERTA SUR MRMR		
GEOTECNICAL UNIT	**Oxides**	**Sulphides**
VCB (CARBONATE VEIN)		76
TON (TONALITE)	51	74
PTF (FINE PORPHYRY)	51	74
PTC (CROWDED PORPHYRY)	48	72
BXI (IGNEOUS BRECCIA)	48	82
PAN (ANDESITIC DYKES)		69

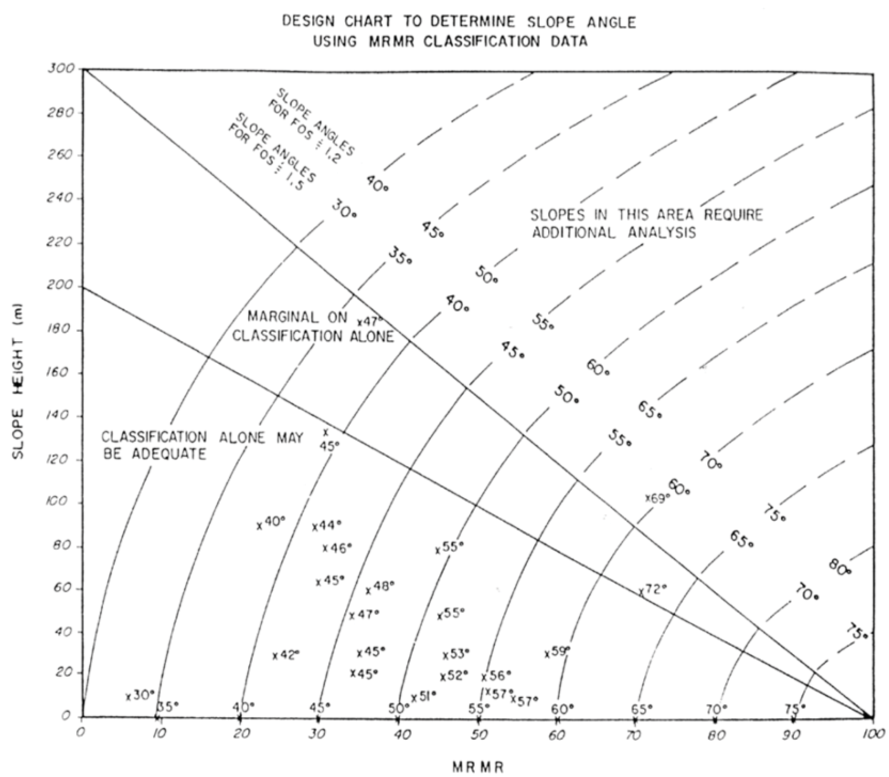
Table 24.6: MRMR Geotechnical Quality vs Slope Angle

MRMR	100	90	80	70	60	50	40	30	20	10	0
Slope Angle	>75°	75°	70°	65°	60°	55°	50°	45°	40°	35°	<35°

24.1.1 ESTIMATION OF SLOPE ANGLES

The authors A. Haines and P. Terbrugge in their publication Estimación Preliminar de Estabilidades de Taludes en Roca Usando Sistemas de Clasificación de Macizos Rocosos provide a practical and simple tool to determine slope angles based on multiple real data of design angles, slope heights and the adjusted classification of Laubscher. GraphNº2 shows the abacus design, where an empirical relationship between slope height, the adjusted rock quality (MRMR) and slope angles is established (Fig 24.3)

Figure 24.3: Graph N° 2: Abacus design MRMR vs Slope Height



The abacus is divided in three sections:

Lower Section: using the classification alone may be adequate to design the slope height/angle configuration from the MRMR.

Intermediate Section: corresponding to a marginal decision from where the MRMR classification alone could be used.

Upper Section: it is recommended to perform more rigorous additional analytical analysis for these slope configurations.

In this abacus there are two superimposed sets of angle values in each design curve. The value of one of the curves corresponds to civil engineering projects, where the Security Factor used is 1,5. The second superimposed value corresponds to mining projects with Security Factors of 1,2.

Table 24.7 shows the slope angles values with an associated intermediate Security Factor of 1,3, typical of mining situations for MRMR values associated with an overall height of 100m, for each design region.

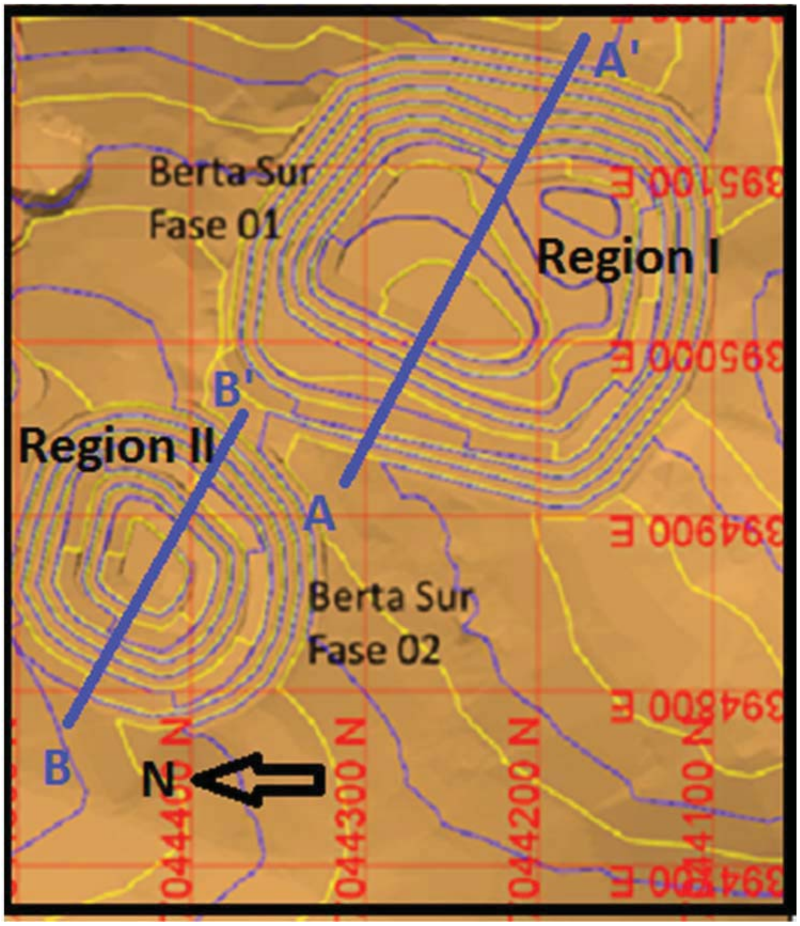
Table 24.7: Recommended Slope Angles for the Berta Sur, FINAL PIT (MRMR)

Pit o Región	Angulo Recomendado (º)	Tipo de Material	Angulos Factibles de Alcanzar (º)
Región I	50	Tonalita y Pórfidos	> 60
Región II	50	Brecha y Pórfidos	55

Design Regions

Based on available information, the Berta Sur Phase 1 and Phase 2 Pits would comprise two design regions given the disposition of the rock units in each Pit. Region I has two rock units; an upper Tonalite Zone with oxides and Porphyry, primary sulphide zone. Region II also has two rock units; an upper hydrothermal breccia and Porphyry in the primary sulphide zone. Two sections named A (Region I) and B (Region II) were designed in order to evaluate the variation of the slope orientation the materials and/or rock types present in each of them, and their respective heights. Both abovementioned regions are shown in Figure 24.4

Figure 24.4: Design Regions



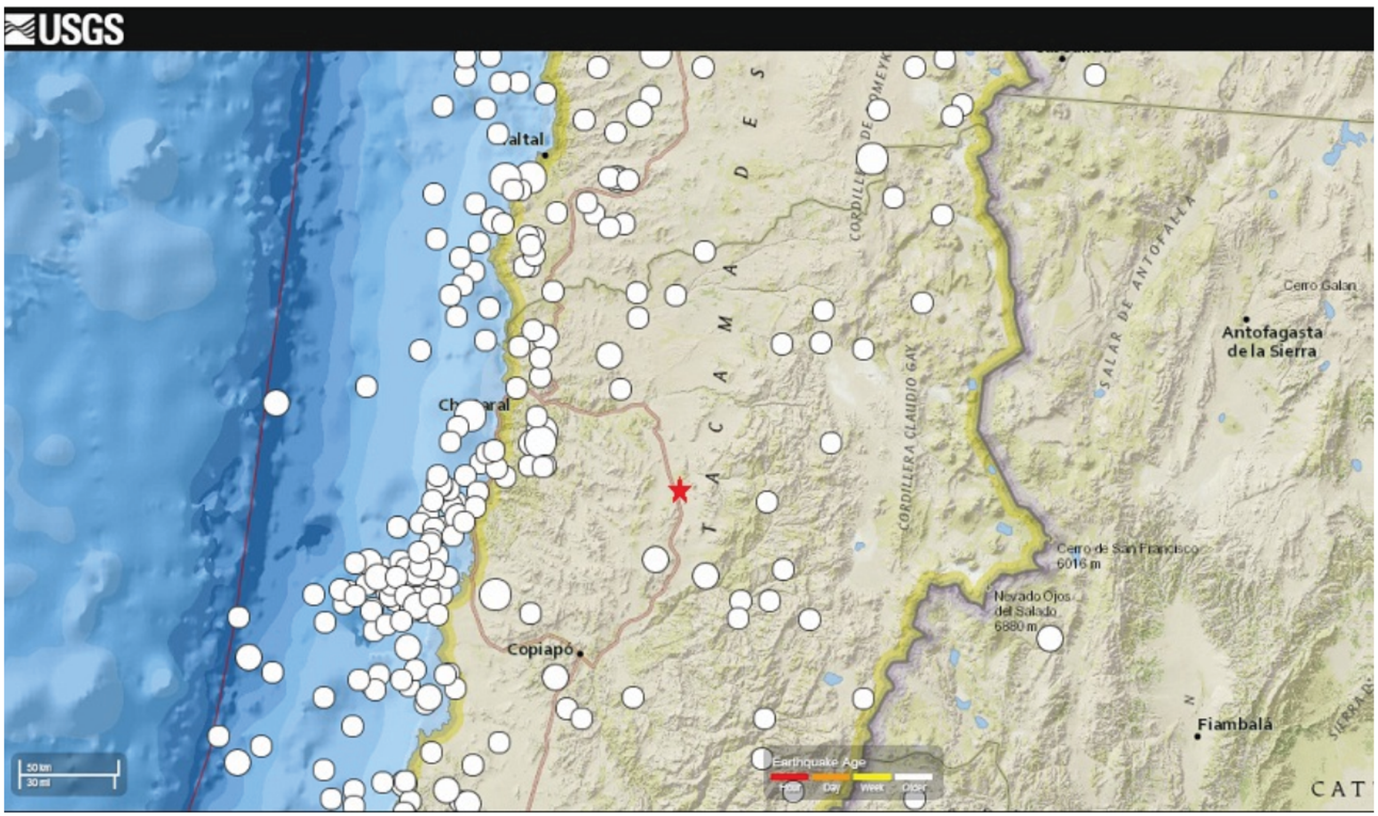
Seismic Risk

In the period 1570-2005, corresponding to a continuous period of 435 years, the North, Central and South regions of Chile were affected by numerous violent earthquakes including some with estimated magnitudes of greater than 8.0. Figure 24.5 shows Earthquakes location over 5 on the Richter Scale in that region.

The epicentres of the largest earthquakes have always been located in the Pacific Ocean, whose focuses are originated at depth, associated with the Benioff Zone. The Earthquakes with epicentres in the mainland are of magnitudes less than 7.0.

The following list corresponds to major earthquakes that have affected the North, Centre and South of Chile during the period 1570-2005, with probable epicentres between southern Peru and southern Chile (Grave, 1964; Kausel, 1982; Lomnitz, 1971).

Figure 24.5: Location of earthquakes of > 5 on the Richter Scale since 1900



To estimate the seismic risk, Labbe and Saragoni (1976) have established the parameters a and b of the Gutemberg-Richter relationship as expressed by the formula:

$$\text{Log } N = a - M\,b$$

Where "N" is the frequency of occurrence of earthquakes, with Richter scale greater than or equal to "M".

"a" and "b" are constants that relate the intensity of the movement with the Richter scale and the focal distance in a given seismic zone.

The aforementioned authors have defined three seismic regions in Chile from an engineering point of view, considering only the events of magnitude major than 7.0, with the purpose of evaluating the risk of occurrence of large earthquakes.

The defined zones are shown in the following Table 24.8

Table 24.8: Seismic Regions in Chile

Zone	Latitude	a	b	Seismicity
Arica-Taltal	18ºS-26ºS	7,68	1,14	Major
Taltal-P.Aysén	26ºS-45ºS	6,36	0,99	Intermediate
P.Aysén-C.Hornos	45ºS-56ºS	4,46	0,81	Minor

Based on the information in this Table, the probability of occurrence of an earthquake, magnitude greater than 7.0 for the Taltal -Puerto Aysén zone, is shown in Table 24.9

Table 24.9: Richter Scale vs number of earthquakes per year for Taltal – Puerto Aysén zone.

Magnitude Range	Number of earthquakes per year
7,0 - 7,5	0,17
7,6 - 7,9	0,05
8,0 or more	0,027

Determination of Seismic Design

For the determination of seismic design the “a” and “b” parameters, shown in the table were used in the following equation, which provides the number of earthquakes of a given magnitude.

$$\log N' = a - bI$$

Where:

a= 6,36

b= 0,99

N' = Accumulated Frequency

I= Magnitude

For an 8.0 magnitude earthquake, there is an accumulated frequency of 0.0275 per year, meaning an earthquake every 36.31 years, while for a 7.7 magnitude there is an earthquake every 18.32 years (Table 24.10: Return Period

That is:

Table 24.10: Return Period

Magnitude	Accumulated Frequency (Earthquakes/year)	Period (years)
7.7	0.0545	18.32
8.0	0.0275	36.31
8.5	0.0088	113.50

The earthquake types that could affect the project area are then established for each of the seismic sources:

Quake I: Interplate Subduction Earthquake (MCE).

A typical seismic event of subduction with similar characteristics to the 1918 earthquake in Taltal-Caldera. The level of the Maximum Credible Earthquake (MCE) will be represented in this case by a design magnitude Ms = 8.5. The hypo-central distance to consider will be R = 60 [km].

Quake II: Interplate Earthquake of Intermediate Depth (MCE).

A typical interplate seismic event of intermediate depth with similar characteristics to the 1939 earthquake in Copiapó. The Maximum Credible Earthquake (MCE) will be in this case represented by design magnitude Ms = 7.5. Its Hypo-centre will be located to the minimum distance to the interplate zone, where an earthquake of this magnitude can occur. In this case is estimated to be located at a hypo-central distance of 80 [km].

Attenuation Relationships of Maximum Horizontal Accelerations

The estimation of the maximum accelerations induced by earthquakes requires the attenuation relationships to be determined empirically. These relationships consider that value of this parameter reduces with increasing distance from the epicentre of the earthquake.

As different types of seismic sources have their own characteristics, it is necessary to distinguish adequate attenuation relationships for each one of them. The form that these attenuation relationships take varies enormously depending on the considered seismic source.

Interplate Subduction Earthquake Type

Considering the last earthquake data to date, Ruiz and Saragoni (2005) disaggregated the database of interplate and intraplate earthquakes, obtaining for interplate subduction earthquake (“thrust type”) the following attenuation relationships for material of “rock and hard soil” type (360 [m/s] <= Vs<=1.500 [m/s]) for accelerations and maximum induced velocities:

Maximum Horizontal Acceleration (Ruiz and Saragoni, 2005):

$a_{h\ max} = 2 \times e^{(1,28xMs)}/(R+30)^{1,09}$ (cm/s^2)

Where, M is the Richter scale and R the hypo-central magnitude in km.

Intraplate Earthquake type of Intermediate Depth

For Chilean earthquakes, Ruiz and Saragoni (2005) developed the following attenuation relationships specifically for intraplate earthquakes of intermediate depth, considering material of “rock or hard soil” type (360[m/s]<= Vs <=1.500[m/s]) for maximum accelerations and induced velocities:

Maximum Horizontal Acceleration (Ruiz y Saragoni, 2005):

$a_{h\ max} = 3.840 \times e^{(1,2xMs)}/(R+80)^{2,16}$ (cm/s^2)

As in the previous case, M is the Richter scale and R is the hypo-central distance in km.

Table 24.11 shows a summary of the maximum acceleration values expected in the Project area for the defined designed earthquakes.

Table 24.11: Summary of Designed Earthquakes

Designed Earthquake	M_s	R (km)	A_{hmax} (g)
Interplate (Thrust)	8.5	60	0.80
Intermediate Depth Intraplate	7.5	80	0.55

The data presented in Table 24.11 indicates that for the interplate (Thrust) earthquake that accelerations are greater than those of the intraplate earthquake of intermediate depth. Therefore, the Interplate (Thrust) earthquake type of magnitude Ms=8.5 for the designed earthquake; with maximum horizontal acceleration of 0.80g will be used.

Designed Seismic Coefficient

The designed seismic coefficient will be determined for use in the pseudo-static analysis and slope stability design in coarse granular material, soils, tailings, leached mineral material stockpiles and rock.

Calculation of Designed Seismic Coefficient

The designed seismic coefficient for pseudo-static analysis is determined through an equation that relates maximum horizontal acceleration of the earthquake controlling the design, with the seismic coefficient.

The following equation is calibrated for Chilean seismicity (Saragoni, 1993):

$$K_h=0,30 \times a_{hmax}/g \qquad ;a_{hmax}<=0,67g$$

$$K_h=0,22 \times (a_{hmax}/g)^{0,5} \qquad ;a_{hmax}> 0,67g$$

The maximum horizontal acceleration obtained in the analysis of the considered designed earthquake, corresponds to interplate earthquake type Thrust, $a_{hmáx}$ = 0,80 [g],and the following horizontal coefficient of design is obtained:

$$K_h = 0,20$$

24.1.2 STABILITY ANALYSIS (LIMIT EQUILIBRIUM)

To carry out the stability analysis using the equilibrium limit method, the Slide program of Rocscience was used. This programproducesSecurity Factors by different methods of analyses, for example, simplified Bishop, simplified Janbu, GLE/Morgesten y Price, Cuerpo de Ingenieros, Spencer, etc. It is also possible to perform probabilistic analysis of sensibility, to incorporate pseudo-seismic analysis, water conditions, different supports for increasing stability, surface analyses of circular and non-circular failure, etc.

Based on the results of the economic pit, a representative section for each domain was chosen, with the defined geometry, lithological conditions and most representative structures of the pit configurations resulting from the economic evaluation.

In the analyses the GLE/Morgentern-Pricemethod was used to obtain the Security Factor. For the analyses performed on circular failure surfaces an automatic search grid was used, which generated about 10,116 surfaces of analysis, and in that surface where the lowest Security Factor was obtained, (using the estimated mean values of the resistant properties assigned to each material), a probabilistic analysis was performed, using 1,000 samples randomly generated by Monte-Carlo, again utilizing estimated mean values assigned to the properties of each material involved in the analysis.

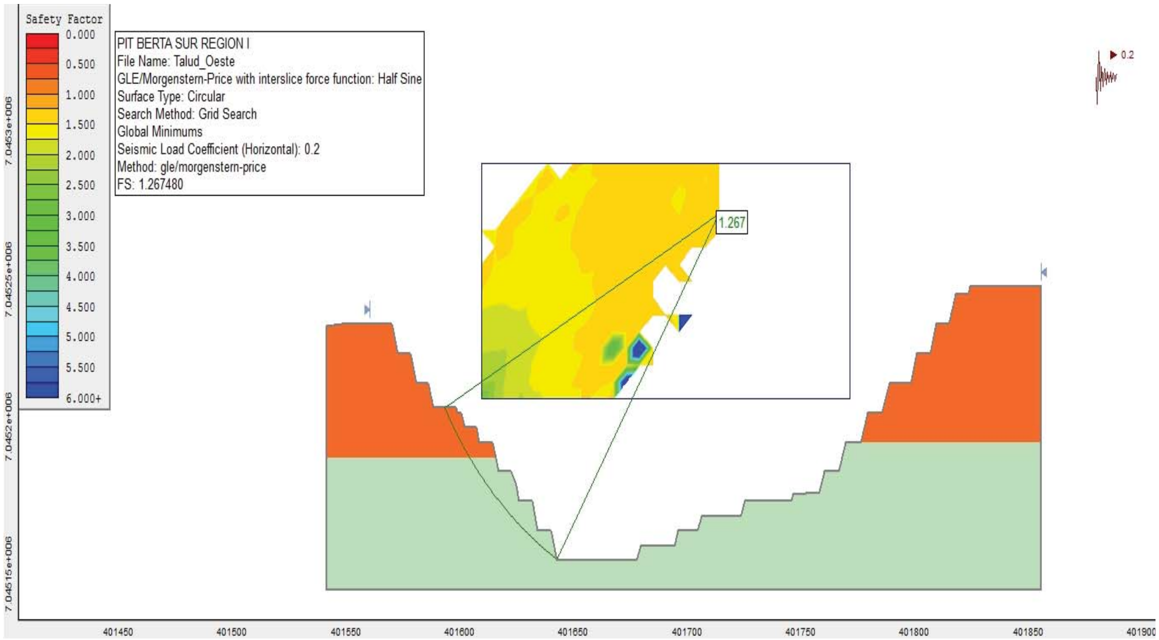
Table 24.12 shows estimated values of the physical properties (Unit weight, Cohesion, and Angle of Internal Friction) of the rocks in the slopes.

Table 24.12: Material Properties

Lithology	Unit Weight (KN/m^3)	Cohesion (KN/m^2)	Angle of Friction (°)
BXI (ox)	20	48	50
TON (ox)	25	50	50
PTF (pri)	25	51	58
PTC (pri)	25	55	52

Figure 24.6 shows a typical graphical output of the results of circular analysis, in which are shown the colour scale assigned for the surface with the lowest Security Factors (SF), over whicha probabilistic analysis was completed which shows the probabilistic results of Security Factor, and its probability of failure.The seismic coefficients applied in the analysis are also shown.

Figure 24.6: Circular Analysis



Geometry of the Sections considered for the Analyses

Based on the designs, the geometries of slopes with bench face angles of 75o for all slopes were determined and four representative cross sections for the aim of this analysis were prepared. Table 24.13: Geometrical Characteristics of the Analysed Sections

Summarizes the ranges of the geometrical characteristics of these sections that were considered for the analyses.

Table 24.13: Geometrical Characteristics of the Analysed Sections

PIT or Region	Prof. (m)	h Banco (m)	A Berma (m)	A Rampa (m)	α Global (°)	α Cara Banco (°)
REGION I	100	5	3	5.7	50	75
REGION II	70	5	3	5.7	50	75

Results of Stability Analyses

The stability analyses were carried out in sections A for Region I and B for Region II of Table 24.14 shows summaries of the achieved results for the circular analyses of each analysed layout, corresponding to 4 analyses.

Table 24.14: Summary of Results of Circular Analyses of Region I and Region II for the East and West Slopes

Pit or Región	Type of analysis	Direction	Security Factor
Region I			
Slope E AA'		To the E	1.77
Slope E AA'	pseudostatic	To the E	1.28
Slope W AA'	static	To the W	1.49
Slope W AA'	pseudostatic	To the W	1.27
Región II			
Slope E BB'	static	To the E	1.86
Slope E BB'	pseudostatic	To the E	1.35
Slope W BB'	static	To the W	1.91
Slope W BB'	pseudostatic	To the W	1.40

According to the results of stability analysis, it can be concluded that the slopes with bench face angles of 75º and overall pit angles of 50º, the East and West slopes of both pit phases are stable.

The Security Factors (FS) are all superior to FS>1,0 inboth static and pseudo-static analyses.

Faulting Structurally Controlled Method

No structural measurements from geotechnical core logging have been obtained to date. The mapped Berta Sur faults, generally oriented WNW and ENE and subvertical, separate Regions I and II. These faults are shown on Figure 24.7and Figure 24.8, and should not constitute a stability problem in the design of the final Berta Sur pit. However, a more detailed analysis will be required for control of this portion of the pit.

Figure 24.7: Showing the Principal Structures of the Berta Project

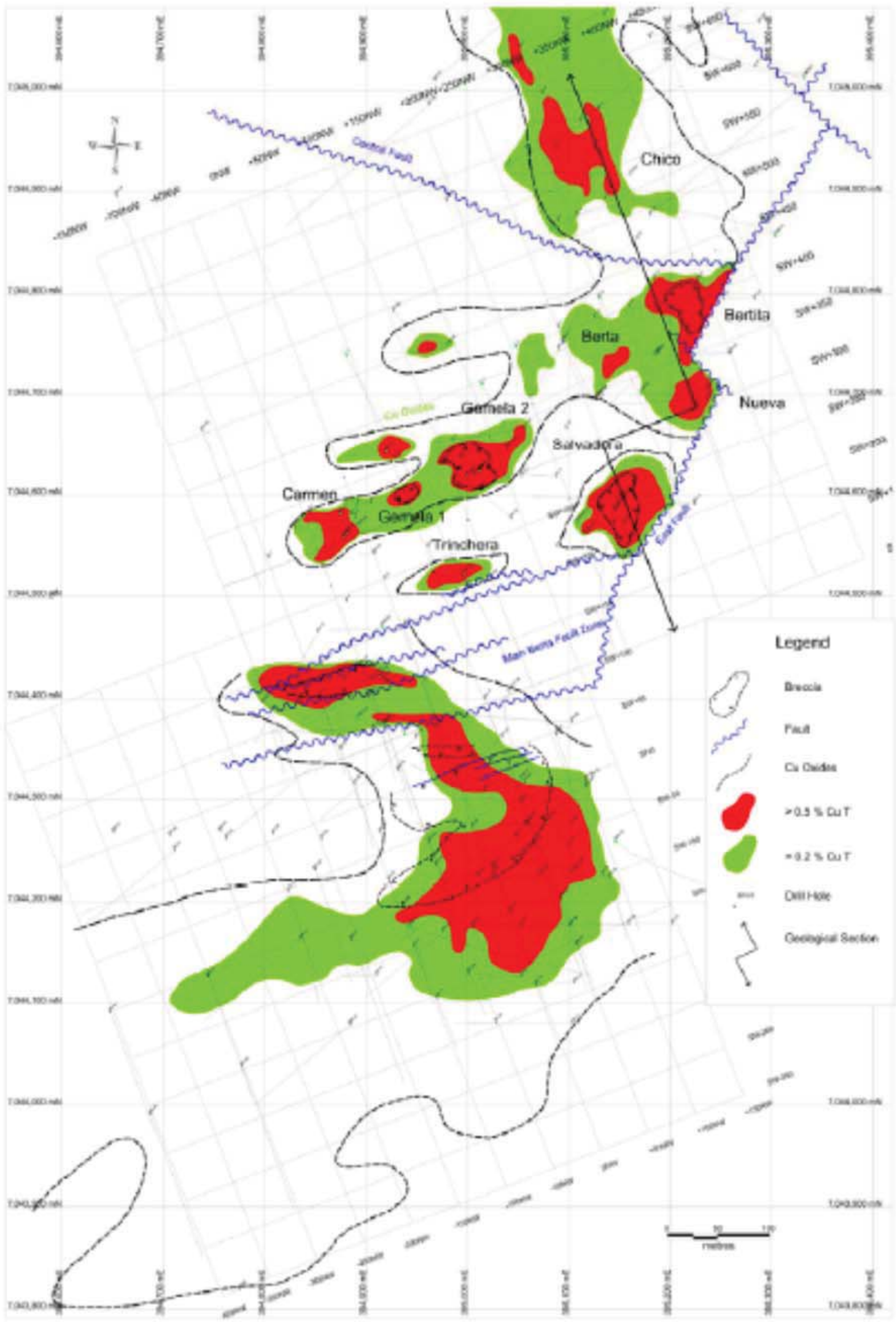
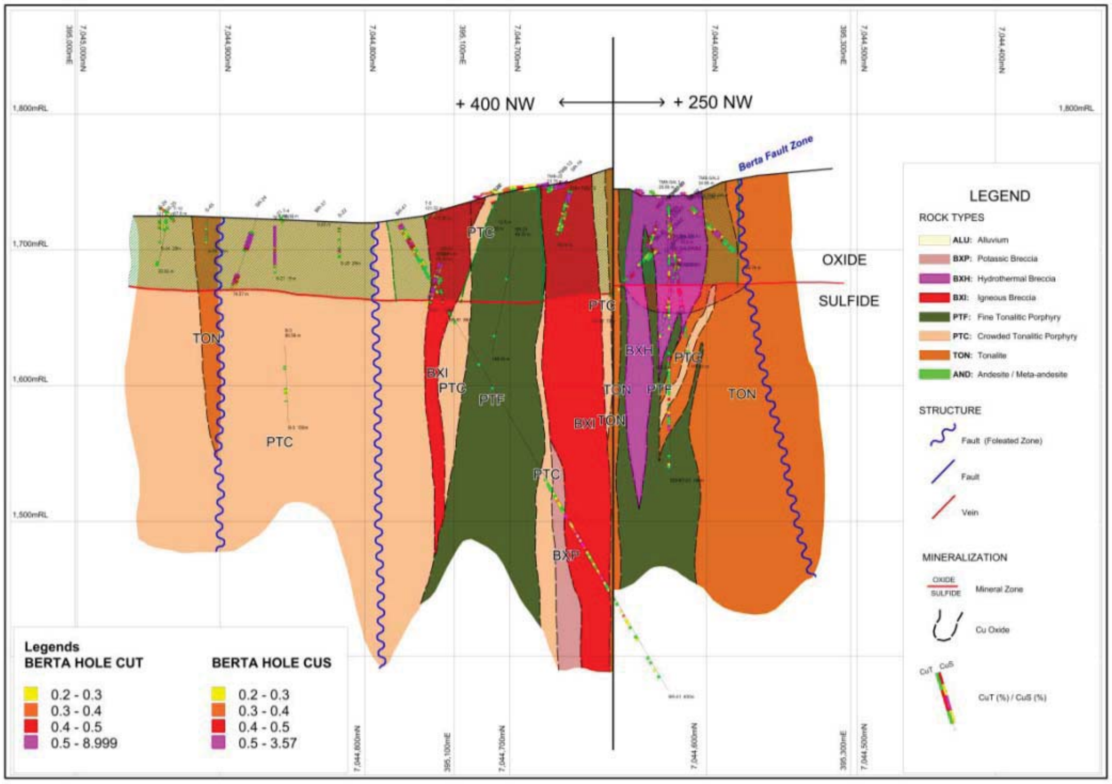


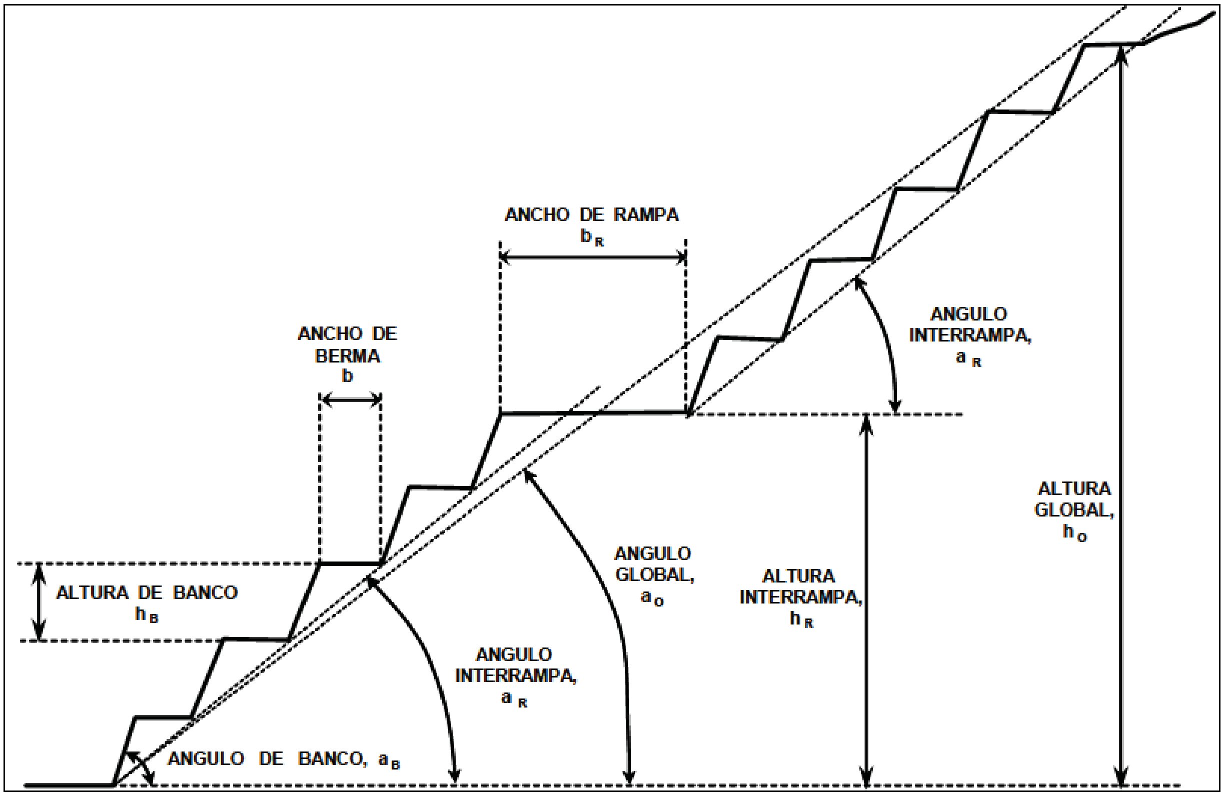
Figure 24.8: Cross Section Showing the Berta Sur Faults



24.1.3 GEOMETRICAL DEFINITION OF THE PIT

Figure 24.9 shows the parameters that define the pit geometry, overall slope angle, overall height, inter-ramp angle, inter-ramp height, ramp width, berm width, bench face angle and bench height.

Figure 24.9: Parameters that Define the Geometry of the Pit



In the 1960's, Ritchie formulated an empirical approach for obtaining the berm width required to contain any falling rocks, originally done for road cuts and civil engineering works. Later it was adapted for mining open pits, being defined by the following formulas:

b=0,2Hb+2, for heights of benches less than or equal to 9m, and

b=0,2Hb+4,5 for heights of benches greater than or equal to 9m, where

Hb = Bench Height

Applying the first formula for the case of a bench of 10m;

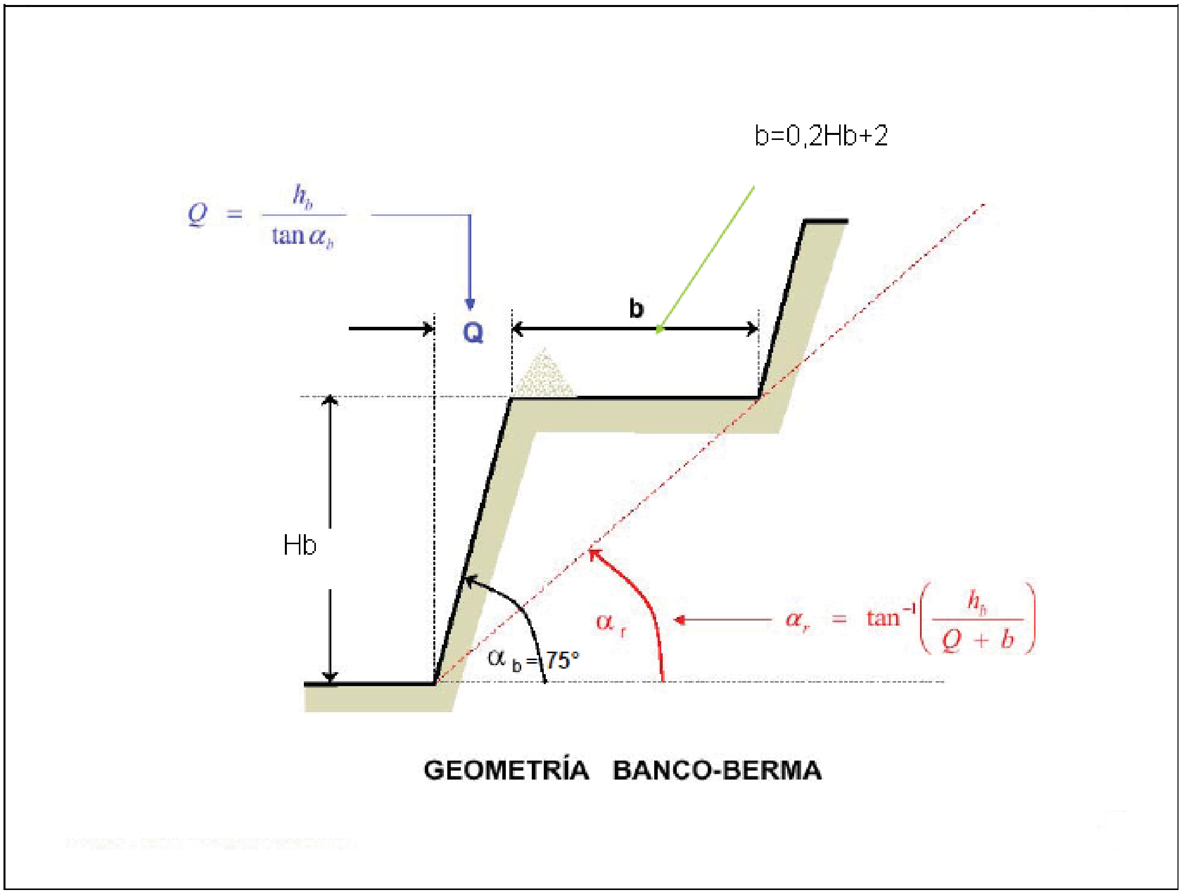
b= 0,2*10+2

b= Berm width = 2.4m,

For a bench height of 5m, the recommended berm width is 3.0m.

Figure 24.10 shows the parameters that define the bench-berm geometry.

Figure 24.10: Bench-Berm Geometry Parameters



Based on the bench angle and height, adopting and applying the Ritchie Criteria in the berm width definition, the inter-ramp angle is defined as:

For the NW wall with bench face angle of 75º at the same height of 5m, the inter-ramp angle is:

Hb=5 m

Q= Hb/tan α_b

Q= 2.3

b = 3,0m,

α_r=66°

The geometric configuration for the Phase I and II pits, for their respective slopes, assumes approximately 8-9 benches, 10m high with inter ramp angles of up to 66°. In the detailed design stage, the definitive design will be established.

Bench Face Angle = 75°

Bench Height = 10m

Berm Width = 3,0 m

Ramp Width = 5.7 m

Overall Slope Height, Phase I=100 m

" " " Phase II – 70m

24.1.4 GEOTECHNICAL CONCLUSIONS AND RECOMMENDATIONS

- The rock mass of the Phase I and Phase II Berta Sur pits, according to the core logging of the three drill-holes that are located in both pits, calculated by MRMR presents rocks that have regular to very good geotechnical quality.
- The geotechnical units and qualities defined by logging of the drill-holes corroborate the units observed in the field.
- The overall slope angles of 50º for the pit walls are within the accepted ranges for MRMR geotechnical qualities for the rock in Berta Sur Pits.
- Based on the stability analysis results, slopes with bench face angles of 75º and overall angle of 50º are stable for both Regions of design.
- Security Factors are all greater than FS>1,0, both for static and pseudo-static analyses.
- It is recommended to carry out sampling for laboratory tests to further characterize the rocky massif and component structures resistance.
- It is necessary to complete 3 or 4 drill holes with the objective of further defining the geotechnical qualities of the rocks, including sampling and laboratory analysis of rock mass resistance and more detailed structural interpretation
- It is recommended to maintain a permanent geological, geotechnical and structural control of the pits focussing on the Berta Sur structures, via mapping. This will allow for improved operational performance during the mine’s life.
- During the operation drilling and blasting (pre-cut and load density for the different types of rocks) must be strictly controlled, because this work is fundamental to conserve the slopes of Berta Sur Pits.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 GENERAL

The Amended Updated PEA includes a revised open pit mine plan, new operating and capital costs and financial analysis for the Berta project which contemplates the treatment of 13.1 Mt of mineral material within the proposed pit boundary by conventional heap leach, solvent extraction and electrowinning methods to produce an average of 4,700ktpy of copper cathode for a period of 8 years. Phase I includes the trucking of high grade material to the Nora Plant, while Phase 2 assume leaching all material at the Berta site.

The mineral resources here described are located in mining claims originally optioned to MCC and transferred to SCMB, which has rights to acquire 100% of the property. The acquisition of the property is contingent upon making the underlying option payments.

The economic results of this PEA, at a constant $2.80/lb copper price, are summarized in Table 25.1.

Table 25.1: Summary of Project Economics

	Revised Mine Plan		
	Phase 1	**Phase 2**	**LOM**
Copper Price	US$2.80/lb		
Copper Production	2,988	34,833	37,821
Duration	11 months	7 years	8 years
Cash Costs	$1.75/lb	$1.57/lb	$1.59/lb
CAPEX ($million)	$7.15	$12.6	$23.0[1]
Pre-tax:			
NPV (8%)	$46.4 million		
IRR	83%		
After-tax			
NPV (8%)	$35.2 million		
IRR	75%		

Readers are advised that more detailed engineering studies have not been completed for the Berta project and so the normal progression from PEA to Preliminary Feasibility Study to Feasibility Study has not been followed in respect of making a production decision. Therefore, investors are cautioned that no mineral reserves have been declared and the level of confidence in the resources, metallurgy, engineering and cost estimation is not at a level normally associated with a project reaching a production decision.

25.2 GEOLOGY AND MINERAL RESOURCES

The geology of the Berta Sur and Berta Central deposits are reasonably well understood, in terms ofgenesis, mineralization controls and structure. Copper oxide mineralizationextends to depths of 30 to 100 m with mineralization outcropping at surface and with effectively no overburden. It also has a simple mineral material and gangue mineralogy, excellent response to leaching and fairly continuous Cu grades and sharp contacts with low-grade margin mineralization.

To separate the zones with different statistical behavior, solids were constructed torepresent two mineralization types: Oxide body and Low gradeOxide body.Metallurgical test work considered copper grades for both type of mineralization.

This Berta report model is based on 22,213 m of drilling, mainly reverse circulation (RC) and mostly drilled by MCCin three stages completed during 2011 and 2012. Other drill holes included in the resource estimate were completed during the 1990's by Empresa MineraMantos Blancos S.A. (Anglo American) and Outokumpu anddiamond drilling completed by Grandcru in 2006 and 2007. Drilling and sampling procedures, sample preparation and assay protocols for all the drilling campaigns were verified and found to be generally acceptable and all available information was used in the resource evaluation without limitation.

The resource estimate was completed in September 2013 at a variety of total copper (%CuT) grades, as shown on Table 25.2 below.

Table 25.2: Resource Estimate

Berta Project Resource Estimate													
Zone	Cutoff	Measured			Indicated			Measured & Indicated			Inferred		
		kt	% CuT	% CuS	kt	% CuT	% CuS	kt	% CuT	% CuS	kt	% CuT	% CuS
Berta Sur & Central	0.10	16,498	0.34	0.23	8,653	0.23	0.14	25,150	0.30	0.20	4,845	0.24	0.15
	0.15	13,275	0.39	0.27	5,780	0.27	0.18	19,055	0.36	0.24	3,249	0.30	0.20
	0.20	10,487	0.45	0.31	3,336	0.35	0.23	13,822	0.43	0.29	2,039	0.38	0.25
	0.25	8,355	0.51	0.36	1,961	0.44	0.30	10,316	0.50	0.35	1,402	0.45	0.31
	0.30	6,791	0.56	0.40	1,289	0.52	0.36	8,080	0.56	0.39	932	0.53	0.37
Berta Sur	0.10	10,972	0.32	0.21	4,423	0.18	0.11	15,394	0.28	0.18	2,105	0.18	0.11
	0.15	8,853	0.37	0.25	2,800	0.21	0.13	11,653	0.33	0.22	1,296	0.22	0.13
	0.20	6,892	0.42	0.29	1,332	0.26	0.16	8,225	0.39	0.27	720	0.26	0.16
	0.25	5,385	0.47	0.33	561	0.31	0.20	5,946	0.46	0.32	343	0.29	0.18
	0.30	4,288	0.53	0.37	261	0.36	0.24	4,549	0.52	0.36	127	0.33	0.21
Berta Central	0.10	5,526	0.38	0.26	4,230	0.27	0.17	9,756	0.33	0.22	2,740	0.29	0.19
	0.15	4,422	0.45	0.31	2,980	0.33	0.22	7,402	0.40	0.27	1,953	0.35	0.24
	0.20	3,594	0.51	0.36	2,003	0.41	0.27	5,598	0.47	0.33	1,318	0.44	0.30
	0.25	2,969	0.57	0.40	1,401	0.49	0.34	4,370	0.55	0.38	1,059	0.50	0.34
	0.30	2,503	0.63	0.45	1,028	0.56	0.39	3,531	0.61	0.43	805	0.57	0.40

Geoinvestment considered the basis for determining the reasonable prospects for eventual economic extraction of the Berta Sur and Central resources by completing a series of pit optimizations using the Lersch & Grossmann algorithm based on the following technical and economic parameters; mining cost of $2.09/t, processing Cost of $4.74/t ,SXEW cost of $0.102/lb, G & A cost of $0.045/lb , sales & marketing cost of $0.041/lb, metallurgical recovery of 80% (based on results obtained from the metallurgical test work), inter ramp pit slope of 50o , and a variety of copper prices. For a base case using a $3.00/lb copper price, and a 0.1%CuT cut off grade, the optimum pits were determined to contain Measured and Indicated Resources of 17.6 million tons at a grade of 0.37%CuT and an overall stripping ratio of 0.49:1, as detailed in Table 25.3 below.

Table 25.3: In pit Resource Estimate based on $3/lb Cu, 0,1% CuT cutoff

Berta Project In Pit Resource												
Zone	Pit	Measured			Indicated			Measured & Indicated			Waste	Strip
		kt	% CuT	% CuS	kt	% CuT	% CuS	kt	% CuT	% CuS	kt	Ratio
Berta Sur	Berta Sur	8,929	0.35	0.23	1,427	0.19	0.11	10,356	0.33	0.21	2,609	0.25
Berta Central	Trinchera-Salvadora	2,242	0.48	0.30	527	0.47	0.29	2,769	0.48	0.30	2,499	0.90
	Carmen-Gemela	982	0.51	0.36	562	0.38	0.26	1,544	0.47	0.32	1,852	1.20
	Nueva	219	0.43	0.29	295	0.34	0.22	514	0.38	0.25	375	0.73
	Berta II	853	0.37	0.24	150	0.36	0.23	1,003	0.37	0.24	572	0.57
	Chico	900	0.30	0.18	518	0.25	0.14	1,418	0.29	0.17	762	0.54
Berta Sur & Central	**Total**	**14,125**	**0.38**	**0.25**	**3,479**	**0.29**	**0.18**	**17,604**	**0.37**	**0.23**	**8,669**	**0.49**

This Amended updated PEA is further optimizing the project using the new operating parameters shown Table 25.4 Mine plan also assumes a first phase using a variable cut-off grade in year 1 of between 0.60% and 0.70%CuT, in order to maintain a constant feed to the existing Nora crusher for a period of 11 months, thus postponing part of the capital investment until year 2 of operations. A total of 0.4mt at 0.83%Cu will be mined and trucked to the Nora plant while 1.2mt of lower grade heap leach material and 0.6mt of ROM will be stockpiled for processing in year 2. In addition, the Nora plant will reprocess some Ripios stockpiles from the previous 2009-12 operation at a rate of ~30 tpm of copper cathode during Phase 1 as described in section 17 of this chapter.

Phase 2 after eleven months considers all the copper oxide material from the open pits will be treated through a heap leach process with capacity of 1 million tonnes of mineral material per year (including crushing, agglomeration and permanent pads), and the processing of 1.2 million tonnes per year of Run of Mine (ROM) material directly onto dump leach pads.

Table 25.4: Design Criteria and Mine Planning

Variable	BERTA	NORA	ROM
Mining Cost (USD/ton)	2.32	2.32	2.32
Hauling (USD/ton)	0.00	0.00	0.00
Processing Cost (USD/ton)	7.91	12.29	1.82
SX-EW Cost (USD/lb)	0.250	0.250	0.250
G&A (USD/lb)	0.090	0.090	0.090
Selling Cost (USD/lb)	0.050	0.050	0.050
Recovery	78.0%	78.0%	45.0%
Selling Price	$3.00	$3.00	$3.00

The final optimized pit contains 7.2 million tonnes @ 0.547%CuT of heap leachable material, 6.6 million tonnes @ 0.20%CuT of ROM and 7.1 million tonnes of waste, as shown below in Table 25.5 by sector. This represents a mining recovery of 89.4% of the heap leach resources and 38.8% of the ROM resources contained in the Berta resource estimate.

Table 25.5: Pit Optimization by Sector.

Sector	HL Material	CuT%	CuS%	ROM	CuT%	CuS%	Waste	Total
Berta Sur	4.178.240	0,529	0,375	4.175.360	0,203	0,122	1.448.139	9.801.739
Trinchera-S	1.130.880	0,786	0,560	1.096.000	0,186	0,111	2.663.429	4.890.309
Carmen-G	786.240	0,588	0,422	314.880	0,196	0,117	1.931.798	3.032.918
Nueva	223.360	0,567	0,401	205.440	0,209	0,126	271.977	700.777
Berta II	509.760	0,522	0,367	308.160	0,204	0,123	434.526	1.252.446
Chico	395.200	0,492	0,343	533.440	0,196	0,117	434.770	1.363.410
Total	**7.223.680**	**0,574**	**0,407**	**6.633.280**	**0,200**	**0,120**	**7.184.640**	**21.041.600**

25.3 METALLURGICAL TESTWORK

A total of twelve 2m column tests have been completed on material from Berta Sur at Geomet SA and three 2m columns from Berta Central material at the Hydrometallurgy Laboratory of the University of Santiago de Chile, and this testwork was used to estimate recoveries of 78% of total copper for the heap leach and 45% of total copper for the dump leach material.

25.4 MINING

The Project contemplates an open pit mine to extract oxide material from the Berta Sur and Central deposits using mining contractors, followed by crushing, agglomeration and heap leaching of higher grade (>0.3%CuT) material and dump leaching of lower grade (0.1-0.3%CuT) material. The resulting pregnant leach solution (“PLS”) would then be transported by 6"-54kmpipeline to the Nora SXEW plant for recovery of copper cathode. Water and raffinate would be returned by 10"-54km pipeline from Nora to Berta. Overall material contained in the mine plan developed by Geoinvestment has 7.22mt of heap leach material, with an average grade of 0.57% CuT and 6.63mt of dump leach material with an average grade of 0.20%CuT. Annual average material movements represent a strip ratio of approximately 0.52:1 waste: ore.

This Amended updated PEA considers a Phase 1 using a variable cut-off grade in year 1 of between 0.60% and 0.70%CuT, in order to maintain a constant feed of material trucked to the Nora crusher for a period of 11 months.

The Berta mine plan & cathode production schedule is shown on Table 25.6 below;

Table 25.6: Berta Mine Plan

Production Profile		Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Tot
Nora Crushed	Ton	399.258	-	-	-	-	-	-	-	**399.258**
	CuT%	0,83	-	-	-	-	-	-	-	**0,83**
	CuS%	0,61	-	-	-	-	-	-	-	**0,61**
	Rec%	80,97	-	-	-	-	-	-	-	**81,0**
	Cu Cathode, t	2.673	-	-	-	-	-	-	-	**2.673**
Ripios Line	Cu Cathode, t	315								**315**
Berta Crushed	Ton	84.932	1.002.740	1.000.000	1.000.000	1.000.000	846.925	1.000.000	828.737	**6.763.334**
	CuT%	0,55	0,51	0,55	0,50	0,51	0,79	0,61	0,48	**0,56**
	CuS%	0,39	0,36	0,39	0,35	0,36	0,56	0,43	0,34	**0,40**
	Rec%	0,79	0,78	0,78	0,77	0,77	0,78	0,79	0,77	**0,78**
	Cu Cathode, t	366	4.025	4.271	3.838	3.945	5.241	4.783	3.074	**29.544**
Berta ROM	Ton	109.353	1.537.653	1.163.006	975.679	598.340	490.499	470.580	603.613	**5.948.725**
	CuT%	0,18	0,20	0,21	0,19	0,19	0,19	0,21	0,20	**0,20**
	CuS%	0,11	0,12	0,13	0,11	0,11	0,11	0,13	0,12	**0,12**
	Rec%	45	45	45	45	45	45	45	45	**45**
	Cu Cathode, t	90	1.387	1.101	812	501	408	440	541	**5.280**
Total Cu	Cu Cathode, t	3.444	5.412	5.372	4.650	4.446	5.650	5.223	3.615	**37.812**
Stockpiled Material										
Berta ROM	Ton	499.882	499.882	499.882	499.882	499.882	499.882	499.882	499.882	**499.882**
	Cu Cathode, t	490	490	490	490	490	490	490	490	**490**
Berta Leach	Ton	1.090.174	732.799	957.612	650.361	254.530	(0)	115.396	- 0	**- 0**
	Cu Cathode, t	4.036	2.281	2.922	1.896	742	- 0	330	- 0	**- 0**

Equipment and support facilities for this schedule have been costed on a mine contractor basis. The main installations for maintenance will be composed of a maintenance shop dimensions for a fleet of 3 trucks of 25 ton capacity, 2 front end loaders, 1 bulldozers, 1 grader, 1 wheeldozer, 1 water truck and 1 drill rig, representing the most cost effective option.

25.5 INFRASTRUCTURE

At the Nora Plant, power supply will be obtained from the existing electrical grid through a local distributor EMELAT that has confirmed feasibility for conecting to the existing power line. At the Berta mine site power will be supplied by 1.75Mw diesel generators.

Water will be sourced from the CODELCO owned Pampa Austral tailing dams, located 10km north of Nora Plant. The Berta mine site water requirement will be supplied by 10"-54km pipeline.

Sulphuric acid may be sourced from CODELCO´s Potrerillo smelter located 85km to the northwest of Berta mine site or from ENAMI´s Paipote Smelter located 110 km to the south.

25.6 ENVIRONMENTAL AND SOCIAL ISSUES

The Evaluation Commission of the Atacama Region of Chile, part of the Chilean Environmental Evaluation Service (in Spanish, "SEA"), has approved the EID of the Berta copper project and has emitted the corresponding Resolution of Environmental Qualification (in Spanish, "RCA") on 14 October 2014 (See Annex 1)

The RCA for the Nora plant was granted in July 31, 2008. There are no anticipated social issues.

25.7 ECONOMIC AND FINANCIAL ANALYSIS

Operating Costs

Operating cost estimates reflect the current market environment in northern Chile for contract mining, crushing, sulphuric acid, power supply, cathode production by SXEW, and transportation of PLS and water, and are shown on Table 25.7 below.

Principal operating cost components are sulphuric acid at $94/t and power at $222/MW for Berta (generators) and $117/MW for Nora (connected to grid).

Table 25.7: Life of Mine Operating Costs

	$´000			$/lb		
Operating Costs	**Phase 1**	**Phase 2**	**LOM $m**	**Phase 1**	**Phase 2**	**LOM $m**
Mining	2,653	38,700	41,353	0.40	0.50	0.50
Processing	5,478	71,334	76,811	0.83	0.93	0.92
Transport	2,276	2,942	5,218	0.35	0.04	0.06
G&A	1,143	7,762	8,906	0.17	0.10	0.11
Cash Costs C1	**11,549**	**120,739**	**132,288**	**1.75**	**1.57**	**1.59**

Capital Costs

Life of mine capital costs are shown on Table 25.8

Table 25.8: Life of Mine Capital Costs

Area No	AREA TITLE	TOTAL $´000	Yr1 $´000	Yr2 $´000	Yr3 $´000	Yr4 $´000	Yr5 $´000	Yr6 $´000	Yr7 $´000	Yr8 $´000
10	NORA PLANT PURCHASE & STARTUP	5,761	6,467	219						-925
20	BERTA CONSTRUCTION	6,375		6,375						
30	NORA EXPANSION	1,324		1,324						
40	PIPELINE PLS & RAFF/WATER	3,773	107	3,666						
50	Other Owner Cost									
	Sustaining Capex	1,308	163	163	163	163	163	163	163	163
	Option Payments	2,300	574	1,156	570					
	Closure Costs	2,200								2,200
	GRAND TOTAL	23,040	7,311	12,903	733	163	163	163	163	1,438

Pre- Financing Financial Analysis

The Project has been evaluated on both a pre-tax basis and after all Chilean taxes and a 1.5% royalty due to the Berta claim owner at a base case copper price of $2.80/lb and for sensitivity, at prices of $2.60/lb and $3.00/lb as shown on Table 25.9 The project economics contemplated by this Amended Updated PEA are summarizedon Table 25.10

Table 25.9: Berta Economic Evaluation Summary

Cu Price	$2.60/ lb		$2.80/ lb		$3.00/ lb	
NPV ($ millions)	Pre tax	After tax	Pre tax	After tax	Pre tax	After tax
5%	42.3	32.3	55.2	41.8	68.1	52.1
8%	35.1	26.8	**46.4**	**35.2**	57.7	44.3
10%	31.1	23.7	41.5	31.5	51.8	39.9
IRR	62%	56%	**83%**	**75%**	106%	98%

Table 25.10: Summary Economics

	Revised Mine Plan			Prior
	Phase 1	**Phase 2**	**LOM**	**LOM**
Copper Price	US$2.80/lb			US$3.00/lb
Copper Production	2,988	34,833	37,821	38,400
Duration	11 months	7 years	8 years	8 years
Cash Costs	$1.75/lb	$1.57/lb	$1.59/lb	$2.03/lb
CAPEX ($million)	$7.15	$12.6	$23.0(1)	$20.3
Pre-tax:				
NPV (8%)	$46.4 million			$34.3m
IRR	83%			55.2%
After-tax				
NPV (8%)	$35.2 million			$26.6m
IRR	75%			46.9%

Readers are advised that more detailed engineering studies have not been completed for the Berta project and so the normal progression from PEA to Preliminary Feasibility Study to Feasibility Study has not been followed in respect of making a production decision. Therefore, investors are cautioned that no mineral reserves have been declared and the level of confidence in the resources, metallurgy, engineering and cost estimation is not at a level normally associated with a project reaching a production decision.

26.0 RECOMMENDATIONS

Sufficient metallurgical test work has been completed for a PEA. However, a detailed assessment of the mine plan and testing of specific samples based on the early years of production is recommended in phase 1.

For Berta Central, which will be exploited towards the end of the mine life in this plan, further drilling is necessary to investigate if more HG material is available for continuing the strategy of initial capital deferring. Also test work is necessary to confirm the anticipated metallurgical performance.

There still some potentially available dump material within trucking distance of the Nora plant which should be evaluated as feed for the plant in early stagephase 1 and when Berta is being developed.

An alternative to the diesel generators proposed for mine site power supply could include solar power generation, similar to those currently being built in the area, and this should be evaluated.

Despite the execution of initials agreements, itis recommended that SCMB should conclude a sulphuric acid contract with either of the smelters located in the region.

Disclaimer and Risks

The opportunity to acquire the plant and accept the financing occurred while SCMB was finalizing the PEA for the Project and this has been modified to incorporate the proposed plant acquisition and revised capital and operating costs. There has been insufficient time to complete more detailed engineering studies and so the normal progression from PEA to Preliminary Feasibility Study to Feasibility Study has not been followed.

Therefore, investors are cautioned that no mineral reserves have been declared and the level of confidence in the resources, metallurgy, engineering and cost estimation is not at a level normally associated with a project reaching a production decision. This may result in the production rates, copper recoveries and operating costs stated in this PEA not being realized.

Geoinvestment's assessments are preliminary in nature, mineral resources are not mineral reserves and do not have demonstrated economic viability, and there is no assurance the preliminary assessments will be realized. The outcome of this PEA may be materially affected by the closing of the financing, copper pricing, environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. Inferred mineral resources are considered too speculative geologically to have economic considerations applied to them that enable them to be categorized as mineral reserves.

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28.0 DATE AND SIGNATURE PAGE

The undersigned prepared this Technical report, titled Geology and Minerals Resources Estimation of Berta Project, Inca de Oro, Chile with an effective date of September 24th, 2015. The format and content of the report are intended to conform to Form 43-101F1 of National Instrument 43-101 (NI 43-101) of the Canadian Securities Administrators.

Signed

September 24th, 2015

29.0 ANNEXES

Annex1: Resolution of Environmental Qualification Notification, COREMA of Atacama, October 14, 2014.

ANNEX 1

: RESOLUTION OF ENVIRONMENTAL QUALIFICATION NOTIFICATION, COREMA OF ATACAMA, OCTOBER 14, 2014.

Servicio de Evaluación Ambiental

Gobierno de Chile

ACTA DE NOTIFICACION PERSONAL

En la ciudad de Copiapó, a 14 del mes de Octubre de 2014, en dependencias de la Dirección Regional del Servicio de Evaluación Ambiental de la Región de Atacama, domiciliada en calle Yerbas Buenas N° 295, Copiapó, concurre personalmente a notificarse de la Resolución de Calificación Ambiental N°236 de fecha 14 de Octubre de 2014 de la Comisión de Evaluación de la Región de Atacama, don Marcelo Claudio Cortes Pantoja, cédula nacional de identidad N°10.375.586-7, en representación de Sociedad Contractual Minera Berta del Proyecto "Proyecto Berta".

Para constancia del presente acto se hace entrega de copia íntegra de la Resolución de Calificación Ambiental N°236 de fecha de 14 de Octubre de 2014 de la Comisión de Evaluación de la Región de Atacama del Proyecto "Proyecto Berta".

MARCELO CLAUDIO CORTES PANTOJA

COMISIÓN DE EVALUACIÓN
SECRETARIO
REGIÓN DE ATACAMA

MARCO ANTONIO CABELLO MONTECINOS
SECRETARIO
COMISIÓN DE EVALUACIÓN
REGIÓN DE ATACAMA