# Altar Project, San Juan Province, Argentina

NI 43-101 Technical Report
October 4, 2010
Amended March 21, 2011

Prepared by: Nilsson Mine Services Ltd. & Geosim Services Inc.

# TECHNICAL REPORT ALTAR PROJECT, SAN JUAN PROVINCE, ARGENTINA

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**OCTOBER 4, 2010**

**AMENDED MARCH 21, 2011**

**IMPORTANT NOTICE**

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# 1.0 EXECUTIVE SUMMARY

This updated technical report on the Altar Project has been prepared by Nilsson Mine Services Ltd. (NMS), Geosim Services Inc. (Geosim) and Hydrometal Inc. (Hydrometal) at the request of Peregrine Metals Ltd. ("Peregrine", "Peregrine Metals" or the "Corporation"), which is based in Vancouver B.C., Canada. The report was written in compliance with disclosure and reporting requirements set forth in the Canadian Securities Administrators' National Instrument 43-101, Companion Policy 43-101CP, and Form 43-101F1. In general, the information in this report is current as of October 4, 2010. An updated resource estimate is provided based upon the current geological interpretation and exploration results of the past year.

This report was amended on Mar 21, 2011 to include details on the Rio Cenicero concessions that were not included in the previous report. Several statements clarifying the arsenic content of the mineral resource have also been added.

## 1.1 Introduction

The Altar Project (or the "Project") is a copper ± gold ± molybdenum porphyry deposit located in San Juan Province, Argentina, approximately 10 km from the Argentina–Chile border and 180 km west–southwest of the city of San Juan.

This project was the subject of earlier reports prepared by NMS-Geosim (September 2008 and October, 2009) and AMEC Americas Limited (June, 2007). Sections of these earlier reports have been referenced directly in this report or retained for continuity where only minor changes have taken place in the past year.

The Project is currently accessed via 170 km of gravel road leading westward from the nearest settlement, the town of Calingasta, along the Rio Calingasta. There is no rail or air access to the Project. The closest airports are in the cities of San Juan and Mendoza (220 km southeast of the Project). The site is very remote, and has no local infrastructure apart from the road constructed into the Property.

The climate is continental semi-arid. Due to the location in the High Andes, the exploration field season is normally restricted to the six-month period from November to April.

The Altar Project consists of six mining concessions, one exploration permit and two mining rights of way (servidumbres). It also includes an option on the five adjacent Rio Cenicero concessions. The six mining concessions collectively cover an area of about 4,924 ha and the Rio Cenicero concessions cover an additional 3,705 ha. The Corporation's interest in the six mining concessions was held under an option agreement with Rio Tinto Mining and Exploration Ltd. ("Rio Tinto"). The Corporation exercised the option by completing a cash payment to Rio Tinto of US$1,650,000 that was due on or before July 20, 2008. Pursuant to a transfer agreement dated March 6, 2009 between Rio Tinto and Minera Peregrine Argentina S.A. ("Minera Peregrine"), a subsidiary of the Corporation, in accordance with Argentinean law, the rights and obligations with respect to the six mining concession and the two servidumbres (the "Rio Tinto Interests") were transferred from Rio Tinto to Minera Peregrine. Rio Tinto retains a 1% net smelter return ("NSR") royalty in respect of the Rio Tinto Interests, referred to herein as the “Rio Tinto Royalty”. Pursuant to an underlying agreement with Rio Tinto, which has been superseded by a notification of assignment dated March 10, 2009, Juan Carlos Robledo and Otto Wilko Simon also hold an NSR royalty of 1% (referred to herein as the “Robledo Royalty”) in respect of certain concessions known as Loba, Santa Rita, RCA II and RCA VII), which the Corporation has the right to purchase at any time for US$1,000,000. If a mine is not in production by April 21, 2010, or if the Corporation has not previously purchased the Robledo Royalty, then payments of US$80,000 per annum must be made to Robledo and Simon until commercial production is achieved. On the date of commencement of commercial production, the annual payments cease, and the Robledo Royalty becomes due. The annual payments are in addition to, and not an advance on, the Robledo Royalty.

Rio Tinto provided an exploration-level Environmental Impact Assessment in 2003, updated it in 2005 through 2007. Peregrine provided an updated baseline environmental study in November 2009.

On August 14, 2008 Minera Peregrine signed an option agreement (the “Rio Cenicero Option Agreement”) to acquire a 100% interest in exploration and exploitation rights to the 3,705 hectare Rio Cenicero concessions from the “Instituto Provincial de Exploraciones y Explotaciones Mineras” (“IPEEM”) of the Province of San Juan, Argentina. Four out of the five Rio Cenicero concessions adjoin the core properties comprising the Altar Project on three sides to the east, south and west. According to the terms of the agreement there is a five year exploration term after which Minera Peregrine has the option to convert its 100% interest in the concessions from exploration to exploitation rights. During the exploration phase the requirements to keep the option in good standing are:

- Option payments to IPEEM of US$2,500 per month.
- US$1.7 million in total exploration expenditures over five years from the date of signing.

Commencing with the exercise of the exploitation option and the signing of the exploitation agreement, the requirements to keep the option in good standing are option payments to IPEEM of US$7,500 each month up until commencement of commercial production, after which the option payments cease and an NSR royalty of 1% is payable on all mineral products from the Rio Cenicero concessions.

## 1.2 Geology and Mineralization

The Altar mineralized system is developed at the unconformable contact of Permo-Triassic Choiyoi Group basement and Tertiary volcanic rocks. It is associated with Miocene intermediate-composition porphyries that intrude rhyolitic ignimbrites and fine-grained andesite flows. Alteration comprises a core of potassic alteration, which in the upper levels of the deposit has been overprinted and almost obliterated by a later porphyry-related sericite event. The sericitic alteration passes outwards into little-altered ignimbrite or chloritized andesitic volcanics, both of which were subjected to varied degrees of supergene kaolinization as a result of acid attack during pyrite oxidation. The outer limit of jarositic limonite after pyrite was mapped and defines the Altar hydrothermal system to be some 3.5 km x 3 km in surface dimensions. The Project contains porphyry-style copper and lesser gold and molybdenum mineralization, deposited in an environment that transitions from the basal section of a high-sulphidation epithermal lithocap to a copper porphyry environment.

Stockwork and disseminated copper mineralization is associated with the sericitic and potassic alteration phases, and is hosted in a range of rock types, including porphyry, andesite and andesitic volcanic to hydrothermal breccias. Sulphide minerals generally exhibit a consistent vertical zonation pattern: minor pyrite–enargite at the higher levels; pyrite–chalcocite–covellite–bornite–digenite assemblages at intermediate levels; and pyrite–chalcopyrite–bornite and molybdenite assemblages at deeper levels. There is minor supergene chalcocite, but very little secondary copper oxide mineralization.

## 1.3 Exploration History

Work completed prior to Peregrine's involvement in the project included:

- From 1995–1996, undertaken by CRA Exploration Argentina S.A. (CRA), construction of 18 km of new access road, reconnaissance stream sediment sampling (36 samples), rock chip sampling (485 samples), talus fines sampling (491 samples), geological mapping and acquisition of heli-borne magnetic geophysical data.
- From 1999–2003, Rio Tinto completed geological mapping (1:10,000 scale), alteration studies using Aster imagery, a ground magnetic survey (line spacing of 100 m and sensor height of 2 m), diamond drilling (total of 2,841 m in seven widely-spaced holes), reverse circulation drilling (four widely-spaced holes), and petrographic examination of selected diamond core samples.

From 2005–2009 Peregrine completed geological, alteration and structural mapping (1:5,000 scale), an induced polarization (IP) and resistivity survey (23.4 line-km with 200 m dipole spacing), and diamond drilling (3,302 m of drilling in eight holes). A further 25 drill holes for 10,408 m were completed by Peregrine from January to April

2007. Peregrine completed another 24 drill holes (and deepened one pre-existing hole) for 12,741 m between January and April 2008. Mapping and sampling was also carried out on the Rio Cenicero concessions bordering the property to the south, west and east.

From Jan to May 2010 Peregrine completed 76 core holes (26,030 m) and deepened 2 previous holes (318 m). Trenching and sampling were also carried out on peripheral targets.

## 1.4 Drill Hole and Assay Database

The Altar drill hole database consists of 140 core holes totalling 55,640 m drilled since 2003 by Rio Tinto and Peregrine Metals. Core recovery has been excellent averaging 98%. A total of 26,392 intervals have been assayed. The database includes interval tables for lithology, alteration, degree of oxidation, mineralogy and vein and fracture intensity.

QA/QC sampling and analysis for the drilling programs has been ongoing during the course of the exploration. Current procedures have been clearly documented by Peregrine Metals. These procedures and results have been independently reviewed. The results from 2010 have also been reviewed by Geosim prior to development of the resource estimate.

## 1.5 Metallurgical Testing

An initial set of density determinations was completed in 2006 using ACME Analytical Laboratories, with a follow-up program in 2007-2008 using Alex Stewart (Assayers) Ltd. The initial metallurgical work was also undertaken in 2007. This included preliminary flotation testwork that was conducted by Dawson Metallurgical Laboratories. Due to the limited amount of material, flotation parameters were not fully optimized, especially with respect to final concentrate grades. Initial copper bottle roll leaching studies were separately conducted by McClelland Laboratories, Inc.

Both the flotation and leaching studies focused on composites that represented two of the major sulphide ore types: an enriched material high in covellite (designated CC-CV) and a lower grade primary material dominated by chalcopyrite (designated CP-BN).

In 2010-2011 the metallurgical program has been greatly expanded and is currently underway. This work includes column leach studies (now being completed) and further flotation studies (in preparation stage, to begin in second quarter, 2011) on a range of representative samples. It also included cyanide bottle roll leach tests to recover gold from the oxidized cap that overlays the sulphide material. Finally, the 2010 testing included initial comminution studies and water solubility testing. Most of these activities have been undertaken at McClelland Laboratories, Inc. The comminution work was done at Phillips Enterprises LLC. In 2011 a more detailed SAG mill/ball mill comminution and flotation optimization study has been undertaken at G&T Metallurgical Services. A summary of the findings from the 2007 non-optimized testing program and the 2010 metallurgical program is provided as follows:

- The non-optimized, single, 2007, CC-CV composite had a head grade of 0.86% Cu and produced a final concentrate that assayed 26% Cu and 0.2% As at an overall copper recovery of 90%.

- The non-optimized, single, 2007, CP-BN composite had a head grade of 0.56% Cu and produced a final concentrate that assayed 17% Cu and 0.6% As at an overall copper recovery of 89%.

- Mineralogical examination results indicated that the sulphide minerals were well liberated at the primary grind size used. Important copper minerals were chalcopyrite, covellite, enargite with minor tetrahedrite and bornite.

- The bottle roll tests on a very limited selection of material indicated that the low arsenic CC-CV ore type should be a good candidate for a heap leach operation, subject to confirmatory column leach tests.

- Crushing work indices ranged from 5.9 to 7.9 kW-h/mt and show that the Altar ore types range from very soft to moderately soft and should be easy to crush.

- Ball mill grinding work indices ranged from 11.8 to 13.4 kW-h/mt and show that the Altar ore types have moderate grindability by copper industry standards.

- The abrasion indices for the Altar ore types ranged from 0.07 to 0.15 and show that most materials should cause only limited abrasion on metal surfaces.

- Specific gravity tests on whole and crushed core showed that the Altar ore types have a high degree of internal porosity and void space, which averages about 12%. This should allow good penetration of leach solution into the interior of the ore fragments.

- The oxidized leach cap responds well to the cyanide bottle roll tests. Gold extraction was generally rapid and exceeded 80% in half the samples. However, there was some falloff in extraction as depth increased. Reagent consumption during the gold leaching was generally low.

- All Altar samples tested contained some water soluble material that averaged nearly 2%. The major water soluble constituent is anhydrite ($CaSO_4$), but other soluble metal sulphates and chlorides are also present in many samples. These water soluble species will build up in either a mill circuit or a leach circuit. As a result, water treatment may be required to avoid scaling or corrosion problems.

- A variety of column leach tests are nearing completion to assess how ore lithology, head grade, crush size and copper solubility affect copper extraction and acid consumption. Because the tests are not yet complete, it is difficult to draw final conclusions. However, none of the composites being tested has had as high a leach rate as the original bottle roll tests. This may be largely a matter of the much coarser ore size in the column tests.

## 1.6 Resource Estimate

The Resource Estimate was jointly prepared by Nilsson Mine Services Ltd. and GeoSim Services Inc. of Vancouver, Canada, utilizing drill results for the 140 core holes (55,640 metres) drilled at Altar to date. Copper and gold grades were estimated by ordinary kriging constrained by an optimized pit shell using metal prices of $2.80 per pound of copper and $850 per ounce of gold. Block dimensions were 15 by 15 by 15 metres. Assays were composited in 10 metre down-hole intervals. Copper grades were capped at 5% and gold grades at 0.3 g/t prior to compositing to remove outlier values.

Wireframe models of the leached cap and major lithologies were created based on cross sectional interpretation. The density values assigned to the major lithologies were based on 1,877 bulk density measurements of drill core.

The model was validated by comparing to nearest neighbour estimated and composite grade distributions, swath plots and visual inspection of sections and plans. Resources were classified as measured, indicated or inferred based upon a number of constraints including, zone, drilling density, elevation and distance to nearest composite.

Resources are summarized in the table below for a range of cutoff grades. The cutoff grade selection in operating copper mines is sensitive to a large range of economic factors and can be impacted by processing and mining method selection and scale. Cutoff grades are not static during a typical mine life and hence a resource estimate that provides a summary over a range of cutoff grades is useful in assessing potential development options. The Altar deposit may be amenable to both bulk open pit and underground mining methods. Portions of the deposit are amenable to conventional milling and copper flotation to concentrate and may be amenable to heap leach / solvent extraction processing as well. A preliminary economic assessment is required to develop trade off studies of the various mining and processing options available.

The cutoff grade of 0.3% total copper equivalent is considered a reasonable starting point for reporting the global mineral inventory at Altar at this level of study.

**Table 1-1 Altar Project mineral resource summary**

| Cut-off | Quantity | Grade | | | Contained Metal | |
|---|---|---|---|---|---|---|
| Copper Equivalent (%) | Tonnes (Millions) | Copper (%) | Gold (g/t) | Copper Equiv (%) | Copper (Billion lbs) | Gold (Million oz) |
| **MEASURED** | | | | | | |
| 0.2 | 669 | 0.38 | 0.057 | 0.40 | 5.62 | 1.22 |
| **0.3** | **491** | **0.43** | **0.061** | **0.45** | **4.69** | **0.96** |
| 0.4 | 278 | 0.51 | 0.070 | 0.53 | 3.12 | 0.62 |
| 0.5 | 126 | 0.61 | 0.082 | 0.63 | 1.69 | 0.33 |
| **INDICATED** | | | | | | |
| 0.2 | 541 | 0.33 | 0.050 | 0.34 | 3.92 | 0.87 |
| **0.3** | **311** | **0.40** | **0.057** | **0.41** | **2.72** | **0.57** |
| 0.4 | 137 | 0.49 | 0.061 | 0.51 | 1.47 | 0.27 |
| 0.5 | 60 | 0.56 | 0.058 | 0.58 | 0.74 | 0.11 |
| **MEASURED+INDICATED** | | | | | | |
| 0.2 | 1,210 | 0.36 | 0.054 | 0.37 | 9.54 | 2.09 |
| **0.3** | **802** | **0.42** | **0.059** | **0.44** | **7.41** | **1.53** |
| 0.4 | 414 | 0.50 | 0.067 | 0.52 | 4.59 | 0.89 |
| 0.5 | 186 | 0.59 | 0.074 | 0.62 | 2.43 | 0.44 |
| **INFERRED** | | | | | | |
| 0.2 | 906 | 0.33 | 0.053 | 0.34 | 6.53 | 1.56 |
| **0.3** | **465** | **0.42** | **0.058** | **0.44** | **4.32** | **0.88** |
| 0.4 | 252 | 0.50 | 0.054 | 0.52 | 2.80 | 0.44 |
| 0.5 | 135 | 0.57 | 0.048 | 0.58 | 1.69 | 0.21 |

*The copper equivalent ("CuEq") calculation is based on a copper price of $2.80/lb and gold price of $850/oz. It also includes a factor to compensate for an assumed gold recovery of 65% and a 90% recovery for copper.

## 1.7 Conclusions & Recommendations

An updated evaluation of the exploration programs and results available to the effective date of this report indicates that:

- The geology is sufficiently well understood to support the resource estimation presented in this report and summarized in the section above.
- The oval-shaped sulphide-bearing portion of the 3.5 km x 3 km Altar hydrothermal system has dimensions of about 2.9 km x 1.7 km, based on IP data.
- The deposit remains open at depth; the deepest drill hole to date, ALD-43, reached 1009.9 m and remains in mineralized rock.
- Data collection to May 2010 at the drill site is acceptable.
- Drill holes from all programs intersected a leach cap ranging from 0 m to 258 m in thickness. This was underlain by a zone of primarily sulphide mineralization that is

variably affected by supergene enrichment generally within 105 m of surface. Sulphide mineralization thicknesses range from about 75 m to at least 805 m which is at the vertical limit of the present drilling.

- Average grades for all the assays returned as of the report effective date were 0.26 % Cu, 0.07 g/t Au and 0.002% Mo. Copper grades ranged from below detection to 12.38% Cu.
- The database contains all diamond drilling data collected on the project to date and has been structured for resource estimation.
- QA/QC with respect to the results received to date for the Peregrine 2010 exploration program is acceptable and protocols have been well documented.
- Preliminary metallurgical test work has been completed. The utilized methodologies are in line with industry best practice, and are appropriate for the deposit type.
- Assaying of cyanide and acid soluble copper has been conducted in areas where there may be potential for the application of solvent extraction and electro-winning process technology.

Studies to support a Preliminary Economic Assessment (PEA) are presently underway and all recommendations pertaining to this from the previous Technical Report are being followed.

Follow-up work on the QDM target including trenching, sampling and initial drilling are already being planned.

Additional recommendations arising from this Technical Report are as follows:

- Additional drilling is warranted to define the lateral extents of shallower copper mineralization to the south and east which could reduce the overall strip ratio.
- Further work to develop a gold heap or dump leach process for gold-bearing leached cap material which would need to be removed as part of the pre-stripping.

# 2.0 INTRODUCTION

Peregrine Metals Ltd. ("Peregrine Metals", the "Corporation" or "Peregrine")) requested that Nilsson Mine Services Ltd. (NMS), Geosim Services Inc. (Geosim) and Hydrometal Inc. (Hydrometal) provide a Technical Report update on the Altar Project (the Project), located in San Juan Province, Argentina.

The current work by NMS-Geosim-Hydrometal entailed the preparation of a Technical Report as defined in NI 43–101 and in compliance with Form 43–101F1 (the "Technical Report").

The Qualified Persons responsible for the preparation of the Technical Report are:

- John Nilsson P.Eng – Nilsson Mine Services Ltd.
- Ronald Simpson P.Geo – Geosim Services Inc.
- W. Joseph Schlitt, P.Eng – Hydrometal, Inc.

This Technical Report references two previous reports prepared by the Nilsson-Geosim titled "Altar Project, San Juan Province, Argentina, Preliminary Resource Estimate" dated September 1, 2008 and "Technical Report – Altar Project, San Juan Province, Argentina" dated October 20, 2009. It also references an earlier report prepared by AMEC Americas Limited (AMEC) entitled "Altar Project San Juan Province, Argentina Report on Exploration", June 30, 2007. Sections of this earlier AMEC report have been referenced in this report or retained for continuity where only minor changes have taken place in the past three years.

This report was amended on Mar 21, 2011 to include details on the Rio Cenicero option that were recently obtained. Several statements clarifying the arsenic content of the mineral resource have also been added, as well as a description of the expanded metallurgical program.

## 2.1 Terms of Reference

NMS, Geosim, and Hydrometal are independent of Peregrine Metals, and have no beneficial interest in the Altar Project. 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.

In preparing this report, the authors relied on geological maps, reports and miscellaneous technical data listed in the References section at the conclusion of this report.

John Nilsson P.Eng, Ronald Simpson P.Geo., and W. Joseph Schlitt P.Eng. conducted a site visit to the Altar Project from 19 to 22 March 2010. Nilsson and Simpson also visited the site from 18 to 20 March 2008. The purpose of the visits was to review the geology and mineralization encountered on surface and in the drill holes completed to date. In addition, drilling, sampling, quality assurance/quality control (QA/QC), sample preparation and analytical protocols and procedures, and database structure were reviewed. The site visits also included a tour of the core storage facility and the sample preparation laboratory in Mendoza operated by Acme Analytical Laboratories.

W. J. Schlitt has made multiple visits to McClelland Laboratories and Dawson Metallurgical Laboratories during the course of the metallurgical testing,

The Effective Date of the Technical Report is October 4, 2010.

All measurement units used in this report are metric, and currency is expressed in United States dollars unless stated otherwise.

# 3.0 RELIANCE ON OTHER EXPERTS

## 3.1 Introduction

The authors of this Technical Report, confirm that they are qualified persons for those areas as identified in the "Certificates of Qualified Persons" attached to this report. The authors have relied, and believe there is a reasonable basis for this reliance, upon the reports referenced below, which provided information regarding mineral rights, surface rights, permitting, and environmental issues in sections of this Technical Report.

## 3.2 Mineral Tenure

The authors have not reviewed the mineral tenure, nor independently verified the legal status or ownership of the Project area or underlying property agreements. The authors have relied upon the AMEC summary and Peregrine Metals experts for this information through the following documents.

- Peregrine Diamonds Ltd., 2005: Property Option Agreement between Rio Tinto Mining and Exploration Limited and Peregrine Diamonds Ltd ("Peregrine Diamonds"): confidential option agreement between Rio Tinto and Peregrine Diamonds, dated 20 April 2005 (Section 4.3 of this report).
- Peregrine Diamonds Ltd., 2005: Property Option Agreement between Rio Tinto Mining and Exploration Limited and Peregrine Diamonds, Schedule F, Underlying Option Agreement: confidential option agreement between Rio Tinto and Peregrine Diamonds, dated 20 April 2005, (Section 4.3 of this report).
- Allende and Brea, 2005: Altar Project 27 April 2005: letter summarizing due diligence findings from Buenos Aires-based lawyers Allende and Brea, addressed to Peregrine Diamonds, dated 27 April 2005 (Sections 4.3 of this report).
- Peregrine Metals Ltd., 2005: Acquisition Agreement between Peregrine Diamonds and Peregrine Metals: confidential acquisition agreement, dated 22 September 2005 (Section 4.3 of this report).
- Peregrine Diamonds Ltd., 2006: Amendment of Altar Option Agreement, 18 October 2006: confidential amendment to option agreement between Rio Tinto and Peregrine Diamonds, dated 18 October 2006 (Section 4.3 of this report).
- Toohey, J., 2007a: Altar 20 April 2007 Payment of US$800,000 to Rio Tinto: internal email to AMEC regarding option payments as at 31 May 2007, dated 29 May 2007 (Section 4.3 of this report).

- Koffman Kalef, 2007: Peregrine Metals Ltd – Altar Property: letter summarizing vesting of rights in and to Altar Property, from Koffman and Kalef lawyers, addressed to Peregrine Metals, dated 12 June 2007.
- Allende and Brea, 2007: Altar Project 3 July 2007: email summarizing status of title of Altar Project from Buenos Aires-based lawyers Allende and Brea, addressed to Peregrine Diamonds, dated 3 July 2005 (Sections 4.3.1 and 4.3.2 of this report).
- Toohey, J., 2008: Altar 08 July 2008 Payment of US$1,650,000 to Rio Tinto: internal email to NMS regarding option payments as at 08 July 2008 (Section 4.3 of this report).
- Transfer Agreement between Rio Tinto and Peregrine, March 6, 2009 regarding the Rio Tinto interests.

## 3.3 Surface Rights, Access and Permitting

The authors have relied on information regarding Surface Rights, Access and Permits, including the status of the granting of surface rights by the Province of San Juan for land designated for future potential mining, milling, dumps and tailings impoundments supplied by Peregrine's representatives as follows:

- Allende and Brea, 2005: Altar Project 27 April 2005: letter summarizing due diligence findings from Buenos Aires-based lawyers Allende and Brea, addressed to Peregrine Diamonds, dated 27 April 2005 (Section 4.3.3 of this report).
- Toohey, J., 2007: Permits: internal email to AMEC detailing status of permits as at 31 May 2007, dated 31 May 2007 (Sections 4.3.3 and 4.5 of this report).
- Allende and Brea, 2007: Altar Project 3 July 2007: email summarizing status of title of Altar Project from Buenos Aires-based lawyers Allende and Brea, addressed to Peregrine Diamonds, dated 3 July 2005 (Sections 4.3.3 and 4.5 of this report).

## 3.4 Environmental and Socioeconomics

The authors have relied upon opinions of experts retained by Peregrine Metals, through the following documents:

- Allende and Brea, 2005: Altar Project April 27, 2005: letter summarizing due diligence findings from Buenos Aires-based lawyers Allende and Brea, addressed to Peregrine Diamonds, dated 27 April 2005 (Section 4.4 of this report).

- Parizek, B., 2005: Primer Actualización Bianual y Segunda Auditoria Ambiental Proyecto Altar: Environmental Impact Assessment document prepared by Vector Argentina S.A. for Rio Tinto, February 2005 (Section 4.4 of this report).
- Allende and Brea, 2007: Altar Project 3 July 2007: email summarizing status of title of Altar Project from Buenos Aires-based lawyers Allende and Brea, addressed to Peregrine Diamonds, dated 3 July 2005 (Sections 4.4 of this report).
- Vector Argentina 2008: Calidad de Aire Proyecto Altar, March, 2008
- Vector Argentina 2008: Estudio de Línea de Base - Resumen ejecutivo, October 2008
- Vector Argentina 2008: Estudio de Línea de Base - Calidad de Aire, August, 2008
- Vector Argentina 2009: Estudio de Línea de Base – Calidad de Agua, Revision 01, February 2009
- Vector Argentina 2008: Estudio de Línea de Base – Esudio Hidrogeológico Etapa I, August, 2008
- Vector Argentina 2008: Estudio de Línea de Base – Esudio Hidrogeológico Etapa II, August, 2008
- Vector Argentina 2008: Estudio de Línea de Base – Limnología, August, 2008
- Vector Argentina 2008: Estudio de Línea de Base – Flora, August, 2008
- Vector Argentina 2008: Estudio de Línea de Base – Fauna, August, 2008
- Vector Argentina 2008: Estudio de Línea de Base - Arqueología, August, 2008

# 4.0 PROPERTY DESCRIPTION AND LOCATION

## 4.1 Location

The Altar Project is located in Argentina (Figure 4-1), within the San Juan Province, about 1 km from the Argentina–Chile border (Figure 4-2), and some 180 km west-southwest of the city of San Juan.

The Project is centered on approximate coordinates 6517000N and 2359800E with datum set to Argentina Campo Inchauspe Zone 2. Elevations within the Project area range between 3,300 masl and 4,000 masl. The centre of the deposit is at an elevation of about 3,400 masl.

## 4.2 Overview of Argentina

The Republic of Argentina is located in the southeastern portion of South America. Argentina is bordered to the south and west by Chile and to the north by Bolivia, Paraguay and Brazil. From north to south, the east side of Argentina is bordered by Brazil, Uruguay and the Atlantic Ocean (Figure 4-2).

Argentina is the second-largest country in South America after Brazil and the eighth largest in the world. The population of the country is about 39.5 million; approximately 16 million live in and adjacent to the capital city, Buenos Aires (CIA, 2007).

### 4.2.1 Metal Mining in Argentina

Historically metal mining has not played a dominant role in Argentina's economy, but this situation has changed during the last five years. While industrial minerals and building materials accounted in the past for nearly two thirds of the total mining production, Argentina's gold production increased to 1.1 Moz of gold in 2003, becoming the 14$^{th}$-largest world producer (fourth in Latin America, with 7% of the gold output of the region; Torres, 2004). Since then, large gold mines Gualcamayo and Veladero have begun mining, and Cerro Morro, Agua Rica, Pascua-Lama and El Pachon are moving through development.

Argentina is one of three producers of primary aluminium in Latin America, accounting approximately for 12% of production. The country was Latin America's third leading producer of lead (after Peru and Mexico) and steel (after Brazil and Mexico). It was the fourth leading producer of copper (after Chile, Peru, and Mexico) and primary iron and pig iron (after Brazil, Mexico, and Venezuela). Argentina was the fifth leading producer of silver (Torres, 2004).

**Figure 4-1 Argentina Location Plan**

**Figure 4-2 Project Location Plan**


Note: Figure from Peregrine Metals Ltd.

Argentina and Chile signed an agreement in 1997 regulating the mining activities on shared border zones (Tratado de Integración y Complementación Minera Chileno-Argentina, the "Mining Integration Treaty"). Under this agreement, exploration and mining have dramatically increased (Torres, 2004). The most recent development in this sense corresponds to a protocol signed in January 2006 to facilitate the exploration of the Amos–Andrés gold-copper-molybdenum and the Vicuña gold projects, both located along the border between Central Chile and Argentina.

### 4.2.2 Mining Industry and Legislation

Information in this section is taken from Garcia (2007) and Torres (2004) and has not been independently verified by AMEC, NMS or Geosim.

The Argentine Mining Code, which dates back to 1886, is the legislation which deals with mining in the country. Special regimes exist for hydrocarbons and nuclear minerals. In the case of most minerals, the Mining Code dictates that the owner of the surface is not the owner of the mineral rights; these are held by the State. The State is also bound by the Code to grant to whoever discovers a new mine the rights to obtain a “mining concession”.

Owners must comply with three conditions: payment of an annual fee, investment of a minimum amount of capital, and the carrying out of a reasonable level of exploitation. Failure to do so could lead to forfeiture of the property back to the State.

The administrative organization for mining – specific regulation, at Federal level, is the Federal Ministry of Planning, Public Works and Investment, which has a Mining Department headed by the Secretary of Mines. At Provincial level, there are mining departments, or mineral courts, depending on the jurisdictions, that deal with the granting of exploration permits, mining concessions and have jurisdiction on mining permitting, in general. The Argentine Mining Code is a federally drafted law implemented by all Provincial governments under the National Constitution of Argentina. Argentine Provinces retain sole jurisdiction on matters of procedural regulations, but cannot change the Mining Code.

In 1980, an amendment recognized the need for modernizing the production and classification of minerals, size of individual mining areas (to encourage development of low-grade deposits) and elimination of miners’ rights to encourage foreign companies to engage in mining operations through public tenders.

Between 1993 and 1995, Argentina implemented a new Mining Investment Law (No. 24,196, as amended), a Mining reorganization Law (No. 24,224), a Mining Modernization Law (No. 24,498), a Mining Federal Agreement (No. 24,228), and financing and Devolution pf IVA (Law No. 24,402). Decree 456/97 implemented a unified text of the Mining Code with all amendments made by the aforementioned legislation. These amendments offered attractive incentives for exploration and mining to foreigners, and include both financial and tax guarantees, such as import duty exemptions, unrestricted repatriation of capital and profits and a 3% cap on Provincial royalties. This group of laws also creates the basis for federal-provincial harmonization of the procedural regulations.

In 2001, Law 25.429 “Update of the Mining Investment Law” was passed, and in March 2004 approval was reached for a key provision of the Law allowing refund of the IVA (or value added tax) for exploration related expenses incurred by companies registered under the Mining Investment Law.

In 1995, Law No 24.585 Environmental Protection (Mining Code) was passed and provides regulation for operations and environmental reporting at the exploration and exploitation levels.

In summary, the major changes to the Mining Code encompass:

- Exploration areas have been increased to a maximum of 100,000 ha per company and per province.

- Exclusive aerial prospecting areas of 20,000 $km^2$ are also permitted.
- A guarantee of tax stability for 30 years has been granted.
- Expenditures made in prospecting, exploring and construction of mining installations are tax deductible and value added taxes are recoverable.
- Imports of capital goods, equipment and raw material are exempt from import duties.
- Royalties will not exceed 3% of the ex-mine value of the extracted mineral.
- Environmental funds to correct damage are required and are deductible from income taxes; a National system of permanent mining environmental monitoring is set up. Implementation at the provincial level has been variable and in 2004–2005 San Juan province began to increase staffing for monitoring purposes.
- Municipal taxes on mining were eliminated.
- Systemization and digital conversion of mining property registers has been implemented to varying degrees of success in each province and the definition by geographic co-ordinates now establishes mining rights.

In addition to the legal changes discussed above, the Argentine government has taken political steps to raise the profile of mining within the domestic industrial regime. A position of Secretary of Mines has been established, giving a seat at federal cabinet to the government's mining portfolio. This change was also made at the provincial level in San Juan Province in early 2004. The Secretariats are also commissioned to foster mining investment, participate in cooperation between international and inter-jurisdictional departments, and to oversee environmental, labour and hygiene issues related to mining. They respond to and govern initiatives of the National Mining Commission (which supervises the country's mining policy) and oversee the National Geological Service Board (SEGEMAR, which functions as a national Geological Survey).

### 4.2.3 Mineral Property Title

Information in this section is taken from Garcia (2007) and Torres (2004) and has not been independently verified by NMS or Geosim.

Among other functions, the Mining Code constitutes the system to obtain exploration rights or concessions. Characteristics of an exploration concession, referred to as a *cateo*, include:

- Exclusivity – the holder of the cateo has rights to any mineral discoveries, including those made by a third party within the boundaries of the cateo.

- Extent – cateos are measured in 500 ha units, or fractions thereof. No single cateo may exceed 10,000 ha (20 units), and no person may hold more than 200,000 ha (20 cateos) in a single province. The exploration area within a cateo may be contiguous or separated.
- Time – the holder of a cateo must assess the mineral potential within its exploration boundary (and apply for an exploration right) within a time period based on the size of the cateo. The exploration term is 150 days for the first 500 ha. (1 unit) or fraction thereof, and an additional 50 days for each additional unit (or fraction thereof) within the cateo. After 300 days, 50% of the exploration area over 2,000 ha. (4 units) within the cateo must be relinquished. At 700 days, 50% of the remaining area over 2,000 ha. (4 units) must be dropped. Time extensions may be granted to allow for inclement weather, difficult access, etc
- Work – the holder of a cateo must present to the mining authority a minimum exploration work program and schedule. The cateo may be revoked if the requirements of the work program and schedule are not met.

A single-time fee of ARS $400 (400 Argentina Pesos) per 500 ha (one unit) must be paid upon application for a cateo.

The Mining Code also regulates exploitation rights (mining concessions). Priority for receiving a mining concession is given to the registered discoverer of the mine, i.e. the holder of the cateo. A mining concession unit area, or pertenencia, is 6 ha. for some types of minerals (mainly, gold, silver, copper, and, generally, hard rock minerals), in common deposits, and 100 ha. for the mentioned type of minerals if found in disseminated mineral bodies; each mining concession may consist of one or more units. The application to the mining authority must include official cartographic coordinates of the mine location and of the reconnaissance area, and a sample of the mineral discovered. The reconnaissance area, which may be as much as twice the surface area projection of the mine, is intended to allow for the geological extent of the ore body and for site layout and development. Excess area is released once the survey plans are approved by the mining authority.

*Estaca Minas* (Mine Estacas) are 6 ha extensions to the existing surveyed mining concessions that were granted under previous versions of the Mining Code, and are the equivalent to mining concessions. The granting of new estacas was eliminated from the Mining Code in August 1996.

Once the application for a mine has been submitted, the applicant may commence works on the reconnaissance area of the application. Any person, or company, opposed to the application for the new mine, whether a holder of an overlapping cateo, a mining title holder with conflicting claims, a partner in the discovery that claims to have been neglected, among others, may submit his opposition, following publication of the application in the Boletin Oficial or official publication of the Provincial jurisdiction. The person, or company, opposed to the mining concession application must present evidence of his claim to the Provincial mining authority. The Provincial mining authority resolves on the opposition, and such a resolution can be appealed to the Provincial mining law courts.

Within 30 days after the term to file certain statutory exploration works on the reconnaissance area of the mining concession application, the applicant must submit a legal survey of the units (pertenencias) requested for the new mine, within the maximum property limits allowed by the Mining Code. The request is published in the Boletín Oficial and may also be subject to dispute, to be resolved under similar rules as mentioned with regard to opposition to the application for mining concessions. Approval and registration of the legal survey request by the Provincial mining authority constitutes formal title to the mining property.

### 4.2.4 Royalties and Taxes

Rio Tinto retains the Rio Tinto Royalty, being an NSR royalty of 1% on all mineral products from the properties comprising the Rio Tinto Interests (“the “Altar Property”). The original underlying concession owners Juan Carlos Robledo and Otto Wilko Simon (“Robledo and Simon”) also hold the Robledo Royalty, being an NSR royalty of 1% on all mineral products from the mining concessions known as Loba, Santa Rita, RCA II and RCA VII. The Corporation has the right to purchase the Robledo Royalty at any time for a payment of US$1,000,000. If a mine is not in production by April 21, 2010, payments of US$80,000 per annum must be made to Robledo and Simon until commercial production is achieved unless the Corporation exercises its option to purchase the Robledo Royalty. On the date of commencement of commercial production, the annual payments cease and the Robledo Royalty becomes due. The annual payments are in addition to, and not an advance on, the the Robledo Royalty.

Information in this section is taken from Garcia (2007), Beretta and Garcia (2007) and Torres (2004) and has not been independently verified by The authors.

A new mining operation is entitled to national, provincial, and municipal tax exemptions for five years. The exemptions commence with the awarding of formal title to the mine. Additional royalty payments to the government are subject to exemptions of three years as described below.

New exploitation concessions may also be awarded for mines that were abandoned or for which their original exploitation concessions were declared to have expired. In such cases, the first person claiming interest in the property will have priority. A new exploitation concession will be awarded, for the mine in the condition left by the previous holder, after payment of the royalties owed since abandonment or expiry of the previous concession.

The mining operation must fulfill three conditions as part of its exploitation concession:

- payment of mining royalties
- provision of minimum investment
- reactivation of the mine if it is shut down for more than four years, if so demanded by the mining authority.

According to the Mining Investment Law (Law No 24.196), mining royalties in adherent Provinces (San Juan included) are no more than 3% (to be negotiated) of the mineral's mouth-of-mine value. Generally the Provinces determine mining royalty benefits in order to promote added-value activities, according to the following:

- Mining ore only: royalty of no more than 3% (to be negotiated).
- Milling: royalty between 1% and 2%. This category applies to companies producing doré bars and concentrates.
- Final elaboration: between 0% and 1%. This royalty rate range applies to companies making final products such as jewellery.

Mining royalties are paid to the state (national or provincial) under which the exploitation concession is registered, and are paid in equal instalments twice yearly. A mining operation that has not paid its royalty within two months of the due date will be served a notice by the mining authority. The exploitation concession under which the mine operates will expire if the overdue royalty has not been paid within 45 days of the notice.

The royalty is set by national law (presently Law No 24.224 of the Mining Reorganization) according to the category of the mine. In general, the royalty due per year is ARS $80 per 6 ha pertenencia for common ore bodies held by the exploitation concession, or ARS $800 per 100 ha pertenencia for disseminated ore bodies. The discoverer of the mine is exempt from paying royalties for 3 years from the date on which formal title was awarded to the mine.

The holder of the exploitation concession must also commit to investing in the property fixed assets of at least three hundred times the value of the annual mining royalty, over a period of five years. In the first two years, 20% of the total required investment value (i.e., the annual royalty for each year) must be made each year. For the final three years, the remaining 60% of the total required investment may be distributed in another manner. The exploitation concession expires if the minimum required investment schedule is not met.

#### 4.2.5 Surface and Private Property Rights

Information in this section is taken from Garcia (2007) and Beretta and Garcia (2007) and has not been independently verified by The authors.

Access over surface property rights in Argentina is obtained through the Ministry of Mines, which is required to communicate with the surface owners and ensure that they cooperate with the activities of the exploration/mining companies. Notice can be difficult due to delayed filing of personal property title changes and registry as well as limited staffing and mobility of the relevant authorities.

Private property rights are secure rights in Argentina, and the likelihood of expropriation is considered low. The Argentine legal and constitutional system grants mining properties all the guarantees conferred on property rights, which are absolute, exclusive and perpetual. Mining property may be freely transferred and purchased by foreign companies.

### 4.2.6 Environmental Regulations

The Environmental Protection Mining Code of Argentina, enacted in 1996, establishes the guidelines for preparing the environmental impact statement for mining projects. The federal nature of the Argentine government leaves the application of this law to each Province. Initially the provinces adhered to the mining law, and established the provincial mining secretary as the application authority. However, starting in 2002 several of the provinces have re-evaluated their approach to mining and have shifted the environmental criteria and authority to the environmental secretary.

A party wishing to commence or modify any mining-related activity as defined by the Mining Code, including prospecting, exploration, exploitation, development, preparation, extraction, and storage of mineral substances, as well as property abandonment or mine closure activity, must prepare and submit to the Provincial Environmental Management Unit (PEMU) an Informe de Impacto Ambiental or Environmental Impact Assessment (EIA) prior to commencing the work. Each EIA must describe the nature of the proposed work, its potential risk to the environment, and the measures that will be taken to mitigate that risk. The PEMU has a sixty-day period to review and either approve or reject the EIA; however, the EIA is not considered to be automatically approved if the PEMU has not responded within that period. If the PEMU deems that the EIA does not have sufficient content or scope, the party submitting the EIA is granted a thirty-day period in which to resubmit the document.

If accepted by the PEMU, the EIA is used as the basis to create a Declaración de Impacto Ambiental or Declaration of Environmental Impact (DEI) to which the party must agree to uphold during the mining-related activity in question. The DEI must be updated at least once every six months. Sanctions and penalties for non-compliance to the DEI are outlined in the Environmental Protection Mining Code, and may include warnings, fines, suspension of Environmental Quality Certification, restoration of the environment, temporary or permanent closure of activities, and removal of authorization to conduct mining-related activities.

***San Juan Province Environmental Regulations***

Under Argentine Mining Law, the State Mining Secretary (SMS) of the San Juan Province manages the environmental approval system for new mining projects. The applicable evaluation process of the EIA is defined by the Escribanía de Minas (EDM) according to the size of the mining project.

The new Decree of the Provincial Law 1679 SMS, dated October 2006, states that for small and medium mining projects in San Juan Province, the EIA must be presented together with a feasibility study. This allows the SMS to determine the size of the deposit in order to set up the members of the Evaluation Commission, as well as the corresponding terms of reference.

After obtaining an EIA, a mining applicant must apply and obtain various permits and authorizations from the Province of San Juan to proceed with Project development. The permits and authorizations demonstrate compliance with current legislation for the construction and operation of mining operations.

## 4.3 Property Description

### 4.3.1 General

The Altar Project consists of six mining concessions, one exploration permit (Table 4-1, Figure 4-3.) and two mining rights of way (Servidumbres). It also includes an option on the five adjacent Rio Cenicero concessions. The six mining concessions collectively cover an area of approximately 4,924 hectares and the Rio Cenicero concessions cover an additional 3,705 ha. The Project is located less than 10 km east of the Argentina–Chile border, with the westernmost tenement corner of the Leona concession abutting the border.

**Table 4-1 Tenure Details**

| Concession Number | Concession Name | Concession Type | Area (ha) |
|---|---|---|---|
| **Mining Tenure** | | | |
| 1597-C-95 | Leona | Mina | 203 |
| 1598-C-95 | Loba | Mina | 383 |
| 1042-C-96 | Santa Rita | Mina | 1,597 |
| 1118-R-96 | Pampa | Mina | 2,741 |
| Subtotal Mining | | | 4,924 |
| **Rights of Way** | | | |
| 0116-F-28 | | Servidumbre | 30 |
| 0098-F-28 | | Servidumbre | 123 km |
| Subtotal Easements | | | 30 (excludes 0098-F-28) |

Two additional concessions which originally covered the areas of Loba and Santa Rita are not listed in Table 4-1 as they have been superseded by the more recent concessions. These are known as RCA II (338641-I-92) and RCA VII (338646-I-92).

Exploration permit 1124 548-M-08 was staked by Peregrine in 2008.

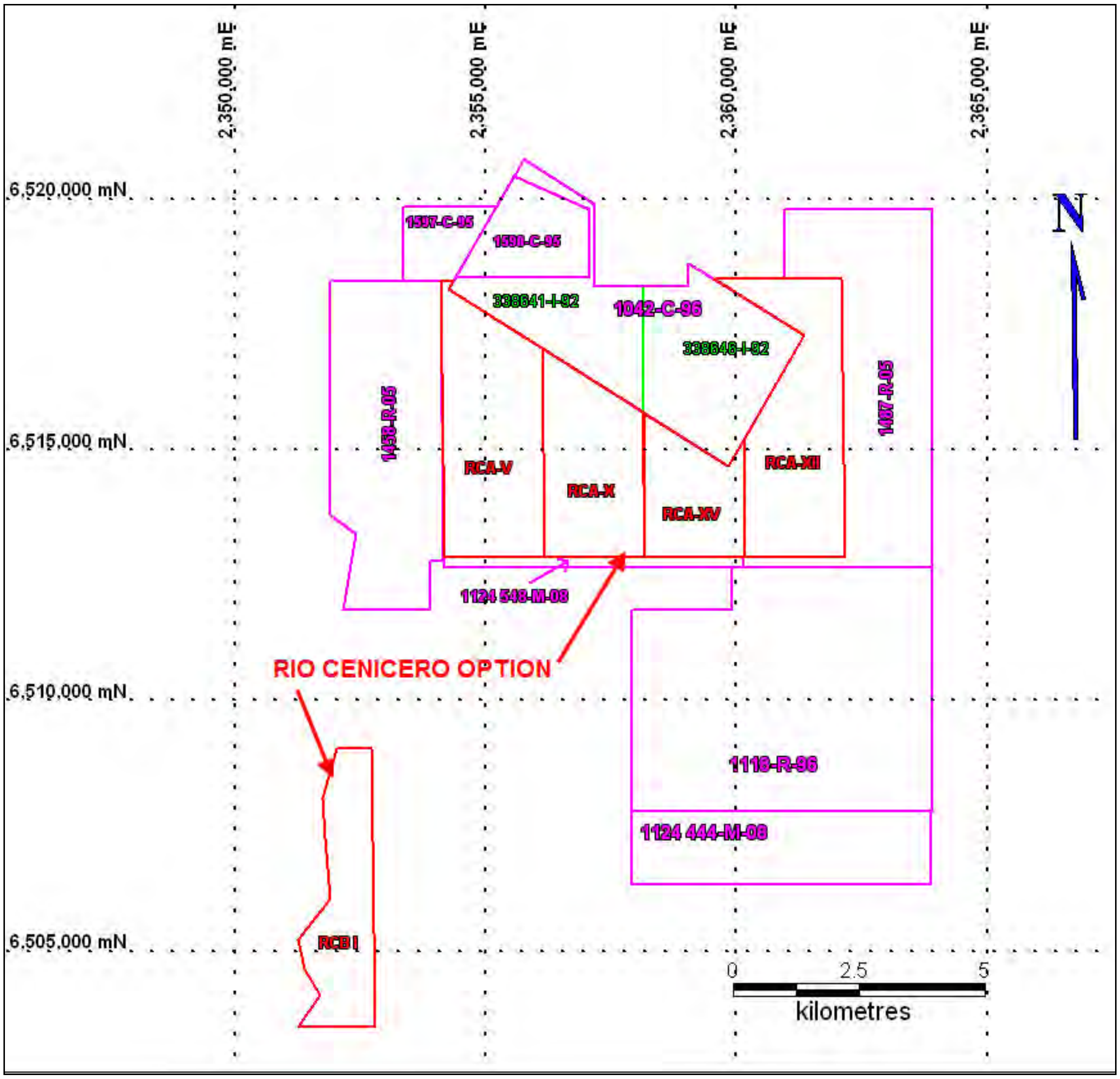
One right-of-way (Servidumbre), 0116-F-28, is about 30 ha in area. Peregrine is in the process of obtaining the documentation for the second, 123 km long, right-of-way number 0098-F-28.

The Altar Project is situated in the eastern portion of the Santa Rita tenement as illustrated in Figure 4-3. Two kilometres to the south is the physically-separated Pampa Mina with an area of 2,471 ha. The exploration camp is situated in the Pampa Mina.

***CRA and Rio Tinto***

On 1 August 2003, CRA Exploration Argentina SA ("CRA") assigned its rights in an option agreement with the Altar Cateo concession owner Juan Carlos Robledo ("Robledo") to Rio Tinto.

**Figure 4-3 Tenure Plan**




### *Rio Tinto and Peregrine Metals*

Under the terms of an option agreement signed 20 April 2005 (the "Altar Option Agreement") between Peregrine Diamonds and Rio Tinto, Peregrine Diamonds had the right to acquire a 100% interest in the Altar Property from Rio Tinto subject to, among other things, the Robledo Royalty. The agreement was amended on 18 October 2006.

To exercise the option, Peregrine Diamonds agreed to assume all of Rio Tinto's obligations to Robledo, and to undertake a series of option payments on a prescribed schedule.

**Peregrine Metals and Peregrine Diamonds**

In January, 2006 Peregrine Diamonds transferred its metals assets, including its interest in the Altar Property option, to a new private company, Peregrine Metals Ltd. subsequent to an Option Agreement signed in September 2005 between the two companies.

The Altar Option Agreement has been subsequently assigned to the Peregrine Metals Ltd. subsidiary company Minera Peregrine Argentina SA under Argentine law.

***Robledo – Rio Tinto Agreement***

The obligations to the original owner, Robledo, comprised:

- A payment of US$70,000 on or before 21 July 2005.
- A payment of US$800,000 on or before 21 April 2007.
- If a mine is not in production by 21 April 2010, then payments of US$80,000 per annum must be made until commercial production is achieved. On the date of commencement of commercial production, the annual payments cease, and a royalty becomes due. The annual payments are in addition to, and not an advance on, the royalty.
- A net smelter return (NSR) royalty of 1% on all mineral products from the Altar Property.

Under the terms of the agreement between Rio Tinto and Robledo, Rio Tinto had the right to purchase the 1% NSR royalty at any time, for a payment of US$1 million. On exercise of the option between Peregrine Diamonds and Rio Tinto, Peregrine Diamonds acquired the NSR royalty purchase right.

On May 23, 2006 Robledo transferred 50% of his rights to the underlying royalty (and the US$80,000 annual payments commencing in April 2010) to Mr. Otto Wilko Simon

Peregrine has confirmed that the original owner obligation payments of US$70,000 due 21 July 2005, and US$800,000 due 21 April 2007, were made by Rio Tinto on behalf of Peregrine Diamonds.

***Rio Tinto Agreement***

The obligations to Rio Tinto under the Altar Option Agreement prior to the amendment on October 18, 2006 were:

- Payment of US$50,000 on completion of a due diligence period.
- Payment of US$50,000 on or before three months following 20 April 2005.

- Payment of US$50,000 on or before the first anniversary of the 20 April 2005 date.
- Expenditure of not less than US$350,000 on or before the first anniversary of the 20 April 2005 date.
- Payment of US$825,000 on or before the second anniversary of the 20 April 2005 date.
- Issue of a number of common shares of Peregrine Diamonds on or before the second anniversary of the 20 April 2005 date. The number of shares is fixed by a formula relating to division of US$825,000 by the market price per common share of Peregrine Diamonds, discounted by 10%.
- An NSR royalty of 1% on all mineral products from the Altar Property.

Peregrine completed the US$50,000 due diligence, US$50,000 20 April 2005, and US$50,000 20 April 2006 payments as per the schedule above. Peregrine also completed the US$350,000 expenditure requirement for the first anniversary period.

Unlike the Robledo agreement, there is no agreement to purchase the 1% Rio Tinto royalty.

***Amendment to Rio Tinto Agreement***

On 18 October 2006, Peregrine amended the agreement with Rio Tinto as follows:

- To exercise the option, Peregrine was required to complete a cash payment to Rio Tinto of US$1,650,000 due on or before 20 July 2008.
- Notification of assignment of Peregrine Diamonds' interest in the option to Peregrine Metals Ltd.

The US$1.65 million payment due 20 July 2008 replaced two clauses in the original agreement—the requirement to pay US$825,000 on or before the second anniversary of the 20 April 2005 signing date and requirement to issue common shares in Peregrine Diamonds on or before the second anniversary of the 20 April 2005 signing date.

Peregrine confirmed that the original owner obligation payment of US$800,000 due April 21, 2007 was made by Rio Tinto on behalf of Peregrine Metals.

On July 9, 2008 Peregrine Metals completed the final US$1,650,000 payment that was due to Rio Tinto by July 20, 2008, thereby exercising the option to acquire a 100% interest in the Altar Property.

On November 25, 2008 Peregrine Metals amended the April 20, 2005 agreement with Rio Tinto assigning Peregrine Metals' interests, rights and obligations with respect to the Altar Property to its Argentinean subsidiary Minera Peregrine. This amendment also established that Rio Tinto Mining and Exploration Ltd. – Sucursal Mendoza ("Rio Tinto Exploration Argentina") was the Argentinean Branch of Rio Tinto and was bound by the terms and conditions of the April 20, 2005 agreement between Peregrine Diamonds and Rio Tinto.

On March 6, 2009, Minera Peregrine signed a title transfer agreement with Rio Tinto Exploration Argentina, in accordance with Argentinean law, that transferred the rights and obligations with respect to the Altar Property from Rio Tinto Exploration Argentina to Minera Peregrine. This agreement also established that the price of the transfer was US$2,670,000 and that this amount was received by Rio Tinto Exploration Argentina previous to the execution of the title transfer agreement.

***Rio Cenicero Option***

The Corporation presently holds an option on five additional mineral concessions, four of which border the Altar Property to the east, south and west and one which lies southwest of the property. The total area covered by these concessions is 3,705 hectares. These concessions are collectively referred to as the "Rio Cenicero concessions" (Table 4-2 and Figure 4-3).

**Table 4-2 Rio Cenicero Mineral Concessions**

| Expediente Number | Name | Type | Area (ha) |
|---|---|---|---|
| 338.644-I-92 | RCA V | Mina | 966 |
| 338.649-I-92 | RCA X | Mina | 709 |
| 338.651-I-92 | RCA XII | Mina | 942 |
| 338.654-I-92 | RCA XV | Mina | 465 |
| 338.637-I-92 | RCB I | Mina | 624 |

The option agreement was signed on August 14, 2008 between Minera Peregrine and the IPEEM of the Province of San Juan. The following terms apply:

Stage 1 – Exploration

5 year term then Peregrine has the option to convert from Exploration to Exploitation rights.

- US$1.7 million in total exploration expenditures over 5 years from date of signing:
- US$100,000 on or before first anniversary date of signing of option agreement
- Additional US$100,000 on or before second anniversary date of signing of option agreement

- Additional US$500,000 on or before third anniversary date of signing of option agreement
- Additional US$1,000,000 on or before fifth anniversary date of signing of option agreement
- Option payments to IPEEM of US$2,500 each month.

Stage 2 – Exploitation

Beginning upon exercise of exploitation option and signing of exploitation agreement:

- Option payments to IPEEM of US$7,500 each month up until commencement of commercial production.
- Upon commencement of commercial production 1% NSR (option payments cease).

### 4.3.2 Tenure History

From 1988 to 2003 the original underlying "Altar" mineral concession was the subject of litigation between its owner Robledo, and the provincial mining company IPEEM. On 21 April 2003, the conflict was resolved by resolution in favour of the concession owners and the property passed to CRA, who held rights under an option agreement signed with the owners in 1995. The Cateo "Altar" has subsequently expired.

Rio Tinto is the successor company to CRA. In 1995, Rio Tinto staked the "Leona" and "Loba" concessions and, in 1996, the "Santa Rita" concession.

Rio Tinto also staked the "Pampa" concession in 1996 to cover a potential exotic copper target and to protect the broad valley area for possible future plant and tailings disposal sites. These subsequent mining applications, as well as the Rio Cenicero concessions, are the current source of exploration and mining rights for the area, as the Altar cateo has expired.

### 4.3.3 Surface Rights

***Camp Easement***

CRA (presently Rio Tinto) applied on 2 February 1996 (# 116-F-28-C-96) for an area of about 30 ha. that could be used as an exploration camp and equipment storage area. On October, 8, 2004, Rio Tinto requested the publication of a notice of the easement claim against the corresponding landowner (M.L. Correa G. de Errázuriz), holding surface rights on the area.

The permit is currently under review of the Provincial state attorneys (at the Mining Department of San Juan) to review compliance with legal formalities. The permit is in the process of being granted. Granting of this easements claim will call for a payment indemnification to the landowners to cover damages to areas covered with surface rights.

***Right of Way***

A 123 km. right- of- way easement was applied for by CRA (currently Rio Tinto) on February 9, 1996, statutory notices were published and the permit is currently pending grant.

Pachon SA Minera (Pachon) objected to the application, and a later objection was filed by IPEEM.

The Pachon objection was a formal objection, but as a copy of an agreement between Pachon and Rio Tinto to share easement rights was filed, it is not considered a material objection. The IPEEM objection has to be resolved; however there is a report from the Legal Department within the Mining Directorate that is favourable to Rio Tinto.

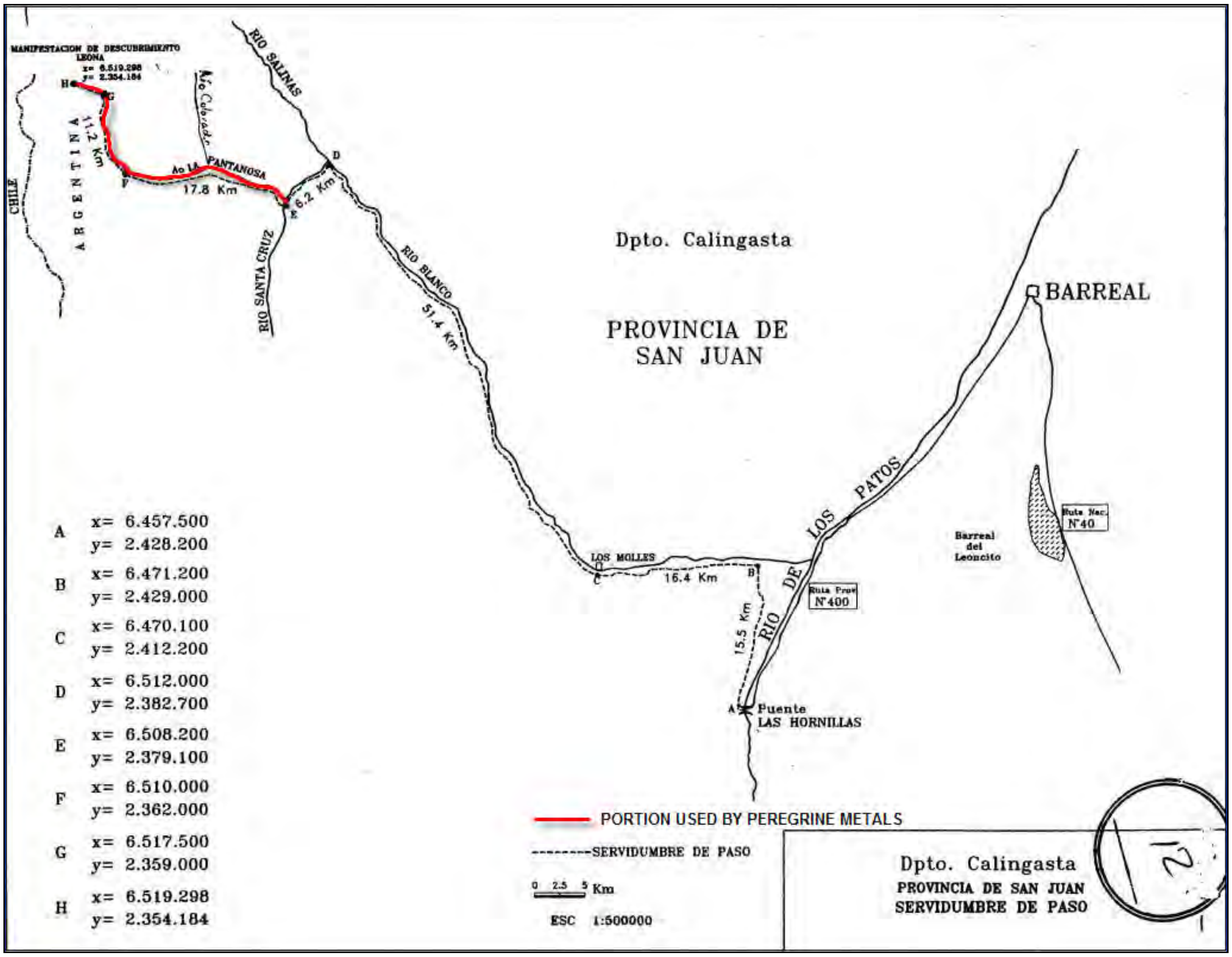
Certain landowners have been identified as being affected by the right-of-way application. The San Juan mining authority is in the course of conducting an inspection the area and a report is pending on such an inspection.

The right-of-way application is currently under review of the Provincial state attorneys (at the Mining Department of San Juan) to review compliance with legal formalities. Granting of this right-of-way easement will call for a payment indemnification to the landowners to cover damages to areas covered with surface rights.

Bonds must be posted once the application is accepted, to cover possible damages to the areas held by the surface rights owners.

A portion of the present right-of-way easement is illustrated in Figure 4-4. Peregrine is endeavouring to locate more comprehensive maps and documentation.

**Figure 4-4 Portion of Road Easement**




## 4.4 Environment

Rio Tinto provided an initial exploration-stage Environmental Impact Assessment (EIA) report on 23 April 2003, and an updated exploration-stage EIA in 2005. The 2005 assessment was completed by Vector Argentina S.A. (Vector; Parizek, 2005). The report covers all claims and concessions that are held by Rio Tinto in the Altar area, and incorporates the easement claims.

Rio Tinto continued to administer the environmental permitting aspects for the project until 2007 year end. As the Altar Option Agreement has been fully exercised as of July 2008, Peregrine now has responsibility for future updates.

In May, 2009, Vector Argentina S.A. completed the baseline environment study on the Altar Project that was begun during the field season of 2008. This report will be submitted to the environmental authorities of the Province of San Juan in November, 2009.

Studies of flora, fauna, limnology, archaeology, and water quality were carried out on the Rio Cenicero concessions between February 22nd and February 27th, 2009 by a team of scientists under the direction of Dr. Jorge Gonnet of Mendoza.

## 4.5 Permits

The Altar Project currently has an approved exploration-stage EIA in place (Section 4.4). The most recent update was for the 2009/2010 field season. Another update to the environmental permit is required for the 2010/2011 field season.

The current water permit allows for extraction of water for drilling purposes for the 2010 calendar year. The Corporation plans to apply for renewal of the permit for 2011.

Peregrine has entered into an agreement to use and maintain in good access conditions a common exploration access road that is shared among several exploration companies working in the region.

## 4.6 Socioeconomics

Politically the Project site lies within the Department of Calingasta, in the Province of San Juan. The population base of the Calingasta Department is characterized as a dispersed rural population with concentration in certain localities.

By population size, the principal localities are Villa Calingasta and Barreal with, in 2005, about 75% of the total Department population of 8,456 residing there.

The Department of Calingasta has a basic or limited economic structure. In general, there is a lack of employment and personal income is low. The main economic activity in the zone is agriculture.

There are no villages or settlements near the Project; the closest community is Calingasta, 170 km by road to the east.

# 5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

## 5.1 Accessibility

The Altar Project is currently accessed via 170 km of gravel road leading westward from the town of Calingasta along the Rio Calingasta. The route crosses the Cordillera de La Totora at the headwaters of the Rio Calingasta, turns southward along the upper tributaries of the Rio Blanco and then westward again along the Rio Pantanosa to the Property (Figure 5-1).

The route to Altar from Calingasta takes six hours by 4x4 pick-up and involves crossing ten rivers and two 4,200 m high mountain passes. Some of the river crossings require regular maintenance with a front-end loader or bulldozer to keep them passable for truck traffic.

Peregrine has identified a viable access route from Chile via the town of Illapel (Figure 5-1). Opening this route would require the construction of approximately 15 km of new road, 5 km of which is in Argentina and 10 km in Chile. Its advantages are that there are no major river crossings or high passes.

## 5.2 Climate

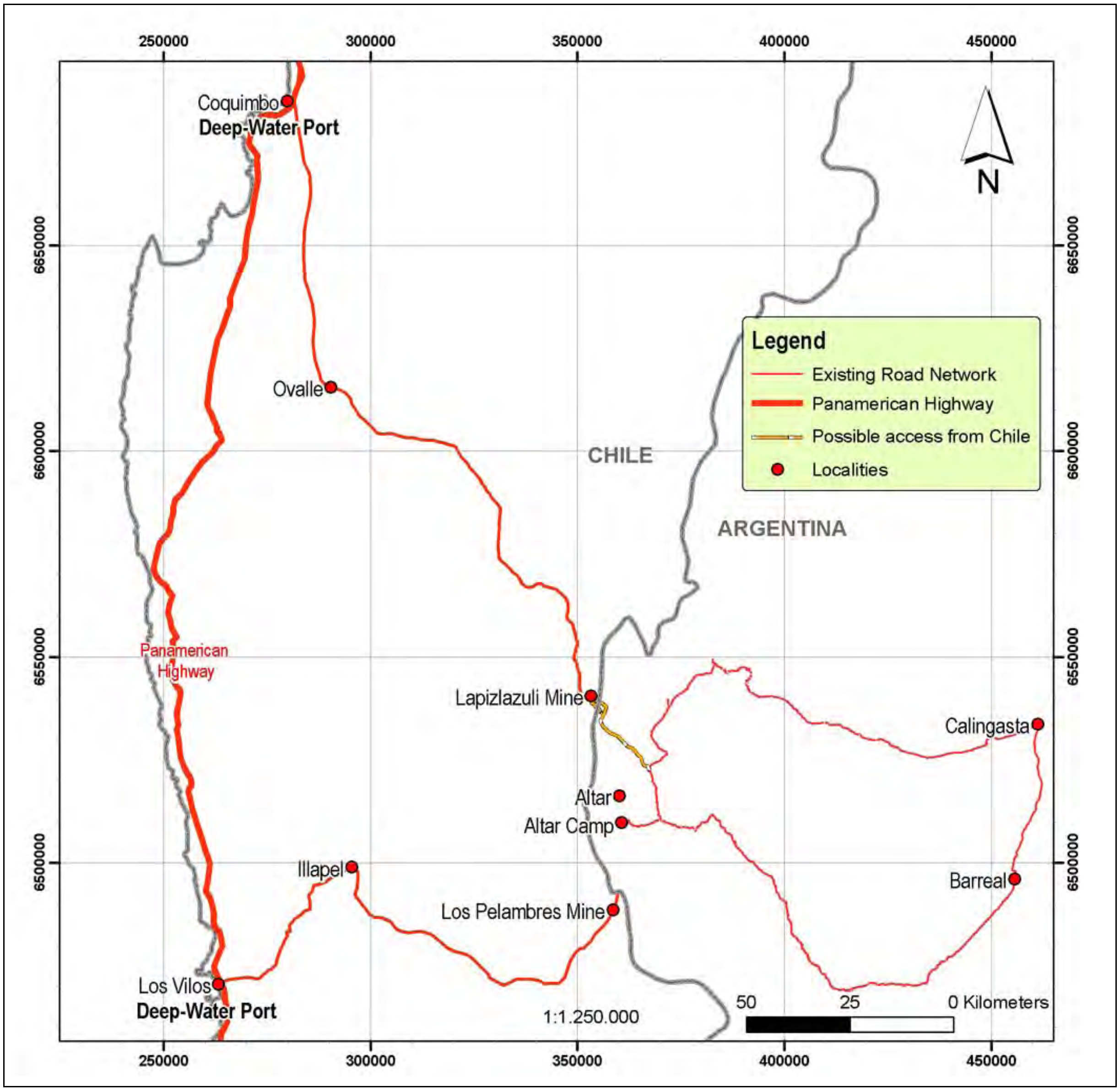
In 2008, Peregrine installed a remote solar-powered weather station at the camp site. Prior to this there was no site-specific weather data collected. The National Meteorological Service has in the past recorded data in the Rio de los Patos valley, and some weather data have been recorded at the El Pachon exploration camp in the Rio Pachon valley, 25 km due southwest of the Project.

The climate is continental semi-arid, characteristic of elevations above 2,500 masl in the Central Andes. Temperatures are low during the entire year ranging from -3°C to 15ºC in summer and from -25°C to 7°C in winter. Precipitation ranges from 600 mm/a to 1,000 mm/a with frequent storms bringing rain and snowfall, along with strong winds, mainly in the winter (May–August). In contrast, the summers are generally dry. Net evaporation rates are high, and exceed annual rainfall by a significant margin.

The Pacific Ocean has a strong effect on the climate of the region. Low pressure centres forming in the eastern Andes cause the movement of air masses from the Pacific eastward through the mountain passes. Storm fronts coming from the west may bring snowfall as early as mid-March. Snowstorms in the region can last for several days.

The exploration field season is normally restricted to the six-month period from November to April.

**Figure 5-1 Road Access Networks**




Note: Figure from Peregrine Metals Ltd.

## 5.3 Local Resources

The nearest centre of population to the Project is Calingasta, a village of approximately 2,000 people, located on the provincial highway connecting Uspallata and San Juan, in San Juan Province. Calingasta offers basic supplies and simple accommodation. It is 170 km by road east of the project.

The closest major population centre to the Project is the city of San Juan (population about 450,000) in San Juan Province, some 180 km to the east–northeast. The city is a major centre providing full hospital services and educational facilities to university level. The Universidad Nacional de San Juan has a century-old mining engineering and geology faculty, as well as diverse science and humanities programs and a medical school.

The most important commercial centre in the region is the city of Mendoza, in Mendoza Province, situated about 220 km southeast of the Project. Mendoza has approximately 773,000 inhabitants, an international airport, as well as a selection of drilling contractors, assay laboratories and accounting and legal services. Mendoza is about a four-hour drive from Calingasta.

## 5.4 Infrastructure

### 5.4.1 Regional

The closest international airports in Argentina are in the cities of San Juan and Mendoza, located 180 km due northeast, and 220 km due southeast of the Project respectively. In Chile, the closest international airport is in Santiago, about 250 km to the due southwest.

There is no rail or air access to the Project. The closest ports are on the Chilean coast, at Los Vilos (120 km due east) and Coquimbo (170 km to the northeast). Los Vilos is the deepwater port currently used by the Los Pelambres mine, 25 km due southeast of the Altar Project, which pumps concentrate via a slurry pipeline from the mine to the coast. On the Argentinean coast, the closest port is at Bahia Blanca, about 700 km due south–southeast of the project.

The Project falls within the Treaty area designated by the Chilean and Argentinean governments (Section 4.2.4) for facilitation of cross-border mining activities. Peregrine has prepared the required documentation and is proceeding with an application to establish a border crossing on the proposed new access road, under the terms of the Treaty.

There is no existing power infrastructure.

### 5.4.2 Local

The site is very remote, and has no local infrastructure apart from the gravel road constructed into the Project. There are no settlements closer than Calingasta.

There is sufficient area within the Pampa Mina for future construction of a plant, related infrastructure, tailings disposal and waste disposal.

Roads provide access to the drill sites and to the ridge tops.

## 5.5 Physiography

### 5.5.1 Relief

The Altar Project is characterised by high-mountain glacial geomorphology with steep-walled cirques and U-shaped valleys. The cirque walls are scree-covered and moraine deposits are common. Rock glaciers are evident at the higher elevations. Nearby mountain peaks reach 4,500 masl. To the east of the property a relatively small number of deeply-incised river valleys are responsible for draining large areas of the high Andes.

### 5.5.2 Water

The Altar Project lies within the upper reaches of the Rio Blanco drainage system. Rio Blanco flows to the southeast from the high Andes, joining downstream with the Rio de los Patos, which flows northward to join the Rio Calingasta and, eventually, the Rio San Juan.

A preliminary limnological survey on Rio Cenicero concessions was carried out in 2009 by Dr. Patricia Peralta. It involved the establishment of 6 monitoring sites on the property. Consideration was given to monitor areas of direct and indirect influence of the project as well as those streams of 1$^{st}$, 2$^{nd}$, 3$^{rd}$ order including swampland tributaries.

According to the results and taking into account the guideline values for human consumption and aquatic biota recommended by the Ministry of Water Resources of the Nation, the waters exceed these levels, especially arsenic. The stream from the project (PMA3) also showed high concentrations of cadmium, copper, iron and zinc.

Nitrate concentration was found to be high, probably due to the high productivity of the plains environment. It is estimated fecal contamination in PMA1, PMA2 and PMA4, due to the presence of Escherichia coli. The source may be due to the livestock (horses and goats) common in the area.

Rio Pantanosa is the local tributary to the Rio Blanco system. It flows eastward, passing a few kilometres to the south of the Altar Project. Internal drainage of the immediate project area is provided by the Arroyo Altar.

It runs from north to south and empties into the broad valley of the Rio Pantanosa about 5 km south of the Altar mineralized system, where it has formed a large alluvial fan. These streams are all permanent.

There are no flow data available; however water levels are very dependent on the season, with the highest flows during spring and summer run-off between November and January.

Arroyo Altar drains the area of the Altar Project and has a low pH as evidenced by the Fe-oxides deposited in the alluvial sediments downstream.

The aquifers in the area have not been studied; however substantial subterranean water resources are expected in valleys such as that of the nearby Rio Pantanosa, which is filled with large volumes of permeable Pleistocene moraine sediments and Holocene gravels.

Water for drilling at the Project has been taken from two small streams in the immediate vicinity of the deposit. Water for the exploration camp comes from the nearby Rio Pantanosa.

### 5.5.3 Flora

A flora survey on the Rio Cenicero concessions was carried out in 2009 under the direction of Dr. Jorge M. Gonnet.

The region belongs to Biome Altoandino or High Andean Ecosystem. It is characterized by shrub steppe vegetation dominated (plants <0.7 m) or mixed grassland or, in more or less dense patches on the slopes. There is a high heterogeneity of microhabitats on the steppes generated by the combination of factors such as exposure, color and size of the clasts from the substrate, slope and water availability. Permafrost is commonly observed on the ridge tops, which are generally devoid of vegetation.

### 5.5.4 Fauna

Fauna and limnological surveys were carried out on the Rio Cenicero concessions during the summer of 2009 under the direction of Dr. Jorge M. Gonnet. A total of 32 species of tererestrial vertebrates were identified through a combination of specimen collection and identification of indirect signs. These included 27 birds, 3 mammals and 1 reptile. Notable was the absence or scarcity of large wild herbivores such as guanacos, Suris and Piuquenes.

A considerable number of livestock were encountered including some 3900 goats and 86 horses along the access road from the Project within 50 km of the camp.

It was concluded that the ecosystem of the valley adjacent to the areas of exploration is highly influenced by livestock. Signs of overgrazing are evident on the steppes and in meadows. Grazing and the presence of domestic dogs place constraints on the selection of optimal foraging habitat and refuge for wildlife.

### 5.5.5 Seismicity

The region of San Juan, including the area of the Project, is in an active tectonic area, having experienced two large-scale earthquakes of magnitude (M) 7.0 or greater, over the last sixty years. In particular, this region had been struck by a M7.4 earthquake in 1944, causing nearly 10,000 casualties and leaving half the inhabitants of the province homeless (Zaragoza, 1999). Similarly, a M7.0 earthquake occurred in 1977, resulting in seventy people killed and up to 40,000 left homeless in Western Argentina. Records indicate that a large-scale earthquake event is recurrent in the region, at intervals consisting of about of 40 to 50 years (Zaragoza, 1999). Better building construction techniques and codes have accounted for the major improvement in death-toll statistics since the early 1900s. All facilities must now be built to withstand Richter M7 earthquakes to Argentine codes equivalent to seismic design UBC4 or better (Zaragoza, 1999).

# 6.0 HISTORY

The Altar deposit was discovered in the mid-1990s by CRA. CRA completed access road construction, surface sampling (rock chip, talus fines and stream sediment), and geological mapping in the period 1995-1996. Geophysical data from a helicopter-borne aeromagnetic and radiometric survey over the property was acquired and interpreted.

Work ceased until mid-1999, when Rio Tinto re-evaluated the area. Rio Tinto completed geological mapping, alteration studies, a ground magnetic survey, and seven diamond drill holes (DDH) for 2,845 m.

Peregrine optioned the property in 2005, and carried out a 23.4 km induced polarization (IP) survey followed by eight DDH totalling 3,302 m during the 2005-2006 summer field season. In the first quarter of 2007, Peregrine carried out a second drilling campaign comprising 25 core holes (10,408 m).

Peregrine carried out a third drilling campaign in the first quarter of 2008 comprising 24 core holes and deepening of one pre-existing hole (12,741 m).

Between January and May 2010 Peregrine completed 76 core holes (26,026 m) and deepened 2 previous holes. Trenching and sampling were also carried out on peripheral targets.

There is no record of previous exploration field work carried out on the Rio Cenicero concessions prior to Peregrine's 2009 field season.

# 7.0 GEOLOGICAL SETTING

## 7.1 Regional Geology

### 7.1.1 Cordillera Principal

The Andean Cordillera extends for about 5,000 km along the western coast of South America, attaining a maximum width of about 700 km in the Central Andes of Bolivia. Tectonism in the Cordillera varies both along strike, and across the range; along-strike variations reflect changing plate geometry along the Pacific margin, whereas across-strike variations generally assigned to four sub-domains reflect the generally eastward migration of Andean arc magmatism and deformation through time. In general terms, there are three units within each sub-domain, from west to east: a fore-arc zone, a magmatic arc, and a back-arc region.

In the southern flat-slab sub-domain of the Central Andes (from 28°S to 33°30'S), the fore-arc zone is a steady rise to the crest of the Andes, which is formed by an inactive magmatic arc and thrust belt (Frontal Cordillera or Cordillera Principal). The Triassic magmatic (rift) arc has a general northwest–southeast trend. The foreland consists of an active, thin-skinned fold-thrust belt (Pre-cordillera) and zone of basement uplifts (Sierras Pampeanas, with altitudes ranging from 2,000–6,000 m). The Altar Project is located in the Cordillera Principal.

Basement rocks in the Project area have been assigned to the Choiyoi Group, of Permo–Triassic age; the Choiyoi Group covers about 500,000 $km^2$ in Argentina. It comprises an upper and lower volcanic sequence, intruded by shallow-level plutons, stocks, and dyke-like bodies (Lambias, 1999). The lower volcanic sequence comprises calc-alkaline andesite–dacites that represent the products of a subduction-related magmatic arc, which is overlain by an upper sequence of peraluminous rhyolites, related to a period of post-orogenic extensional collapse. Composition of the volcanics trends from mafic to acidic through time. Both sequences are propylitically-altered and contain fracture-controlled epidote, chlorite, albite, and calcite veining. The volcanic sequence was intruded by peraluminous A-type and S-type granites that are considered coeval with the rhyolitic volcanics and likewise typically exhibit low-grade propylitic alteration.

Generally, Jurassic marine sediments that consist of red-bed sandstones and claystones infill the Triassic rift, and unconformably overlie the Choiyoi Group; however Jurassic sediments are not known in the immediate surroundings of the Altar Project. Within the Project area, rhyolitic ignimbrites and andesitic volcanics of the Pachon Formation overlie the Choiyoi basement sequence with age dates of 20-22 Ma (Miocene).

The Pachon volcanics are equivalent in age to country rock sequences at Los Pelambres and El Pachon (Pachon Minera S.A., 1999). The Pachon Formation is known in Chile as the Abanico Formation and they are time-stratigraphic equivalents (Fernandez et al., 1974).

## 7.2 Property Geology

The Altar porphyry copper±gold±molybdenum deposit is associated with Miocene intermediate composition porphyries that intrude mid-Miocene rhyolitic ignimbrites and fine-grained andesite flows of the Pachon Formation. Elevated gold, silver and molybdenum values are associated with the copper mineralization.

### 7.2.1 Permo–Triassic Choiyoi Group Volcanic Rocks

***Dacite and Andesite***

Propylitically-altered dacite and andesite have been mapped at the western limit of alteration. The units have a strong resemblance to rocks of the lower Choiyoi Group volcanic sequence and may have a similar Permo–Triassic age as the Choiyoi volcanics.

***Choiyoi Andesite***

The Choiyoi Andesite is part of the lower Choiyoi Group volcanic sequence and outcrops along the western margin of the drilled area. It comprises black to dark brown andesitic to dacitic lava flows with aphanitic to fine-grained porphyritic textures. Fiamme can be observed in road outcrops, as well as in drill core. The Choiyoi Andesite is concordant with the overlying Vitric Tuff and is steeply westward-dipping as found in road outcrops in the northwestern part of the Central Zone.

***Piuquenes Volcaniclastic Breccia***

The presumed Permo–Triassic-aged Piuquenes Volcaniclastic Breccia outcrops in the northwestern portion of the deposit. In texture and occurrence, it is very similar to the Pachon Volcaniclastic Breccia in the East Zone. The breccia is a matrix-supported volcanic breccia which is locally moderately silicified. The matrix is composed of fine feldspar crystals in a microcrystalline groundmass. Clasts, comprising aphanitic and fine-grained porphyritic andesites range from 1 cm to 5 cm. Some clasts have been strongly silicified, so that primary textures are no longer discernable.

The Piuquenes Volcaniclastic Breccia was informally termed the Choiyoi Breccia in the 2005–2006 drilling program. The unit is in the process of being re-defined by Peregrine from drill core intercepts, and with further drilling data; it may be that the Piuquenes Volcaniclastic Breccia and the Pachon Volcaniclastic Breccia are part of the same lithological unit.

***Rhyolite Ignimbrite – Vitric Tuff***

For the purposes of mapping and core logging the rhyolitic ignimbrites have been assigned the field name "Vitric Tuff". The rhyolitic ignimbrite is fine- to medium-grained, moderately to strongly welded, and has diagnostic medium-grained embayed quartz eyes. The quartz eyes are sub-rounded to subangular with embayed fine-grained feldspar fragments and crystals. The ignimbrite is steeply west-dipping at 60° to 85° as seen in the western half of the Altar system, where several areas of measurable fiamme have been noted from road cuts and limited outcrops (Sillitoe, 1999). The ignimbrites may be of a similar age to the Choiyoi volcanics, if the correlation of the underlying dacites and andesites with the Choiyoi Group is correct. Within the area of Miocene porphyry intrusions, the ignimbrite is quartz-veined and altered to sericite.

### 7.2.2 Miocene Pachon Formation – Andesites and Volcaniclastics

Unconformably overlying the Vitric Tuff is an andesite assemblage of Miocene age, (20-22 Ma) which is interpreted to be part of the Pachon Formation. This formation was previously believed to be Cretaceous in age.

***Pachon Volcaniclastic Breccia***

The basal portion of the andesite sequence consists of volcaniclastic breccia, the "Pachon Volcaniclastic Breccia", which consists of clasts of the underlying Vitric Tuff unit and andesite in an andesitic matrix. This breccia is found at the head of the Arroyo Altar and in lower road cuts that access the eastern half of the Altar mineralized system. It strikes roughly north–south, and is east- to southeast-dipping at 10° to 30°. The volcaniclastic breccia appears to be of variable thickness with surface exposures reaching up to 140 m in thickness.

***Pachon Andesite***

Overlying the Pachon Volcaniclastic Breccia is a series of fine-grained to aphanitic andesite and andesite porphyry flows with thin (1 cm to 20 cm) partings, the "Pachon Andesite." The andesite porphyry contains fine-grained feldspar phenocrysts in an aphanitic to fine-grained groundmass that is weakly to moderately welded. The Pachon Andesite sequence is found in the eastern half of the Altar mineralized system where it is weakly to moderately sericite-altered. Tertiary Porphyry Intrusives

***Quartz Diorite Porphyry, Early Quartz Diorite Porphyry, Inter-mineral Quartz Diorite Porphyry***

Intruding the Vitric Tuff, Pachon Volcaniclastic Breccia, and Pachon Andesite are two phases of Miocene (10-12 Ma) intermediate-composition porphyries, which, from IP data, form a single stock that is interpreted as the source of the alteration and mineralization at the Altar deposit.

Petrographic work by Rio Tinto has determined compositions ranging from diorite to tonalite to dacite for these porphyries. The rocks characteristically contain medium-grained phenocrysts of internally-zoned feldspars and finer-grained sparse quartz phenocrysts in a fine-grained to aphanitic groundmass. The feldspar phenocrysts comprise up to 80% of the rock, often giving it a crowded texture that is distinctive and identifiable through all types and intensities of alteration.

The porphyries are subdivided into early and inter-mineral varieties. Based on past mapping (Sillitoe, 1999, and Almandoz et al., 2003), surface mapping during the 2005-2006 field season, and limited petrographic examination of core, the major distinguishing feature that separates early and inter-mineral porphyries is intensity of alteration. The field names “Early Quartz Diorite Porphyry” and “Inter-mineral Quartz Diorite Porphyry” have been assigned to these two varieties; when the types are indistinguishable, the term “Quartz Diorite Porphyry” is used.

A recent Ar–Ar age date of 10.38 Ma was obtained by Rio Tinto from sericite hosted in Early Quartz Diorite Porphyry from drill hole ALD-01 at 293 m depth (Peregrine Metals Ltd, 2006). The Miocene age date is similar to the K–Ar date of 9.8 Ma obtained by Sillitoe (1977) at the nearby Los Pelambres Mine in Chile.

Early and Inter-mineral Quartz Diorite Porphyry have been distinguished on the basis of quartz vein density within the zone of oxidation and by the presence of strong disseminated chalcocite mineralization below the zone of oxidation. Early Quartz Diorite Porphyry generally hosts quartz veinlet stockworking that comprises more than 40% of the rock, whereas Inter-mineral Quartz Diorite Porphyry hosts less than 20% quartz veinlets. Fine-grained disseminated chalcocite, ranging from 1% to 2% of the rock by abundance, is characteristic of mineralized Early Quartz Diorite Porphyry below the oxidized zone and within sericitic alteration.

Inter-mineral Quartz Diorite Porphyry was also found to host medium to coarse grained, Carlsbad-twinned feldspar phenocrysts. The complex twinned phenocrysts appear to be diagnostic, as they are always present within Inter-mineral Quartz Diorite Porphyry, but are not identifiable within Early Quartz Diorite Porphyry.

Inter-mineral Quartz Diorite Porphyry is mapped within the outer fringes of sericitic alteration, where weakly propylitically-altered plugs of dacite porphyry have undergone advanced argillic alteration.

For exploration convenience, the porphyry stock has been divided on a purely geographical basis into three zones. The Central Zone corresponds to a central ridge in the Altar Cirque, and the East and West Zones correspond to the eastern and western slopes of the cirque, respectively (Figure 7-1).

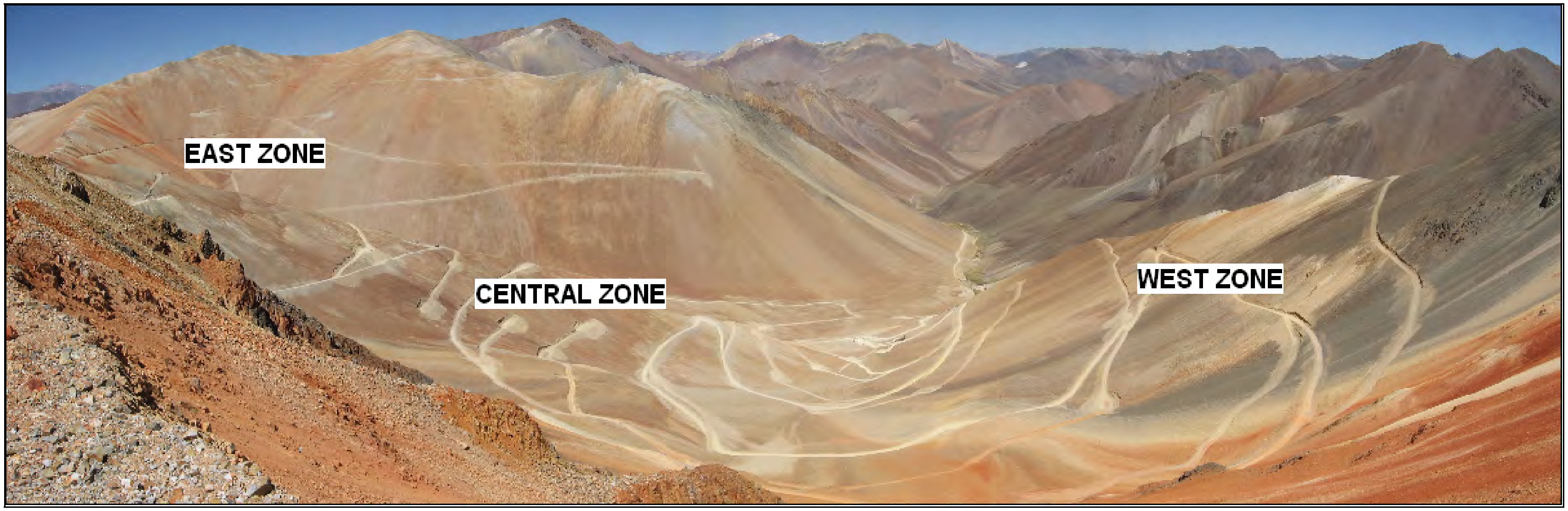
***Magmatic Breccia***

Associated with the Quartz Diorite Porphyry is a magmatic breccia that is only found in drill core, and consists of coarse clasts of strongly quartz-veined Quartz Diorite Porphyry and altered Vitric Tuff, within an aphanitic matrix.

### 7.2.3 Colluvium and Alluvium

The Altar area was subjected to regional alpine glaciations, which resulted in several moraines and significant glacial sediments that cover most low-lying areas. There is little outcrop within the altered and mineralized area due to scree and talus cover on steep slopes and glacial sediments in the valley bottoms. Glacial sediments in the Arroyo Altar area can reach up to 30 m in thickness, as demonstrated from core drilling in ALD-05.

**Figure 7-1 Surface Exposure, Altar Project, Looking South**




Note: From Peregrine Metals, (2007). The photograph field of view covers about 2 km from east to west.

## 7.3 Alteration

All of the lithological units described in Section 7.2 have undergone varying degrees of hydrothermal alteration. The strongest alteration is found within Early Quartz Diorite Porphyry which underwent early potassic alteration (K-feldspar–secondary biotite–quartz) overprinted by intense pervasive sericitic alteration (quartz–sericite–pyrite–tourmaline).

Using the vein terminology for porphyry deposits of Gustafson and Hunt (1975), at least three generations of A-veins and locally B-veins, related to potassic alteration are seen in drill core. These veins constitute at least 40% of the overall rock mass.

The Inter-mineral Quartz Diorite Porphyry and Magmatic Breccia locally underwent potassic alteration within the core of the altered area, and are also found to have undergone weak to moderate propylitic alteration (chlorite–specularite–quartz–hematite) peripheral to the center of the system.

A-type quartz veining and K-feldspar replacement within Inter-mineral Quartz Diorite Porphyry is weak to moderate, with quartz veins generally constituting less than 20% of the resulting rock mass. Potassic alteration of the Quartz Diorite Porphyry phases is only preserved at depth in several of the deeper drill holes, and in all cases has at least a weak sericitic alteration overprint.

Vitric Tuff and Pachon Andesite also underwent potassic alteration where these units occur in proximity to Quartz Diorite Porphyry intrusions, but in all cases the resulting A-vein density was substantially less than that found in the nearby intrusions. This potassic alteration has been strongly overprinted by sericitic alteration, so that the early potassic alteration minerals are no longer readily identifiable.

The sericitic alteration passes outwards into little-altered ignimbrite or chloritized andesitic volcanics, both of which were subjected to varied degrees of supergene kaolinization as a result of acid attack during pyrite oxidation. The outer limit of jarositic limonite after pyrite was mapped and defines the Altar hydrothermal system to be at least 3.5 km x 3 km in outcropping surface area.

Although specular hematite is ubiquitous as a late, fracture-controlled mineral in the Altar system, it is particularly abundant in the peripheral chloritized rocks and appears to constitute a halo to the sericitic core. Epidote is apparently absent from the peripheral alteration zone, except in the northern aphanitic andesites.

Intense pervasive silicic alteration is observed in drill holes in the East Zone, where silicification ranges from 25% to 100% over intervals of tens of metres. This silicification does not appear to be related to D-vein density, and is likely associated with the advanced argillic alteration that affected the lithocap. Intense silicic alteration correlates with elevated Au grades, as observed in holes ALD-04 and ALD-13.

***Stockworks***

Stockwork quartz veining was initially identified in surface mapping (Sillitoe, 2003). It comprises grey to pinkish-grey, translucent A-type quartz veinlets (using the nomenclature of Gustafson and Hunt, 1975) as well as a lesser number of more laterally extensive B-type quartz veinlets characterized by central sutures. The A-type veinlets attain 3 cm in width. The stockwork zones are cut by generally minor D-type veinlets with prominent sericitic haloes; the most extensive observed examples from surface exposures in the western stockwork zone strike north–northeasterly, parallel to the zone itself.

***Siliceous Ledges***

The 3 km long arcuate ridge along the eastern side of the Altar cirque is characterized by the basal part of an advanced argillic lithocap. The lithocap remnant is defined by numerous structurally-controlled siliceous ledges separated by chloritized andesitic volcanics. In 2008 Approximately 50 principal ledges were mapped to define a broadly radial pattern centred on the eastern stockwork zone. Mapping by Peregrine in 2009 defined over 200 ledges. Ledges are confined to the ridge top, at elevations above 3,600 m and do not continue far down the talus-covered slopes (Sillitoe, 2003).

Most ledges are steep and contain central zones, ranging from 10 cm to 2 m-wide, of quartz–alunite alteration flanked outwards by quartz–kaolinite. Locally, pyrophyllite is observed as a transitional zone. A few ledges contain pods of vuggy residual quartz along their centre lines in which enargite and, less commonly, barite and native sulphur occur. Stringers of massive enargite are also present in places in quartz-rich quartz–alunite ledges lacking vuggy quartz. The hypogene quartz–kaolinite haloes to the ledges are transitional outwards to supergene kaolinization developed at the expense of chlorite–smectite alteration. The most extensive ledge, 500 m long, terminates in a small hydrothermal breccia pipe displaying intense quartz–alunite alteration.

## 7.4 Structure

Outcrop mapping in the northern part of the main Altar cirque has identified a north-northeast-striking fault projected to cut through the centre of the mineralized system between the Central Zone and the East Zone. Grey fault gouge exposed in a road cut at Gauss Kruger coordinates 2,359,750E/6,516,830N provides further evidence for the continuity and importance of this fault zone. This is also reflected by zones of intense fracturing intersected in drill holes ALD-01, ALD-03, ALD-08 and ALD-09.

The silica ledges (see Section 7.3), which define the basal part of the advanced argillic lithocap, are much more numerous on the ridge crests on the east side of the fault than on the west side, where only a few silica ledges occur at the highest elevations.

This suggests west-side up displacement across the fault system. Deeper levels of the stratigraphic succession are exposed on the west side of the fault, again implying west-side up displacement.

## 7.5 Lithology Codes

The geological model was updated in 2010 based upon the drilling to the end of the 2010 field season. The current interpretation of the lithology distribution is illustrated in plan view in Figure 7-1 and cross section in Figure 7-3 and Figure 7-4. Primary lithologic codes are shown in Table 7-1.

**Table 7-1 Lithology Codes**

| Code | Symbol | Lithology |
|---|---|---|
| 1 | Tap | Miocene Andesite Porphyry |
| 3 | Tpa | Miocene Andesite |
| 5 | Ka | Miocene Rhyolite |
| 8 | Tdp | Quartz Diorite Porphyry |
| 10 | Ovbdn | Overburden |

**Figure 7-2 Lithology Plan**

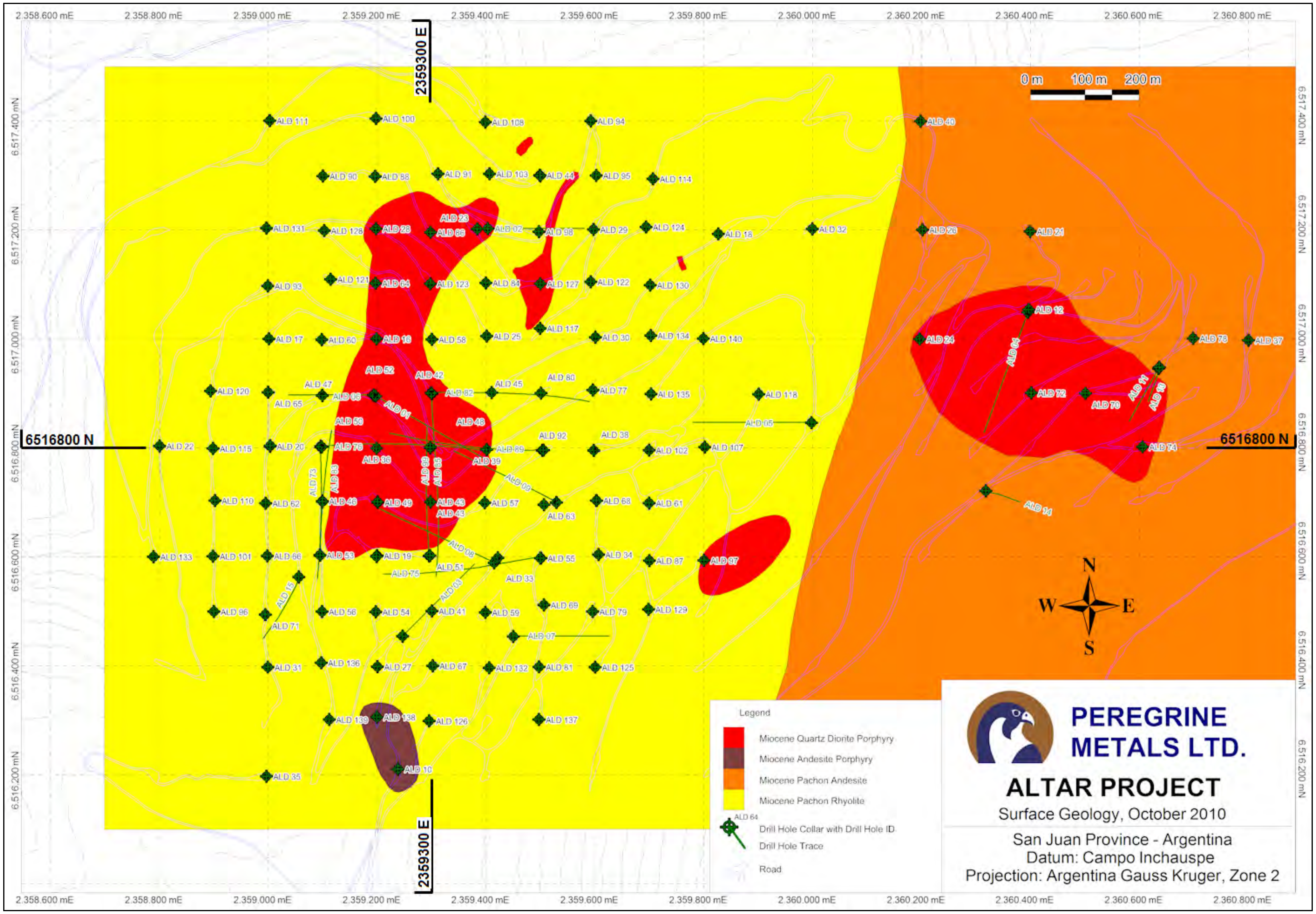


**Figure 7-3 Geological Section 6516800 N**

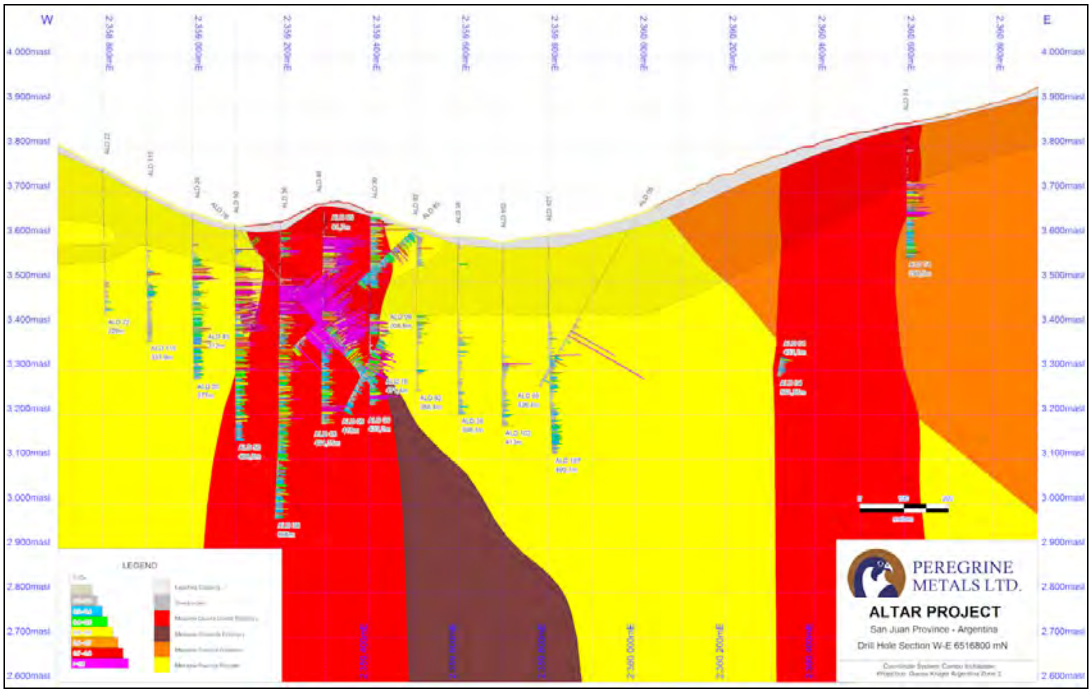
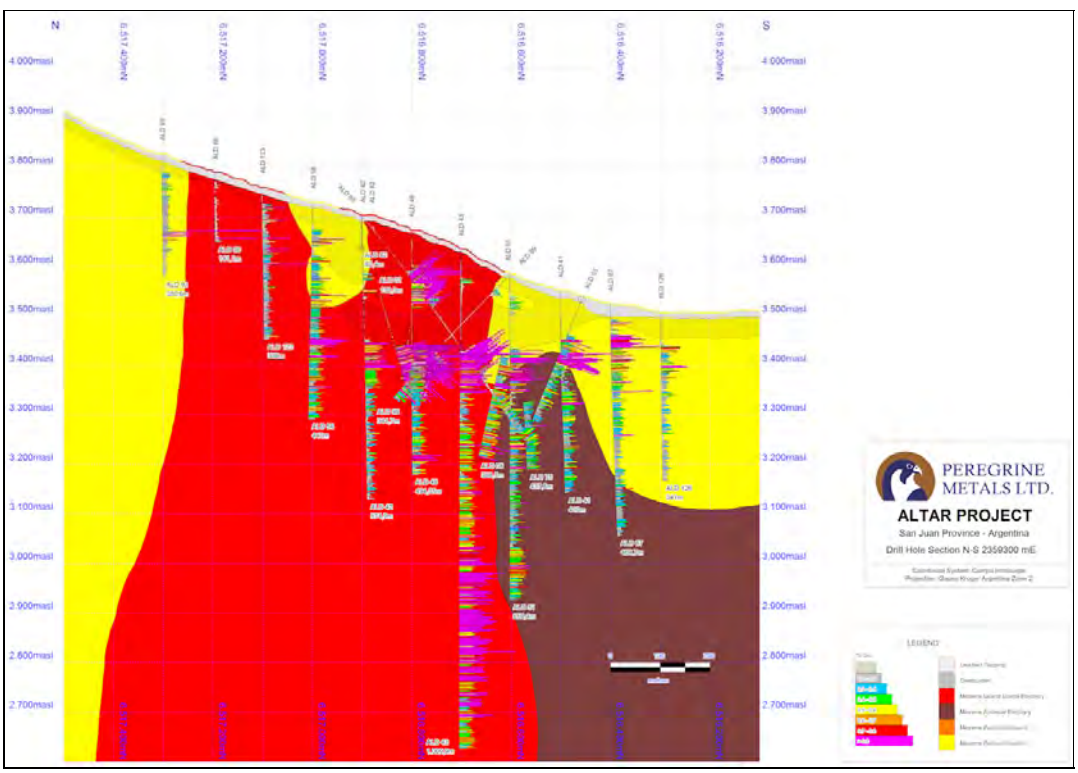



**Figure 7-4 Geological Section 2359300 E**



# 8.0 DEPOSIT TYPES

The Altar Project contains copper±gold±molybdenum sulphide mineralization that was deposited in an environment that transitions from the basal roots of a high-sulphidation epithermal lithocap to a subvolcanic copper porphyry environment at depth. The Altar Project is described as "telescoped" because of the close spatial distance between the porphyry and high-sulphidation alteration systems.

## 8.1 High-Sulphidation Epithermal Deposits

These deposits typically form in subaerial volcanic complexes or composite island arc volcanoes above degassing magma chambers. The deposits are also genetically related to high-level intrusions. Multiple stages of mineralization are common, presumably related to periodic tectonism with associated intrusive activity and magmatic hydrothermal fluid generation. High-sulphidation deposits can also be developed in second-order structures adjacent to crustal-scale fault zones, both normal and strike-slip, as well as local structures associated with subvolcanic intrusions. The deposits tend to overlie and flank porphyry copper–gold deposits and underlie acid-leached siliceous, clay and alunite-bearing 'lithocaps'.

Host rocks are typically volcanic pyroclastic and flow rocks, most commonly subaerial andesite to dacite and rhyodacite, and their subvolcanic intrusive equivalents. The deposits range in age from Tertiary to Quaternary; less commonly, Mesozoic and rarely Palaeozoic volcanic belts may be hosts. The rare preservation of older deposits reflects rapid rates of erosion before burial of subaerial volcanoes in tectonically active arcs.

Mineralization is developed in multiple, cross-cutting veins and massive sulphide replacement pods and lenses, stockworks and breccias. Deposits may have irregular shapes, as deposit geometry is determined by host rock permeability and the orientation of deposition-controlling structures. Principal minerals comprise pyrite, enargite/luzonite, chalcocite, covellite, bornite, gold, and electrum; lesser minerals can include chalcopyrite, sphalerite, tetrahedrite/tennantite, galena, marcasite, arsenopyrite, silver sulphosalts, and tellurides including goldfieldite. Two types of ore are commonly present: massive enargite–pyrite and/or quartz–alunite–gold.

Typically, alteration consists of quartz, kaolinite/dickite, alunite, barite, hematite; sericite/illite, amorphous clays and silica, pyrophyllite, andalusite, diaspore, corundum, tourmaline, dumortierite, topaz, zunyite, jarosite, Al–P sulphates (such as hinsdalite, woodhouseite, crandalite) and native sulphur. Advanced argillic alteration is characteristic, and can be a really extensive and visually prominent.

Quartz occurs as fine-grained replacements and, more characteristically, as vuggy residual silica in acid-leached rocks.

## 8.2 Porphyry Copper Deposits

Porphyry copper deposits tend to form in orogenic belts at convergent plate boundaries, and are commonly linked to subduction-related magmatism. They may also form in association with emplacement of high-level stocks during extensional tectonism related to strike-slip faulting and back-arc spreading following continent margin accretion. Host rocks tend to be high-level (epizonal) stock emplacement levels in volcano-plutonic arcs, commonly oceanic volcanic island and continent-margin arcs. Virtually any type of country rock can be mineralized, but commonly the high-level stocks and related dykes intrude their coeval and cogenetic volcanic piles. Porphyry deposits in the Andes are generally Tertiary in age; globally, deposits can range in age from Archaean to Quaternary.

Intrusions range from coarse-grained phaneritic to porphyritic stocks, batholiths and dike swarms, but are rarely pegmatitic. Compositions range from calc-alkaline quartz diorite to granodiorite and quartz monzonite. Commonly, there are multiple emplacements of successive intrusive phases and a wide variety of breccias. Deposits generally comprise large zones of hydrothermally-altered rock that contain quartz veins and stockworks, sulphide-bearing veinlets; fractures and lesser disseminations in areas up to 10 km$^2$ in size. Deposits can be wholly or in part coincident with hydrothermal or intrusion breccias and dike swarms. Deposit boundaries are determined by economic factors that outline ore zones within larger areas of low-grade, concentrically zoned mineralization.

Pyrite is the predominant sulphide mineral; in some deposits the iron oxide minerals magnetite, and rarely hematite, are abundant. Economically-important minerals are chalcopyrite; molybdenite, lesser bornite and rare (primary) chalcocite. Subordinate minerals are tetrahedrite/tennantite, enargite and minor gold, electrum and arsenopyrite. In many deposits late veins commonly contain galena and sphalerite in a gangue of quartz, calcite and barite.

Early formed alteration can be overprinted by younger assemblages. Central and early formed potassic zones (K-feldspar and biotite) commonly coincide with ore. This alteration can be flanked in volcanic host rocks by biotite-rich rocks that grade outward into propylitic rocks. The biotite is a fine-grained, 'shreddy' looking secondary mineral that is commonly referred to as an early-developed biotite (EDB) or a 'biotite hornfels'. These older alteration assemblages in cupriferous zones can be partially to completely overprinted by later biotite and K-feldspar and then phyllic (quartz–sericite–pyrite) alteration, less commonly argillic, and rarely, in the uppermost parts of some ore deposits, advanced argillic alteration (kaolinite–pyrophyllite).

# 9.0 MINERALIZATION

## 9.1 General

Surface geological and alteration mapping, IP geophysical surveys and drilling have identified porphyry- and high-sulphidation-related alteration and sulphide mineralization at Altar, extending over an area of 2.9 km x 1.7 km within a 3.5 km x 3 km zone of hydrothermal alteration.

Mineralization is closely associated with the intrusive stock, but is hosted not only in the porphyry phases, but also in the Vitric Tuff, Pachon Volcaniclastic Breccia and Pachon Andesite. The well-developed copper mineralization shows a strong relationship to the distribution and intensity of sericitic and potassic alteration.

Peregrine geologists have interpreted the mineralization paragenesis as follows.

Stage 1: Potassic alteration, accompanied by deposition of pyrite–chalcopyrite–bornite and pyrite–molybdenite mineralization.

Stage 2: Sericitic alteration overprint, accompanied by reconstitution of the Stage 1 mineralization as assemblages of pyrite, chalcocite, covellite, bornite and digenite.

Stage 3: Deposition of pyrite–enargite vein systems.

Acidic high-sulphidation conditions prevalent in an advanced argillic lithocap were superimposed on an underlying potassic alteration zone as a result of telescoping. Sulphides generally exhibit a consistent vertical zonation pattern: pyrite–enargite at the higher levels; pyrite–chalcocite–covellite–bornite–digenite assemblages at intermediate levels; and pyrite–chalcopyrite–bornite and pyrite–molybdenite assemblages at deeper levels. Recent petrographic work has also identified tennantite-tetrahedrite from intermediate level samples.

The copper mineralization associated with the potassic alteration, mainly porphyry-style chalcopyrite–bornite mineralization, was reconstituted as hypogene assemblages of pyrite, chalcocite, covellite and bornite within the sericitic alteration zone. Magnetite originally present in the potassic alteration zone was pyritized during the high-sulphidation overprint. Sulphide minerals found within sericitic alteration include pyrite, chalcopyrite, chalcocite, enargite, bornite, covellite, digenite, and molybdenite. Latest-stage pyrite–enargite veins related to a high-sulphidation epithermal system cut through the Stage 1 and 2 mineralization, but contribute a minor proportion of the copper mineralization.

Pyrite is ubiquitous with contents ranging from 2% to 15% but generally falling between 3% and 6%. It occurs as disseminations in wall rock, as quartz–pyrite veins, in late pyrite–enargite veins and occasionally as massive pyrite veins up to 2 cm thick.

The main style of hydrothermal alteration observed on the Rio Cenicero property corresponds to a potential high-sulphidation epithermal Au system that is located in the northern half of the RCA XII concession. Advanced argillic alteration is associated with the epithermal Au system and is accompanied by widespread silicification in the form of “silica ledges”, which are steeply dipping structures filled with epithermal quartz.

Of secondary importance is a zone of sericitic and argillic alteration that occurs in the northern part of the RCA V concession.

Peregrine mapped a large number of silica ledges within the zone of advanced argillic alteration in the RCA XII concession. The silica ledges are characterized by vuggy silica, multi-stage episodic hydrothermal breccias (crackle breccias and rotational breccias, and breccias with well-rounded clasts that indicate forceful expulsion of high-pressure fluids), colloform and crustiform banded quartz, deposition of native sulphur, alunite, barite, enargite and tennantite, limonite and boxworks after sulphides. These are indicators of the high-level acid-sulphate environment that overlies high-sulphidation epithermal Au deposits.

## 9.2 Mineralization Thickness

At Altar a leached cap zone has been intersected by drilling at depths ranging from 0 m to 258 m. Below the leached cap is a zone of primarily sulphide mineralization that is variably affected by weathering, mainly in the form of oxidation developed along narrow joints and fractures. This transitional zone ranges in drilled thickness from 4 m to 86 m, averaging about 11 m.

Sulphide mineralization has been logged at depths ranging from 9 m to 1010 m. Drilled thicknesses range from about 110 m in drill hole ALD-009, to 805 m in ALD-043. All but 4 holes completed to date that have reached target depth have ended in sulphide mineralization, and the mineralization remains open at depth.

# 10.0 EXPLORATION

Exploration prior to 2009/2010 is described in the summarized in Section 6 and detailed in the previous technical report (NMS-Geosim, 2009)

During the 2009/2010 exploration field season Peregrine Metals constructed 24 kilometres of new roads on the Altar and Rio Cenicero properties to provide access for drilling, excavator trenching, rock chip sampling, and geophysics.

A total of 4,360 metres of excavator trenching was completed over an epithermal Au-Ag target located along the eastern margin of the Altar porphyry Cu system in early 2010, with 2,390 metres on the Rio Cenicero property and 1,970 metres on the Altar property. Continuous 2-metre rock chip samples collected from these trenches totalled 2,679, with 1,195 samples collected on the Rio Cenicero property and 1,484 samples collected on the Altar property.

At the Quebrada de la Mina (“QDM”) target area, the company collected a total of 169 “grab-style” rock chip samples, of which 146 were collected on the Altar property and 23 on the Rio Cenicero property.

The company completed a total of 22,900 metres of induced polarization (“IP”) geophysical surveys in early 2010, of which 14,660 metres was carried out on the Altar property and 8,240 metres on the Rio Cenicero property.

The new IP lines run in early 2010 expanded the pre-existing IP coverage from the initial 2005 survey to the east and north around the margins of the Altar porphyry Cu system, and extended the IP anomalies in those directions. In addition a new IP target was identified at the Quebrada de la Mina alteration zone defined as a 300-metre wide chargeability anomaly extending north-south over 900 metres. This anomaly remains open to the north and south and is coincident with an area of outcropping leached capping in dacite porphyry that has returned significant Au values in grab-type rock chip samples described in section 10.1.

## 10.1 QDM Rock Chip Sampling

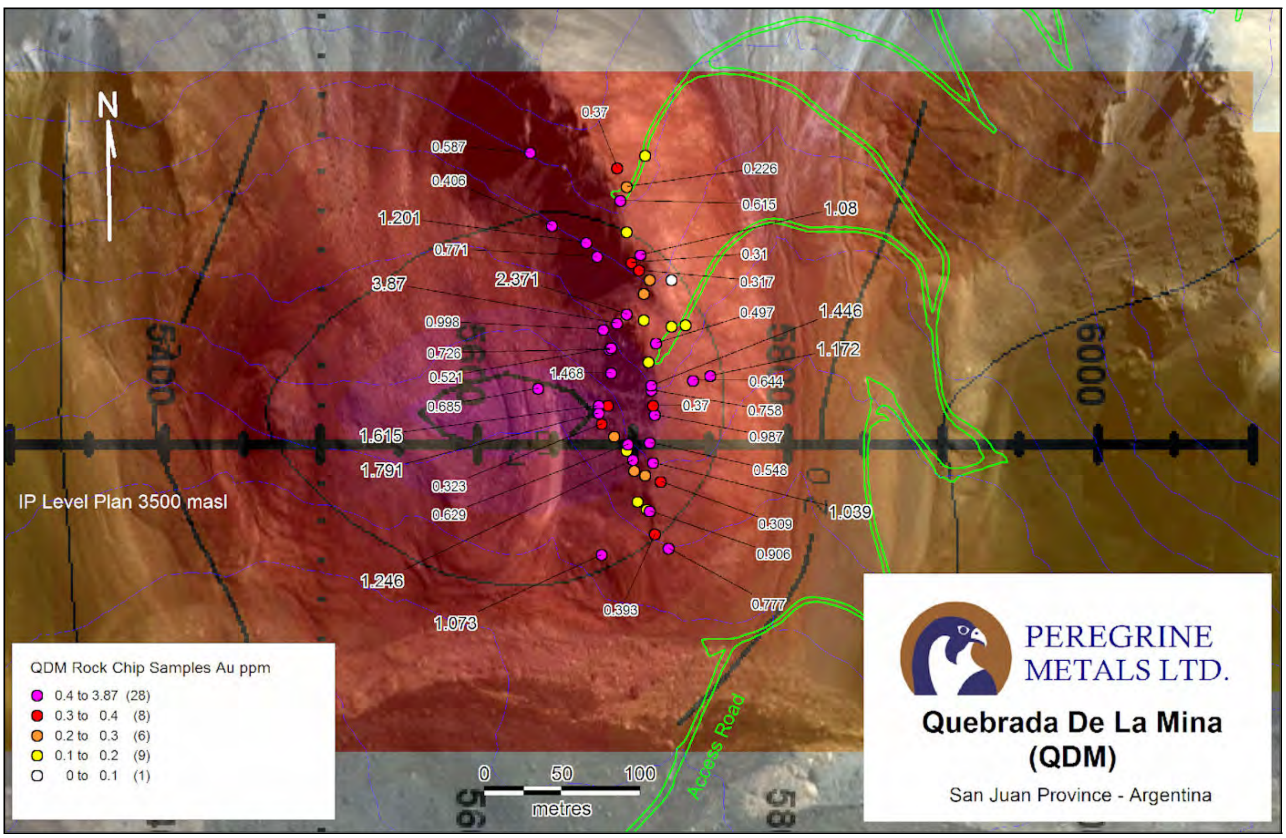
The Quebrada de la Mina (“QDM”) prospect is located on the Altar property approximately 2 kilometres to the northwest of the Altar porphyry Cu system. QDM is underlain by the same andesitic volcanic sequence that forms the country rock sequence at Altar. The volcanics are intruded by a circular dacite porphyry stock approximately 700 metres in diameter and host a large alteration footprint centred on the porphyry stock. Surface rock exposures at QDM are characterized by pervasive quartz-sericite-tourmaline alteration with disseminations and veinlet stockworks of jarosite after pyrite and less-abundant quartz veinlets.

Visible oxide copper mineralization occurs in the alteration zone and includes malachite, chalcanthite, neotocite and azurite impregnating fractures. Well-defined hydrothermal breccias also occur at two localities within the prospect area. Three campaigns of geochemical rock chip sampling have consistently returned gold grades ≥0.5 g/t along with low copper grades reflecting the fact that the rocks at surface are leached of more mobile copper while leaving behind the immobile gold.

In early 2010, Peregrine Metals Ltd. collected 52 rock chip grab samples from a 150 metre by 300 metre area of abundant outcrop in the center of the QDM prospect (Figure 10-1). The assays returned gold values averaging 0.7 g/t and ranging up to 3.9 g/t. This area of sampling is coincident with the centre of an induced polarization ("IP") geophysical anomaly that measures 300 metres by 900 metres, as defined by the 20 millivolt per volt chargeability contour on the 3,500 metre elevation level plan. The IP survey was also carried out in early 2010.

Immediate plans for follow-up of these encouraging preliminary results include excavator trenching and channel sampling, along with initial drill testing of the QDM target during the coming field season (November 2010 to May 2011).

**Figure 10-1 QDM Rock Chip Sample Locations**



# 11.0 DRILLING

## 11.1 Introduction

Five phases of diamond drilling have been completed to date on the Altar Project: the first, by Rio Tinto, in 2003, the second, by Peregrine, during 2005–2006, the third, by Peregrine during 2006-2007 and the fourth by Peregrine during 2007-2008. The fifth and most recent program was carried out between January and May of 2010. Drill programs prior to 2010 have been documented in the previous technical report and are summarized in Table 11-1.

**Table 11-1 Altar drill program summary**

| Year | Company | Holes Drilled | Total metres | Comments |
|---|---|---|---|---|
| 2003 | Rio Tinto | 7 | 2,841.13 | |
| 2006 | Peregrine | 8 | 3,302.20 | |
| 2007 | Peregrine | 25 | 10,408.15 | |
| 2008 | Peregrine | 24 | 12,740.60 | + 1 hole extended |
| 2010 | Peregrine | 76 | 26,348.55 | + 2 holes extended |
| **Total** | | **140** | **55,640.63** | |

Drill hole locations are documented in Table 11-2, and illustrated in Figure 11-1 below.

**Table 11-2 Drill hole collar locations**

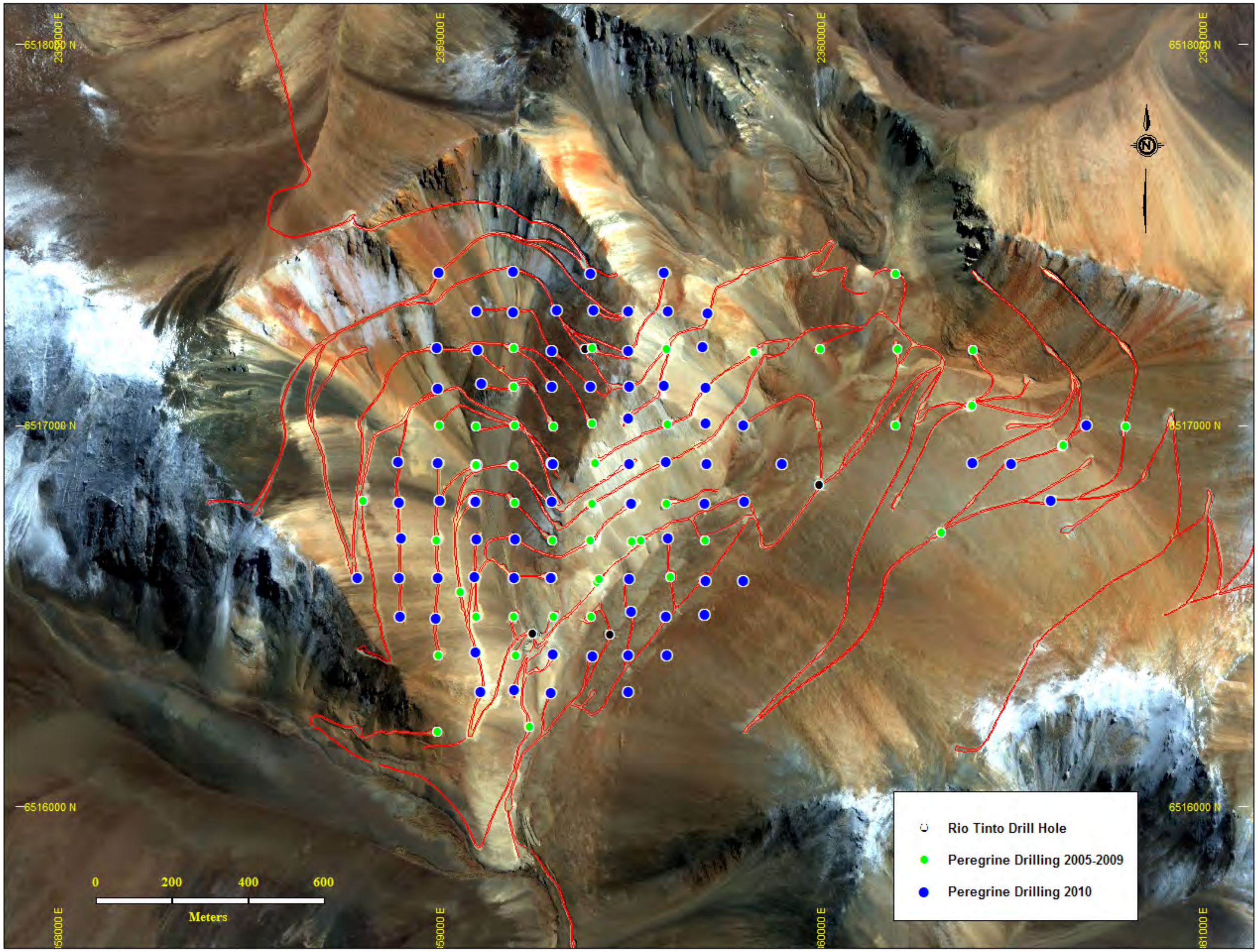
| HoleID | Easting | Northing | Elevation | Length | Azimuth | Dip | Year | Company |
|---|---|---|---|---|---|---|---|---|
| ALD 01 | 2359193.58 | 6516897.46 | 3657.45 | 466.55 | 120.00 | -60.00 | 2003 | Rio Tinto |
| ALD 02 | 2359384.00 | 6517202.51 | 3817.59 | 393.60 | 90.00 | -60.00 | 2003 | Rio Tinto |
| ALD 03 | 2359247.00 | 6516454.75 | 3541.26 | 376.33 | 45.00 | -60.00 | 2003 | Rio Tinto |
| ALD 04 | 2360395.43 | 6517049.88 | 3729.33 | 501.85 | 200.00 | -62.00 | 2003 | Rio Tinto |
| ALD 05 | 2359997.93 | 6516846.86 | 3643.75 | 436.40 | 270.00 | -60.00 | 2003 | Rio Tinto |
| ALD 06 | 2359194.24 | 6516897.32 | 3657.33 | 314.45 | 270.00 | -60.00 | 2003 | Rio Tinto |
| ALD 07 | 2359450.17 | 6516454.28 | 3534.50 | 351.95 | 90.00 | -60.00 | 2003 | Rio Tinto |
| ALD 08 | 2359415.32 | 6516589.76 | 3554.16 | 496.00 | 295.00 | -62.00 | 2006 | Peregrine |
| ALD 09 | 2359529.54 | 6516700.24 | 3569.96 | 418.00 | 300.00 | -60.00 | 2006 | Peregrine |
| ALD 10 | 2359238.28 | 6516210.99 | 3512.35 | 400.40 | 0.00 | -90.00 | 2006 | Peregrine |
| ALD 11 | 2360635.69 | 6516946.29 | 3804.00 | 209.50 | 209.00 | -58.00 | 2006 | Peregrine |
| ALD 12 | 2360397.06 | 6517054.78 | 3729.38 | 398.70 | 0.00 | -90.00 | 2006 | Peregrine |
| ALD 13 | 2360637.04 | 6516948.65 | 3804.14 | 495.00 | 195.00 | -80.00 | 2006 | Peregrine |
| ALD 14 | 2360318.25 | 6516721.55 | 3799.33 | 469.20 | 115.00 | -80.00 | 2006 | Peregrine |
| ALD 15 | 2359056.36 | 6516563.61 | 3613.50 | 415.40 | 198.00 | -70.00 | 2006 | Peregrine |
| ALD 16 | 2359198.95 | 6517000.90 | 3694.13 | 435.00 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 17 | 2359001.54 | 6517001.07 | 3691.52 | 322.00 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 18 | 2359826.80 | 6517193.53 | 3673.42 | 362.30 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 19 | 2359200.40 | 6516602.74 | 3580.34 | 404.00 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 20 | 2359003.29 | 6516803.86 | 3655.63 | 377.00 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 21 | 2360399.70 | 6517198.21 | 3718.97 | 476.05 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 22 | 2358800.62 | 6516804.29 | 3753.78 | 329.00 | 0.00 | -90.00 | 2007 | Peregrine |

| HoleID | Easting | Northing | Elevation | Length | Azimuth | Dip | Year | Company |
|---|---|---|---|---|---|---|---|---|
| ALD 23 | 2359403.24 | 6517203.36 | 3824.22 | 595.00 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 24 | 2360197.04 | 6517000.47 | 3685.08 | 445.80 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 25 | 2359401.61 | 6517005.87 | 3776.74 | 500.00 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 26 | 2360202.05 | 6517201.12 | 3680.45 | 464.45 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 27 | 2359200.89 | 6516399.00 | 3539.50 | 436.90 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 28 | 2359198.11 | 6517203.36 | 3774.05 | 421.00 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 29 | 2359597.49 | 6517201.88 | 3785.71 | 406.80 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 30 | 2359600.98 | 6517004.12 | 3705.71 | 409.50 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 31 | 2358999.47 | 6516398.13 | 3656.08 | 400.85 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 32 | 2359999.57 | 6517202.44 | 3661.06 | 350.30 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 33 | 2359421.69 | 6516598.72 | 3555.05 | 419.00 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 34 | 2359606.54 | 6516604.87 | 3574.20 | 449.00 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 35 | 2358997.01 | 6516197.56 | 3604.94 | 352.40 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 36 | 2359199.15 | 6516799.94 | 3632.77 | 666.00 | 0.00 | -90.00 | 2007/2008 | Peregrine |
| ALD 37 | 2360800.57 | 6516998.24 | 3858.80 | 415.70 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 38 | 2359597.96 | 6516796.06 | 3599.79 | 398.30 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 39 | 2359401.58 | 6516797.42 | 3654.69 | 433.20 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 40 | 2360198.61 | 6517400.43 | 3701.98 | 358.00 | 0.00 | -90.00 | 2007 | Peregrine |
| ALD 41 | 2359300.73 | 6516501.00 | 3551.37 | 410.00 | 0.00 | -90.00 | 2008 | Peregrine |
| ALD 42 | 2359300.81 | 6516900.28 | 3703.15 | 574.90 | 0.00 | -90.00 | 2008 | Peregrine |
| ALD 43 | 2359298.35 | 6516701.30 | 3635.62 | 1009.90 | 0.00 | -90.00 | 2008/2010 | Peregrine |
| ALD 44 | 2359498.91 | 6517301.06 | 3880.57 | 215.75 | 0.00 | -90.00 | 2008 | Peregrine |
| ALD 45 | 2359410.57 | 6516902.41 | 3715.65 | 494.00 | 0.00 | -90.00 | 2008 | Peregrine |
| ALD 46 | 2359099.44 | 6516702.83 | 3604.67 | 601.80 | 0.00 | -90.00 | 2008 | Peregrine |
| ALD 47 | 2359099.32 | 6516897.41 | 3648.11 | 538.00 | 0.00 | -90.00 | 2008 | Peregrine |
| ALD 48 | 2359297.59 | 6516800.72 | 3673.39 | 494.35 | 0.00 | -90.00 | 2008 | Peregrine |
| ALD 49 | 2359200.97 | 6516700.31 | 3605.95 | 655.90 | 0.00 | -90.00 | 2008 | Peregrine |
| ALD 50 | 2359097.11 | 6516802.05 | 3626.91 | 484.80 | 0.00 | -90.00 | 2008 | Peregrine |
| ALD 51 | 2359297.18 | 6516602.89 | 3585.29 | 659.40 | 0.00 | -90.00 | 2008/2010 | Peregrine |
| ALD 52 | 2359197.47 | 6516895.54 | 3657.79 | 458.00 | 0.00 | -90.00 | 2008 | Peregrine |
| ALD 53 | 2359094.99 | 6516603.77 | 3595.24 | 501.40 | 0.00 | -90.00 | 2008 | Peregrine |
| ALD 54 | 2359197.56 | 6516499.66 | 3553.39 | 455.00 | 0.00 | -90.00 | 2008 | Peregrine |
| ALD 55 | 2359501.19 | 6516598.14 | 3546.19 | 638.00 | 0.00 | -90.00 | 2008 | Peregrine |
| ALD 56 | 2359099.21 | 6516500.16 | 3592.01 | 494.00 | 0.00 | -90.00 | 2008 | Peregrine |
| ALD 57 | 2359398.07 | 6516699.88 | 3610.53 | 677.50 | 0.00 | -90.00 | 2008 | Peregrine |
| ALD 58 | 2359301.04 | 6516999.70 | 3731.08 | 440.00 | 0.00 | -90.00 | 2008 | Peregrine |
| ALD 59 | 2359399.08 | 6516498.67 | 3530.08 | 512.00 | 0.00 | -90.00 | 2008 | Peregrine |
| ALD 60 | 2359098.52 | 6516999.36 | 3679.95 | 419.00 | 0.00 | -90.00 | 2008 | Peregrine |
| ALD 61 | 2359699.71 | 6516699.51 | 3589.40 | 583.00 | 0.00 | -90.00 | 2008 | Peregrine |
| ALD 62 | 2358995.23 | 6516698.74 | 3648.85 | 470.00 | 0.00 | -90.00 | 2008 | Peregrine |
| ALD 63 | 2359506.39 | 6516696.53 | 3570.52 | 660.90 | 0.00 | -90.00 | 2008 | Peregrine |
| ALD 64 | 2359197.05 | 6517102.83 | 3741.26 | 392.00 | 0.00 | -90.00 | 2008 | Peregrine |
| ALD 65 | 2358999.96 | 6516902.85 | 3669.90 | 397.60 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 66 | 2358998.50 | 6516602.30 | 3645.51 | 547.10 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 67 | 2359302.16 | 6516400.65 | 3524.96 | 469.70 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 68 | 2359603.40 | 6516704.25 | 3569.05 | 814.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 69 | 2359506.65 | 6516512.23 | 3551.22 | 757.60 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 70 | 2360501.15 | 6516900.72 | 3797.54 | 404.00 | 0.00 | -90.00 | 2010 | Peregrine |

| HoleID | Easting | Northing | Elevation | Length | Azimuth | Dip | Year | Company |
|---|---|---|---|---|---|---|---|---|
| ALD 71 | 2358995.05 | 6516494.74 | 3649.68 | 452.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 72 | 2360401.57 | 6516901.87 | 3775.07 | 500.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 73 | 2359096.42 | 6516802.01 | 3626.94 | 367.00 | 180.00 | -50.00 | 2010 | Peregrine |
| ALD 74 | 2360606.10 | 6516802.72 | 3851.90 | 298.50 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 75 | 2359499.91 | 6516598.53 | 3546.16 | 502.90 | 270.00 | -55.00 | 2010 | Peregrine |
| ALD 76 | 2359096.50 | 6516803.54 | 3626.77 | 471.40 | 90.00 | -50.00 | 2010 | Peregrine |
| ALD 77 | 2359596.54 | 6516906.70 | 3652.97 | 299.80 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 78 | 2360699.05 | 6517002.08 | 3809.37 | 299.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 79 | 2359596.28 | 6516499.97 | 3569.38 | 640.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 80 | 2359501.29 | 6516901.08 | 3679.58 | 408.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 81 | 2359497.45 | 6516398.65 | 3546.26 | 517.80 | 90.00 | -50.00 | 2010 | Peregrine |
| ALD 82 | 2359299.16 | 6516901.86 | 3703.01 | 508.20 | 90.00 | -50.00 | 2010 | Peregrine |
| ALD 83 | 2359095.66 | 6516603.49 | 3595.13 | 313.00 | 0.00 | -45.00 | 2010 | Peregrine |
| ALD 84 | 2359399.93 | 6517103.30 | 3795.04 | 302.10 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 85 | 2359299.87 | 6516901.10 | 3703.08 | 577.10 | 180.00 | -50.00 | 2010 | Peregrine |
| ALD 86 | 2359298.10 | 6517196.58 | 3790.13 | 141.10 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 87 | 2359700.23 | 6516593.91 | 3589.38 | 630.50 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 88 | 2359196.39 | 6517299.19 | 3804.69 | 193.80 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 89 | 2359503.52 | 6516796.21 | 3621.44 | 393.10 | 270.00 | -45.00 | 2010 | Peregrine |
| ALD 90 | 2359100.30 | 6517299.51 | 3808.79 | 164.20 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 91 | 2359311.77 | 6517303.73 | 3828.52 | 250.60 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 92 | 2359505.69 | 6516796.35 | 3621.39 | 368.40 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 93 | 2358999.32 | 6517097.71 | 3723.77 | 333.60 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 94 | 2359592.79 | 6517400.86 | 3842.34 | 170.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 95 | 2359602.49 | 6517300.95 | 3829.63 | 328.20 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 96 | 2358900.22 | 6516499.98 | 3701.20 | 267.45 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 97 | 2359800.57 | 6516594.30 | 3609.84 | 383.50 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 98 | 2359497.71 | 6517197.72 | 3842.32 | 276.50 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 99 | 2359295.67 | 6516602.29 | 3584.82 | 344.70 | 0.00 | -45.00 | 2010 | Peregrine |
| ALD 100 | 2359197.90 | 6517405.58 | 3839.28 | 196.50 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 101 | 2358898.54 | 6516601.78 | 3701.50 | 410.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 102 | 2359699.81 | 6516796.76 | 3587.20 | 413.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 103 | 2359406.40 | 6517304.04 | 3868.52 | 188.10 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 104 | 2359199.29 | 6516601.30 | 3580.54 | 156.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 105 | 2359499.82 | 6517300.33 | 3880.44 | 84.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 106 | 2359201.25 | 6516701.37 | 3605.99 | 182.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 107 | 2359801.80 | 6516802.15 | 3602.17 | 490.70 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 108 | 2359398.87 | 6517398.78 | 3904.79 | 214.90 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 109 | 2359099.80 | 6516701.77 | 3604.50 | 146.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 110 | 2358902.29 | 6516703.95 | 3699.65 | 419.50 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 111 | 2359002.78 | 6517401.64 | 3872.24 | 223.70 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 112 | 2359098.49 | 6516802.40 | 3626.72 | 202.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 113 | 2359297.79 | 6516801.68 | 3673.47 | 148.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 114 | 2359706.85 | 6517294.77 | 3772.92 | 230.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 115 | 2358898.72 | 6516799.67 | 3703.12 | 337.90 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 116 | 2359099.72 | 6516896.72 | 3648.11 | 180.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 117 | 2359499.61 | 6517020.35 | 3748.27 | 293.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 118 | 2359900.96 | 6516899.17 | 3617.84 | 443.00 | 0.00 | -90.00 | 2010 | Peregrine |

| HoleID | Easting | Northing | Elevation | Length | Azimuth | Dip | Year | Company |
|---|---|---|---|---|---|---|---|---|
| ALD 119 | 2359004.06 | 6516804.45 | 3655.67 | 152.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 120 | 2358894.37 | 6516905.21 | 3715.06 | 292.50 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 121 | 2359114.98 | 6517110.12 | 3741.61 | 299.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 122 | 2359592.46 | 6517105.78 | 3759.03 | 271.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 123 | 2359297.61 | 6517102.02 | 3760.43 | 308.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 124 | 2359694.05 | 6517206.43 | 3742.40 | 226.50 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 125 | 2359600.42 | 6516398.11 | 3573.07 | 353.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 126 | 2359295.75 | 6516299.26 | 3507.63 | 341.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 127 | 2359500.11 | 6517102.20 | 3802.39 | 281.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 128 | 2359102.96 | 6517199.38 | 3768.07 | 261.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 129 | 2359698.72 | 6516504.31 | 3592.20 | 513.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 130 | 2359701.77 | 6517099.55 | 3695.57 | 238.50 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 131 | 2358996.83 | 6517204.31 | 3772.45 | 136.50 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 132 | 2359405.83 | 6516396.95 | 3523.93 | 545.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 133 | 2358789.49 | 6516601.07 | 3767.55 | 306.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 134 | 2359702.61 | 6517006.97 | 3670.19 | 277.50 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 135 | 2359703.18 | 6516899.48 | 3619.48 | 288.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 136 | 2359097.34 | 6516405.89 | 3597.09 | 371.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 137 | 2359497.70 | 6516301.98 | 3546.25 | 439.40 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 138 | 2359199.71 | 6516306.68 | 3532.64 | 449.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 139 | 2359111.97 | 6516302.08 | 3587.19 | 380.00 | 0.00 | -90.00 | 2010 | Peregrine |
| ALD 140 | 2359799.42 | 6517001.55 | 3631.76 | 250.50 | 0.00 | -90.00 | 2010 | Peregrine |
| | | **Total** | **140** | **55,641** | **m** | | | |

**Figure 11-1 Project Drill Hole Location Plan 2003–2010**



## 11.2 2010 Drill Program

Boart Longyear of Mendoza was contracted to provide drilling services during the 2010 field season. They provided 2 wheel-skid mounted BBS37A diamond drill rigs and 2 wheel-skid mounted BBS56A diamond rigs, all 4 rigs with hydrostatic driver. The drilling commenced on January 15, 2010 with drill hole ALD 65 and finished with dill hole ALD 140 on May 2. A total of 26,348.55 metres was drilled. Sixty-seven holes were collared vertical with depths up to 814 metres. Two existing vertical holes were extended, ALD 43 to 1009.9 m and ALD51 to 659.4 m. Eight angle holes were drilled with dips ranging from -45 to -55°. All holes were collared with HQ and extended as deep as possible before reducing to NQ.

Eight of the holes were drilled for metallurgical samples and were twins of pre-existing core hole completed in prior years. Two water monitoring holes were also completed (50.9 m).

### 11.2.1 Collar and Down Hole Surveys

The survey data is discussed in further in Section 10 and Section 14.

Collar locations for Peregrine drilling were picked up by a professional surveyor, using a differential GPS system with ± 10 cm accuracy.

Once the downhole survey is completed, a cement slab monument is constructed at the drill hole collar to permanently preserve its location and identification. The monument is engraved with the drill hole ID, azimuth, dip, total depth and the date of the drill hole's completion.

For drill holes that may possibly require re-entry at a later date in order to be extended to greater depth, the casing is left in the hole, otherwise the casing is removed and replaced by plastic piping prior to constructing the cement slab monument.

### 11.2.2 Core Handling and Transportation

At the drill site, the drill core is removed from the core tube by the drilling contractor's personnel and placed into 1-metre long wooden core boxes, each box having capacity for approximately 3 metres of HQ core or 4 metres of NQ core. The end of each drill run is marked by a small wooden block on which the down-hole depth is written using a permanent waterproof marker. The driller's depth markers are routinely checked for errors by the geologist or an assistant present at the drill site and discrepancies are recorded and corrected in consultation with the drilling contractor's supervisor on site. When full, each core box is sealed with a tight-fitting wooden lid and stacked in sequential order in a safe location at the drill site.

Drill core is picked up at the drill site by a geologist or an assistant on a regularly basis at least twice daily and transported by 4-wheel drive pick-up to the core logging area located at the exploration camp approximately 7 kilometres from the centre of the drilling area.

At the logging area, the core is laid out in sequential order on racks organized according to drill rig and by drill hole. The core boxes for each hole are numbered in down-hole sequence and labelled with the drill hole ID and the starting and ending depth of the core contained in each box. A written record is made of the depth interval that corresponds to each box number to facilitate subsequent searches for specific core intervals after the core has been archived.

The driller's depth markers are again checked by the geologists and/or assistants at the core logging area and any discrepancies are recorded and corrected. The core is re-pieced to allow accurate core recovery measurements, which are made for each drill run, and RQD measurements are also made for each drill run.

Following core logging the core boxes are re-sealed with their wooden tops and labelled with aluminium Dymotape strips imprinted with the Drill Hole ID, the box number and the end depth of that box. They are then stacked in columns to be inventoried prior to being loaded on a truck and transported to Peregrine's core splitting and storage facility in Mendoza. Peregrine has contracted a 4-wheel drive truck with capacity for 6 tonnes of cargo that is dedicated full time to transporting the drill core from the core logging facility at the exploration camp to the core splitting facility in Mendoza. The truck is fitted with a specially constructed sealed steel box into which the core boxes are placed and carefully packed to avoid shifting. The tight fitting door is securely closed and locked with 3 padlocks following inspection and final inventory by a geologist. Numbered strap locks also seal the door alongside each padlock and the corresponding strap lock numbers are included in the core shipment transmittal form signed by the Project Manager that accompanies each core shipment. Keys for the padlocks sealing the door to the core transportation truck are only held by the Project Manager at the exploration site and by the Supervisor at the core splitting facility in Mendoza. The trip from Altar to Mendoza is approximately 430 km and takes an average of 17 hours.

Upon arrival of a core shipment to the core splitting facility, the Supervisor there first checks the numbers on the 3 strap locks that seal the door of the truck against those recorded on the core shipment transmittal form accompanying the shipment. After opening the locks, the core boxes are inspected and inventoried against the shipment transmittal list. Any damaged or missing boxes or numbering discrepancies are immediately reported to the Project Manager at Altar.

Prior to splitting, the whole core is photographed using a Canon digital single-lens reflex camera with a 10.1 megapixel imaging sensor. Each photograph includes 3 core boxes laid out in sequential order in a specially-constructed photographic stage.

The photographs are stored on a computer and backed up digitally on two separate DVD back up disks kept in different locations.

Following core splitting and sampling the core boxes are re-sealed with their wooden tops and stacked sequentially in steel racks in the core storage warehouse of the splitting facility in Mendoza.

Following laboratory analyses, all sample pulps and rejects are stored for a short period of time by Acme Analytical Laboratories in a neat and orderly manner inside the preparation lab. Peregrine promptly retrieves the pulps and rejects from Acme (once a day during periods of high sample volume) and returns them immediately to the Peregrine core splitting facility where they are catalogued and permanently stored. The pulps are archived in cardboard boxes placed on the core racks next to the drill core for the corresponding hole. The rejects are kept in specially made bins built on pallets and stacked in order in a section of the core storage facility with one drill hole to each bin.

To ensure smooth handling, the core boxes are transported between the core splitting building and the adjoining core storage warehouse on pallets using a hand operated forklift, which is also used to move and stack the reject bins.

The Mendoza core splitting and storage facility is housed within a secure private property with a single street entrance protected by 2 locked gates (Figure 11-2). The perimeter of the property has a high concrete and brick wall topped with barbed wire. Both the core splitting room and the core storage warehouse are completely enclosed and sealed concrete and brick buildings with single entrances and large sliding steel doors that are kept locked at night. Strategically-located inside the buildings are several highly visible fire extinguishers. There is a 24hr safety guard controlling vehicles and persons arriving and leaving the warehouse.

**Figure 11-2 Mendoza storage facility**

### 11.2.3 Core Logging Procedure

Core logging is carried out by teams of two geologists working together on individual drill holes. The logging involves recording lithological units, lithological descriptions, alteration type and intensity, mineralization type and intensity, sulphide mineralogy, veining and fracturing type, and structural orientation and intensity. Information is entered into hand written field logs and each evening transferred to digital versions of the same field log format as Excel spreadsheets.

Measurements of core recovery and RQD are carried out by experienced geological assistants under the supervision of the geologists. Recovery and RQD measurements are first recorded in hand written sheets and subsequently entered into individual Excel spreadsheets.

The core is subsequently marked by a geologist with a cutting line to avoid sampling bias that may occur due to veins or veinlet clusters oriented obliquely to the drill core axis. The logging geologists are also responsible for laying out, measuring and marking the core sample intervals and for laying out the sequencing of reference standards, blanks and core duplicates to be inserted into the sample stream in the core splitting facility in Mendoza.

When the logging of a specific hole is completed and all data has been recorded and assembled, a master log of the hole is created comprising an independent Excel spreadsheet for each hole with worksheet tabs corresponding to a Header sheet (with the drill hole ID, collar coordinates and elevation, drilling contractor and rig number and the start and completion dates of the hole), followed by sheets for Down-hole Surveys, Sample Intervals, Box Registry, Recovery, RQD, Field Log, Assays, Lithology, Alteration, Veins and Fractures.

At the project site the geologists also prepare and regularly update a Sample Batch Control Sheet for each drill. These sheets track the samples through the entire sampling process and keep a record of the Laboratory Work Order, Laboratory Report Date, Peregrine Batch Number, Drill Hole ID, Depth Interval, Sample ID, Laboratory Delivery Date, Reject Return Date, and Pulp Return Date. They also show the predetermined position of all Peregrine Reference Materials within the sample stream. The Project Manager emails the updated control sheets from Altar to the Supervisor at the core splitting facility on a regular basis.

When sample assay results have been received and merged into the master logs, all of the data is transferred to a single Drill Hole Database file designed to facilitate computer plotting of assay data and geological parameters. All final digital data is backed up onto two separate hard drives kept in different locations.

### 11.2.4 Core Sampling

Drill core is sampled in a continuous sequential fashion commencing at the beginning of core recovery and terminating at the end of the hole. Drill core samples are routinely taken at regular 2.00-metre intervals. Individual sample lengths vary from this rule for the first and last sample of a drill hole when the depth of overburden (and hence the depth to the beginning of the first sample of the hole, is not a multiple of 2.00 metres) or when a hole ends at a depth that is not a multiple of 2.00 metres.

The drill core is split by well-trained and experienced personnel using 3 industry standard circular rotary rock saws with diamond saw blades, and using a constant flow of fresh clean water to cool and lubricate the saw blades. For each sample interval the core is split in half according to the cutting line marked by the logging geologists at the project site. One half of the split core is stored in the wooden core boxes and the other half, constituting the sample to be assayed, is placed in clean new transparent high-strength plastic sample bags.

The outside of each sample bag is marked with a specific individual sample number using a permanent waterproof marker and triplicate pre-printed waterproof paper tickets with the same sample number are added to the bag. In the Primary Laboratory, one sample ticket accompanies the pulp that goes for analysis, one remains with the pulp reject and one remains with the coarse reject. A fourth identical sample ticket is stapled to the inside of the core box to mark the position of the beginning of each sample, and the fifth and final sample ticket remains in the assay sample ticket booklet and is marked with the drill hole ID and the depth interval of the corresponding sample.

After the sample and the triplicate sample tickets have been placed in the sample bag, the bag is doubly sealed with two plastic strap locks. An ordinary strap lock is first used to close the bag as tightly as possible low around the neck. A second custom labelled strap lock engraved with Peregrine's name and a unique number matching the sample number on the sample tickets is then secured in such a way that it pierces the neck of the bag above the first ordinary strap lock. Using this tamper-preventative measure, it is impossible to slip the ordinary strap lock over the neck of the bag without first removing the custom numbered strap lock, which cannot be removed without breaking it or cutting the bag. The laboratory is required to notify Peregrine immediately if any samples do not arrive with the bags in good condition and both seals intact.

The doubly-sealed sample bags are placed in new rice sacks in sequence and the rice sacks are then sealed with strap locks and stacked inside the core storage warehouse awaiting transportation to the Primary Laboratory. Each rice sack is clearly labelled using a permanent waterproof marker with Peregrine's name, the contained sample number sequence, and a unique sequential Batch Number. Each dispatch of samples submitted to the Primary Laboratory is accompanied by a Sample Dispatch Transmittal Form, which is prepared and signed by Peregrine's core facility supervisor and then signed by the receiving laboratory representative upon unpacking, inventorying, and confirming that all samples listed in the transmittal form are received in good condition with seals intact.

Core splitting is carried out under the direct supervision of the core splitting facility supervisor and is also monitored on a regular basis by a geologist.

### 11.2.5 Recovery

The core recovery for the drilling undertaken by Peregrine, ALD-16 through ALD-140 includes 17443 determinations and averages 97.3% with a median of 100.00%. Recovery data for the first 15 historic holes could not be located. A total of 15,904 intervals also had RQD measurements which averaged 54%.

## 11.3 Core Photography

Up until 2007, core was not photographed. However, Peregrine has photographed wet core, prior to core splitting, for all of the 2007-2010 drilling. All split core from holes ALD-1 through ALD-40 was photographed in 2007. The photography setup is illustrated in Figure 11-3.

**Figure 11-3 Drill Core Photography (2008)**

## 11.4 Data Entry

Altar has implemented an Excel-based database, where logging and sampling data are manually entered, and assay and survey data are digitally entered.

## 11.5 Sample Security During Core Cutting

Sample transportation and security is discussed in Section 11.2.4 Core Sampling above.

# 12.0 SAMPLING METHOD AND APPROACH

## 12.1 Stream Sediment

There is very limited documentation on the CRA stream sediment sampling program. No information was available to AMEC for the sampling method and approach for this sample type.

## 12.2 Talus

There is very limited documentation on the CRA talus sampling program. The sample fraction analysed was -80 mesh.

## 12.3 Rock Chip

There is very limited documentation on the CRA rock chip sampling program. Samples taken from road cuts averaged 10 m in length. The grab sampling was completed over the entire project, but a concentration of sampling occurred over the zone of silica ledges. In general, sample weights were about 3 kg to 5 kg.

## 12.4 Drilling

Drill core is mechanically split in half with an industry-standard diamond saw. One half of the drill core is stored for reference, and the other half is bagged, numbered and submitted to the primary laboratory for preparation and analysis. The geologists define the half-core cutting line in the project during core logging. The saw operator cuts the core as indicated by the cutting lines. (no hydraulic splitters were used in 2008 or in the 2010 season.

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# 13.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

## 13.1 Rock Chip

Rock chip samples were analysed by Bondar Clegg laboratory (now part of the ALS Chemex group). Sample preparation and analytical methodology is not documented in information provided to Peregrine.

Samples were analysed by Bondar Clegg using inductively coupled plasma (ICP) Analytical Suite G33, comprising Au (ppb), Ag (ppm), Al (%), $Al_2O_3$ (%), As (ppm), Ba (ppm), Bi (ppm), Br (ppm), Ca (%), CaO (%), Cd (ppm), Ce (ppm), Co (ppm), $CO_3$ (%), Cr (ppm), $Cr_2O_3$ (%), Cs (ppm), Cu (ppm), Eu (ppm), F (ppm), Fe (%), $Fe_2O_3$ (%), FeTotal (%), Ga (ppm), Ge (ppm), Hf (ppm), Hg (ppm), In (ppm), Ir (ppm), K (%), $K_2O$ (%), La (ppm), Li, LOI (%), Lu (ppm), Mg (%), MgO (%), Mn (ppm), MnO (ppm), Mo (ppm), Na (%), $Na_2O$ (%), Nb (ppm), Nd (ppm), Ni (ppm), P (ppm), $P_2O_5$ (%), Pb (ppm), Pd (ppb), Pt (ppb), Rb (ppm), Re (ppm), S (%), Sb (ppm), Sc (ppm), Se (ppm), $SiO_2$ (%), Sm (ppm), Sn (ppm), Sr (ppm), Ta (ppm), Tb (ppm), Te (ppm), Th (ppm), Ti (ppm), $TiO_2$ (%), Tl (ppm), U (ppm), V (ppm), W (ppm), Y (ppm), Yb (ppm), Zn (ppm), Zr (ppm). Lower analytical detection limits for these elements are not available to AMEC.

An Analytical Suite G31 is also documented in the Excel spreadsheet provided to Peregrine, detailing the rock chip sampling program; however, the columns for analytical results are empty, and no other information is available.

## 13.2 Drilling

Peregrine uses ACME Analytical Laboratories (ACME), based in Santiago as the primary analytical laboratory. ACME is an international analytical service, and is corporately accredited to ISO 9001:2000 (ACME, 2007).

Samples are prepared by ACME at a preparation facility in Mendoza. A person contracted by Peregrine regularly delivers the samples from the core-cutting facility to the ACME preparation laboratory, after which the pulps are transported to Santiago for assaying. NMS-Geosim visited the sample preparation facility in Mendoza in 2008 and reviewed its procedures.

A chain of custody procedure is strictly followed. Samples are organized in batches of 34 samples. Upon receipt, samples are sorted by sample number and individually weighed, and a job order is issued. Sample preparation is as follows:

- Drying to 60°C on a large electric oven with automatic temperature control
- Crushing to better than 85% passing -2 mm using a Rocklabs Boyd jaw crusher

- Splitting using a Labtech Hebro rotary splitter to obtain a 0.5 kg subsample; if samples are small, then a Jones splitter is used

- Pulverizing using a Labtechnics LM-2 puck-and-ring pulverizer to 95% passing 0.106 mm.

After preparation, a 150 g pulp sample is bagged and submitted to ACME in Santiago for Au, Cu, Mo, Ag and As analysis.

Protocols supplied by ACME are summarised in Table 13-1. Au is determined by fire assay with an AAS finish, using a 30 g aliquot (detection limits: 5 ppb to 10,000 ppb). The total copper (CuT), Mo, Ag and As determinations use hot aqua regia digestion and AAS finish on a 1 g aliquot. Copper and molybdenum are assayed to minimum detection limits of 0.001% for CuT and Mo. Silver and As values are reported using the same digest and finish, but to a detection limit of 0.5 ppm for Ag and 10 ppm for As, using the "geochemical method".

**Table 13-1 Analytical Protocols, ACME (supplied by ACME Sudamerica)**

| Analytical Protocol Number | Procedure |
|---|---|
| AAS-009 — Au | Add 180 g of flux every crucible; weigh 30 g of sample; mix flux with sample; transfer the rack on the furnace fusion for 1 h; transfer the rack to the cupellation furnace for 45 min; transfer the prills to the digestion rack; add 1 ml $HNO_3$ 1:1 and 1.5 ml HCl; read by AAS. |
| AAS-001 — Cu–Mo | Weigh 1.0 g sample into 100 ml flask. Add 50 ml aqua regia mixture to test tube. Digest in water bath for 3 h (90ºC) and mix every 15 min. Add 2 ml $AlCl_3$ and $H_20$. Cool. Mix the solution thoroughly then decant into a numbered test tube. Discard the excess solution. Read by AAS. |
| AAS-005 — Ag– As | Weigh 0.5 g sample into test tube. Add 5 ml aqua regia acid mixture to test tube. Digest in water bath for 60 min (90ºC) and mix every 15 min. Add 5 ml $H_20$ and read by AAS. |

From time to time, Peregrine personnel collect the coarse and fine reject bags for permanent storage at the Mendoza core storage facility.

### *Quality Assurance/Quality Control (QA/QC)*

#### ***General***

Samples are organized into batches of 34 samples made up of 29 routine samples and 5 reference samples. The 5 Peregrine reference samples include two standards, one coarse blank, and two core (field) duplicates. The positioning of the reference materials and core duplicates within the sample stream in each batch is predetermined by the geologists at the project site and incorporated into the field logs during the core logging process. Sample Batch Control Sheets showing the location of the reference materials within the sample number sequence for each batch are prepared on site and regularly updated and emailed to the supervisor at the core splitting facility. Personnel in the core splitting facility complete the sample stream in each batch using the specific reference materials and the numbering sequence indicated in these control sheets.

***Standards***

In 2007, Peregrine prepared 5 different standards by blending drill core rejects from prior campaigns at Altar. These are designated STD 3 through STD 7 to distinguish them from STD 1 and STD 2 used in previous drilling programs. They were blended to give resulting grades of approximately 0.3% Cu, 0.5% Cu and 0.75% Cu and designed to range from 0.07 to 0.10 g/t Au. One of the standards was blended to give a resulting grade of approximately 0.015% Mo. Three of the standards were made using the "Quartz Diorite Porphyry¨ host rock lithology and two were made using the ¨Vitric Tuff¨ host rock lithology. The standards are also differentiated according to the different sulphide mineralization types observed at Altar.

The standards were processed by CDN Resource Laboratories Ltd. located in Vancouver, Canada. For each standard, 100 Kg of reject material was used. After blending and homogenization the standards were packaged in approximately 100 g portions in tin-tie kraft envelopes, and further packaged in groups of 20 envelopes into heat-sealed vacuum-packed plastic bags for shipping and storage. Dr. Barry Smee of Vancouver, Canada supervised and monitored the standard preparation procedure and the selection of 7 internationally certified laboratories that participated in the Round Robin assays. Dr. Smee also evaluated the assay results from the seven laboratories and reported to Peregrine the mean and standard deviation of each standard as summarized in Table 13-2.

Based on the core logging observations, the geologists at the project site select 2 different standards that best suit the mineralization host rock, sulphide mineralogy and estimated grade range for each batch of core samples.

**Table 13-2 Certified Standard Reference Material Statistics**

| Standard | Element | Mean | Std Dev |
|---|---|---|---|
| **STD 3** | Cu % | 0.676 | 0.024 |
| | Au ppm | 0.074 | 0.005 |
| | Mo % | 0.0049 | 0.0005 |
| | As ppm | 181.9 | 11 |
| **STD 4** | Cu % | 0.474 | 0.021 |
| | Au ppm | 0.120 | 0.006 |
| | As ppm | 132.9 | 9.5 |
| **STD 5** | Cu % | 0.280 | 0.016 |
| | Au ppm | 0.096 | 0.007 |
| | As ppm | 256.2 | 16.5 |
| **STD 6** | Cu % | 0.452 | 0.02 |
| | Au ppm | 0.077 | 0.006 |
| | Mo % | 0.0037 | 0.0005 |

| Standard | Element | Mean | Std Dev |
|---|---|---|---|
| | As ppm | 438.9 | 22 |
| STD 7 | Cu % | 0.297 | 0.011 |
| | Au ppm | 0.043 | 0.003 |
| | Mo % | 0.0141 | 0.001 |
| | As ppm | 492.5 | 21 |

***Blanks***

In 2007, Peregrine acquired a large quantity of homogeneous granite in the form of slabs from a decorative stone quarry near Mendoza. A total of 21 samples of this material were submitted to Alex Stewart (Assayers) Argentina S.A. in Mendoza, where they were assayed for Cu, Au, Mo, Ag and As to certify the homogeneity of the material and the absence of significant detectable quantities these metals. At the core splitting facility this material is washed and broken into convenient-sized pieces and used for the coarse blank submitted with each sample batch. The position of the coarse blank within each batch is predetermined by the geologists at the project site.

The positions of the 2 core duplicates within each batch are pre-selected in random fashion by the logging geologist at the project site so that there are no discernable patterns of placement. Core duplicates comprise twin quarters of the drill core.

***QC Failures***

The Project Manager at the Altar Project maintains a Quality Control Worksheet that records on a batch by batch basis all of the assay results received from the laboratory, and allows for easy isolation and analysis of results returned for Peregrine Reference Materials. By this method QC Failures are quickly identified along with the corresponding sample batches and affected sample numbers.

Peregrine uses the following rules to define conditions that result in a QC Failure:

Standard Failures: Standards for Cu and/or Au that fall beyond the Round Robin mean +/- 3 standard deviations are Standard Failures due to accuracy. Two adjacent standards for Cu and/or Au that fall beyond the Round Robin mean +/- 2 standard deviations on the same side of the mean are both Standard Failures due to bias.

Blank Failures: Coarse blanks that are higher than the Warning Limit for Cu and/or Au are Blank Failures. The Warning Limit is defined as 5 times the background value for each of Cu and Au. The background values for the Peregrine coarse blanks are 10 ppm for Cu and 0.01 ppm for Au, and the Warning Limits are 50 ppm for Cu and 0.05 ppm for Au.

Core duplicates are not used to pass or fail a batch of samples but are used for the calculation of overall sampling precision.

In response to Standard Failures, the corresponding analytical batches are examined to determine if they occur within an economically-significant mineralized intersection. If this is determined to be true then the analytical batches from the previous acceptable standard to the following acceptable standard are re-assayed from the pulps. If this is not determined to be true then no action is required.

In the case of Blank Failures, the corresponding analytical batches are examined to determine if carryover contamination could affect the grade of samples within an economically-significant mineralized intersection. If this is determined to be true then the analytical batch from the previous acceptable blank to the following acceptable blank is prepared again from the rejects and re-assayed. If this is not determined to be true then no action is required.

When a series of pulps are re-assayed due to a Standard Failure the new batches are prepared using the same QC procedure that was applied to the original routine sample batches. The pulps to be re-assayed are organized into batches of 34 samples made up of 29 routine samples and 5 reference samples. The 5 Peregrine reference samples include two standards, one pulp blank, and two pulp duplicates. In 2007, Peregrine purchased a supply of prepared and certified pulp blanks from Alex Stewart (Assayers) Argentina S.A. in Mendoza. The pulp blanks were packaged in approximately 100g portions in tin-tie kraft envelopes. A total of 10 samples of this material were assayed by Alex Stewart for Au and 39-element ICP-MA to certify the homogeneity of the blank material and the absence of significant detectable quantities these metals.

When a series of rejects are re-assayed due to a Coarse Blank Failure the new batches contain 29 routine samples, plus two standards, one coarse blank and two reject duplicates.

When the results from the re-assayed pulps or rejects are received they are screened for Standard or Blank Failures. If none occur, the new assays replace the corresponding original assays in the database only if the new assays show significant differences with respect to the original assays. All QC Failures and follow-up actions are recorded in a Table of Failures, which documents the Laboratory Work Order, Failure Sample Number, Failure Type, Reason for Failure, Action Taken, Date New Data Received, Work Order of New Data, and Date Entered into Master Database.

***Primary Laboratory Procedures***

The Primary Laboratory is Acme Analytical Laboratories (South America) who operate a sample preparation laboratory in Mendoza and a full analytical laboratory in Santiago, Chile. On a regular basis, Peregrine delivers the batches of doubly-sealed core samples to the Acme sample preparation laboratory in Mendoza, which is located a short distance from the Peregrine core splitting facility.

Upon delivery to the Primary Laboratory the Peregrine batches are combined with the laboratory's internal reference samples (see ¨Primary Laboratory Procedures¨ below) to make up the final laboratory batches.

Core Sample Preparation

All drill core samples are dried with the temperature of the drier not to exceed 70° C. The core is initially crushed to 85% -10 mesh size using a single stage jaw crusher. The jaw of the crusher is cleaned with air between every sample and with coarse quartz at the beginning of every batch and after every 10 samples. A sieve test to ensure quality control is done at least every 30 samples and the results of the sieve test are reported each month to Peregrine. The -10 mesh crushed sample is split with a Jones style riffle splitter to 800g to 1,000g +/- 10% size after initial homogenization. For every batch as defined by Peregrine a preparation duplicate is taken at the splitter. This preparation duplicate is then treated as a normal sample.

The 800g to 1,000g split is pulverized to 95% -150 mesh in an LM-2 pulverizer in a single pass. The pulverizer is cleaned with a quartz wash at the beginning of every batch and after every 10 samples, or more frequently if considered necessary. A sieve test to ensure quality control of the -150 mesh sample is done at least every 30 samples and the results of the sieve test are recorded.

For each Work Order the laboratory analyzes a coarse preparation blank, which has gone through the entire sample preparation procedure, and generates a preparation duplicate for every 40 samples. Acme reports these results on a separate QC page.

Analytical Laboratory

Pulps prepared at the Acme preparation laboratory are transported on a regular basis by Acme to its analytical laboratory in Santiago, Chile, where Acme performs the following analyses on all core samples submitted:

1. 32-element ICP-ES using an Aqua Regia digestion

2. Au by Fire Assay using a 30 gram sample with an AAS finish

3. For all ICP analyses for Cu that exceed 5,000 ppm, the sample is repeat-assayed for Cu by AAS using an Aqua Regia digestion.

The results for all ICP analyses are reported in ppb or ppm units as appropriate, or percent to 3 decimal places. The results for all Au and Cu assays are reported to one decimal of uncertainty- for Au in g/t, to 2 decimals; for Cu in %, to 3 decimals.

For ICP-ES the analytical laboratory analyzes 1 blank, 2 pulp duplicates and 2 geological standards for every 60 samples; for Fire Assay the laboratory analyzes 2 blanks, 2 pulp duplicates and 2 geological standards for every 84 samples; and for AAS the laboratory analyzes 1 blank, 2 pulp duplicates and 2 geological standards for every 34 samples. Acme reports the results of these internal reference materials to Peregrine on a separate QC page with each Work Order. The blanks are accepted within 2 times the detection limit of the corresponding analytical method. The pulp duplicates are accepted within a 10% variation. The geological standards are International Certified Reference Materials which have established means. They are accepted within error limits of 2 standard deviations of the mean.

After completion of the 2010 drill program, core intervals with anomalous molybdenum content from current and past drill programs were re-analyzed using ACME's Group 8TD method involving 4-acid digestion which resulted in significantly higher grades over the ICP method.  Approximately 4300 intervals have been analyzed at the time of this report.

Acme also carried out sequential Cu analyses (Group code G904) on 8895 samples from the upper portion of the mineralized zone.  Results yielded grades for cyanide-soluble (CuCN), acid soluble (CuHS) and residual (CuR) copper.

*Secondary Laboratory*

Peregrine uses Alex Stewart (Assayers) Argentina S.A. in Mendoza as the Secondary Laboratory. The Project Manager makes a random selection of pulps representing 5% of the sample stream from within economically-significant mineralized intersections originally assayed by the Primary Laboratory. These pulps are submitted to Alex Stewart for Check Assay on a regular routine basis. The frequency of these Check Assay submittals is approximately once per month during the drilling campaign.

*Bulk Density Measurements*

When the core has been split and sampled and the assays have been reported and incorporated in the Drill Hole Database, Peregrine geologists select solid pieces of split drill core from 20 to 30 locations within each hole to be submitted to Alex Stewart Laboratories in Mendoza for bulk density measurement.

In the laboratory the core samples are first cleaned with dry compressed air and then oven dried for 2 hours at 105º C. They are then allowed to cool to ambient temperature and weighed. Following this dry weighing they are dipped in a bath of melted solid paraffin at 60º C and then left to drain and cool on racks before repeating the weighing with the paraffin coating. Following this they are weighed in an empty pignometric balance and then re-weighed in the pignometric balance with distilled water.

The bulk density is calculated by the formula: Bulk Density (g/ml) = Dry Weight of Sample / (Volume of Water Displaced x Volume of Paraffin). The values are reported to 2 decimal places.

*Database Backup System*

Peregrine has in place both a digital and hard-copy data backup system. The site for primary data backup is Peregrine's Santiago, Chile office where original hard copies of all paper field entry forms, drill logs and documents are filed. On a regular basis, the digital database is also backed up onto a portable hard drive kept in the Santiago office. The secondary backup site is Peregrine's Vancouver, Canada office where an identical file archive is kept of hard copy data and a second portable hard drive is located for storage of digital data.

Peregrine has in-house computer servers and networks in both the Vancouver and Santiago offices and these servers are being used for regular data backup. All hard copy data is scanned and stored digitally in the two locations as well as in paper form.

## 13.3 Bulk Density

In 2010, 1005 core samples of the various lithologies, alteration and mineralization styles were measured for specific gravity ("SG") at Alex Stewart laboratories. Results of this work are summarized in Section 17 and a description is provided of how this information has been used to establish block model values for resource reporting.

## 13.4 Security

In 2010 a new 2000 $m^2$ warehouse was acquired close to the ACME preparation laboratory. Advantages of the new site include:

- Larger size
- Truck loading-unloading facilities
- 24hr security and a safety guard

- Security cameras at all 4 sides and motion sensors connecte to an alarm system
- Office washroom and kitchen facilities
- WIFI installed
- Bright open space with skylights
- Closed rooms for core sawing

Drill core is stored in racks within the main part of the warehouse (Figure 13-1). All pulp rejects are stored in two separate rooms, ordered in small racks and well-labeled cartons. All coarse rejects are stored in labelled rice bags in wooden bins – each labelled on all 3 sides with the hole ID's.

**Figure 13-1 Drill Core Storage**

## 13.5 Conclusions

The authors are of the opinion that the sample preparation, security and analysis meets or exceeds industry standards and is adequate to support a mineral resource estimate as defined under NI 43-101.

# 14.0 DATA VERIFICATION

AMEC conducted a number of independent verification checks on the Altar Project.

Checks comprised a site visit, a review of the geological and mineralization interpretations for the Project, drill core inspection, review of core logging, sampling and assay protocols and methods, review of sample security measures and sample storage, verification of QA/QC protocols and methods, validation of QA/QC data received to date on the Peregrine drilling programs, confirmation of the appropriateness of the proposed metallurgical test work that is based on existing core samples, and a review of, and contributions to, the proposed Peregrine work programs for 2007–2008.

AMEC did not take independent samples of the mineralization on the project, or review the Rio Tinto QA/QC programs, and has relied on external experts for the information on tenure, environmental and surface rights.

In March 2008 and May 2010, NMS - Geosim undertook site visits including a review of the geological and mineralization interpretations for the Project, drill core inspection, review of core logging, sampling and assay protocols and methods, review of sample security measures and sample storage, QA/QC protocols and methods. In addition spot checks of assay entries were undertaken and data validation was done on sections and plans during model preparation.

## 14.1 Geology, Mineralization and Deposit Model

NMS – Geosim reviewed the geology and mineralization interpretation on section and plan and utilized this information to develop a three dimensional model of the deposit lithology and grade distributions.

## 14.2 Sample Security and Sample Storage

Peregrine sample security protocols for 2010 are summarized Section 11.2 and further detailed in “QA-QC Procedural Guidelines, Altar Project” by Jeff Toohey, M.Sc. P. Eng.

## 14.3 Assay Data

Peregrine has established the digital data duplicate facility described in the section above. NMS – Geosim has checked 10% of the assay data prior to development of the resource model.

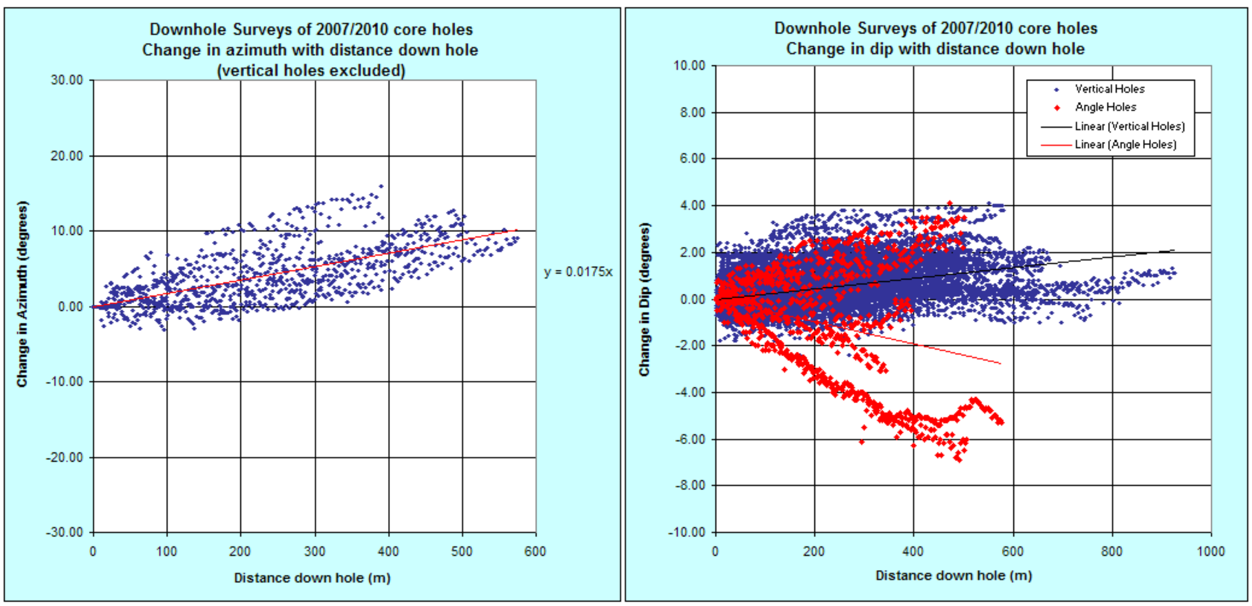
## 14.4 Drill Hole Collars

NMS – Geosim checked collar locations on sections and plans relative to the updated topographic surface during model development as well as GPS collar location checks in the field during the site visit.

## 14.5 Down Hole Surveys

NMS – Geosim checked analyzed the down hole survey data for trends and errors. No issues of concern were identified. The range of downhole survey deviations are illustrated in Figure 14-1. The angle holes tended to veer to the right and could either steepen or flatten with depth by as much as 2 degrees over 100m. Vertical holes stayed relatively straight averaging about 1 degree of flattening every 400m.

**Figure 14-1 Downhole survey analysis**




## 14.6 Independent Sampling

Independent samples were not submitted for verification testing. However NMS-Geosim did observe many areas of visible copper mineralization in drill core that was consistent with copper grades reported by the laboratory.

## 14.7 QA/QC

QA/QC procedures for historic drill programs were reviewed in the previous technical report.

***Protocols***

The QC protocol used at the Altar Project included the insertion of a single standard, made by Alex Stewart in the 2005 program, and two standards also made by Alex Stewart in the 2006-2007 program. The 2005 program also inserted field duplicates. The 2006-2007 drill program also included insertion of a blank sample to monitor possible contamination. All drill programs included insertion of a field duplicate.

A table of logic was used to define failed QC samples.

**Rule 1:**

Standards for Cu and/or Au beyond the mean ± 3 SD limits are failures (accuracy)

**Rule 2:**

Two adjacent standards for Cu and/or Au that are more than 2 SD on the same side of the mean are both failures (bias). These may not be the same standard.

**Rule 3:**

Field blanks that are more than the “Warning Limit” for Cu and/or Au are failures.

*Note: The “Warning Limit” is defined as 5 times the background value for each of Cu and Au.*

*Note: Field duplicates are not used to pass or fail batches of samples, but are used for the calculation of overall sampling precision.*

**Blank Results**

There were 7 blank failures in Cu and 2 failures in. All affected batches were re-analyzed. No evidence was found for sample contamination in the preparation facility. Several of the failures were suspected sample number mix-ups and other Cu failures attributed to rare minor Cu content in the blank coarse granite material.

**Standard Sample Results**

Monitoring of reference standard performance resulted in a total of 19 failures for Cu and 45 for Au. Most of the Au failures were in batches containing no significant Au mineralization and it was not considered necessary to re-assay these. Several instances were identified where the wrong standard label was entered. A total of 14 batches were re-assayed due to Cu standard issues, 11 batches due to Au failures and 3 batches due to combined Cu/Au failures.

### Duplicate Results

A total of 832 field duplicates were analyzed from the 2010 drill program. Precision was calculated by the Thompson – Howarth (T-H) method using the average difference between duplicates vs the mean grade of the duplicates to determine relationship between precision and grade Figure 14-2 and Figure 14-3.

The overall precision for Cu is just below 15% for field duplicates at 1% Cu. This is a reasonable level for a mineral deposit of this type. Both prep and pulp duplicates show very good precision of 5% or less.

The precision of Au duplicates is about 27% for field duplicates, 12% for coarse rejects and 6% for pulp duplicates. This is regarded as acceptable precision.

**Figure 14-2 Thompson-Howarth Plot, Precision of Copper Assays**

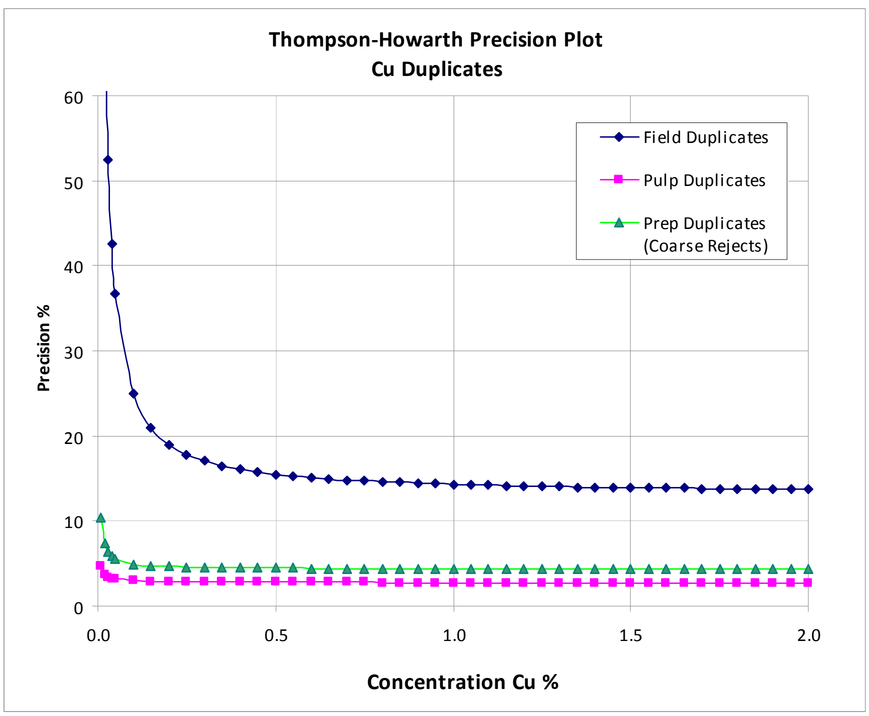
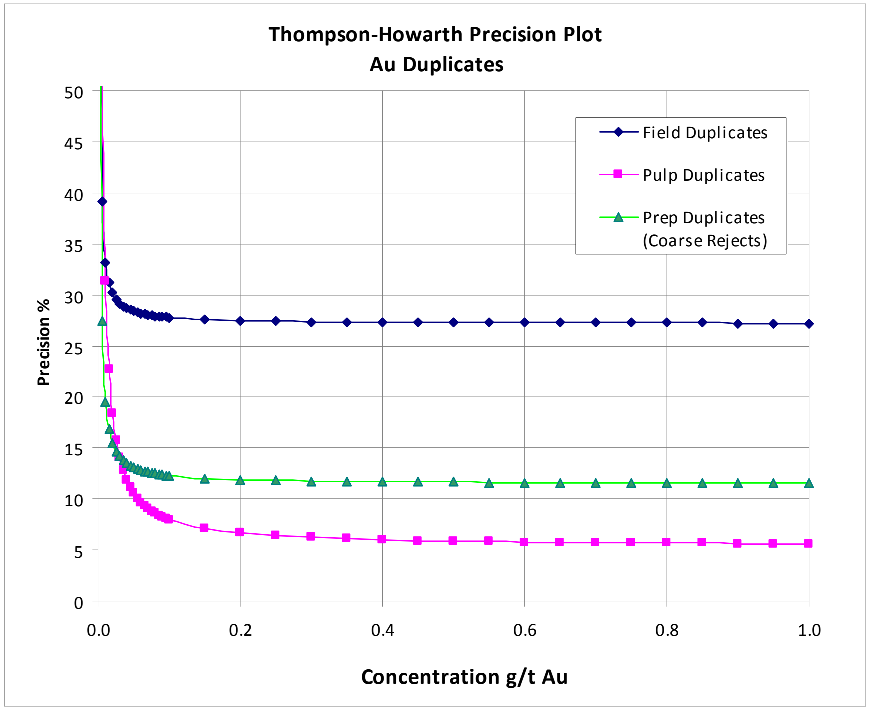


**Figure 14-3 Thompson-Howarth Plot, Precision of Gold Assays**




## 14.8 Density Determinations

Peregrine have expanded the density database during 2010 with determinations to appropriate industry standards.

## 14.9 Metallurgical Testwork

The metallurgical testwork program results are summarized in Section 16 of this report.

# 15.0 ADJACENT PROPERTIES

The Altar Project is located in a known mining area. It is surrounded by early-stage exploration properties, none of which have more than historic reconnaissance-stage drilling.

The two closest projects to the Altar Project are both porphyry copper–molybdenum deposits of Miocene age: the open pit Los Pelambres mines (Antofagasta plc (Antofagasta) and partners) in Chile and the El Pachon project, currently under development by Xstrata plc (Xstrata), in Argentina. Both projects report resources and reserves under the Australian Joint Ore Reserves Committee (JORC) Code.

The Lost Pelambres deposit is developed in a sequence of andesitic lavas, flow breccias and volcaniclastic sediments that has been intruded by small irregular dioritic to granodioritic plutons, and by an approximately 5 x 2 km quartz diorite stock.

A number of related alteration and mineralization events are associated with emplacement of multiple centres of small (~200 m in diameter), vertically-zoned bodies of aplite, pegmatite, and hydrothermal breccia within the quartz diorite. Each centre formed a bornite-rich core, grading out through chalcopyrite to pyrite Hypogene mineralization commenced prior to the cessation of magmatism, taking the form of quartz stockwork veining, potassic alteration and breccia pipes. Late mineralisation occurs as pyrite veining with sericite halos. Mineralisation in both types is present in veins, with the only disseminated ore being in the alteration halos of these veins. Supergene enrichment is important at Los Pelambres, with five blanket-like zones formed vertically (Perello et al., 2007; Porter Geoconsultancy, 2003).

Los Pelambres commenced operation in 1999, with a 30 year projected mine-life. In the first quarter of 2010 production rate was expanded to 175,000 tonnes per day. Proven and Probable Reserves at the end of 2009 comprised 1.5 Gt at 0.64% Cu, 0.018% Mo, 0.03 g/t Au. Mineral resources total 6.1 Gt averaging 0.52% Cu, 0.011% Mo and 0.03 g/t Au (Antofagasta, 2009). Copper concentrate is transported by pipeline 120 km to the coast, where it is dewatered, dried and stored prior to shipment by sea. The molybdenum concentrate is packed in drums and shipped from the concentrator plant to roasters.

The El Pachón porphyry copper and molybdenum deposit is contained within an area of 2 x 2 km. Mineralization occurs mainly in andesite, diorite, tuff, and a siliceous breccia; the main intrusive phase is a diorite (Cambior, 1997). Some late stage intrusions, (Quartz–Biotite–Feldspar Porphyry and Tourmaline Breccias) are barren and cross the mineralized units (Falconbridge, 2005). Copper is associated with fracture filling, whereas molybdenum is primarily found within quartz veinlets. Approximately 94% of the mineralization lies within the primary zone (Cambior, 1999).

Resources are based on the results of 46,237 m of drilling in 247 holes and 662 m of underground work in two adits, completed during various campaigns between 1969 and 2003. El Pachon has current Measured and Indicated Resources, reported to a cut-off grade of 0.3% Cu totalling 0.81 Gt at 0.65% Cu and 0.02% Mo, with a further 0.57 Gt grading 0.5% Cu and 0.01% Mo in the Inferred category (Xstrata, 2009). Xstrata is currently developing the project; the deposit has the potential to produce on average over 200,000 t/a of copper in concentrate over a proposed 23-year mine life (Xstrata, 2007).

# 16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

## 16.1 Introduction

A preliminary metallurgical testwork program was initiated and completed during 2007 to assess two potential treatment routes: production of a flotation concentrate and heap leaching. Doug Halbe Consultant P.C. guided the metallurgical evaluation of the flotation option to produce a sulphide concentrate and Dr. W. Joseph Schlitt, Principal at Hydrometal, guided the option to produce copper by leaching crushed whole ore with an acidic ferric sulphate-based lixiviant.

Dawson Metallurgical Laboratories (DML) of Salt Lake City, Utah was selected to perform flotation testwork. DML was also selected to prepare the samples for flotation testwork and the individual samples for leach testing at McClelland Laboratories Inc. (MLI) of Sparks, Nevada.

The next phase of the metallurgical testwork began in 2010 and is carrying over into 2011. It also being guided by Dr. Schlitt and is considerably more extensive. It includes both bottle roll and column leach tests on various sulphidic ore samples using an acidic ferric sulphate lixiviant to extract copper. The same samples will also be subjected to milling and flotation testing to produce a sulphide concentrate. A separate testing program includes a series of bottle roll leach tests using a cyanide lixiviant to extract gold from the overlaying leach cap. Additional diagnostic work is being conducted in parallel. The diagnostic studies include comminution tests to establish the crushing and grinding work indices, the abrasion indices and the specific gravity of the different lithological units: quartz diorite porphyry, rhyolite and andesite. The diagnostic work also includes a determination of the water soluble content of different ore samples.

The comminution program is being performed by Phillips Enterprises LLC of Golden, Colorado (Phillips). The balance of the current testwork is being performed by MLI.

The results of the testwork programs done to-date are summarized in the sections below. The 2007 work is discussed first, followed by a synopsis of the current program.

## 16.2 Preliminary Flotation Testwork

### 16.2.1 Composites

The flotation studies focused on composites that represented two of the major sulphide ore types: an enriched material with mixed mineralization, but dominated by covellite (designated CC-CV) and a lower grade primary material dominated by chalcopyrite, (designated CP-BN). These were made up from contiguous 2 m intervals from representative depths and reasonable copper grades. These head grades were 0.86% Cu for CC-CV and 0.56% Cu for CP-BN

### 16.2.2 Flotation Testwork

Due to the limited amount of material, the flotation parameters were not fully optimized. Flotation testwork was limited to one set of grinding conditions and two sets of rougher flotation conditions, with cleaner tests run on the better of the two rougher flotation tests.

Staged rougher flotation tests were performed under the following conditions:

1. Grind of approximately $P_{80}$=150 mesh (105 micron).
2. Collector Scheme 1: Thionocarbamate-Mercaptobenzothiozole (TCB)/MBT;pH 9-10.
3. Collector Scheme 2: Sodium Isopropyl Xanthate/Dithiophosphate (NaIPX/DTP), pH 10-10.5.
4. Float in 4 stages, with total flotation time of 12 minutes.

One 2-stage cleaner test was conducted on each of the three composites, using the TCB/MBT scheme with rougher concentrate re-grind to $P_{80}$ 325 mesh (44 micron).

The overall results of the flotation tests are shown in the Table 16-1.

**Table 16-1 Flotation Testwork Results for Final Concentrate**

| | Composite CC-CV | | Composite CP-BN | |
|---|---|---|---|---|
| | Concentrate Grade - % | Percent Recovery | Concentrate Grade - % | Percent Recovery |
| **Copper** | 26 | 90 | 17 | 89 |
| **Iron** | 22 | 19 | 32 | 22 |
| **Arsenic** | 0.2 | 69 | 0.6 | 84 |

Other than the arsenic, the CC-CV and CP-BN composites produced clean concentrates. Final copper recoveries were encouraging at 89 to 90%, although the final concentrate grades need further optimization. Target concentrate grades are those achieved in the second cleaner stages - - 32% Cu with CC-CV and 25% Cu with CP-BN.

Arsenic head grades in CC-CV and CP-BN were 0.007% and 0.021%, respectively. By comparison, the global average arsenic level in the deposit is 0.028% As, a level that can generally be maintained by blending in the mine or at the concentrator. (See Section 17.8.1 for further details on the arsenic grade-tonnage relationship). Based on the Dawson test results with concentration ratios of 30:1 to 40:1, the average arsenic content in the final concentrate should range between 0.75 and 1.0% As. While these levels may incur a minor smelter surcharge, most smelters can accept such material. Conventional smelting will remove arsenic via the off gas or a low temperature roast can be used to remove arsenic prior to smelting.

### 16.2.3 Enargite Testwork

The deposit contains a relatively small amount of enargite, a copper-bearing arsenic mineral. To confirm our understanding of the leachability and flotation response of enargite, it was decided to prepare a sample that was anomalously high in enargite. Therefore, a third atypical composite was prepared by blending selected 2-m intervals of material containing enargite taken from different drill holes.

It is important to realize that this third composite was a special sample that did not represent any particular area of the deposit. This composite had a head grade of 0.21% As. Material with this As grade and higher only represents about 0.05% of the total tonnage in the Altar resource.

Flotation results with the high-arsenic composite were about as expected. The head grades were 0.94% Cu and 0.21% As. The final concentrate contained 23% Cu and 5% As at 94 to 95% copper recovery

### 16.2.4 Microscopy

Microscopic work supporting the DML testwork was undertaken by Dr. Erich Petersen, Acting Chair of the Department of Geology and Geophysics, University of Utah. Based upon photomicrographs, the copper mineralization in the CC-CV composite is almost entirely covellite and chalcopyrite rimmed with covellite. In the CP-BN composite the dominant copper mineral is chalcopyrite with less abundant covellite and enargite. In the high arsenic composite enargite and chalcopyrite are present in roughly equal amounts. Essentially all of the enargite particles in both the rougher and cleaner concentrates are free. In all three composites the dominant sulphide is actually pyrite. All samples represent sulphide mineralization in early stage quartz diorite porphyry.

### 16.2.5 Conclusions

- The CC-CV composite had a head grade of 0.86% Cu and produced a final concentrate that assayed 26% Cu and 0.2% As at an overall copper recovery of 90%.
- The CP-BN composite had a head grade of 0.56% Cu and produced a final concentrate that assayed 17% Cu and 0.6% As at an overall copper recovery of 89%.

- Approximately 50% of the copper losses in the rougher tailings were contained in the minus 20 micron size fraction.
- The CC-CV and CP-BN cleaner concentrates assayed 0.21% and 0.91% As, respectively. Almost all of the arsenic contained in the concentrates was present as enargite, $Cu_3AsS_4$.
- Mineralogical examination results indicated that the sulphide minerals were well liberated at the primary grind size used. Important copper minerals were chalcopyrite, covellite and enargite with minor tetrahedrite and bornite.

## 16.3 Preliminary Bottle Roll Leach Studies

### 16.3.1 Sample Characterization

The samples for testing were provided by Peregrine Metals. These were split from the same composites used in the flotation studies. The samples were from assay reject material from seven holes crushed to 100% minus ten mesh (< 2.0 mm). Samples in this size range are only suitable for bottle roll testing. Coarser material is required for column testing. All samples represent sulphide mineralization in early stage quartz diorite porphyry.

The material types were categorized into two broad types based upon mineralogy. Nevertheless, the samples were obtained from just seven drill holes and should not be expected to represent the entire resource.

- CC-CV came from shallow depths in holes ALD 08, 09, 19, 20, 36 and 39
- CP-BN is only found at great depth in holes ALD 08, 16 and 19.

The high in arsenic composite came from shallow depths in holes ALD 08, 16, 19 and 20. As previously noted in Section 16.2.3, this sample does not represent any particular part of the deposit.

### 16.3.2 Metallurgical Testing

An initial set of bottle roll tests was undertaken to gain an overall impression of the leachability and acid consumption. Nine composites were drawn from seven drillholes representing the samples described in Section 16.3.1.

Splits from the nine composites were run in triplicate for total copper (CuT). A sequential assay was also done on a split from each composite. The purpose of the sequential assay was to determine the acid (CuAS), cyanide soluble (CuCN) copper content and residual copper (CuR). Acid soluble copper species include oxides and other non-sulphide species. Cyanide soluble forms of copper include chalcocite, diginite, covellite, native copper and a portion of the bornite and enargite. The residual copper typically includes the chalcopyrite, the remainder of the bornite and other refractory minerals. The sum of the acid soluble and cyanide soluble copper components can give a reasonable indication of the copper that can be recovered in a typical heap leach operation. However, enargite is largely cyanide soluble but leaches poorly in an acidic ferric sulphate medium. Thus, it can affect the estimate of leachable copper.

A second set of bottle roll tests was undertaken as a parametric study to determine the effects of ferric iron and acid concentration in the leach solution. All of the tests were run on a split from the master CC-CV flotation concentrate to allow for a direct comparison of the leach and flotation recoveries.

***Recovery***

The Phase 1 program showed that the CC-CV material leached well. Recovery averaged 81% of total copper and over 90% of the readily soluble fraction in 55 days. The CP-BN material was more refractory with 24% recovery of total copper in 15 days. As expected, the high arsenic composites leached poorly, averaging only 15% recovery in 15 days. The presence of refractory material did not retard copper extraction from the more leachable copper minerals.

The Phase 2 program tests were run at concentrations of 0.5, 1.0, 2.5, and 5.0 g/l ferric iron and 2.5, 5.0, 10.0 and 15.0 g/l sulphuric acid. These concentrations cover typical ranges found in most commercial operations. All samples reached 72% recovery when leached for 48 days. Gangue acid consumption increased stepwise as the acid concentration increased. Hence, operation at low acid concentration would be preferred as it would give the same recovery at lowest acid consumption. The recoveries appeared to be independent of the ferric iron content. However, this may be misleading, as the tests were all run at 40% solids. Thus, even at the lowest ferric concentration, there was sufficient iron available to leach much of the copper.

***Acid Consumption***

Gross acid consumption averaged about 30 kg/t ore for eight of the nine composites. Bottle roll tests overstate acid consumption due to the fine particle size, so a commercial leach operation is estimated to consume about 13 to 15 kg/t acid on a net basis.

***Mineralogy Studies***

A mineralogical study was undertaken on head samples and residues of the Phase 1 program. Dr. Tommy Thompson of Economic Geology Consulting has prepared a report detailing the mineralogy of each of the nine composite samples.

In summary, all nine of the samples contained abundant pyrite. Chalcopyrite was abundant or present in all samples, except two of the high arsenic composites. Covellite was generally more abundant that chalcocite and both minerals are found in most of the composites. As expected, enargite was found in the high arsenic samples and also in the CP-BN sample. Traces of digenite and bornite were found in two composites each.

A comparison of the head and residue mineralogy showed that chalcocite and digenite leached very well unless encapsulated in a refractory constituent. In most samples covellite also leached but the degree of extraction was more varied. Minor leaching of the refractory phases was noted in a few samples. The limited amount of non-sulphide (acid soluble) copper leached very well.

***Water Solubility***

Water solubility tests were undertaken at MLI on the nine Phase 1 composites used in the bottle roll tests. Results showed that one composite, the chalcopyrite-bornite composite (BR-4) contained significant water soluble material. This sample lost 2.6% of its original weight. Multi-element analytical results showed the water soluble material is some form of sulpho-salt. High water soluble material content can have an impact on flotation plant design. Additional investigation has been included in the present metallurgical program.

***Biological Activity***

A third party laboratory, Sierra Environmental Monitoring, doing biological tests used standard culturing techniques to establish and count the bacterial population. Their results of the culturing and counts were negative. No significant bacterial populations were identified. However, as discussed below, current testing has shown that more than 55 days is needed for the indigenous bacterial populations to develop in the Altar ores.

### 16.3.3 Conclusions & Recommendations

The low arsenic CC-CV material appears to be a good candidate for a heap leach operation.

A comprehensive column leach test work program is needed to define additional process parameters and confirm the suitability of this material as feed to a leach operation. Initial tests should focus on effect of crush size on recovery. As discussed below, this program has now been initiated.

## 16.4 Current Metallurgical Testwork Program

Information on the location of the samples used in the program is presented first. In addition, several portions of the current program are now complete. These are reported next and include the comminution tests, gold cyanide leach tests, and the testing of the water soluble content. The column and bottle roll copper leach tests on the various ore types have been underway for about four months, but are expected to continue for at least two more months. Interim results are presented for this part of the program. The flotation work is on hold, pending completion of the mineralogical studies needed to help guide the program. Therefore, no flotation results will be presented at this time.

### 16.4.1 Location of Samples Used in the Metallurgical Program

A number of metallurgical holes were included in the 2009-2010 Altar drilling program. These were twinned off existing holes, with whole HQ core provided for comminution and column leach testing. All holes were located near the center of the mineralized zone and were relatively shallow. In all, a total of 61 two meter intervals from six metallurgical holes were used to prepare the sulphide composites used in the leach columns and related tests. The make-up of the 15 column composites is shown in Table 16-2. Head grades are shown in a later section.

Inspection of the table entries shows that only two of the three main ore lithologies are represented. No mineralized andesite was encountered during the metallurgical drilling; all material is porphyry unless listed as rhyolite. Thus, leach tests on andesite will have to await the next round of testwork.

**Table 16-2 Drill Hole Intervals Used in Metallurgical Sulphide Composites**

| Column No. | Description of Column Test | Sample Location | |
|---|---|---|---|
| | | Drill Hole No. | Interval(s), m |
| AC-1 thru 5 and 20 | Crush Size Composite | ALD 19 | 104-112, 114-116, 118-120, 124-126, 128-132 |
| | | ALD 46 | 88-90, 92-94, 104-106, 108-110 |
| | | ALD 48 | 76-78 |
| AC-6 | Porphyry Composite | ALD 46 | 112-114, 128-130 |
| | | ALD 49 | 168-172 |
| AC-7 | Rhyolite Composite | ALD 50 | 128-136 |
| AC-8 | Low-grade Composite | ALD19 | 102-104, 142-144 |
| | | ALD 49 | 154-156 |
| AC-9 | Mid-grade Composite | ALD-19 | 138-140 |
| | | ALD-48 | 78-80 |
| | | ALD-49 | 142-144 |
| AC-10 | High-grade Composite | ALD-19 | 150-152 |
| | | ALD-46 | 90-92, 122-124 |

| Column No. | Description of Column Test | Sample Location | |
|---|---|---|---|
| | | Drill Hole No. | Interval(s), m |
| AC-11 | 85% Solubility Composite | ALD-19 | 120-122, 132-134, 140-142 |
| AC-12 | 70% Solubility Composite | ALD-46 | 130-132 |
| | | ALD-49 | 152-154, 156-158 |
| AC-13 | 50% Solubility Composite | ALD-46 | 136-138 |
| | | ALD-49 | 174-178 |
| AC-14 | Porphyry Refractory I | ALD-48 | 116-122 |
| AC-15 | Porphyry Refractory II | ALD-48 | 108-110, 122-124, 140-142 |
| AC-16 | Porphyry Refractory III | ALD-46 | 142-146 |
| | | ALD-48 | 142-144 |
| AC-17 | Rhyolite Refractory I | ALD-50 | 96-98, 196-200 |
| AC-18 | Rhyolite Refractory II | ALD-47 | 50-54, 126-128 |
| AC-19 | Rhyolite Refractory III | ALD-47 | 56-58, 132-136 |

No metallurgical drilling was done in the oxidized leach cap. Therefore, only assay reject material was available for the gold leach tests. Due to the fine particle size of this material (minus 2.0 mm), only bottle roll testing was possible. Samples for these tests were taken from both the East Zone and the Central Zone and represented material from all three lithologies. Information on the location of the samples and the lithology is shown in Table 16-3 and Table 16-4. Head grades are given in a later section.

**Table 16-3 Central Zone Drill Hole Intervals Used in Leach Cap Gold Samples**

| Bottle Roll No. | Description | Sample Location | |
|---|---|---|---|
| | | Drill Hole No. | Interval, m |
| CY-14 | Porphyry | ALD-36 | 108-110 |
| CY-15 | Porphyry | ALD-43 | 44-46 |
| CY-16 | Porphyry | ALD-43 | 48-50 |
| CY-17 | Porphyry | ALD-43 | 134-136 |
| CY-18 | Porphyry | ALD-85 | 246-248 |
| CY-19 | Rhyolite | ALD-42 | 92-94 |
| CY-20 | Rhyolite | ALD-25 | 108-110 |
| CY-21 | Rhyolite | ALD-46 | 24-26 |
| CY-22 | Rhyolite | ALD-33 | 64-66 |

**Table 16-4 East Zone Drill Hole Intervals Used in Leach Cap Gold Samples**

| Bottle Roll No. | Description | Sample Location | |
|---|---|---|---|
| | | Drill Hole No. | Interval, m |
| CY-1 | Porphyry | ALD-12 | 16-18 |
| CY-2 | Porphyry | ALD-13 | 112-114 |
| CY-3 | Porphyry | ALD-24 | 100-102 |
| CY-4 | Porphyry | ALD-70 | 250-252 |

| Bottle Roll No. | Description | Sample Location | |
|---|---|---|---|
| | | Drill Hole No. | Interval, m |
| CY-5 | Rhyolite | ALD-72 | 62-64 |
| CY-6 | Rhyolite | ALD-72 | 74-76 |
| CY-7 | Rhyolite | ALD-72 | 86-88 |
| CY-8 | Rhyolite | ALD-72 | 120-122 |
| CY-9 | Rhyolite | ALD-72 | 198-200 |
| CY-10 | Andesite | ALD-13 | 16-18 |
| CY-11 | Andesite | ALD-13 | 18-20 |
| CY-12 | Andesite | ALD-13 | 20-22 |
| CY-13 | Andesite | ALD-13 | 56-58 |

### 16.4.2 Comminution Tests

Phillips determined the crushing work index (CWi), the ball mill grinding work index (BMWi) and the abrasion index (Ai) on samples of material representing each major lithology in the deposit. Specific gravities (SG) were also determined as part of the testing.

The porphyry and rhyolite determinations were made on whole core from the same samples used to prepare the column test composites. The porphyry came from drill hole ALD 46, 96 to 102 m. The rhyolite came from drill hole ALD 50, 116 to 122 m. As the metallurgical drilling did not encounter any andesite, the comminution tests were run on unmineralized andesite from drill hole ALD 35, with pieces of core from between 338 and 352.4 m.

***CWi Tests***

As the CWi test has a high degree of variability, each test was run on 20 pieces of competent core measuring between 50 and 75 mm in all dimensions. The average value for all 20 pieces is then given in kilowatt hours per metric ton (kW-h/mt). Values obtained for each lithology were reported as follows:

Porphyry - - 7.89 kW-h/mt Rhyolite - - 5.86 kW-h/mt Andesite - - 7.03 kW-h/mt

Typical CWi values for copper ores range from about 5.5 to 20.0 kW-h/t. Therefore the rhyolite would be considered very soft and the porphyry and andesite would be moderately soft.

***BMWi Tests***

The BMWi tests were performed on representative splits from each lithology, with core samples crushed and ground to less than 6 mesh (~3.35 mm). Only a single test on about 6 kg of material is required for each lithology, with values reported in kW-h/mt.

These tests were run with a closing screen size of 100 mesh (150 µm). Results for each lithology were reported as follows:

Porphyry - - 11.80 kW-h/mt  Rhyolite - - 13.26 kW-h/mt  Andesite - - 13.39 kW-h/mt

By copper industry standards, such ores would be considered to have medium grindability.

### *Ai Tests*

Broken material from the CWi tests is used for the abrasion tests. Each test requires about a 2 kg sample with material sized between 12 and 19 mm in all dimensions. As the abrasion measures the weight difference in the test coupon at the start and end of the test, it is reported without units. Results for each lithology were reported as follows:

Porphyry - - 0.0925  Rhyolite - - 0.0750  Andesite - - 0.1497

Materials with Ai values below 0.1 are considered to have low abrasiveness. Thus, the rhyolite and porphyry would cause limited wear on metal surfaces. Highly abrasive materials would have Ai values of 0.3 to 0.4. The andesite falls in the middle and would have medium abrasion characteristics.

### *SG Tests*

Without definition, the SG measurement can be somewhat misleading. Therefore, Phillips was asked to provide two measurements; whole core SG and crushed core SG. The objective for these two measurements is to show the degree of porosity inherent in the ore.

Since the whole segments were obviously porous, the samples were wax coated before testing. For these large samples, the SG values were determined by immersion. Once the whole core tests were complete, the samples were de-waxed and crushed to minus 8 mesh (~2.4 mm). Then the SG values were determined by using a pycnometer. Both sets of results are compared in Table 16-5.

**Table 16-5 Comparisons of Specific Gravities on Whole and Crushed Core**

| Lithology | Whole Core SG | Crushed Core SG | Difference |
|---|---|---|---|
| Porphyry | 2.50 | 2.855 | 0.36 |
| Rhyolite | 2.53 | 2.825 | 0.30 |
| Andesite | 2,54 | 2.834 | 0.29 |

The results show that the SG is essentially independent of lithology. However, all three lithologies exhibit a high internal void fraction, which averages around 12%. Such a structure will provide many planes of weakness and undoubtedly contributes to the low CWi values. The high void space should also benefit leaching by providing pathways for the leach solution to penetrate well into the rock fragments and contact the internal copper mineralization.

***Conclusions***

The Altar ore types should be easy to crush and require only a moderate power input for grinding.

The ores are generally soft and should cause only limited abrasion of metal surfaces.

All three lithologies have a high degree of internal porosity, which could enhance leaching by allowing the lixiviant to penetrate into the interior of the rock fragments.

Comminution parameters should be measured on a broader range of samples, including material from the oxidized leach cap.

### 16.4.3 Gold Leach Tests

***Test Procedures***

A total of 22 gold-bearing samples (~3 kg each) from the oxidized leach cap were received for bottle roll testing. All were screened to determine their crush size distribution. Of these, eight contained more than 20% plus 1.7 mm material and were stage crushed to minus 80% 1.7 mm so that all samples had the same nominal size distribution. After screening or crushing and screening, each sample was recombined and blended. Then a rotary splitter was used to remove a 1,000 g sample of material for testing.

Each 1,000 g charge was subjected to a standard 96-h agitated leach test using a bottle roll procedure. The sample was placed in the bottle roll container and diluted to 40% solids with water. The slurry was briefly agitated without reagent addition to determine the natural pH of the sample. Then sufficient lime was added to raise the pH to 11 (pH range 10.8 to 11.2). Finally sufficient sodium cyanide (NaCN) was added to give an initial NaCN concentration of 1 g/L.

Once agitation began, it was suspended briefly after 2, 6, 24, 48 and 72 h to allow the pulp to settle so that a pregnant solution sample could be removed. This was checked for pH and cyanide concentration, then assayed for gold, silver and copper using an AA technique. After sampling, solution equal to the sample volume was added, the cyanide concentration was restored to the initial concentration, and lime was added as necessary to raise the pH to the desired range. Then rolling resumed.

After 96 hours, the pulp was filtered to separate liquids and solids. The final pregnant solution volume was measured and sampled for gold, silver and copper analysis. Final pH and cyanide concentrations were determined. The leached residue was washed, dried, weighed, and assayed in triplicate to determine residual gold, silver and copper content.

Tests on ALD-72, 74-76 m, and ALD-33, 64-66 m, were conducted in triplicate to check the experimental variability of the bottle roll procedure. Tests on ALD-13, 16-18 m, and ALD-85, 246-248 m, were repeated as head assay checks. Results for all tests are summarized in Table 16-6 to Table 16-10.

***Test Results***

In general, the results are positive. Ignoring the results of the replicate tests, which would skew the values, gold recovery averaged 74.6% after 96 h, with a range of 90% to 44%. However, extraction from several samples was continuing when the tests were terminated. In addition, the average was pulled down by a few samples with low recoveries. Recovery in 11 of the 22 samples exceeded 80%. The three samples with the lowest recoveries all had low head grades (<0.20 g/mt).

Silver recovery was erratic; it ranged from 57% to <2%, with an average of about 25%. Copper recovery also varied widely from <4% to over 40%. The average value was just over 20%.

Cyanide consumption was generally low. It was below 1 kg/mt in all cases and was as low as 0.04 kg/mt. The average was 0.40 kg/mt. Lime consumption was more erratic, ranging from 1.0 to 9.5 kg/mt. The average lime consumption was 4.0 kg/mt.

**Table 16-6 Bottle Roll Results for East Zone Porphyry Samples**

| Metallurgical Results | Samples | | | |
|---|---|---|---|---|
| | CY-1 | CY-2 | CY-3 | CY-4 |
| Predicted Head, gAu/mt[1] | 0.41 | 0.20 | 1.42 | 0.6 |
| Calc'd Head, g/mt | | | | |
| Au | 0.41 | 0.20 | 1.44 | 0.39 |
| Ag | 2.1 | 2.7 | 6.5 | 2.4 |
| Cu | 148 | 238 | 70 | 204 |
| Tail Assay, g/mt[2] | | | | |
| Au | 0.04 | 0.04 | 0.48 | 0.16 |
| Ag | 1.3 | 1.7 | 6.0 | 2.3 |
| Cu | 139 | 221 | 64 | 172 |
| Au Extraction, % of total | | | | |
| in 2 h | 75.0 | 67.5 | 44.8 | 19.2 |
| in 6 h | 80.3 | 72.1 | 54.0 | 32.0 |
| in 24 h | 85.6 | 69.1 | 62.5 | 45.6 |
| in 48 h | 87.2 | 73.1 | 64.1 | 52.2 |
| in 72 h | 88.5 | 77.2 | 65.5 | 55.2 |
| in 96 h | 90.0 | 80.0 | 66.7 | 59.0 |
| Ag Extraction, % of total | 38.1 | 37.0 | 7.7 | 4.2 |
| Cu Extraction, % of total | 6.1 | 7.1 | 8.6 | 15.7 |
| NaCN Consumed, kg/mt ore | 0.14 | 0.14 | 0.36 | 0.46 |
| Lime Added, kg/mt ore | 2.1 | 3.3 | 2.5 | 4.6 |
| Natural pH | 6.6 | 6.6 | 6.5 | 5.4 |

[1] Supplied by Peregrine [2] Average of triplicate tail assays

**Table 16-7 Bottle Roll Results for East Zone Rhyolite Samples**

| Metallurgical Results | Samples | | | | |
|---|---|---|---|---|---|
| | **CY-5** | **CY-6** | **CY-7** | **CY-8** | **CY-9** |
| Predicted Head, gAu/mt[1] | 0.41 | 0.82 | 0.60 | 0.94 | 0.20 |
| Calc'd Head, g/mt | | | | | |
| Au | 0.39 | 0.73 | 0.55 | 0.73 | 0.21 |
| Ag | 3.8 | 3.7 | 4.2 | 1.4 | 2.7 |
| Cu | 274 | 390 | 386 | 246 | 224 |
| Tail Assay, g/mt[2] | | | | | |
| Au | 0.09 | 0.13 | 0.14 | 0.26 | 0.10 |
| Ag | 3.3 | 2.3 | 3.7 | 1.3 | 1.7 |
| Cu | 246 | 316 | 354 | 224 | 186 |
| Au Extraction, % of total | | | | | |
| in 2 h | 50.0 | 49.3 | 46.4 | 26.7 | 42.9 |
| in 6 h | 61.0 | 62.9 | 57.7 | 42.8 | 45.8 |
| in 24 h | 72.5 | 77.2 | 69.6 | 57.9 | 48.8 |
| in 48 h | 76.9 | 79.9 | 71.1 | 63.4 | 51.7 |
| in 72 h | 76.9 | 80.4 | 72.5 | 62.9 | 54.7 |
| in 96 h | 76.9 | 82.2 | 74.5 | 64.4 | 52.4 |
| Ag Extraction, % of total | 13.2 | 37.8 | 11.9 | 7.1 | 37.0 |
| Cu Extraction, % of total | 10.2 | 19.0 | 8.3 | 8.9 | 17,0 |
| NaCN Consumed, kg/mt ore | 0.30 | 0.38 | 0.45 | 0.22 | 0.30 |
| Lime Added, kg/mt ore | 3.0 | 2.9 | 3.5 | 2.8 | 3.3 |
| Natural pH | 5.8 | 5.7 | 5.0 | 5. | 1 5.2 |

[1] Supplied by Peregrine [2] Average of triplicate tail assays

**Table 16-8 Bottle Roll Results for East Zone Andesite Samples**

| **Metallurgical Results** | **Samples** | | | |
|---|---|---|---|---|
| | **CY-10** | **CY-11** | **CY-12** | **CY-13** |
| Predicted Head, gAu/mt[1] | 0.72 | 0.41 | 4.67 | 0.20 |
| Calc'd Head, g/mt | | | | |
| Au | 0.55 | 0.48 | 4.75 | 0.17 |
| Ag | 4.6 | 4.1 | 6.5 | 5.5 |
| Cu | 152 | 213 | 619 | 156 |
| Tail Assay, g/mt[2] | | | | |
| Au | 0.08 | 0.09 | 1.16 | 0.03 |
| Ag | 4.0 | 3.0 | 5.0 | 3.3 |
| Cu | 140 | 196 | 587 | 142 |
| Au Extraction, % of total | | | | |
| in 2 h | 40.9 | 53.1 | 60.3 | 70.6 |
| in 6 h | 57.2 | 63.0 | 68.0 | 75.2 |
| in 24 h | 82.7 | 73.3 | 74.3 | 88.7 |
| in 48 h | 85.0 | 77.9 | 75.7 | 85.1 |

| Metallurgical Results | Samples | | | |
|---|---|---|---|---|
| | CY-10 | CY-11 | CY-12 | CY-13 |
| in 72 h | 87.1 | 79.2 | 75.6 | 80.9 |
| in 96 h | 85.5 | 81.3 | 75.6 | 82.4 |
| Ag Extraction, % of total | 13.0 | 26.8 | 23.1 | 40.0 |
| Cu Extraction, % of total | 7.9 | 8.0 | 5.2 | 9.0 |
| NaCN Consumed, kg/mt ore | <0.07 | 0.70 | 0.85 | 0.15 |
| Lime Added, kg/mt ore | 1.0 | 2.4 | 8.7 | 2.2 |
| Natural pH | 5.9 | 6.0 | 6.0 | 6.3 |

[1] Supplied by Peregrine [2] Average of triplicate tail assays

**Table 16-9 Bottle Roll Results for Central Zone Porphyry Samples**

| Metallurgical Results | Samples | | | | |
|---|---|---|---|---|---|
| | CY-14 | CY-15 | CY-16 | CY-17 | CY-18 |
| Predicted Head, gAu/mt[1] | 0.80 | 0.59 | 0.39 | 0.20 | 0.98 |
| Calc'd Head, g/mt | | | | | |
| Au | 0.81 | 0.53 | 0.38 | 0.18 | 0.45 |
| Ag | 2.9 | 4.1 | 1.8 | 3.1 | 5.8 |
| Cu | 323 | 170 | 179 | 256 | 250 |
| Tail Assay, g/mt[2] | | | | | |
| Au | 0.16 | 0.10 | 0.05 | 0.10 | 0.12 |
| Ag | 2.3 | 2.7 | 1.0 | 3.0 | 5.7 |
| Cu | 276 | 102 | 106 | 192 | 197 |
| Au Extraction, % of total | | | | | |
| in 2 h | 18.5 | 34.0 | 59.2 | 33.3 | 33.3 |
| in 6 h | 38.3 | 50.4 | 71.1 | 35.6 | 45.6 |
| in 24 h | 70.4 | 76.3 | 83.5 | 46.2 | 61.9 |
| in 48 h | 76.7 | 78.3 | 84.5 | 49.0 | 69.0 |
| in 72 h | 79.4 | 80.0 | 85.4 | 51.8 | 69.8 |
| in 96 h | 80.2 | 81.1 | 86.8 | 44.4 | 73.3 |
| Ag Extraction, % of total | 20.7 | 34.1 | 44.4 | 3.2 | 1.7 |
| Cu Extraction, % of total | 14.6 | 40.0 | 40.8 | 25.0 | 21.2 |
| NaCN Consumed, kg/mt ore | 0.45 | 0.31 | 0.53 | 0.61 | 0.65 |
| Lime Added, kg/mt ore | 3.5 | 2.1 | 2.4 | 7.8 | 7.2 |
| Natural pH | 6.2 | 5.8 | 5.9 | 3.9 | 3.5 |

[1] Supplied by Peregrine [2] Average of triplicate tail assays

**Table 16-10 Bottle Roll Results for Central Zone Rhyolite Samples**

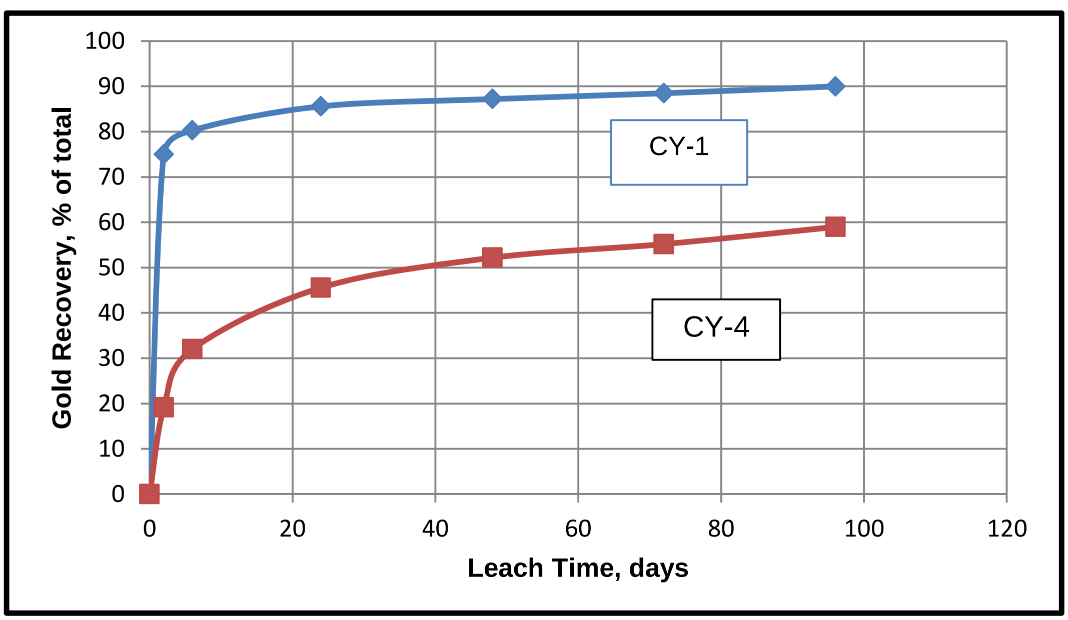
| Metallurgical Results | Samples | | | |
|---|---|---|---|---|
| | CY-19 | CY-20 | CY-21 | CY-22 |
| Predicted Head, gAu/mt[1] | 0.50 | 0.60 | 0.22 | 0.35 |
| Calc'd Head, g/mt | | | | |
| Au | 0.33 | 0.61 | 0.18 | 0.33 |
| Ag | 3.0 | 11.4 | 3.5 | 1.5 |
| Cu | 187 | 342 | 126 | 193 |

| Metallurgical Results | Samples | | | |
|---|---|---|---|---|
| | CY-19 | CY-20 | CY-21 | CY-22 |
| Tail Assay, g/mt[2] | | | | |
| Au | 0.06 | 0.14 | 0.02 | 0.14 |
| Ag | 1.3 | 11.0 | 1.7 | 1.3 |
| Cu | 139 | 301 | 120 | 183 |
| Au Extraction, % of total | | | | |
| in 2 h | 63.6 | 59.0 | 75.0 | 22.7 |
| in 6 h | 68.0 | 65.6 | 80.0 | 28.8 |
| in 24 h | 76.9 | 74.6 | 85.0 | 44.3 |
| in 48 h | 77.1 | 76.7 | 90.0 | 51.6 |
| in 72 h | 76.9 | 76.2 | 86.7 | 59.3 |
| in 96 h | 81.8 | 77.0 | 88.9 | 57.6 |
| Ag Extraction, % of total | 56.7 | 3.5 | 51.4 | 13.3 |
| Cu Extraction, % of total | 25.7 | 12.0 | 4.8 | 5.2 |
| NaCN Consumed, kg/mt ore | 0.35 | 0.96 | 0.04 | 0.34 |
| Lime Added, kg/mt ore | 2.2 | 9.5 | 2.9 | 7.3 |
| Natural pH | 5.5 | 4.1 | 4.9 | 4.0 |

[1] Supplied by Peregrine [2] Average of triplicate tail assays

In general, the gold leached rapidly. In about a quarter of the samples, gold extraction exceeded 60% in the first two hours. An example is CY-1, which reached 75% extraction in just 2 h. For most samples, at least 80% of the final extraction occurred within 24 h. However, a few samples proved to be more refractory and leached more slowly. A good example is CY-4, which is the deepest of all the samples. Leach curves for CY-1 and CY-4 are compared in Figure 16-1.

**Figure 16-1 Leach Curves for Fast and Slow Leaching Samples**



The average values of the key parameters determined for all zones and lithologies are shown in Table 16-11. A comparison of these values permits an assessment of the impact of these two variables on leach performance.

**Table 16-11 Comparison of Key Parameters for the Five Sample Types**

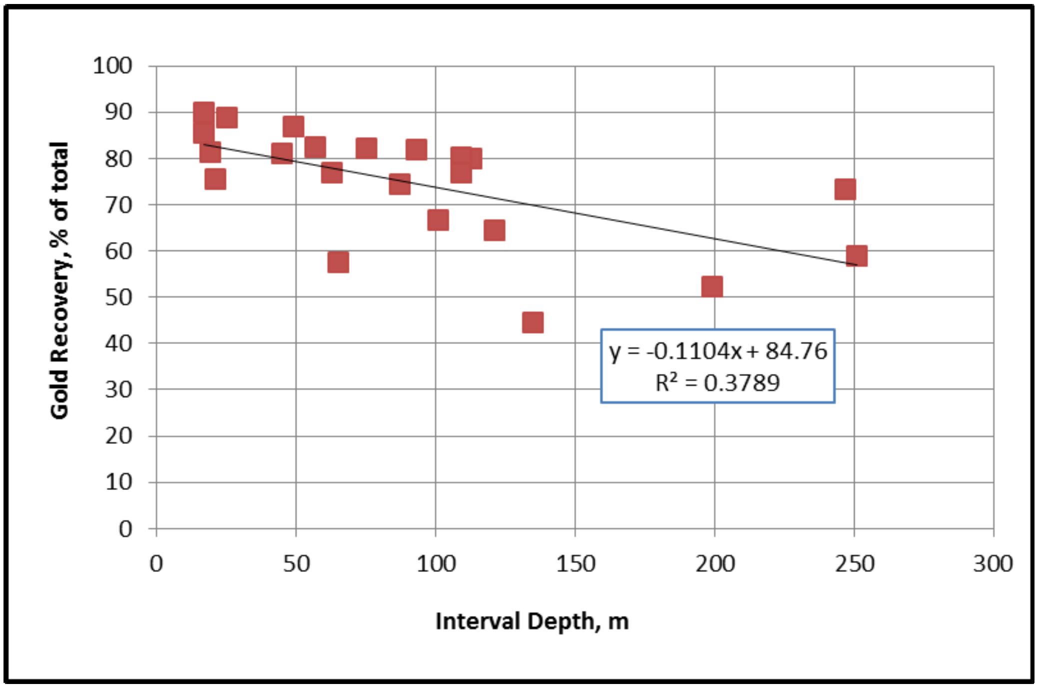
| Sample ID. | Gold Extraction | | Copper Extraction | | NaCN Consumed kg/mt | Lime Used, kg/mt |
|---|---|---|---|---|---|---|
| | % of Total | g/mt | % of Total | g/mt | | |
| East Porphyry | 73.9 | 0.43 | 9.4 | 16 | 0.28 | 3.1 |
| East Rhyolite | 70.1 | 0.38 | 12.7 | 39 | 0.33 | 3.1 |
| East Andesite | 81.2 | 1.15 | 7.5 | 19 | 0.44 | 3.5 |
| Cent'l Porphyry | 73.2 | 0.36 | 28.3 | 61 | 0.51 | 4.6 |
| Cent'l Rhyolite | 76.3 | 0.27 | 11.9 | 26 | 0.42 | 5.5 |

Given the scatter in the data, neither the zone nor the lithology appears to have a significant impact on gold recovery. The average recoveries for each zone are nearly identical. The East Zone andesite material appears to give better results. However, this group contains only very shallow samples, which apparently leach better than deeper samples (see below).

The five groups all have about the same average copper head grade. However, the East Zone does appear to have a lower level of copper extraction. It also appears to exhibit lower reagent consumption, although these differences should be confirmed through further testing. The lower cyanide consumption is likely related to the lower amount of copper extraction. The lower lime usage may be associated with the higher natural pH levels for the East Zone samples. It is not clear if the degree of copper extraction is high enough to cause problems in gold recovery.

One variable that does appear to affect gold extraction is the sample depth. As shown in Figure 16-2, extraction tends to drop as the depth of the sample interval increases. This inverse trend seems reasonable, as deeper material is likely to be less well oxidized and therefore more refractory. However, this should be confirmed in further testing, as the correlation is rather weak (low $R^2$ value). As shown in Figure 16.1, CY-4 is the deepest sample tested and exhibits the slowest leach rate. For this sample there was still significant extraction occurring when the test was terminated after 96 h.

**Figure 16-2 Gold Extraction by Depth**




### *Analytical Accountability*

As shown in Tables 16.6 through 16.10, the back calculated and predicted head grades agreed within 10% for 17 of the 22 samples. New head assays were run in triplicate on the other five samples. In three of these the new assays agreed closely with the back calculated assays. The other two tests were rerun and results matched the initial tests. Thus, there do not appear to be any serious analytical issues associated with the bottle roll tests.

### *Experimental Reproducibility*

Two of the bottle roll tests were run in triplicate to check the variability of the bottle roll procedure. Key parameters are compared in Table 16-12. As can be seen, metal extraction is highly reproducible, varying by 3%, or less. The back calculated gold head assays agree within 0.03 g/mt and the tail assays agree within 0.02 g/mt. However, reagent consumption appears more variable. Even so, a reasonable level of confidence can be placed in the test results.

**Table 16-12 Comparison of Key Parameters for the**

| Test No. | Gold Extraction, % of total | Assay, g Au/mt | | Copper Extraction, % of total | NaCN Usage, kg/mt | Lime Usage, kg/mt |
|---|---|---|---|---|---|---|
| | | Head | Tail | | | |
| CY-6 Orig. | 82.2 | 0.73 | 0.13 | 19.0 | 0.38 | 2.9 |
| CY-6 2nd | 80.3 | 0.76 | 0.15 | 19.3 | 0.36 | 4.6 |
| CY-6 3rd | 80.3 | 0.76 | 0.15 | 18.6 | 0.30 | 4.1 |

| Test No. | Gold Extraction, % of total | Assay, g Au/mt | | Copper Extraction, % of total | NaCN Usage, kg/mt | Lime Usage, kg/mt |
|---|---|---|---|---|---|---|
| | | Head | Tail | | | |
| CY-22 Orig. | 57.6 | 0.33 | 0.14 | 5.2 | 0.34 | 7.3 |
| CY-22 2nd | 55.9 | 0.34 | 0.15 | 5.3 | 0.42 | 8.5 |
| CY-22 3rd | 55.9 | 0.34 | 0.15 | 5.1 | 0.34 | 8.1 |

***Conclusions***

The Altar oxidized leach cap appears to be amenable to cyanide leaching for gold recovery. The median gold recovery exceeds 80% and cyanide consumption is low. Additional column leach tests should be undertaken to develop the project further.

Gold recovery is largely independent of either lithology or the ore zone, but tends to decrease with increasing depth.

The gold leaches rapidly, with most of the extraction occurring within 24 h.

Silver recovery is erratic, but low, and is not likely to contribute to the project significantly.

Copper recovery is highly variable. If high enough, it could complicate gold recovery.

The bottle roll procedure is reproducible and testing was not subject to serious analytical issues.

### 16.4.4 Water Solubility Tests

During the initial flotation tests, DML was unable to get closure on the mass balance because the combined weight of the concentrates and tailings was less than the initial weight of the flotation feed. Investigation showed that the weight difference was due to the presence of water soluble material in the head sample. Therefore, the water soluble content of the three MLI composites was also checked. The chalcopyrite-bornite composite was found to have about 2% water soluble material, while the other two had less than 1% water soluble content.

***Test Procedure***

Further testing seemed to be warranted and all 15 composites being used in the column leach tests have been checked for water soluble content. The procedure was straight forward and started by drying the pulverized sample to constant weight at low temperature. Then 50 g (exact weight recorded) of solids were slurried with sufficient deionized water to give a slurry with 10% solids. This was heated to 35º C and agitated gently for 30 minutes. After 30 minutes, the slurry was hot filtered to avoid possible crystallization of solids. The solid residue was then dried to constant weight and reweighed. Any weight loss was recorded as water soluble content. The filtrate was retained for analysis to determine the content of any water soluble material.

***Test Results***

The water solubilities are shown in Table 16.13. The soluble constituents in each sample are shown in Table 16.14. The results show that all Altar material is likely to have a water soluble component. The average water soluble content was 1.90%, with the refractory porphyry material running nearly four percent. Based on the high calcium and sulphate levels, the water soluble component in the refractory porphyry is anhydrite. However, the other composites contain much lower calcium levels and on a relative basis a much higher mix of cations. These include considerable potassium and sodium, with lesser amounts of magnesium, iron and even strontium. Most of these metals are apparently present as sulphates, but also as chlorides in some composites.

These water soluble constituents will build up in either the mill circuit or a leach circuit due to the processing of new ore and evaporation. The high calcium and sulphate levels are likely to cause scaling problems on wetted surfaces. The chloride may also build up to the point where it can cause corrosion problems.

***Conclusions***

All Altar material apparently contains some water soluble material.

In a high tonnage operation a significant amount of water soluble material will build up in any recirculating process stream. Water treatment may become necessary to avoid serious scaling or corrosion problems.

**Table 16-13 Water Soluble Content of Column Composites**

<table>
<tr><th>Sample Composite</th><th>MLI Test No.</th><th>Water Soluble Content, %</th><th>Sample Composite</th><th>MLI Test No.</th><th>Water Soluble Content, %</th></tr>
<tr><td>Crush Size</td><td>AC 1-5, 20</td><td>0.63</td><td>85% Solubility</td><td>AC 11</td><td>1.06</td></tr>
<tr><td>Low Grade</td><td>AC 8</td><td>1.03</td><td>70% Solubility</td><td>AC 12</td><td>1.34</td></tr>
<tr><td>Mid-Grade</td><td>AC 9</td><td>0.60</td><td>50% Solubility</td><td>AC 13</td><td>0.88</td></tr>
<tr><td>High Grade</td><td>AC 10</td><td>1.38</td><td>Rhyolite</td><td>AC 7</td><td>2.22</td></tr>
<tr><td>Porphyry</td><td>AC 6</td><td>1.41</td><td>Rhyolite Refractory 1</td><td>AC 17</td><td>1.29</td></tr>
<tr><td>Porphyry Refractory 1</td><td>AC 14</td><td>3.94</td><td>Rhyolite Refractory 2</td><td>AC 18</td><td>2.76</td></tr>
<tr><td>Porphyry Refractory 2</td><td>AC 15</td><td>3.80</td><td>Rhyolite Refractory 3</td><td>AC 19</td><td>2.35</td></tr>
<tr><td rowspan="2">Porphyry Refractory 3</td><td rowspan="2">AC 16</td><td rowspan="2">3.81</td><td colspan="2">Average</td><td>1.90</td></tr>
<tr><td colspan="2">Median</td><td>1.38</td></tr>
</table>

**Table 16-14 Water Soluble Constituents of Column Composites**

| Composite | Constituent, ppm | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | Ca | Fe | K | Mg | Na | Sr | $SO_4$ | Cl |
| Crush Size | 15 | 0.2 | 17 | 1.4 | 4.8 | 0.3 | 82 | 2.0 |
| Low Grade | 11.5 | 0.1 | 1 | 1.4 | 0.6 | 0.3 | 69 | 4.2 |
| Mid-Grade | 15.3 | * | 17 | 1.2 | 7.6 | 0.3 | 81 | 2.7 |
| High Grade | 9.2 | * | 14 | 1.1 | 9.0 | 0.3 | 60 | 5.3 |
| Porphyry | 5.3 | * | 10 | 0.6 | 10.6 | 0.1 | 23 | 6.6 |
| Porphyry Refractory 1 | 602 | 5.7 | 25 | 7.2 | 7.5 | 4.8 | 1,762 | * |
| Porphyry Refractory 2 | 619 | 2.9 | 24 | 5.5 | 9.0 | 6.4 | 1,757 | 2.5 |
| Porphyry Refractory 3 | 549 | 5.4 | 29 | 5.5 | 10.7 | 4.5 | 1,606 | 1.8 |
| 85% Solubility | 13.5 | * | 17 | 1.0 | 5.5 | 0.2 | 70 | 3.8 |
| 70% Solubility | 1.0 | * | 3 | 0.3 | 8.3 | * | 14 | 5.3 |
| 50% Solubility | 1.3 | * | 6 | 0.2 | 9.1 | * | 10 | 7.0 |
| Rhyolite | 2.4 | * | 12 | 0.3 | 12.0 | 0.1 | 34 | 6.1 |
| Rhyolite Refractory 1 | 7.5 | * | 14 | 1.2 | 8.5 | 0.1 | 55 | 6.3 |
| Rhyolite Refractory 2 | 0.3 | * | 13 | 0.1 | 8.2 | * | 25 | 11.6 |
| Rhyolite Refractory 3 | 0.5 | * | 18 | 0.1 | 7.7 | * | 31 | 8.3 |

* Denotes value is below detection limit.

### 16.4.5 Column Leach Program

The column leach program is the follow-on to the earlier bottle roll testing described in 16.3. The main focus is to determine how the various resource parameters affect leach performance of the sulphide ores. Parameters include lithology (porphyry and rhyolite), head grade, and the degree of solubility. The latter refers to the solubility ratio, often expressed as a percentage. The ratio is the sum of acid soluble copper plus cyanide soluble copper divided by total copper content, with all three values determined by the sequential copper assay technique. The solubility ratio serves as a rough guide to the percentage of the total copper that is readily leachable. Material having a low solubility would be considered refractory and not be expected to leach as rapidly or completely as material with a high ratio. The current program includes samples with solubility ratios of 85%, 70% and 50%. In addition, there are three more refractory porphyry samples and three more refractory rhyolite samples. One test is being run in duplicate to check experimental reproducibility.

All samples for the column program came from the metallurgical drilling program. As mentioned above, the drilling did not encounter any mineralized andesite. As a result, testing of this lithology will have to await the next phase of the metallurgical program.

Also, crush size is the only process parameter currently being tested. Other process parameters, such as column height and irrigation (flow) rate, will also be evaluated at a later date. In the current program, the two main parameters being evaluated are the rate of copper extraction and the acid consumption.

### *Test Procedures*

The material to be used for preparing each composite was received as random lengths of whole HQ core. The core was first crushed to a $P_{100}$ of 25 mm, then screened into three fractions: 25 by 12.5 mm, 12.5 by 6 mm, and minus 6 mm. Each was weighed. Column charges (approximately 20 kg each) were then prepared by withdrawing the appropriate amount of material from each fraction. Depending on the desired top size in the test, the coarsest portion was normally stage crushed to the desired $P_{80}$ particle size. . All tests are being run with a 12.5 mm top size, except for the crush size tests. For these the top sizes are 25, 19, 12.5 and 6.3 mm.

Following crushing and blending, a sample was the split out for a full head assay. Then the sample was rescreened to determine the screen size distribution down to 75 µm, with a sample of each size fraction split out for assaying. Finally, the fractions were reblended and charged into a column 150 mm in diameter by 2 m high. Once the column was loaded, it was cured by adding synthetic raffinate containing enough acid to provide a 5 kg/mt cure. After the 24 h cure cycle, leaching began at an irrigation rate of 0.14 L/min/$m^2$. The lixiviant contained 2.5 g/L free acid, as measured by titration, and 1.5 g/L ferric iron.

Daily PLS volumes are being measured and sampled for pH, free acid, Eh, Cu and Fe (total and ferrous). After sampling, PLS from each test is stored for solvent extraction (SX) processing. SX processing of PLS solution from each column is conducted twice each week, or as needed. Once a sufficient volume of the resulting raffinate has been generated for each test, raffinate with any necessary make-up reagents is recycled to the appropriate test. This type of testing is referred to closed cycle. Through the repeated recycling of solution the soluble constituents in the ore build up to their steady state concentrations, replicating actual leach conditions.

### *Test Results*

Testing has been underway for about four months, with a planned duration of six months. Therefore, only interim results are available. Recoveries are being estimated from the amounts of copper in solution and the assayed head grades. The more accurate back calculated head grade (copper in solution plus copper in the residue) cannot be determined until the test is terminated and the residue can be sampled.

Cumulative results through the first 130 days are presented in Table 16.15. The table includes a description of each column, the estimated head grade, the amount (weight) of copper extracted, the total and soluble percent copper extractions, and the total and net acid consumptions. The total copper extraction is based on the total copper head assay, while the soluble recovery is based on just the acid and cyanide soluble contents from the sequential assays. The total acid consumption includes all acid consumed by the reactive gangue and any side reactions which involve acid. The net acid consumption is simply the total acid consumption less a credit for the acid that is recovered when copper is recovered by solvent extraction-electrowinning (SX-EW). The credit is 1.54 kg of acid per kg of copper extracted.

Detailed interpretation of the results is difficult, as the tests are expected to run for another two months. There is clearly a wide difference in the leach performance of the different composites. Recovery of total copper ranges from nearly 70% down to less than 20%, with an average of 48%. In part, the low recovery reflects the very slow leaching of the refractory samples. If the refractory results are deleted from the data base, the average recovery rises to 58% of the total copper. By comparison, over two thirds of the soluble copper has been extracted.

Even the best results are lower than the recoveries observed in the earlier bottle roll tests. This is no doubt due in large part to the difference in particle size. Even the finest column charges are quite coarse compared to the bottle roll samples. The latter had a top size below 2 mm and much of the material was finer than 150 µm. This is typical of flotation plant feed and would be expected to leach quite rapidly, as most of the copper sulphides would be either fully or at least partially liberated.

The percentage recovery can also be misleading, unless viewed as part of a broader evaluation of leach performance. For example, the low grade composite has the highest percentage of total copper recovery. However, this is merely a consequence of the low head grade. More than twice as much copper has been extracted from the crush size composites, but the percentage of copper being recovered is lower for these because the head grades are so much higher.

One trend is clear, even though the column tests are still underway. Initially there was little difference in recovery between the coarse and fine crush size composites. Now the recoveries are starting to diverge. The coarsest column charge (25 mm top size) has achieved a total copper recovery of 49%. However, the 6.5 and 12.5 mm charges have reached about 61% extraction, a difference of almost 25%.

Another observation is that the presence of copper arsenides complicates interpretation of the test results. Enargite and to a lesser extent tetrahedrite-tennantite have a high cyanide solubility. Thus, they are classed as soluble copper by the sequential assay technique. However, both leach very slowly in acidic ferric sulphate.

As can be seen in Table 16-15, the refractory rhyolites are leaching almost twice as fast as the refractory porphyry samples. Yet the two sets of samples have almost identical assay head grades and soluble copper ratios. The difference lies in the arsenic contents. The refractory porphyry samples have nearly 800 ppm As, indicating that these have considerable enargite present. This will show up as soluble copper, but will leach poorly. In contrast, the refractory rhyolite samples contain only about

100 ppm As and would have little enargite, so much of the soluble copper would be leachable.

In addition to copper recovery, acid consumption is a parameter of significance. Fortunately, the acid consumption levels are encouraging. Total acid consumption currently averages 14 kg/mt, while net acid consumption averages less than 7 kg/mt. Only five columns have values above 10 kg/mt. Initially much of the acid consumption was caused by reaction with the gangue mineralization. Most of the reactive gangue is now gone. Current acid consumption is due mainly to bacterial activity that reoxidizes ferrous iron to the ferric form. The reaction consumes acid as follows:

$$4Fe^{+2} + O_2(\text{liquid}) + 4H^+ \rightarrow 4Fe^{+3} + 2H_2O$$

In the reaction, the balancing anion is sulphate, which is not shown for simplicity.

***Conclusions and Recommendations***

- Interpretation of the column leach results is difficult, given the fact that the leach cycle has another two months to go. Recommendations for further development of the leach program should await completion of the tests.
- The copper extraction rate varies widely with the characteristics of the ore.
- Even the best extraction rates are slower than where observed in the earlier bottle roll tests. One contributing factor is the much coarser rock size in the column tests.
- The presence of arsenic complicates interpretation of the leach results.
- Acid consumption is low.

**Table 16-15 Column Test Results after 130 Days**

| Test Description | Cu Head Assay, %Cu | | Cu Extraction, g or % | | | Acid Consumed, kg/mt | |
|---|---|---|---|---|---|---|---|
| | Soluble | Total | Weight | Soluble | Total | Total | Net |
| Crush Size Composite-25 mm | 1.22 | 1.40 | 138 | 56.7 | 49.3 | 14.6 | 4.0 |
| Crush Size Composite-19 mm | 1.24 | 1.42 | 149 | 60.3 | 52.8 | 17.3 | 5.8 |
| Crush Size Composite-12.5 mm | 1.20 | 1.39 | 159 | 66.3 | 57.3 | 17.2 | 4.9 |
| Crush Size Composite-12.5 mm[1] | 1.20 | 1.39 | 170 | 71.1 | 61.5 | 18.7 | 5.6 |
| Crush Size Composite – 6.3 mm | 1.23 | 1.43 | 162 | 66.4 | 57.3 | 18.9 | 5.8 |
| Low Grade Composite | 0.40 | 0.48 | 64 | 80.2 | 67.1 | 13.3 | 8.4 |
| Mid-Grade Composite | 0.94 | 1.07 | 123 | 66.3 | 58.6 | 17.2 | 7.5 |
| High Grade Composite | 0.96 | 1.13 | 127 | 66.1 | 56.3 | 16.3 | 6.5 |
| Porphyry Composite | 0.59 | 0.76 | 90 | 76.1 | 59.0 | 19.0 | 12.1 |
| Porphyry Refractory 1 | 0.32 | 0.71 | 27 | 41.7 | 19.1 | 7.6 | 5.5 |
| Porphyry Refractory 2 | 0.37 | 0.65 | 24 | 32.3 | 18.2 | 4.8 | 2.9 |
| Porphyry Refractory 3 | 0.33 | 0.65 | 26 | 37.9 | 19.6 | 11.9 | 10.1 |

| Test Description | Cu Head Assay, %Cu | | Cu Extraction, g or % | | | Acid Consumed, kg/mt | |
|---|---|---|---|---|---|---|---|
| | Soluble | Total | Weight | Soluble | Total | Total | Net |
| Rhyolite Composite | 0.50 | 1.21 | 125 | >100 | 64.4 | 15.5 | 5.9 |
| Rhyolite Refractory 1 | 0.28 | 0.60 | 39 | 67.1 | 31.9 | 13.3 | 10.4 |
| Rhyolite Refractory 2 | 0.24 | 0.50 | 30 | 61.9 | 30.1 | 7.0 | 4.6 |
| Rhyolite Refractory 3 | 0.36 | 0.64 | 55 | 75.6 | 42.5 | 5.3 | 1.1 |
| 85% Solubility Composite | 0.57 | 0.69 | 68 | 59.2 | 49.0 | 9.2 | 4.0 |
| 70% Solubility Composite | 0.53 | 0.71 | 92 | 86.7 | 64.7 | 17.5 | 10.4 |
| 50% Solubility Composite | 0.27 | 0.42 | 40 | 74.1 | 47.7 | 18.4 | 15.3 |
| Average | 0.66 | 0.93 | 94 | 68.6 | **48.4** | 14.0 | 6.7 |

[1] Duplicate column

# 17.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

## 17.1 Introduction

The Altar Project is now at an advanced stage of exploration. This section describes the updated mineral resource estimate prepared with the latest drilling results and geological interpretation. The primary economic item is copper but gold values also contribute to the total value of the deposit. Molybdenum is a minor component and it is uncertain if there would be existing tonnage of sufficient grade to justify the inclusion of a Mo circuit at present day prices.

There are no mineral reserves existing for the Altar Project.

## 17.2 Exploratory Data Analysis

The Altar Project drill hole database consists of 140 core holes drilled on the deposit since 2003. Core recovery has been excellent averaging 97.3% (median=100%). A summary of the drilling and sampling is presented in Table 17-1.

**Table 17-1 Altar core drilling summary**

| Year | Company | Holes Drilled | Total metres | Comments |
|---|---|---|---|---|
| 2003 | Rio Tinto | 7 | 2,841.13 | |
| 2006 | Peregrine | 8 | 3,302.20 | |
| 2007 | Peregrine | 25 | 10,408.15 | |
| 2008 | Peregrine | 24 | 12,760.60 | + 1 hole extended |
| 2010 | Peregrine | 76 | 26,328.55 | + 2 holes extended |
| **Total** | | **140** | **55,640.63** | |

The database includes interval tables for lithology, alteration, degree of oxidation, mineralogy, and vein and fracture intensity. Lithologic codes used for Altar are listed in Table 17-2. The terminology has been slightly modified since 2008 to reflect the better understanding of the respective lithologic units.

**Table 17-2 Altar lithologic codes**

| Code | Symbol | Lithology |
|---|---|---|
| 1 | Tap | Tertiary Andesite Porphyry |
| 3 | Tpa | Tertiary Pachon Andesite |
| 4 | Tpr | Tertiary Pachon Rhyolite |
| 8 | Tdp | Tertiary Diorite Porphyry |
| 10 | Ovbdn | Overburden |

Box plots were prepared comparing grade distribution to lithology in both unweathered rock and in the leached cap (Figure 17-1 and Figure 17-2). Within the sulphide zone the Tdp unit encompasses the bulk of the higher grades for Cu, Au and Mo, however, all other units contain a significant distribution of higher grades as well. In the leached cap the Cu is depleted but Au and Mo shows some enrichment, although not in potentially economic concentrations that would support a stand-alone mining operation.

**Figure 17-1 Cu distribution by lithology**

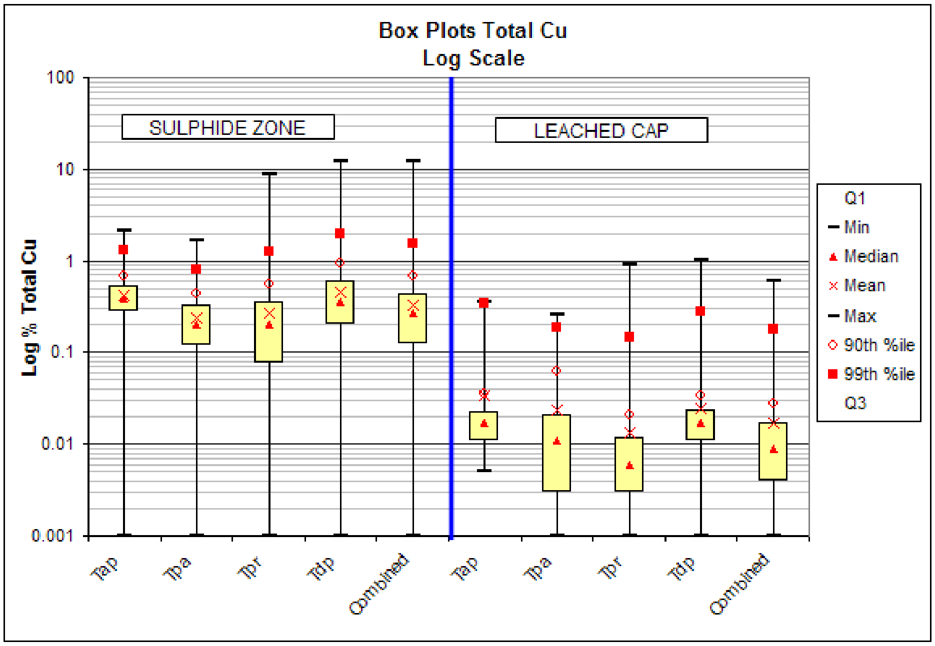



**Figure 17-2 Au distribution by lithology**

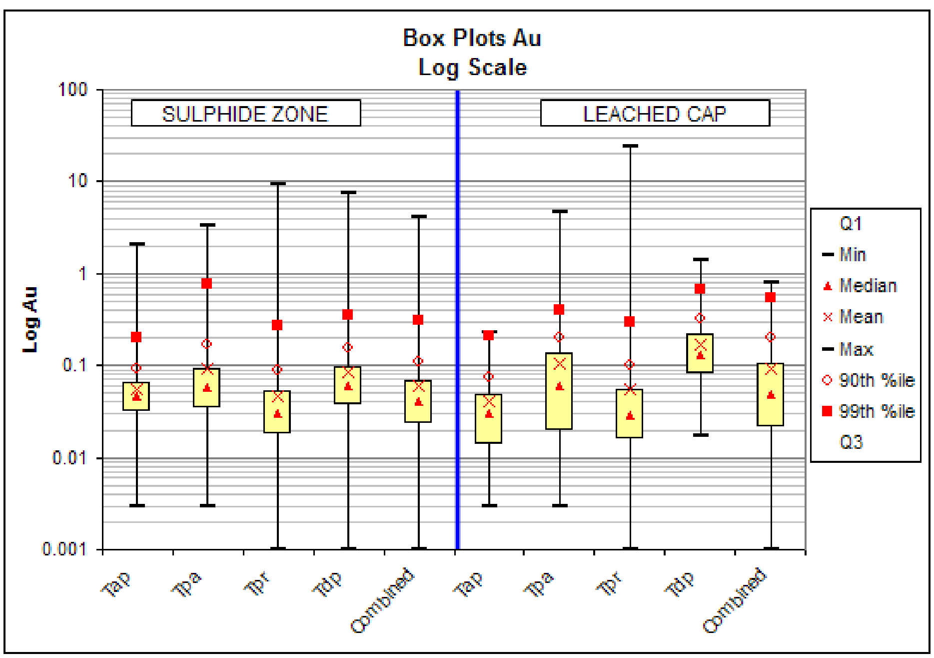
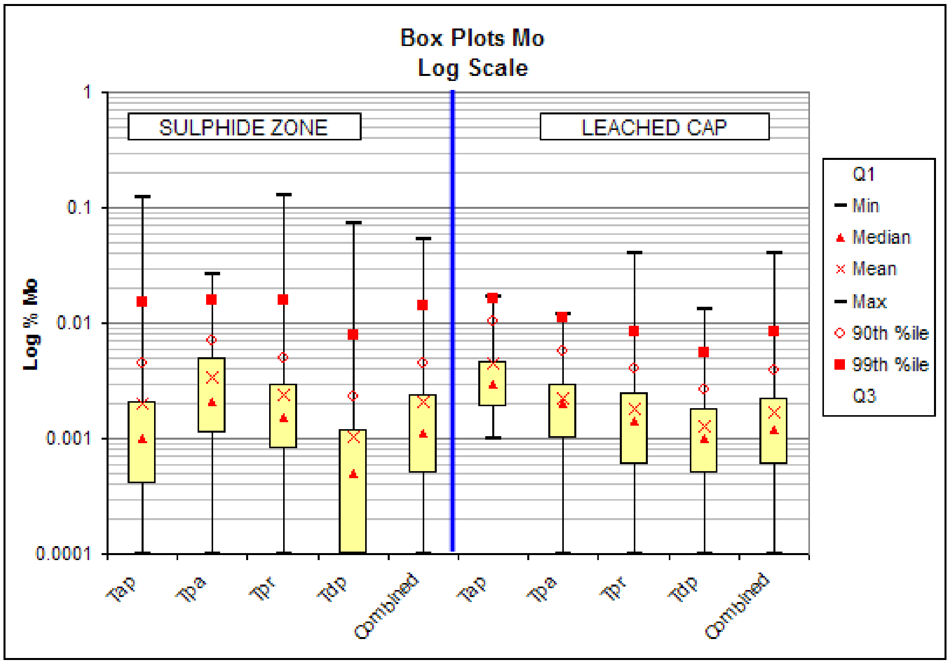


**Figure 17-3 Mo distribution by lithology**




Lithologic contacts were examined on cross sections and compared to grade distribution. No significant differences in grade occurred across contacts with the exception of the leached zone where Cu was depleted and Au enriched locally.

Grades for Cu, Au and Mo were compared with alteration type and intensity both statistically and on cross sections. Within areas of potentially economic grades no significant correlations were found.

Sample data from the sulphide (unweathered) zone extracted for analysis and compositing. Statistics for Cu, Au, Ag, Mo and As are shown in Table 17-3. Histograms for Cu and Au exhibit skewed populations approaching log normality with no apparent bimodal character (Figure 17-4 to Figure 17-6).

**Table 17-3 Descriptive statistics of sample data**

| | Sulphide Zone | | | | |
|---|---|---|---|---|---|
| | Cu | Au | Ag | Mo | As |
| **Number of samples** | 19954 | 19954 | 19954 | 19953 | 19954 |
| **Minimum value** | 0.00 | 0.00 | 0.00 | 0.000 | 0.00 |
| **Maximum value** | 12.38 | 4.13 | 88.80 | 0.053 | 1.00 |
| | | | | | |
| **25.0 Percentile** | 0.13 | 0.02 | 0.30 | 0.001 | 0.00 |
| **50.0 Percentile (median)** | 0.27 | 0.04 | 0.60 | 0.001 | 0.01 |
| **75.0 Percentile** | 0.45 | 0.07 | 1.10 | 0.002 | 0.04 |
| **90.0 Percentile** | 0.68 | 0.11 | 1.90 | 0.005 | 0.09 |
| **95.0 Percentile** | 0.87 | 0.16 | 2.60 | 0.006 | 0.13 |
| **98.0 Percentile** | 1.23 | 0.23 | 3.70 | 0.010 | 0.19 |
| **99.0 Percentile** | 1.56 | 0.32 | 5.00 | 0.014 | 0.25 |
| | | | | | |
| **Mean** | 0.34 | 0.06 | 0.90 | 0.002 | 0.03 |
| **Variance** | 0.12 | 0.02 | 3.56 | 0.000 | 0.00 |
| **Standard Deviation** | 0.34 | 0.12 | 1.89 | 0.004 | 0.07 |
| **Coefficient of variation** | 1.02 | 2.08 | 2.10 | 1.761 | 2.08 |

**Figure 17-4 Frequency distribution of Cu in sulphide zone**

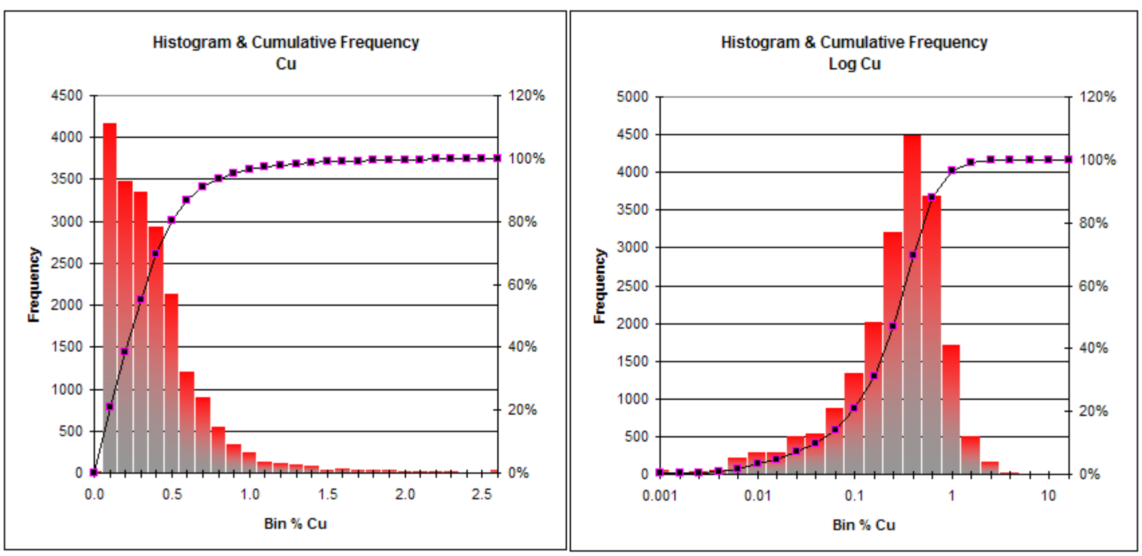


**Figure 17-5 Frequency distribution of Au in sulphide zone**

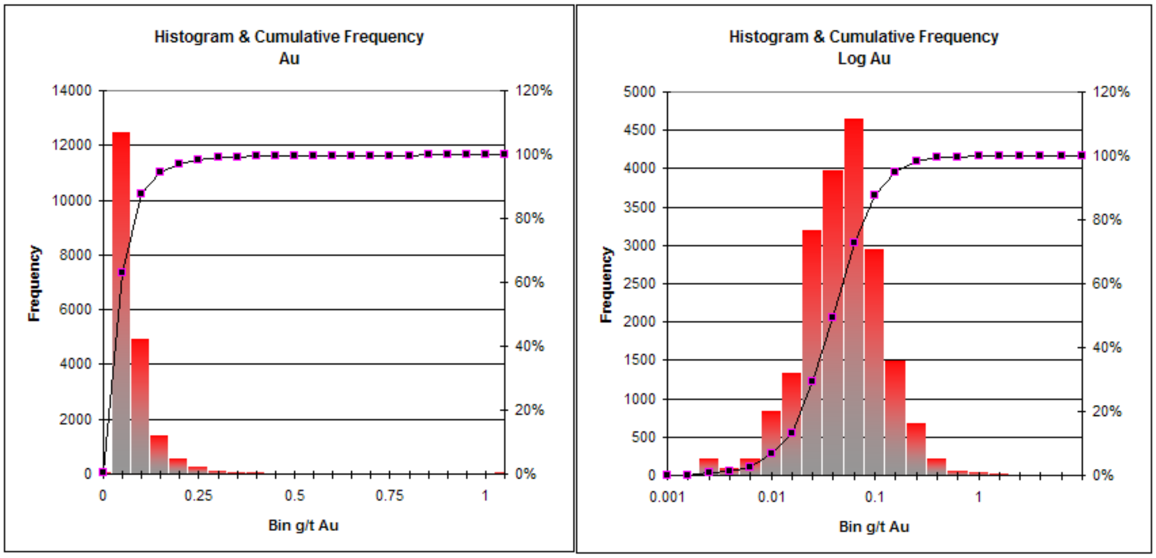
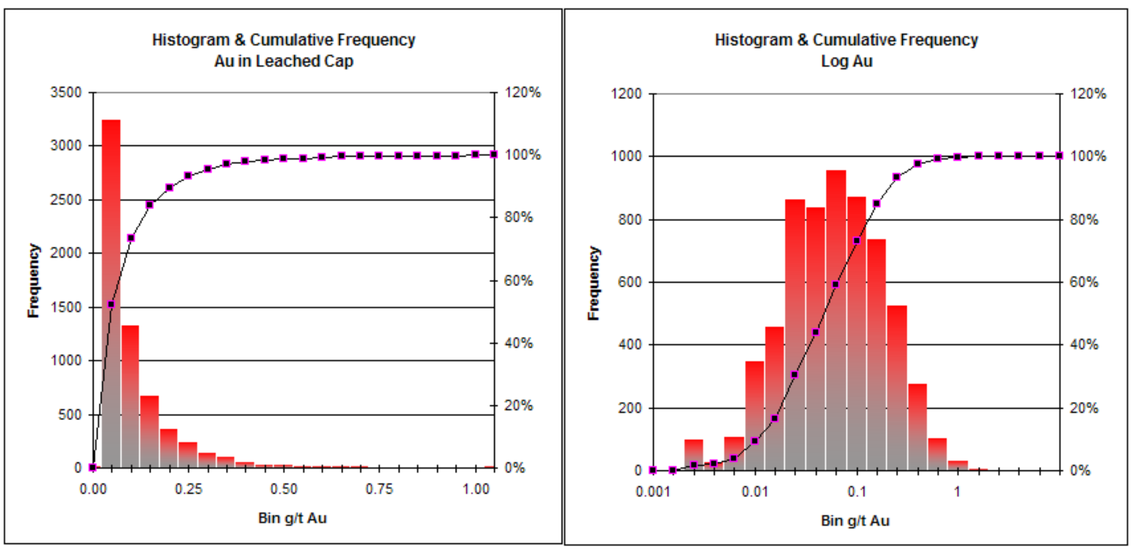



**Figure 17-6 Frequency distribution of Au in leached cap**

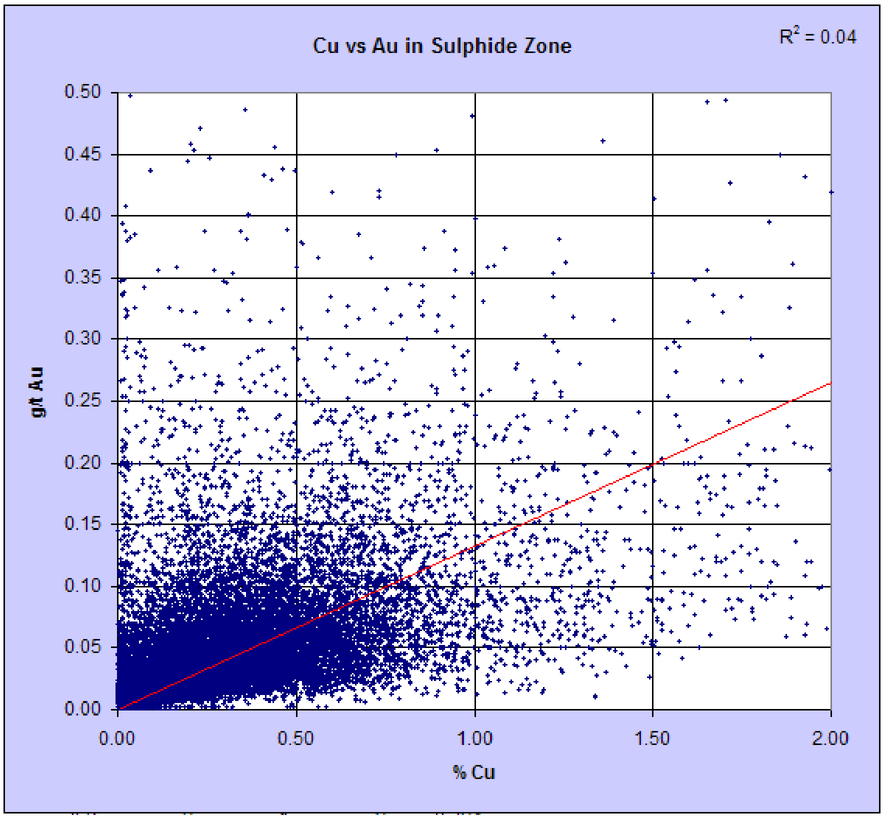



Copper shows weak to moderate positive correlations with arsenic and silver, a weak correlation with gold and no significant correlation with molybdenum (Table 17-4). The Cu/Au correlation is further illustrated in the Scatterplot (Figure 17-7) with an $R^2$ value of only 0.04. Gold shows an extremely weak negative correlation with molybdenum.

**Table 17-4 Correlation matrix**

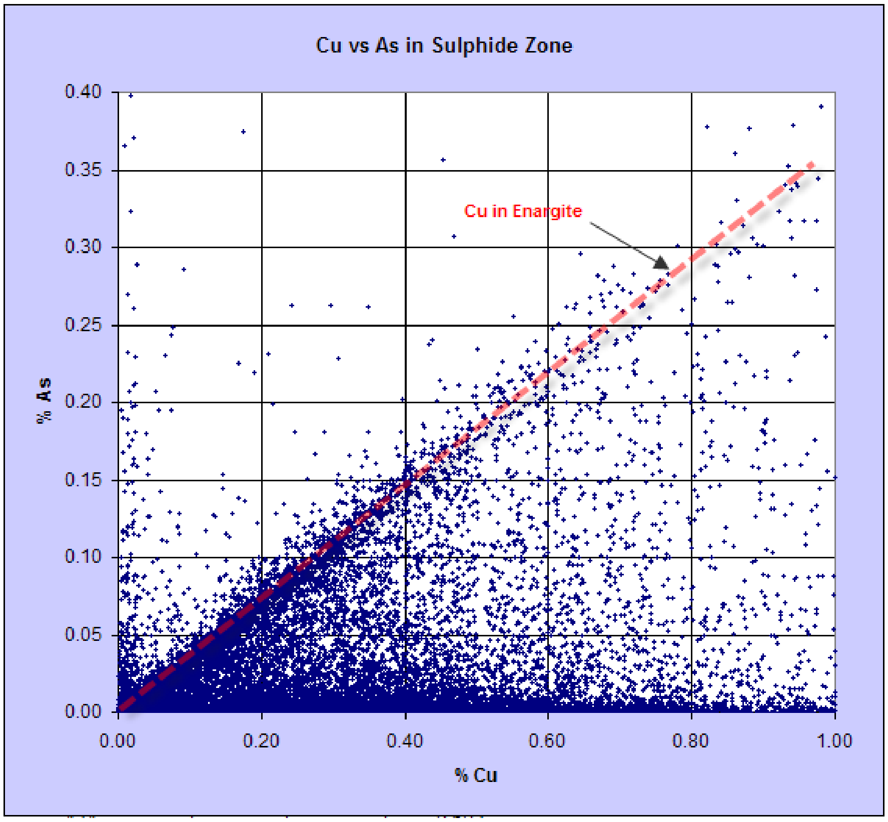
| | *Cu* | *Au* | *Ag* | *Mo* | *As* |
|---|---|---|---|---|---|
| Cu | 1.00 | | | | |
| Au | 0.25 | 1.00 | | | |
| Ag | 0.44 | 0.18 | 1.00 | | |
| Mo | 0.02 | -0.01 | 0.03 | 1.00 | |
| As | 0.41 | 0.19 | 0.42 | 0.04 | 1.00 |

**Figure 17-7 Scatterplot of Cu vs Au in sulphide zone**




The presence of enargite is clearly evident in the scatterplot between Cu and As values shown in Figure 17-8. The cloud below this line represents Cu in a combination of other sulphides, mainly chalcopyrite. Recent petrographic studies have also identified tennantite-tetrahedrite in metallurgical composites (Economic Geology Consulting, 2010). The presence of As not associated with Cu is indicated by the scattered points above the enargite line (possibly in minerals such as arsenopyrite or realgar/orpiment).

**Figure 17-8 Scatterplot of Cu vs As in sulphide zone**



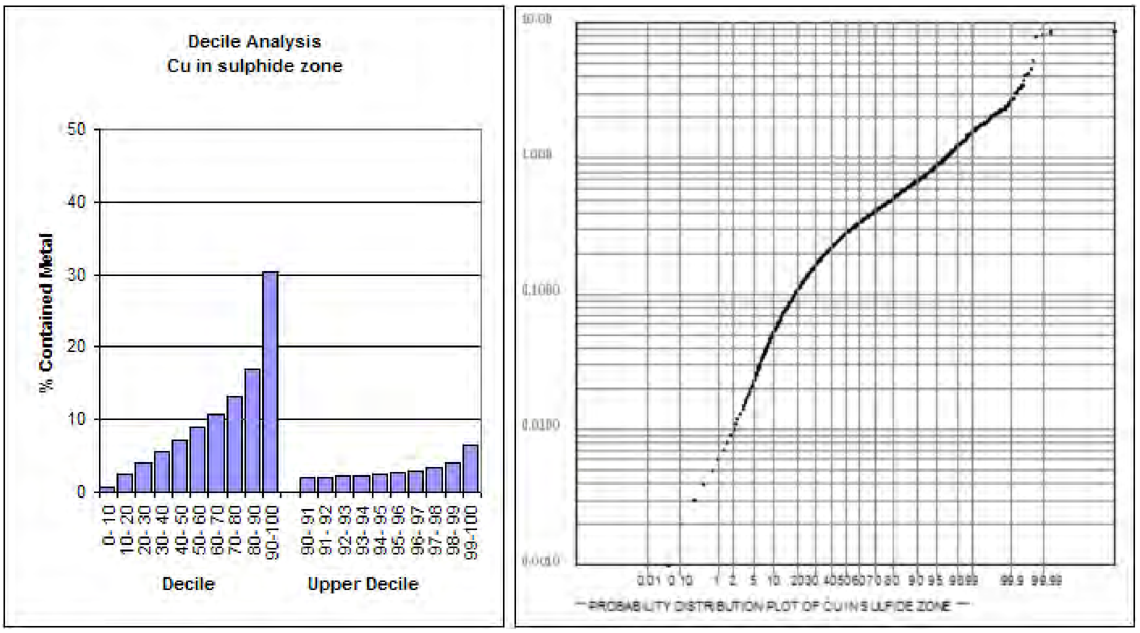

## 17.3 Grade Capping

Before compositing, the grade distribution in the raw sample data was examined to determine if grade capping or special treatment of high outliers was warranted. Log probability plots were examined for outlier populations and decile analyses were performed for Cu, Au, and Mo within the zone domains. As a general rule, the cutting of high grades is warranted if:

- the last decile (upper 10% of samples) contains more than 40% of the metal; or
- the last decile contains more than 2.3 times the metal of the previous decile; or
- the last centile (upper 1%) contains more than 10% of the metal; or
- the last centile contains more than 1.75 times the next highest centile.

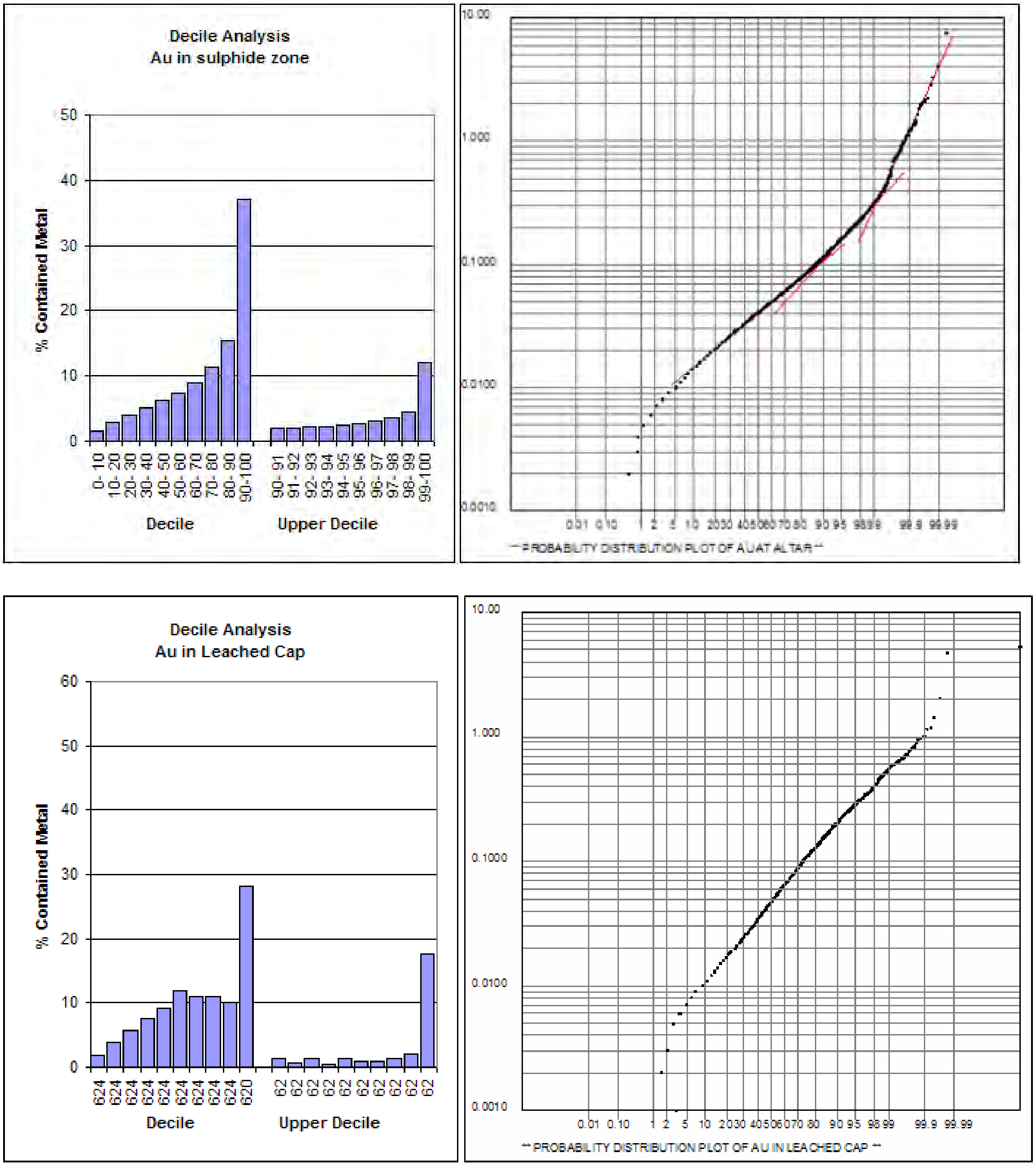
It was found that none of these criteria applied for Cu in the sulphide zone. However, it was deemed appropriate to cap six scattered outliers visible in the log probability plot at a level of 5% Cu (Figure 17-9).

**Figure 17-9 Decile and log probability plots for Cu**




The decile statistics for Au revealed that over 10% of the contained metal is within the upper percentile suggesting that capping is warranted (Figure 17-10). The log probability curve shows a marked inflection around 0.3 g/t which is close to the cap value of 0.32 suggested by the decile analysis at the 99$^{th}$ percentile level. Based on this a topcut value of 0.3 g/t was selected for Au and the cap was applied prior to compositing. This cap affected 202 samples or about 1% of the data. For Au in the leached cap, no inflection point was apparent at the 99$^{th}$ percentile and a higher cap grade of 2 g/t was imposed affecting 4 samples.

**Figure 17-10 Decile and log probability plots for Au**




Decile analyses for Mo and Ag indicated that both elements had over 10% contained metal in the upper decile. It was decided to cap both elements at the 99th percentile value of 5 g/t for Ag and 160 ppm for Mo prior to compositing.

## 17.4 Deposit Modeling

Lithology was interpreted on 13 E-W geologic cross sections and 19 N-S cross sections by Peregrine geologic staff. Modeled surfaces included the leached cap and overburden boundaries. A bedrock geologic plan was also produced based on a combination of outcrop mapping and drill hole information.

Solid models of the Tap, Tpa and Tdp units were created from the sectional interpretations and downhole lithology using Leapfrog3d software (Figure 17-11). The Tpr unit was not modeled as it was assumed to make up the remainder of the bedrock. A models of the leached cap was also formed (Figure 17-12). The solid models were used to code the model blocks for lithology and weathering zone.

**Figure 17-11 Lithologic models**

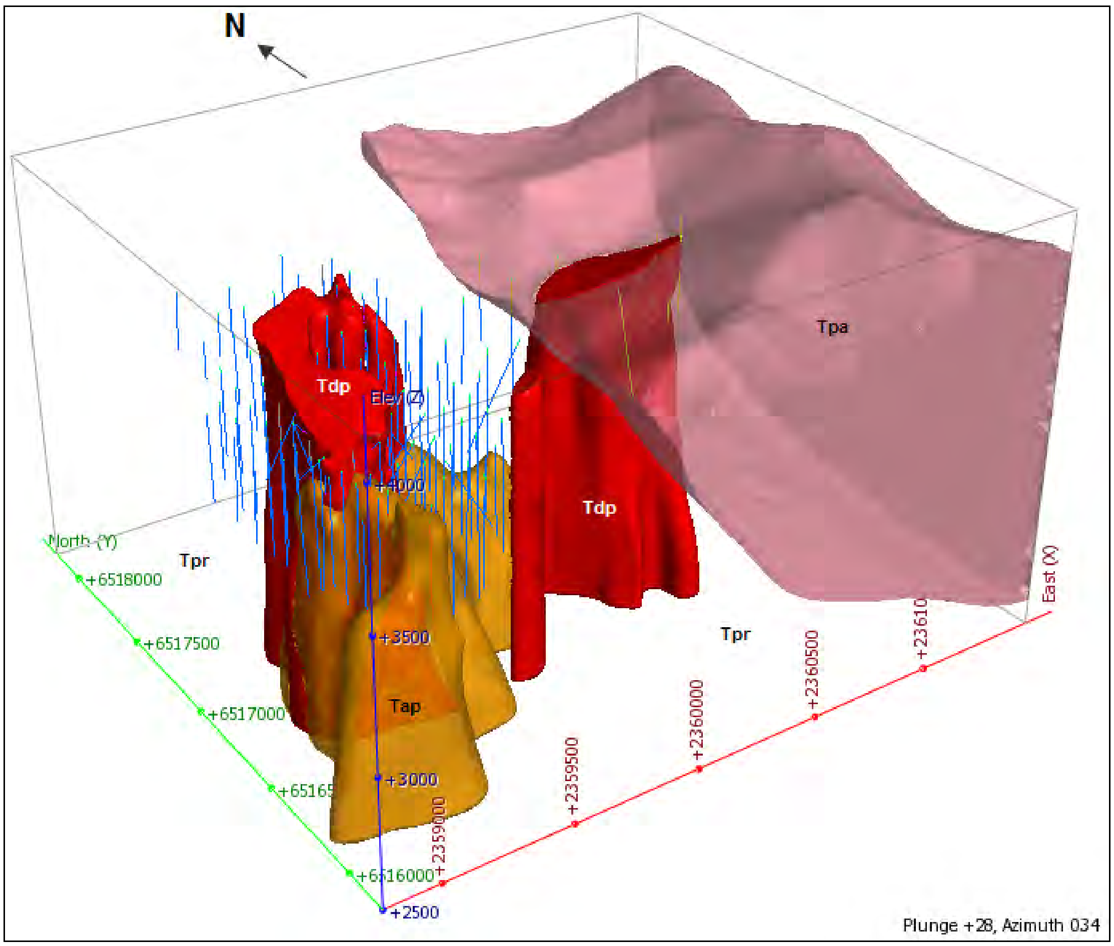


**Figure 17-12 Leached cap model**

## 17.5 Compositing

Fixed length downhole composites of Cu, Au, Ag Mo and As grades were generated using 10 metre intervals. A hard boundary was used between the leached cap and the sulphide zone for Cu and Au. Statistics for composites are summarized in the following table. Nine weakly mineralized holes on the periphery were excluded from this statistical analysis.

**Table 17-5 Composite Statistics**

| Zone | Sulphide | | Leached Cap | Combined | | |
|---|---|---|---|---|---|---|
| **Element** | **Cu %** | **Au g/t** | **Au g/t** | **Mo %** | **Ag g/t** | **As %** |
| Top Cut | 5 | 0.3 | 2.000 | 0.016 | 5 | - |
| n | 3862 | 3862 | 1294 | 5285 | 3994 | 3994 |
| min | 0.002 | 0.000 | 0.003 | 0.000 | 0.0 | 0.000 |
| max | 2.337 | 0.300 | 0.628 | 0.016 | 4.7 | 0.614 |
| mean | 0.345 | 0.056 | 0.086 | 0.002 | 0.8 | 0.031 |
| median | 0.299 | 0.045 | 0.052 | 0.001 | 0.7 | 0.015 |
| Var | 0.069 | 0.002 | 0.009 | 0.049 | 0.4 | 0.002 |
| Std Dev | 0.263 | 0.041 | 0.093 | 0.002 | 0.7 | 0.045 |
| COV | 0.762 | 0.730 | 1.090 | 1.039 | 0.8 | 1.446 |

## 17.6 Density

In 2010, 1005 core samples of the various lithologies, alteration and mineralization styles were measured for specific gravity ("SG") at Alex Stewart laboratories. This data was combined with previous 872 measurements and statistics were generated comparing SG to lithology and weathering (Figure 17-13).

Model blocks within the sulphide zone and leached cap were assigned a bulk density for volume to mass conversion based on the median value of the corresponding rock type and zone as shown in Table 17-6.

**Figure 17-13 Box plots showing SG distribution by rock type**

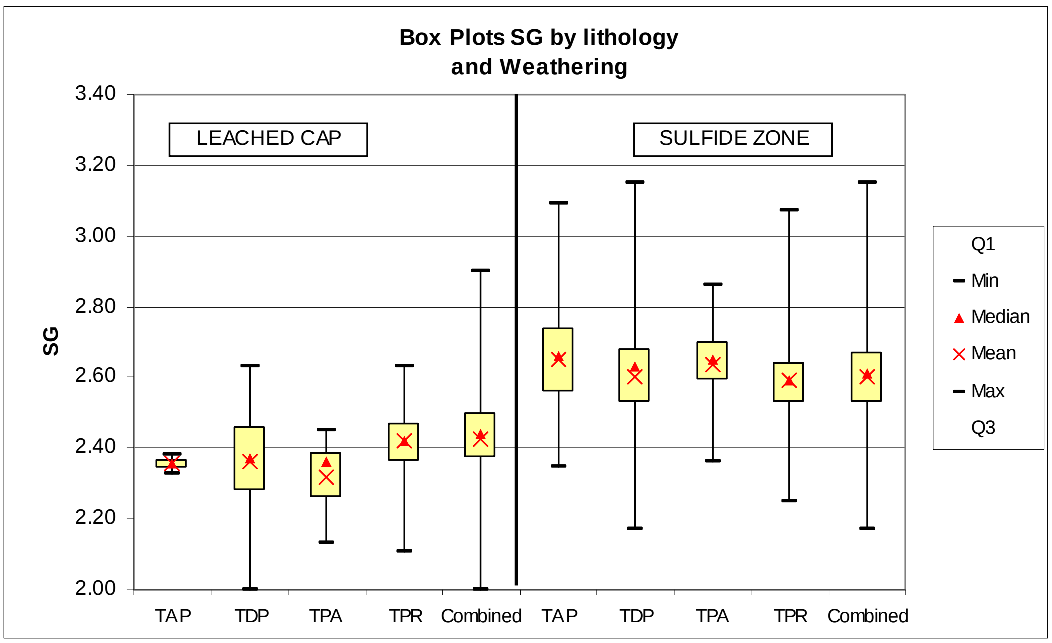



**Table 17-6 Specific gravity assignments**

| Unit | Subdivision | SG |
|---|---|---|
| 10 | Overburden | 2.00 |
| 1 | TAP LC | 2.36 |
| 8 | TDP LC | 2.37 |
| 3 | TPA LC | 2.36 |
| 4 | TPR LC | 2.42 |
| 1 | TAP | 2.66 |
| 8 | TDP | 2.63 |
| 3 | TPA | 2.65 |
| 4 | TPR | 2.59 |

## 17.7 Variogram Analysis

Normal semi-variograms for Cu and Au were modeled using composites falling within the zone constraint in order to determine kriging parameters, search parameters and anisotropy (Table 17-7). The Cu model showed a moderate anisotropy with the major and semi-major axis oriented vertically and in the N-S direction with the minor axis E-W. No significant anisotropy was evident in the models for the other elements.

**Table 17-7 Semi-variogram models**

| Model Item | Direction | co | c1 | a1 | c2 | a2 |
|---|---|---|---|---|---|---|
| Cu | -90->000° | 0.019 | 0.026 | 110 | 0.0287 | 365 |
| | 0->000° | 0.019 | 0.026 | 110 | 0.0287 | 365 |
| | 0->090° | 0.019 | 0.026 | 110 | 0.0287 | 246 |
| Au | Isotropic | 0.0004 | 0.0002 | 101 | 0.0011 | 392 |
| Au LC | Isotropic | 0.0019 | 0.004 | 247 | | |
| Ag | Isotropic | 0.108 | 0.09 | 35 | 0.18 | 305 |
| Mo (ppm) | Isotropic | 153 | 116 | 128 | 305 | 400 |
| As | Isotropic | 0.001 | 0.0006 | 79 | 0.0005 | 390 |

## 17.8 Block Model and Grade Estimation Procedures

A block model was created in Surpac Vision software using a block size of 15x15x15 metres. The parameters of the model are summarized in the following table.

**Table 17-8 Block model parameters**

| | Min | Max | Dist | size | # blocks |
|---|---|---|---|---|---|
| x | 2357500 | 2362000 | 4500 | 15 | 300 |
| y | 6515000 | 6519005 | 4005 | 15 | 267 |
| z | 2500 | 4405 | 1905 | 15 | 127 |

The partial percentage of each block below the topographic and overburden surfaces was calculated and stored as a block attribute.

Model blocks were initially assigned a lithologic code based on the majority of each block within solid models of the Tap, Tdp and Tpa units. All remaining unassigned blocks below the bedrock surface were then categorized as Tpr. Blocks were also given a zone code to differentiate the sulphide (hypogene) zone from the leached cap. Block density was assigned based on the median values for each lithology and zone.

### 17.8.1 Grade Models

***Total Cu and Au***

Cu and Au grades for blocks within the sulphide zone domain were estimated in two passes using the ordinary kriging method. Search parameters used for each pass are shown in Table 17-9. Octant searches were used in the first pass to limit grade extrapolation. Au grades were interpolated in the leached cap domain using a single kriging pass and hard boundaries.

**Table 17-9 Block estimation parameters for Cu and Au**

| Item | Domain | Kriging Pass | Search Type | Max Search Dist | Min # Composites | Max # Composites | Min Octants Required | Max per hole | Grade Cap |
|---|---|---|---|---|---|---|---|---|---|
| **Cu** | **Sulphide** | 1 | Octant | 274 | 6 | 32 | 5 | | 5% |
| | | 2 | Ellipsoidal | 274 | 6 | 32 | na | 4 | 5% |
| **Au** | **Sulphide** | 1 | Octant | 274 | 6 | 32 | 5 | | 0.3 g/t |
| | | 2 | Ellipsoidal | 300 | 6 | 32 | na | | 0.3 g/t |
| **Au** | **Leached Cap** | 1 | Ellipsoidal | 185 | 5 | 32 | na | 4 | 2 g/t |

The following set of figures illustrates the block grade distribution for Cu and Au in plan, section and perspective views. The frequency distributions of block grades are shown in Figure 17-19 and Figure 17-20.

**Figure 17-14 Block model grade distribution - Cu**

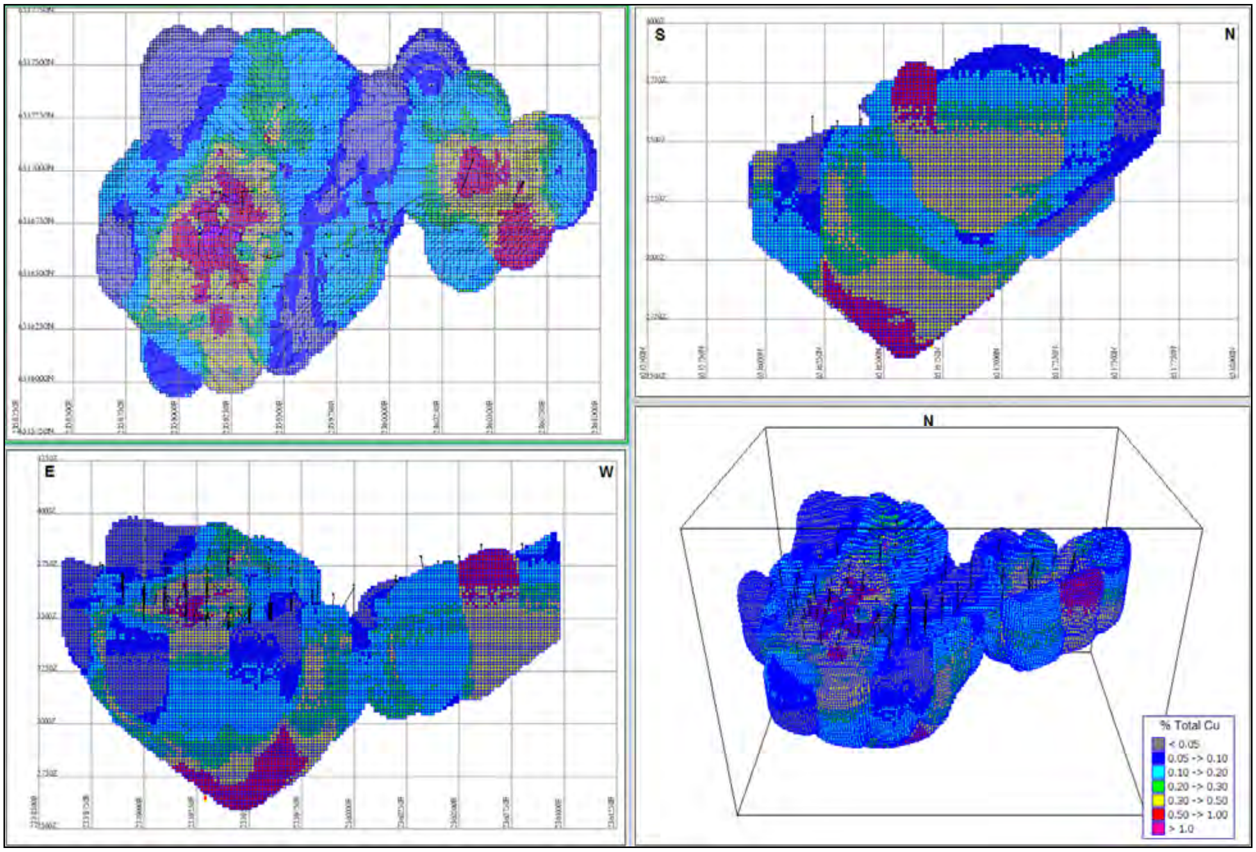


**Figure 17-15 Block model Cu grades - Section 6516700 North**

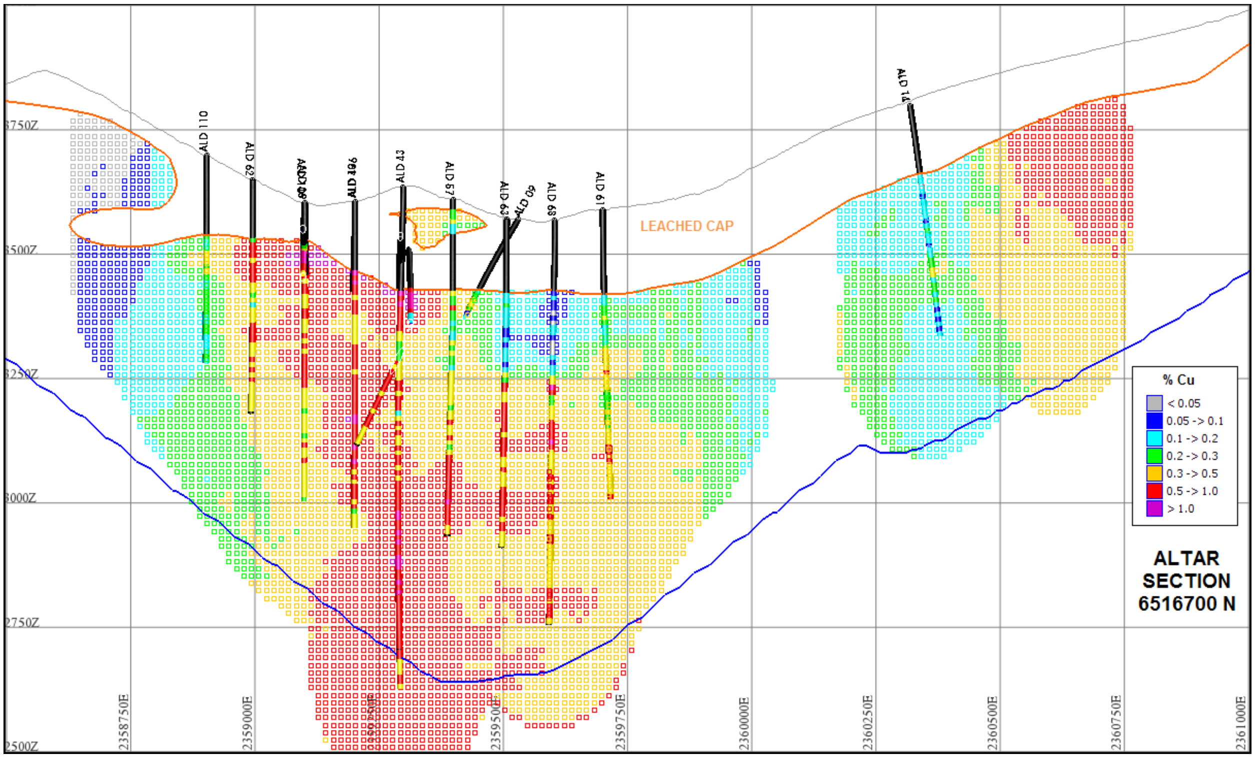


**Figure 17-16 Block model Cu grades - Section 2359300 East**

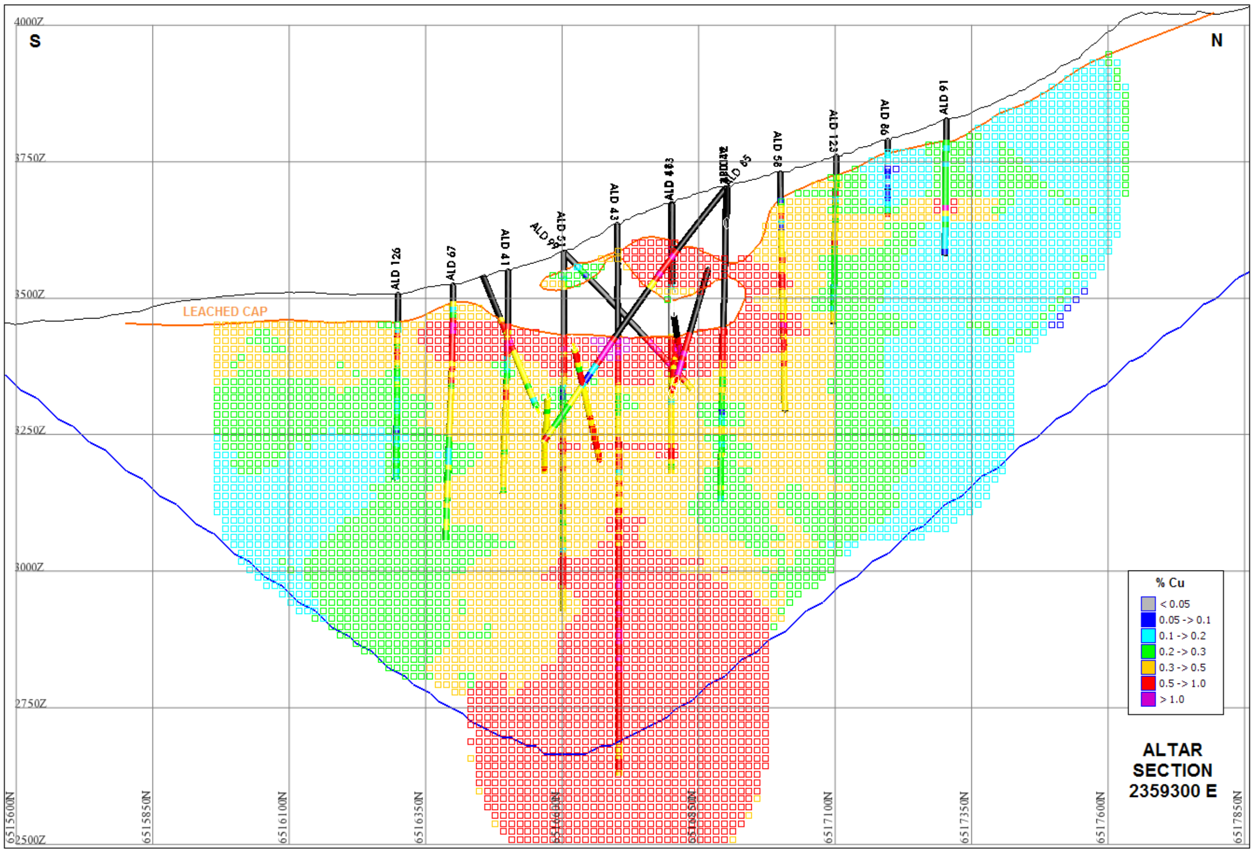


**Figure 17-17 Block model grade distribution - Au**

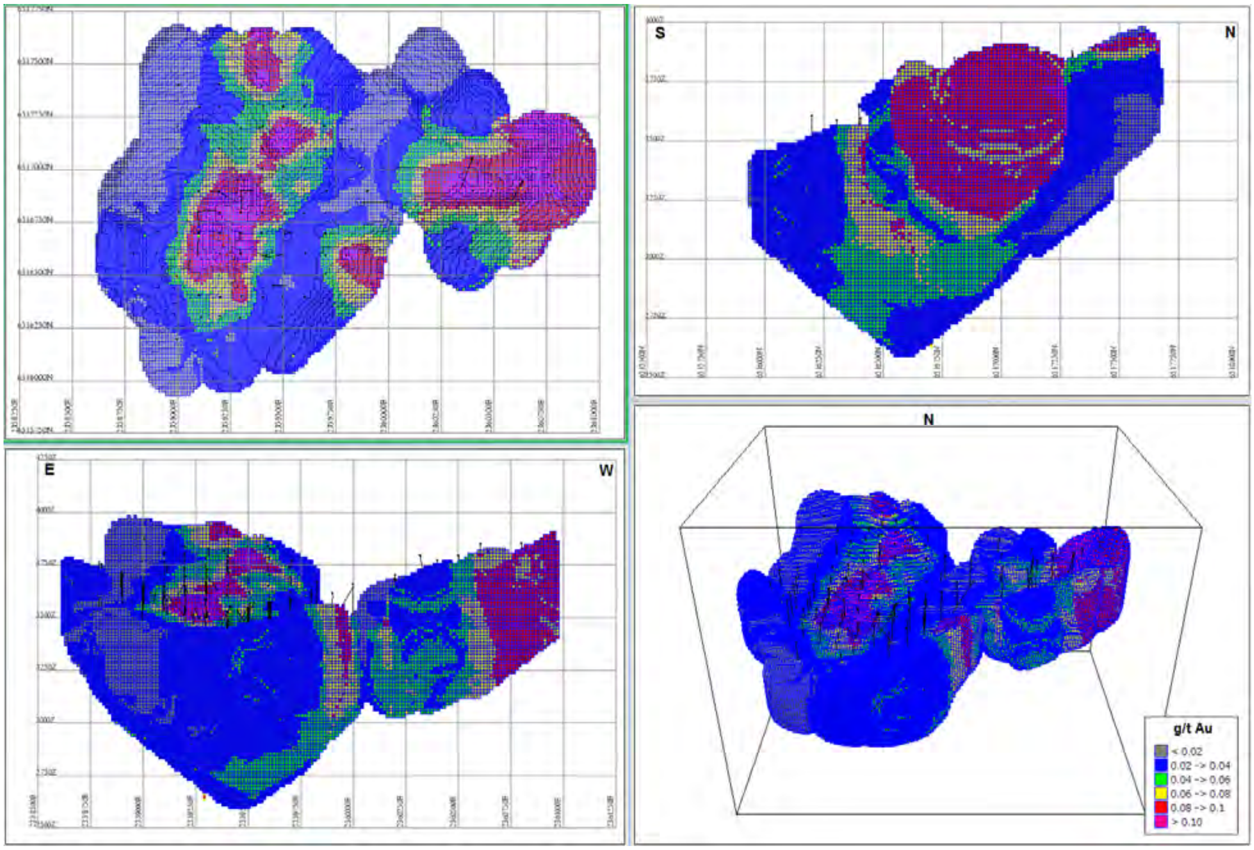


**Figure 17-18 Block model Au grades in sulphide zone - Section 6516700 North**

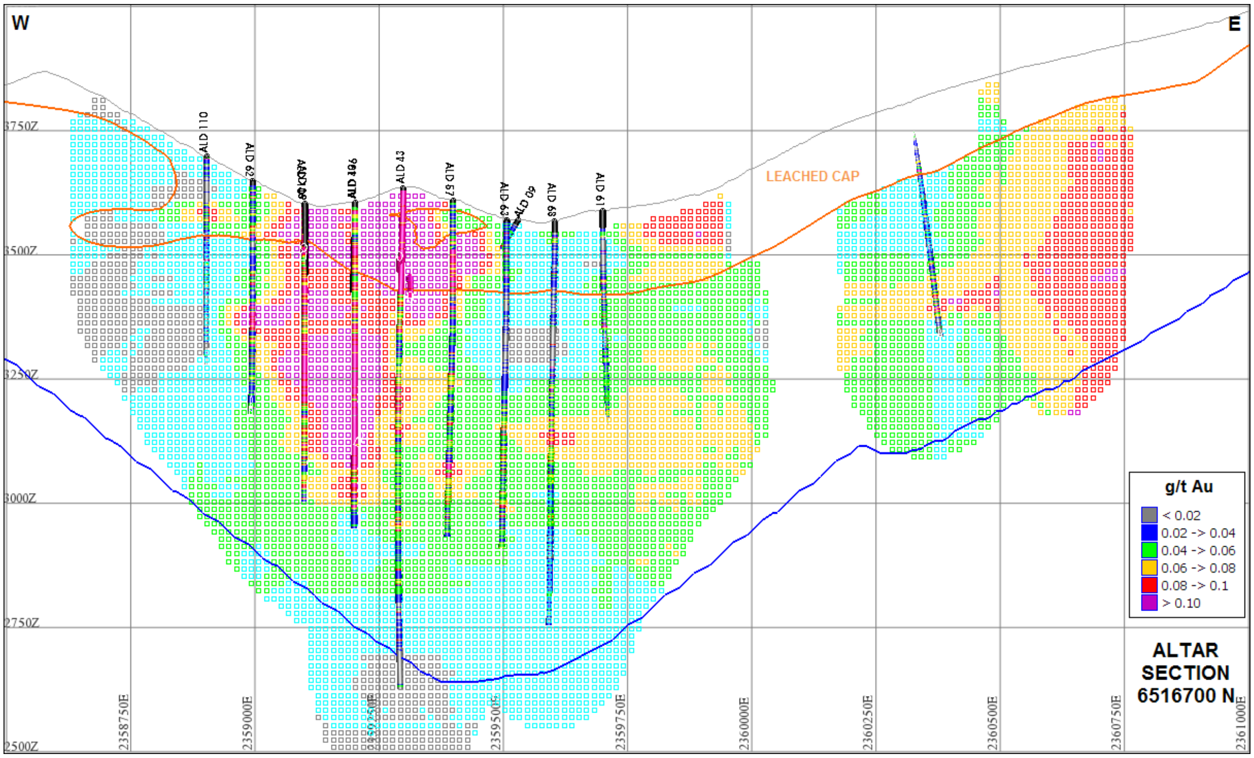



**Figure 17-19 Frequency distribution of Cu grades in block model**

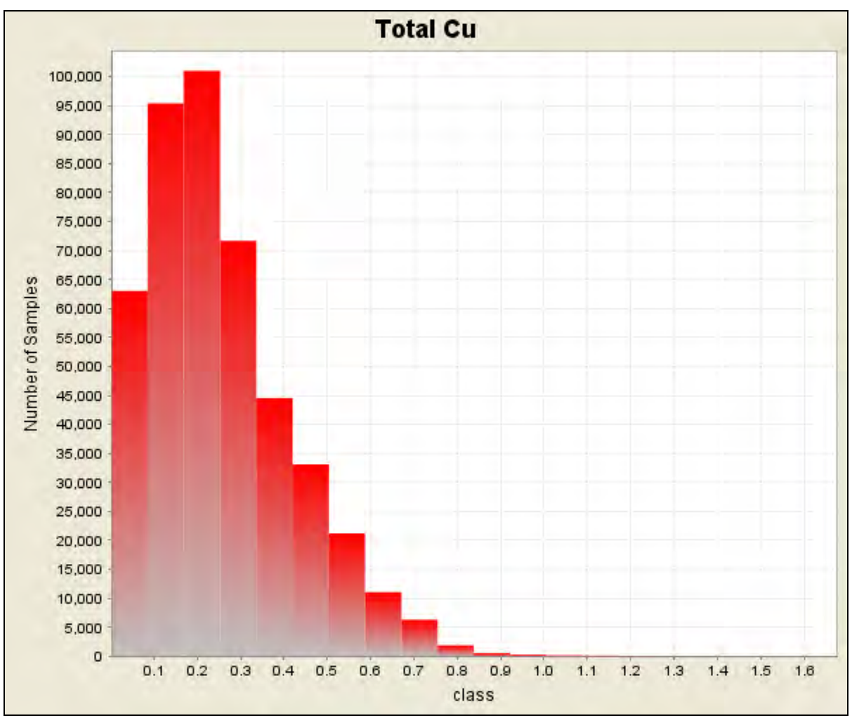
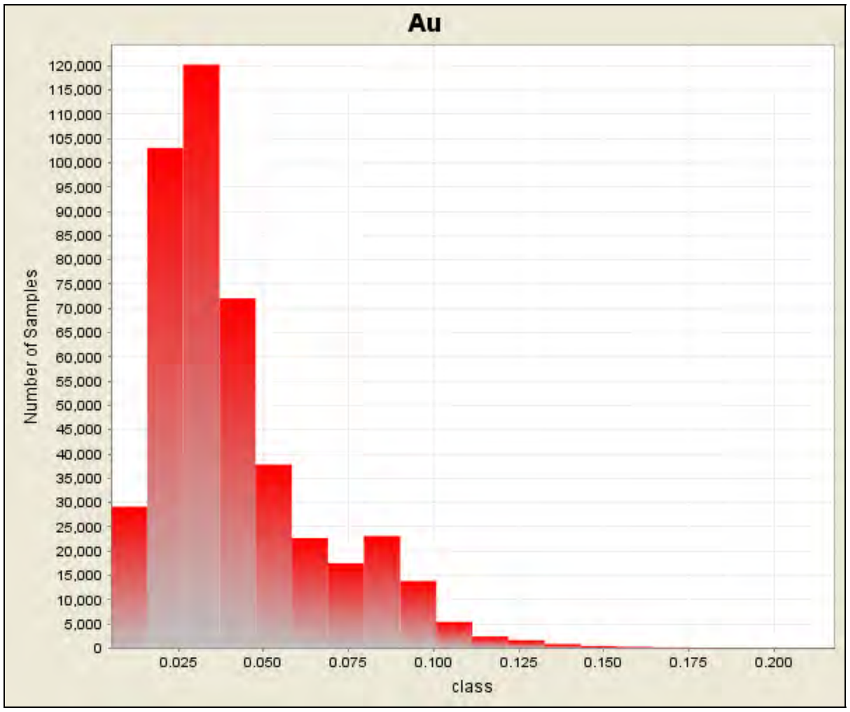


**Figure 17-20 Frequency distribution of Au grades in block model**




***Soluble Copper***

A zone of potentially leachable Cu was defined based on sample results and distribution for the Main (western) Zone. It was generated from sectional profiles based on the deepest occurrence of >0.05% CuCN in each analyzed drill hole. Within the zone of potentially leachable material it was found that there were significantly more CuT assays than sequential Cu analysis (about 23% more). This was partly because there is no sequential Cu data for the first 16 historic holes but there were also data gaps from more recent holes.

A statistical comparison was carried out on the intervals within the zone that contain both sets of data. Results show that the CuT assays average 3.7% higher than the calculated total from the sequential Cu analyses. This is not unexpected as the aqua regia digestion is not as aggressive as the 3 or 4 acid digestion used for the CuT assays. About 24% of the calculated Cu (sequential) grades are greater than the CuT assay but most of the differences are minor or at very low Cu levels. About 9% are greater by over 0.01% for CuT values exceeding 0.1% and about 1% of these vary from 0.05 to as much as 0.52% higher.

The methodology for estimating the CuCN and CuHS values in the block model involved the calculation of the relative percent of each item compared to the calculated total seq Cu value then applying this percent to the estimated CuT of each block in the zone. If CuCN and CuHS were estimated directly then it would inevitably result in a number of areas where soluble Cu grades exceeded the estimated CuT value. This was due primarily to the missing sequential Cu data but also in rare cases where the sequential Cu total exceeded the original CuT assay. It can be demonstrated that using just the sequential Cu data will overestimate the grade by comparing the global average values. The average of all the calculated total sequential analyses at 0.385% is higher than the average of the original CuT assays of 0.357% (using 23% more data).

The modeling procedure is outlined as follows:

- CuT block values are estimated using ordinary kriging as described above.
- All intervals with sequential Cu values in the zone are composited to 10m.
- Relative percentages of CuCN and CuHS are calculated for each composite based on the following formulas.

  %CuCN=CuCN/(CuCN+CuHS+CuR)

  %CuHS=CuHS/(CuCN+CuHS+CuR)

- The percentage values are interpolated into the block model using the inverse distance squared method.
- The CuCN and CuHS values for each block in the zone are calculated by applying the percentage factor to CuT.

  CuCN=%CuCN*CuT

  CuHS=%CuHS*CuT

- The calculated CuCN and CuHS values are then reduced by 3.7% which is the average difference between the composited CuT assays and the calculated total sequential Cu values.

The extent of the model estimated for soluble Cu and the grade distribution is illustrated in Figure 17-21. Block model statistics for the estimated region are summarized in Table 17-10.

**Figure 17-21 Extents and grade distribution of Soluble Cu model**

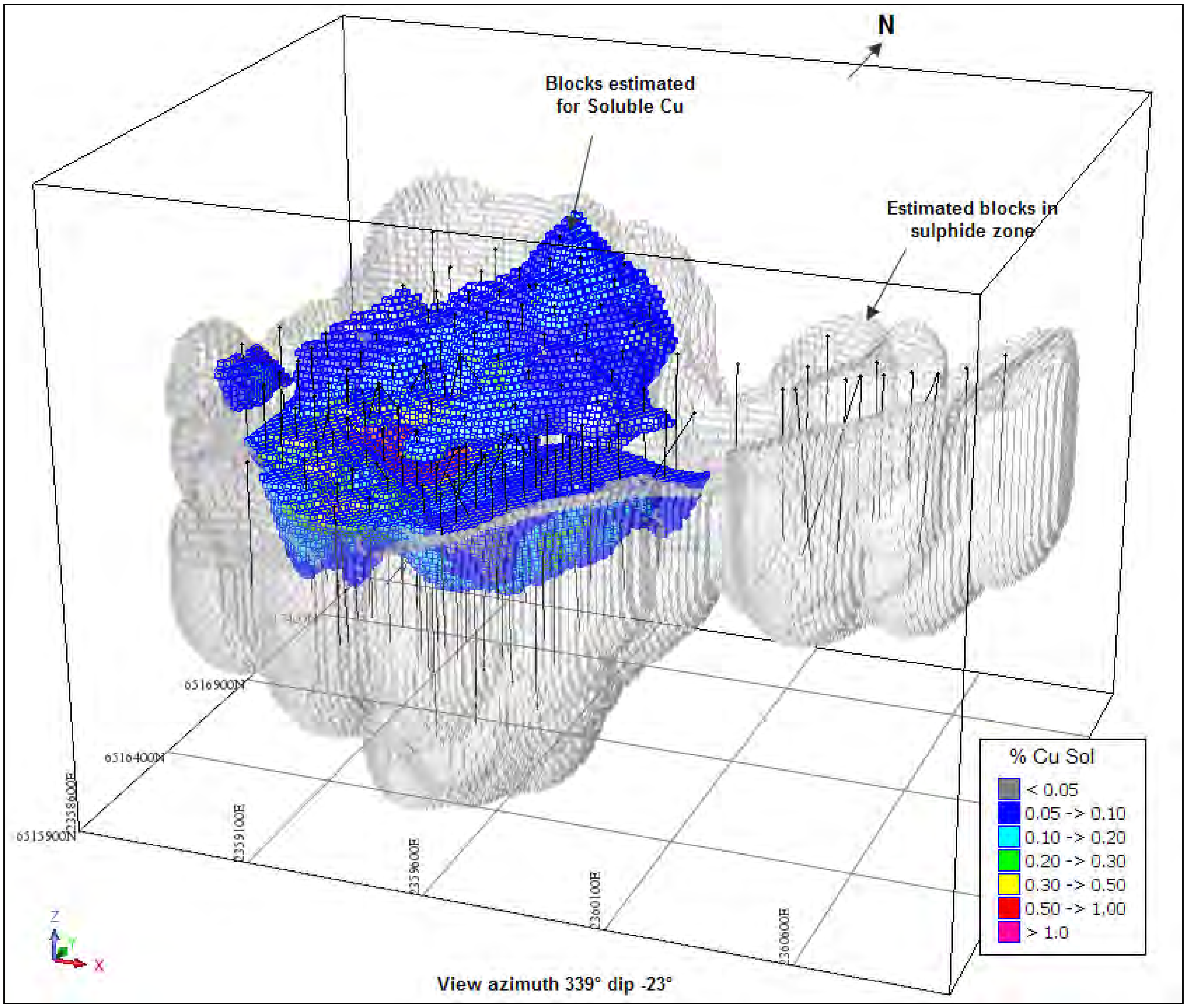



**Table 17-10 Block model statistics for soluble Cu**

| Cutoff Grade % CuSol | KTonnes ≥ Cutoff | Total Cu Sol % | Cu CN % | Cu AS % | T Cu % |
|---|---|---|---|---|---|
| 0.00 | 418,063 | 0.15 | 0.13 | 0.03 | 0.31 |
| 0.05 | 376,144 | 0.17 | 0.14 | 0.03 | 0.34 |
| 0.10 | 247,508 | 0.21 | 0.18 | 0.04 | 0.41 |
| 0.15 | 156,414 | 0.27 | 0.22 | 0.05 | 0.48 |
| 0.20 | 95,698 | 0.32 | 0.27 | 0.06 | 0.57 |
| 0.25 | 60,986 | 0.38 | 0.32 | 0.07 | 0.64 |
| 0.30 | 39,433 | 0.44 | 0.37 | 0.07 | 0.72 |

Metallurgical test work on the potential leachable material composites is still incomplete and the soluble Cu is not reported as part of the Altar mineral resource at this time.

***Mo, Ag and As***

Grades for Mo, Ag and As are not reported as part of the mineral resource but were estimated in the block model. Parameters used for the grade estimation are summarized in Table 17-11.

**Table 17-11 Block estimation parameters for Mo, Ag and As**

| Item | Domain | Kriging Pass | Search Type | Max Search Dist | Min # Composites | Max # Composites | Min Octants Required | Grade Cap |
|---|---|---|---|---|---|---|---|---|
| Mo | All | 1 | Octant | 274 | 6 | 32 | na | 160 ppm |
| | | 2 | Ellipsoidal | 300 | 6 | 32 | 5 | 160 ppm |
| Ag | All | 1 | Octant | 274 | 6 | 32 | na | 5 g/t |
| | | 2 | Ellipsoidal | 300 | 6 | 32 | 5 | 5 g/t |
| As | All | 1 | Octant | 274 | 6 | 32 | na | - |
| | | 2 | Ellipsoidal | 300 | 6 | 32 | 5 | - |

Anomalous Mo grades tend to occur in a halo surrounding the higher grade Au blocks as illustrated in Figure 17-23.

Silver content is generally low, averaging 1.4 g/t at a cut-off grade of 0.3% Cu equivalent. The highest estimated block Ag grades are near the centre of the main zone just below the leached cap and grade just marginally above 3 g/t.

The higher As grades (> 500 ppm) occur in the leached cap above the core of the West Zone and in the upper portion of the West Zone where enargite is more common. The average arsenic content of the mineral resource is 0.028%. Only 1% of the resource blocks have estimated As content exceeding 0.1% and the highest estimated block grade is 0.27%. Table 17-12 shows the arsenic distribution in resource blocks above a 0.3% Cu equivalent cut-off grade. The frequency distribution of As in the resource blocks is illustrated in Figure 17-22.

**Table 17-12 Arsenic distribution**

| COG %As | KTonnes > COG | Avg %As content | % of Total Tonnes |
|---|---|---|---|
| 0.00 | 1,270,546 | 0.028 | 100% |
| 0.01 | 1,094,255 | 0.032 | 86% |
| 0.02 | 705,487 | 0.041 | 56% |
| 0.03 | 455,851 | 0.051 | 36% |
| 0.04 | 305,048 | 0.058 | 24% |
| 0.05 | 174,981 | 0.068 | 14% |
| 0.06 | 88,052 | 0.082 | 6.9% |
| 0.07 | 53,837 | 0.093 | 4.2% |

| COG %As | KTonnes > COG | Avg %As content | % of Total Tonnes |
|---|---|---|---|
| 0.08 | 34,139 | 0.103 | 2.7% |
| 0.09 | 22,350 | 0.113 | 1.8% |
| 0.10 | 14,208 | 0.123 | 1.1% |
| 0.11 | 9,074 | 0.133 | 0.7% |
| 0.12 | 5,572 | 0.144 | 0.44% |
| 0.13 | 3,471 | 0.155 | 0.27% |
| 0.14 | 1,988 | 0.171 | 0.16% |
| 0.15 | 1,164 | 0.191 | 0.09% |
| 0.20 | 472 | 0.232 | 0.04% |
| 0.25 | 306 | 0.239 | 0.02% |
| 0.28 | 0 | - | 0.00% |

**Figure 17-22 Frequency distribution of As in classified resource blocks**

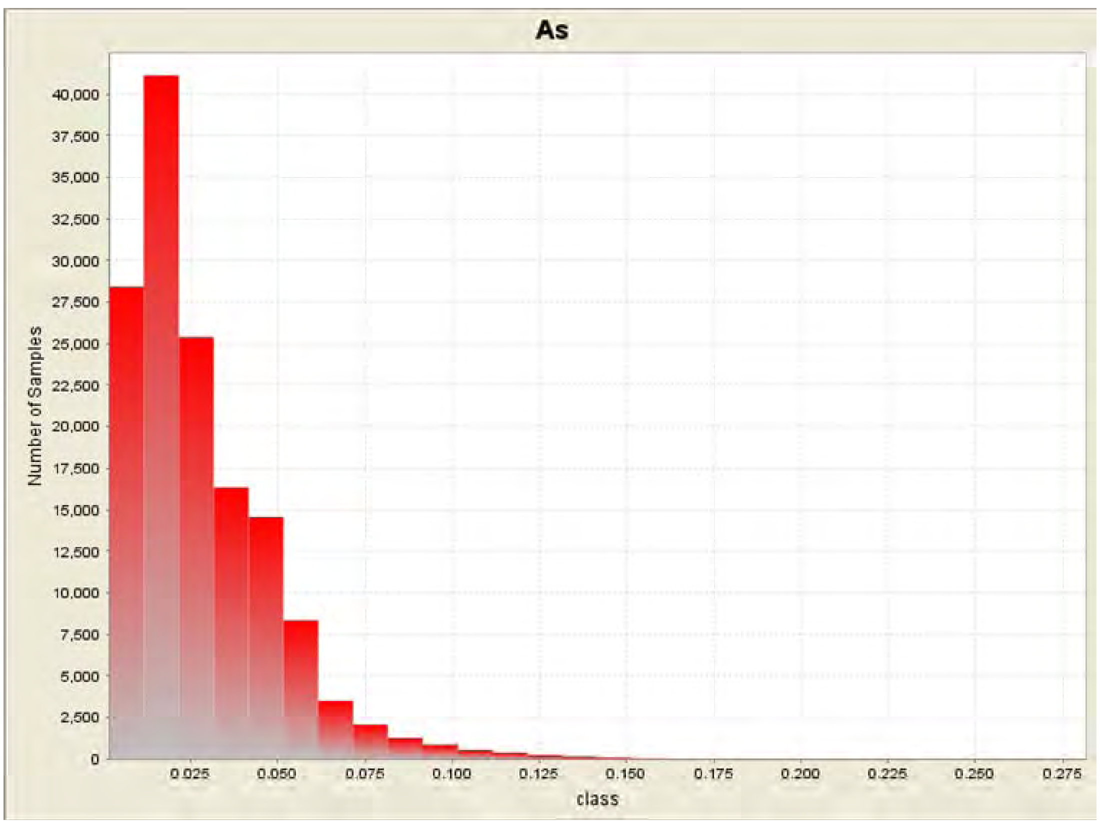
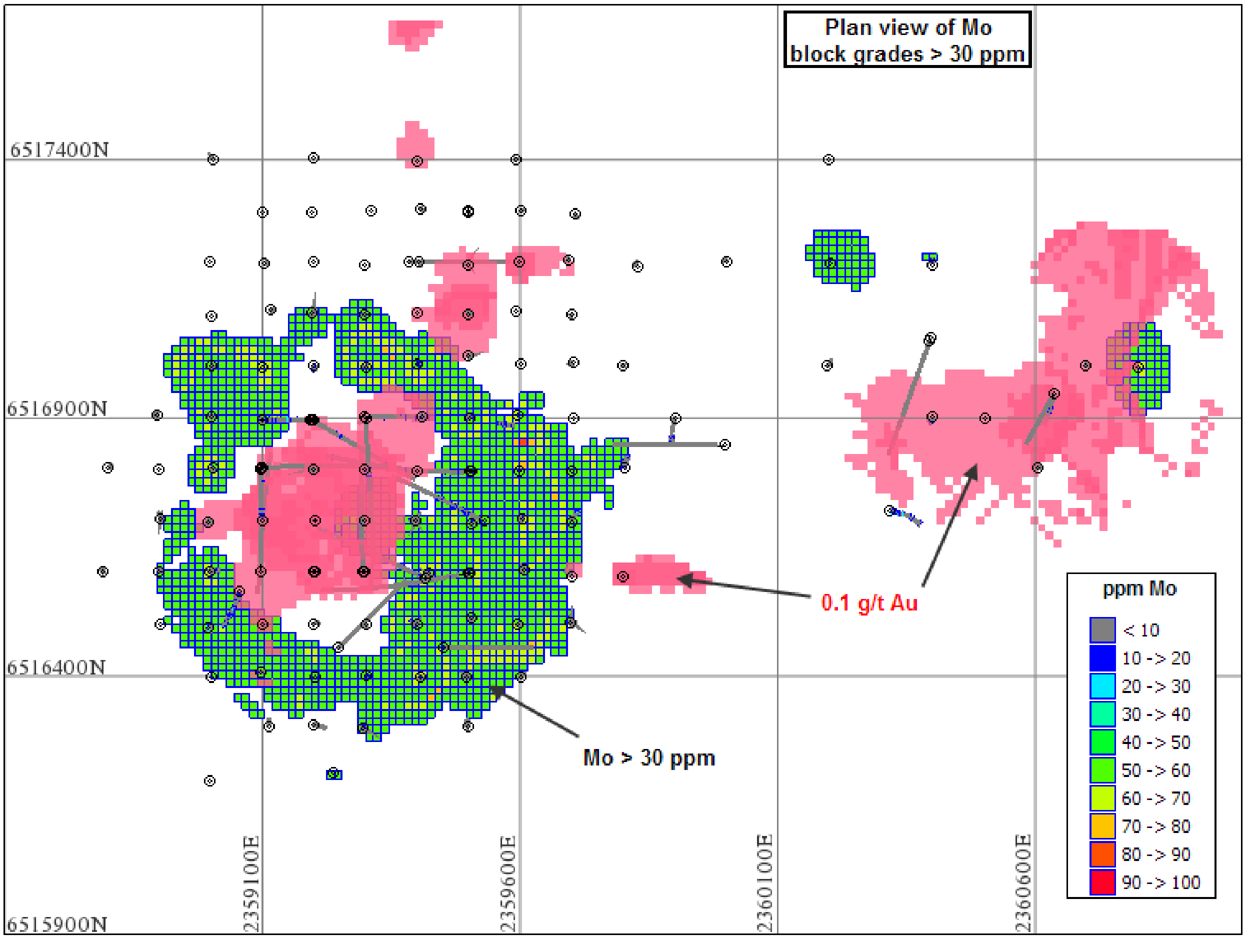


**Figure 17-23 Anomalous Mo grade distribution**




## 17.9 Mineral Resource Classification

Resource classifications used in this study conform to the following definition from National Instrument 43-101:

***Mineral Resource***

*A Mineral Resource is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.*

***Measured Mineral Resource***

*A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and* *economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.*

***Indicated Mineral Resource***

*An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.*

***Inferred Mineral Resource***

*An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.*

In order to meet the requirements of NI43-101 with respect to reasonable prospects of economic extraction, by open pit mining methods and a conventional copper concentrator, a 38°wall slope floating cone pit was generated to constrain the resource within the block model. Metal prices assumed were $2.80/lb for copper and $850/ounce for gold. Metallurgical recoveries assumed were 90% for copper and 65% for gold. Metallurgical test-work is currently in progress and will provide improved estimates of recovery for future resource evaluation. General & Administration, Processing and Ore Mining costs were assumed to be $6.89/tonne. Base waste mining costs were assumed to be $1.28/tonne. Ore and waste mining costs were incremented by $0.015/tonne/bench above and below the entrance bench elevation of 3500 metres.

Blocks estimated in the first pass for Cu using an octant search were classified as 'Measured' if the following criteria were met:

1. Within the sulphide (hypogene) zone
2. Within the area of 100m grid drilling in the western zone
3. Above the 2900 metre elevation level and a superimposed depth limit modeled visually on cross sections.

Blocks estimated in the first or second pass that were not assigned to the measured category were classified as 'Indicated' under the following conditions:

4. Within the sulphide (hypogene) zone

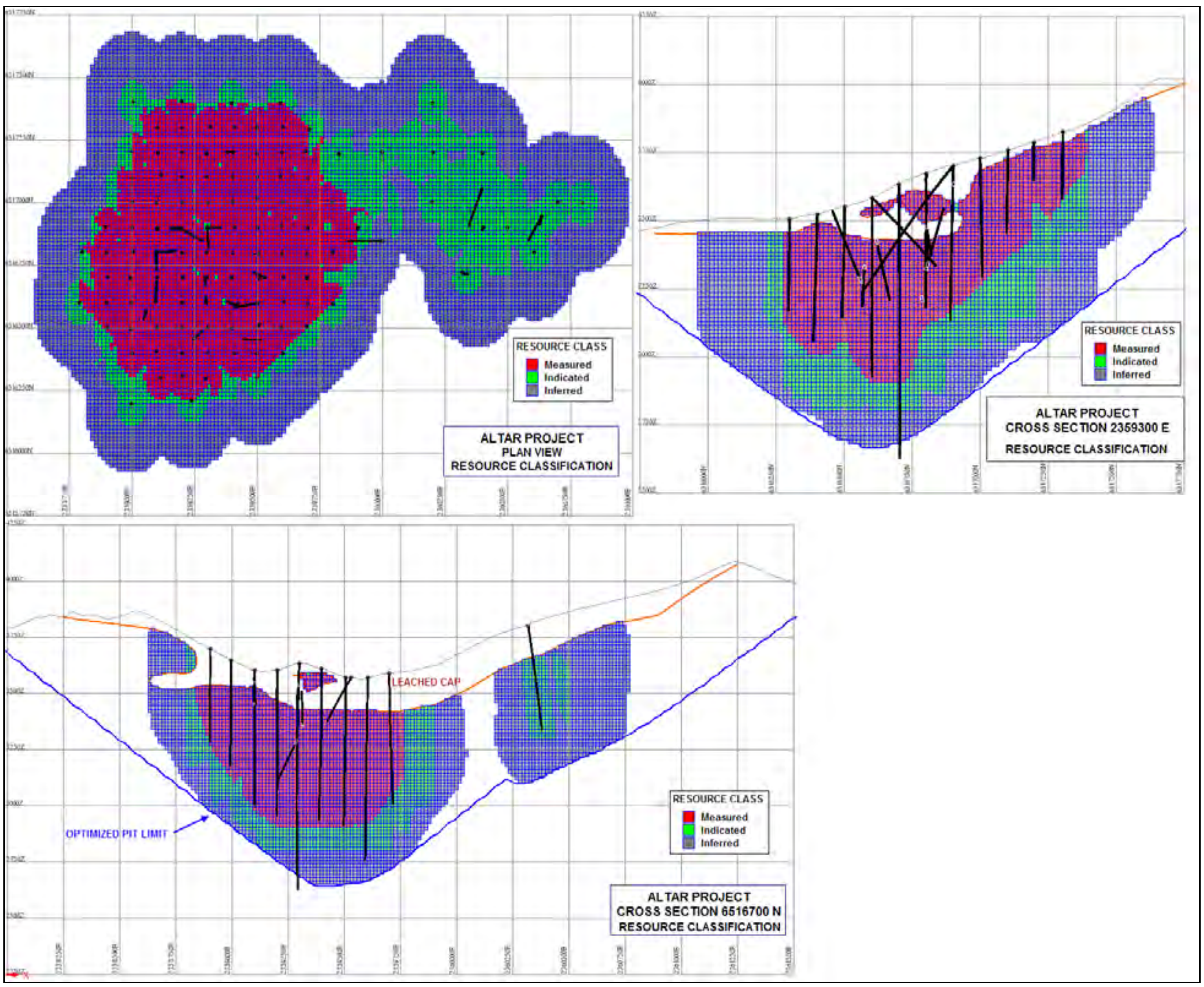
5. No more than 90m from the closest composite

6. Above the 2800 metre elevation level

All remaining estimated were assigned to the ‘Inferred’ category with the exception of blocks lying beyond the boundaries of the optimized pit which remained unclassified.

Figure 17-24 shows the distribution of the three classes in plan and section views.

**Figure 17-24 Model Classification**

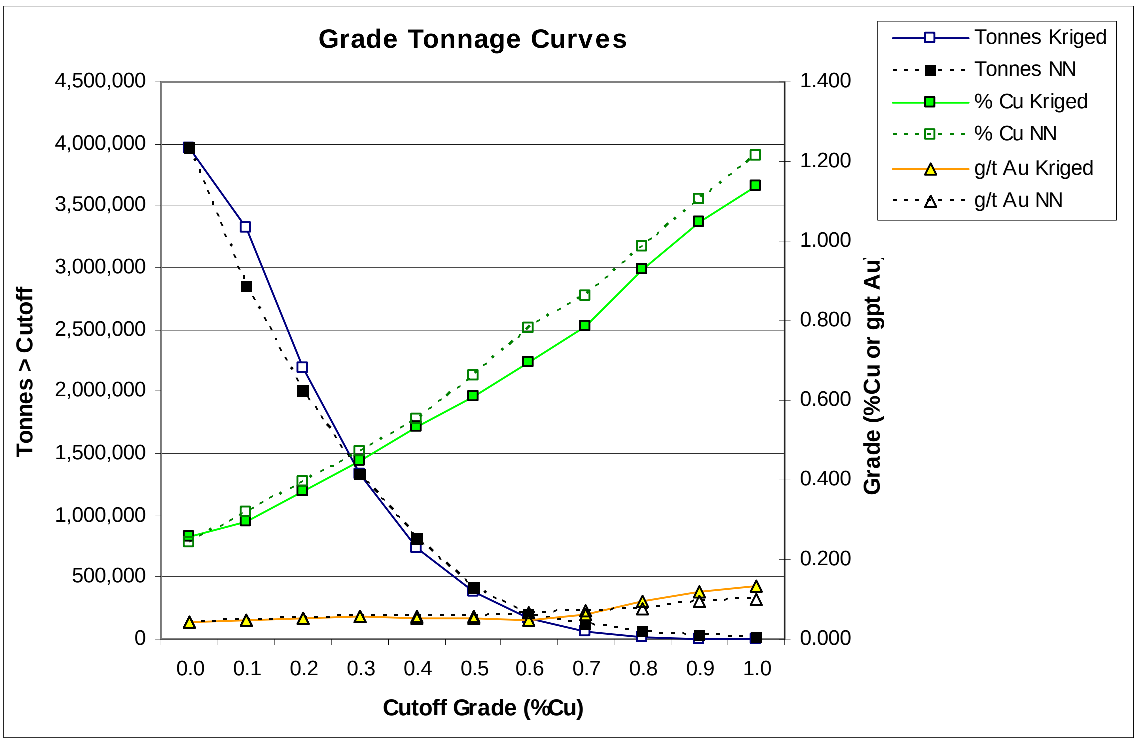


## 17.10 Model Validation

Model verification was initially carried out by visual comparison of blocks and sample grades in plan and section views. The estimated block grades showed good correlation with adjacent composite grades.

The model was also estimated using the Nearest Neighbour method (NN). The search strategy was identical to the Kriged runs. A comparison of the grade-tonnage curves (Figure 17-25) shows very close agreement between the Kriged and NN models with only marginally higher average Cu grades for NN. It is concluded that the block model constraints and search parameters have resulted in an acceptable level of smoothing in the Kriged estimate.

**Figure 17-25 Comparison of Kriging to Nearest Neighbour grade estimates**




Swath plots were generated to assess the model for global bias by comparing Kriged and nearest neighbour estimates on panels through the deposit. Results show a good comparison between the methods, particularly in the main portions of the deposit indicated by the bar charts (Figure 17-26 to Figure 17-29).

**Figure 17-26 Swath plot (E-W) at 6516700 North - Cu**

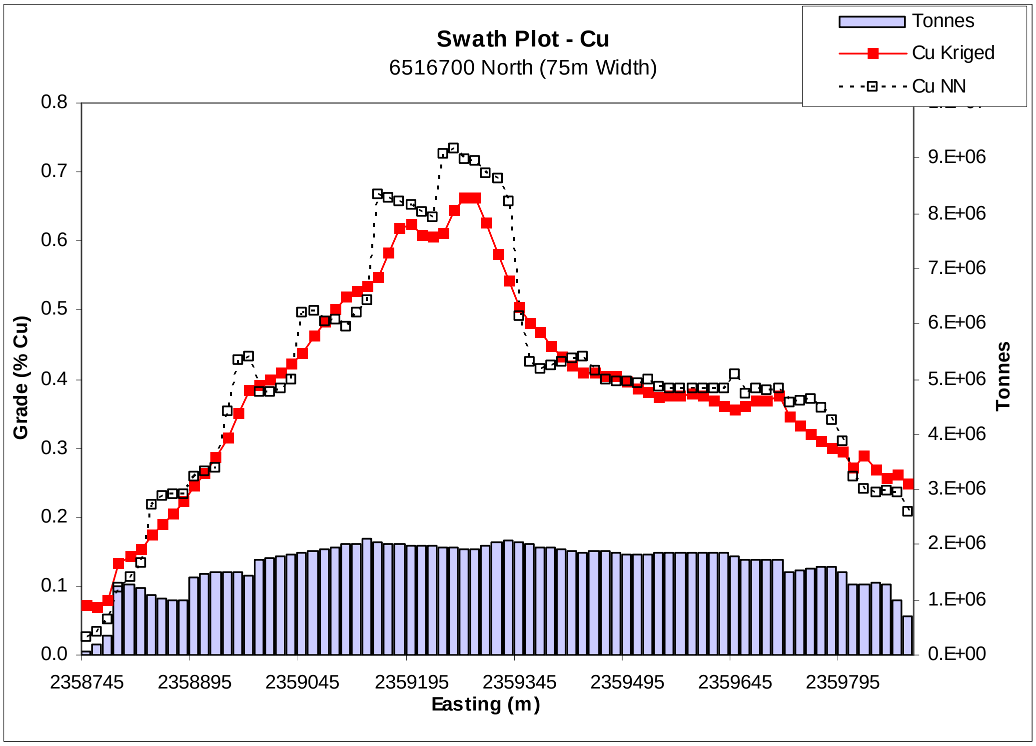



**Figure 17-27 Swath plot (E-W) at 6516700 North - Au**

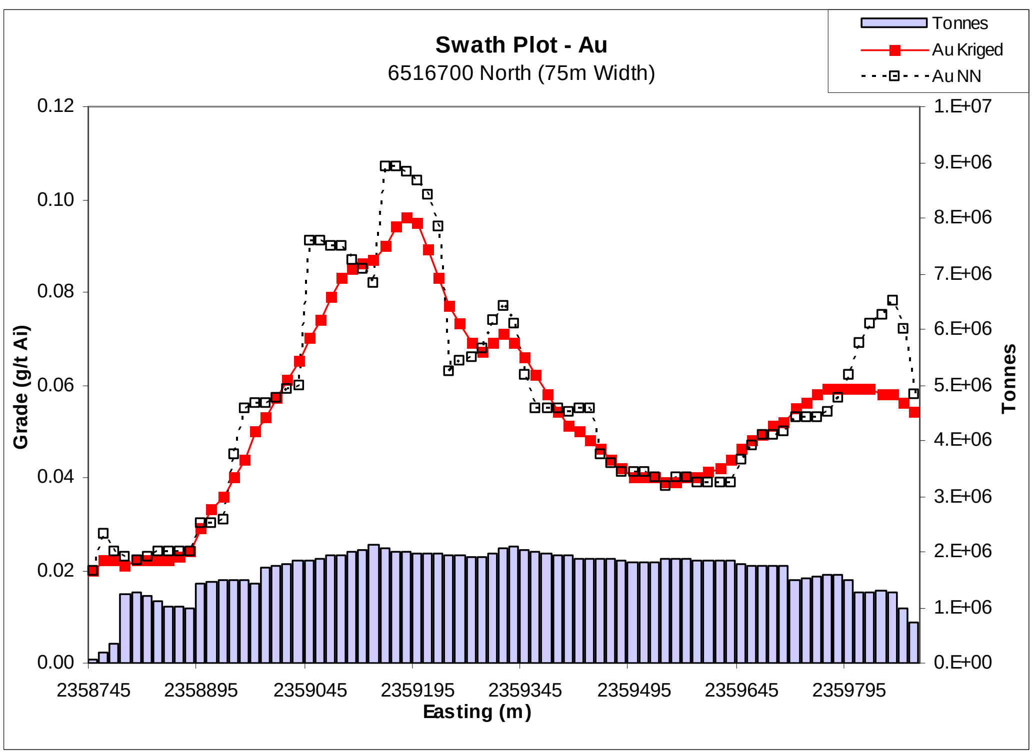


**Figure 17-28 Swath plot (S-N) at 2359300 East - Cu**

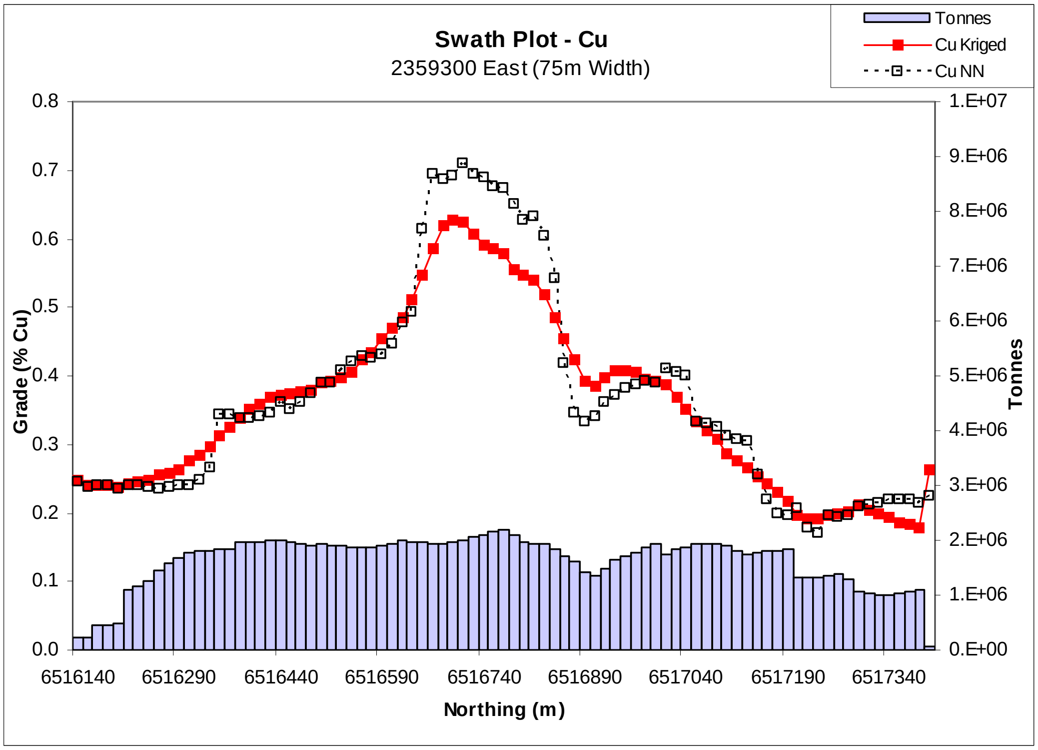
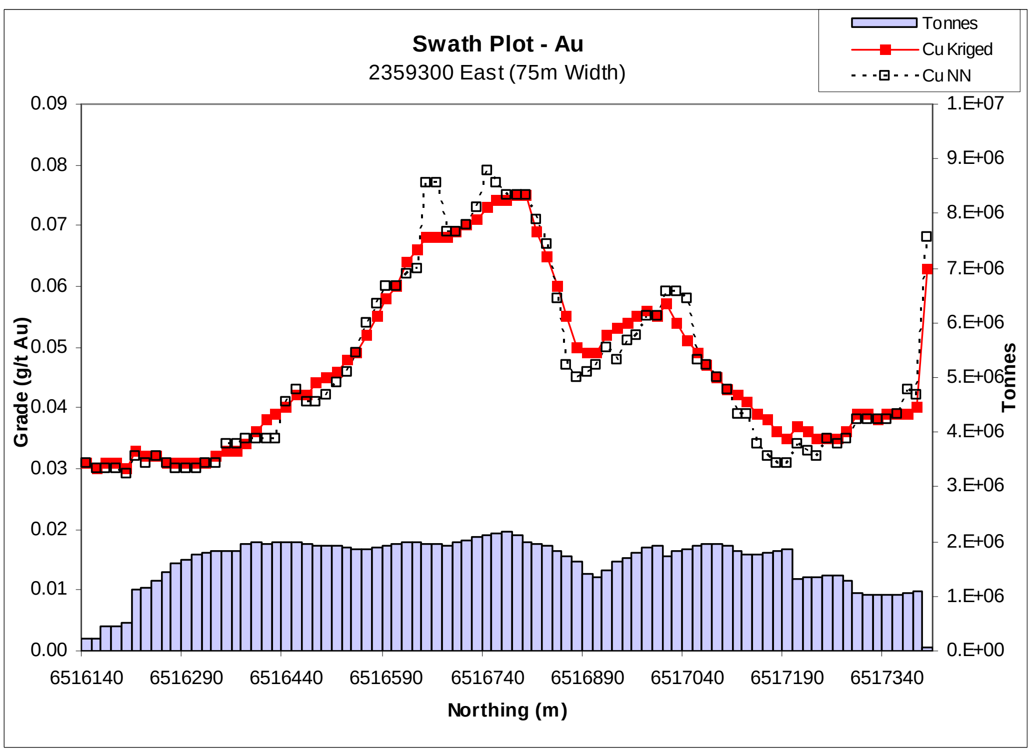



**Figure 17-29 Swath plot (S-N) at 2359300 East - Au**



## 17.11 Mineral Resource Summary

The following tables present the mineral resource estimate for the Altar Project at a range of cut-off grades with the base case in bold face. The selected base case cut-off grade of 0.3% copper equivalent is considered consistent with other mineral deposits of similar characteristics, scale and location.

**Table 17-13 Altar Project mineral resource summary**

| Cut-off | Quantity | Grade | | | Contained Metal | |
|---|---|---|---|---|---|---|
| Copper Equivalent (%) | Tonnes (Millions) | Copper (%) | Gold (g/t) | Copper Equiv (%) | Copper (Billion lbs) | Gold (Million oz) |
| **MEASURED** | | | | | | |
| 0.2 | 669 | 0.38 | 0.057 | 0.40 | 5.62 | 1.22 |
| **0.3** | **491** | **0.43** | **0.061** | **0.45** | **4.69** | **0.96** |
| 0.4 | 278 | 0.51 | 0.070 | 0.53 | 3.12 | 0.62 |
| 0.5 | 126 | 0.61 | 0.082 | 0.63 | 1.69 | 0.33 |
| **INDICATED** | | | | | | |
| 0.2 | 541 | 0.33 | 0.050 | 0.34 | 3.92 | 0.87 |
| **0.3** | **311** | **0.40** | **0.057** | **0.41** | **2.72** | **0.57** |
| 0.4 | 137 | 0.49 | 0.061 | 0.51 | 1.47 | 0.27 |
| 0.5 | 60 | 0.56 | 0.058 | 0.58 | 0.74 | 0.11 |
| **MEASURED+INDICATED** | | | | | | |
| 0.2 | 1,210 | 0.36 | 0.054 | 0.37 | 9.54 | 2.09 |
| **0.3** | **802** | **0.42** | **0.059** | **0.44** | **7.41** | **1.53** |
| 0.4 | 414 | 0.50 | 0.067 | 0.52 | 4.59 | 0.89 |
| 0.5 | 186 | 0.59 | 0.074 | 0.62 | 2.43 | 0.44 |
| **INFERRED** | | | | | | |
| 0.2 | 906 | 0.33 | 0.053 | 0.34 | 6.53 | 1.56 |
| **0.3** | **465** | **0.42** | **0.058** | **0.44** | **4.32** | **0.88** |
| 0.4 | 252 | 0.50 | 0.054 | 0.52 | 2.80 | 0.44 |
| 0.5 | 135 | 0.57 | 0.048 | 0.58 | 1.69 | 0.21 |

* The copper equivalent (“CuEq”) calculation is based on a copper price of $2.80/lb and gold price of $850/oz. It also includes a factor to compensate for an assumed gold recovery of 65% and a 90% recovery for copper.

The average arsenic content of the mineral resource is 0.028%. Only 1% of the resource blocks have estimated As content exceeding 0.1% and the highest estimated block grade is 0.27%.

# 18.0 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORT ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

The Property is at an exploration-stage; therefore this section is not applicable to this Technical Report.

# 19.0 OTHER RELEVANT DATA AND INFORMATION

There are no other data known to NMS or Geosim that are relevant to this Technical Report: therefore there are no relevant data or information presented in this section. Furthermore, there are no known factors or issues that materially affect the estimate of mineral resources.

# 20.0 INTERPRETATION AND CONCLUSIONS

**Geology**

The Altar mineralized system is developed at the unconformable contact of Permo-Triassic Choiyoi Group basement and Cretaceous volcanic rocks. It is associated with a Miocene intermediate-composition porphyry stock that has intruded rhyolitic ignimbrites and fine-grained andesite flows. Mineralization is closely associated with the intrusive stock, but is hosted not only in the porphyry phases, but also in the Vitric Tuff and Pachon Andesite.

Peregrine geologists have interpreted the mineralization paragenesis as follows.

Stage 1: Potassic alteration, accompanied by deposition of pyrite–chalcopyrite–bornite and pyrite–molybdenite mineralization.

Stage 2: Sericitic alteration overprint, accompanied by reconstitution of the Stage 1 mineralization as assemblages of pyrite, chalcocite, covellite, bornite and digenite.

Stage 3: Deposition of pyrite–enargite vein systems.

High-sulphidation conditions prevalent in a serecitic alteration event were superimposed on an underlying potassic alteration zone. Sulphides generally exhibit a consistent vertical zonation pattern: pyrite–enargite at the higher levels; pyrite–chalcocite–covellite–bornite–digenite assemblages at intermediate levels, associated with sericite alteration; and pyrite–chalcopyrite–bornite and molybdenite assemblages at deeper levels, associated with potassic alteration.

- There is a well-defined copper, gold and molybdenum anomaly on surface at Altar.

- Mineralization styles have been defined and are appropriate to the deposit-type analogues, namely porphyry-copper and high-sulphidation epithermal mineralization styles.
- The exploration programs completed to date at Altar have defined a telescoped high-sulphidation epithermal–porphyry-copper system that has an area, based on alteration mapping of at least 3.5 km x 3 km.
- Within this system, IP surveys have defined sulphide-rich mineralization of 2.9 km x 1.7 km dimensions. The cores of the central and eastern zones have been confirmed by drilling.
- The sulphide mineralization is divided on a geographical basis into the Central, East and West Zones, and comprises three sulphide zones: pyrite–enargite (paragenetic phase 3), pyrite–chalcocite–covellite (paragenetic phase 2) and pyrite–chalcopyrite–bornite–molybdenite (paragenetic phase 1).
- Mineralization is hosted in quartz diorite porphyry, vitric tuff, andesite, and volcanic and magmatic breccias.
- Sericitic and potassic alteration are associated with the mineralization.
- Drilling is currently on approximate 100 m centres.
- Elevated copper, molybdenum and gold values have been intersected to depths of up to 1010 m
- All holes which reached targeted depths have ended in pyrite–chalcopyrite–bornite–molybdenite sulphide mineralization associated with potassic alteration.
- A number of holes have also ended in pyrite–chalcocite–covellite mineralization, associated with sericitic alteration. This mineralization has yet to be depth-constrained.
- The deposit remains open to the south of the West Zone and east and south of the East Zone as well as at depth.

**Metallurgy**

- Copper recoveries into rougher concentrate ran 93 to 94% at a primary grind of 80% passing 96 to 116 µm. Arsenic recovery ranged from 86 to 95%.
- For the atypical high-arsenic composite, the copper and arsenic recoveries into the rougher were about 96% and 97.5%, respectively, at a primary grind of 107 µm.
- Approximately 50% of the copper losses in the rougher tailings occurred in the minus 20 µm size fraction.

- The CC-CV rougher concentrate assaying 8.3% Cu and 0.06% As was upgraded to 32.2% Cu and 0.21%As in two stages of cleaning after regrinding the rougher concentrate to 80% passing 40 to 50 µm.

- The CP-BN rougher concentrate assaying 5.4% Cu and 0.20% As was upgraded to 25.2% Cu and 0.91% As by cleaning under the same conditions,

- The atypical high-arsenic rougher concentrate assaying 7.6% Cu and 1.67% As was similarly upgraded to 29.1% Cu and 6.32% As.

- Mineralogical examination results indicated that the sulphide minerals were well liberated at the primary grind size used. Important copper minerals were chalcopyrite, covellite, enargite with minor tetrahedrite and bornite.

- The bottle roll tests on a very limited selection of material indicated that the low arsenic CC-CV ore type should be a good candidate for a heap leach operation, subject to confirmatory column leach tests.

- Crushing work indices ranged from 5.9 to 7.9 kW-h/mt and show that the Altar ore types range from very soft to moderately soft and should be easy to crush.

- Ball mill grinding work indices ranged from 11.8 to 13.4 kW-h/mt and show that the Altar ore types have moderate grindability by copper industry standards.

- The abrasion indices for the Altar ore types ranged from 0.07 to 0.15 and show that most materials should cause only limited abrasion on metal surfaces.

- Specific gravity tests on whole and crushed core showed that the Altar ore types have a high degree of internal porosity and void space, which averages about 12%. This should allow good penetration of leach solution into the interior of the ore fragments.

- The oxidized leach cap responds well to the cyanide bottle roll tests. Gold extraction was generally rapid and exceeded 80% in half the samples. However, there was some falloff in extraction as depth increased. Reagent consumption during the gold leaching was generally low.

- All Altar samples tested contained some water soluble material that averaged nearly 2%. The major water soluble constituent is anhydrite ($CaSO_4$), but other soluble metal sulphates and chlorides are also present in many samples. These water soluble species will build up in either a mill circuit or a leach circuit. As a result, water treatment may be required to avoid scaling or corrosion problems.

- A variety of column leach tests are currently underway to assess how ore lithology, head grade, crush size and copper solubility affect copper extraction and acid consumption. Because the tests are not yet complete, it is difficult to draw final conclusions. However, none of the composites being tested has had as high a leach rate as the original bottle roll tests. This may be largely a matter of the much coarser ore size in the column tests.

**Resources**

- The geology is sufficiently well understood to support the resource estimation presented in this report and summarized in the section above
- The oval-shaped sulphide-bearing portion of the 3.5 km x 3 km Altar hydrothermal system has dimensions of about 2.9 km x 1.7 km, based on IP data
- The deposit remains open at depth; the deepest drill hole to date, ALD-43, reached a depth of 1009.9 m and remains in mineralized rock with the average grade of the final 100m exceeding 0.5% Cu.
- Data density and data reliability are considered adequate to support a mineral resource estimate as defined under NI 43-101
- Drill holes from all programs intersected a leach cap ranging from 0 m to 258 m in thickness. This was underlain by a zone of primarily sulphide mineralization that is variably affected by supergene enrichment generally within 105 m of surface. Sulphide mineralization thicknesses range from about 75 m to at least 805 m which is at the vertical limit of the present drilling.
- Average grades for all the assays returned as of the report effective date were 0.26 % Cu, 0.07 g/t Au and 0.002% Mo. Copper grades ranged from below detection to 12.38% Cu.
- The database contains all core data collected on the project to date and has been structured for resource estimation
- QA/QC with respect to the results received to date for the Peregrine 2010 exploration program is acceptable and protocols have been well documented.
- Preliminary metallurgical test work has been completed. The utilized methodologies are in line with industry best practice, and are appropriate for the deposit type.
- Assaying of cyanide and acid soluble copper is ongoing in areas where there may be potential for the application of solvent extraction and electro-winning process technology.
- Measured and Indicated mineral resources total 802 million tonnes grading 0.42% Cu and 0.059 g/t Au at a 0.30% Cu Equivalent cutoff grade

- Inferred mineral resources total 465 million tonnes averaging 0.42% Cu and 0.058 g/t at a 0.30% Cu cutoff grade
- A preliminary economic assessment should be initiated
- Results of the preliminary economic assessment should be used to guide the ongoing exploration plan including the ongoing drilling and data collection process to be used to support a pre-feasibility study.

## 21.0 RECOMMENDATIONS

Studies to support a Preliminary Economic Assessment (PEA) are presently underway and all recommendations pertaining to this from the previous Technical Report are being followed.

Follow-up work on the QDM target including trenching, sampling and initial drilling are already being planned.

Additional recommendations arising from this Technical Report are as follows:

- Additional drilling is warranted to define the lateral extents of shallower copper mineralization to the south and east which could reduce the overall strip ratio.
- Further work to develop a gold heap or dump leach process for gold-bearing leached cap material which would need to be removed as part of the pre-stripping.
- The occurrence of arsenic-bearing minerals within the deposit, such as enargite, needs to be defined. Such material would impact negatively on total recovery in a heap leach operation and on concentrate arsenic grade in a flotation operation.

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# 23.0 DATE AND SIGNATURE PAGE

The effective date of this Technical report, entitled “Technical Report, Altar Project, San Juan, Argentina, is October 4, 2010. The date of the amended report is March 21, 2011.

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